Safety in Mines Research Advisory Committee

Final Project Report

Stability and support requirements for stope backs in the shallow depth mining of steeply dipping vein/tabular deposits

Authors – alphabetically

Johnson R
Quaye GB
Roberts MKC

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Executive Summary

A number of smaller mines in South Africa have orebodies, which are steeply dipping vein or tabular deposits. A variety of mining methods are used to extract these orebodies but many have common stability problems with stope backs and crown or sill pillars. Many of the mines have little rock engineering expertise available to address these problems. This investigation has identified mines which fall within the defined “steeply dipping vein or tabular deposit” as those which have orebodies that dip in excess of 40° and have a width of less than 8 m. During the course of the investigation it was apparent that in some cases the use of an inappropriate mining method was the main cause of the rock engineering hazards. Other rock engineering hazards that were identified were:

- fallout of large structurally controlled blocks
- crown/sill pillar failure
- use of poor, inadequate or no support
- failure to modify mine layout as spans, depth or conditions change
- inappropriately located haulages with respect to the ore body

A detailed literature review was undertaken in order to identify world best practice covering the scope of the project and this was reviewed by Professor Peter Lilly from The Western Australian School of Mines and Dr. Nick Barton of the Norwegian Geotechnical Institute.

Following the recognition that inappropriate mining methods have been used on some of these South African orebodies it was considered that a methodology for the selection of a suitable mining method or for testing if an existing mining method was appropriate, needed to be set out in this project.

Guidelines for the support of stope backs and sidewalls have been addressed by the methodology defined by the empirical stability graph method. A second alternative approach, which is semi-empirical, has also been described.

A method of designing crown and sill pillars to suit the prevailing conditions is described. The support of these pillars draws upon the same support methodology used to support stope backs.
Even with regular systematic support installed in stope backs and pillars, structural features can require additional support. These would typically be large structurally controlled wedges and an approach to identifying such unstable wedges is described as is a method of determining the correct support requirements to stabilise such wedges.

In view of the lack of rock engineering expertise on these mines much of the content of this report has been summarised in a small, easy to use handbook, which will have a role in assisting the mine official with respect to the support of stopes, the design of pillars and the identification of unstable ground. Importantly however it is emphasised that in some situations specialist expertise will be required.
Acknowledgements

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# Table of contents

<table>
<thead>
<tr>
<th>Section</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Executive Summary</td>
<td>ii</td>
</tr>
<tr>
<td>Acknowledgements</td>
<td>iv</td>
</tr>
<tr>
<td>Table of contents</td>
<td>v</td>
</tr>
<tr>
<td>List of Figures</td>
<td>vi</td>
</tr>
<tr>
<td>List of tables</td>
<td>x</td>
</tr>
<tr>
<td>List of tables</td>
<td>x</td>
</tr>
<tr>
<td><strong>1 Introduction</strong></td>
<td>1</td>
</tr>
<tr>
<td>1.1 Research problem statement</td>
<td>1</td>
</tr>
<tr>
<td>1.2 Background and motivation</td>
<td>2</td>
</tr>
<tr>
<td>1.3 Structure of research</td>
<td>3</td>
</tr>
<tr>
<td><strong>2 Identification of rock engineering hazards</strong></td>
<td>4</td>
</tr>
<tr>
<td><strong>3 World’s best practice – literature review</strong></td>
<td>7</td>
</tr>
<tr>
<td>3.1 Introduction</td>
<td>7</td>
</tr>
<tr>
<td>3.2 Characteristics of vein type deposits</td>
<td>8</td>
</tr>
<tr>
<td>3.3 Typical extraction methods</td>
<td>8</td>
</tr>
<tr>
<td>3.3.1 Factors considered for the selection of a mining method</td>
<td>9</td>
</tr>
<tr>
<td>3.3.2 Overhand shrinkage stoping method</td>
<td>10</td>
</tr>
<tr>
<td>3.3.3 Sublevel open stoping</td>
<td>12</td>
</tr>
<tr>
<td>3.3.4 Sublevel caving</td>
<td>14</td>
</tr>
<tr>
<td>3.3.5 Square-set mining method</td>
<td>14</td>
</tr>
<tr>
<td>3.3.6 Cut-and-fill mining method</td>
<td>16</td>
</tr>
<tr>
<td>3.3.7 Classification of the mining methods</td>
<td>18</td>
</tr>
<tr>
<td>3.4 Stability of stope crown and sidewall</td>
<td>19</td>
</tr>
<tr>
<td>3.5 Support strategies of stope back and sidewalls</td>
<td>23</td>
</tr>
<tr>
<td>3.5.1 Reinforcement</td>
<td>23</td>
</tr>
<tr>
<td>3.5.2 Pre-reinforcement</td>
<td>25</td>
</tr>
<tr>
<td>3.5.3 Design of reinforcement systems</td>
<td>25</td>
</tr>
<tr>
<td>3.5.4 Rock mass classification systems</td>
<td>25</td>
</tr>
<tr>
<td>3.5.5 Stability graph method</td>
<td>28</td>
</tr>
<tr>
<td>3.6 Backfill</td>
<td>30</td>
</tr>
<tr>
<td>3.7 Technicalities of sill pillar extraction</td>
<td>31</td>
</tr>
<tr>
<td>3.8 Summary</td>
<td>34</td>
</tr>
</tbody>
</table>
List of Figures

<table>
<thead>
<tr>
<th>Figure</th>
<th>Description</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Figure 1.1</td>
<td>Research methodology</td>
<td>3</td>
</tr>
<tr>
<td>Figure 2.1</td>
<td>Flowchart highlighting the component in the methodology to be discussed in this chapter and its position in the sequence</td>
<td>4</td>
</tr>
<tr>
<td>Figure 3.1</td>
<td>Flowchart highlighting literature review as the focus of this chapter in the research methodology</td>
<td>7</td>
</tr>
<tr>
<td>Figure 3.2</td>
<td>An illustration of the shrinkage stoping method (source: Atlas Copco, 1980)</td>
<td>11</td>
</tr>
<tr>
<td>Figure 3.3</td>
<td>An illustration of the sublevel open stoping method (source: Atlas Copco, 1980)</td>
<td>13</td>
</tr>
<tr>
<td>Figure 3.4</td>
<td>An illustration of the cut-and-fill mining method (source: Atlas Copco, 1980)</td>
<td>17</td>
</tr>
<tr>
<td>Figure 3.5</td>
<td>Schematic representation of the geometry of a cut-and-fill stope</td>
<td>20</td>
</tr>
<tr>
<td>Figure 3.6</td>
<td>Profile of stresses developed around the crown of a cut-and-fill stope for various stope geometries, and for various field stresses</td>
<td>21</td>
</tr>
<tr>
<td>Figure 3.7</td>
<td>Profile of stresses developed around the sidewall of a cut-and-fill stope for various stope geometries, and for various field stresses</td>
<td>21</td>
</tr>
<tr>
<td>Figure 3.8</td>
<td>Schematic representation of a filled stope with sill pillar in place</td>
<td>31</td>
</tr>
<tr>
<td>Figure 3.9</td>
<td>Extraction of a sill pillar by cut-and-fill mining using horizontal drill holes</td>
<td>32</td>
</tr>
<tr>
<td>Figure 3.10</td>
<td>Extraction of a sill pillar by retreat mining using vertical drill holes</td>
<td>33</td>
</tr>
<tr>
<td>Figure 4.1</td>
<td>Flowchart highlighting the component in the methodology to be discussed in this chapter and its position in the sequence</td>
<td>36</td>
</tr>
<tr>
<td>Figure 4.2</td>
<td>Data collection and analysis process</td>
<td>38</td>
</tr>
<tr>
<td>Figure 4.3</td>
<td>The mining method selection process</td>
<td>40</td>
</tr>
<tr>
<td>Figure 5.1</td>
<td>Flowchart highlighting the component in the methodology to be discussed in this chapter and its position in the sequence</td>
<td>48</td>
</tr>
<tr>
<td>Figure 5.2</td>
<td>Support design methodology for the support of stopes</td>
<td>50</td>
</tr>
<tr>
<td>Figure 5.3</td>
<td>Stability graph, no support</td>
<td>51</td>
</tr>
<tr>
<td>Figure 5.4</td>
<td>Stability graph with support</td>
<td>52</td>
</tr>
<tr>
<td>Figure 5.5</td>
<td>Stability graph showing cablebolt design zone for open stopes</td>
<td>53</td>
</tr>
</tbody>
</table>
Figure 7.4. Cross section of an excavation in a rock mass containing two joint sets and the corresponding stereographic projection ..................................................84

Figure 7.5. Cross section of an excavation in a rock mass containing three joint sets and the corresponding stereographic projection ................................................85

Figure 7.6. An illustration of the forces acting on a symmetric triangular roof prism .......87

Figure 7.7. An illustration of the forces acting on an asymmetric triangular roof prism ....88

Figure 7.8. Design of a reinforcement system to prevent sliding of a triangular block.....89

Figure 7.9. Methodology to determine the force required to reinforce an unstable wedge ..................................................................................................................90
List of tables

Table 3.1. Similarities and differences between typical extraction methods for mining narrow-vein reefs ................................................................. 19

Table 3.2. Advantages and disadvantages of some of the most commonly used types of reinforcements in relation to mining ........................................... 24

Table 3.3. Assessing support requirements using the MRMR ........................................... 28

Table 4.1. Explanation of ranking values ........................................................................ 39

Table 4.2. Geometry of deposit and grade distribution .................................................... 40

Table 4.3. Rock characterization ......................................................................................... 41

Table 4.4. Ranking of geometry/grade distribution for different mining methods .......... 41

Table 4.5. Ore zone ........................................................................................................... 41

Table 4.6. Hangingwall ........................................................................................................ 42

Table 4.7. Footwall ................................................................................................................ 42

Table 4.8. Ranking of mining methods based on increasing unit mining cost ............ 42

Table 4.9. Ranking of mining methods based on orebody geometry and grade distribution ................................................................. 44

Table 4.10. Ranking of mining methods based on rock mechanics characteristics of ore zone ......................................................................................... 44

Table 4.11. Ranking of mining methods based on rock mechanics characteristics of the hangingwall ........................................................................... 45

Table 4.12. Ranking of mining methods based on rock mechanics characteristics of the footwall .................................................................................. 45

Table 4.13. Total rank values ............................................................................................. 45

Table 4.14. Q value equivalent to method selection criteria ............................................. 47
1 Introduction

A number of smaller mines in South Africa have orebodies which are steeply dipping vein or tabular deposits. For the purposes of this project, steeply dipping has been taken to include all tabular deposits dipping at greater than 40°, and narrow to include deposits generally having a width of less than 8 m. A variety of mining methods are used to extract these orebodies but many have common stability problems with stope backs and the stability of crown and sill pillars. These types of ore deposits are mined in many parts of the world. In this project it is intended to identify “best practice” with respect to the support of stope backs and crown and sill pillars, as implemented on these mines and to determine what is applicable to South African mines.

Many of these South African mines do not have in house rock engineering expertise. There is a requirement therefore to translate these “best practices” for the support of stope backs and sill and crown pillars into a methodology that can be used by mining personnel. This will take the form of a readily understood booklet providing guidance on the support of stope backs and crown and sill pillars. It will also allow the identification of other potential problems such as incipient wedge failure. The findings from this research will also include material for use in compiling codes of practice.

1.1 Research problem statement

The main tasks of this project are the following:

- Identify mines in South Africa, which have orebodies that are steeply dipping vein or tabular deposits and determine pillar and back stability hazards.

- In order to address these problems, it is essential to determine the world’s best practices for support of stope backs and crown and sill pillars and select and modify, if necessary, the most appropriate for South African conditions and circumstances.

- Compile a readily understood/comprehended booklet that can be used by mine personnel to evaluate the support of stope backs, and crown and sill pillars.
1.2 Background and motivation

The primary motivation for this project is to improve health and safety, particularly in small mines where accident rates are perceived to be high. (By understanding the best techniques for mining “other mines”, improved stability and safety may be achieved).

Many accidents in this sector of South African mining have as their root cause personnel responsible for planning such mines who are deficient in rock engineering and mining practice knowledge. In addition, limited financial resources available to many of these mines lead to the selection of inexpensive options, rather than better, more cost effective solutions. Furthermore there is the perception that lack of discipline, non adherence to best practice standards and risk taking occur on these mines. The incidence of such problems can be reduced by having an appropriate, well comprehended code of practice in place, which to a large extent leads to self regulation.

It was also considered that an easily understood, well indexed and illustrated handbook and guide on the above topics should be produced. This would enable personnel, particularly on small mines, to select the best options for problems associated with regional and local instability, requiring appropriate mine layout and local support solutions. Such a document would be useful for mine management, production personnel, in house and external consultants and DME officials in achieving improved mining method selection, pillar design and back support.

Given that formal, on-mine rock engineering expertise is limited, the handbook should provide the information for safe mine design and operation in an easy to understand format for use by mine management, planners and other technical personnel. This also indicates when more formal rock engineering expertise should be consulted.

It should also be a guide to “best practice” mining which would assist significantly in overcoming these problems, particularly on mines where staff have limited exposure to and experience of various mining methods and rock engineering problems.

Another of the challenges of this work is to assist on-mine staff to ensure timely identification of potential problems and their causes.
1.3 Structure of research

The research follows the flow chart shown in Figure 1.1. Firstly a survey was undertaken to determine the scale of mining of shallow, steeply dipping, vein/tabular orebodies in South Africa as well as to identify the pillar and back related problems for the various identified mining methods.

**Figure 1.1. Research methodology**

A literature review was undertaken to determine the scale and extent of the problem of stability of stope backs and pillars in these types of mines internationally. A further objective of this literature survey was to determine the world’s best practice with respect to the support of stope backs and the design and support of stope pillars. Determining the suitability of the mining method is an important step in the overall rock engineering assessment as an inappropriate mining method can lead to unnecessary problems and be hazardous. Therefore a simplified methodology to assist in the selection of a suitable mining method was defined. Furthermore, methodologies for the design of stope back support and crown/sill pillars were established. Finally, an approach that can be used to identify and evaluate local instabilities is presented.
2 Identification of rock engineering hazards

![Flowchart]

**Figure 2.1. Flowchart highlighting the component in the methodology to be discussed in this chapter and its position in the sequence**

This project was initiated to address rock engineering and related hazards associated with mining steeply dipping tabular/vein deposits in South Africa. In particular, hazards associated with crown and sill pillar design and support as well as the support of stope hangingwall and back areas. For the purposes of this project, steeply dipping has been taken to include all tabular deposits dipping at greater than 40°, and narrow to include deposits generally having a width of less than 8 m.

It was apparent soon after the outset of the project that these conditions do not apply universally, or indeed very widely in the South African mining industry, but that many small mines fall in this category.

In order to make this work as broadly applicable as possible, all mines that could conceivably fall into this category were contacted. All potential mines were obtained from:

- DME lists of mines from regional offices,
- SIMOT list of contributors,
• DME mining directory,
• SA mining directory,
• discussion with senior mining industry representatives,
• people involved in other SIMOT projects (Stacey (2000), Peake (1999))

These mines were contacted in a telephonic survey (see Appendix B). As none of the mines had full-time rock mechanics input, a number of questions were asked of the mine geologist, mine manager or other available technical official. The questions asked included the following:

• operation surface or underground?
• deposit tabular/vein or massive?
• tabular/vein?
• typical dip?
• single or multi-reef/vein?
• depth range of current mining?
• approximate annual production?
• mining method(s) used?

Finally a total of 14 mines, or groups of mines were identified as relevant to this project (see list in Appendix A).

In carrying out this survey, numerous problems were encountered, not least:
• that many mines had closed or partly closed - e.g. many asbestos mines,
• that some mines were temporarily closed - e.g. Klipwal - in the process of being sold,
• that none of the mines had resident rock engineering input, and several had no resident geologist,
• that several mines had no technical expertise sufficient to answer even the simplest of technical questions.
Following an initial questionnaire/screening, visits were made to relevant mining districts and DME offices. In several cases, the mine responses with respect to possible rock engineering hazards did not correspond well with the corresponding inspectorate view. Given this, and the fact that this project was motivated by the high accident record of these mines, this survey was complemented by a review of all available accident records for these mines at DME offices, to gain a better understanding of safety related aspects of such mining. The following hazards/potential hazards were identified as a result of the visits and survey:

- fallout of large structurally controlled blocks,
- poor ground conditions and weak/weathered material,
- haulage location with respect to the orebody,
- cross-cut support at the orebody contact,
- use of inappropriate mining methods,
- stress “bridging” or arching - potential regional failure,
- use of poor, inadequate or no support,
- poor understanding of regional stress distribution,
- crown/sill pillar failure,
- poor access control into working stopes,
- failure to adapt mine layout as spans, depth or conditions change.
3 World’s best practice – literature review

Figure 3.1. Flowchart highlighting literature review as the focus of this chapter in the research methodology

3.1 Introduction

Mining of narrow, steep vein-type deposits is less common in South Africa than the exploitation of flatter tabular deposits. Vein-type deposits are often but not always steeply inclined. Little is known about the few mines of this type that exist in South Africa and there are virtually no formal publications to describe these operations. Effort has therefore been expended in reviewing similar operations elsewhere in the world.

The focus of this review is on narrow vein deposits. In this review, the characteristics of this type of deposit are briefly explained and the typical extraction methods used are summarized. The factors considered in selecting these mining methods are reviewed and the conditions under which a particular mining method is applicable are emphasized. Some information on the stability of stope crowns is also given. General support strategies and techniques usually adopted to stabilize stope backs and sidewalls are presented. Finally, the technicalities of exploiting sill pillars are highlighted.
3.2 Characteristics of vein type deposits

In order to understand the reasons for some of the difficulties encountered in mining vein type deposits, it was considered necessary to briefly review the geotechnical characteristics of these orebodies.

A vein deposit refers to a long narrow body usually emplaced in fissures in the earth’s crust, by hydrothermal and metamorphic processes. Sometimes a clay gouge fringes the vein. Veins have an irregular shape. On the basis of their mode of occurrence, they can be classified as simple veins, complex veins or vein systems. They are termed simple veins if they have no offshoots. Those that have offshoots are referred to as complex veins. The term vein system is used to describe an aggregate of veins that are grouped together.

Depending on the width, a vein can be described as being narrow or wide. In this report, narrow veins include all those that are less than 8 m thick. In addition, narrow veins are considered to include all mineral deposits that are mostly steeply inclined, with dips exceeding the angle of repose of the broken ore. In general, there are certain problems associated with the mining of narrow veins. These problems include:

- complex geology and grade distribution, meaning that reserves are generally difficult to assess without underground development
- a high percentage of wall rock dilution (both planned and additional)
- a low ratio of stope tonnage to metres developed
- low overall efficiencies because of the relatively small tonnages involved
- the inapplicability of large-scale mechanization, particularly in stopes.

3.3 Typical extraction methods

The major mining methods that are used to extract steeply dipping vein deposits include the following:

- overhand shrinkage stoping method
- sublevel open stoping
- sublevel caving
- Alimak Vein Mining (AVM)
- square-set
- cut-and-fill mining method
In all cases mining is undertaken by the drill-blast-muck cycle. The main differences between the above methods stem from the degree of development that must be done before mining can begin and the ore handling strategies associated with each method. In addition, some of these methods easily lend themselves to mechanization whereas others do not.

The sublevel caving method is usually employed to mine orebodies of large horizontal extent, but it has been included in the above list because some mines in South Africa apply it.

For a particular vein deposit there are factors that may favour or preclude the choice of some of the above mining methods. These factors are stated in the following sub-section, and a detailed selection methodology is set out in section 4.

### 3.3.1 Factors considered for the selection of a mining method

There are numerous factors that influence the selection of a mining method. These factors may include the following:

- vein geometry and the physical characteristics of the vein and wall rock
- method adaptable to irregular ore limits, yielding good recovery and minimal dilution
- vein strike length and dip
- consistency of the orebody width
- the dimensions and regularity of ore shoots
- the geomechanical properties of the wall rocks and ore
- nature of the grade distribution
- hydrogeology

In addition to the factors mentioned above, results from the analysis of the pre-mining stress field, the analysis of the stress redistribution induced by mining activity and the analysis of the rock quality and strength and the nature of geological discontinuities are all considered.

A thorough consideration of all these factors ensures that the best mining method is selected, which yields the maximum profit whilst providing acceptable levels of safety. Fortunately a great deal of work has been done in the past and the factors that favour each of the mining methods mentioned above have been established. Each of these mining methods will be briefly reviewed in the following sub-sections.
3.3.2 Overhand shrinkage stoping method

This is a stoping method in which a small percentage of the broken ore is drawn as mining progresses to make room for subsequent in-stope activities including drilling and blasting. Most of the blasted ore is left to accumulate in the stope and drawn after the stope is completely mined.

Suitable orebody characteristics

The suitable characteristics of the orebody and the surrounding rock mass that favour the adoption of the shrinkage mining method include:

- steeply dipping; the dip must exceed the angle of repose
- must be strong and resistant to crushing and degradation during draw
- comparatively stable hangingwall and footwall
- regular ore boundaries
- must be completely inert with no tendency for oxidation, hydrolysis, dissolution or development of cementitious materials

The above characteristics are necessary to ensure that once the ore is blasted, it will remain mobile and amenable to free granular flow during its residence time in the stope.

Key operational features

The key operational features of the shrinkage mining method include the following:

- the excavation is performed in overhand fashion
- no ore-pass development is required in the orebody footwall. Rather an extraction system must be developed at the base of the stope.
- the ore is broken in the stope by stripping the crown of the excavation, with miners working directly under the stope crown.
- some broken material must be drawn through chutes or draw-points to make room for drilling. The draw-down must ensure that about 2 – 2.5 m void is left between the top of the shrink pile and the back of the stope.
- the hangingwall and footwall are stabilized to some degree by the broken ore in the stope.
- about 75% of the ore is locked up until work in the stope is completed.

The method is illustrated in Figure 3.2.
Primary problems

Some of the major problems encountered in stopes where the shrinkage mining method is used include the following:

- The drawdown is often accompanied by in-stope hang-ups caused by cementing of the pile, locking and oversize blocks. Cementation of the shrink is usually related to in situ oxidation and/or mud-like minerals that make the ore sticky. This often results in abandonment of the stope.

- Locking of the shrink can be caused through squeezing of the shrink by gradually failing wall rocks. Shrink pile locking is time-dependent. The longer ore is left undrawn, the greater the likelihood of problems.

- Compaction of the shrink pile during stoping can be exacerbated by horizontal blast holes as the blast force is downwards. Vertical holes allow the sideways or lateral force of the blast to minimize compaction effects.

- In some instances, rockbolts are used to support the hangingwall. Such rockbolts have lengths ranging between 1.8 m and 3 m.
Stope support

In areas where a collapse of the hangingwall is deemed possible, rockbolts are installed within the stopes to give support during drawdown. In this regard, resin and in particular Split-Sets have proved their worth in the United Kingdom.

Safety issues

- Boxhole blockage is common. Sometimes this requires blasting which consequently damages the timber frame.
- Settlement of the shrink pile after drawdown results in the ground running away when miners are drilling.

3.3.3 Sublevel open stoping

This is a method of stoping in which the ore is blasted by benching, ring drilling, or long hole; most of the ore is drawn off as it is blasted, leaving an open stope.

Suitable orebody characteristics

The suitable characteristics of the orebody and the surrounding rock mass that favour the adoption of the sublevel open stope mining method include:

- Steeply dipping deposits, the inclination of the footwall must exceed the angle of repose.
- The hangingwall and footwall must be strong.
- The ore must be competent.
- The boundaries of the orebody must be regular.
- The grade of the ore must be fairly uniform.

Key operational features

A comprehensive development is carried out of haulage drifts, raises, drilling drifts, a loading-drawpoint system and a slot raise. The slot raise is enlarged to the width of the stope. This is done by parallel-hole blasting. The ore is fragmented in the stope using ring-drilled or long parallel blast holes, expanding from the free surfaces provided by the stope slot. The broken ore converges to a drawpoint for extraction. The stope faces and side walls remain unsupported during ore extraction, while local and near field support for the country rock is developed as pillars are generated by stoping.

The method is illustrated in Figure 3.3.
Primary problems

- Rapid changes in vein width or payability may not be recognized and longholes may wander ‘off vein’ if ground conditions change. Strong geological control is required to ensure that ring patterns can be designed to fit the orebody shape.

- Localized falls of rock can be experienced within the drawpoints, but this hazard can be controlled by the use of rockbolts and mesh.

- There are no means to recover mineralization in the wall rock.

Stope support

Backfill of various qualities may be placed in the primary stope voids, and pillar mining may be performed by exploiting the local ground control potential of the adjacent fill. Alternatively, pillars may be blasted into adjacent stope voids, with the possibility of extensive collapse of the local country rock.
Rib and sill pillars that provide lateral support may be left to support the mined-out area. The thickness of the crown pillar varies with the room width, length and stability of the ore. The width of the rib pillar depends on the strength and stability of the ore and the thickness of the orebody.

3.3.4 Sublevel caving

This is an induced caving method in which the ore is blasted by ring drilling from drifts; overlying rock is expected to cave as the ore is drawn.

Suitable orebody characteristics

The suitable characteristics of the orebody and the surrounding rock mass that favour the adoption of the sublevel caving mining method include:

- The method is generally used to mine low-grade ores.
- The rock material in the deposit should be moderately competent, such as a jointed or fractured rock with some joint strength.
- The rock should not be free-caving, but when it is broken small fragments should be formed.
- The hangingwall and footwall can be irregular.
- It can also be applied, within limits, to exploit soft sticky ores which have a tendency to repack.

Key operational features

Operations in the orebody are undertaken in headings developed at comparatively small vertical intervals. Ore is fragmented using blast holes drilled upwards in fans from these headings. Ore is extracted selectively, with front-end loaders operating in the drill heading, from the local concentration of fragmented orebody rock contained within the caved waste. As broken ore is extracted at the drawpoint, fragmented ore and enclosing caved waste displace to fill the temporary void. The success of draw, and the method itself, is determined by the relative mobilities of caved waste and fragmented ore.

3.3.5 Square-set mining method

This is a method in which timber squares are formed to replace the rock mined and to support the surrounding rock. The method includes other timbered stoping methods, such as stull stoping.
Suitable characteristics of the orebody and surrounding rock mass

The suitable characteristics of the orebody and the surrounding rock mass that favour the adoption of the square-set mining method include:

- Grade of ore must be such that the cost of using the square set method will be less than the value of ore.
- Applicable in high grade ores where selective mining and maximum sorting of waste in the stopes is required.
- Applicable in altered or broken enclosing rocks and structurally weak ore, which are not able to support themselves.
- Applicable to orebodies of any size, shape or dip.
- Applicable to ores with a tendency to oxidize when left in the stope. Ore broken by using the square-set method is usually brought to the surface as soon as possible.

Key operational features

The key operational features of the square-set mining method include the following:

- Ore can be mined in underhand or overhand configuration. In running or very loose ground, the ore would probably be mined by downward vertical slices. If the ore has just sufficient strength to stand over a short period, or the ground pressure is largely from the top, upward vertical sections would probably be mined. In still stronger ore or where the pressure is from the walls, horizontal or rill sections would be taken. In narrow veins, horizontal sections would be taken.

- The space created by removal of the ore is timbered with framed sets. The sets are usually square in horizontal section.

- The stope may or may not be filled with waste rock as stoping progresses.

Stope support

Primary stope support comprises framed timber sets with or without waste rock filling.
Safety issues

The degree of safety in square-set stopes depends largely on how well the timber sets are placed. In placing square-sets the most important factor is blocking and pre-stressing or wedging. Wedges should be properly and firmly driven and they should receive attention from time to time to see that they do not become loosened by ground movement or by blasting.

Blast holes must not be overloaded otherwise blasting may damage the timber framework. During the drilling of vertical holes, care must be taken to ensure that drillers are well protected from any loose rock that might fall from the stope back.

It is important to emphasize that the principle of square-set timbering is to render the entire structure of timber as rigid as possible so as not to permit any subsidence or lateral ground movement that might pose danger to miners.

3.3.6 Cut-and-fill mining method

This is a stoping method in which each slice of rock is removed after blasting and is then replaced with some type of fill material, leaving space to mine the next slice.

Suitable orebody characteristics

The suitable characteristics of the orebody and the surrounding rock mass that favour the adoption of the cut-and-fill mining method include:

- The orebody must be steeply dipping, between 35 and 90 degrees.
- The ore must be reasonably firm.
- The method is suitable for orebodies with irregular and discontinuous contours requiring selective mining.
- The geomechanical conditions of the orebody must be good so as to maintain stoping width spans.
- High grade ore is best suited to the method.
- The method is suitable for low, rock mass strength conditions.
Key operational features

The key operational features of the cut-and-fill mining method include the following:

- The excavation is usually performed in overhand configuration. Horizontal slices of the ore are taken. Drilling and blasting, removal of broken ore, filling of void, and sectional flooring typify the mining cycle.

- Driving and further extension of ore chutes is essential.

- High degree of selectivity required to extract high grade ore.

- Sectional flooring is required with steel sheets, wooden boards, or a suitable cloth or else gunited.

- Backfilling of the excavation is achieved with waste rock from development ends, mill tailings, sand etc. Close control of rock mass displacement is achieved with backfill placement.

- The operations can be mechanized and hydraulic filling may be used.

The method is illustrated in Figure 3.4.

Figure 3.4. An illustration of the cut-and-fill mining method (source: Atlas Copco, 1980).
Stope support

- The void is filled with hydraulic fill, broken rock, soil, sand and/or gravel.

Primary problems

- When vertical holes are drilled and the blasted ore is removed, the headroom in the stope is increased to about 7 m. The blasting creates a ragged back that is difficult to control. This endangers the safety of the miners. Horizontal drilling is thus preferred.

- Where hydraulic filling is adopted, high slimes content of the fill may pose problems. For this reason, de-slimed mill tailings are preferred.

Safety issues

- The blast sometimes produces boulders, which may require secondary blasting. Without the secondary blast, the boulders often choke chutes. This may require entry into chutes endangering human life.

3.3.7 Classification of the mining methods

In view of the actual mining practices, general safety considerations and the necessary support strategies, the above methods are often classified into entry and non-entry methods. The entry methods, which include square set, cut-and fill and shrinkage mining methods, require that workers enter into the stopes. In contrast, non-entry mining, examples of which are the sublevel open stope and sublevel caving mining methods, does not require workers to enter the stopes.

So as to identify the issues related to each of the above mining methods that are of rock engineering significance, the main similarities and differences are presented in Table 3.1 below.

It can be deduced from Table 3.1 that both the entry and non-entry mining methods require the design of a stable crown/sill pillar. In addition, the sidewalls and stope backs in the entry mining methods need to be reinforced. The design considerations for a crown/sill pillar and the strategies that are adopted to reinforce the sidewalls and stope backs will be reviewed in the following sub-sections.
Table 3.1. Similarities and differences between typical extraction methods for mining narrow-vein reefs

<table>
<thead>
<tr>
<th>Entry method</th>
<th>Non-entry method</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cut-and-fill</td>
<td>Crown and sill pillar design is necessary.</td>
</tr>
<tr>
<td></td>
<td>Crown pillar design is necessary.</td>
</tr>
<tr>
<td></td>
<td>Stope back is supported.</td>
</tr>
<tr>
<td></td>
<td>Sidewalls are supported.</td>
</tr>
<tr>
<td></td>
<td>Extraction system is developed at the base of the stope prior to mining.</td>
</tr>
</tbody>
</table>

3.4 Stability of stope crown and sidewall

The success of a cut-and-fill mining method and its variant, shrinkage stoping, is crucially dependent on effective control of the stope crown and sidewall. A prerequisite for effective ground control is a fairly good knowledge of the evolution of stresses in the crown and sidewalls of the excavation, during mining.

Consider Figure 3.5 which represents the geometry of a cut-and-fill stope; according to Brady and Brown (1993) the points of particular importance in indicating the stope boundary state of stress are point A, in the centre of the sidewall, and point B, in the centre of the crown of the excavation.
Figure 3.5. Schematic representation of the geometry of a cut-and-fill stope

Assuming that the boundary stresses at points A and B are $\sigma_A$ and $\sigma_B$ respectively, Brady and Brown (1993) using a plane strain analysis stated that:

$$\sigma_A / p = 1 - K + 2W / H$$  \hspace{1cm} (3.1)

$$\sigma_B / p = K - 1 + 2K(H / W)^{1/2}$$  \hspace{1cm} (3.2)

K is the ratio of horizontal and vertical field stresses
$\sigma_A / p$ is the stress concentration factor on the sidewall

$\sigma_B / p$ is the stress concentration factor on the crown

Considering the stope geometry shown in Figure 3.5, equation 3.1 evaluates the stope sidewall stress, while equation 3.2 evaluates stope crown stress by considering that some local curvature develops in the stope crown.

By using equations 3.1 and 3.2, the crown and sidewall stresses developed around a cut-and-fill stope for various stope geometries, and for various field stress conditions are shown in Figures 3.6 and 3.7 respectively.
Figure 3.6. Profile of stresses developed around the crown of a cut-and-fill stope for various stope geometries, and for various field stresses

Figure 3.7. Profile of stresses developed around the sidewall of a cut-and-fill stope for various stope geometries, and for various field stresses
Based on the fact that k-ratios in the vicinity of mineral veins are often more than unity, Brady and Brown (1993) concluded that low states of stress, which are frequently tensile, are generated in the sidewalls of the excavation. A jointed or fractured rock mass will de-stress and may disintegrate due to loss of shear strength.

Usually, mining of mineral veins proceeds in situations where the k-ratio is greater than unity. By considering the usual stope height to width ratios, Brady and Brown (1993) indicated that crown stress concentration factors exceeding 10 are to be expected. This can give rise to very high stresses. If the uniaxial compressive strength of the vein is less than the prevailing stress, local fracturing may occur in the stope crown. Barton (2000) suggests that applying the Q-system under such conditions, would give rise to highly unfavourable values of the stress reduction factor (SRF) due to the unfavourable stress concentration. Brady and Brown further explained that because the crown stress concentration factor increases with the stope height to width ratio, the geomechanical state of the active mining domain must deteriorate as stoping proceeds up-dip.

A major deduction from the discussions so far is the increasing need for crown support and reinforcement as mining progresses up-dip. In stopes where the height to width ratio is low, the stope crown may require little or no support. If the characteristics of mining induced fractures are such that local instabilities exist, rockbolting may be used to secure the loose rock surface. At deeper levels in shallow mines, the generation of extension fractures may occur and their interaction with the rock structure might render rockbolting ineffective and cable bolting could perhaps be considered. Otherwise, up-dip mining strategies might have to be abandoned and other alternatives considered to allow for further exploitation of the orebody. However, under these conditions Barton (2000) suggests that the dilatant nature of the fractured rock should not be neglected.

The first alternative may involve changing the direction of mining from an up-dip to down-dip configuration. The second alternative is to leave a pillar above the stope crown and resume mining at a higher elevation. The size of the pillar is determined by several factors including the strength characteristics of the orebody, the prevailing rock stress and pillar contact conditions with hangingwall and footwall. These include geometry of the surfaces, planar or irregular, and cohesion between ore and host rocks. A height of twice the stoping width is often used. Subsequently, the resulting floor pillar might be recovered by some other method (this will be discussed in detail later). The most practical
alternative would require cable reinforcement of the stope crown and allow mining to proceed in the overhand fashion.

The last alternative which appears to be more practically feasible, highlights the need for crown support and reinforcement as mining progresses up-dip along a mineral vein. The following sub-sections will be aimed at reviewing some of the available techniques used to reinforce the sidewall and the stope crown.

3.5 Support strategies of stope back and sidewalls

The discussions so far have indicated that one of the greatest challenges encountered in the exploitation of steeply dipping narrow vein deposits is to provide adequate support of the stope back and sidewalls. The appropriate support strategies will ensure the safety of miners and stability of the workings. In the mining of vein-type deposits, three main ground control measures are used. In broad terms, these can be categorized into reinforcement, pre-reinforcement and backfill techniques.

3.5.1 Reinforcement

Potentially unstable rock near the boundary of mine excavations may be reinforced with penetrative elements such as cable-bolts, grouted tendons, or rock anchors. This type of ground control generates a locally sound stope boundary within which normal production activity can proceed. On the basis of the different anchoring techniques, rockbolts can be grouped into the following categories:

- mechanically anchored rockbolts
- grouted rockbolts
- grouted cable bolts
- friction anchored rockbolts

Table 3.2 lists the advantages and disadvantages of some of the most commonly used types of reinforcements.
Table 3.2. Advantages and disadvantages of some of the most commonly used types of reinforcements in relation to mining

<table>
<thead>
<tr>
<th>Category of bolt</th>
<th>Type of bolt</th>
<th>Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mechanically anchored rockbolts</td>
<td>Expansion shell anchor</td>
<td>Relatively inexpensive. Immediate support action after installation. The bolt can serve as semi-permanent reinforcement. In hard rock, quite high bolt loads can be achieved. Assuming hard rock conditions it’s a versatile system for reinforcement.</td>
<td>Limited to use in moderately hard to hard rock. Difficult to install reliably. Must be monitored and checked for proper tension. Loses bearing capacity as a result of blast vibrations and corrosion.</td>
</tr>
<tr>
<td></td>
<td>Rebar</td>
<td>The bolt gives quite rapid support action after installation. If a fast-setting resin is used for bottom anchoring of the bar, the fully grouted rockbolt can be tensioned. High corrosion resistance in permanent installations.</td>
<td>Difficulties with resin cartridges in underground environment, which can affect installation reliability. Resin can be messy and hazardous to handle as well as wasteful. Resin has a limited shelf life.</td>
</tr>
<tr>
<td></td>
<td>Dywidag steel</td>
<td>Properly installed, it is a competent and durable reinforcement system. Effect of corrosive environments is minimal. The system gives high bolt loads in various rock conditions.</td>
<td>It is expensive. Tensioning of the bolt is possible only if special installation procedures are followed. Use of standard cement in the grout requires several days curing before the bolt can take load. Quality of grout is difficult to check and maintain constant.</td>
</tr>
<tr>
<td></td>
<td>Flexirope</td>
<td>Properly installed, it is a competent and durable reinforcement system, and also economic. Can be installed to any length in narrow areas. The system gives very high capacity in various rock conditions, as well as high corrosion resistance in permanent installations.</td>
<td>Tensioning of the cable bolt is possible only if special installation procedures are followed. Use of standard cement in the grout requires several days curing before the cable can be loaded.</td>
</tr>
<tr>
<td></td>
<td>Split Set</td>
<td>Simple installation. Gives immediate support action after installation. No hardware other than a jackleg or jumbo boom for installation.</td>
<td>Relatively expensive. Borehole diameter is crucial in the prevention of failure during installation and in the provision of the intended holding force. Less capacity than grouted bolts Successful installation of longer bolts can be difficult. Cannot be used in long-term installations unless protected against corrosion.</td>
</tr>
<tr>
<td></td>
<td>Swellex</td>
<td>Rapid and simple installation. Gives immediate support action after installation. Can be used in a variety of ground conditions. The installation causes contraction in the bolt length. This effectively tensions the face plate against the rock surface.</td>
<td>Relatively expensive. Corrosion protection required if used in long-term installations. Requires a pump for installation.</td>
</tr>
</tbody>
</table>
3.5.2 Pre-reinforcement

In cases where circumstances do not allow adequate support or reinforcement of the rock mass to be provided immediately after excavation, it may be practicable to pre-reinforce the rock mass. This however demands that suitable access be available.

Generally, the pre-reinforcement technique is mostly applied in cut-and-fill mining operations. The materials that are often used include grouted rods or cables. These materials are not pre-tensioned and as such, their behaviour is more passive than active. Such pre-reinforcement is considered to be effective because it allows the rock mass to deform a small amount in a controlled manner, and mobilize its strength, but limits the amount of mass dilation and loosening that can occur.

In using cables to pre-reinforce stope crowns, the installation pattern is dependent on the rock mass quality. For general pre-reinforcement purposes, they are installed normal to the rock surface of the most unfavourable discontinuity. In areas where shearing on a particular discontinuity is anticipated, it is recommended that the cables be installed at an angle of $20^0-40^0$ to the discontinuity, to help mobilize both shear and normal stress when the cables are in tension.

3.5.3 Design of reinforcement systems

The preceding sub-sections have highlighted the importance of rock reinforcement and pre-reinforcement. The advantages and disadvantages of the materials that are used in providing these types of support have also been summarized.

Often, the design of reinforcement systems is based on precedent practices or previous experience in similar geological settings. Recent advances in this area have produced scientific methodologies that allow for more comprehensive design calculations regarding rock-support interaction. These take into account the deformation and strength properties of the reinforcement system and the post-peak properties of the rock mass. Some of these techniques will be reviewed briefly in the following sub-sections.

3.5.4 Rock mass classification systems

It is believed that there is no single parameter or index, which can fully and quantitatively describe a jointed rock mass, which may surround an underground excavation. Consequently, some combination of factors such as RQD and the influence of clay filling and weathering appear to be necessary. A number of rock mass classification systems
starting from Terzaghi’s rock load factor have therefore been proposed over the years. The ultimate aim of these classification systems is to give an approximate quantitative measure of the average strength of the rock mass. This information, when compared to the expected resultant stress due to the mining activities, would facilitate the recommendation of appropriate support systems.

The most commonly used systems, which have found various applications in mining include the NGI Q system, Bieniawski’s rock mass rating system (RMR), and the mining rock mass rating (MRMR). A brief summary of each of these systems is given below. Interested readers are referred to Bieniawski (1989) and Barton and Grimstad (1994) for further detailed information about these systems.

**The NGI, Q system**

The rock mass quality is a function of six parameters, each of which has a rating of importance, which can be updated during subsequent excavation. The six parameters are as follows, the RQD index, the number of joint sets, the roughness of the least favourable joints, the degree of alteration or filling along the least favourable joints, and two further parameters, which account for the stress/strength ratio, and degree of water inflow. In combination these parameters represent the relative block-size, the inter-block shear strength, and the active stress. These six parameters chosen to describe the rock mass quality Q are combined in the following way:

\[
Q = \frac{RQD}{J_n} \times \frac{J_r}{J_w} \times \frac{J_a}{SRF}
\]

RQD = rock quality designation
J_n = joint set number
J_r = joint roughness number
J_a = joint alteration number
J_w = joint water reduction factor
SRF = stress reduction factor

The ratings for each parameter were developed by trial-and-error using more than 1000 case records in order to relate their rock mass quality index to the behaviour and support requirements of numerous underground excavations. Barton et al (1974) defined an additional quantity, called the equivalent dimension \(D_e\) of the excavation. This quantity can be simply expressed as follows:
The ESR is related to the use for which the excavation is intended and the extent to which some degree of instability is acceptable. Barton and Grimstad (1994) have given updated values for ESR and their associated excavation categories. For various values of the rock mass quality index, Q and $D_e$, the support requirements of the excavation can be assessed from a design chart. The maximum unsupported spans for different ESR can also be estimated as follows:

$$Span = 2 \times ESR \times Q^{0.4}$$

Various other parameters such as seismic velocity and static deformation modulus have since been linked to the Q-value by Barton (1995). These relationships enhance the validity of Q as a quantitative measure of rock mass stability and behaviour.

**The CSIR geomechanics rock mass classification system (RMR)**

This classification system was proposed and later updated by Bieniawski (1989). In his classification, Bieniawski contends that the six most significant parameters that can describe the behaviour of jointed rock masses are uniaxial compressive strength of rock material, drill core quality, RQD, spacing of joints, orientation of joints, conditions of joints, and ground water inflow.

For a particular rock mass condition, rating values are estimated for each of the above six parameters. In the case of RMR these rating values are added to achieve an overall rating value for the rock mass. The overall rating is adjusted for joint orientation by applying some correction factors. The overall RMR rating can also be related to the stand-up time of an active unsupported span (i.e. the length of advance) of an excavation without support.
The mining rock mass rating (MRMR)

Laubscher introduced the MRMR system in 1977. It was introduced as a further development of the RMR geomechanics classification system to cater for diverse mining situations. The fundamental difference was the recognition that rock mass ratings (RMR) had to be adjusted according to the mining environment so that the final ratings (MRMR) could be used for mine design. The adjustment parameters are weathering, mining-induced stresses, joint orientation, and blasting effects. The geological parameters that must be assessed include the intact rock strength (IRS), joint/fracture spacing, and joint condition/water.

In applying the MRMR to assess the stability of open stopes, the combination of the joint condition rating and joint orientation data can be expressed as a percentage. These percentages can be used to define areas requiring support as suggested in Table 3.3.

<table>
<thead>
<tr>
<th>Percentage range</th>
<th>Support requirements</th>
</tr>
</thead>
<tbody>
<tr>
<td>60% - 70%</td>
<td>Highly unstable, collapse with blast, requires pre-support.</td>
</tr>
<tr>
<td>70% - 80%</td>
<td>Unstable, time dependent falls, may require pre-support.</td>
</tr>
<tr>
<td>80% - 90%</td>
<td>Relatively stable, requires support or scaling, even light blasting.</td>
</tr>
<tr>
<td>90% - 100%</td>
<td>Stable</td>
</tr>
</tbody>
</table>

3.5.5 Stability graph method

In this method, many of the key parameters characterizing the rock mass and stress conditions are quantified in order to determine optimum stope dimensions. Assuming that all factors having a major effect on stability are taken into account, this type of analysis allows the empirical quantification of the supporting effect of cable bolts. Furthermore, this method can assist in the determination of the most suitable cable system and orientation by analysing the possible failure mechanism.

In applying the stability graph design method, two factors are developed: the (modified) stability number representing the capacity of the rock mass to stand up under a given stress condition, and the hydraulic radius accounting for stope plane size and shape. These two factors can be plotted on the revised stability graph in order to assess stope stability.
Potvin et al (1989) have applied the stability graph method to develop guidelines for cable bolt design in open stopes. In developing the guidelines, they used the stability graph to analyse sixty-six case histories collected in Canadian open stope mines. According to their work, the essential factors for the design of a cable bolt support system include density of bolting, bolt length, bolt orientation, and the original rock mass competency. The results obtained from the analyses of the case histories have enabled them to develop empirically calibrated charts from which the essential parameters can be estimated.

**Density of bolting**

The primary aim of using a cable bolt is to prevent significant movement along existing discontinuities. For a highly discontinuous rock mass, a greater number of cable bolts would be needed to stabilize the rock mass. Hence the density of bolting must be related to the intensity of discontinuities. This can best be done if the relative block sizes are known. According to Potvin et al (1989), this can be estimated by computing the ratio of the block size parameter (RQD/Jn) and the hydraulic radius of the stope surface. The hydraulic radius is calculated by dividing the stope plane area by its perimeter. Knowing the ratio of the block size and hydraulic radius, the bolt density can be read off design chart. It must be mentioned that the block size parameter (RQD/Jn) is the same as that defined in the Q system.

**Bolt length**

For effective ground control, the length of a cable bolt must be long enough to anchor firmly in the undisturbed rock mass. From some numerical modelling investigations, Potvin et al (1989) believe that a rough relationship is expected to exist between the hydraulic radius of the stope surfaces and the length of the cable bolt used. For a hydraulic radius range of 1 to 10 m, the estimated bolt length lies in a range of 3 m to about 18 m respectively. Plotting a graph of cable length against hydraulic radius for the ranges indicated, the relationship is linear.

**Bolt orientation**

According to Potvin et al (1989), there are several possible modes of failure and the orientation of the cable bolts should be designed to best resist the most likely mode of failure. The three modes of failure include gravity fall, sliding, and slabbing or buckling.
For conditions, which are conducive to gravity fall or slabbing, the cable should be
installed vertically. In the case of sliding, the most efficient design is when the cables are
installed at an angle between 17 degrees and 27 degrees to the shear direction. This
according to Barton (2000) is approximately the expected range of the residual friction
angle for the expected failure surface. He therefore suggests that force diagrams be used
to check such designs.

Details of the stability graph method are given in Section 5 of this report.

3.6 Backfill

In areas where either conventional cut-and-fill mining or open stoping is adopted to exploit
vein type deposits, backfill can be used as a support medium. In the case of the cut-and-
fill mining method, the fill is introduced periodically, during the progressive extension of
the stope. The effectiveness of the fill is assessed on the basis of its capacity to produce
a stable working surface soon after its emplacement in the stope. Where backfill is used
in an open stoping operation, fill placement in a particular stope is delayed until production
from it is complete.

Brady and Brown (1993) are of the view that a successful performance of a fill mass
requires that during pillar recovery, free-standing walls of fill, capable of withstanding
static and transient loads associated with adjacent mining activity, must be sustained by
the medium.

In the design of backfill, Brady and Brown (1993) recommend that great care be taken to
ensure that significant pore pressure cannot develop in the body of the backfill. Otherwise
this may lead to complete loss of shear resistance and subsequent liquefaction of the
medium. Consequently, the potential for catastrophic flow of fill under high hydrostatic
head may result.

A survey of Canadian mining practice (Thomas et al., 1979) indicated that about 76% of
all mine fill is derived from mill tailings, and that 84% of all fill is transported and distributed
in stopes as an hydraulic suspension. This type of fill is called hydraulic fill or sandfill.
Sandfill is a cohesionless material, with purely frictional resistance to deformation. It is
prepared from concentrator tailings by hydrocyclone treatment to remove the slimes, or
clay-size fraction.
The other types of fill are cemented sandfill and rockfill. Cemented sandfill is obtained by adding various cementing agents to the sand mass. This can provide a significant cohesive component of strength at a relatively low proportional addition to the medium. The resulting mix is transported underground as a suspension at about 70% solids.

In areas where the demand for fill material is higher than the available supply, the mine void can be filled simultaneously with aggregates or similar dry rockfill and cemented sandfill. Some cost savings relative to using only cemented fill can be achieved in the process. The composite fill is placed by discharging cemented sandfill slurry and rockfill into the stope simultaneously.

3.7 Technicalities of sill pillar extraction

It has been mentioned earlier that the cut-and-fill mining method requires the design of a sill pillar. A sill pillar is an ore or rock pillar disposed at the base of a block, extending vertically between the haulage and floor levels. Sill pillars cut in high-grade ore are done so with the intention of extracting them at a later stage. Often, this happens when the stope above is completely mined and filled (see Figure 3.8).

![Figure 3.8. Schematic representation of a filled stope with sill pillar in place](image)
In extracting the sill pillar, people work directly below the emplaced fill in the stope above. This means that the bottom slice of the fill must be stabilized to ensure safety of personnel. In some cases, this is done just after mining and placing the first layer of backfill in the stope above. In other cases this is done only at the sill pillar extraction stage.

Whatever the case may be, the mining of the sill pillar is associated with many problems. These may include failure of the rock due to high stresses, difficulties in controlling the fill above, overall stability of the stope and special requirements regarding the technique for backfilling.

Marklund and Andersson (1998) have evaluated the requirements and techniques for stabilizing the bottom slice of fill. In their investigation, they had to make a choice between the two principal methods of mining the sill pillar. These methods are conventional cut-and-fill mining with horizontally drilled rounds and retreat mining. These methods are illustrated in Figures 3.9 and 3.10 respectively.

In making a choice between these mining methods, the quality of the fill, the rock conditions in the ore and sidewall as well as the geometry of the sill pillar are considered.

\[Figure\ 3.9.\ \textit{Extraction \ of \ a\ sill\ pillar\ by\ cut-and-fill\ mining\ using\ horizontal\ drill\ holes}\]
Retreat mining method

There are three alternatives for retreat mining of the sill pillar. The first alternative is applicable in cases where the bottom slice of the fill above the sill pillar is stabilized. Successful extraction of the entire sill pillar depends on the stability of the stabilized fill across its entire length.

The second alternative can be applied if the fill above is not stabilized. In this case, as mining progresses, the fill is allowed to cave. This caving is acceptable if the geometry of the ore and the properties of the fill are such that fill dilution can be kept at an acceptable level during mucking.

The third alternative is to leave a thin slice of ore to support the fill in the stope above. Due to the fact that the slice of ore left might have been damaged by blasting and high stresses, some form of reinforcement is needed to provide support.

Figure 3.10. Extraction of a sill pillar by retreat mining using vertical drill holes
Conventional cut-and-fill mining

There are four alternative methods for conventional cut-and-fill mining of sill pillars. All these alternatives involve the drilling of horizontal drill holes. The bottom slice of the fill above the sill pillar may have been stabilized or not.

In the case where it has been stabilized, it must be strong enough to allow mining without any further stabilization or reinforcement. If the quality of the fill does not allow this, rock bolting, shotcreting and even spiling can be applied to achieve the desired stability.

If however, the fill is not stabilized in advance by cement addition, the available stabilization alternatives are extensive rock support, stabilization using cement injection or freezing as mining progresses, leaving a thin slice of ore, or a combination of these.

3.8 Summary

The review has revealed that because of the nature of their occurrence, narrow vein-type deposits do not easily lend themselves to extensive mechanization. The most widely adopted mining methods used to extract these deposits are shrinkage mining, Alimak vein mining, sublevel stoping, and cut-and-fill mining methods.

In recent times, the cut-and-fill mining method has been the one most often applied, especially in high grade ores where some degree of selectivity is required. General stope support strategies include the use of various qualities of backfill. The different types of backfill have been mentioned. Strategies that are often adopted to ensure proper emplacement of the fill and to derive maximum benefits from the fill have also been discussed.

The stope crown and sidewalls can be stabilized with different types of reinforcing techniques and materials. Conditions under which pre-reinforcement are applicable have been highlighted. The common rock mass classification schemes that are mostly used in mining applications have been summarized. Since the ultimate aim of these classification systems is to prescribe support requirements for specific ground conditions, the different reinforcing materials with their advantages and disadvantages have been discussed.

Among the reinforcements currently used, cable bolts have proved their reliability and are most favoured. It was established during the review that most of the currently used
reinforcement systems were designed on the basis of experience. So as to improve and facilitate these designs, some of these experiences and historical data have been used to develop design charts. A typical example is the stability graph method, which allows stope stability studies to be conducted. Using the results from these analyses and some other design charts, the density, length and orientation of cable bolts can be determined for open stopes.

Furthermore some attention has been paid to the techniques for the successful extraction of sill pillars. It has been established that sill pillars can be extracted by one of two methods. These methods are conventional cut-and-fill mining and retreat mining. The choice of the most suitable method is governed by the quality of fill, rock conditions in the ore and sidewalls as well as the geometry of the sill pillar.

A major deduction that can be made from this review is that significant knowledge exists about appropriate mining and support strategies that ensure the safe exploitation of narrow, steeply dipping vein-type deposits. However, this information still needs to be tested to ascertain its applicability in similar South African mining environments.
4 Determination of the suitability of a mining method

![Flowchart](image)

Figure 4.1. Flowchart highlighting the component in the methodology to be discussed in this chapter and its position in the sequence

4.1 Introduction

The focus on the mining method in this section stems partially from seeing an inappropriate mining method being used on one of the mine visits. It was clear that the weakness of the ore and the surrounding rock was giving rise to major hazards with respect to the workers in this section of the mine.

The broad range of factors which are considered in choosing a mining method for extracting vein-type deposits were identified and listed in the literature review section of this report. Although all these factors deserve some degree of consideration, detailed analysis of the most influential parameters must be conducted during the feasibility study stage.

These parameters include the geometry and grade distribution of the deposit, and the rock mechanics characteristics of the ore zone, hangingwall and footwall. Primarily, the mining method selection should be based on these parameters and on the mining and
capitalization cost. However, in view of the significant influence of the rock mechanics characteristics on the stability of the excavation, this be given the first priority.

In the following sub-sections, a numerical method is discussed, which facilitates the identification of two or three potential mining methods for extracting a narrow orebody. The processes leading to the selection of these mining methods are explained and an example is given for illustration purposes. It must also be highlighted that the purpose of using the numerical method is not to select the final mining method, but to indicate two or three potential mining methods, which must go through economic evaluation.

4.2 Data requirements and analysis

For the selection of the potential mining methods and initial mine layout, the most important data required are geologic sections and level plans, a grade model of the deposit, and rock mechanics characteristics of the deposit, footwall, and hangingwall. This data is collected from drill cores during the initial core logging or assaying and can be analysed as follows:

- Identify major rock types, alteration zones, and major structures, such as faults, veins, and fold axes from drill cores.
- Prepare geologic sections and level maps to highlight major differences in the above parameters.
- Define the alteration zones on separate set of maps, which can then be overlain onto the rock type geology maps.
- The geologic sections and level maps should be prepared to the same scale as will be used for mine planning.
- Draw sections to true scale without any exaggerations so as to make it easier to visualize the relative layout of mine workings.
- The area included on the maps should extend, horizontally in all directions, 1.75 times the depth beyond the limit of the orebody. This will ensure that there is sufficient information for evaluating the limits of ground surface movement due to mining.
- Develop a grade model of the vein deposit to the same scale as the geologic sections and level maps. Superimpose the grade model on the geologic sections and level maps.
• Contour the combined maps by grade or colour the blocks by grade categories. This will indicate the dominant rock types, as well as their spatial relationships to the orebody.

The above process is simplified in Figure 4.2 below. Having identified the potential mining blocks and delineated zones characterized by major geological structures and rock types, the next step will be applying the numerical method to indicate the two or three potential mining methods.

**Figure 4.2. Data collection and analysis process**

### 4.3 Mining method selection – A numerical approach

In view of the high capital investment needed to open up a new mineral deposit, it is of paramount importance that the mining method selected after the feasibility studies should have a high probability of attaining the projected production rates. It is therefore imperative that all the potential mining methods be thoroughly examined and the most suitable mining method selected to meet the economic goals of the operation. For this reason, a numerical method developed by Nicholas is suggested, which considers some of the most critical parameters that impact on mining method selection.
In the numerical method, there are four ranks, namely preferred, probable, unlikely and eliminated. The ranks and their meaning, and their respective rank values are given in Table 4.1. In Table 4.1, values for the eliminated rank value were chosen so that if the sum of the characteristics values equalled a negative number, the method would be eliminated. A zero value was chosen for the unlikely rank because it does not add to the chance of using the method, but neither does it eliminate the method. The values used for probable and preferred were chosen so that the characteristics for one parameter could be ranked within a mining method and between mining methods.

<table>
<thead>
<tr>
<th>Rank</th>
<th>Description</th>
<th>Ranking value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Preferred</td>
<td>The characteristic is preferred for the mining method</td>
<td>3 – 4</td>
</tr>
<tr>
<td>Probable</td>
<td>If the characteristic exists, the mining method can be used</td>
<td>1 – 2</td>
</tr>
<tr>
<td>Unlikely</td>
<td>If the characteristic exists, it is unlikely that the mining method should be applied, but does not completely rule out the method</td>
<td>0</td>
</tr>
<tr>
<td>Eliminated</td>
<td>If the characteristic exists, then the mining method could not be used</td>
<td>-49</td>
</tr>
</tbody>
</table>

In applying the numerical method, it is suggested that the mining method selection process be done mainly in six steps:

- **Step 1**: Identify and list the potential mining methods.
- **Step 2**: Identify the characteristics, which best describe the orebody geometry and grade distribution from Table 4.2.
- **Step 3**: Identify the rock mechanics characteristics of the ore zone, hangingwall and footwall as described in Table 4.3.
- **Step 4**: For each potential mining method rank the characteristics of the deposit, as defined in Tables 4.4.
- **Step 5**: For each potential mining method rank the rock mechanics characteristics for the ore zone, hangingwall, and footwall according to Tables 4.5, 4.6 and 4.7 respectively.
- **Step 6**: Add up the values from steps 4 and 5 for the combination of characteristics defined in steps 2 and 3 respectively to obtain the final ranking.

Figure 4.3 is presented to facilitate the understanding of the above steps.
**Figure 4.3. The mining method selection process**

**Table 4.2. Geometry of deposit and grade distribution**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Description</th>
<th>Other comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>General shape</td>
<td>equi-dimensional</td>
<td>all dimensions are on the same order of magnitude</td>
</tr>
<tr>
<td></td>
<td>platy – tabular</td>
<td>two dimensions are many times the thickness, which does not usually exceed 100 m</td>
</tr>
<tr>
<td></td>
<td>Irregular</td>
<td>dimensions vary over short distances</td>
</tr>
<tr>
<td>Ore thickness</td>
<td>Narrow</td>
<td>&lt;10 m</td>
</tr>
<tr>
<td></td>
<td>Intermediate</td>
<td>20° – 55°</td>
</tr>
<tr>
<td></td>
<td>Steep</td>
<td>&gt;55°</td>
</tr>
<tr>
<td>Depth below surface</td>
<td>Provide actual depth</td>
<td></td>
</tr>
<tr>
<td>Grade distribution</td>
<td>Uniform</td>
<td>The grade at any point in the deposit does not vary significantly from the mean grade for that deposit</td>
</tr>
<tr>
<td></td>
<td>Gradational</td>
<td>Grade values have zonal characteristics, and the grades change gradually from one to another</td>
</tr>
<tr>
<td></td>
<td>Erratic</td>
<td>Grade values change radically over short distances and do not exhibit any discernible pattern in their changes</td>
</tr>
</tbody>
</table>
Table 4.3. Rock characterization

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Description</th>
<th>Other comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rock substance strength</td>
<td>Weak</td>
<td>&lt;8</td>
</tr>
<tr>
<td></td>
<td>Moderate</td>
<td>8 – 15</td>
</tr>
<tr>
<td></td>
<td>Strong</td>
<td>&gt;15</td>
</tr>
<tr>
<td>Fracture Spacing (fractures/m)</td>
<td>&gt;16 (very close)</td>
<td>RQD between 0 and 20</td>
</tr>
<tr>
<td></td>
<td>10 – 16 (close)</td>
<td>RQD between 20 and 40</td>
</tr>
<tr>
<td></td>
<td>3 – 10 (wide)</td>
<td>RQD between 40 and 70</td>
</tr>
<tr>
<td></td>
<td>&lt;3 (very wide)</td>
<td>RQD between 70 and 100</td>
</tr>
<tr>
<td>Fracture Shear Strength</td>
<td>Weak</td>
<td>clean joint with a smooth surface or fill with material whose strength is less than rock substance strength</td>
</tr>
<tr>
<td></td>
<td>Moderate</td>
<td>clean joint with a rough surface</td>
</tr>
<tr>
<td></td>
<td>Strong</td>
<td>joint is filled with a material that is equal to or stronger than rock substance strength</td>
</tr>
</tbody>
</table>

Table 4.4. Ranking of geometry/grade distribution for different mining methods

<table>
<thead>
<tr>
<th>Mining method</th>
<th>General shape T/P</th>
<th>Ore thickness N</th>
<th>Ore plunge I</th>
<th>Grade Distribution U G E</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sublevel Stoping</td>
<td>2</td>
<td>1</td>
<td>1</td>
<td>3 3 1</td>
</tr>
<tr>
<td>Sublevel Caving</td>
<td>4</td>
<td>1</td>
<td>-49</td>
<td>3 3 1</td>
</tr>
<tr>
<td>Shrinkage Stoping</td>
<td>2</td>
<td>1</td>
<td>1</td>
<td>3 2 1</td>
</tr>
<tr>
<td>Cut &amp; Fill</td>
<td>4</td>
<td>4</td>
<td>4</td>
<td>3 3 3</td>
</tr>
<tr>
<td>Square Set</td>
<td>4</td>
<td>4</td>
<td>4</td>
<td>3 3 3</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>T/P = Tabular or platy</th>
<th>N = Narrow</th>
<th>I = Intermediate</th>
<th>S = Steep</th>
<th>U = Uniform</th>
<th>G = Gradational</th>
<th>E = Erratic</th>
</tr>
</thead>
</table>

Table 4.5. Ore zone

<table>
<thead>
<tr>
<th>Mining method</th>
<th>Rock Substance Strength W M S</th>
<th>Fracture Spacing VC C W WW</th>
<th>Fracture Strength W M S</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sublevel Stoping</td>
<td>-49 3 4</td>
<td>0 0 1 4 4 4 0 2 4</td>
<td></td>
</tr>
<tr>
<td>Sublevel Caving</td>
<td>0 3 3</td>
<td>0 2 4 4 0 2 2</td>
<td></td>
</tr>
<tr>
<td>Shrinkage Stoping</td>
<td>1 3 4</td>
<td>0 1 3 4 0 2 2</td>
<td></td>
</tr>
<tr>
<td>Cut &amp; Fill</td>
<td>3 2 2</td>
<td>3 3 2 2 3 3 2</td>
<td></td>
</tr>
<tr>
<td>Square Set</td>
<td>4 1 1</td>
<td>4 4 2 1 4 3 2</td>
<td></td>
</tr>
</tbody>
</table>
### Table 4.6. Hangingwall

<table>
<thead>
<tr>
<th>Mining method</th>
<th>Rock Substance Strength</th>
<th>Fracture Spacing</th>
<th>Fracture Strength</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>W M S</td>
<td>VC C W VW</td>
<td>W M S</td>
</tr>
<tr>
<td>Sublevel Stoping</td>
<td>-49 3 4</td>
<td>-49 0 1 4</td>
<td>0 2 4</td>
</tr>
<tr>
<td>Sublevel Caving</td>
<td>3 2 1</td>
<td>3 4 3 1</td>
<td>4 2 0</td>
</tr>
<tr>
<td>Shrinkage Stoping</td>
<td>4 2 1</td>
<td>4 4 3 0</td>
<td>4 2 0</td>
</tr>
<tr>
<td>Cut &amp; Fill</td>
<td>3 2 2</td>
<td>3 3 2 2</td>
<td>4 3 2</td>
</tr>
<tr>
<td>Square Set</td>
<td>3 2 2</td>
<td>3 3 2 2</td>
<td>4 3 2</td>
</tr>
</tbody>
</table>

### Table 4.7. Footwall

<table>
<thead>
<tr>
<th>Mining method</th>
<th>Rock Substance Strength</th>
<th>Fracture Spacing</th>
<th>Fracture Strength</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>W M S</td>
<td>VC C W VW</td>
<td>W M S</td>
</tr>
<tr>
<td>Sublevel Stoping</td>
<td>0 2 4</td>
<td>0 0 2 4</td>
<td>0 1 4</td>
</tr>
<tr>
<td>Sublevel Caving</td>
<td>0 2 4</td>
<td>0 1 3 4</td>
<td>0 2 4</td>
</tr>
<tr>
<td>Shrinkage Stoping</td>
<td>2 3 3</td>
<td>2 3 3 2</td>
<td>2 2 3</td>
</tr>
<tr>
<td>Cut &amp; Fill</td>
<td>4 2 2</td>
<td>4 4 2 2</td>
<td>4 4 2</td>
</tr>
<tr>
<td>Square Set</td>
<td>4 2 2</td>
<td>4 4 2 2</td>
<td>4 4 2</td>
</tr>
</tbody>
</table>

### Table 4.8. Ranking of mining methods based on increasing unit mining cost

<table>
<thead>
<tr>
<th>Mining method</th>
<th>Description of unit mining cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sublevel stoping</td>
<td>Cheapest</td>
</tr>
<tr>
<td>Sublevel caving</td>
<td></td>
</tr>
<tr>
<td>Shrinkage stoping</td>
<td></td>
</tr>
<tr>
<td>Cut-and-fill</td>
<td></td>
</tr>
<tr>
<td>Square-set</td>
<td>Most expensive</td>
</tr>
</tbody>
</table>

### 4.4 Important note on using the numerical system

It is important to mention that the purpose of the numerical method is not to choose the final mining method. It is intended to indicate those methods that will be most effective given the geometry/grade distribution and rock mechanics characteristics, and which will require further economic evaluation. In addition it will also help mining engineers and miners to identify what characteristics are important for the mining methods being considered.

**Example of a mining method selection**

In a feasibility study, the width of a mineral deposit is found to be narrow and estimated to be 8 m. The deposit is tabular in shape and plunges at about 70° to the horizontal. Initial assay values indicate that the grade distribution is uniform. For core samples of the ore,
hangingwall, and footwall at a depth of 1000 m, their respective rock substance strengths can be described as moderate. Fractures in the ore zone are widely spaced whilst those in the hangingwall and footwall are close. Further observations indicate that the joint surfaces in the ore zone are smooth with no in-filling material. In contrast, the in-fill material in the hangingwall and footwall joints is stronger than the rock substance strength.

For the above given information, use the numerical method to identify the two most suitable mining methods for this mineral deposit.

**Suggested solution**

All the mining methods that can be used to extract the given deposit are identified. Based on the orebody geometry and grade distribution information given, Table 4.9 can be deduced from Table 4.4. Considering the rock mechanics aspects of the ore zone, Table 4.10 can be extracted from Table 4.5. Similarly Tables 4.11 and 4.12, which consider the rock mechanics characteristics of the hangingwall and footwall, can be prepared from Tables 4.6 and 4.7 respectively. Table 4.13 shows the total rank values for all the mining methods.

Some useful information can be deduced for Table 4.13.

- On the basis of the orebody geometry and shape considered in this example, sublevel caving and cut-and-fill mining methods might be the least and most suitable methods respectively. According to the numerical method, sublevel caving will be eliminated because the sum of the characteristic values is a negative number.

- The combination of the rock mechanics characteristics of the ore zone, hangingwall, and footwall suggests that the shrinkage stoping method may be the most suitable but not significantly better than the other methods.

- For the combination of all the parameters, the total rank values suggest that the cut-and-fill mining method may be the most suitable.

- The second most suitable mining method may either be shrinkage stoping or square set because they both have the same total rank values. Under such
circumstances the unit mining cost of each method should be considered. Hence from Table 4.8, the shrinkage mining method may be selected over square set.

Based on this analysis initial mine plans of the shrinkage stoping and cut-and-fill mining methods will be drawn to provide a better estimate of the mining and capitalization costs and to determine cut-off grade and mineable reserves.

### Table 4.9. Ranking of mining methods based on orebody geometry and grade distribution

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Description</th>
<th>Sublevel stoping</th>
<th>Sublevel caving</th>
<th>Shrinkage stoping</th>
<th>Cut-and-fill</th>
<th>Square set</th>
</tr>
</thead>
<tbody>
<tr>
<td>General shape</td>
<td>Tabular</td>
<td>2</td>
<td>4</td>
<td>2</td>
<td>4</td>
<td>2</td>
</tr>
<tr>
<td>Ore thickness</td>
<td>Narrow</td>
<td>-49</td>
<td>1</td>
<td>4</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td>Ore plunge</td>
<td>Steep</td>
<td>4</td>
<td>4</td>
<td>4</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Grade distribution</td>
<td>Uniform</td>
<td>3</td>
<td>4</td>
<td>3</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Total of values</td>
<td></td>
<td>10</td>
<td>-37</td>
<td>10</td>
<td>15</td>
<td>12</td>
</tr>
</tbody>
</table>

### Table 4.10. Ranking of mining methods based on rock mechanics characteristics of ore zone

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Description</th>
<th>Sublevel stoping</th>
<th>Sublevel caving</th>
<th>Shrinkage stoping</th>
<th>Cut-and-fill</th>
<th>Square set</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rock substance strength</td>
<td>Moderate</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>Fracture spacing</td>
<td>Weak</td>
<td>1</td>
<td>4</td>
<td>3</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td>Fracture strength</td>
<td>Moderate</td>
<td>2</td>
<td>2</td>
<td>2</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Total of values</td>
<td></td>
<td>6</td>
<td>9</td>
<td>8</td>
<td>7</td>
<td>6</td>
</tr>
</tbody>
</table>
### Table 4.11. Ranking of mining methods based on rock mechanics characteristics of the hangingwall

<table>
<thead>
<tr>
<th>Rock mechanics characteristics of hangingwall</th>
<th>Potential mining methods</th>
</tr>
</thead>
<tbody>
<tr>
<td>Parameter</td>
<td>Description</td>
</tr>
<tr>
<td>Rock substance strength</td>
<td>Moderate</td>
</tr>
<tr>
<td>Fracture spacing</td>
<td>Close</td>
</tr>
<tr>
<td>Fracture strength</td>
<td>Strong</td>
</tr>
<tr>
<td>Total of values</td>
<td>7</td>
</tr>
</tbody>
</table>

### Table 4.12. Ranking of mining methods based on rock mechanics characteristics of the footwall

<table>
<thead>
<tr>
<th>Rock mechanics characteristics of footwall</th>
<th>Potential mining methods</th>
</tr>
</thead>
<tbody>
<tr>
<td>Parameter</td>
<td>Description</td>
</tr>
<tr>
<td>Rock substance strength</td>
<td>Moderate</td>
</tr>
<tr>
<td>Fracture spacing</td>
<td>Close</td>
</tr>
<tr>
<td>Fracture strength</td>
<td>Strong</td>
</tr>
<tr>
<td>Total of values</td>
<td>6</td>
</tr>
</tbody>
</table>

### Table 4.13. Total rank values

<table>
<thead>
<tr>
<th>Mining method</th>
<th>Geometry/grade distribution</th>
<th>Rock mechanics characteristics</th>
<th>Total rank values</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sublevel stoping</td>
<td>10</td>
<td>19</td>
<td>29</td>
</tr>
<tr>
<td>Sublevel caving</td>
<td>-37</td>
<td>22</td>
<td>-15</td>
</tr>
<tr>
<td>Shrinkage stoping</td>
<td>10</td>
<td>23</td>
<td>33</td>
</tr>
<tr>
<td>Cut-and-fill</td>
<td>15</td>
<td>22</td>
<td>37</td>
</tr>
<tr>
<td>Square set</td>
<td>12</td>
<td>21</td>
<td>33</td>
</tr>
</tbody>
</table>
4.5 Equivalent Q-values for mining method selection

The mining method selection described above makes use of three geotechnical measures applied to the ore zone, the hangingwall and the footwall of the orebody. These are:

- rock substance strength
- fracture strength
- fracture spacing.

As much of the work in this project makes use of Barton’s NGI Q rating system, an equivalent range of Q values has been developed in Table 4.14 to describe the characteristics of the parameters mentioned above. It must however, be mentioned that in developing Table 4.14, some assumptions were made.

These assumptions include:

- The joint water reduction, J_w number is set to 1, corresponding to generally dry excavations,
- The joint set number, J_n is set to 4, corresponding to two distinct joint sets
- Rock density is 3000 kg/m^3
- k-ratio = 2 at shallow depths being considered in this project.
<table>
<thead>
<tr>
<th>Fracture spacing</th>
<th>Fracture strength</th>
<th>Rock strength</th>
<th>RQD low</th>
<th>RQD high</th>
<th>Jr/Ja low</th>
<th>Jr/Ja high</th>
<th>SRF low</th>
<th>SRF high</th>
<th>Q range</th>
<th>Q rating</th>
<th>Q rating</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very close</td>
<td>Weak</td>
<td>Weak</td>
<td>10.00</td>
<td>30.00</td>
<td>0.05</td>
<td>0.33</td>
<td>15.00</td>
<td>15.00</td>
<td>0.01</td>
<td>0.17</td>
<td>Extremely poor</td>
</tr>
<tr>
<td>Very close</td>
<td>Weak</td>
<td>Moderate</td>
<td>2.50</td>
<td>7.50</td>
<td>0.05</td>
<td>0.33</td>
<td>15.00</td>
<td>0.70</td>
<td>0.01</td>
<td>3.54</td>
<td>Extremely poor</td>
</tr>
<tr>
<td>Very close</td>
<td>Weak</td>
<td>Strong</td>
<td>2.50</td>
<td>7.50</td>
<td>0.05</td>
<td>0.33</td>
<td>0.70</td>
<td>0.50</td>
<td>0.18</td>
<td>4.95</td>
<td>Very poor</td>
</tr>
<tr>
<td>Very close</td>
<td>Moderate</td>
<td>Weak</td>
<td>2.50</td>
<td>7.50</td>
<td>0.15</td>
<td>1.00</td>
<td>15.00</td>
<td>15.00</td>
<td>0.03</td>
<td>0.50</td>
<td>Extremely poor</td>
</tr>
<tr>
<td>Very close</td>
<td>Moderate</td>
<td>Moderate</td>
<td>2.50</td>
<td>7.50</td>
<td>0.15</td>
<td>1.00</td>
<td>15.00</td>
<td>0.70</td>
<td>0.03</td>
<td>10.71</td>
<td>Extremely poor</td>
</tr>
<tr>
<td>Very close</td>
<td>Strong</td>
<td>Weak</td>
<td>2.50</td>
<td>7.50</td>
<td>1.00</td>
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*Note: RQD stands for Rock Quality Determination, Jr/Ja stands for Joint Rake Angle, SRF stands for Stress Reduction Factor.*
5 Detailed design methodology – stope support

Figure 5.1. Flowchart highlighting the component in the methodology to be discussed in this chapter and its position in the sequence

Empirical rock classification systems such as RMR and Q were developed from databases based on civil engineering tunnels and tended therefore to specify conservative designs for mining excavations. Mathews et al (1981) and later Potvin (1988) and Nickson (1992) developed the stability graph method, which allows the stability of mining excavations to be assessed and determines what support is required. This method allows the classification of the rock mass and a function of the excavation size to be plotted as a y-axis function and an x-axis function respectively. Case studies define zones in these graphs where no support is required, support is required or the excavation is unsupportable.

5.1 The stability graph method for the support of stopes and stope backs

Many researchers have developed empirical methods for classifying the rock mass in order to determine representative mechanical properties for a rock mass. A classification of the rock mass and a quantification of the excavation geometry is achieved by the
The modified stability number, $N'$ is defined as follows:

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

where $Q' = \frac{RQD}{J_n} \times \frac{J_r}{J_a}$

where $RQD$ is calculated from drill core as the sum of core lengths greater than 10 cm divided by the total length of core run (cm)

$J_n$ is a measure of the number of joint sets

$J_r$ is the joint roughness number

$J_a$ is the joint alteration number

$\frac{RQD}{J_a}$ is a measure of block size for a jointed rock mass

$\frac{J_r}{J_a}$ is a measure of joint surface strength and stiffness

$A$ is a measure of the ratio of intact rock strength to induced stress

$B$ is a measure of the joint orientation to the excavation face

$C$ is a measure of the influence of gravity on the face or backs being considered.

Values of $N'$ make up the y axis of the stability graph, the x axis is determined by values of the hydraulic radius, $HR$, which is a function of the size and shape of an excavation. The $HR$ is defined as the area divided by the perimeter of the excavation.

$$HR = \frac{w \times h}{2w + 2h}$$

The hydraulic radius needs to be treated with caution as different shapes can have the same hydraulic radius. Examples would be where $w = 20$ m and $h = 20$ m, the $HR = 5$, similarly $w = 15$ m and $h = 30$ m the $HR = 5$. 
The methodology followed in the design of stope support and stope backs is shown in the form of a flow diagram in Figure 5.2.

Figure 5.2. Support design methodology for the support of stopes

Case histories of 176 unsupported open stopes were collated by Potvin (1988) and 13 by Nickson (1992) and these are plotted on the stability graph shown in Figure 5.3 below.
These case histories had no support installed and were rated as stable or caved with a transition zone defining a zone between stable and caved.

![Stability graph, no support](image)

**Figure 5.3. Stability graph, no support**

A further 66 case histories were collated by Potvin (1988) and Potvin and Milne (1992) and in this case cablebolting had been used as support in the stopes. Nickson (1992) added a further 46 case studies to this database.
Nickson imposed a transition zone where the upper curve is the maximum stable hydraulic radius for cablebolted stopes reducing in confidence to the lower curve where cables no longer provide any degree of support. This is shown in Figure 5.4.

If the two Figures, Figures 5.3 and 5.4, are superimposed then a cablebolt design zone can be defined as a zone where cable bolts are needed and can be effective. This is shown in Figure 5.5.
Figure 5.5. Stability graph showing cablebolt design zone for open stopes

Clearly within the cablebolt design zone if the hydraulic radius increases or the modified stability number decreases then the risk of failure increases.

5.1.1 The support of open stopes and stope backs

5.1.1.1 Empirical support design

Potvin (1988) proposed empirical guidelines for the determination of cablebolt length, he plotted the cablebolt length against the hydraulic radius for each of his case studies and this is shown in Figure 5.6 below. A representative line based on current practice is shown and corresponds to:

Length = 1.5 x HR up to a practical maximum of 15 m and a HR of 10.

This empirical design curve can therefore be used to determine cablebolt lengths for various values of hydraulic radius.
Nickson (1992) correlated cablebolt densities with the parameter $N/HR$ for numerous case studies. Hutchinson and Diederichs (1996) incorporated a conservative and non-conservative zone based on the plotted data and this is shown in Figure 5.7. The conservative zone would be applicable to cablebolt design for stope backs above drilling horizons and other areas where entry is permitted.
Figure 5.7. Determination of cablebolt spacing and density

The flow chart in Figure 5.8 summarises the process described above for the support of stopes and stope backs.
5.1.1.2 Semi-empirical support design

Hutchinson and Diederichs (1996) have shown that it is possible to combine the empirical methods described above with a mechanistic approach, which they refer to as a semi-empirical cable bolt design approach. The cablebolt design zone, defined earlier in Figure 5.5, has been divided into three sub zones as in Figure 5.9, namely the reinforcement zone, the support zone and the retainment zone. The other two zones remain: where no support is required and where the rock is unsupportable.
Hutchinson and Diederichs (1996) have developed cablebolt design charts for single (15.2 mm diameter) and double strand cables in the cablebolt design zone shown in Figure 5.9. Note that the recommended cablebolt spacing is for an equivalent square pattern. These two design charts are shown below as Figure 5.10 and Figure 5.11.
Figure 5.10. Cablebolt design chart for single strand cablebolts
Hutchinson and Diederichs also addressed the issue of the minimum cablebolt length in the cablebolt design zone. Based on parametric analysis and using conservative values of $N'$ and assuming that the cablebolt is required to extend 2 m beyond the stabilized beam or failed zone, the design chart for cablebolt length is shown below as Figure 5.12.
Figure 5.12. Cablebolt design chart for minimum cablebolt length

The flow chart shown in Figure 5.13 is a summary of the semi-empirical approach for the design of support for stopes and stope backs which was described above.
5.2 The use packs, poles and mechanical props

Most of the foregoing analysis has been concentrated on the use of cables. Few of the South African operations visited and considered in this project make much use of cables for systematic ground support. Internationally most mines similar to those under consideration here make a great deal of the use of systematic bolting – which has been the focus in this project – to identify world’s best practice.

By contrast, South African operations make far more use of sticks, poles, packs and mechanical props, perhaps because most operations are in good ground. There were,
however, several instances where the use of cables was indicated and would considerably improve safety, conditions and ground control.

The use of poles, packs and mechanical props which is quite common in South African mines deserves some comment. These are certainly effective and do have a place in designs, however, these do not replace the need for cable support if, indeed this is required. In Figure 5.9, such support is effective largely in the retention zone.

5.3 A worked example of the stability graph method for the support of stopes and stope backs

Consider an ore body dipping at 45° and is 6 m wide. A cut and fill mining method is being employed with the competent ore in the immediate hangingwall and not requiring any support. However it is clear that the sidewall rock dipping over the stope is less competent and could well require support. The strike distance of the stope is 50 m, the height of the sidewall rock is 6 m after cleaning and the mining depth is 700 m.

![Figure 5.14 Section through stope and ore body.](image)
In order to determine the support requirements of the hangingwall rock, the stability graph method will be used. First the modified stability number $N'^1$ is required to be determined from measured data shown in bold.

The modified stability number, $N'^1$ is defined as follows:

$$N'^1 = Q'^1 \times A \times B \times C$$

where $Q'^1 = \frac{RQD}{Jn} \times \frac{Jr}{Ja}$

where $RQD$ is calculated from drill core as

$$\frac{\text{sum of core lengths greater than } 10 \text{ cm}}{\text{total length of core runway (cm)}} = 80$$

$J_n$ is a measure of the number of joint sets = 4

$J_r$ is the joint roughness number = 2

$J_a$ is the joint alteration number = 4

therefore $Q'^1 = \frac{RQD}{Jn} \times \frac{Jr}{Ja} = \frac{80}{4} \times \frac{2}{4} = 10$

$A$ is the rock stress factor and is a measure of the ratio of intact rock strength to induced stress. The uniaxial compressive strength was found to be low at 45 MPa. Numerical modelling indicated that the maximum induced compressive stress in the sidewall would be 70 MPa. The ratio of

$$\frac{\text{uniaxial compressive strength}}{\text{maximum induced compressive stress}} = \frac{40}{70} = 0.57$$

The rock stress factor, $A$, (Potvin 1988) is then determined graphically and is 0.1.

$B$ is the joint orientation factor and is a measure of the joint orientation to the excavation face. In this case the joint orientation is at an angle of 30° with the sidewall of the excavation as shown in Figure 2 below.
Figure 5.15. A section showing the joints and sidewall of the stope.

The angle between the face and the joint is $\alpha$ which is $30^\circ$. From Potvin (1988) $B = 0.2$

$C$ is the gravity adjustment factor and is a measure of the influence of gravity on the face or backs being considered. In the case being considered the most likely mode of structural failure will be gravity falls. From Potvin (1988) the gravity adjustment factor for a face dip of $45^\circ$ would be $4$.

$A$ as $N' = Q' \times A \times B \times C$

Then $N' = 10 \times 0.1 \times 0.2 \times 4 = 0.8$

Values of $N'$ make up the y axis of the stability graph, the x axis is determined by values of the hydraulic radius, $HR$, which is a function of the size and shape of an excavation. The $HR$ is defined as the area divided by the perimeter of the excavation.

$$HR = \frac{w \times h}{2w + 2h}$$

The hydraulic radius needs to be treated with caution as different shapes can have the same hydraulic radius.

In the case under consideration the $HR = \frac{50 \text{ m} \times 6 \text{ m}}{2 \times 50 \text{ m} + 2 \times 6 \text{ m}} = 2.67$
Figure 5.16. Design zones for stopes. Hutchinson and Diederichs (1996)

The results of the analysis above plots at A in the stability graph shown in Figure 3.

A number of methodologies are available to determine cable bolt length and spacing, however in this worked example the design chart for single strand cablebolts support, derived by Hutchinson and Diederichs, will be used to determine cablebolt spacing. This is shown in Figure 4 where A is once again plotted and the cablebolt spacing is required to be 2.0 m.
Figure 5.17. Cablebolt design chart for single strand cablebolts. Hutchinson and Diederichs (1996)

The required cablebolt length has been determined by Potvin using data where:

\[ \text{Length} = 1.5 \times HR \] up to a practical maximum of 15 m at a hydraulic radius of up to

\[ HR = 100. \] In this case the HR was determined to be 2.67 and therefore the cablebolt length would be required to be 4.0 m.
6 Detailed design methodology – crown/sill pillars

Figure 6.1. Flowchart highlighting a methodology to design crown/sill pillars as the focus of this chapter in the research methodology

6.1 Background

Most of the mining methods used for steeply dipping, narrow, tabular deposits make use of crown/sill pillars as an integral part of the process. These pillars serve to compartmentalise the different levels of the orebody. In so doing they also provide much needed regional stability and separate different working levels. While site visits to such mines in South Africa have shown few direct current problems with such pillars, this may be somewhat misleading. A number of pillars viewed show a lack of systematic, scientific planning posing potential future problems. In addition, mining of most of these deposits is progressing deeper which will necessitate a more rigorous approach to pillar design.
In considering crown/sill pillar design, the lack of systematic design is not entirely surprising as little definitive information is available for such designs (see literature review in Section 3). Conventional pillar designs could be applied, but would yield excessively conservative designs. In this section of this work, design principles are established and a systematic design methodology described. It should be noted that these design principles have yet to be validated and proven through detailed back analysis and in situ monitoring which falls beyond the scope of this project.

### 6.2 Design principles

It is the view of the authors that in the light of this investigation the following principles should be followed:

- Pillar stability is primarily influenced by the strength characteristics of the ore zone itself.
- The strength of the immediate hangingwall and footwall material is only significant if it is significantly weaker than that of the ore zone.
- Pillar stability is significantly influenced by the orebody geometry (3D shape).
- Pillar stability can be compromised by the ore zone sidewall contact material (often weak shear material).
- It is generally accepted (Brady and Brown, 1993) that stresses in the crown/sill pillar will exceed the strength of the ore zone material as mining progresses, giving rise to stress fracturing.
- As mining progresses, support requirements increase.
- The usual design “rule of thumb” is that the pillar height should be twice the orebody width.
- More generally, the height of the pillar is governed by the orebody (and surround) strength and prevailing stress conditions (which change as mining advances).
6.3 Proposed design methodology

1. Establish reasonable average intact rock strength values for the ore zone, $\sigma_{ore}$; hangingwall, $\sigma_{hw}$; and footwall, $\sigma_{fw}$.

2. Assume reasonable values for in situ vertical stress, $\sigma_v$ and k-ratio, k.

3. If the ore strength is significantly greater than that of the surrounding rock ($\sigma_{ore} >> \sigma_{hw}$ or $\sigma_{fw}$), or if the 3D geometry is unfavourable, or if the orebody/rock mass contact is weak then there is the potential for damage to the orebody surround (punching). In such cases the methodology set out below is also followed, however, the surrounding rock mass and the pillar require support.

4. Compute $N' = Q' \times A \times B \times C$ for the ore zone,

   where $A = (\sigma_{ore}/\sigma_{max})$

   $\sigma_{max}$ is the maximum stress in the crown/sill pillar.

5. Compute the hydraulic radius, $HR = (2 \times w + 2 \times h)/w \times h$. 


6. Depending on the mining method being used, the crown pillar will need to be supported to an appropriate level of confidence. eg. high confidence when stope entry is required (cut and fill or shrinkage); some confidence for open stoping or non-entry methods.

7. Using Figure 6.2 and selecting the appropriate support confidence as set out above, compute \( \sigma_{\text{max}} = (Q' \times B \times C)/(N' \times \sigma_{\text{ore}}) \)

8. Assuming a general stope geometry as shown in Figure 6.3, only the pillar height, \( d \), is unknown.
9. 3D numerical modelling (MAP3D) can be used in pillar design in assisting to obtain the pillar height, as illustrated in Figure 6.4, for a regular geometry.

*Figure 6.3. Schematic illustration of stope and pillars*
The MAP3D model of the crown pillar assumes a k-ratio of 2, a density of 3000 kg/m$^3$ and is run at a depth of 500 m. The orebody is assumed vertical. Pillar heights considered are 40 m, 20 m, 16 m, 12 m, 10 m, 8 m, 6 m & 4 m. Strike extents are increased from 32 m to 120 m. Orebody thicknesses considered are 2 m, 4 m, 6 m and 8 m. Results from these runs are shown in Figures 6.5 to 6.8 corresponding to different orebody thicknesses.
Figure 6.5. MAP3D model results: Maximum skin stress at centre of strike span and pillar for the 2 m orebody

Figure 6.6. MAP3D model results: Maximum skin stress at centre of strike span and pillar for the 4 m orebody
11. Given the orebody geometry and stress computed in step 7 above, the curve matching these values corresponds to the pillar height that can be supported using the previously defined methodology. It should also be noted here that specific
conditions have been modelled, however there are innumerable combinations and modelling of these falls beyond the scope of this project.

12. If the geometry is significantly different from the above, then problem specific 3D modelling needs to be used to obtain the pillar height corresponding to the computed $\sigma_{\text{max}}$ value and the same methodology is followed.

13. The support methodology in Section 5 is followed to define the support required.

14. If modelling is unavailable, and plane strain conditions can be applied to the geometry, then the peak stress for a pillar having a geometry as illustrated in Figure 6.9 can be computed using the Brady –Bray equations (Brady & Brown, 1993). Firstly, the rectangular stopes can be approximated as two ellipses (Figure 6.9(a) Brady and Brown). The stress concentration in the crown as a result of these two ellipses can be computed by adding the influence of the two excavations at this point separately – see Figure 6.9(b) & (d). The same equations can be used to compute the stress in inclined geometries and under unusual 2D stress states - see Figure 6.9(c). These equations, however are very complex, so not repeated in this text. Readers are referred to Brady and Brown, 19 for the full equation. Once again it is suggested that this design be done, or at least reviewed by a rock engineering professional.
15. If modelling is unavailable, and plane strain conditions can be applied to the geometry, then the average stress for a pillar having a geometry as illustrated in Figure 6.10 can be computed (after Salamon, 1964).
Figure 6.10. Geometry amenable to plane strain approximation

The equations are rather complex, so a graphical format has been produced (see Figure 6.11).

The horizontal axis is $e = (T - d)/T$
where $T$ is level spacing and $d$ is the pillar height. No account is taken of the orebody thickness.

Taking the stress computed in step 7, and dividing by $k \sigma_v$ allows the $e$ value to be read off the graph below, subject to the limitations inherent.
Figure 6.11. The graph indicates the average stress in the pillar for different geometries

The vertical axis is average pillar stress/kσ, so making several assumptions, it is possible to compute a pillar height, \( d = T - T_e \). It should be noted, however, that this approach makes many assumptions making this of limited practical use. In any event, the 3D modelling approach is preferred and the use of professional rock engineering input is strongly indicated.

16. Again, the support methodology in section 5 is the followed to define the support required.
7 Assessment of local instability

Figure 7.1. Flowchart highlighting the evaluation of local instability as the focus of this chapter in the research methodology

7.1 Introduction

Initial design of support systems is based on information collected from drill cores. How representative this information is depends on factors such as the distance between adjacent boreholes and the accuracy of drilling. As mining progresses, geological discontinuities, which might not have been detected from drill cores, could be exposed. In view of the fact that information about these discontinuities was not incorporated into the initial support design, instability problems, either on a small or large-scale, may be encountered.

Large-scale instabilities might indicate that mining has progressed into a different geotechnical area, requiring re-evaluation of the adopted support system. In contrast, if the degree of instability is localised, wedge analysis might be essential. In this section of the report, a technique that can assist in identifying potential wedges and their mode of failure is discussed. Theoretical concepts used to determine the force required to stabilize
these wedges are presented and the complexity involved in using these formulae is highlighted.

### 7.2 Identification of local instability

In a preceding chapter a methodology to design appropriate back support was established. It is assumed at this stage that the methodology has been adopted and used to design a support system in a stope. At the time of designing the support system, the information used as input might have been collected from specific geotechnical environments and hence reflects the local geology.

As mining progresses, additional geological structures, which were inconspicuous during the initial stages of mining, might be exposed. If no information about these structures was available during the support design stage, the adopted support strategies might not work effectively in this environment. Consequently, local instabilities might occur, which if not stabilized could pose a threat to miners. The degree of instability dictates the extent to which the support being used must be re-evaluated.

It is therefore essential that as mining progresses, a routine evaluation is made of support performance in relation to new geotechnical conditions. In areas where the adopted support strategies are found to be inadequate and general support performance is undesirable, there is the need to identify the driving factors.

In order to facilitate the identification of the factors that might be causing the local instabilities, routine inspections should be undertaken. Geological structures daylighting into the stope should be noted and the extent of damage to the host rock caused by the effects of blasting should also be assessed. To be able to evaluate the effect of these discontinuities on the stability of the stope, the characteristics of these discontinuities would have to be quantified by a discontinuity survey.

### 7.3 Geotechnical mapping

If the factors causing the local instability are identified, their characteristics could be quantified to provide information for further analysis. These characteristics might include the orientation of the discontinuities and the nature of their surface. The number of joint
sets, persistence and spacing should also be determined. This information could provide a crude means of estimating the potential block sizes that could be isolated. The type of infill material, if any, should also be noted because it is an important parameter in assessing the failure potential of an isolated block. Other important parameters might include the strike and dip of the orebody.

In practice, information about most of the parameters mentioned in the preceding paragraph is collected on a routine basis by the geology department and is therefore easily accessed. Data gathering procedures are well understood by geologists. It is therefore not the intention to describe these procedures in this document. However, it is important to mention that for stability analysis, the purposes of the data collection must be clearly defined. The location of instabilities either in the stope back or sidewalls must also be noted.

### 7.4 Data analysis – stereographic projection

Discontinuity data can be analysed using stereographic projections. Basically two types of projections namely equal-angle and equal-area can be used and these projections can be done either on the lower or upper hemisphere. However, for the analysis of joint orientation data, the lower hemisphere equal-area projection technique is widely used. For a detailed description of these projection techniques, the interested reader is referred to Hoek and Brown (1980).

After plotting the data, the resulting scattered data points on the stereonet should be contoured and the mean orientation of discrete data sets determined. The major planes representing the mean orientations should be constructed on the stereonet to assess the potential modes of failure. Computer programs such as DIPS, if available could be used for this exercise.

### 7.4.1 Potential modes of failure

Identifying the potential mode of failure of a blocky rock mass is important because it enables some useful information to be deduced. Firstly, the geometric shape of the block can be inferred and its volume can therefore be estimated. Secondly the failure mode can
be used to determine the orientation of reinforcement elements that can be adopted to maintain the integrity of the surrounding rock mass.

In determining the mode of failure, the direction of gravity needs to be indicated on the stereonet. This can be visualized as a vertical line passing through the centre of the stereonet and can be represented with a plus sign. The major planes representing the mean orientation of discrete data sets should be plotted.

For illustration purposes, three major planes, supposedly representing the mean orientations of three joint sets, have been plotted in Figure 7.2. Assuming the centre of the net is labelled O, the three lines of intersection of three joints can be referred to as OA, OB, and OC. It can be observed that the plus sign, which indicates the direction of gravity, lies within the closed loop formed by the intersection of the three joint planes. Thus the structural condition illustrated in Figure 7.2 results in the generation of a tetrahedral wedge, which is kinematically free to fall vertically into the underlying excavation.

Figure 7.2. Stereographic projection of three joint sets forming a tetrahedral wedge, which represents a collapse mode of free vertical displacement
Figure 7.3 illustrates a second case where the direction of gravity lies outside the wedge trihedral angle. This condition precludes free vertical displacement of the wedge under its own weight. The possible displacement of the wedge involves slip on the plane of weakness I.

**Figure 7.3. Stereographic projection of three joint sets forming a tetrahedral wedge, which represents a collapse mode of slip on base plane**

**Example**

For demonstration purposes, consider Figure 7.4. Figure 7.4a shows a two-dimensional illustration of an excavation cut in a jointed rock mass. Two joint sets have been indicated and they intersect to form potential blocks both in the crown and sidewall of the excavation.
Figure 7.4. Cross section of an excavation in a rock mass containing two joint sets and the corresponding stereographic projection

By means of geotechnical mapping, the dip and dip directions of the joints illustrated in Figure 7.4a can be measured and their surface characteristics, persistence, and infilling material can also be noted. The strike of the excavation can also be measured.

Assuming joint set I dips at about 45 degrees towards north-west, joint set II is inclined at about 45 degrees towards south-west, and the long axis of the excavation strikes in the north-east-south-west direction, then this information can be plotted as shown in Figure 7.4b.

It can be observed in Figure 7.4b that the joints’ intersections with the roof strike parallel to the excavation axis. It can also be noted that the line of intersection of the joints is parallel to the equatorial plane, which represents the roof of the excavation. The joints intersect in the roof to form a triangular prism. For the condition illustrated in Figure 7.4b, depending on the surface characteristics of the joints, the triangular prism may experience free vertical displacement under gravity load, or slip on the joint surface with the shallow dip.
Figure 7.5a illustrates a condition where an isolated block might slide down into the excavation from the sidewall. Three joint sets have been indicated. It has been assumed that the excavation is long and strikes in the north-east-south-west direction. The lower hemisphere stereographic projection has been used to plot the orientation of the joints in Figure 7.5b. The three joints intersect to form a tetrahedral wedge. If AB represents the southeastern sidewall of the excavation, the joint geometry allows slip of the tetrahedral wedge, on the base plane I, into the excavation. If AB represents the northwestern wall of the excavation, the feasible mode of wall collapse involves block slip in a direction parallel to the lines of intersection of planes II and III.

\[ \text{Figure 7.5. Cross section of an excavation in a rock mass containing three joint sets and the corresponding stereographic projection} \]

### 7.5 Implementation of appropriate support strategies

The implementation of appropriate support strategies will depend on the degree of instability. If underground observations reveal that mining has progressed into a different geotechnical area, which is characterised by different rock mass behaviour, then the entire support system may have to be redesigned. Factors that can cause significant
differences in rock mass behaviour may include increased intensity of jointing, unfavourable orientation of major faults, weak bedding plane contacts, etc. Under these conditions, the incidence of rock falls may increase and the support system may have to be redesigned by the methodology defined in Section 5 of this report.

However, if the instability only occurs locally, then the potentially unstable block will have to be stabilized by some form of reinforcement. Adopting an effective reinforcement strategy requires thorough understanding of the mechanics involved.

Brady and Brown (1993) present an extensive analysis of the stability of wedges both in the crown and sidewall of underground stopes. In their analysis they assumed that fissure water was absent and that the block surface forces could be determined by some independent analytical procedure. It was further assumed that the block mass could be determined from the joint orientations and the excavation geometry.

On the basis of these assumptions, they explained a methodology that can be used to design support systems to prevent falling of symmetrical and asymmetrical blocks from the stope crown. In addition to these they derived another method that can be used to design a support system to prevent sliding of a triangular wedge.

The shear and normal forces acting on a symmetrical block are shown in Figure 7.6a. The weight of the block is denoted by w and it has been assumed that there is a reactive force R provided by an initial reinforcement. The difference between these two forces is denoted by P, which is assumed to be a limiting force required to stabilize the block. A complete force diagram is illustrated by Figure 7.6b to account for the effect of an internal horizontal force $H_o$. N and S denote the normal and shear forces acting on the inclined surfaces, AB and AC, of the prism respectively. The acute angle formed by the intersection of the two inclined planes is defined as $2\alpha$ and their surfaces have been assumed to have a friction angle of $\phi$. 
For a symmetrical block as shown in Figure 7.6, Brady and Brown explained that if the joint normal stiffness is much greater than the joint shear stiffness, then the magnitude of the limiting force required to establish equilibrium is given by:

\[ P_1 = \frac{2H_o \sin \alpha \sin (\phi - \alpha)}{\sin \phi} \]  

(6.1)

By considering the relation:

\[ P = W - R \]  

(6.2)

The factor of safety can be estimated as:

\[ \text{Factor of Safety} = \frac{P_1}{P} \]  

(6.3)

Hence for a given factor of safety, a support system, which will be able to provide the required reinforcement, can be determined. It must however be mentioned that
computing $P$ from equation 6.1 is much more complicated and requires a very good understanding of stress analysis. For this reason it is suggested that experts in this area of work be consulted to provide the needed assistance.

An asymmetric block is illustrated in Figure 7.7. It has been assumed that the frictional properties and the surface forces acting on each of the inclined planes are different.

![Diagram of an asymmetric block](image)

**Figure 7.7. An illustration of the forces acting on an asymmetric triangular roof prism**

For the asymmetric block shown in Figure 7.7, Brady and Brown established that the magnitude of the limiting force could be estimated by:

$$P = \frac{H_o \sin \alpha \sin (\phi_1 - \alpha)}{\sin \phi_1} + \frac{H_o \sin \alpha \sin (\phi_2 - \alpha)}{\sin \phi_2}$$

(6.4)

Similarly, equation 6.2 and 6.3 can be used to estimate the factor of safety. The comments made earlier about the complexity of using equation 6.1 also apply to equation 6.4.
Brady and Brown extended their analysis to include a method for design of a rockbolt or cable system to prevent sliding of a triangular block. For the geometry shown in Figure 7.8, they suggest that the total force require $T$ to maintain a given factor of safety can be estimated as:

$$T = \frac{W(F \sin \varphi - \cos \varphi \tan \phi) - cA}{\cos \theta \tan \phi + F \sin \theta}$$  \hspace{1cm} (6.5)

Where $W$ = weight of the block, $A$ = area of the sliding surface, $T$ = total force on the bolts or cables, $\Psi$ = dip of the sliding surface, $\theta$ = angle between the plunge of the bolt or cable and the normal to the sliding surface, $c$, $\phi$ = cohesion and angle of friction on the sliding surface.

Figure 7.8. Design of a reinforcement system to prevent sliding of a triangular block
7.6 Summary

As mining of a vein deposit progresses, two instability problems may be encountered. In a case where mining enters into a new geotechnical area, large-scale falls of ground may be encountered. This might require re-evaluation of the entire support system. A suggested methodology has been given in Section 5 of this report.

However, if only local instabilities occur, wedge analysis to identify potential modes of failure might be essential. If the mode of failure is identified, the mass of the block should be determined and the limiting force required to ensure equilibrium should be achieved. Theoretical formulae to assist in this regard are given in the text. A methodology to simplify the procedure is given in Figure 7.9.

It has been mentioned in the text that estimating the limiting force is much more complicated than has been indicated in this report. It has therefore been suggested that whenever the need arises an expert should be consulted to assist.

**Figure 7.9. Methodology to determine the force required to reinforce an unstable wedge**
8 Conclusions and recommendations

It became apparent, in carrying out this work, that there are relatively few current mining operations that fall within the brief of this project, namely: shallow depth, steeply dipping vein deposits. It further came to our attention that many of these operations have a good rock related safety record. However, during the course of our investigations a number of problems or potential problems of relevance were identified.

The primary finding of visits to mines and survey work are the following hazards/potential hazards:

- fallout of large blocks bounded by geological structures,
- poor ground conditions and weak/weathered material,
- haulage location with respect to the orebody,
- cross-cut support at the orebody contact,
- use of inappropriate mining methods,
- stress “bridging” or arching - potential regional failure,
- use of poor, inadequate or no support,
- poor understanding of regional stress distribution,
- crown/sill pillar failure,
- poor access control into working stopes,
- failure to adapt mine layout as spans, depth or conditions change.

The problems above are likely to be exacerbated as the mines mature and as the lateral and vertical extent of mining increases.

Several aspects covered in detail in this project (e.g. changing mining method, implementing systematic support design or pillar optimisation) may be difficult for current operations to implement. It is believed that these will be of benefit in a number of ways in the future:

- As mining depth increases the need will arise for designs and support to be systematic and scientifically based to be effective.
- Many current mines are either exploring extensions to existing orebodies or similar geological structures.
- It is believed that many smaller mines are likely to come into operation on orebodies having such characteristics, and that these too will have limited technical rock engineering input available. The guidelines for mining method selection, support and pillar design discussed here and provided in the handbook will be of assistance to
these mines and can be used by the DME to assist in technical evaluations of new mining licence applications.

A detailed literature review has been completed in which world best practice has been documented in detail. Through the use of international reviewers (Dr's Lilly and Barton) these “best practice” approaches have been modified and adapted to be more suited to local application.

Although marginal to the brief of this project, the choice of appropriate mining method is seen as a fundamental issue which impacts on the behaviour of the rock mass. There are several instances where the use of an inappropriate mining method compromised safety and caused practical problems and impacted on economics through high dilution. A systematic and quantified method of selecting the most appropriate mining methods is set out in this project.

Two systematic methodologies have been set out for the design of support. The separate methods apply to different ground conditions and orebody characteristics. Importantly, it should be noted that large structurally controlled wedges are not catered for by the methodologies. The support design only considers general ground condition. An approach to identifying such unstable wedges and addressing the problem has been put forward.

No effective method for designing crown/sill pillars was found in the literature and was confirmed by Lilly and Barton.

Guidelines that follow a stepwise approach to designing a crown pillar of appropriate dimensions have been developed as part of this project. The support requirements for this pillar use the same methodology as for back and hangingwall/footwall support. It needs to be stressed that the designs are complex, generally necessitating the use of 3D numerical modelling. In all but the simplest cases, rock engineering technical expertise needs to be consulted. It should also be noted that this approach has yet to be proven by detailed back analysis and in situ monitoring.

The k ratio and state of in situ stress have been assumed in much of this work. Very little is known about such values around these mines and this remains an area that needs some attention in the longer term.
It should be noted that the method is founded on theoretical considerations, which include a number of simplifying assumptions. It is strongly recommended that a validation process involving back analyses of existing pillars be carried out. Also, only people with sound understanding of rock mechanics and numerical modelling should apply this method. The proper design and support of such pillars is not a trivial issue.

Given that many of these mines are small and have no direct technical rock engineering staff, key findings have been set out in an easy to use handbook.
9 References


APPENDIX A. List of mines considered and contacted using telephonic questionnaire

Shaded items indicate mines that meet the criteria set for this project.

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APPENDIX B. Questionnaire used to identify mines for this project

**SIMOT QUESTIONS**

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<th>Mineral</th>
<th>Tel</th>
<th>Person</th>
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1. Operation surface or underground?  
   - Surface  
   - Underground

2. Deposit tabular/vein or massive?  
   - Massive  
   - Tabular/vein  
   Thickness? m

3. Typical dip?  
   - ° < 35°  
   - ° > 35°

4. Single or multi-reef/vein?

5. Depth range of current mining?  
   - Minimum m  
   - Maximum m

6. Approximate annual production?

7. Mining method(s) used?

8. Other??