Appendix A Pro forma rock mass and hazard rating system

A.1 Support capacity of hangingwall rock mass and installed support

The ability of the rock mass surrounding an excavation to remain stable depends on the capacity of the hangingwall rock mass AND the capacity of the support installed in the excavation to accommodate the imposed loading conditions.

A.1.1 Support capacity of hangingwall rock mass

The support capacity of the hangingwall rock mass could be affected by factors such as the geology of the rock mass, drilling and blasting, lateral confinement of the rock mass and beam geometry.

Effect of geology on strength of hangingwall rock

Main Category	Description				Possible Rating	Rating
Intact strength of hangingwall	Competent host rock only				4	
	Weak geological intrusions				*1	
	Thin (less than 30 cm thick), disseminated plates/beams				*1	
Weathering of rock	No signs of weathering				4	
	Rock weathers with time				2	
Geological structure	Large geological structures	No major faults, dykes or shear zones present			4	
		Description of fault, dyke or shear zone	Dip of major structures	Steep dipping > 60°	3	
				45° to 60°	2	
				Flat dipping < 45°	*1	
			Strike of major structure	Approx. ⊥ to pillars	4	
				Approx. 45° to pillars	2	
				Approx. ∥ to pillars	*1	
			Strength of structure	No infilling, tight	4	
				Weak with infilling	2	
	Parting planes in immediate hangingwall	No parting planes less than 2,0 m in hangingwall			4	
		Description of parting planes	Thickness of layers caused by parting planes	> 1,0 m	3	
				0,3 – 1,0 m	2	
				< 0,3 m	*1	
			Strength of parting planes	"Welded"	3	
				Tight, no infilling	2	
				Weak with infilling	*1	
	Jointing	Dip of joints	Only steep dipping joints (> 60°)		4	
			45° - 60°		2	
			Some flat dipping joints or domes (< 45°)		*1	
		Joint spacing	Joints spaced more than 1,0 m apart		3	
			Joints spaced less than 1,0 m apart		2	
		Joint filling	None		4	
			Sheared infill material		2	
		Joint surface	Stepped		4	
			Undulating		3	
			Smooth planer		2	
Groundwater conditions	No groundwater present or anticipated				4	
	Groundwater present or anticipated				*1	
TOTAL – EFFECT OF GEOLOGY ON CAPACITY OF HANGINGWALL ROCK						

Effect of drilling and blasting on capacity of hangingwall rock

				Possible Rating	Rating
Application of drilling and blasting	Drilling of blastholes	Length of shotholes	Optimum hole length being drilled	4	
			Shotholes drilled too long	2	
		Direction of shotholes	Optimum direction	4	
			Incorrect hole direction	2	
		Spacing of shotholes	Optimum borehole spacing	4	
			Holes spaced too far / too close apart	2	
	Charging of blastholes	Optimum charge length		4	
		Holes over / under charged		2	
	Timing of blastholes	Correct timing		4	
		Incorrect timing leading to out-of-sequence firing		2	
TOTAL – EFFECT OF DRILLING AND BLASTING ON CAPACITY OF HANGINGWALL ROCK					

Effect of boundary conditions on capacity of hangingwall rock

Horizon control	No brows or overhangs created	4	
	Brows or overhangs created	*1	
Depth below surface	More than 100 m deep	4	
	50 – 100 m deep	2	
	Less than 50 m deep	*1	
Width of excavation	Less than 15 m	4	
	15 – 20 m	3	
	20 – 25 m	2	
	More than 25 m	*1	
TOTAL – EFFECT OF CONFINEMENT ON CAPACITY OF HANGINGWALL ROCK			

Effect of beam geometry on capacity of hangingwall rock

Ratio of beam length to beam thickness	Less than 20:1	4	
	20:1 to 25:1	3	
	25:1 to 30:1	2	
	More than 30:1	*1	
TOTAL – EFFECT OF BEAM GEOMETRY ON CAPACITY OF HANGINGWALL ROCK			

Example: Beam length = 28 m and beam thickness = 0,8 m
Ratio of beam length to beam thickness = (28 ÷ 0,8):1 = 35:1

All panels with sub-categories rated as *1 should be declared as "Special Areas"

All panels with sub-categories rated as 2 should be declared as "Moderate Areas"

All panels with sub-categories rated as 3 or 4 should be declared as "Good Areas"

A.1.2 Capacity of installed support

The capacity of installed support could be affected by factors such as the standard of support installation, the ability to assess ground conditions correctly, the removal / damage / loosening of support after installation and the engineering properties of the support elements.

Standard of support installation

Support installation procedure	Knowledge about support installation procedure	Description of support installation procedure	Procedure described in mine standards	4	
			Procedure not described in mine standards	*1	
		Communication of procedure	Procedure communicated to workers responsible for support installation	4	
			Procedure not properly communicated	*1	
		Competence of workers responsible for support installations	Workers properly trained and found to be competent in support installation	4	
			Workers incompetent to install support	*1	
	Support installation equipment	Supply of installation equipment (e.g. special airleg, impact wrench, etc.)	Necessary equipment available to install support correctly	4	
			Support installation equipment inadequate	*1	
		Application of support installation equipment	Equipment used correctly	4	
			Equipment not used correctly	*1	
	Supervision of support installations	Supervision during support installation of a high standard		4	
		Inadequate supervision		*1	
Support installation sequence	Permanent support	Permanent support installed less than 1,0 m from the face before shotholes are drilled		4	
		Permanent support installed less than 2,0 m from the face before shotholes are drilled		3	
		Permanent support installed more than 2,0 m from the face before shotholes are drilled		*1	
	Temporary support	At least 2 mechanical props installed not more than 1,0m on either side of machine operator during drilling operations		4	
		Temporary support not to standard during drilling operation		*1	
Support type	Support type being used is according to standard			4	
	Support type being used is not according to standard			*1	
Support pattern	Support pattern for normal ground conditions	Support installed according to mine standard		4	
		Support not installed according to mine standard		*1	
	Support pattern for abnormal conditions	Additional support installed as required by mine standard for abnormal ground conditions		4	
		Additional support not installed as required by mine standards for abnormal conditions		*1	
Quality of support material	Quality of support material being used adequate			4	
	Quality of support being used inadequate			*1	
Support length	Length of support being used is according to mine standard			4	
	Length of support being used not according to mine standard			*1	
Support inclination	Support installed approximately ⊥ with orientation of hangingwall or weakness plane (rock studs at least 60 degrees)			4	
	Inclination of support too flat (rockstuds less than 60 degrees)			*1	
Support diameter or strength	Diameter or strength of support being used according to mine standard			4	
	Diameter or strength of support being used not according to mine standard			*1	
TOTAL – STANDARD OF SUPPORT WORK AND THE EFFECT ON SUPPORT CAPACITY					

Assessment of ground conditions

Information regarding quality of hangingwall rock mass	Sufficient information available to assess quality of hangingwall rock mass	4	
	Lack of information regarding quality of hangingwall rock mass	2	
Supervision regarding assessment of ground conditions	Supervision regarding assessment of ground conditions generally of a high standard	4	
	Supervision not to standard	*1	
Competence of workers	Workers properly trained in the identification of hazardous rock conditions, and the support required for different ground conditions	4	
	Workers not properly trained in the identification of hazardous rock conditions – apply standard support to all rock conditions	*1	
Attitude of workers	Workers assess ground conditions continuously and install additional support where required	4	
	Workers tend to ignore changing ground conditions	*1	
TOTAL – ASSESSMENT OF GROUND CONDITIONS AND THE EFFECT ON SUPPORT CAPACITY			

Removal / damage / loosening of support after installation

Permanent support	Rockstud support	Rockstuds damaged / loosened by blasting	Rockstuds do not loosen during blasting - properly tensioned and thread not protruding more than 2 cm	4	
			Some rockstuds tend to loosen during blast	*1	
	Mine poles or elongates	Support removed by blasting	Support removed during blast	4	
			Some support removed during blasting	*1	
		Support removed by scraper	Support not removed by scraper	4	
			Some support removed by scraper	*1	
Temporary support	Support removed by means of a remote release tool and from a well supported and safe area			4	
	Support removed without using a remote release tool			1	
TOTAL – REMOVAL / DAMAGE / LOOSENING OF SUPPORT AFTER INSTALLATION AND THE EFFECT ON SUPPORT CAPACITY					

A.2 Loading of hangingwall rock mass and support units

Excessive loading of the hangingwall rock mass surrounding an underground excavation and installed support could lead to instability when the loading exceeds the capacity of the rock mass and installed support. It is therefore important to maximise the capacity (strength) of the hangingwall rock mass and installed support and to reduce the loading of these systems.

A.2.1 Loading of hangingwall rock mass

Loading of the hangingwall rock mass could adversely be affected by a reduction in stress causing potentially loose blocks of ground to dislodge. Potentially unstable blocks of ground could also dislodge due to mining induced disturbance of the hangingwall (e.g. during installation / removal of temporary support).

Loading of hangingwall rock mass due to abnormal hangingwall stress

Excavation span	Less than 15 m	4	
	15 – 20 m	3	
	20 – 25 m	2	
	More than 25 m	*1	
Proximity of excavation to surface	Excavation more than 100 m below surface	4	
	Excavation between 50 and 100 m below surface	2	
	Excavation less than 50 m below surface	*1	
Freedom of movement	Continuous beam with no brows or overhangs	4	
	Freedom of movement due to brows or overhangs	*1	
TOTAL – LOADING OF SUPPORT SYSTEM DUE TO DECREASE IN HANGINGWALL STRESS			

Loading of the hangingwall due to a mining induced disturbance

Installation / removal of temporary support	Probability of disturbing the hangingwall during installation of temporary support **low**.	4	
	Probability of disturbing the hangingwall during installation of temporary support **high**.	*1	
Installation of services	Probability of disturbing the hangingwall during installation of services **low**.	4	
	Probability of disturbing the hangingwall during installation of services **high**	*1	
Installation of permanent support	Probability of disturbing the hangingwall during installation of permanent support **low**.	4	
	Probability of disturbing the hangingwall during installation of permanent support **high**.	*1	
Mechanical impact	Probability of disturbing the hangingwall due to mechanical impact from scraper **low**	4	
	Probability of disturbing the hangingwall due to mechanical impact from scraper **high**	*1	
TOTAL – LOADING OF HANGINGWALL DUE TO MINING INDUCED DISTURBANCE			

A.2.2 Loading of hangingwall support units

The loading of support units could adversely be affected by increasing the spacing between support units and the last row of permanent support and the face.

Loading of support units due to spacing between units

Spacing between support units	Support spaced closer than required by mine standard	4	
	Support spaced further apart than required by mine standard	*1	
Spacing between last row of permanent support and face	Permanent support to face distance less than 2,0 m after the blast	4	
	Permanent support to face distance less than 3,0 m after the blast	3	
	Permanent support to face distance less than 4,0 m after the blast	2	
	Permanent support to face distance more than 4,0 m after the blast	*1	
TOTAL – EFFECT OF SUPPORT SPACING ON LOADING OF SUPPORT UNITS			

Pillar stability

The stability of stope pillars depends on the capacity or strength of the pillar material to support the load imposed on the pillars. Pillar instability could occur if one or more of the factors discussed below changes significantly.

Pillar loading

Depth below surface	Pillar sizes and spacing being used correspond with the current depth of mining	4	
	Depth of mining has changed significantly without increasing the pillar sizes or reducing the pillar spacing accordingly. (possibly caused by rapid change in topography)	*1	
Spacing between pillars	Spacing between pillars are according to mine standard	4	
	Pillar spacing greater than permitted by mine standard	*1	
TOTAL – EFFECT OF PILLAR LOADING ON PILLAR STABILITY			

Pillar strength

Pillar material strength	Pillar strength not affected by adverse geology in seam, footwall or hangingwall rock mass	4	
	Pillar strength lower that normal due to adverse geology in pillar seam, footwall or hangingwall.	*1	
Pillar width and length	Pillar width and length in accordance with mine standard for the depth being mined at.	4	
	Pillar width and length less than required by mine standard (possibly caused by pillar scaling, pillar robbing, mining off-line, etc.)	*1	
Pillar height	Stoping width in accordance with mine standard	4	
	Stoping width more than described in mine standard	*1	
TOTAL – EFFECT OF PILLAR STRENGTH ON PILLAR STABILITY			

Appendix B Questionnaire

QUESTIONNAIRE

Investigation of Factors Governing the Stability of Stope Panels

1. General Information:	
Name of Mine	Black Mountain Mineral and Development Co. (Pty) Ltd.
Postal address	Private Bag X01, Aggeneys, 8893
Tel No.	(054) 983 2571
Fax No.	(054) 983 2382
E-mail address.	
Date of SRK visit	30, 31 March 1998
Contact people: - Manager; - Production Manager; - Underground Manager; - Senior RM Eng.; - Chief Geologist; - Senior Planning Off.	 LG van Biljon M (Mike) McLaren PS(Peter) Westcott R (Rocco) Human JE (Pottie) Potgieter Danie Grobler
Location of mine	The Broken Hill (Cu, Pb, Zn and Ag) deposit is situated on the farm Aggeneys, between Springbok and Pofadder in the Northern Cape Province.
Commodities mined	Copper, silver, lead, zinc.
Datum elevation	
Depth of mining activities: - opencast; - underground.	 627m below shaft collar maximum (21 level). Current mining between 1 level (53m above shaft collar) and 21 level. Maximum exploration depth is 800m.
Describe access to the mine.	A vertical hoisting shaft, an access decline and a conveyor sub-decline below shaft bottom which derves the flatter part of the orebody.

2. Copies of Documents / Plans Required:	
Mine's COP to combat rockfall accidents.	
Mine standards.	
Stope plan (total mine)	
Plans and sections of mine's geology, including structure and stratigraphy.	
Contour plan and section of surface topography (natural and man-made).	
Copies of technical reports on rockmass description, geotechnical parameters, etc,	
Plan showing different geotechnical areas.	
FOG accident reports over last 10 years.	
Plan showing location of FOG accidents and incidents over last 10 years.	
Reports on instability problems in stopes such as pillar collapses, back breaks, etc.	

3. FOG Accident Statistics:	
No. of workers injured/year due to FOG accidents (minor, reportable and fatal) for last 10 years.	
Reportable Injury Frequency Rate for last 10 years.	
Fatal Injury Frequency Rate for last 10 years.	
Are accidents properly investigated and root causes identified?	
What are the major causes of FOG accidents?	
Describe typical dimensions of FOG's: - length; - width; - thickness.	
Is the mine's COP to reduce FOG accidents based on a baseline risk assessment?	Rockfall hazards are associated with rock types: - Competent Non-schist formations
Is the COP compiled such as to reduce the risk of rockfall accidents?	
What is the level of rockfall hazard awareness?	
Are PTO's , CTI's or other techniques being used as part of the mine's continuous rock-related risk assessment?	
What is the level of strata control and rock mechanics knowledge on the mine?	
What is being done to improve the current level of rock mechanics on the mine?	

4. **Mining Method:**	
Short description of method/s	*Blasthole open stoping (BHOS)* was employed in the early years in the steeply dipping, wide, high grade zone of the orebody. This low cost stoping method, combined with the higher grade of ore, favoured more rapid returns on capital. Stopes were 24m wide, leaving 20m wide pillars in between. These pillars were then extracted after backfilling of the adjacent stopes. Drill drives were spaced 35m vertically. Ore was blasted into collecting troughs.
	Cut and Fill (CAF) stoping was initially employed in ore of reasonable width, with breast faces generally 10 to 20m wide. Lifts of 4m were advanced over a 1m undercaut between the previous back and the fill floor. Access to the stopes was by ramp crosscuts, which were slashed down for each successive lift. CAF mining is very selective and breast faces were advanced under full geological control, the boundaries between ore and waste being marked off daily by the mine geologist. Stopes were silled on alternate main levels, thus at 70m vertical intervals. As mining progressed, the available ore became narrower and more severly affected by folding. Also, the footwall conditions had deteriorated in certain areas. These changing conditions necessitated various modifications to the original CAF layouts.
	Ramp in Stope (RIS): This is a variation on CAF. Waste development is largely eliminated. The access ramp is developed in the orebody itself as part of the stoping operation. A stoping block is extracted in 2 phases: the underhand phase during which the ramp is established, followed by the overhand phase. Initially, access to the orebody is via a development crosscut or drive on the sill elevation at one end of the proposed stope. A 5m high sill drive is developed along strike of the stope and then silled out to the full width of the orebody. Once the fill drainage system has been established, the ramp is started by building of a waste rock pile across the width of the stope to a height of 4m, and 20m from the proposed start point of the ramp to allow the required gradient of 1:5. Stoping of the next lift then starts on incline up the ramp until such time as the back is 4m above the previous lift back. Then, a 4m high breast is advanced horizontally through the remainder of the underhand lift. The sequence is repeated until the level above is reached, at which time a complete ramp is available within the stope. Stoping of the overhand side of the ramp can now start from the bottom upwards. With the ramp used as a lower access, the remainder of the lift at the sill elevation is silled out. After eastablishing a ventilation

	raise and drainage facilities as before, the lower access is closed off by backfilling. After the overhand sill has been filled, conventional breasting of the overhand section is conducted in 4m lifts.
	Crown Ramp in Stope (CRIS): Is employed where the orebody has a long strike length in poor host rock. This creates multiple stopes on strike, with access to all the stopes within the orebody itself. The continued access is provided by a crown drive protected by a crown pillar. The crown drive is developed along the footwall of the orebody on each main level elevation. A 4m thick crown pillar is left permanently above the crown drive. Short ramps are developed up to sill elevation for each individual stoping block.
	Bench and Fill (BAF): This method has been introduced into one of the zones where folding has created a thicker, flat dipping orebody. Ore was silled out at the top and bottom of a 20m strike block, 15m wide. Following the installation of 15m cable bolts in the back of the upper sill, a slot raise was bored and 165mm blastholes were employed to blast the ore as a bench. Cleaning is by remote controlled LHD (collecting cones could have been considered as well). Once the stope back has been mined out, the void will be filled with 20:1 cemented backfill, allowing the adjacent 20m block to be mined.
	Scraper Stoping: This method was introduced in a flat dipping part of the orebody where the width is between 1 and 2m thick. This is too narrow for standard mechanized equipment and would cause excessive dilution. The layout consists of stopes extending 20m on strike, separated by 5m dip pillars. The stopes extended on dip between ore drives on successive levels 35m vertically apart. 5m wide strike pillars were also left to protect the ore drives. 2 stope raises were developed on each side of alternate dip pillars, from which stope faces were advanced using hand drilling and scraper cleaning to the intermediate pillars. Holings were made at regular intervals along the lower drive pillar to allow the ore to be scraped into the drive, where the loadinjg was done by LHD. The competency of the hangingwall is critical to the success of the stoping method.
Reason for using this method/s	
Which other methods could be considered?	
Describe exploration drilling (intervals, spacing, etc.)	CAF: Initial stope layouts are based on diamond drilling at 25m intervals
Are these holes used for geotechnical purposes as well?	

Average % extraction of different methods	
Average % dilution of different methods	
Reasons for dilution	
Total tons produced per year over last 10 years.	1,5 Mt pa from Broken Hill.
Description of mining sequence/cycle:	
Drilling - equipment - methods used - length of holes - diameter - spacing	CAF: electo-hydraulic rigs BHOS: between drill drives spaced 35m vertically. BHOS: 165mm
Blasting - explosive types; - charge / hole; - initiation; - detonation; - average face advance per blast; - average tons produced per blast; - extent of damage.	CAF: 1000t /m of working face.
Cleaning - method - equipment	BHOS: Ore blasted into collecting troughs, from where it was transported by LHD's to the ore passes. CAF: LHD's into 25t trucks

5. Geology:	
Description of Stratigraphy.	At Broken Hill, the stratigraphy is reversed due to folding.
Description of structure.	Four phases of deformation (F1 - F4) have been identified. The F2 fold phase often duplicates the ore horizons. The F3 phase of deformation resulted in alarge open fold structure which is responsible for the change in the diip of the orebodies from 60 to 20 degrees. Shearing and pegmatite intrusions are often associated with the F3 deformation. Movement along F4 associated fractures and faults is right lateral with minor displacements. In the footwall (10 - 34m) of the LOB, a concordant 2-10m wide graphitic-clay schist, known as the Weak Zone, is present. This is an early thrust fault.
Description of orebodies mined.	Ore is mined from 2 superimposed mineralised horizons, known as the Upper and Lower Orebody (UOB and LOB) respectively. Both orebodies comprise a well mineralised massive sulphide core, enveloped by iron formation containing disseminated sulphides. The UOB varies from 2 - 30m in thickness and mineralisation may extend up to 5m into the adjacent iron formation. The LOB is 1 - 15m, but the surrounding iron formations are more extensively mineralised. The economic horizons of the UOB are predominantly massive sulphide, magnetic quartzite and magnetic amphibolite, with magnetic quartzite comprising the hangingwall and schist or massive magnetite the footwall. The economic horizons of the LOB are predominantly massive sulphide, magnetic quartzite, magnetite amphibolite, garnet magnetite and sulphidic quartzite, with schist, pegmatite or massive magnetite comprising the hangingwall.
Main ore minerals.	Galena, sphalerite, chalcopyrite
Geometry of orebody: - dip length; - strike length; - thickness.	 UOB: 1000m; LOB: 600m Strike length decreases with depth. Mineable width is 5 to 50m, but economic factors play an important role in defining the mineable width.
Dip / plunge: - angle; - direction.	 20 - 60 degrees. The dip varies from almost vertical in well defined steeply folded zones to almost flat in some sections E-NE
Strike orientation	Generally striking E-W

Description of ore deposit and host rock	Consists of 3 major mineralized zones namely Broken Hill, Black Mountain and.... The Broken Hill ore resources are contained in 2 conformable orebodies seperated by a 5 to 30m wide intermediate schist in the east and merging into one strongly folded zone in the west. Both orebodies comprise high grade, massive sulphide lenses close to a geologically defined schist footwall, and medium to low grade disseminated mineralization in magnetite rich rocks defined by an economic hangingwall.
Description of major geological structures. (show on plan)	No faulting has been encountered in the orebodies, but folding is fairly intense. The orebody is characterized by a weak schist footwall and a strong, magnetite rich hangingwall.
Princpal ore minerals	Galena, sphalerite and chalcopyrite, in order of decreasing abundance

6. Geotechnical Information:	
Describe different geotechnical areas.	Orebody very complex and very difficult to divide it into differnt geotechnical areas. Rock mechanics problems are largely encountered in flat dipping areas. Here a combination of the flat dipping foliation and F3 and F4 shears and fractures can result in roof collapse if the stope span is too wide. The Broken Hill deposit may however be sub-divided into 4 geological domains, namely: - Domain A: < 60m, highly oxidised part of orebody. The weathered nature of the rocks causes unstable roof conditions. - Domain B: Part od orebody having steeper dips (50 - 80 degrees) and which have largely been mined out by blast hole stoping. Flat dipping areas also occur due to the folded nature of the orebody. - Domain C: Central flat dipping (20 - 50 degree) portion of orebody. Regular stability pillars have to be left due to flat dip and to prevent roof collapse. - Domain D: Complex folded area in western part of the orebodies. Here flat dips often cause poor roof conditions. Variations in rock type, strength and competencies are encountered in both orebodies as well as the hangingwall and footwall. Rockwall hazards are associated with some of these rock types.
Young's Modulus: - $E_{hangingwall}$; - $E_{footwall}$; - E_{reef} .	
Poissons's Ratio: - $v_{hangingwall}$; - $v_{footwall}$; - $v_{reef.}$	
Rock density: - $\rho_{hangingwall}$; - $\rho_{footwall}$; - $\rho_{reef.}$	
Uniaxial Compressive Strength: - $UCS_{hangingwall}$; - $UCS_{hangingwall}$; - $UCS_{hangingwall}$.	
Cohesion: - $C_{hangingwall}$; - $C_{footwall}$; - C_{reef} .	
Rock Quality Designation: - $RQD_{hangingwall}$; - $RQD_{footwall}$; - RQD_{reef} .	

Describe hydraulogy, geohydrology and influence on underground excavations.	Surface water may enter the shallower underground workings through continuous fractures from surface. Backfill water does drain through joints and fractures from stopes being filled, weakening the country rock and causing weathering along joints, faults and slips. This is more significant in formations where pyrite and pyrrhotite are prone to oxidation.
Describe different joint sets in terms of: - strike orientation; - dip; - dip direction; - spacing/frequency; - continuity; - strength.	
Describe potential failure mechanisms.	
Rock Mass Rating: - $RMR_{hangingwall}$; - RMR_{reef} ; - $RMR_{hangingwall}$;	
Mining Rock Mass Rating: - $MRMR_{hangingwall}$; - $MRMR_{reef}$; - $MRMR_{hangingwall}$;	
Rock Mass Strength: - $RMS_{hangingwall}$; - RMS_{reef} ; - $RMS_{hangingwall}$;	
Design Rock Mass Strength: - $DRMS_{hangingwall}$; - $DRMS_{reef}$; - $DRMS_{hangingwall}$;	
Stability Index (plan area divided by the perimeter of the excavation): - $SI_{hangingwall}$; - SI_{reef} ; - $SI_{hangingwall}$	
How does the mine ensure that structural features and mineralization zones, which could influence local/regional stability, are identified pro-actively?	
Describe significance of shear or weak zones, joint orientation, etc. and their effect on structural stability.	
In situ stresses: - principal stress directions; - principal stress magnitudes; - measured or estimated.	

7. Pillars:	
Description of pillar types being used (e.g. crush pillars)	
Typical pillar dimensions: - pillar widths; - pillar lengths; - pillar heights.	
Typical pillar spacings: - dip spacings; - strike spacings.	
Description of pillar design methodology.	
Range of pillar stresses and how it is normally calculated.	
Range of pillar strengths and how it is normally calculated.	
Minimum allowed FOS	
Description of any other pillar design methods used in the past or planned for the future	
Does the mine use numerical analyses in the design process? - name of software; - input parameters.	
Has the mine experienced pillar failure in the past and why? - pillars design incorrect; - sub-standard pillars; - change in rock strength; - change in structure.	
Describe mode/s of failure.	
Actual FOS of failed pillars (back analyse)	
Describe failure in vicinity of pillar failure (e.g. footwall heave, roof spalling, etc.)	
Is pillar extraction being done or planned for the future?	
Describe physical interaction of opencast / other topographical features on underground workings.	

8. Stope Spans:	
Mine standard for stope spans: - dip; - strike.	
Description of design methodology (e.g. empirical method, RM classification method, beam theory, etc.)	
Does the mine use numerical analysis in the design process? - name of software; - input parameters.	
Design parameters (e.g. depth, thickness of stratification, etc.)	
Has the mine experienced hangingwall failure in the past and why? - design incorrect; - sub-standard spans; - change in rock strength; - change in rock structure.	
Description of any relevant instrumentation such as closure-ride meters, extensometers, etc.	

9. In-Stope Support:	
Description of support types being used: - temporary/primary; - permanent/secondary; - additional/tertiary.	*CAF:* According to ground conditions with friction rockbolts supplemented by 15m cable bolts where required. In certain wide areas, post pillars were left for regional support. A full lift of cemented hydraulic fill, reinforced with pinned cross cables and suspended weldmesh, was placed in each sill to facilitate mining from below. Thereafter, successive lifts were filled with 3m cycloned dune sand, supplemented with development waste, and topped with a 0,25m layer of 30:1 cemented backfill, followed by 0,75m of 8:1 cemented backfill. The cemented backfill consisted of mixtures of uncycloned mill tailings and Portland cement). *BHOS*: Cemented backfill with a strength of around 0,7MPa in the stopes and low strength cemented backfill in the pillars.
Design strength of support units: - temporary/primary; - permanent/secondary; - additional/tertiary.	
Area supported per unit: - temporary/primary; - permanent/secondary; - additional/tertiary.	
Describe design methodology.	

10. Underground Visit/s:	
Name/s of working places	

11. References:

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Appendix C Summary of mapping data at Mines A, B and C

SUMMARY OF DETAILED MAPPING AT EASTERN CHROME (MINE A)

Mine Name	Locality	Site Name	Rock Type	Depth below Surface (m)	Direction of Stoping	Stability	Failure Area (m)	Failure Height (m)	Failure Volume (m^3)	Mass of Failure (Kg)	Width of weak zone (m)	Intact Rock Strength (MPa)	Joint Description: No of sets	Spacing	Joint Orientation: Orientation	Mean Orientation	Range	Std Deviation	Joint Condition: Infill type	Infill thickness (mm)	Rough. Small Scale	Rough. Large Scale	C	phi	RMR	Adjustments: Weathering	Orientation	Induced Stress	Blasting	Total Adjustments	MRMR	Barton (Q)	GSI	E (S+P) (MPa)	Excavation Geometry: Span length	Span width	Height	Hydraulic radius	Stability Index	Regional Support: Pillar Size	Condition	Local Support: Elongate Spacing	Instope Pillar Size	Pillar Condition	Split sets	Shepherds Crooks	Rock bolts	Associate Geological Structure	Water	Comments
Steelpoor	1	33/15A-1	pyroxenite		Breast - strike parallel	stable							2+1(random)	2.75	2 subV	J1=81/013 J1a=85/338 J2=65/054 R=43/209	61/295-71/150 75/329-82/170 54/033-65/253 28/187-53/235	94.1 99.53 2.52 12.02	clean	0	smooth -rough und	planar			62	1	0.9	1	0.95	0.855	53	6.6	69	29 854	40	12	1.5-2.5	5.6	Stable	3X3	good	0	3x3	good	-	-	1x1.5 diamond pattern	edges of zone of influence of FZ	dry	Increase in the number of discontinuos random joints
Steelpoor	2	33/15A-2	chromite		Breast - strike parallel	stable							2	4.3	2 subV	J1=86/326 J1a=86/302 J2=89/025 J2a=44/032	74/131-64/340 75/293-79/131 79/015-80/219 21/360-65/060	101.8 1.41 8.19 -	clean	0	smooth planar	planar			49	1	0.9	1	0.95	0.855	41	9.1	56	14 128	40	12	1.5	5.6	Stable	3x3	good	0	3x3	good	-	-	1x1.5 diamond pattern	Top and bottom contact of chrome seam form distinct weakness planes	dry	Most of the individual joint are not continuous through the chrome-pyroxenite contact. But the J1 joints in the chromite refelects same orientation but is generally more closely spaced and have a lower joint condition rating.
Steelpoor	3	33/15A-3	pyroxenite		Breast - strike parallel	stable							3 + random	5.2	2 subV + 1 incl	J1=84/020 J1a=85/308 J2=88/037 Jr=63/194									55	1	0.8	0.9	0.95	0.855	36	8.3	52		40	12	1.5-3.5	5.6	Stable	2.9x3	good	0	0	good			1x1.5 diamond pattern	In inflence zone of Fault zone	dry	The zone of influence of the fault is about 7m wide on either side of the fault. This zone is well jointed but many of the joint are sympathertic to the fault orientation and non continuous.
Steelpoor	4	33/15A-4	pyroxenite		Breast - strike parallel	failure	4x2.5	1			7	220	2+ random	8	2 subV + 1 incl	J1=87/312 J1a=85/336 J2=89/057 Jr=50/200	61/290-65/151 73/331-83/166 59/042-66/254 36/174-68/229	92.39 1.41 101.5 12.4	serpentinite	1-4	smooth planar	planer - und			50	1	0.8	0.75	0.95	0.6	25	2.07	39	5 309	40	15	1.5-3.5	1.6	Transition	3x3	good	0	3x3		-	-	1x1.5 diamond pattern	FOG only occurred wherenot support by pillar	dry	
Steelpoor	5	33/16S-1	pyroxenite		Breast - strike parallel	stable					2		2+ random	7.1	2 subV + 1 incl	J1=66/261 J1a=79/296 J2=83/018	43/231-86/112 60/103-44/323 63/359-70/214	10.79 91.21 84.2	serpentinite	1-2	smooth - rough planar	planar			49	1	0.8	0.85	0.95	0.646	32	4.5	39	5 309	50	5	1.5-2.5	6.4	Transition	3x3	good	0	3x3	good	-	-	1x1.5 diamond pattern	Fault zone - very small FOG	dry	Very small FOG associated with FZ tiny wedge formed by conjugated joint sets (J1)
Steelpoor	6	33/16S-2	pyroxenite		Breast - strike parallel	stable							3	4	2 subV + 1 subH	J1=56/274 J1a=72/110 J2=82/018	62/098-80/317 29/235-80/305 30/235-79/301 62/356-69/214 62/100-80/323 62/357-68/214	13.41 108.9 76.55	clean	0	rough und	planar			61	1	0.9	1	0.95	0.855	51	10.6	70	31 623	50	15	1.5-2.5	6.4	Stable	3x3	good	0	3x3	good	-	-	1x1.5 diamond pattern	Out of FZ influence	dry	Representative of most of the rockmass in the mine.
Steelpoor	7	30/14S-3S-	pyroxenite		Breast - strike parallel	Failure	1.0x5				1		2+ random	13.2	2 subV + 1 incl	J1=86/279 J1a=66/107 J2=84/023	59/260-82/122 48/091-81/127 72/013-84/213	9.048 9.074 23	serpentinite and talc	2	smooth und	planar - und			52	1	0.8	0.85	0.95	0.646	34	2.08	47	8 414	10	25	1.5	6.75	Transition	rib 25m	good	-	2.9x3.2	good	-	-	?	Fault zone sub parallel to J	dry	Small FOG associated with FZ tiny wedges formed by conjugated joint sets (J1)
Steelpoor	8	30/14S-3S-	pyroxenite		Breast - strike parallel	Failure	0.5x5				0.5		3	8	3 subV	J1=86/132 J1a=72/120 J1b=44/314 J2=88/027 J3=53/277	76/117-80/324 61/104-79/141 31/301-59/329 76/017-78/215 34/251-76/299	5.51 8.89 11.34 126.6 1.41	serpentinite and talc	1	smooth und	planar - und			49	1	0.8	0.85	0.95	0.646	31	3.9	46	7 943	10	20-25	1.5	6.75	Transition	rib 25m	good	-	3x3	good	-	-	?	Fault zone sub parallel to J	dry	Small FOG associated with FZ tiny wedge formed by conjugated joint sets (J1).
Steelpoor	9	30/14S-4S-	pyroxenite		Breast - strike parallel	Stable							3 + random	1	1 subV + 1 incl	J1=85/240 J1a=84/28 J2=60/274	- - -	- - -	clean	0	smooth und	planar - und			77	1	0.9	1	0.95	0.855	65	19	70	31 623	25	25	1.5	11.2	Stable	rib 25m	good	-	3x2.5	good	-	-	-	out of FZ zone	dry	Edge of FZ well defined.
Steelpoor	10		pegmatite			Failure	3x2.5				7		2 + random	6	2 subV + 1 incl	J1=84/290 J2=75/184 J1incl=46/282	- - -	- - -	serpentinite and talc	2-10	smooth und to slickensided und	und			42	1	0.85	0.85	0.95	0.686	29	1.3	47		10	25	1.5	6.25	Transition	rib 25m	good	1.5x2m diamond					1x1.5 diamond pattern		dry	
Lannex	11	14S16-1	pyroxenite		Up dip	Failure							3+ random	6	2 subV + 1subH + 1 incl	J1=89/322 J1a=88/305 J1b(r)=57/296 J2=68/234 J3=15/331 Jran=15/138	68/312-69/154 76/297-82/133 47/283-69/308 70/244-69/064 02/258-32/022 07/101-23/177	105.7 1 104.8 18.46 - -	serpentinite and talc	5-10	smooth-rough und	planar - und			44	1	0.8	0.85	0.95	0.646	28	3.5	43		10	15	2		transitional	rib 15m	good	0	2.5x2.5	good	-	-	1x1.5 diamond pattern	Fog associated with Fault	dry	FOG associated with a fault perpendicular to the direction of mining
Lannex	12	14S16-2	pyroxenite		Up dip	stable							3 + random	1.2	2 subV + 1 subH	J1=88/133 J1a=78/279 J2=87/188 Jr1=32/012 Jr2=11/318	77/297-70/145 65/270-86/287 80/000-75/196 23/354-42/029 02/279-20/014	101.1 - - - -	serpentinite and talc	0-1	rough planar-und	planar			62	1	0.9	0.95	0.95	0.812	50.3	7	80																dry	
Lannex	13	14S16-3	pyroxenite		Up dip	failure							2 + random	11	2subV	J1=85/279 J1a=85/129 J1b=88/156 J2=89/232 J3 (FZ)=34/31 Jr1=64/172 Jr2=30/023	60/258-63/113 61/296-58/143 79/323-76/169 80/044-81/240 17/289-51/344 47/149-77/203 22/354-41/058	90.86 84.5 105.3 9.897 - 14.97 22.63	serpentinite and talc	1-2	smooth slickenside	planar			42	1	0.8	0.9	0.95	0.684	29	1.5	37						transitional										dry	
Lannex	14	14S13-1	pyroxenite		Up dip	failure							3	8.3	2subV + 1 subH	J1=87/277 J2=88/158 J3=44/316 Jr=60/018	65/258-63/114 78/328-76/168 33/303-52/330 55/008-70/028	90.72 125.9 - -	serpentinite and talc and fault guoge	2-30	smooth slickenside	planar			41	1	0.85	0.85	0.95	0.686	28	2	40						transitional										moist	
Lannex	15	14S5-1	pyroxenite		Up dip	stable							3	2.5	2subV + 1 subH	J1=85/136 J2=89-232 J1 (FZ)=30/31 Jr=63/165	70/118-79/339 79/223-81/060 14/284-47/350 46/148-78/190	99.11 - 8.142 9.45	clean	0	rough und	planar			63	1	0.9	1	0.95	0.855	53	3.5	70																dry	
Lannex	16	14/26	pyroxenite		Up dip	Failure							4	18.5	2 subV + 1 subH + 1 inc	J1=78/330 J1a=89/301 J1 inc=57/155 J2a=88/175 J4-FZ=34/300 J1r=65/089	59/312-79/164 79/290-89/129 40/120-77/184 82/346-78/185 12/248-57/339 47/077-87/105	6.33 - - 22.28 18.81 2.944	serpentinite	1-10	smooth planar- rough und	planar - curved			36	1	0.8	0.9	0.95	0.684	25	1.5	26						transitional										dry	
Lannex	17	14/25	pyroxenite		Up dip	Failure									2 subV + 1 subH + 2 inc	J1=87/132 J1a=88/278 J2a=79/348 J3(FZ)=24/291 J4(FZ)=57/088 J5=83/049 J8r=14/090 J2r=63/004	69/112-72/334 76/087-71/286 68/337-84/180 07/227-40/325 38/060-74/116 71/039-84/240 01/044-28/131 48/352-75/016	105.9 131.5 0.58 14.67 13.43 2.12 6.36 2.12							36	1	0.8	0.9	0.95	0.684	25		26						transitional										dry	
Lannex	18	6757	pyroxenite		Up dip	Failure							5	5.5	2 subV + 1 subV + 1 inc	J1=82/320 J3(FZ)=23/294 J4(FZ)=56/084 J5=83/049 Jr=09-085 J1r=82/282 J2r=59/006	73/306-81/154 08/256-39/328 38/057-74/114 72/038-82/240 03/020-19/133 72/273-89/110 51/352-68/016	7.07 9.82 12.12 2.12 - - -	serpentinite	1-2	rough und	planar			45	1	0.75	0.85	0.95	0.606	28	1	46						transitional										moist	
Steelpoor	19		Pyroxenite									220	3	2.5	2 subV + 1 subH	J1=85/267 J2=78/347 J3=15/279	- - -	- - -	serpentinite and talc and clean	0-5	smooth und	planar			57	1	0.9	0.9	0.95	0.684	44	9	67		85	27	1.5	10.3	Stable	rib	good	2x2.5	none	-				Dyke sub parallel to J1 - release	dry	Failure on the contact with the dyke which has a thick clay infill
Steelpoor	20		Pyroxenite									220	3	4.5	2 subV + 1 SubH	J1=83/284 J2=79/358 J3=12/265	- - -	- - -	clean	0	smooth und	planar			53	1	0.9	0.85	0.95	0.726	39	3	42		85	27	1.5	10.3	transitional	rib		2x2.5	none	-				Jointing Sympatetic to dyke orien	dry	A dyke contact this zone is symapathetic th the orientation of the dyke. Yhis zone provides the instope failue.
Tweefontei	21		Pyroxenite		breast-parallel	failure	14.5x24	1.25m				220	3	2	2 subV + 1 subH	J1=80/307 J2=84/246			serpentinite and talc fault guoge	2-20	rough und	planar			56	1	0.9	0.95	0.95	0.812	45	1.9	47		120	20	1.9	8.6	stable	rib	good	2x3	none	-				Weakcontinuous joints and J2 para	dry	The failed zone plots in thesatble section of the Stability index curve - but since the failure occurred on a geological feature the rating at the contact can be lower to bring the MRMR into the transitional zone. This is achived by reducing the stress reduction factor 0.8. We have assumed that there is a significant change in stresses across the fault plane.

SUMMARY OF DETAILED MAPPING AT DILOKONG (MINE B)

Mine Name	Locality	Site Name	Rock Type	Depth below Surface (m)	Direction of Stoping	Stability	Failure Width (m)	Failure length (m)	Failure Height (m)	Failure Volume (m^3)	Mass of Failure (t)	Width of weak zone (m)	Intact Rock Strength (MPa)
Dilakong	1	N1	chromite	200	Breast - strike parallel	stable							90
Dilakong	2	N2	pyroxenite	200	Breast - strike parallel	stable							200
Dilakong	3	N3	pyroxenite	200	Breast - strike parallel	failure	8	10	0.3	24	75.024		200
Dilakong	4	N4	pyroxenite	200	Breast - strike parallel	failure	5	5	0.3	7.5	23.445		220
Dilakong	5	N5	pyroxenite	200	Breast - strike parallel	stable							200
Dilakong	6	N6	pyroxenite	200	Breast - strike parallel	failure							200
		N7	pyroxenite	200		failure	10	10	0.5	50	156.3		200
Dilakong	7	N8	pyroxenite	200	Breast - strike parallel	beam failure	15	60	1.5	1350	4220.1	1	200
Dilakong	8	N10	pyroxenite	200	Breast - strike parallel	failure	10	15	1.2	180	562.68	0.5	200
Dilakong	9	N11	pyroxenite	200	Breast - strike parallel	failure	10	10	1.2	120	375.12		200
Dilakong	11	S4-1	pegmatite	250	Reef drive	stable						7	200
Dilakong	12	S4-2	pyroxenite	250	Reef drive	stable							200
Dilakong	13	S4-3	pyroxenite	250	Breast - strike parallel	failure dome							200
Dilakong	14	S4-4	pyroxenite	250	Breast - strike parallel	failure	3	5	0.5	7.5	23.445		200
Dilakong	15	S4-5	pyroxenite	250	Breast - strike parallel	stable							
Dilakong	16	S4-6	pyroxenite	250	Breast - strike parallel	dome	15	15	1	225	703.35		200
Dilakong	17	S4-7	pyroxenite	250	Breast - strike parallel	stable							
Dilakong	18	S4-8	pyroxenite	250	Breast - strike parallel	dome	3	3	0.5	4.5	14.067		200
Dilakong	19	S4-9	pyroxenite	250	Breast - strike parallel	dome	3	5	0.5	7.5	23.445		200
Dilakong	20	S4-10	Pyroxenite	250	Breast - strike parallel	failure dome	1.2	10	0.4	4.8	15.005		200
Dilakong	21	S4-11	Pyroxenite	250	Breast - strike parallel	stable							200
Dilakong	21	S4-12	Pyroxenite	250	Breast - strike parallel	stable							200
Dilakong	22	S4-13	Pyroxenite	250	Breast - strike parallel	failure dome	5	6	0.4	12	37.512		200
Dilakong	23	S4-15	Pyroxenite	250	Breast - strike parallel	stable							200
Dilakong	27	UL1	Pyroxenite	50	Breast - strike parallel	stable							200
Dilakong	28	UL2	Pyroxenite	50	Breast - strike parallel	failure	2	2	0.4	1.6	5.0016		200
Dilakong	29	UL3	Pyroxenite	50	Breast - strike parallel	failure	1	3	0.2	0.6	1.8756		200
Dilakong	30	UL4	Pyroxenite	50	Breast - strike parallel	failure	2	3	0.4	2.4	7.5024		200
Dilakong	31	UPL5	Pyroxenite	50	Breast - strike parallel	failure	3	3	0.35	3.15	9.8469		200
Dilakong	32	L1-north	Pyroxenite	250	F/W drive	stable							200
Dilakong	33	L2	Pyroxenite	250	F/W drive	stable							200

Site Name	Joint Description: No of sets	Spacing	Joint Orientation: Orientation	Mean Orientation	Range	Std Deviation	Joint Condition: Infill type	Infill thickness (mm)	Rough. Small Scale	Rough. Large Scale
N1	4	5	2 subV + 1 subH + 1 incl	J1=85/329 J2=81/204 J2A=56/250 J3=11/316	80/139-67/341 65/180-86/048 68/235-38/272 07/230-31/056	128.7 15.57 7.365 145.9	clean	0	smooth -rough und	planar
N2	3	3.5	2 subV + 1 subH	J1a=89/259 J2=79/193 J3=17/278	71/053-67/280 72/182-88/205 38/205-03/326	101.9 4.24 7.07	clean	0	smooth - rough und	planar
N3	3	6.3	2 subV + 1 subH	J1=80/276 J2=81/200 J3=09/278	68/265-88/282 70/185-81/038 01/224-19/333	- 6.93 -	clean minor calcite	<1mm	smooth - rough und	planar
N4	3	6.8	2 subV + 1 subH	J1=79/109 J1a=68/276 J2=79/195 J3=16/280	67/099-88/117 58/267-78/286 70/178-88/208 27/246-3/333	- - 5.77 19.8	serpentinite	1-4	smooth planar	planer - und
N5	3	3.1	2 subV + 1 subH	J1=76/100 J2=80/191 J3=12/282	64/086-89/112 71/183-90/219 2/218-25/343	10.79 91.21 84.2	serpentinite	1-2	smooth - rough planar	planar
N6	3	7	2 subV + 1 subH	J1a=65/255 J2=66/192 J3=11242	73/242-54/265 51/181-80/210 03/179-21/288	- 6.43 -	clean	0	rough und	planar
N7	3	4.6	2 subV + 1 subH	J1a=65/255 J2=66/192 J3=11242	73/242-54/265 51/181-80/210 03/179-21/288	- 6.43 -	serpentinite and talc	2-10	rough und	planar
N8	3	4.25	2 subV + 1 subH	J1=69/263 J2=76/189 J3=8/278	53/248-88/278 64/180-87/198 1/213-19/357	1.41 - -	serpentinite and talc	2	smooth und	planar - und
N10	2	5	1 subV + 1 subH	J2=70/213 J3=26/306	53/195-89/228 39/270-9/346	7.78 18.92	serpentinite and talc	1	smooth und	planar - und
N11	4	7	2 subV + 1 subH + 1 incl	J1a=85/258 J2=84/183 J3=5/292 J3a=35/251	73/070-63/270 74/171-85/017 16/245-01/026 26/234-45/268	125.2 4.24 - -	clean	0	smooth und	planar - und
S4-1	5	6.2	3 subV + 2 subH	Jl=85/320 J1a=89/248 J2=79/056 J2a=60/230 J3=17/057 J3a=21/315	72/309-80/159 78/262-68/120 64/040-88/078 46/194-82/250 05/000-38/103 06/278-35/341	88.97 106.4 56.74 16.29 30.53 20.51	serpentinite and talc	2-10	smooth und to slickensided und	und
S4-2	4	4.6	3 subV + 1subH	J1=65/109 J1a=79/141 J2=88/196 J3=24/008	42/092-85/123 71/130-88/147 76/182-75/032 46/348-03/044	5.72 - 77.86 5.83	serpentinite and talc	5-10	smooth-rough und	planar - und
S4-3	4	3	2 subV + 2 subH	J2=89/016 J2a=55/056 J2b=60/226 J3=16/331 J3a=11/181	77/297-70/145 65/270-86/287 80/000-75/196 23/354-42/029 02/279-20/014	103.71 14.71 - - 16.62	serpentinite and talc	0-1	rough planar-und	planar
S4-4	3	3	1subV + 2 subH	J1a=26/112 J2=89/196 J3=14/311	35/091-15/131 77/185-89/205 20/270-4/354	- - -	serpentinite and talc	1-2	smooth slickensided	planar
S4-5	3	2.8	1 subV + 2 subH	J2-82/025 J3=22/357 J3a=18/208	70/014-89/036 35/324-11/026 27/180-10/239	0.71 246.1 -	serpentinite and talc and	2-30	smooth slickensided	planar
S4-6	2	2.5	1subV + dome	J2-76/069 d=21/007	67/056-84/081 14/295-35/054	15.56 141.9	clean	0	rough und	planar
S4-7	3	5.3	1 subV + 1 subH + 1 incl	J1=65/089 J2=74/210 J3=25/322	56/078-76/099 64/189-86/226 38/279-14/107	6.33 - -	serpentinite	1-10	smooth planar- rough und	planar - curved
S4-8	1	5	1 subV + dome	J1=74/152 J1a=81/348	56/142-87/164 72/337-89/00	1.73 2.12				
S4-9	2	4.5	1 subV + 1 subH + dome	J1=75/163 J3=12/262 d=35/011	63/149-87/179 22/207-06/320 42/354-25/02	10.61 25.46 -	serpentinite	1-2	rough und	planar
S4-10	4	4.2	1 subV + 1 subH + dome	J1=73/282 J2=62/218 J3=15/336 d=45/015	60/272-85/290 50/204-77/230 03/293-23/011 55/003-37/027	1.41 1.41 - -	serpentinite and talc and clean	0-5	smooth und	planar
S4-11	3	3.3	1 subV + 1 SubH + incl	J1a=70/261 J1b=68/101 J3=13/171	55/248-85/272 58/090-79/111 01/135-26/222	0.58 - 8.49	clean	0	smooth und	planar
S4-12	3	8	2 subV + 1 subH	J1=87/28 J2=87/356 J3=20/318	81/092-74/304 77/159-71/016 10/292-30/343	135.1 169.5 -	clean	0	smooth und	planar
S4-13	4	6.5	2 subV + 1 subH + dome	J1=68/137 d=60/323 J2=80/056 J3=08/007	44/121-83/162 48/310-70/334 66/047-89/067 04/282-19/073	16.44 - - -	serpentinite and talc fault gouge	2-20	rough und	planar
S4-15	3	5	2 subV + 1 subH	J1=76/282 J2=82/185 J3=13/290	62/267-89/295 74/164-89/205 28/258-02/011	5.77 16.26 30.41	clean	0	smooth und	planar
UL1	3	6.5	2 subV + 1 subH	J1=58/138 J2=73/045 J3=26/291	76/125-41/154 56/033-86/058 36/272-16/310	6.03 4.57 7.07	clean + minor calcite	<1mm	smooth - rough und	planar
UL2	3	7	2 subV + 1 subH	J1=88/310 J2=73/045 J3=44/088	80/116-73/329 56/033-86/058 27/070-50/096	96.71 4.57 9.9	clean + minor calcite	<1mm	smooth und	planar
UL3	3	3	2 subV + 1 subH	J1=82/104 J2=86/022 J3=19/292	76/253-58/125 79/187-72/037 27/266-8/326	74.11 93.22 2.08	calcite clay	1-2mm	smooth und	planar
UL4	3	10	2 subV + 1 subH	J1=88/111 J2=87/197 J3=27/257	75/099-80/305 72/180-71/037 42/240-14/278	4.04 97.09 2.12	clean + minor calcite	<1mm	smooth und	planar
UPL5	3	5.2	2 subV + 1 subH	J1=83/087 J2=81/012 J3=26/279	76/077-89/098 70/001-89/021 34/258-16/296	0.58 2.51 1.41	clean + minor clay	1-2mm	smooth und	planar
L1-north	4	3.5	3 subV + 1 subH	J1=70/118 J2=74/212 J2a=79/066 J3=15/107	54/108-87/131 65/202-86/222 78/232-55/082 06/270-37/101	3.46 2.12 118.1 155.3	clean	0	smooth und	planar
L2	3	6.8	2 subV + 1 subH	J1=70/123 J2=76/210 J3=06/258	59/106-81/138 64/189-84/053 02/189-16/342	7.07 14.14 2.83	clean	0	smooth und	planar

Site Name	C (Mpa)	phi	RMR	Adjustments: Weathering	Orientation	Induced Stress	Blasting	Total Adjustments	MRMR	Barton (Q)	RMR Bieniawski 1989	GSI	E (S+P) (MPa)
N1	7.44	54.97	58	1	0.9	1	0.94	0.846	50	11.26	70	65	23 714
N2	7 7	54.18 54.18	59	1	0.9	1	0.94	0.846	50	10.55	67	62	19 953
N3			56 56	1 1	0.9 0.75	1 1	0.94 0.8	0.846 0.6	47.376 33.6	11.26	67	62	19 953
N4	7.44	54.97	59 59	1 1	0.85 0.7	0.8 0.75	0.94 0.94	0.6392 0.4935	37.7128 29.1165	11.26	70	65	23 714
N5	8.367	56.21	62	1	0.8	0.85	0.95	0.646	40.052	11.73	75	70	31 623
N6	6.208	52.24	57	1	0.9	1	0.95	0.855	48.735	11.26	60	55	13 335
N7	6.748	53.64	51	1	0.9	0.95	0.94	0.8037	40.9887	11.26	65	60	17 783
N8	7.143	54.45	65	1	0.8	0.85	0.95	0.646	41.99	11.26	68	63	21 135
N10	8.367	56.21	63 63	1 1	0.8 0.7	0.85 0.7	0.95 0.95	0.646 0.4655	40.698 29.3265	11.26	75	70	31 623
N11	7.44	54.97	61 61	1 1	0.9 0.7	1 0.7	0.95 0.95	0.855 0.4655	52.155 28.3955	10.9	70	65	23 714
S4-1				1	0.85	0.85	0.95	0.686375	0	1.3			
S4-2				1	0.8	0.85	0.95	0.646	0	3.5			
S4-3				1	0.9	0.95	0.95	0.81225	0	7			
S4-4	8.374	54.51		1	0.8	0.9	0.95	0.684	0	1.5	72	67	26 607
S4-5	8.032	53.99	59	1	0.9	1	0.94	0.846	49.914	10.55	70	65	23 714
S4-6	7.583	53.18	62 62	1 1	0.9 0.85	1 0.9	0.94 0.94	0.846 0.7191	52.452 44.5842	11.25	67	62	19 953
S4-7	8.032	53.99	59	1	0.9	1	0.94	0.846	49.914	10.07	70	65	23 714
S4-8	7.319	52.63	62 62	1 1	0.9 0.7	1 1	0.94 0.8	0.846 0.56	52.452 34.72	11.73	65	60	17 783
S4-9	8.967	55.27	65 65	1 1	0.9 0.7	1 1	0.94 0.75	0.846 0.525	54.99 34.125	11.73	75	70	31 623
S4-10	7.583	53.18	59 59	1 1	0.9 0.85	1 0.85	0.94 0.94	0.846 0.67915	49.914 40.06985	10.07	67	62	19 953
S4-11	8.032	53.99	70	1	0.9	1	0.94	0.846	59.22	11.85	70	65	23 714
S4-12	8.032	53.99	59	1	0.9	1	0.94	0.846	49.914	10.55	70	65	23 714
S4-13	7.725	53.45	62 62	1 1	0.9 0.7	1 0.8	0.94 0.94	0.846 0.5264	52.452 32.6368	10.55	68	63	21 135
S4-15	8.967	55.27	62	1	0.9	1	0.94	0.846	52.452	11.73	75	70	31 623
UL1	3.132	66.25	57	1	0.9	1	0.94	0.846	48.222	4.31	68	63	21 135
UL2	2.261	65.24	57 57	1 0.9	0.9 0.7	1 0.8	0.94 0.94	0.846 0.47376	48.222 27.00432	8.62	60	55	13 335
UL3	2.261	65.24	65 65	1 0.9	0.9 0.7	1 0.8	0.94 0.94	0.846 0.47376	54.99 30.7944	8.89	60	55	13 335
UL4	2.747	65.92	54 54	1 0.9	0.9 0.7	1 0.8	0.94 0.94	0.846 0.47376	45.684 25.58304	7.29	65	60	17 783
UPL5	1.919	64.42	57 57	1 0.9	0.9 0.7	1 0.8	0.94 0.94	0.846 0.47376	48.222 27.00432	8.8	55	50	10 000
L1-north	8.967	55.27	62	1	0.9	1	0.94	0.846	52.452	11.85	75	70	31 623
L2	8.032	53.99	59	1	0.9	1	0.94	0.846	49.914	10.9	70	65	23 714

Site Name	Excavation Geometry: Span length	Span width	Area	Perimeter	Height	Hydraulic radius	Stability Index	Regional Support: Pillar Size	Condition	Local Support: Elongate Spacing	Instope Pillar Size	Pillar Condition	Split sets	Shepherds Crooks	Rock bolts	Associate Geological Structure	Water
N1	260	43	11180	606	1.3	18.4	Transition		good	2x2.5 diamond			-	-	-	massive unaffected by weak zones	dry
N2	240	39.5	9480	559	1.3	17.0	Transition		minor stress fracturing	2x3 diamond			-	-		massive unaffected by weak zones	dry
N3	210	16	3360	452	1.3	7.4	Stable Transitional			2x3 diamond						false hanging - disseminated orebody	dry
N4	120	12	1440	264	1.3	5.5	Stable transitional			2x2.5 diamond			-	-		false hanging - disseminated orebody	dry
N5	260	32	8320	584	1.3	14.2	stable			2x2.5 diamond			-	-		massive unaffected by weak zones	dry
N6	260	32	8320	584	1.3	14.2	Stable			2x2.5 diamond			-	-		false hanging - disseminated orebody	wet
N7	90	75	6750	330	1.3	20.5	transitional			2x3 diamond						Unfavourably oriented Joint set with clay infill	dry
N8	130	38	4940	336	1.3	14.7	transitional			2x2.5 diamond			-	-	?	Large beam failure - defined by clay infilled fault and joints.	dry
N10	12	10	120	44	1.3	2.7	stable stable			2x2.5 diamond			-	-	?	key block defined by clay infilled joints	dry
N11	12	12	144	48	1.3	3.0	Stable stable			2x2.5 diamond			-	-	-	key block defined by clay infilled joints	
S4-1	100	3	300	206	1.3	1.5				1.5x2m diamond						Closely jointed	dry
S4-2	100	3	300	206	1.3	1.5				2x3 diamond			-	-		Closely jointed	dry
S4-3	136	34	4624	340	1.3	13.6				2x3 diamond						Small scale doming	dry
S4-4	136	34	4624	340	1.3	13.6				2x3 diamond						massive unaffected by weak zones	dry
S4-5	136	34	4624	340	1.3	13.6	stable			2x3 diamond						massive unaffected by weak zones	dry
S4-6	50	38	1900	176	1.3	10.8	stable transitional			2x3 diamond						Large scale dome failure associated with pegmatite	dry
S4-7	15	30	450	90	1.3	5.0	stable			2x3 diamond						massive unaffected by weak zones	dry
S4-8	48	20	960	136	1.3	7.1	stable transitional			2x3 diamond						Doming	dry
S4-9	48	20	960	136	1.3	7.1	stable transitional			2x3 diamond						Doming	dry
S4-10	82	29	2378	222	1.3	10.7	Stable transitional			2x2.5 diamond						Doming	dry
S4-11	94	27	2538	242	1.3	10.5	stable			2x2.5 diamond						massive unaffected by weak zones	dry
S4-12	84	29	2436	226	1.3	10.8	stable			2x2.5 diamond						massive unaffected by weak zones	dry
S4-13	25	29	725	108	1.3	6.7	stable transitional			2x2.5 diamond						Doming	dry
S4-15	103	27	2781	260	1.3	10.7	stable			2x2.5 diamond						massive unaffected by weak zones	dry
UL1	260	2.9	754	525.8	1.3	1.4	stable			2x2.5 diamond						Slightly weathered rock mass	dry
UL2	25	15	375	80	1.3	4.7	stable transitional			2x2.5 diamond						Failure due to weathered joint surface	dry
UL3	15	10	150	50	1.3	3.0	stable stable			2x2.5 diamond						Failure due to weathered joint surface	dry
UL4	15	10	150	50	1.3	3.0	stable transitional			2x2.5 diamond						Failure due to weathered joint surface	dry
UPL5	13	12	156	50	1.3	3.1	stable transitional			2x2.5 diamond						Failure due to weathered joint surface	dry
L1-north	100	4	400	208	1.3	1.9	stable			2x2.5 diamond						massive unaffected by weak zones	dry
L2	100	3.5	350	207	1.3	1.7	stable			2x2.5 diamond						Closely jointed zone	dry

Site Name	Comments
N1	There is no stress fracturing in the rib pillar at this site. Although some blast damage is evident some blast damage is evident.
N2	The majority of the joints abut against the chrome seam. The same joint orientation is reflected in the chrome seam but the spacing is generally, but not always similar.
N3	ailure of the hanging wall due to the 20 cm thick disseminated chrome layer that is locally developed above the seam and i to s the false hanging. Failure occurred between the local support. In cases such as this adequately designed and installed elongates (closer than the current spacing) could prevent these failures. Elsewhere in the mine this false hanging has been blasted to prevent local failures.
N4	The rib pillar is stress fractured indicating that the regional support is not sufficient in this area. The rib pillar at this locality is between 1-1.5m thick. It is important to realise that the failure of this pillar will result in an increased stress being placed on the adjacent pillars. This could lead to a domino effect failure of many of the pillars.
N5	
N6	In this entire areas there elongate support is damaged. This largely due to 2 factors. The high incidence of inclined joints with weak clay infill and the loosening of the hanging wall due to an impending beam failure.
N7	In this entire areas there elongate support is damaged. This largely due to 2 factors. The high incidence of inclined joints with weak clay infill and the loosening of the hanging wall due to an impending beam failure.
N8	Very Large Beam failure affecting at least one panel, estimated size is 38x100m. The mechanism of failure is the slipping of the entire 1.5m beam on a fault plane that it infilled with 2-20mm of clay and serpentinite. Clay infilled joint surfaces act as release planes. The beam failure has been "stopped" by the rib pillars.
N10	Small hanging wall failure - related to the failure of a key block.
N11	Small hanging wall failure between local instope support (elongates)
S4-1	Well jointed region but the span is narrow enough to prevent failure
S4-2	Well jointed region but the span is narrow enough to prevent failure
S4-3	In the southern region of the mine the incidence of doming is greater and there are numerous small dome failures. This is one of them.
S4-4	In the southern region of the mine the incidence of doming is greater and there are numerous small dome failures. This is one of them.
S4-5	Region of stable ground - with no dome development
S4-6	Very large scale failed dome which comprises several smaller domes. None of the existing rock mass rating systems reflected the likely hood of failure. Therefore the MRMR adjustments, particularly the joint orientation and the induced stress were reduced. To "force" the system to predict the failure.
S4-7	Region of stable ground - with no dome development
S4-8	These small scale domes usually form part of a larger system of domes of varying scale which ride up on each other. Instope pillars are usually cut to prevent the failure of the bigger domes where they are recognised in the stope.
S4-9	It is important to realise that the failure of the small domes are more likely to cause injury because of the higher incidence of occurrence.
S4-10	The inability to reliably identify let alone predict the occurrence of dome structure essentially means that in this situation a classification approach to the problem will not achieve significant success. Therefore a risk assessment approach, which will deal with the issue of doming more successfully, will be used.
S4-11	Region of stable ground - with no dome development
S4-12	Region of stable ground - with no dome development
S4-13	One of the approaches to the doming problem is the change the direction of mining to reduce the distance over which the doming influences the stope. A problem at Dilakong is that the long axis of the domes are oriented in 2 major directions, perpendicular to each other. Therefore even if the breast panels were changed to up dip panels some dome structure would still be oriented disadvantageously to the excavation.
S4-15	Region of stable ground - with no dome development
UL1	Near surface therefore the rock mass is more weathered.
UL2	The weathering is concentrated on the joint surfaces reducing the strength of the joints. The clamping stress at a shallow depth is not significant and consequently the rock mass has a lower overall strength.
UL3	The weathering is concentrated on the joint surfaces reducing the strength of the joints. The clamping stress at a shallow depth is not significant and consequently the rock mass has a lower overall strength.
UL4	The weathering is concentrated on the joint surfaces reducing the strength of the joints. The clamping stress at a shallow depth is not significant and consequently the rock mass has a lower overall strength.
UPL5	The weathering is concentrated on the joint surfaces reducing the strength of the joints. The clamping stress at a shallow depth is not significant and consequently the rock mass has a lower overall strength.
L1-north	The narrow span of the footwall drive in comparison to the span of the stopes, and the good rock mass conditions contribute to the stability of this exaction.
L2	The narrow span of the footwall drive in comparison to the span of the stopes, and the good rock mass conditions contribute to the stability of this excavation.

SUMMARY OF DETAILED MAPPING AT WESTERN CHROME; ELANDSDRIFT (MI

Mine Name	Locality	Site Name	Rock Type	Depth below Surface (m)	Direction of Stoping	Stability	Failure Area (m)	Failure Height (m)	Failure Volume (m^3)	Mass of Failure (Kg)	Width of weak zone (m)	Intact Rock Strength (MPa)	Joint Description: No of sets	Spacing	Joint Orientation: Orientation	Mean Orientation	Range	Std Deviation
Elandsdrift	1	e1	pyroxinite	200	up dip	dome						195	4	4.5	1 subV	J1=68/241	45/226-87/260	6.831
						large FOG									+ 1 incl	J2a=41/352	30/341-48/005	-
															+dome	J3=14/343	06/283-22/025	-
																dome=23/178	09/151-40/216	20.13
Elandsdrift	2	e2	pyroxinite	205	up dip	dome						195	3	1.5	2 subV	J1=81/254	58/246-78/087	2.626
						small FOG									+ 1 incl	J2=47/337	82/295/24/024	185.5
															+dome	J2a=36/176	23/147-55/200	9.074
																J3=13./249	19/258-02/338	-
Elandsdrift	3	e3	pyroxinite	206	up dip	failure						195	2	2.5	2 subV	J1=66/25	52/242-79/26	2.646
																J2=85/187	73/176-89/194	-
Elandsdrift	4	e4	pyroxinite	210	up dip	stable						195	3	3	2 subV	J1=63/250	52/230-74/273	16.26
															+ 1 incl	J2=87/311	74/277-79/169	94.5
																J3=20/245	30/223-11/270	-
Elandsdrift	5	e5	pyroxinite	209	up dip	failure						195	3	5.5	2 subV	J1=75/256	83/247-64/264	-
						FOG									+ 1 incl	J2=80/165	69/157-87/175	-
																J2a=16/008	27/345-04-044	-
																J3=48/002	57/348-38/016	0
Elandsdrift	6	e6	dolerite	221	up dip	failure						195	2	4.2	2 subV	J1=40/069	48/055-30/086	116.7
						fracture zone										J1a=50/235	39/222-58/249	-
																J2=45/157	33/141-56/172	-
Elandsdrift	7	e7	pyroxinite	221	up dip	failure	3x4					195	3	3.7	2 subV	J1=85/256	77/246-75/088	-
						- beam									+ 1 incl	J2=70/161	59/149-83/171	-
																J3=09/314	21/266-03/023	-
Elandsdrift	8	e8	pyroxinite	221	up dip	shear						100	3	4.2	2 subV	J1=81/081	68/065-76/274	7.778
						failure									+ 1 incl	J2=80/157	68/147-89/168	-
																J3=20/225	30/201-08/257	-
																thrust=11/016	22/338-06/063	-
Elandsdrift	9	e9	pyroxinite	195	up dip	small FOG						195	2	2.3	2 subV	J1=69/256	43/243-88/269	-
																J1a=49/075	36/060-61/090	-
																J2=70/152	57/140-85/163	-
Elandsdrift	10	e10	pyroxinite	195	up dip	failure	2x3	0.2				195	3	2.5	2 subV	J1=89/107	78/094-78/301	-
															+ 1 incl	J2=85/008	73/355-80/203	-
																J3=09/351	19/304-02/063	-
Elandsdrift	11	e11	pyroxinite	209	up dip	dome						195	3	3	2 subV	J1=79/77	68/065-88/088	-
															+ 1 incl	J2=70/145	60/1332-82/157	0
															+dome	J3=09/353	18/317-02/038	1
																DOME=08/291	{ 42/253-23/287 }	140.7
																	{ 27/040-08/105 }	
Elandsdrift	12	e12	pyroxinite	140	up dip	failure						195	3	7.3	2 subV	J1=80/072	88/063-67/082	-
						FOG									+ 1 incl	J2=80/197	66/187-90/206	-
																J3=15/337	27/304-06/020	-
Elandsdrift	13	e13	pyroxinite	140	up dip	failure	20	30				195	3	2.5	2 subV	J1=85/103	64/070-88/090	-
						dome									+ 1 incl	J2=80/197	89/188-67/208	-
																J3=14/007	21/326-05/061	-
Elandsdrift	14	e14	pyroxinite	140	up dip	failure				0.7		195	2	7	2 subV	J1=82/253	75/241-89/265	2.121
						fracture zone										J1a=85/346	77/337-88/356	-
																J2=79/203	67/192-87/213	-
																fault=57/239	46/229-67/250	-
Elandsdrift	15	e15	pyroxinite	140	up dip	stable						195	3	2	2 subV	J1=85/103	73/082-81/298	12.02
															+ 1 incl	J2=67/001	57/349-76/014	-
																J3=15/340	26/312-03/020	-
Elandsdrift	16	e16	pyroxinite	152	up dip	dome FOG						195	3	2.6	2 subV	J1=78/251	65/233-82/093	8.578
															+ 1 incl	J2=70/131	77/120-59/143	-
															+dome	J3=15/327	09/310-22/342	-
Elandsdrift	17	e17	pyroxinite	143	up dip	stable						195	3	1	2 subV	J1=70/256	57/240-83/270	-
															+ 1 incl	J2=80/191	68/182-87/202	-
																J3=14/338	27/292-06/026	-
Elandsdrift	18	e18	pyroxinite	144	up dip	stable						195	3	2	2 subV	J1=84/251	74/239-89/265	-
															+ 1 incl	J2=83/154	72/145-89/345	-
																J3=10/345	23/284-01/056	-
Elandsdrift	19	e19	pyroxinite	145	up dip	failure				1.5		195	2	9.1	2 subV	J1=75/269	64/256-87/278	-
						dome										J2=65/349	76/340-53/360	-
Elandsdrift	20	e20	pyroxinite	209	up dip	failure				0.5		195	3	2	2 subV	J1=89/284	80/272-78/114	-
						block									+ 1 incl	J2=89/349	80/160-76/358	-
																J3=23/330	33/305-10/002	-
Elandsdrift	21	e21	pyroxinite	209	footwall	failure						195	3	10	2 subV	J1=71/240	79/226-58/251	-
					drive										+ 1 incl	J2=75/163	83/153-62/173	-
															+fault	J3=09/043	19/351-04/108	-
																fault=70/061	08/052-55/069	-
Elandsdrift	22	e22	pyroxinite	209	footwall	dome						195	3	4	2 subV	J1=67/260	49/241-87/283	9.899
					drive										+ 1 incl	J2=69/172	55/163-80/182	-
															+dome	J3=15/072	04/016-28/119	-
																dome=53/080	37/041-69/110	24.04
Elandsdrift	23	e23	pyroxinite	209	footwall	stable						195	3	1	1 subV	J1=75/224	62/211-88/233	-
					drive										+occasional	J2=69/172	57/160-82/183	-
															J2 and J3	J3=16/032	26/360-05/079	-
Elandsdrift	24	e24	pyroxinite	209	winze	dome						195	1	2.5	1 subV	J1=75/246	59/236-86/257	-
						weathered hw									+dome	dome=15/157	23/123-04/203	-
Elandsdrift	25	e25	pyroxinite	209	winze	failure						195	3	7	2 subV	J1=80/241	66/229-88/256	-
						dome									+ 1 incl	J2=65/009	75/000-53/019	-
																J3=15/072	24/030-03/125	-
Elandsdrift	26	e26	pyroxinite	209	footwall	stable						195	3	0.6	2 subV	J1=81/237	68/227-86/068	-
					drive										+ 1 incl	J2=75/163	64/153-85/173	-
																J3=09/352	20/316-02/053	-
Elandsdrift	27	e27	pyroxinite	209	up dip	dome						195	3	0.5	2 subV	J1=78/240	85/228-65/251	-
															+ 1 incl	J2=82/340	71/341-88/180	-
																J3=10/336	23/302-04/044	-

Site Name	Joint Condition: Infill type	Infill thickness (mm)	Rough. Small Scale	Rough. Large Scale	C	phi	RMR Laubscher	Adjustments: Weathering	Orientation	Induced Stress	Blasting	Total Adjustments	MRMR	Barton (Q)	RMR Bieniawski 1989	GSI	E (S+P) (MPa)
e1	clean,	>1mm	smooth und	planar	6.431	59.53	68	1	0.9	1	0.94	0.846	58	4	77	72	35 481
	clay on J3			dome- curved				1	0.7	0.8	0.94	0.526	36				
e2	clean and	0	smooth und	planar	6.49	59.39	65	1	0.9	1	0.94	0.846	55	8.4	77	72	35 481
	serp			dome- curved													
e3	clean	0	smooth und	planar	5.175	57.61	68	1	0.9	1	0.94	0.846	58	6.3	67	62	19 953
								1	0.8	0.8	0.94	0.602	41				
e4	clean	0	smooth und	planar	5.646	58.17	68	1	0.9	1	0.94	0.846	58	12.6	72	67	26 607
e5	serp	2	smooth und to planar	planar	4.236	54.89	60	1	0.9	1	0.94	0.846	51	1.2	59	54	12 589
								1	0.8	0.85	0.94	0.639	38				
e6	clean	0	smooth planar	planar	5.364	57.13	65	1	0.9	1	0.94	0.846	55	7.2	69	64	22 387
								1	0.8	0.85	0.94	0.639	42				
e7	minor white	1mm	smooth und	planar	4.053	53.16	68	1	0.9	1	0.94	0.846	58	3.2	54	49	9 441
	infill							1	0.8	0.85	0.94	0.639	43				
e8	clean	0	smooth und	planar	5.788	57.83	68	1	0.9	1	0.94	0.846	58	12	72	67	26 607
								1	0.85	0.9	0.94	0.719	49				
e9	clean	0	smooth und	planar	5.788	57.83	65	1	0.9	1	0.94	0.846	55	12.6	74	69	29 854
e10	clean	0	smooth und	planar	6.366	59.69	68	1	0.9	1	0.94	0.846	58	4.2	77	72	35 481
								1	0.7	0.7	0.94	0.461	31				
e11	clean,	0	smooth und	planar	3.646	52.15	62	1	0.9	1	0.94	0.846	52	2.8	49	44	7 079
	some clay on			dome- curved				1	0.8	0.85	0.94	0.639	40				
	J3																
e12	clean	0	smooth und	planar	2.739	54.72	65	1	0.9	1	0.94	0.846	55	8.8	47	42	6 310
	J2 calcite	5						1	0.75	0.8	0.94	0.564	37				
e13	clean	0	smooth und	planar	4.712	60.72	70	1	0.9	1	0.94	0.846	59	6.3	72	67	26 607
				dome- curved				1	0.75	0.85	0.94	0.599	42				
e14	clean, but	2	smooth und	planar	3.22	57.4	59	1	0.9	1	0.94	0.846	50	5.3	57	52	11 220
	with serp							1	0.8	0.85	0.94	0.639	38				
	close to																
	fault																
e15	clean	0	smooth und	planar	4.712	60.76	72	1	0.9	1	0.94	0.846	61	13.9	72	67	26 607
e16	joints clean	0	smooth und	planar	4.214	59.21	68	1	0.9	1	0.94	0.846	58	6	67	62	19 953
	dome: clay	2		dome- curved				1	0.75	0.85	0.94	0.599	41				
e17	clean	0	smooth und	planar	5.688	61.48	72	1	0.9	1	0.94	0.846	61	12.6	77	72	35 481
e18	clean	0	smooth und	und to planar	4.103	59.57	65	1	0.9	1	0.94	0.846	55	31.6	67	62	19 953
e19	clean	0	smooth und	planar	3.354	57.38	56	1	0.9	1	0.94	0.846	47	8	58	53	11 885
				dome- curved				1	0.8	0.85	0.94	0.639	36				
e20	clean	0	smooth und	planar	5.209	57.52	70	1	0.9	1	0.94	0.846	59	12.6	69	64	22 387
e21	joints clean	0	smooth und	planar	3.97	53.88	62	1	0.9	1	0.94	0.846	52	2.5	55	50	10 000
	fault clay	2						1	0.7	0.7	0.94	0.461	29				
e22	clean	0	smooth und	planar	5.209	57.52	65	1	0.9	1	0.94	0.846	55	6	69	64	22 387
				dome- curved				1	0.7	0.7	0.94	0.461	30				
e23	clean	0	smooth und	planar	6.547	59.26	70	1	0.9	1	0.94	0.846	59	25	77	72	35 481
e24	calc & serp	10	smooth und	planar	3.91	53.6	67	1	0.9	1	0.94	0.846	57	6.3	54	49	9 441
				dome- curved				1	0.7	0.75	0.94	0.494	33				
e25	clean	0	smooth und	planar	4.474	55.78	59	1	0.9	1	0.94	0.846	50	8	62	57	14 962
				dome- curved				1	0.75	0.85	0.94	0.599	35				
e26	clean	0	smooth und	planar	6.547	59.26	72	1	0.9	1	0.94	0.846	61	12.6	77	72	35 481
e27	calc & clay	10	smooth und	planar	3.952	55.36	70	1	0.9	1	0.94	0.846	59	3.2	58	53	11 885
				dome- curved				1	0.7	0.7	0.94	0.461	32				

Site Name	Excavation Geometry: Span length	Span width	Height	Hydraulic radius	Stability Index *	Regional Support: Pillar Size	Condition	Local Support: Elongate Spacing	Instope Pillar Size	Pillar Condition	Split sets	Shepherds Crooks	Rock bolts	Associate Geological Structure	Water	Comments
e1	30	28	1.3	7.2	stable	6x16	good		4x3				2x2.5	dome FOG	dry	
					transitional											
e2	30	28	1.3	7.2	stable	6x15	good		4x3				2x2.5	dome	dry	J1 joints discontinuous.
																Pillar cut to support rising domes.
																Small FOG only, therfore hydraulic radius falls in stable zone.
e3	60	28	1.3	9.5	stable	8x30	good		-				1x2	fracture zone	dry	J1 fracture zone. J1 's seem to pinch out on each ot
					transitional											
e4	60	28		9.5	stable	8x30	good		-				2x2.5		dry	
e5	70	30	1.3	10.5	stable				-				2x2.5	fracture zone	dry	Joints open. Serpentinization on joint surfaces.
					transitional											Moderately weathered.
e6	70	30	1.3	10.5	stable				-				1x2	dolerite dyke	dry	Late stage dolerite dyke (large quantity of waste.
					transitional											Tin hat structure formed by J1 and J2 joints.
e7	70	30	1.3	10.5	stable				-				1x2	close to dolerite dyke	dry	Beam failure.
					transitional											
e8	70	30	1.3	10.5	stable				-				2x2.5	shear zone	dry	Shearing has thrust the chrome seam upwards. Sheared material biotite rich.
																Sheared nature of material gives decrease in strength.
e9	10	2	1.3	0.8	stable	see comment			2x2, 2x5				2x2.5	dolerite dykes, pothole	dry	Joints associated with dolerite dyke.
																Updip chromite with pegmatoid floats.
e10	10	2	1.3	0.8	stable	see comment			2x2, 2x5	blast damage			1x2		dry	Mini bean failures 20 cm thick
					transitional											Slight pillar damage most likely due to blasting. Pothole present
																support: at moment support regional as still mining, will be ribs 6 m wide.
e11			1.3		stable				-		sticks		2x2.5	dome	moist	Dome just up dip from instope pillar.
					transitional								2x2.5			Some spalling on the down slope side of the pillar, penetrates 0.5m into pillar (stress fracturing)
																J1's not forming sig. fracture zone. J3's more closely spaced in chrome.
																support: at moment support regional as still mining, will be ribs 6 m wide.
e12	90	25	1.3	9.8	stable	6x15	good		-				1x2	J1 fracture zone	moist	Fracture zone, but no fall of ground recorded
e13	90	25	1.3	9.8	stable	6x15	good		-				1x2	dome FOG	moist	
					transitional											
e14	90	25	1.3	9.8	stable	6x15	good		-				1x2	fault and fracture zone	moist	Fault: brecciated rock, calcite and slickensides, but no apparent displacement of chrome seam,
					transitional											therefore probably strike slip movement. Small FOG's associated with these. Water on fault.
e15	90	25	1.3	9.8	stable	6x15	good		-				2x2.5		wet	Representative of zones between faults and fracture zones.
e16	70	25	1.3	9.2	stable	6x18	good		-				2x2.5	dome FOG	dry	
					transitional											
e17	70	25	1.3	9.2	stable	6x18	good		-				2x2.5		moist	Good, with smooth hanging wall
e18	70	25	1.3	9.2	stable	6x15	good		-				2x2.5		moist	Good
e19	70	25	1.3	9.2	stable	6x15	good		-				1x2	J1 fracture zone	moist	J1 fracture zone, some J2's produce domes.
					transitional											
e20			1.3		stable				-				1x2	local fracture zone	dry	Local J1 fracture zone 0.5 m wide, in foot wall drive.
					transitional											
e21	80	3	1.3	1.4	stable	regional support			-				1x2	fracture zone with micro fault	dry	Blocky zone in this area only, associated with micro faulting.
					transitional											
e22	80	3	1.3	1.4	stable	regional support			-				2x2.5	dome	dry	Other than dome failure, this area is good.
					transitional											
e23	80	3	1.3	1.4	stable	regional support			-				2x2.5		dry	Good smooth roof with widely spaced joints typical of raise area except for localised domes
																and fracture zones.
e24	20	25	1.3	5.6	stable	regional support			-				2x2.5	dome	dry	Domes running from domes found in raise line.
					transitional											
e25	30	30	1.3	7.5	stable	regional support			-				1x2	J2 fracture zone	dry	Pegmatoid in roof decreases hanging wall conditions.
					transitional											
e26	25	15	1.3	4.7	stable	regional support			-				2x2.5		dry	Good conditions
e27	22	15	1.3	4.5	stable				2x2				2x2.5		dry	Very few joints but very continuous white clay infill makes joints very weak.
					transitional											Also large number of domes present.

Appendix D Fault-Event Tree methodology approach to risk assessment

Fault-Event Tree methodology approach to risk assessment

D.1 Introduction

The failure of any system, e.g. a fall of ground in an underground excavation, is seldom the result of a single **cause**, or **fault**. Failure usually results after a combination of faults occurs in such a way that the factor of safety of the system falls to below unity. A disciplined and systematic approach is therefore required to determine the correct logic that controls the failure of the system and to analyse the potential consequences of failure. One such approach, the **Fault-Event Tree Analysis**, is discussed in this appendix.

D.2 Cause/Fault Tree Analysis

Fault Tree Analysis (FTA) is a quantitative or qualitative technique by which conditions and factors that can contribute to a specified undesired incident (called the **top fault**) are deductively identified, organised in a logical manner, and presented pictorially. It can also be defined as a deductive failure analysis, which focuses on one particular undesired fault and which, provides a method for determining causes of the fault.

FTA affords a disciplined approach that is highly systematic, but at the same time sufficiently flexible to allow analysis of a variety of factors. The application of the top-down approach focuses attention on those effects of failure that are directly related to the top fault. FTA is especially useful for analysing systems with many interfaces and interactions.

Starting with the top fault, the possible causes or failure modes (**primary faults**) on the next lower system level are identified. Following the step-by-step identification or undesirable system operation to successively lower levels, **secondary faults, tertiary faults**, etc. are identified.

In order to determine the correct logic that controls the failure of the system, the faults are not initially given probabilities of occurrence. In this form the "tree" is referred to as a "**cause tree**". Once the cause tree is considered to correctly reflect the combinations of faults necessary to result in failure, probabilities are either calculated or assigned to the faults. In this form, the "tree" is referred to as a "**fault tree**".

Thus, a fault tree represents a quantitative or qualitative evaluation of the probabilities of various faults leading to the calculation of the **top faults,** which result in failure of the system.

D.3 Probability evaluation in fault tree

The fault tree is a complex of entities known as **gates** which serve to permit or inhibit the passage of fault logic up the tree. The gates show the relationships of faults needed for the occurrence of a higher fault. **AND** gates and **OR** gates denote the type of relationship of the input events required for the output event.

- **AND** gates are used where faults are statistically dependent. If it is necessary for *n* secondary faults to occur in order for a primary fault to result, then the probability of occurrence, *p*, is represented by:

- ***p[primary fault] = p[secondary fault 1] x p[secondary fault 2] x …x p[secondary fault n]***

- **OR** gates are used where faults are statistically independent. If a primary fault can result as a consequence of the occurrence of any *n* secondary faults, then the probability of occurrence is determined from the calculation as follows:

- ***p[primary fault] = 1 - (1 – p[secondary fault 1]) x (1 – p[secondary fault 2]) … (1 – p[secondary fault n])***

D.4 Event tree analysis

The potential damaging consequences of a top fault are known as **events** and the systematic display of the events is referred to as an **event tree**. The probability of occurrence of a top fault together with relative weighting for the associated potentially adverse events, enable their likely occurrence to be determined. The product of the probability of occurrence and severity of the damage of an event is defined as the **risk**.

The systematic nature of the Fault-Event Tree enables the sensitivities of the potentially adverse consequences to any of the causative hazards to be evaluated. This enables the most threatening causative hazards to be identified and eliminatory measures to be defined.

D.5 Allocation of probabilities of occurrence

Three measures are available for measuring reliability in engineering design, viz:

- the factor of safety;
- the reliability index, and;
- the probability of failure.

The factor of safety is a clearly understood and a numerically sensitive measure. It is, however, not a consistent measure and is not determined in terms of consistent processes. The reliability index is a consistent measure and is based on consistent processes for determining operational values. Its meaning is, however, not clearly understood. It is also not numerically sensitive, especially not with regard to higher orders of reliability.

The probability of failure is a consistent and numerically sensitive measure and is based on consistent processes for the determination of operational values. The numerical sensitivity of the probability of failure, however, detracts from the clarity of its meaning. The probabilities of various kinds of losses of life, property, etc. vary exponentially over many orders of magnitude between very large and very small values. The meaning of such a measure is often difficult to understand.

The difficulties that designers have in selecting acceptable thresholds for probability of failure can be resolved by using the norms and guidelines for selecting acceptable probabilities of failure for design, presented in a paper entitled: “Review of norms for probability of failure and risk in engineering design”, (Kirsten, 1994). The acceptable lifetime probabilities of total loss of life described by Kirsten (1994) are summarised below.

Degree of risk	Acceptable lifetime probabilities (after Cole, 1993)
Very Risky	0,7
Risky	0,07
Some risk	0,007
Slight chance	0,000 7
Unlikely	0,000 07
Very unlikely	0,000 007
Practically impossible	0,000 000 7

In certain cases, probabilities of occurrence could also be determined more accurately by assigning probability density functions to primary faults. This is particularly important in geotechnical engineering designs where input parameters, especially those that are affected by geology, are often not known accurately and the influence of their variability should be accounted for. However, probabilistic analyses of multiple variables require sophisticated numerical techniques that are beyond the scope of this project.

A simplified approach is to assign probabilities based on engineering judgement and past experience with this type of work. Probabilities assigned to certain levels of risk as described in the above table could be used as a guideline. The final result will then show if a more accurate assessment of the probability of occurrence would be necessary. It is likely that the detailed assessment will only be required for key sensitive areas which will be revealed by sensitivity analysis.

It is important to note that probabilities of occurrence may not have unique or discreet values. It is possible for a probability of a particular fault (or event) to change in sympathy with another probability that it is coupled with. This is best illustrated by means of an example:

> *Take the example of a "wrong support installation procedure" being used in an underground excavation. The probability of a wrong support installation procedure being used depends upon the probability that:*

- *the knowledge about the correct support installation procedure is lacking, or;*
- *the equipment being used for support installations is out of order, or;*
- *the discipline and supervision are poor.*

The probability that the knowledge about the correct support installation procedure is lacking in turn depends on the probability that:

- *the support installation procedure is not defined by the mine standards, or;*
- *the support installation procedure is not communicated to the workers, or;*
- *the workers are incompetent.*

The probability that the workers are incompetent depends on the probability that:

- *inadequate training is provided, or;*
- *the workers are untrainable.*

The probability of a wrong support installation procedure being used could be different for different parts or sections of the mine. For example, the equipment being used for support installation in one section could be more reliable than the equipment being used in another section.

The acceptability of probabilities of failure for particular design applications can be evaluated in terms of the magnitudes and distributions of actual frequencies of total losses of life, property and money. For example, the lifetime frequencies of fatalities due to unstable ground in gold and coal mines in South Africa in 1993 amounted to approximately 7,9% and 2,8% respectively (Kirsten, 1994). (These correspond with fatality rates/1000 at work of 0,76 and 0,37 respectively.) According to Cole (1993), an acceptable lifetime probability of loss of life in respect of voluntary employment in underground mines would be 0,7%.

Ground conditions are known to carry potentially high risks and uncertainty. According to Sowers (1993) a study of 500 geotechnical failures revealed that 88 percent of the failures were produced by human shortcomings and that 75 percent of the failures originated in the design process. It is for these reasons that Kirsten (1994) suggested that acceptable levels for probabilities of failure for which designs may be prepared should be significantly smaller than the actual probabilities of failure observed for similar situations.

Appendix E Fault-Event Tree Analysis – Risk of Panel Instability