PRO FORMA CODE OF PRACTICE TO

COMBAT ROCKFALL ACCIDENTS IN

STEEP VEIN MINES

INTRODUCTION

This document has been prepared as part of a Safety in Mines Research Advisory Committee (SIMRAC) research project OTH 602 entitled “Best practice rock engineering handbook for other mines including underground and open pit mines and quarries”.

The objective of this document is to assist employers at steep vein mines in preparing a code of practice to combat rockfall and rockburst accidents (COP) in accordance with the requirements of the Mine Health and Safety Act (MHSA), 1996 (Act No. 29 of 1996), and in accordance with the Department of Minerals and Energy (DME) Guideline Reference No. 7/4/118-AB1 (Tabular Metalliferous Mines) issued by the Chief Inspector of Mines on 16 October 1996. At the time of compiling this pro forma COP, the revised Guideline for tabular metalliferous mines had not been completed yet. It is recommended that the new Guideline be consulted when it becomes available.

DEFINITION OF STEEP VEIN MINING

For the purpose of this COP, steep vein mining is defined as underground mining operations in which vein type orebodies are extracted which dip at greater than 55 degrees and do not exceed 3m in width. Orebodies, which may typically fall into this category, are fissure diamond mines and vein gold mines.

STRUCTURE OF DOCUMENT

This document presents the requirements of a COP as defined in the Guideline as boxed text.

REQUIREMENTS AS BOXED TEXT

The requirements are followed by an example of how the actual COP document may be written to satisfy the requirements. Additional notes on particular issues that require attention are presented in *italics.*

FORMAT AND CONTENT OF COP

TITLE PAGE	
• *Name of mine*	*Y*
• *Heading: Mandatory COP to Combat Rockfall and Rockburst Accidents in Steep Vein Mines*	*Y* *Y*
• *Statement: The COP was drawn up in accordance with DME Guideline, Reference No. 7/4/118 AB1 issued by Chief Inspector of Mines on 16 October 1996.*	
• *Mine's reference number*	*Y*
• *Effective date*	*Y*
• *Revision date*	*Y*

ABC DIAMOND MINE (PTY) LTD

MANDATORY CODE OF PRACTICE TO COMBAT ROCKFALL AND ROCKBURST ACCIDENTS IN STEEP VEIN MINES

This code of practice (COP) was drawn up in accordance with DME Guideline, Reference No. 7/4/118 AB1 issued by Chief Inspector of Mines on 16 October 1996.

REF. No.	:	ABC COP1/2001
EFFECTIVE DATE	:	FEBRUARY 2001
REVISION DATE	:	FEBRUARY 2002

CONTENTS OF CODE OF PRACTICE	
Table of contents	*Y*

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STATUS OF MANDATORY COP	
State that:	
• *The COP has been drawn up in accordance with the relevant guideline issued by the Chief Inspector of Mines.*	***Y***
• *This is a mandatory COP in terms of Section 9(2) of the Mine Health and Safety Act, 1996 (Act 29 of 1996).*	***Y***
• *This COP may be used in accident investigation/inquiry to ascertain compliance and also to establish whether the COP is effective and fit for purpose.*	***Y***
• *This COP supersedes all previous COP's in this regard.*	***Y***
• *All managerial instructions or recommended procedures and standards on the relevant topics must comply with the COP and must be reviewed to ensure compliance.*	***Y***

1 STATUS OF MANDATORY CODE OF PRACTICE (COP)

- The COP has been drawn up in accordance with DME Guideline, Reference No. 7/4/118 AB1 issued by Chief Inspector of Mines on 16 October 1996.
- This is a mandatory COP in terms of Section 9(2) of the Mine Health and Safety Act, 1996 (Act 29 of 1996).
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- This COP supersedes all previous COP's in this regard.
- All managerial instructions or recommended procedures and standards on the relevant topics must comply with the COP and must be reviewed to ensure compliance.

MEMBERS OF DRAFTING COMMITTEE	
• *Full names*	*Y*
• *Designation*	*Y*
• *Professional qualifications, experience and affiliation*	*Y*
• *Must include a competent RE practitioner*	*Y*

2 MEMBERS OF DRAFTING COMMITTEE

The Manager of ABCt Mine, after consultation with the Health and Safety Committee (H&SC), appointed a committee for the drafting of this COP to combat rockfall accidents. Combating rockbursts does not form part of this COP, since the likelihood of rockbursts occurring at ABC Mine is minimal.

The full names, designation, professional qualifications and/or experience and affiliation of the COP Drafting Committee members are:

Mr BLA Stall ***(Production Manager)***

- National Higher Diploma - Metalliferous Mining (Technikon Witwatersrand)
- National Diploma - Metalliferous Mining (Technikon Witwatersrand)
- Mine Managers Certificate of Competency
- Mine Overseers Certificate of Competency
- Associate Member of Mine Managers Association.
- Thirteen years mining experience in various positions in the production environment.

Mr CLE Wirr ***(Rock Engineering Practitioner)***

- National Diploma - Metalliferous Mining (Technikon Witwatersrand)
- Chamber of Mines Certificate in Rock Mechanics
- Member of the South African National Institute for Rock Engineering
- Fifteen years mining rock engineering experience on a tabular mine.

Mr STOF Engate ***(Mine Overseer)***

- Mine Overseers Certificate of Competency
- Experience in tabular mining:
  - 5 Years Shift Boss Mine A
  - 4 Years Shift Boss Mine B
  - 3 Years Mine Overseer Mine C

Mr PA Soppa ***(Safety Officer)***

- N.O.S.A S.A.M.T.R.E.C.
- Experience in tabular mining:
  - 5 Years Stoper Mine A
  - 3 Years Shift Boss Mine B
  - 2 Years Safety Officer Mine C

Mr SHI Bas ***(Shift Supervisor)***

- Experience in tabular mining:
  - 2 Years Onsetter
  - 20 Years Developer Mine A
  - 5 Years Shift Boss Mine B
  - 2 Years Safety Officer Mine C
  - 2 Years Shift Boss Mine D

Mr Wilnet Boor ***(Rockdrill Operator)***

- Experience in tabular mining:
  - 2 Years Stope Timber Man Mine A
  - 6 Years Rockdrill Operator Mine B

GENERAL INFORMATION	
Include locality map, indicating: • *location relative to towns* • *existing infrastructure* • *other relevant features, e.g. common boundaries, dams, rivers and other topographical features which could influence the strategies adopted.*	 *Y* *Y* *Y*
Describe geological structures, such as: • *faults* • *dykes* • *stratigraphy (around individual orebodies or seams)*	 *Y* *Y* *Y*
Highlight any dangerous or difficult strata.	*Y*
Include typical section.	*Y*
Include map showing major geological features in relation to mining outlines and shafts.	*Y*
Give general description of orebodies or seams being mined, including relevant information such as: • *average mining depth* • *range of mining depths* • *orebody width* • *dip* • *strike*	 *Y* *Y* *Y* *Y* *Y*
Describe regional hydrology such as the occurrence of any significant groundwater and/or any relevant information.	*Y*
Describe ground control districts based on: • *known geological hazards* • *structures* • *jointing* • *changes in rock type* • *changes in rock strength* • *any other factors which may impact on mining.* *(Include nature of virgin stress field, occurrence of significant pore water and any other local geological features)*	 *Y* *Y* *Y* *Y* *Y* *Y*
Depict location and extent of above information on a plan.	*Y*
Tabulate 5 year history of rock-related: • *casualties* • *non-casualty incidents (where available).* *(Categorise according to rockfalls per 1000 employees at work for both surface and underground operations.)*	 *Y* *Y*
Present above information graphically, depict annual statistics and highlight trends.	*Y*

State who is responsible for: • *completion of accident report forms* • *maintenance and interpretation of mine accident statistics.*	 *Y* *Y*
Use accident report form 13 and ID root causes of fatal and reportable accidents	*Y*
Store above information in mine's data bank	*Y*

3 GENERAL INFORMATION

3.1 Locality

The ABC Diamond Mine is located 105 km North of Kimberley and 75 km west of Warrenton (Figure 1). The western portion of the 4 km east-west trending ABC Mining Lease cuts through the Theronskop Hill which forms a prominent feature 3 km west of the R370 main road. Access to the mine is via a gravel road 8 km north of Mount Rupert. The northern boundary of the mine is common with the Messina Diamond Mine. There are no rivers or dams on the property which may influence the strategies adopted.

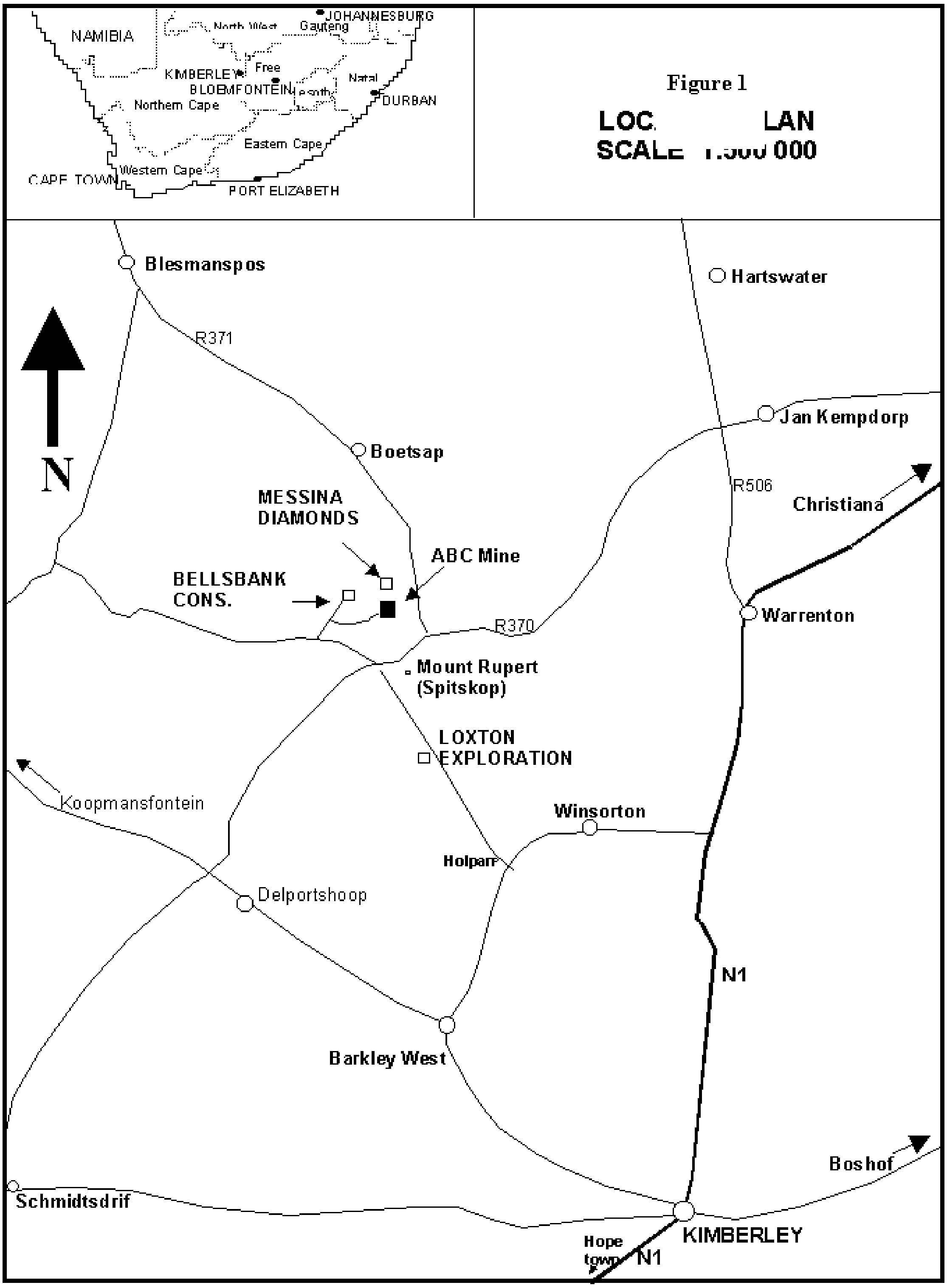

Figure 1 **Locality map**

3.2 Geological setting

ABC Diamond Mine extracts diamonds from near vertical kimberlite dykes locally known as fissures. The near vertical host fissure system to the kimberlite dykes trend 25 degrees east of true north and form part of the regional NE trending tensional system. Further south, the overall direction of this system becomes more easterly around Kimberley. The main fissure zone hosting the near vertical to steeply westerly dipping kimberlite dykes is a south-western extension of the Bobbejaan Fissure Zone. The main fissure zone has been mined to a distance of 900 metres south of the ABC Mine boundary.

The host rocks to the mine's ore bodies are dolomitic argillites and quartzites overlying Ventersdorp lavas which can be correlated with the lower Chuniespoort section of the Transvaal Sequence of the Pretoria System.

With reference to Figure 2, dolomites interlayered with argillite and quartzite form the dominant westerly dipping host rock unit to a depth of 250 metres. A unit of quartzites containing minor shale and dolomite lenses occurs between 250 and 290 metres below surface. This unit is called the Upper Geotechnical Area.

The base of these quartzites at 290 metres below surface forms an unconformity with the underlying Ventersdorp volcanic sequence and is intersected by the workings between the 800 and 1000m level. The lava country rock is called the Lower Geotechnical Area.

3.2.1 Structure

The major geological features at ABC Diamond Mines are shown in Figure 2 and can be described as follows:

- The Big Boy Fault displaced the kimberlite dykes and country rock by 20m laterally. This zone is associated with poor ground conditions and water inflow.

- Versit zones which are locations where the kimberlite dyke intrusion pinches out and is intruded into an adjacent fissure. Overlap of mineable kimberlite may occur in these zones. Versit zones do not present any hazards in terms of rock stability, but special precautions are required when the parting between overlapping stopes is smaller than 3m.

In addition to the major structures, the country rock in the upper geotechnical area is characterised by well bedded strata containing at least three near vertical joint sets with average strikes of N 33^0 W, N 28^0 E, and N 88^0 E. The spacings of these joints within each of the three sets vary between 0,3 and 1,5m on average. The lengths of the joints range from a few centimetres to several tens of metres. The joints do not posses significant waviness. In the vicinity of the kimberlite dyke the joint intensity parallel to the fissure trend increases significantly, up to 20 joints per metre.

The country rock in the lower geotechnical zone contains two steeply dipping joints sets trending at N 25^0 W and N 22^0E. The joints are spaced between 1,2 and 3,2 m apart. The lengths of the joints range from a few centimetres to several tens of metres. The joints do not posses significant waviness. In the vicinity of the kimberlite dykes the joint intensity increases and alteration of the lava occurs resulting in a weak disturbed zone which may be up to 2m wide on either side of the orebody.

The potential hazards may therefore be summarised as:

- **Poor ground associated with the Big Boy fault;**
- **Poor ground immediately adjacent to the kimberlite veins in lava;**
- **Instability when Versit zone overlap by less than 3 m.**

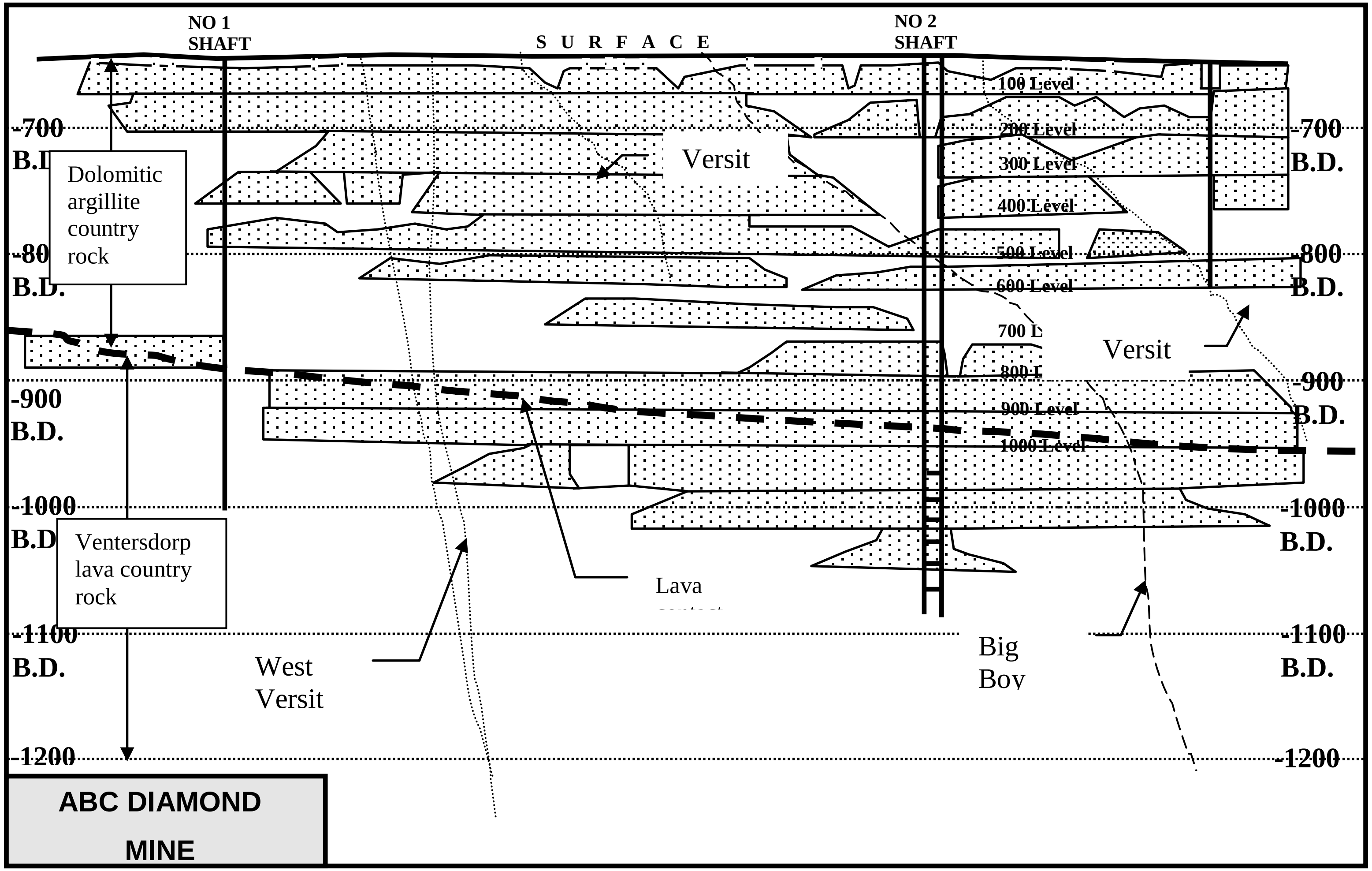

Figure 2 **Major geological features**

3.3 Orebodies mined

Only one orebody is mined, known as the Main Fissure. The mining method is underhand shrinkage stoping. The Main Fissure has been mined over a continuous strike length of 1200 meters to a depth of 400 meters below surface. The orebody dips at between 80^0 and 90^0 in a westerly direction and trends at 25^0 east of true north. The ore body consists of a series of overlapping kimberlite lenses averaging approximately 150 meters in vertical and horizontal extent, narrowing from a central width of 55-70 cm to 10 cm stringers on the peripheral margins. Stoping widths vary between 90 cm and 150 cm, depending on the condition of the country rock. Each overlapping lens is mined as a separate stoping unit with a waste parting of between 5-20 meters separating the lens margins. The near vertical lenses therefore do not provide a continuous vertical slot between surface and the lowest mining level.

3.4 Regional hydrology

The gently easterly sloping drainage of the almost flat dolomite capping of the Ghaap Plateau presents no topographical features which could channel water into the mine.

Drainage from the mines on the parallel Bellsbank fissure system situated 2 km to the west drains over the 100 meter fall of the escarpment into Hartsrivier valley well east of the surface workings.

The main source of water ingress to the underground workings is from surface run-off. This easterly flowing sheet-water finds its way down via surface workings and fracture systems.

The Big Boy fault zone is the major structure retaining water in the mine. This water has to date been controlled and caused no major disruption to mining operations.

The dolomitic argillites and quartzites are impervious to water and there are no other known water bearing fractures or cavities caused by dolomite weathering.

3.5 Seismological setting of the mine

The mine is set in a stable geological environment of shallow westerly dipping Karoo sedimentary units which lie unconformably on the stable base of the Ventersdorp Volcanic unit. There has been no record of any natural seismological activity in this mining area. There have been no reports of mining induced seismicity or strain bursting since mining commenced at ABC Mine in 1978. No mining induced seismicity is therefore expected over the next two years.

3.6 Ground control districts

The mine has been divided into two ground control districts based on the composition of the country rock. The upper ground control district is confined to the dolomitic argillite rocks which have historically sustained mining provided support has been installed to allow future access. The lower ground control district is located in lava, which contains varied ground conditions adjacent to the fissure and is generally very good away from the fissure. The two ground control district are described in Table 1 and are illustrated in Figure 3.

Table 1 Ground control districts at ABC Mine

LEVEL DEPTH	NAME	DESCRIPTION
Surface to 900 Level (0-250m)	**Upper ground control district**	Country rock is dolomitic argillite. Contains well-developed fracture cleavage parallel to the kimberlite dyke. Sidewalls of underground stopes will not stand without support due to slabbing and collapse of shale bands. Zone of deformation confined only to kimberlite margins (max. 30 cm each side).
Below 900 level (>250 metres)	**Lower ground control district**	Country rock is metavolcanic lavas consisting of tuffaceous, breccias, vesicular, and fine andesites interlayered with fine-grained quartzites and shales. Kimberlite dyke wall rock margins can have varied width of alteration due to exfoliation of lavas, and secondary cleavage direction, causing blocky ground in sidewalls of excavation. Highly fractured coarse textured lavas can create unpredictable ground conditions.

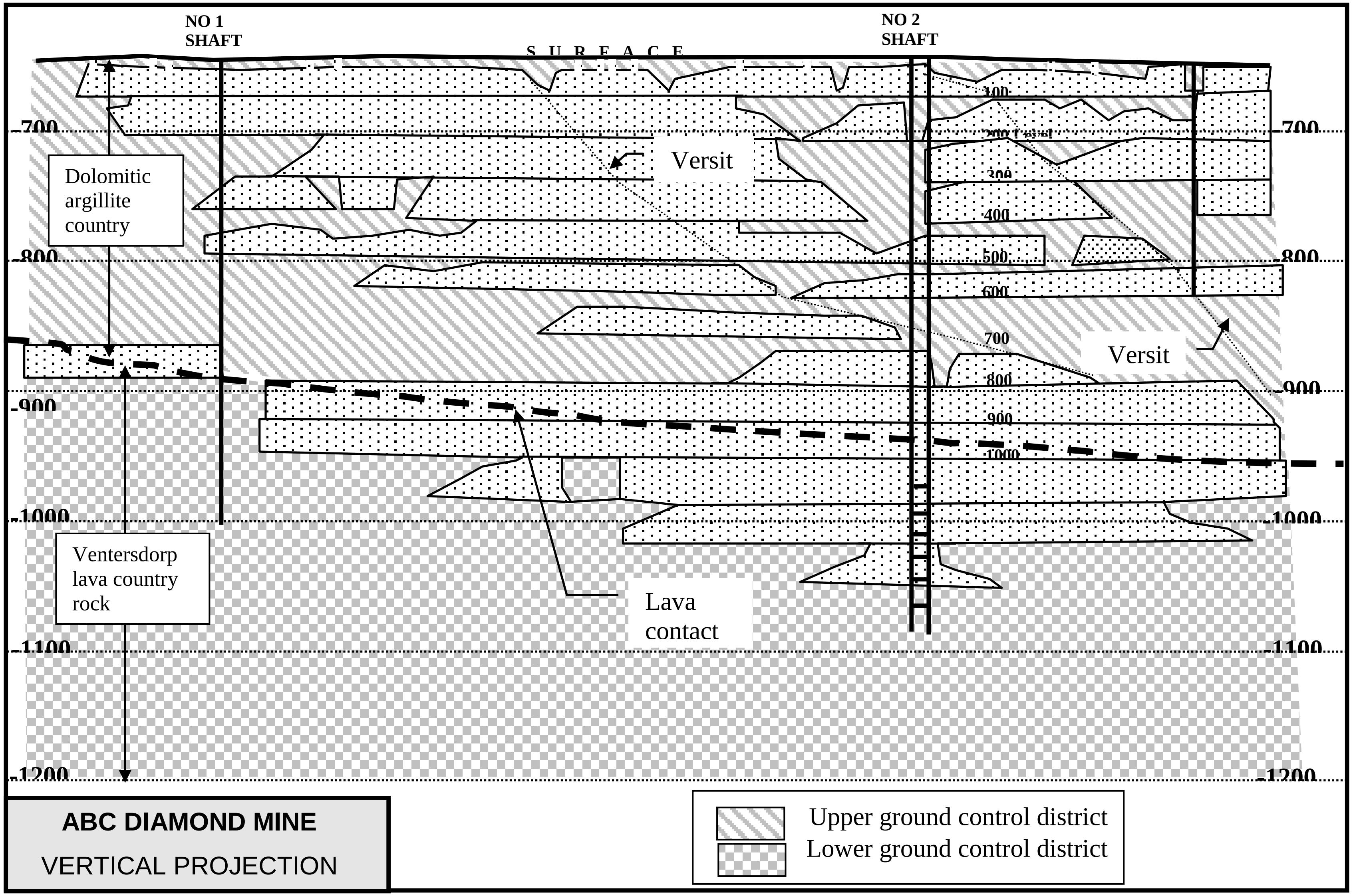

Figure 3 Ground control districts

3.7 Mine rockfall accident analysis

3.7.1 Rock-related accident statistics

The mine's rockfall accident records between 1994 and 1999 are summarised in Table 2. The trend in the number of rock related accidents per year is shown in Figure 4.

Figure 5 is a graphic representation of the origin of rocks causing rockfall accidents shows that raise development is the most frequent cause of rock related accidents. The overhand stope face and development hanging wall are the next most frequent sources of rockfall accidents.

Table 2 Rock-related accidents (1995 – 1999)

Date	Working Place	Accident Type	Excavation Type	Location	Origin	Cause
12/3/95	800/6w x-cut	Reportable	Development	Face	Hanging	Lack of (or inadequate) standards / procedures
3/9/95	300/2E	Reportable	Stope	Face	Face	Inadequate examination
22/11/95	600/12W	Reportable	Stope	Face	Rolling rocks	Inadequate examination
13/12/95	500/13W	Reportable	Raise	Face	Face	Inadequate examination
9/1/96	400/24E drive	Reportable	Development	Face	Hanging	Failure to comply with standards
12/4/96	400/29E	Fatal	Stope	Face	Face	Support not installed
28/5/96	800/5E x-cut	Reportable	Development	Face	Hanging	Inadequate examination
7/6/96	800/12E	Reportable	Raise	Face	Face	Inadequate examination
2/7/96	300/12W	Reportable	Raise	Face	Face	Inadequate examination
9/11/96	800/23E drive	Reportable	Development	Face	Hanging	Inadequate examination
13/2/97	800/28E drive	Reportable	Development	Face	Hanging	Support ineffective
3/3/97	500/12W	Reportable	Stope	Face	Sidewall	Inadequate examination
18/8/97	600/12W	Reportable	Raise	Face	Face	Inadequate examination
22/9/97	700/12W	Reportable	Stope	Face	Rolling rocks	Inadequate examination
14/10/97	600/3E	Reportable	Raise	Face	Face	Inadequate examination
3/12/97	800/12W	Reportable	Raise	Face	Face	Inadequate examination

8/3/98	300/6W	Reportable	Stope	Face	Face	Inadequate examination
26/6/98	300/12E	Reportable	Travelling way	Back area	Rolling rock	Inadequate examination
19/11/98	800/24W	Reprotable	Raise	Face	Face	Inadequate examination
7/3/99	900/17E	Reportable	Development	Face	Hanging	Insufficient support
19/9/99	1100/2W	Reportable	Stope	Face	Face	Inadequate examination
6/11/99	700/22E	Reportable	Raise	Face	Face	Inadequate examination

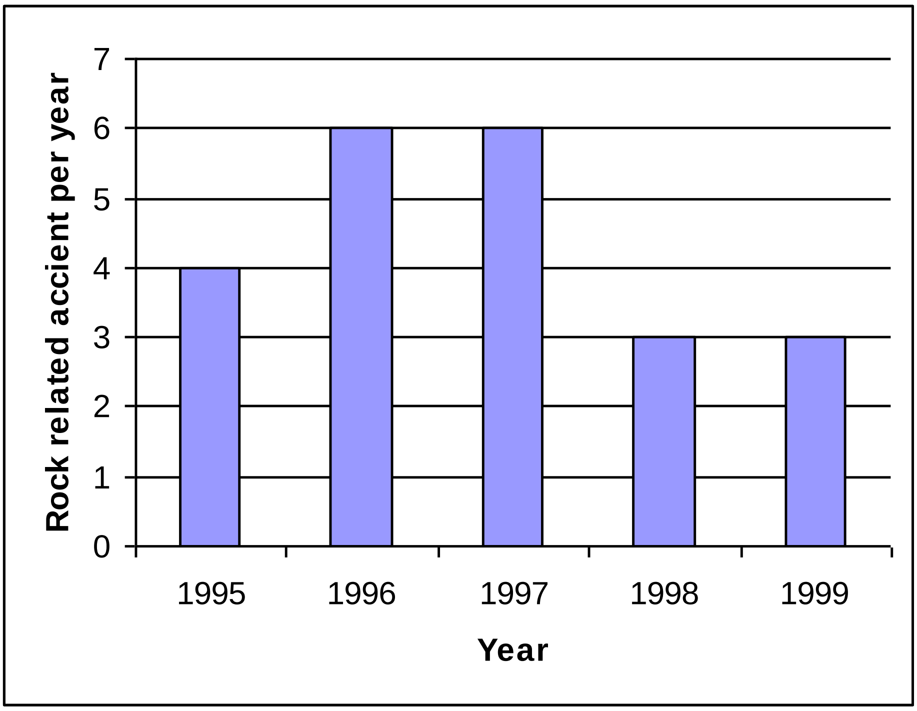

Figure 4 Trend of accidents per year

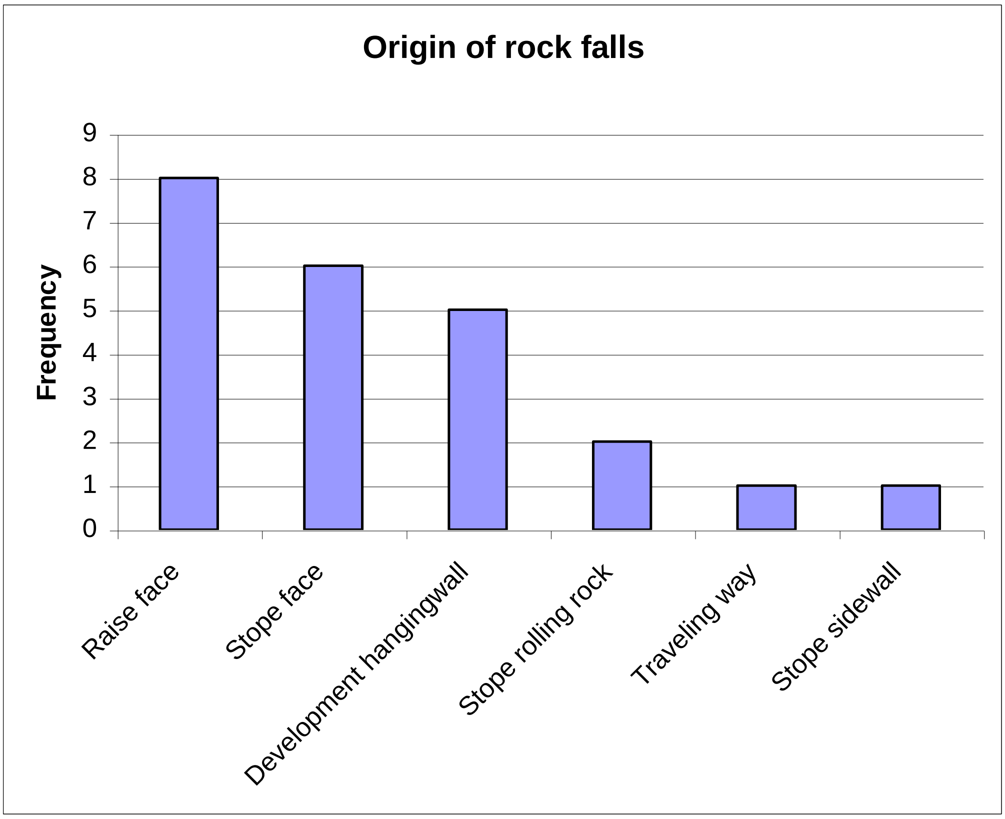

Figure 5 **Origin of rockfalls**

The activities of persons at the time of accidents and the location of the persons are summarised in Figure 6. The results indicate that drilling in raises is an area requiring particular attention. In shrinkage stopes drilling operations and installing support are the most likely activities during accidents.

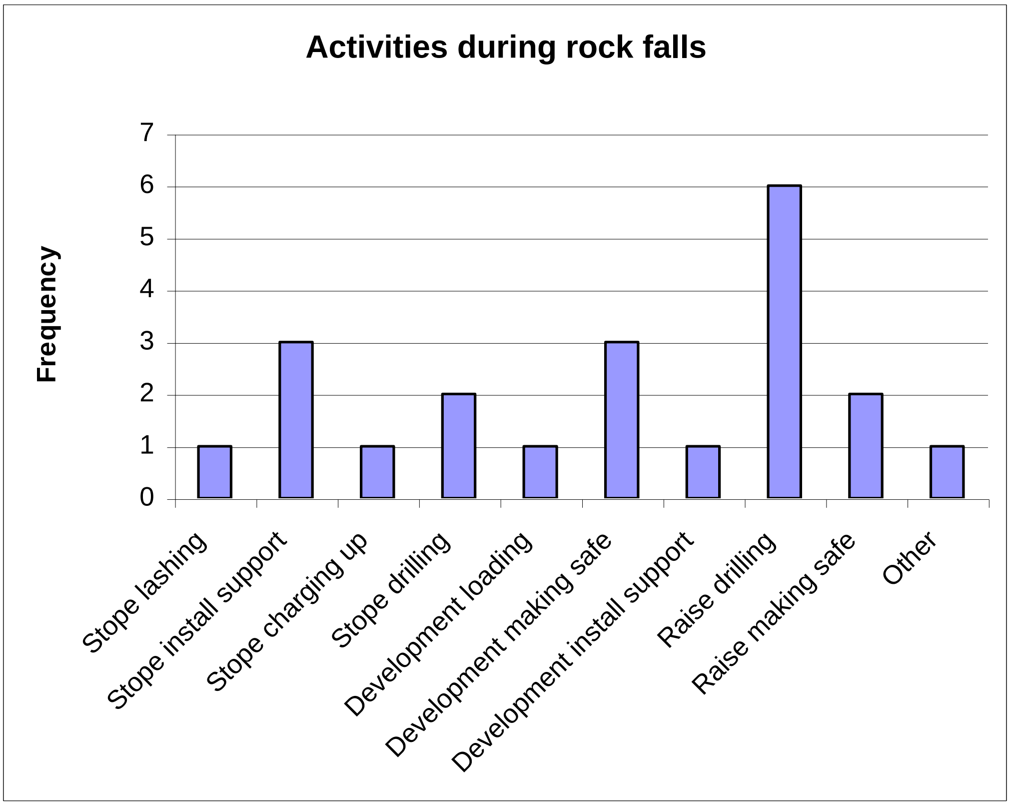

Figure 6 **Activities during rockfall accidents**

3.7.2 Fall of Ground Accident Records

As required by the Mine Health and Safety Regulation 34.1, ***accident report Form 13 must be completed for all incidents that result in the loss of 14 or more shifts***. This form is formulated in a manner which facilitate identification of the root causes of fatal and reportable accidents.

The responsibilities of safety representatives, supervisors, shift bosses and other officials with regards to accident investigations, completion of accident report forms, maintenance and interpretation of mine accident statistics are as follows:

- **Safety Representative responsible for area where accident occurred**:
  - He must assist with all F.O.G. accident investigations.
  - He must assist the injured person□σ immediate supervisor with the completion of the accident report form.

- **The injured person□s immediate supervisor:**
  - He must assist with all F.O.G. accident investigations.
  - He must complete the accident report form.

- **The shift boss in charge of the area where the accident occurred:**
  - He must arrange for an in loco investigation of all F.O.G accidents.
  - He must inform the responsible safety representative, supervisor and safety officer and other officials (if necessary) of the time and place of the accident investigation.
  - He must attend all in loco F.O.G. accident investigations.
  - He must ensure that the accident report form is completed by the injured person□s immediate supervisor.
  - He must complete his section of the accident report form.
  - On completion of the accident report form by the responsible mine overseer, the shift boss must forward the accident report form to the safety officer.

- **The mine overseer in charge of the area where the accident occurred:**
  - He must attend all in loco investigations for F.O.G. reportable accidents.
  - He must complete his section of the accident report form.

- **The safety officer:**
  - He must attend all in loco investigations for F.O.G. reportable accidents and all other F.O.G. accidents as far as possible.
  - He must receive all F.O.G. accident report forms and ensure that they have been completed properly.
  - He must analyse the accident and report his findings to the Health and Safety Committee at the monthly meeting.
  - He must keep copies of the accident report forms on file.
  - He must update the F.O.G. accident data base on a monthly basis.
  - He must ensure that Accident Form 13 is completed after each F.O.G. accident.

GLOSSARY OF TERMS AND DEFINITIONS	
Incorporate a glossary of terms and definitions	*Y*

4 GLOSSARY OF TERMS AND DEFINITIONS

A glossary of terms and definitions is included in Appendix B. This appendix forms part of the COP.

ROCK-RELATED RISK MANAGEMENT	
• *Identify and describe rock-related hazards which are likely to arise from the mining of each geotechnical area identified.*	*Y*
• *Assess and prioritise the health and safety risks to which workers will be exposed and record findings.*	*Y*
• *Develop and implement reasonably practicable strategies to reduce and manage these risks, based on above risk assessment and accident analysis.*	*Y*
• *Use Tripartite Risk Assessment Guidelines when dealing with the aspects of hazard ID and risk assessment.*	*Y*

5 ROCK-RELATED RISK MANAGEMENT

5.1 Introduction

A baseline rock-related risk assessment has been carried out by the risk assessment team consisting of the following employees:

Name	**Occupation**
M. H. S.	Mine Overseer
A. P. B.	Safety Officer
J. M.	Health and Safety Committee member
J. J. D.	Shift Boss
D. N.	Training Officer
J. N.	Miner
P. T.	Team Leader

The fall of ground accident analysis and identification of all fall of ground hazards were used as a basis of the risk assessment. Risk ranking was carried out in terms of location (geographic), tasks being performed and identified hazards. A risk ranking matrix, shown as Table 3 was used to prioritise the risks. Risk rankings of 7 and higher are considered high risks.

Table 3 Risk ranking matrix

CONSEQUENCE	PROBABILITY/FREQUENCY				
	Common (Daily) A	**Likely (Weekly) B**	**Happens (Monthly) C**	**Unlikely (Yearly) D**	**Rarely (1 - 5 years) E**
Fatality R500 000+ 1	1	2	4	7	11
Reportable Injury R250 000 - R500 000 2	3	5	8	12	16
Disabling Injury R100 000 - R250 000 3	6	9	13	17	20
First Aid Case R50 000 - R100 000 4	10	14	18	21	23
No Injury Nil - R50 000 5	15	19	22	24	25

5.2 Additional risks identified

The risk of large scale collapse of the working was identified as a potential hazard. Since the mine started working from sub-outcrop workings to the current depths no large scale collapses have occurred in any of the mined out areas. Information gathered from surrounding mines which extract similar ore bodies indicate that large scale collapses have not occurred in any of the mines. However, The following potential mode of large scale failure was identified, based on rock engineering principles:

- The large scale failure of the surrounding rock into unsupported open stopes may be initiated by the growth of tensile stress zones around the stope excavations as the stopes increase in span.
- Collapse may occur when the rock within the tensile zone becomes unstable and fails by shearing along a circular slip surface or toppling into the open stopes.

Large scale failure was therefore considered in the baseline risk assessment.

5.3 Results of baseline risk assessment

Tables 4 to 7 below systematically describe the different fall of ground hazards and their associated risk rankings.

Table 4 Risk ranking – Upper ground control district

Fall of ground hazard description			**Frequency**	**Conse-quence**	**Risk Rank**
Stope	Face area	Stope face	C	1	4
		Sidewalls	C	3	13
		Rolling rocks	B	3	9
	Accessway (poleway)	Side walls	D	2	12
		Stope face	E	2	16
	Draw point	Rolling rocks	B	4	14
Horizontal development	Face area	Hanging wall	D	2	12
		Tunnel face	D	3	17
		Side walls	D	3	17
	Back area	Hanging wall	E	3	20
		Side walls	E	3	20
Raise development	Face area	Face	B	2	5
		Sidewalls	D	2	12

Table 5 Risk ranking – Lower ground control district

Fall of ground hazard description			**Frequency**	**Conse-quence**	**Risk Rank**
Stope	Face area	Stope face	C	1	4
		Sidewalls	B	3	9
		Rolling rocks	B	3	9
	Accessway (poleway)	Side walls	D	2	12
		Stope face	E	2	16
	Draw point	Rolling rocks	B	4	14
Horizontal development	Face area	Hanging wall	D	2	12
		Tunnel face	D	3	17
		Side walls	D	3	17
	Back area	Hanging wall	E	3	20
		Side walls	E	3	20
Raise development	Face area	Face	B	2	5
		Sidewalls	D	2	12

Table 6 Task based risk ranking

Fall of ground hazard description		Frequency	Conse-quence	Risk Rank
Stope	Barring	C	1	4
	Drilling face	C	1	4
	Lashing	D	1	7
	Support installation	D	1	7
	Charging up	E	1	11
	Inspection	E	1	11
Horizontal development	Barring	E	1	11
	Drilling face	E	1	11
	Cleaning with loader	E	1	11
	Lashing	E	1	11
	Install support	D	1	7
Raise development	Barring	C	1	4
	Drilling face	C	1	4

Table 7 Hazard based risk ranking

Fall of ground hazard description		Frequency	Conse-quence	Risk Rank
Lava country rock with geologically disturbed contacts	Falls in stopes from friable sidewalls	B	3	9
	Falls in development from friable surrounding rock	C	3	13
Big Boy Fault zone	Falls in stopes from friable sidewalls	C	3	13
	Falls in development from friable surrounding rock	C	3	13
Overlapping stopes less than 3m apart at versit zones	Breakthrough of one stope into another	D	3	17
	Instability of intervening pillar	D	3	17
Large scale collapse of workings	Large scale failure of sidewalls into stope excavations	E	1	11

The results of the risk analysis indicate that the following are the hazards requiring immediate attention in order of priority:

- Falls of ground from the overhand stope face;
- Barring and drilling the stope face
- Barring and drilling raises
- Falls from the face area during raise development;
- Lashing in stopes;
- Support installation in stopes;
- Support installation in development;

Each of these high risk areas is addressed in more detail in the following chapter.

Strategies to Reduce and Manage Rock-Related Risks
• *State department or persons responsible for the execution of the particular strategies or portions thereof.* • *Provide time table for preparation and implementation of strategies.* • *Derive mine standards from strategies.*

Mining Method, Sequence and Overall Mine Stability:
• *Include measures to avoid failures that may injure employees or damage mine excavations or equipment.*
• *Take into account:*
  - *the geotechnical environment*
  - *potential major rock related hazards identified in risk assessment*
• *Describe:*
  - *mining method*
  - *mining sequence to be followed.*
• *Describe strategy adopted to manage risk where mining of one orebody can be expected to have an adverse effect on the other.*
• *Describe in detail the use of ongoing RE input in mine layout design and performance monitoring.*

Describe the design methodology to avoid uncontrolled collapses.
Describe the effects on: • *surface structures* • *topography.*
Describe: • *methodology* • *criteria used for the design of in-panel and barrier pillars.*
Give reasons for selecting a specific type of pillar.

6 STRATEGIES TO REDUCE AND MANAGE ROCK-RELATED RISKS

6.1 Mining Method

The mining method in both geotechnical areas is Overhand Shrinkage Stoping. Changes to the mining methods will be appraised to assess exposure to risk of personnel, production and main mine infrastructure that will include a rock fall hazard appraisal and a rock engineering appraisal. The manager will be responsible to arrange such an appraisal.

6.2 Strategy to ensure overall mine stability

The ABC Diamond Mine may be classed as shallow, hard rock mine extracting a vein type ore body which is nearly vertical. Since no precedents exist of large scale instability, it is not possible to design regional stability measures by referring to past events. Limiting mining spans and monitoring of rock displacements will be used as strategies to ensure overall mine stability. The manager is responsible to ensure that the strategies are followed. Each strategy is described below in greater detail.

6.2.1 Regional stability by limiting mined out spans

The primary strategy to prevent large scale instability will be to limit the mined out spans in future so that they do not exceed current spans that are known to be stable. Mined spans of up to 150 m on dip by 400 m on strike have remained stable in the past. Versit zones in the orebody form natural breaks which reduce the mined out spans. The Versit zones will be utilised to reduce future mined out spans, and will be indicated on the mine plans. Should suitable Versit zones be absent, the manager, in consultation with the rock engineering practitioner, will design an alternative method of limiting the potential for instability.

6.2.2 Monitoring of rock mass stability

Careful monitoring of rock mass behaviour allows early detection of potential instability. Monitoring will initially be carried out by visual observation of open cracks in the rock mass adjacent to the ore body. The mine overseer will be responsible for inspecting excavations and reporting open cracks in the rock mass. Opening of cracks will be reported to the manager, who will record these on a plan. Should an unexpected increase in open cracks occur, the manager in consultation with the rock engineering practitioner, will plan appropriate remedial action. The rock engineering practitioner will inspect the plan on his regular visits to the mine and recommendations for additional monitoring or remedial action will be made by the manager, in consultation with the rock engineering practitioner, as required.

6.2.3 Mining sequence

Owing to the shallow depth and low stress environment, the sequence of mining has a limited effect on the stability of the workings. The overall mining sequence will be to mine away from the shafts. The formation of isolated remnants will be avoided as far as practically possible. Where multiple fissures exist and advancing stope faces are within a horizontal distance of 10m of one another, stoping shall only occur in one fissure at a time. The potential of inadvertently holing of one stope into the other will be avoided. As the pillar between two fissures narrows, the stability of the intervening pillar becomes critical and only mining in one of the fissures at a time reduces exposure of personnel.

The manager will be responsible for ensuring that the overall mining sequence strategy is adhered to.

6.2.4 Surface effects

Owing to the steepness of the orebody and the narrow mining width, surface effects of mining have been minimal. On occasion small sinkholes have formed directly above near outcrop workings. These sinkholes pose no threat in terms of fall of ground accidents. The outcrop area has been fenced off or made safe where necessary. Large scale subsidence or caving has not occurred in the past and is considered highly unlikely. The mine surveyor will be responsible for inspecting the ground surface for the formation of sinkholes and reporting to the manager, who will be responsible for making safe and fencing off.

The Influence of Mining Activities on Neighbouring Mines	
• Describe method to ensure that, where the possibility exists of one mine's activity influencing the activities of another mine, the mines concerned exchange data concerning:	
- *mining methods*	*Y*
- *regional support*	*Y*
- *mining sequence*	*Y*
- *common geological features*	*Y*
- *% extraction*	*Y*
- *the location, magnitude and nature of seismic events.*	*Y*
• Include timing and overall sequencing for the removal of the boundary pillar where applicable.	*NA*

6.3 The Influence of Mining Activities on Neighbouring Mines

Mining activities on neighbouring mines are remote from ABC's boundaries and well outside the zone of influence. Mining on neighbouring mines is therefore not a factor to be considered in this COP. However, this situation will be reconsidered at least with every review of this COP.

Rockburst control	
• *Describe seismic monitoring, seismic emission control and rockburst damage control measures*	*Y*

6.4 Rockburst control strategy

The mine is set in a stable geological environment and natural seismicity that may affect the mine stability is not expected. Owing to the shallow depth of the workings no mining induced seismicity has occurred. It is therefore unnecessary to develop a strategy for rockburst control.

Stope and panel support	
• *Strategy must accommodate the conditions expected from the:* - *geotechnical settings* - *type of accident encountered.*	*Y*
• *Refer to:* - *accident analyses;* - *risk assessment done for the ID of problem areas.*	*Y*
• *Include:* - *gully support design methodology;* - *the excavation sequence for the gully siding, if any, in relation to the face.*	*Y*
• *Describe:* - *location;* - *support of two separate access ways for each panel.*	*Y*
• *Describe methodology to select the most appropriate support design requirements (e.g. energy absorption capability, yieldability, areal coverage) to reduce the risk.*	*Y*
• *When experimenting with a mining or support system, it must be subjected to a risk assessment and fully documented.*	*Y*

6.5 Stope Support Strategy and Design Methodology

6.5.1 Overall strategy for stope support

The accident statistics and risk assessment has shown that gravity driven falls of ground and rolling rocks are the two modes of failure that need to be addressed by the support system in stopes. Since the hazards are similar in both geotechnical areas, the overall strategy is similar in both areas. The strategy is to:

- Eliminate the hazard by removing the unstable blocks in a controlled manner;
- Control the rock fall hazard by supporting potentially unstable blocks that cannot be removed;
- Control the rolling rock hazard by reducing the slope of the shrinkage pile;
- Reduce exposure of employees to hazardous conditions;
- Reduce the risk by training employees in the proper procedures for making safe, installing support and hazard awareness training.

The manager will be responsible for ensuring the strategies are implemented.

6.5.2 Stope support design methodology

The support design methodology is largely empirically based. In order to design support, the *loads* that will be acting on the supports, the *capacity* of the supports and the support *spacing* must be determined. This is the responsibility of the rock mechanics practitioner.

Support loading by blocks released from the stope face: Support will be provided which is able to carry at least the dead weight of the maximum unstable block which has fallen out from the overhanging stope face in the past. It is assumed that an unstable block is free to fall in the vertical direction, and is driven by gravity. The support resistance (R) will be calculated as follows:

$$R = \rho g t$$

where ρ is the density of the rock in kg/m^3, g is gravity acceleration and t is the thickness of the potential rock fall. The support resistance is usually expressed in units of kN/m^2.

Support loading by unstable slabs in the stope sidewall: Support is required to prevent toppling and sliding of unstable slabs of rock into the stope. The support capacity and spacing will be determined using the maximum block size that has resulted in instability in the past. The method of calculating the required support resistance is presented in Appendix 1.

Support spacing : The support spacing is determined by the spacing of natural and blast induced fractures in the rock. If the supports are too widely spaced, rocks will fall out between the supports, in addition, the support resistance requirement may not be met. In the absence of accepted criteria for determining support spacing the experience at ABC Mine will be used, which indicates that large falls in stopes are adequately controlled by supports 2m apart. The support spacing will therefore be based on satisfying the required support resistance as well as experience which indicates that supports should be spaced no more than 2m apart.

Support capacity: The strength of support units is commonly expressed in terms of tons or kN, and refers to the tensile strength of support tendons or compressive strength of mine poles and mechanical props as determined in a laboratory. However, the strength of timber support units, when determined in a laboratory, should be down-rated by 50% to allow for factors such as creep and poor contact owing to irregularities of the rock surfaces.

6.5.3 Support areas

Stope support is required to provide safe working conditions in the face and in travelling ways within the stope. Two areas are defined :

- *The stope face area* is the area between the advancing stope face and the shrinkage pile. The stope face area is also used as the main access way along a stope.

- *Travelling ways* are any other means of access into the stope workings and may include timber ways through the shrinkage pile.

6.5.4 Support of the stope face area

Stope face area to be supported : Owing to the mining method, persons are able to access the face area only. The back area is filled with shrinkage material and no hazard of rock fall injuries exist behind the face area. Stope face support is required to be effective for no more than 2,5m from the face. Owing to the shallow working depth and limited distance requirement for stope support units to be operational, the stope closure is not a concern in support design.

Rock to be supported : The maximum rock fall that has occurred from the stope face at ABC mine was 50 cm thick, the density of the ore is 2800 kg/m^3. Smaller rocks fall from the face, especially after a blast or during drilling. These small rocks are typically 10 cm thick and may be up to 50 cm long. Falls of ground from the sidewall have occurred mainly by slabs of rock that topple into the stope. The thickness of these slabs has seldom exceeded 20 cm.

Control of small rock falls from the stope face: The hazard associated with small unstable rocks, approximately 10 cm thick and up to 50 cm long, is best addressed by elimination. The stope face will be barred and made safe after each blast and before drilling into the face commences to ensure that any small unstable rocks are removed. The face will be mined in such a way that brows will not be formed. The miner will be responsible to ensure that the stope face is made safe and that brows are not formed.

Control of larger rock falls: Larger potentially unstable blocks will be controlled by providing support. Based on the thickness of the largest reported rock fall from the face, the required support resistance is:

$$
\begin{aligned}
R &= 2800 \times 9{,}81 \times 0{,}50 \\
&= 13{,}2 \quad (kN/m^2)
\end{aligned}
$$

The above support resistance may easily be achieved by mine pole support units. The empirical requirement that support units should not be more than 2m apart will be the limiting factor. For example, if 100 kN capacity mine poles are spaced 2m apart down the face and the stope width is 1,2 m, the resulting support resistance *S* will be:

$$S = \frac{50}{1{,}2 \times 2{,}0}$$
$$= 20.8 \quad (kN/m^2)$$

Note that the capacity of the mine pole has been downgraded to 50 kN to allow for the difference between laboratory test results and actual performance in underground excavations.

Prevention of unstable sidewalls: The hazard of fall of ground accidents from unstable sidewalls will be controlled by limiting the height of the exposed sidewall to no more than 2,5m in shrinkage stopes. While every effort will be made to maintain the face to shrinkage pile distance less than 2,5m, over-pulling may result in greater distances. In this case additional support will be installed to limit the vertical unsupported distance to no more than 2m. Sidewall support will be designed as described below.

Support resistance requirements for stope sidewalls : In a vertical stope the support is required to hold loose slabs of rock in position in the sidewalls. The support should generate horizontal forces sufficient to hold slabs of rock in position by preventing toppling or sliding of the slabs into the stope face area. The required horizontal support resistance is nominal, according to the design methodology presented in Appendix 1. A support resistance of 20 kN/m^2 will be more than adequate to control toppling and sliding of slabs up to 1m thick in the sidewalls of a stope.

Specification of temporary supports for stopes: Temporary supports may be installed immediately after the blast in locations where the face or sidewalls are deemed to be unstable after making safe. The miner will be responsible for making safe and installing temporary supports. Temporary supports may be mechanical jacks or similar devices which have a capacity of 100 kN.

Prevention of rolling rock accidents: Control of the angle of the stope face is the best way of preventing rocks from rolling along the top of the shrinkage pile. The overall slope of the stope face shall not exceed 40°. Where local variations exist in the slope angle, it should not exceed 40° over a length of more than 5m. Should these requirements be exceeded, gate stulls should be built at 5m intervals along the shrinkage pile to arrest rolling rocks. The miner will be responsible to ensure that these requirements are met.

6.5.5 Support strategy for stope accessways (pole ways)

Since no fall of ground accidents have been reported in stope accessways, it is concluded that the current methods are adequate, namely providing unsupported raises developed 3m from the fissure. Should ground conditions deteriorate to such an extent that a danger exists of loose rocks falling down a travelling way, the mine overseer shall notify the manager who will who will specify appropriate support in consultation with the rock engineer.

6.5.6 Strategies to reduce exposure to hazards

The following strategies are designed to minimise exposure of workers to hazardous conditions after the blast and prior to the installation of support.

- Nobody is allowed to enter a stope panel prior to the installation of temporary support, except for persons responsible for making safe and installing temporary support. Other workers are allowed to work in supported areas of the panel only and may move in other areas only after the installation of the necessary temporary or permanent support.
- Any area, which is not supported according to mine standard, must be barricaded off. This barricade may only be removed once the area has been made safe and the necessary temporary and/or permanent support has been installed. The miner will be responsible for barricading areas.
- When making safe, work from a safe area.

6.5.7 Modifications to support systems

When experimenting with a mining or support system it must be subjected to a risk assessment by the manager. Where unacceptably high hazard is identified, or there is significant uncertainty, a rock engineering appraisal will be carried out.

Tunnel and service excavation stability	
• *Describe strategy to ensure the safety of personnel working and/or travelling in tunnels.*	*Y*
• *Include:*	
- *opening-up;*	*Y*
- *re-supporting procedures.*	*Y*

6.6 Tunnel and Service Excavation Support Strategy and Design Methodology

6.6.1 Overall strategy for tunnel and service excavation support

The results of the rockfall hazard assessment indicated that falls from the hanging wall of tunnel development may result in minor accidents during drilling and charging up operations. The overall strategy is to:

- Locate service excavations so that the potential for instability is minimised;
- Eliminate the rock fall hazard by removing the unstable rock in a controlled manner;
- Control the rock fall hazard by supporting potentially unstable blocks that cannot be removed;
- Maintain safe conditions in travelling ways and accessways through regular inspection and making safe.
- Reduce exposure of employees to hazardous conditions;
- Reduce the risk by training employees in the proper procedures for making safe, installing support and hazard awareness training.

The manager will be responsible to ensure that the strategies are adhered to.

6.6.2 Tunnel and service excavation support design methodology

Owing to the differing conditions in the two geotechnical areas, support will be designed separately for each geotechnical area. The following method will be used:

- The manager in consultation with the rock engineering practitioner will design support for good, average and poor rock conditions in each geotechnical area.
- The support design will be based on rock mass classification and the use of support design tables associated with the classification system. Once the basic support requirements have been determined from the design tables, modifications may be made to suit local conditions and experience.
- The rock mass will be classified using either the Q-System of classification (Barton, Lien & Lunde, 1974) or the Mining Rock Mass rating (MRMR) of Laubscher (1990).
- The type of support to be installed will be based on either the support chart of Grimstad & Barton (1993) if the Q-System of classification is used or the support guidelines of Laubscher (1990) if the MRMR system is used.
- Support resistance will be determined based on the thickness of the largest expected unstable block in the hangingwall of tunnels. The support resistance (R) will be calculated as follows:

$$R = \rho g t$$

- where ρ is the density of the rock in kg/m^3, g is gravity acceleration and t is the thickness of the potential rock fall. The support resistance is usually expressed in units of kN/m^2.
- The length of support units will be at least one half the width of the excavation being supported.
- The spacing of support units will be determined be considering the frequency of jointing, aerial support, length of support and recommendations of
- The rock engineering practitioner will ensure that the final support designs are in accordance with good rock engineering practice.
- The recommended support will be written into the mine standards.

6.6.3 Location of rock drives

Rock drives will be located a sufficient distance from the orebody so that the pillar formed between the drive and the orebody will remain stable for the service life of the drive. The manager will specify the appropriate distance after consultation with the rock engineering practitioner. When considering an appropriate distance, the performance of drives elsewhere on the mine, the rock mass conditions and the field stresses will be considered.

6.6.4 Support of tunnels and drawpoint cross cuts

The design methodology in Section 6.6.2 will be applied to design suitable support for tunnels and drawpoint cross cuts and will be written into the mine standards.

Where conditions exist that are not catered for in the mine standards, suitably trained personnel will notify the manager, who will consult with the rock engineering practitioner so that a modified support system may be designed.

The quality of permanent support installation will be ensured by proper training and supervision of miners and their crews who are responsible for the installation. The training officer will be responsible to ensure that employees are adequately trained.

The long term stability of the excavations will be ensured by regular inspection and making safe. The shift supervisor will be responsible for maintaining safe travelling and accessways. The long term stability of the excavations will be verified by the rock engineering practitioner on his regular visits to the mine.

When a tunnel is no longer in service, it will be barricaded or sealed off appropriately to prevent access by employees. The manager will be responsible to ensure that access to disused tunnels is prevented.

6.6.5 Support during tunnel development

During development two support types will be used, being temporary support and permanent support.

Temporary support is used to ensure that drilling of support holes for permanent support and charging up of the face occurs under safe working conditions. Mechanical jacks with a capacity of 100 kN are suitable for this application. Owing to the generally good rock conditions temporary support will be installed only where suitably trained personnel deem it necessary. Barring of the rock faces to remove loose rocks prior to commencing work at the face is required. The miner is responsible to ensure that barring is carried out satisfactorily.

When developing a cross-cut towards the fissure, the fissure contact zone represents an increased hazard owing to the highly fractured nature of the rock near the fissure. Temporary supports will be used when the cross-cut is within 3m of the fissure. Extra caution is required when barring and supporting this area.

Permanent support is used to ensure the long term integrity of the tunnels. Permanent support will be installed in tunnels if dictated by rock conditions. The support design methodology will be used to specify the required support and will be written into the mine standards.

6.6.6 Re-support of tunnels

The removal of old supports and installation of new supports shall be carried out under the supervision of a suitably trained person. The manager shall specify the new support to be installed and the procedures for installation, after appropriate consultation with the rock engineering practitioner.

6.6.7 Service excavation support

Service excavations include any excavations other than tunnels or stopes. When service excavations are less than 4m in width and height, they will be treated as tunnels. Support for large service excavations (exceeding 4m width or height) will be specified by the manager after appropriate consultation with rock engineering practitioner. The design will be based on initial estimates of required support using an accepted rock classification technique together with the appropriate support design tables. The rock engineering practitioner will ensure that the support design conforms to accepted rock engineering principles.

Mine access protection	
• *Describe strategy for the protection of shafts and/or the main entrances.*	*Y*
• *Include a summary of the rock engineering appraisal of the existing accessways' current stability.*	*Y*
• *Include procedures employed to monitor ground movement in shafts and accessways where danger of instability exists.*	*Y*
• *Include steps taken to minimise the risk associated with such moments.*	*Y*

6.7 Mine Access Protection

6.7.1 Appraisal of current stability of mine accessways

The mine access consists of vertical lined and unlined shafts. Regular inspections of the shaft over the last five years have indicated that the shafts are stable and there is no need for concern regarding their local stability. Since the operating shafts do not intersect the ore body, shaft protection pillars are not required. However, the shafts are relatively close to the orebody, and large scale caving of the orebody sidewalls or gradual scaling and enlargement of the stope excavations could affect the stability of the shafts. The shafts and their locations relative to the fissure are presented below.

Table 8 Location of shafts relative to fissure

Shaft	Depth	Nearest Distance to fissure	Depth nearest to fissure	Lining
No 1 Shaft	Surface to 170m	18m	234 m	Nil
No 2 Shaft	Surface to 337m	3m	26,5 m	Nil

6.7.2 Monitoring of shaft stability

The stability of the shafts will be monitored regularly by the shaft timberman. Visual inspection will be carried out of the middling between the shaft sidewall and the worked out stope sidewalls on selected levels underground. This will give prior warning of uncontrolled scaling of the open stopes which may affect the shafts. The stability of the shaft walls will be monitored by examining the rock wall conditions during shaft examinations. The manager will specify the frequency of monitoring and locations after appropriate consultation with the rock engineering practitioner. Any instability detected in the rock walls will be reported to the manager who will decide on appropriate remedial action. The results of such monitoring shall be recorded in the shaft examination log books.

6.7.3 Support of shafts

When sinking shafts, the support in the shaft walls will be designed using the techniques described above for service excavations. The manager, after appropriate consultation with the rock engineering practitioner, will be responsible to specify support for sinking shafts.

Special Areas	
• *Describe strategy to ID and deal with an increased risk of rockfalls which may develop during the course of routine mining operations.*	*Y*
• *Describe responsibility of RE in:*	
- *designing the layout*	*Y*
- *mining sequence*	*Y*
- *support*	*Y*
- *monitoring of special areas*	*Y*
• *Indicate:*	
- *where approved procedure, and any subsequent modifications for individual/specific areas are to be located*	*Y*
- *to whom copies of these instructions are to be distributed.*	*Y*

6.8 Strategies for Special Areas

Areas in which the rock conditions are such that an abnormal risk of rock falls exists shall be designated as "special areas". These areas will be subject to additional supervision and support.

6.8.1 Identification of a special area

Where rock conditions are deemed to have deteriorated or a particularly hazardous situation exists, the shift boss will report to the mine overseer and a decision will be made whether to designate an area as a "special area". Examples of special areas would be:

- Stopes or development in the Big Boy Fault zone;
- When the parting between two adjacent overlapping stopes is less than 3m.

The mine overseer will advise the manager in writing that an area has been identified as a potential special area. The manager will consider all the factors, consult with the mine overseer and will be responsible to designate a special area, within a reasonable practical time.

6.8.2 Notification of relevant personnel

The manager will inform the following persons in writing that an area has been declared a special area.

- Mine Overseer
- Shift boss
- Stope Miner
- Safety officer
- Competent Rock engineering practitioner

Once the manager has satisfied himself that conditions have reverted to normal, he shall notify the above persons in writing that the area is no longer a special area.

6.8.3 Demarcation of special areas

A prominent sign shall be erected by the miner at all entrances to a special area stating "You are now entering a special area".

6.8.4 Support and monitoring of special areas

A special area support rule for each special area must be drawn up by the Manager in consultation with the Mine Overseer and the Competent Rock Engineering Practitioner. The following aspects will be considered in designing special area support:

- Rock mass classification;
- Groundwater conditions
- Rock stress and closure rate
- Spacing and orientation of jointing;
- Loading of support units;
- Capacity of support units;
- Stope width;
- Previous instability in the area.

The competent rock engineering practitioner shall visit special areas at intervals as requested by the manager and report to the manager in writing on the safety and compliance to support instructions. The report shall be submitted within two working days of the visit.

Blast Design and Practice	
• *Describe strategy adopted to minimise blast induced damage. This must include:*	*Y*
- *methods to ensure drilling accuracy*	*Y*
- *types of explosives*	*Y*
- *method of initiation*	

6.9 Blast Design and Practice

Stringent control of blasting practices is necessary to preserve the stability of the immediate wall rock of the fissure. The objective of the blasting strategy is to minimize the blast induced fracturing and thereby reduce the possibility of rock falls. The strategy consists of the following activities:

6.9.1 Good blast design

The layout of blast holes, burden, spacing and timing will be designed by suitably qualified personnel. The blasting efficiency and damage to the surrounding rock will be monitored by the shift boss and modifications made to the design as required.

6.9.2 Control of shot holes

The direction, spacing and length of shot holes has a primary effect on blasting efficiency and blast damage to the surrounding rock.

The first step in accurate shot hole drilling is the correct marking of the holes. The miners will be suitably trained so that they adhere to the mine standards regarding burden, spacing and distance from the orebody contacts. Supervision of the correct marking of shot holes will be the responsibility of the Shift Boss.

Accurate drilling of shot holes is the second important stage in good blasting practice. Drill operators will be suitably trained to ensure that they understand the requirements for drilling holes accurately and in the correct direction. The miner will be responsible to ensure that shot holes are accurately drilled to the correct length and diameter.

6.9.3 Selection of explosives

Explosives will be used which are suitable for the rock being broken and the local ground conditions. The type of explosives will be selected so that their potential for damage to the surrounding rock is minimised. The following types of explosives will be used:

- Anfo based explosives: Used in stoping and development owing to their safety and cost effectiveness.
- Emulsion explosives: Used in poor ground and wet conditions.

The selection of appropriate explosives for variable rock conditions will be the responsibility of the Shift Boss.

Monitoring and Control	
• *Describe monitoring strategies which will ensure that:*	
- *the orebody is safely exploited;*	*Y*
- *early warning of changing conditions is communicated to responsible person.*	*Y*

6.10 Monitoring and control

6.10.1 Monitoring strategy

Monitoring of local rock conditions in stopes, tunnels and service excavations will be carried out visually by the shift supervisor of each section and will be reported in his log book on a daily basis. Changes in conditions will be reported to the mine overseer who will be responsible for implementing appropriate remedial action. If conditions are such that the mine standards do not adequately address the conditions, the mine manager must be informed who will plan appropriate action.

The regional stability of the mine will be reviewed once in six months. The mine manager in consultation with the rock engineering practitioner will check actual layouts against proposed layouts. Where deviations exist that will affect the regional stability of the mine, the mine manager in consultation with the rock engineering practitioner will be responsible for planning appropriate remedial action.

• *For each identified hazard outline the:*	
- *controls to be followed;*	*Y*
- *procedures to be followed;*	*Y*
- *the responsible person.*	*Y*
This should be presented in tabulated format.	

6.10.2 Rock related hazard management

The following table summarises the hazards identified at the mine and the control measures to alleviate the risks.

Table 9 Rock-related hazard management

Item	Hazard	Controls	Action required and Mine standards	Responsible person
1	Fall of ground in stopes/ development ends/ tunnels.	Early shift examination	Mine Standard No. 2.3.1	Team leader/miner
		Miners follow-up examination	Mine Standard No. 2.3.2	Miner
		Midshift examination	Mine Standard No. 2.3.4	Team members
		Persons not to work in unsupported area	Mine Standard No. 2.2.1	Miner
		Persons not to enter panel before installation of temporary support	Mine Standard No. 2.2.2	Miner
		Proper support installation	Mine Standard No. 2.4.2 and 6.1.3	Miner
		Good support design	Code of practice Section: 6.5 & 6.6	Manager and rock engineering practitioner
		Good blasting practice	Code of practice Section: 6.9.1	Manager
		Training of all persons responsible	On the job training	Training Officer
2	Rolling rocks in stopes, development ends, loading cross-cuts	Early shift examination	Mine Standard No. 2.3.1	Team leader/miner
		Maintain stope face angle below 40 degrees	Mine Standard No. 2.6.2	Miner
		Miners follow-up examination	Mine Standard No. 2.3.2	Miner
		Midshift examination	Mine Standard No. 2.3.4	Team members
		Training of all persons responsible	On the job training	Manager/ Training officer
3	Fall of ground in raises and ore passes	Daily examination and making safe	Mine Standard No 2.3.6	Miner
		Removal and support of loose rocks	Mine Standard No 2.3.7	Miner
		Proper barring procedures	Mine Standard No 2.3.8	Miner
4	Poor ground associated with lava country rock	See Item 1 above for general controls		
		Stope support standards for lower geotechnical area	Mine Standard No 2.4.2	Miner
5	Poor ground associated with Big Boy fault	Special area precautions	Code of Practice Section: 6.8	Manager/Mine overseer
6	Large scale collapse of workings	Monitoring and mine layout design	Code of Practice Section:6.2	Manager/rock engineering practitioner

Function of a rock engineering service	
• *Describe:*	
- *of rock engineering department/consultant*	*Y*
- *the routine input by the RE personnel in the monitoring process*	*Y*
• *Specify frequency of review of every separate working place with different risk classifications by a competent RE practitioner.*	*Y*
• *Specify frequency of visits to working places with different risk classifications by competent RE personnel.*	*Y*

6.11 The Function of A Rock Engineering Service

According to ABC Diamond Mine's risk profile, a full time rock engineering service is not required on the mine. The manager may acquire the services of a part time competent rock engineering practitioner to assist with mine layout design, support design, investigate fall of ground accidents, and review the Code of Practice annually.

The competent rock engineering practitioner shall visit the mine at least once in six months to verify compliance with the codes of practice and the mine standards. During these visits he will assess the success of the recommended support and mining sequence. He will inspect the mine plans to determine compliance with the overall mining sequence and regional support strategies. He will be required to visit all the special areas. The observations made during each visit will be recorded systematically. A report indicating such observations will be submitted to the manager within five days of the completion of such a visit.

Additional services of a competent rock engineering practitioner or consultant will be at the discretion of the employer.

6.12 Implementation of the Code of Practice

Training will be structured for the different levels of responsibilities and will include strata control and hazard recognition tuition as well as the strategies of this COP. The safety officer will be responsible for developing and implementing the training programme.

The training programme will be controlled to ensure that all the necessary persons have attended and a record of this and each person's assent to having understood their responsibilities shall be maintained. After the period of training required for the initial implementation of the COP, the programme will be continuous and will be presented to all employees on induction and re-induction. Where necessary, employees will also be sent for re-training.

6.13 Conformance to the Code of Practice

The ultimate measure of the proper implementation of and conformance to the COP will be the rock related accident rates. Monitoring and control of strategies and the level of hazard awareness exhibited by underground workers will also indicate whether conformance is being achieved. However, the manager shall appoint people with specific responsibilities for assuring conformance with particular sections/strategies of the COP.

The safety officer will monitor the implementation of and conformance to this COP. In addition, an annual audit will be carried out by a suitably qualified rock engineering practitioner. This will be done prior to the review of the COP.

7 REFERENCES

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APPENDIX 1 : MOTIVATION OF SUPPORT DESIGN METHODOLOGY

1. Background

A vertical orebody has special support requirements not addressed by the support methodology for shallow dipping orebodies proposed in the guidelines for preparing a code of practice. In a near vertical orebody, fall of ground occur from the stope sidewalls and from the stope face. In the case of an overhanging stope face, the support should be able to carry the full weight of the potential rock falls. In the case of sidewall instability, the support does not have to carry the full weight of the loose rocks in the stope sidewalls. The support merely has to hold these blocks in position, so that they do not topple or slide into the stope excavations, as shown in Figure 1-1.

2. Local conditions at ABC mine

At ABC Mine the conditions of the immediate contact between the kimberlite fissure and the country rock is usually poor. The country rock often contains fracturing to a depth of 30cm to 100cm into the sidewall, called the “deformed zone”. The deformed zone is expected to be the main origin of rock falls in the stopes, since the poor sidewalls may loosen when blasting is done. The fracturing in the deformed zone is parallel to the fissure and results in thin slabs of rock in the immediate sidewalls of the stope. These slabs may fail by toppling into the stope excavation. A support system is therefore required to hold these slabs in position so that they do not topple into the stope.

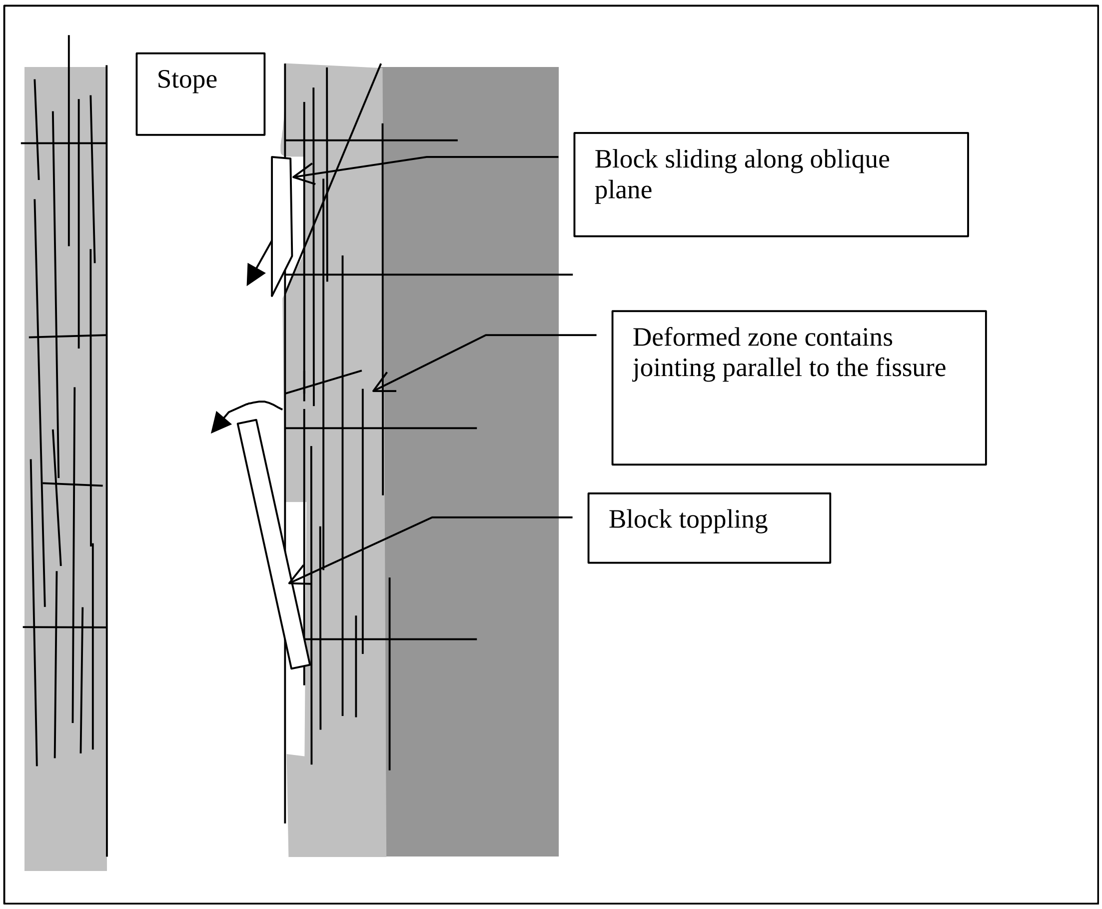

Figure 1-1 Schematic section showing mode of failure in sidewalls of steep stopes

The sliding mode of failure is possible if some of the discontinuities are inclined towards the stope, allowing slabs to slide into the stope. At ABC mine, most of the discontinuities are sympathetic to the fissure and are steeply dipping. Bedding joints exist which dip at about 10^0. Sliding cannot take place along the bedding joints. The existence of joints inclined at between 30^0 and say 80^0 cannot be ruled out and the support system should be designed to accommodate sliding.

3. Support resistance to prevent toppling

Toppling may occur if the joints form long slender slabs parallel to the fissure sides and they are terminated by near horizontal bedding planes. If the stope overhangs slightly, the slabs may tend to topple. This is shown in Figure 1-2.

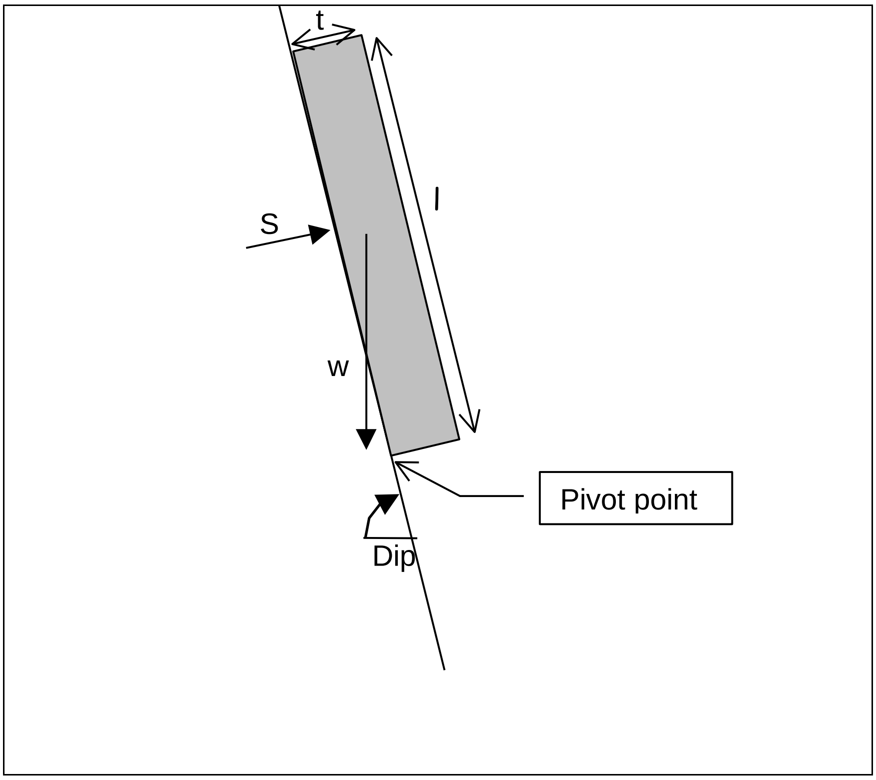

Figure 1-2 Sketch showing toppling failure mode

For example, if the slab in Figure 1-2 is 1m thick and 9m long by 3m in the third dimension, it will have a weight of 715 kN, taking the density as 2700 kg/m^3. Toppling will become possible when the stope dip is less than 83,6^0. Assume the stope dips at 80^0, the lever arm of the weight, causing rotation of the block about the pivot point will only be 0,29m. Assume further that the support force acts through the centroid of the slab. This means that the lever arm of the support will be 4,5m about the pivot point. The support force required to prevent toppling of the block is only: 4,5/0,29 = 0,06 times the weight of the slab. This means that a force of 46,6 kN is sufficient to hold the slab in position. The face area of the slab is 27m^2, resulting in a required support resistance of 1,7 kN/m^2.

In practice such large slabs will be unlikely to be formed since the bedding joints are less than 9m apart. However, the calculation illustrates that very small support resistance magnitudes are required to prevent toppling of slabs.

For design purposes, assume that slabs will be no more than 3m high and the stope dip is 80^0. Toppling will only be possible if the slab thickness is less than 0,26m. The calculated support resistance to prevent toppling is shown in Figure 1-3. It can be seen that when the slab is approximately 15 cm thick the required support resistance is a maximum at 0,3 kN/m^2.

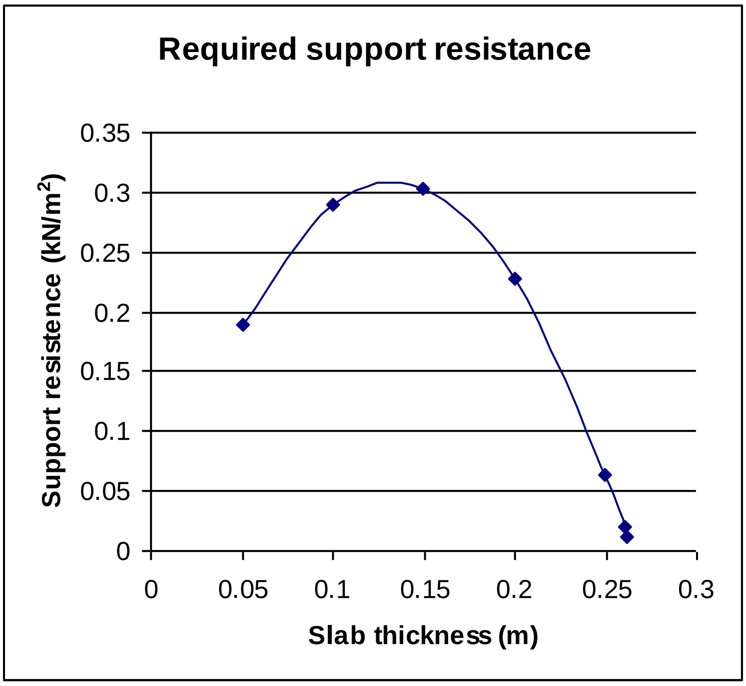

Figure 1-3 Graph showing the support resistance required to prevent toppling of 3m high slabs in a stope with an overhanging sidewall at 80^0

The above arguments show that toppling may easily be prevented by the provision of small lateral support to the slabs. Although the calculations show that a support resistance of only 0,3kN/m^2 is sufficient, the design of sidewall support will be based on a requirement of 1kN/m^2.

4. Support resistance to prevent sliding

If a slab of rock in the stope sidewall is truncated by an oblique plane, the slab may fail by sliding along this plane. The mode of failure is illustrated in Figure 1-4 below. If it is assumed that the full weight of the slab rests on the sliding plane, the sliding force and required support to prevent sliding may be calculated as follows:

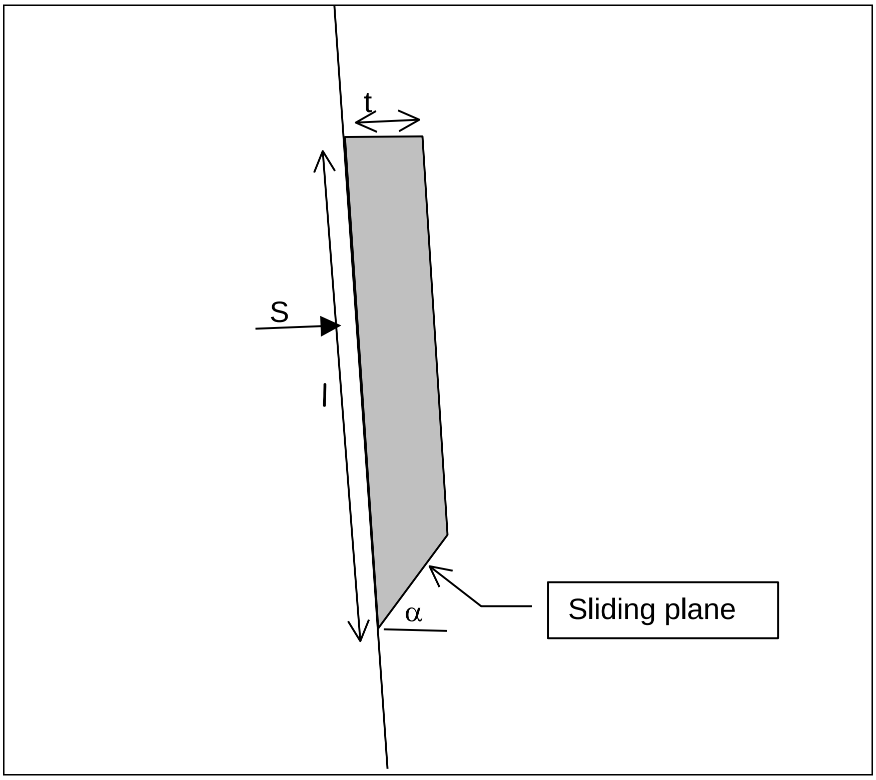

Figure 1-4 Sketch showing sliding mode of failure.

Forces in direction of sliding:

$$F_s = w\sin\alpha - S\cos\alpha$$

Forces resisting sliding:

$$F_r = (w\cos\alpha + S\sin\alpha)\tan(\phi)$$

Where ϕ is the friction angle along the sliding surface. The slab will be stable when the sliding forces and resisting forces are equal. The required support resistance may be calculated by considering a slab 1m thick and 3m high. A representative friction angle of 30^0 was used in the calculations. The dip of the sliding angle was varied. Note that as the dip of the sliding plane changes, the shape and volume of the slab also changes. The results are presented in Figure 1-5, where it can be seen that a support resistance of approximately 13 kN/m^2 is required to hold the slabs in position.

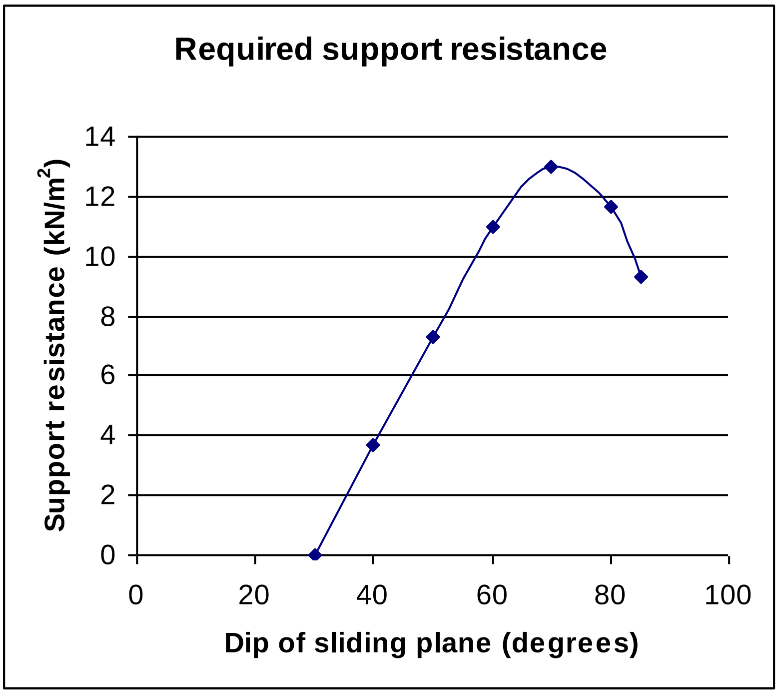

Figure 1-5 Graph showing required support resistance to prevent sliding of a 1m thick slab, 3m high, for different orientations of the sliding plane

For the purpose of support layout design, a support resistance of 20 kN/m^2 will be used.

5. Support resistance to prevent falls from the overhanging stope face

In the case of the stope face, unstable blocks of rock will tend to fall vertically downwards under the force of gravity. The support will therefore be required to carry the full weight of unstable blocks. The required support resistance is calculated by assuming that the stope sidewalls do not contribute to the stability of the block. The required support resistance is given by:

$$R = \rho g t$$

where ρ is the density of the rock in kg/m^3, g is gravity acceleration and t is the thickness of the potential rock fall.

APPENDIX 2 : GLOSSARY OF TERMS AND DEFINITIONS

Active Support:	Any support that is pre-stressed & loaded during installation that maintains or increases the load during its lifetime - hydraulic props are active support units.
Adit:	A horizontal opening, started from a hillside, to reach an orebody.
Back:	This is the orebody between a level and the surface, or between two levels.
Breast Panel:	Stope faces which advance at an angle approximately 90^0 to dip.
Burden:	Distance between an explosive charge and the free surface in the direction of throw.
Closure:	Reduction of the distance between two basically parallel surfaces (usually the hanging wall and the footwall). This includes both the elastic and inelastic movement of the rock mass.
Compressive stress:	Normal stress tending to shorten the body in the direction in which it acts
Controlled blasting:	All forms of blasting designed to preserve the integrity of the remaining rocks (e.g. smooth blasting, pre-splitting post-splitting).
Convergence:	Reduction of the distance between two basically parallel surfaces (usually the hanging wall and the footwall)
Creep:	Time dependent deformation,
Cross-cut:	A horizontal opening, like a tunnel that cuts the rock formation at an angle to the strike in order to reach an orebody.
Crush pillar:	Any portion rock left in situ, on the reef plane, with a width to height ratio of < 3.
Decoupling:	Ratio of the radius of a blast hole to the radius of the charge; this causes a reduction in the amplitude of the strain wave by increasing the space between the charge and the blasthole wall.

Deformation:	A change in shape or size of a solid body.
Dilatancy:	The property of volume increase under loading
Dip:	Angle at which a stratum or other planar feature is inclined from the horizontal
Discontinuity surface:	Any surface across which some property of a rockmass is discontinuous (e.g. bedding planes, fractures).
Drive:	A horizontal opening, like a tunnel, lying in or near the orebody, parallel to the strike.
Dykes:	Linear features which were pushed into pre-existing cracks within the rock mass. Dykes can be associated with shear zones, high stress; water or zones of decomposed low strength material.
Elasticity:	Property of a material whereby it returns to its original form or condition after an applied force is removed.
Fault:	Any geological discontinuity along which relative displacement can be measured. Common faults are either reverse - gain of ground fault on normal - loss of ground faults.
Fissure:	Near vertical ore bearing kimberlite intrusion
Fissure drive:	Tunnel in the fissure
Footwall:	Mass of rock beneath a discontinuity surface (in tabular mining, the rock below the reef plane).
Force:	An action that tries to move an object from a stationery position, or to change its rate of movement or its direction of movement.
Geological Features:	All major geological features or discontinuities, i.e. those having one or more of the following properties. a. A previous history of falls of ground or resulting in adverse ground conditions. This includes brows. b. If water is dripping/seeping from the geological discontinuity. c. At the intersection of any faults, dykes or shear zones.

	d. If the discontinuity contains any infilling greater than 3 mm wide.
	e. A feature that is the opinion of the supervisor, unsafe.
Geotechnical area:	A portion of a mine where similar geological conditions exist which give rise to a unique set of identifiable rock-related hazards for which a common set of strategies can be employed to minimise the risk resulting from mining.
Gully:	An excavation cut in the immediate footwall or hanging wall of the reef for the purpose of enabling the removal of rock from the face or providing access to the face for men or material.
Hangingwall:	Mass of rock above a discontinuity surface (in tabular mining, the rock above the reef plane).
Inelastic deformation:	The portion of deformation under stress that is not annulled by the removal of the stress.
Level:	All openings at a horizon from which the orebody is opened up and mining is started.
Normal Conditions:	Normal conditions refer to ground conditions where virtually no falls of ground occur after the installation of the minimum support requirements.
Normal force:	Force directed normal (perpendicular) to the surface element across which it acts.
Normal stress:	Component of stress normal to the plane on which it acts.
Over break:	The quantity of rock that is removed beyond the planned perimeter of the final excavation.
Passive Support:	Any support that required external loading either through lateral or vertical displacement. E.g. timber sticks, grout packs, timber packs, steel or timber sets, wire meshing and lacing.
Permanent support	Support that once installed is not removed.
Pillar:	Rock left in situ during the mining process to support the local hanging wall, roof or to provide stability to the mine or portion thereof.
Plasticity:	State in which material continues to deform indefinitely whilst sustaining a constant stress.

Poisson's ratio:	Ration of shortening in the transverse direction to elongation in the direction of an applied force in a body under tension below the proportional limit
Primitive (virgin) stress:	State of stress in a geological formation before it is disturbed by manmade operations.
Principal stress (strain):	Stress (or strain) normal to one of three mutually perpendicular planes or which the shear stress (or strain) at the point in the body is zero.
Raise:	Any tunnel having an inclination above horizontal in the direction of the working of more than 5 degrees (but not included under the definition of a shaft).
Regular review:	Assessment of the conditions of an area through discussions, plan critique, planning meetings and *I* or underground visits.
Remnant:	Any reef area isolated on at least three sides by previous mining. An area less than 10 000 m^2 in size that is not part of current scattered mining stoping is treated as a "Special Area".
Rock:	Any naturally formed aggregate of mineral matter occurring in large masses or fragments.
Rockburst:	Seismic event that causes damage to underground workings.
Rock drive:	Tunnel parallel to fissure in country rock
Rockfall:	Fall of a rock fragment or a portion of fractured rock mass without the simultaneous occurrence of a seismic event.
Rockmass:	Rock as it occurs in situ, including its discontinuities.
Rock Engineering Consultant:	A Professional Engineer or a Professional Natural Scientist specialising in Rock Engineering and practicing, or a graduate possessing the Chamber of Mines Certificate in Advanced Rock Engineering who has sufficient experience of rock engineering practice in the industry that he is able to advise management on strategic decisions that affect the industry and has sufficient theoretical knowledge to be able to understand and implement new research findings within the industry.
Rockbolt:	A steel bar placed in a hole to reinforce the strata. The steel bar is bonded to the rock by means of full column cement grout or a mechanical anchor.

Scattered Mining:	A system of reef extraction that advances in more that one direction simultaneously. The mining is initiated from a central raise/winze excavation and stope faces mine outwards advancing faces from an adjacent raise/winze connection: i.e. multiple mining operations on the same level.
Seismic event:	Transient earth motion caused by a sudden release of the energy stored in the rock.
Shaft:	Any tunnel having a cross sectional dimension of 337 m^2 or over and (I) having an inclination to the horizontal of 15 degrees or over, or (ii) having an inclination to the horizontal of less than 15 degrees but more than 10 degrees where the sped of traction exceeds 2 m/s
Shear Zones:	Areas where parallel or sub-parallel slips occur, in close proximity, with little or no cohesion along interfaces.
Slip:	Any geological discontinuity, which divides the rock mass into separate blocks, without relative displacement across the feature.
Solid Pillar:	Any portion of rock left in situ, with the width > 10 x stope width.
Spalling:	Longitudinal splitting in uniaxial compression, or the breaking-off of plate-like pieces from a free rock surface.
Special areas:	During the course of routine mining an increased risk of rockfalls may develop. Such areas requiring additional attention and precautions must be designated special areas.
Spitting:	Violent ejection of splinters of rock from the surface of an excavation.
Stick:	Unturned round timber pole with a minimum diameter of 0.15m which is not pre-stressed.
Stiffness:	Ratio of force versus displacement
Strain burst:	Rockburst at the lower end of the spectrum of violent events occurring essentially at the surface of an excavation.
Strength:	The maximum stress that a material can resist without failing for any given loading-regime.

Stress:	Force acting across a surface element divided by the area of the element.
Strike:	Direction of the azimuth of a horizontal fine in the plane of an inclined stratum (or other planar feature) within a rockmass.
Stope:	An underground excavation made in removal of any ground or mineral other than coal, but does not apply to excavations made for engine rooms and pump chambers or for development purposes such as shafts drives, winzes and raises.
Suitably qualified rock engineering practitioner:	A person who is at least in possession of the Chamber of Mines Certificate in Rock Mechanics.
Suitably trained Personnel:	A person trained in relevant rock engineering/strata control competencies.
Subsidence:	Downward movement of the overburden (soil and/or rock) lying above an underground excavation or adjoining a surface excavation.
Support:	A structure or a structural feature built into or around an underground excavation to maintain its stability.
Support pillar:	Any portion of rock left in situ is a support pillar. There are several types of support pillars, namely crush pillars, barrier pillars, shaft pillars; water projection pillars; ventilation pillars etc.
Swelling:	Constitutive mineralogical nature of the rock by which water is absorbed, causing a measurable increase in volume; swelling can exert very large time dependent forces on rock or support systems and reduce the size of excavations.
Temporary Mechanical Prop:	A prop composed of an inner and outer steel sleeve for easy transportation. The mechanical prop is installed by sliding out the inner tube from the outer tune so that the ends of the steel sleeves are wedged by means of a cam between the hanging wall and footwall. The prop must be equipped with a remote release tool.
Temporary support:	Support which will be removed.
Tensile stress:	Normal stress tending to lengthen a body along the direction in which it acts.

Thickness:	Perpendicular distance between bounding surfaces (e.g. bedding planes).
True dip:	Two points joined, by a straight line, to give the maximum elevation difference along the some plane, will be parallel to the true dip direction.
Updip Panels:	Stope panels that advance, parallel or near parallel to the dip direction, at a positive inclination.
Underhand mining:	Mining method in near vertical fissure which results in the advancing face forming the floor of a stope
Versit:	Horizontal or vertical dislocation of fissure
Weathering:	Process of disintegration and decomposition as a consequence of exposure to the atmosphere, to chemical action, and to the action of frost, water and heat.
Working place:	The place where mine workers normally work or travel.