MT CARBINE BANKABLE FEASIBILITY STUDY

CHAPTER 4: MINING

DECEMBER 2021

Document History

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1. Introduction

1.1. Context

This Chapter 4: Mining shall be read in conjunction with Chapter 1: Executive Summary and additional references as listed in Section 16.

1.2. Purpose

Chapter 4: Mining discusses the approach to mining for the feasibility study. It investigates the optimisation of the mine plan development and equipment and operational strategies to maximise the value of the Mt Carbine ore reserves.

2. Mine Description

2.1. Regional Setting

2.1.1. Location

The Project is located adjacent to the township of Mt Carbine, in the Tablelands Regional Local Government area approximately 60 km NNW of Mareeba.

2.1.2. Climate

The site is subject to two seasons as typical of tropical locations. The dry season and the rain season (November-April). Mean annual minimum and maximum temperatures for Mt Carbine are 18°C to 20°C and 26°C to 28°C respectively. The mean annual rainfall for the Mt Carbine is on average 2,013mm.

2.1.3. Surface Conditions

Most of the lease area is located in the catchment of Manganese Creek which flows to join the Mitchell River approximately 5km from the lease boundary. North-western parts of the lease area drain to the Holmes Creek catchment, and ultimately to the Mitchell River.

The leases are mapped as mainly Hodgkinson Formation arenites, mudstones and cherts, with smaller areas of quaternary alluvial deposits (unit Qa) and mud-silts (unit Qhh).

2.2. Tenure

The Mt Carbine mining area is confined within two Mining Leases, ML4867 and ML4919 totalling approximately 366.39 hectares.

The Mining Leases are 100% percent owned by EQR through its wholly owned subsidiary Mt Carbine Quarries Ptty Ltd.

Tenure is detailed further in Chapter 3: Geology and Resources.

2.3. Site Access

The Mulligan Highway is a sealed highway that runs through the Mining Leases. Site access to the mine from the Mulligan Highway is already established and no further work is required to support the mining activities.

The Mulligan Highway connects to the broader Queensland road network allowing for sealed road access to port facilities and good quality roads for the delivery of goods to the site.

3. Operational Overview

The Mt Carbine mine is a surface operation, with two sources of tungsten ore available – an in-situ open-cut resource and a historical low-grade stockpile. Ore Reserves are 1.26Mt @ 0.71% WO_3 and 10.2Mt @ 0.075% WO_3 respectively.

Extraction from both sources will be undertaken by conventional excavator and truck fleets. Selective ore mining practices will be employed in the insitu open cut, with bulk ore mining of the low grade ore stockpile (LGS) occurring due to local grade variability and lack of historical records.

Ore from both sources will be treated at a dry processing plant prior to concentrate production at the gravity plant. The basic flowsheet for dry processing is as follows:

- All material passes through a Jaw Crusher to -100mm.
- Material is screened at -6mm and -40+6mm (Fines and Ore Sorter material streams respectively).
- +40mm material is crushed and returned to the screen for separation into the Fines and Ore Sorter material streams.
- Fines material stream is crushed to -3mm and prepared for slurry pumping to the gravity plant; and
- Ore Sorter material stream conveyed to the TOMRA COM1200 XRT sorter for processing. Product is trucked to the gravity plant; reject is trucked to the waste dumps.

Through this process, ore grade to the gravity plant is significantly improved, with an associated reduction in mass.

The primary operational constraints are as follows:

- Processing plant capacity of 408kt per annum.
- LGS regulatory approval limit of 1Mt mined per annum; and
- Mobile fleet capacity of 6Mt per annum total material movement.

Based on these constraints a life of mine (LOM) schedule has been developed. Following regulatory approvals, a three-year contract mining operation of the insitu reserves will be undertaken, with supplementary LGS feed via EQR mobile equipment. Upon depletion of the insitu reserves, ore feed will revert solely to the LGS.

It is planned to commence underground mining to supplement the depletion of the open pit insitu ore. This will be the subject of further study and is not considered in this mine schedule.

Table 1 includes the primary physical metrics of the LOM schedule.

Table 1: LOM Primary Physical Metrics

Variable	Unit	Annual Minimum	Annual Maximum	LOM
Total Mined Tonnes	t	786,000	5,999,000	25,201,000
Mined Ore Tonnes	t	783,000	1,003,000	11,382,000
Mined Waste Tonnes	t	10,700	5,129,000	13,819,000
Gravity Plant Feed	t	318,000	408,000	4,774,000
Gravity Plant Head Grade	%	0.18	1.17	0.33
Produced Concentrate	t	950	8,020	26,680

Mobile equipment required for mining operations will comprise of two fleets:

- A primary fleet of 1 x 100-120t class excavator and 5-6 x50t articulated dump trucks. The focus of this fleet will be open cut (OC) waste movement and some LGS ore mining depending on scheduling.

- A secondary fleet of 1 x 50-90t class excavator and 3 x 50t articulated dump trucks, focusing on OC ore recovery and LGS ore mining as required.

The operation will run on a 24hr, 7-day a week roster, with the exception of LGS load and haul requirements which will be 12hr, 7-day shift only. Mine management personnel will be on a standard 5-day work week. Labour requirements and cost per ROM tonne are as per Table 2.

Table 2: Labour Requirements and Cost per Tonne

Group	Number of People	Cost AUD/t
Contract OC Mining	-	4.50 total service contract
Executive & Administrative Team	5	0.60
Operational Management & Technical Services	11	1.43
Total	**16**	**2.03**
Quarry/Mining Supervisor	2	0.17
Excavator Operator	2	0.13
Diesel Fitter/General Maintenance	5	0.34
Articulated Dump Truck Operator	6	0.38
Total	**15**	**1.02**
Crush & Screen Operator/Leading Hand	4	0.47
Ore Sorting Operator	3	0.35
FEL Operator	8	0.47
Total	**15**	**1.28**
Process Plant Operator	12	2.74
Electrician	2	0.54
Total	**14**	**3.28**

The operational costs for each component are summarised in Table 3.

Table 3: Operational Cost by Component

Component	Rate AUD/t	Comment
OC Contract Mining	4.50	All inclusive
Labour (Mining 24hr operations)	0.76	
Labour (Mining 7-day 12hr operations)	0.64	
Labour (ADT Operator - 24hr operations)	0.76	

Component	Rate AUD/t	Comment
Labour (ADT Operator 7-day 12hr operations)	0.38	
Labour (Crush & Screening)	0.47	
Labour (Sorting Plant Operator)	0.35	
Labour (Gravity Processing Plant)	3.28	
Labour (FEL Operator)	0.47	
Excavator Cost - 24hr operations	0.36	
Excavator Cost - 12hr operations	0.27	
ADT Cost - 24hr operations	0.59	
ADT Cost - 12hr operations	0.39	
FEL Cost	0.37	
Crushing & Screening Cost	1.54	
Ore Sorting Cost	0.71	
Gravity Processing Plant Cost	7.73	
Gravity Processing Plant Tailings Cost	0.34	

Primary infrastructure requirements – power, water, processing plant and administration buildings are already on site and functional. Additional operational and maintenance facilities will be required for the contract mining services and have been allowed for in the cost build-up.

Mine capital expenditure is explained in detail in Chapter 12: Capital Cost Estimate. Operational costs are further detailed in Chapter 13: Operating Cost Estimate.

4. Material Characterisation

Material to be extracted at Mt Carbine can be divided into two main lithologies – hornfels and metasediments. The weathered profile of the two rock types varies considerably, the hornfels is ~2-4 metres, whilst the metasediments have a deep (up to 30 metres) weathered profile, particularly adjacent to the South Wall and Iron Duke Faults. As such, weathered metasediments can be classified as a separate 'material type'. Minor occurrences of felsic and andesitic rocks have also been noted as dykes in the western wall of the open pit.

Characteristics of the three main material types is detailed in Table 4.

Table 4: Material Characteristics

Material	Estimated Hardness	Unconfined Compressive Strength (MPa)	Insitu Density	Loose Density	Swell Factor (%)	Contaminants (PAF/NAF)	Comments
Hornfels	R4 to R5	62.5 to 91	2.74	2.28	20	No	
Metasediment	R3 to R4	4 to 91	2.74	2.28	20	Unknown	Likely to slake and desiccate based on field observations
Weathered Metasediment	S4 to R0	0.2 to 7	2.74	2.28	20	Unknown	

A description on estimated hardness is included in Table 5. The below table is an empirical methodology used to validate laboratory testing.

Table 5: Estimated Hardness Methodology

Hardness Type: S = Soil, R = Rock	Description	Comment	Estimated UCS (MPa)
S1	Very Soft	Easily penetrated several inches by a fist.	<0.02
S2	Soft	Easily penetrate several inches by a thumb.	0.02 – 0.05
S3	Firm	Can be penetrated several inches by thumb with moderate effort.	0.05 – 0.1
S4	Stiff	Readily indented by thumb only with a great effort.	0.1 – 0.2
S5	Very Stiff	Readily indented by thumbnail.	0.2 – 0.4
S6	Hard	Indented with difficulty by thumbnail.	0.4
R0	Extremely Soft Rock	Indented by thumbnail.	0.2 – 0.7
R1	Very Soft Rock	Crumbles under firm blow with point of geopick.	0.7 – 6.9
R2	Soft Rock	Shallow indentations made by firm blow of geopick.	6.9 – 27.6
R3	Average Rock	Can be fractured by single firm blow of geopick.	27.6 – 55.2
R4	Hard Rock	Requires more than one blow with geopick to fracture.	55.2 – 110.3
R5	Very Hard Rock	Requires many blows with geopick to fracture.	110.3 – 220.6

The information included in Table 4 and Table 5 is taken from the report *“HD042 Slope Stability Analysis and Design of the Open Pit Slopes”.* Piteau & Associates (1982), which is included in Appendix A.

The other material characteristic of note is the high quartz content of the tungsten bearing veins. The prevention of silicosis (a lung disease caused by inhaling large amounts of crystalline silica dust) has become a priority of Queensland regulatory bodies and the operational management plan will include respirable dust controls. These controls will include specific drill and blast practices and dust suppression through water spraying.

5. Hydrogeology

A series of groundwater bores were drilled around the Mt Carbine local area in 2011, providing a good groundwater monitoring network for the mining operation. Sampling and analysis of the network was undertaken by hydrogeological consultants Rob Lait & Associates, with a report *"Report on Carbine Tungsten Groundwater Study"* delivered in December 2012. The report is included in Appendix B.

The findings of the report are as follows:

- There is low hydraulic conductivity within the Hodgkinson Formation aquifers and minimal groundwater inflow is expected into the open cut pit; and
- Testing of groundwater samples indicates the open cut pit water is better quality than the surrounding groundwater aquifers.

Based on these findings, groundwater is not considered a major risk from either a ground stability or contamination perspective and will be managed via a typical suite of operational controls – pumping, sediment settling dams, dilution, reuse, and approved discharge if necessary.

Accordingly, surface water management, particularly in the wet season is the more relevant issue. Similarly, a standard range of controls, including pit bunding, surface drainage, road design (in particular cambering), pumping and water storage structures will be employed to manage surface water.

Although cyclones are seasonally frequent in the area and can provide short-term issues, overall water management for the site is not considered to be particularly onerous and can be handled at an operational level.

Hydrogeology is detailed further in Chapter 3: Geology and Resources and Chapter 10: Environment and Approvals.

6. Geotechnical

The current pit excavation provides a good opportunity for understanding the future open cut geotechnical performance for the area. Additionally, the underground development has provided further insight into the rockmass condition and has several consultant investigations completed over the years. The previous work has developed a broad understanding, albeit over many decades, during which time changes in geotechnical data collection methodologies and evaluation techniques have evolved.

The dataset provided and reviewed is as follows:

- RQD and defect information from 79 diamond drill holes across deposit;
- Two images and a powerpoint presentation of the above drill hole information; and
- Four geotechnical reports, entitled as follows:
  - GCPL – MC – 160421 – *Preliminary Geotechnical Assessment of Ground Conditions & Remedial Support (2021)*
  - HCOVGlobal – *Brief Review & Structural Assessment/Scoping of Iron Duke – Petersens – Mt Carbine EPM 14872 (2020)*
  - Golder Associates – *Report to R.B Mining Pty. Ltd. On Mt Carbine Mine Review of Rock Mechanics (1984)*
  - HD042 – Piteau & Associates – *Slope Stability Analysis & Design of the Open Pit Slopes (1982).*

Two of the reports (GCPL and Golder Associates) primarily focus on the underground potential of the deposit and leverage off the previous report by Piteau (1982). The HCOV Global report is a preliminary assessment of the structural characteristics present within the mining area and surrounds, with particular emphasis on the Iron Duke prospect. The Piteau & Associates report focuses entirely on the existing open cut and the geotechnical conditions required for stable slope conditions.

Key areas for understanding can be grouped into the following areas:

- Structural defect orientation, persistence and frequency with related defect strength understanding;
- Intact rock strength and weathering characteristics for slope performance;
- Domain rock types and expected geotechnical performance; and
- Groundwater impacts on the rockmass and variability of the medium.

The initial method of assessment is to review the existing data and highlight any critical issues that would materially impact the proposed design.

6.1. Open Cut Mining

6.1.1. Historical Data Evaluation

The Piteau & Associates report is the most comprehensive with reference to open pit mining. An engineering geologist was onsite for approximately six months, collecting and analyzing the following datasets:

- Geological mapping of all accessible open pit benches. Data collection related to lithology, rock strength and geological structures (faults, shears, joints, contacts). Studies of rock competency, degree of fracturing, degree of weathering, rock hardness and bench face angles were also conducted;
- A geotechnical drilling program comprised of two diamond drill holes and eleven percussion drill holes in the area of the south wall. Geotechnical sampling and drill core logging was completed;

- A brief hydrogeological investigation in conjunction with the geotechnical drilling. The study focused on whether there was a need to depressurize the southern wall of the pit to maintain slope stability;
- Strength tests and back analysis of wall failures; and
- Geological structural analysis of the mapped defects.

Based on the collected information and subsequent analysis, Piteau & Associates developed six structural domains with ten associated design sectors. Two separate sets of stability analysis and slope designs were then created – one set for the majority of the pit and the other for walls south of the South Wall Fault.

For the design sectors north of the South Wall Fault, a series of batter angles, bench widths and inter-ramp slope angles were specified, which varied depending on the interrelationship between Relative Level, rockmass lithology and pit geometry.

The slope stability design south of the South Wall Fault incorporated a similar set of design factors, with the addition of mechanical ground stabilisation techniques via grouted cable dowels. To validate the design parameters and effectiveness of the cable dowels, ten pullout tests were conducted at different locations on the south wall.

Table 6 to Table 9 are drawn from the Piteau (1982) report and describe geotechnical information collected during the study. The full report is included in Appendix A.

Table 6: Summary of Unconfined Compressive Strength

Rock Type	**Estimated Average Hardness**	**Drill Hole**	**Depth (m)**	**Uniaxial Compression Tests**		**Point Load Index Tests**	
				UCS (MPa)	Average UCS (MPa)	UCS Parallel Schistocity (MPa)	UCS Normal to Schistocity (MPa)
Fill & residual soil	S4 to R0	-	-	-	-	-	-
Highly weathered schist & greywacke	R1	CB20	24.0	7.04	5.6	-	-
			26.1	4.60			
			28.5	4.53			
			28.7	6.01			
		CB3	16.8	-	-	5.76	
Moderately weathered schist & greywacke	R2	CB20	-	6.48	10.44	-	-
		CB21	4.0	14.4			
Unweathered greywacke	R3 to R4	CB20	51.0 to 56.0	-	-	-	93.4
		CB20	77.0 to 81.0	-	-	29.0	59.8
		CB3	54.3 to 60.4	-	-	-	54.0
Unweathered schistose greywacke	R3 to R4	CB20	122.0 to 127.0	-	-	28.1	73.4
		CB20	131.0 to 137.0	-	-	3.7	91.2
Unweathered hornfels	R4 to R5	CB3	71.0 to 77.13	-	-	-	62.4

Table 7: Orientation of Fault and Shear Sets in the Open Pit

Structural Domain	Total Population	Shear Set SR1			Shear Set SR2		
		Dip Dir.	Dip	%	Dip Dir.	Dip	%
1,2 & 3	9	355	72	>35	-	-	-
4	31	043	33	11	038	70	19
5	35	047	37	14	032	73	13
6	10	-	-	-	-	-	-
4,5 & 6	75	045	35	11	035	72	12

Notes: 1. Orientation is given in terms of dip direction and dip of the peak or average orientation of the discontinuity set.

2. Percent refers to the percent concentration with one percent area of the lower hemisphere for the peak or average orientation of the population.

Table 8: Summary of Back Analysis Results for Slopes South of the South Wall Fault

Slope Height (m)	Dry Slope	Maximum Slope Angle for F = 1.2	
		Moderate Groundwater	High Groundwater
10	60	58	47
15	48	40	31
20	41	29	25

Table 9: Summary of routed Dowels & Pull-out Test Results

Dowel No.	Location (m)			Orientation		Length (m)	Grouted Length (m)	Est. Rock Hardness	Location In Final Wall	Jack Force @ Failure (t)
	North	East	Elev.	Dip Dir.	Incl.					
1	26226.6	22887.1	380.8	160	-30	10	10	R0	1st bench	67
2	26202.8	22804.3	389.7	183	-30	10	10	S5	Top	48*
3	26212.8	22862.3	384.8	172	-30	20	20	R0	Top	64
4	26216.6	22856.2	381.2	171	-30	20	20	1.5	1st bench	70
5	26212.9	22837.2	381.5	171	-30	10	10	R1	1st bench	77
6	26215.3	22807.8	380.5	189	-30	10	10	R1	1st bench	71
7	26228.4	22958.4	379.5	159	-30	20	10	R0	1st bench	76
8	26241.1	22874.2	371.0	174	-30	10	10	R2	2nd bench	71
9	26246.1	22899.1	370.7	181	-30	20	20	R2	2nd bench	76
10	26255.9	22954.6	370.9	173	-30	10	10	R3	2nd bench	78

6.1.2. Current Data Evaluation

Defect and RQD logging of the 79 diamond drill holes (over 20,000 metres of core) was completed by EQ Resources geological personnel. The data was compiled into a three-dimensional model in Leapfrog software and corresponds well with historical fault, shear and fractured zones.

The majority of the fractures observed are associated with the South Wall Fault, being found in the 10-15m zone of foot wall. The South Wall Fault is well exposed in the existing pit and has over eighty intersections recorded in exploration drilling to date. It varies from 0.5 to 2.0m in thickness and is marked by a clay filled fault gouge.

Figure 1 and Figure 2 illustrate the defect logging completed and alignment with historical defect mapping.

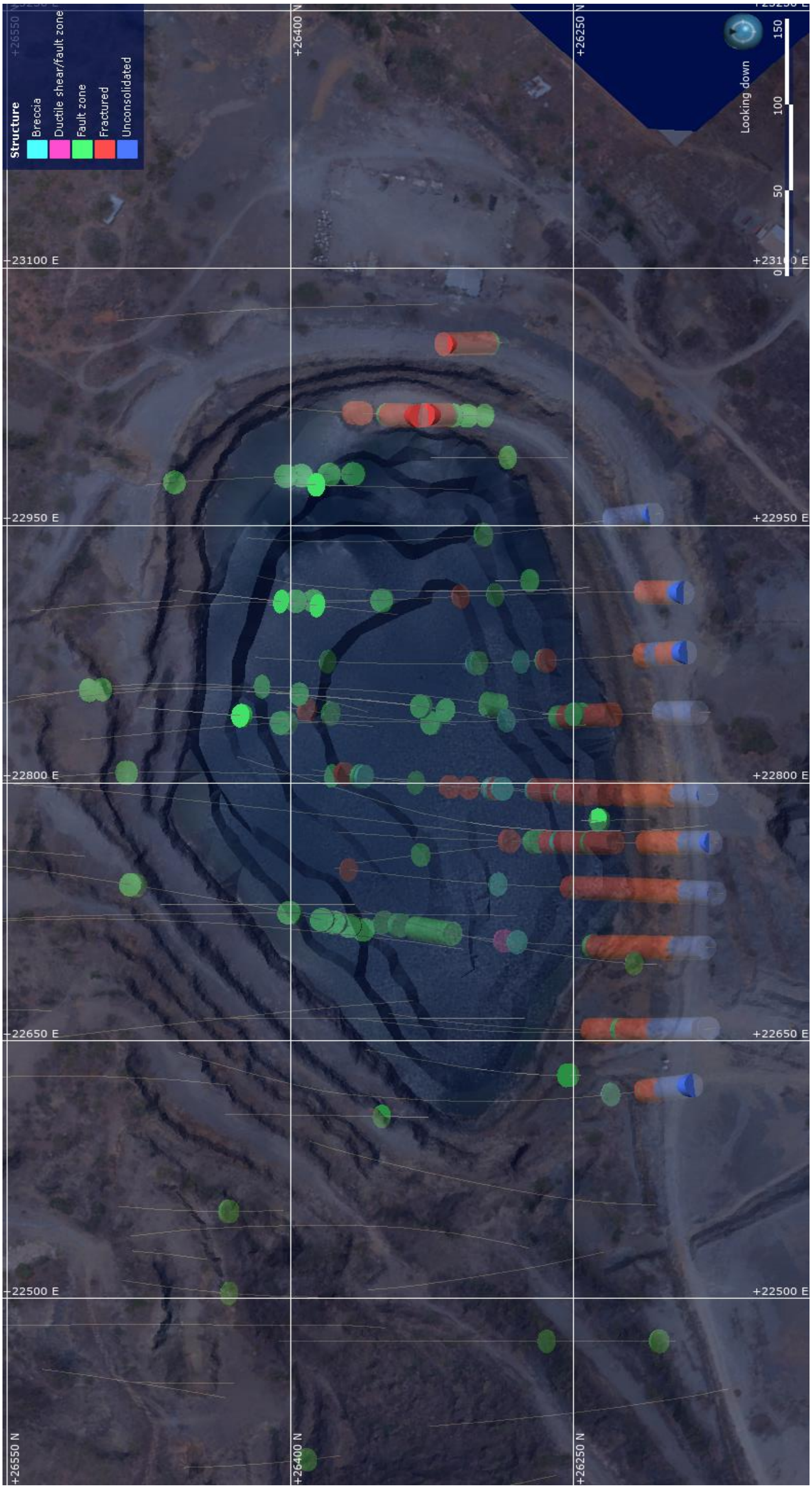

Figure 1: Defect Logging and Structural Model of Exploration Drill Holes (plan view)

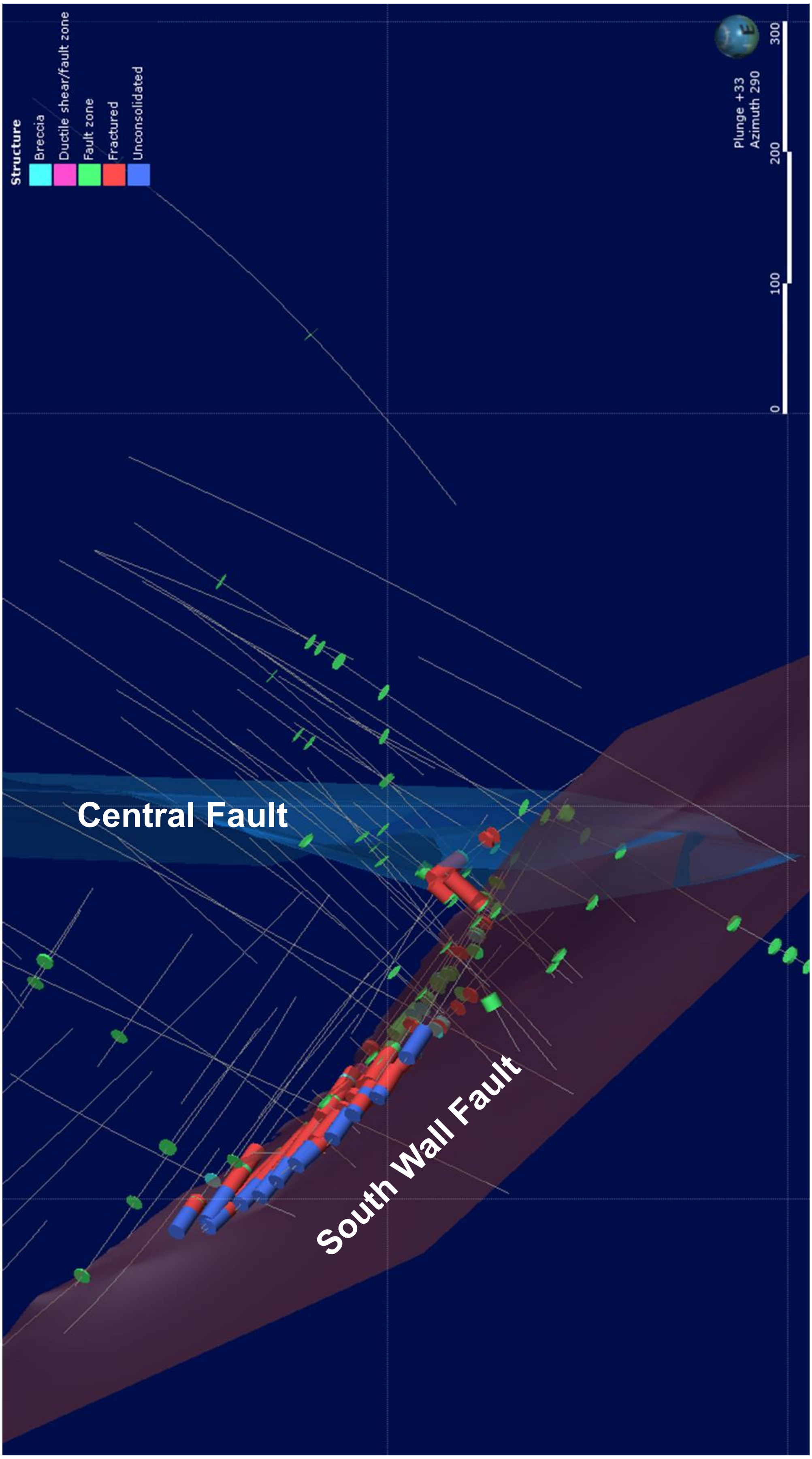

Figure 2: Defect Logging and Relationship to Known Faults (Oblique View Looking West)

6.1.3. Current Pit Design

Critical data items related to the current pit design include:

- The South Wall Fault (SWF) intersects the current pit and the expanded pit design in the south wall. Details include dip/dip directions of 75/345 and >300m offset between the metasediments and host rock volcanic. The structure has unfavorable orientation dipping into the pit and is understood to be a significant shear zone with low contact strength;
- The C1 joints set has shallow dips of 15-30 degrees toward the east and has potential to form a sliding mechanism for the western wall of the pit within the Hornfels. Combinations of this feature with additional structures will create kinematic potential for block/wedge failure formation within the batter slopes;
- The foliation within the metasediments creates an intense, vertically orientated defect through the footwall rockmass south of the SWF. The SWF forms a silicification boundary resulting in a severe reduction in strength south of the structure;
- The Iron Duke Fault (IDF) also appears to act as a boundary for silicification and creates a reduction in hardness east of the feature;
- The weathering profile is much deeper in the southern wall (Metasediments) than the Hornfels (Host rock). The SWF will provide a stepped boundary for the weathering and therefore vary the intact strength. The weathering appears to extend up to 30m below surface within the metasediments and has varying strength with depth;
- The Hornfels are extremely hard and have performed well over time within the current pit exposure;
- Felsic and Andesitic dykes have been reported on the western and eastern boundary of the pit. Dykes have contact boundaries which can serve as weakness planes for failure modes if aligned with pit geometry. There is often a strength change in the material, especially within the weathered zone;
- Six main structural domains have been identified (Piteau 1982) and detail the local structure. Key takeaways are:
    - The continuous nature of joint set B that has persistence over several benches and is the most dominant feature throughout the excavation. Dip/Dip direction of 80/324-360;
    - Joint set G is a low angle feature that can be continuous and may propagate planar failure. Dip/Dip direction 50/340-090; and
    - The joints classed as B and G will most likely dictate the batter bench design required for the pit.
- Material and defect strength tests have been completed within (Piteau 1982) and provide a good basis for the design analysis input values. A record of back analysis from previous failures also provides a refinement of the strengths for use within the mine; and
- Groundwater south of the SWF is higher than groundwater north of the fault. This indicates the SWF forms a type of barrier. Surface water appears to have significant connectivity with the ground south of the SWF. Drainage options for the rockmass maybe required, especially for the southern wall.

The key element of the current pit design is the requirement for mechanical ground stabilisation on the southern wall, behind the South Wall Fault. The ground stabilisation is based off the work completed by Piteau (1982) and comprises the following:

- Horizontal groundwater drainage holes (up to 20m long) at the base of each bench, with associated drainage channel.
- Vertical 10m twin strand cable bolts, two rows at 3 metres spacing, above the 380RL.
- Inclined (-10^0) 10m twin strand cable bolts, four rows at 3 metres spacing, above the 360RL.

- Inclined (-10^0) 10m twin strand cable bolts, six rows at 3 metres spacing, above the 340RL
- Below the 340RL, 10m twin strand cable bolts, five rows at 3 metres spacing for all benches
- Cable bolt loading above the 340RL is 20 tonne, below the 340RL is 50 tonne.

Installation of the grouted cable bolts will be undertaken in conjunction with mining. On every second mining lift a drill rig will drill the required stabilisation pattern (76mm dia) for that bench. Following drilling, the cable bolt will be inserted, grouted into position and tensioned to the appropriate load. Given the high-risk nature of the activity, specialist equipment such as a radio remote controlled drill rig, telehandler with work basket and overhead guard will be deployed.

Geotechnical parameters of the current pit design and ground stabilisation requirements are included in Table 10 and Table 11.

Table 10: Geotechnical Parameters of Open Cut Pit Design

Parameter	Value	Comment
Bench Height (m)	20	Suitable in hornfels material north of the South Wall Fault, see Table 11 for mechanical ground stabilisation requirements south of the fault.
Bench batter angle (^{0})	70	
Bench width (m)	8	
Ramp angle (%)	10	
Inter-ramp slope angle (^{0})	**70**	

Table 11: Ground Stabilisation Requirements for Southern Pit Wall

Toe Elevation (m)	Length of pit wall (m)	Area (m^2)	Horizontal Spacing	No. of Rows	Min. bolt length (m)	No. of bolts	Bolting metres (m)
380	261	2463	3	2	10	174	1740
360	491	9820	3	4	10	655	6547
340	453	9060	3	6	10	906	9060
320	413	8260	3	5	10	688	6883
300	349	6980	3	5	10	582	5817
280	333	6660	3	5	10	555	5550
260	270	5400	3	5	10	450	4500
240	211	4220	3	5	10	352	3517
220	51	1020	3	5	10	85	850
Total	**2832**	**53,883**				**4446**	**44,463**

6.2. Low Grade Stockpile & Waste Rock Dumps

The LGS was constructed in two lifts. At maximum height (NW end), each lift is approximately 15m, at a batter angle approaching 37 degrees. Empirically, the duration of the LGS (almost 40 years) and the lack of any observed instability indicates a high degree of confidence in these design parameters for the rockmass.

Mining benches of the LGS have been designed at 4m high, a minimum of 25m wide and with a nominal 55 to 60-degree inter-bench batter angle. Notwithstanding the potential for minor rilling of the batter angle over time, the overall slope profile of this design is significantly shallower than the existing LGS profile.

Accordingly, the risk of geotechnical failure within the LGS is considered extremely low.

Design parameters for the waste rock dumps have been modelled on the LGS. Dump lifts are 15m in height, with a 20m set back between lifts and a batter angle of 37 degrees. Even allowing for the weaker material characteristics of the metasediment, this design provides a more than adequate factor of safety for the waste rock dumps.

6.3. Key Risks

6.3.1. Low Grade Stockpile and Waste Rock Dumps

No material risks have been identified for the LGS or waste rock dumps. Standard operational geotechnical controls (inspections, surface water management and dig/dump to design compliance) will be sufficient to manage any potential hazards.

6.3.2. Open Cut Mining

No significant risks have been identified for the northern, western, and eastern walls within the Hornfels rockmass. There remains potential for localised failure due to defect orientation but can be managed at an operational level with further detailed data collection and evaluation.

The expansion of the pit will expose a much greater proportion of metasediments south of the SWF. These materials will be weaker and more susceptible to failures developing; as such they constitute a primary risk to the current mine design and production schedule (Figure 3: South Wall Fault Through the Proposed OC Pit).

The current pit is located predominantly within Hornfels material, which is the strongest rockmass, but does include metasediment in the southern wall. As the majority of potential failure modes are kinematically controlled further data collation and analysis prior and during extraction is recommended, in particular for the southern wall to ensure the validity of the mechanical ground stabilisation program.

Currently there is only a basic understanding of how reduction in silicification away from fault zones will impact on metasediment material and defect strength at a localised scale and the associated slope performance. A conservative position on ground stabilisation has been taken to account for this, but further data collection and analysis is required to optimize the program.

6.4. Future Work

6.4.1. Objectives

A greater understanding of the following geotechnical parameters is required prior to operational pit design:

- Metasediments:
    - Structural dataset – defects within the rockmass both proximal and distal from the SWF and IDF;
    - Material strength parameters to allow for geotechnical analysis – again with spatial reference to the SWF and IDF; and
    - Groundwater – specifically porosity and permeability.
- Hornfels:
    - Further structural and material strength data to complement the existing historical dataset; and
    - Groundwater – porosity and permeability.

- Further delineation of the South Wall and Iron Duke Faults – spatial location and material strength parameters.

A localised understanding of the groundwater parameters for the metasediments (porosity and permeability) will inform the specific required drainage requirements for the pit and facilitate the cost and process involved to implement remedial works.

6.4.2. Activities

Investigations should include exploration drilling using a combination of orientated core and open holes to build an understanding of the rockmass. The drilling program should be designed to adequately represent the spatial variability of the rockmass both proximal and distal from the SWF and IDF.

Orientated core holes would be used for the following:

- Structural defect data collation (including RQD); and
- Samples for laboratory material testing (UCS, defect direct shear, slaking).

Open holes would be used to increase the definition of fault locations, material boundaries and silicification zones. Downhole geophysical tools such as Acoustic Televiewer and Sonic should be run on all holes to provide a secondary geotechnical dataset.

Specific drill holes for groundwater parameter testing can be selected following the initial drilling campaign and should be undertaken by a groundwater specialist.

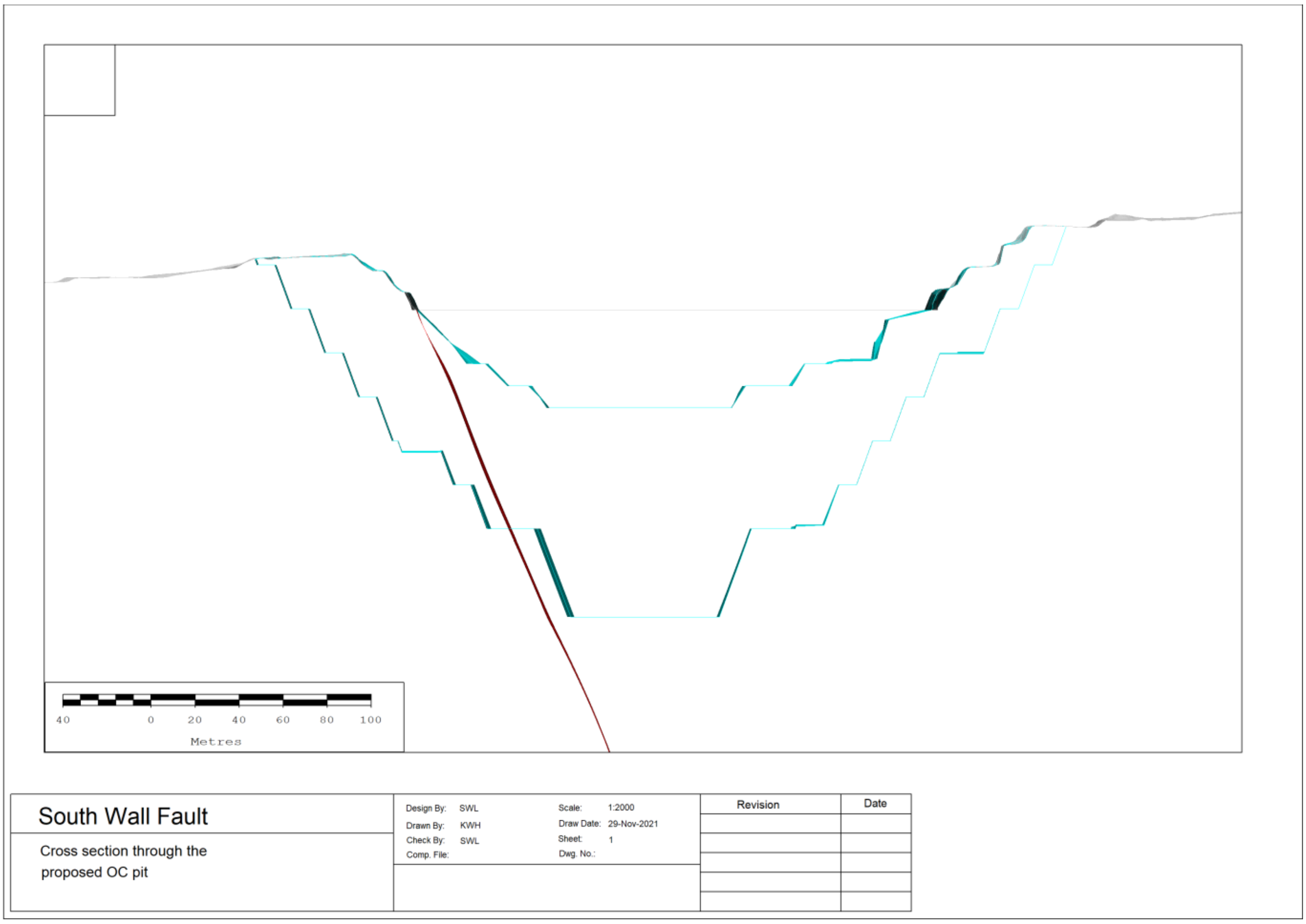

Figure 3: South Wall Fault Through the Proposed OC Pit

7. Resource Model

The two geological models used to develop the final resource model were generated by Measured Group Pty Ltd in August and September 2021 and are titled 'Mt_Carbine_LGS_20210820.bmf' and 'Mt_Carbine_20210918.bmf'.

Development of the Mt Carbine mineable resource was driven by three key factors:

- Utilising all the gravity processing plant capacity to realise maximum revenue for the Project;
- Maximising the amount of high-grade ore from the insitu resource to the gravity processing plant; and
- Handleability and operational efficiency of the LGS material through the mining, crushing & screening circuits, given the 1m+ top size of the material.

The mineable resource model consists of two components – the insitu open-cut resource, and the existing low grade stockpile resource.

7.1. Low Grade Stockpile Mineable Resource

A series of production scale trials were completed on site with different top sizes – 100, 170, 200 & 500mm. Operational efficiencies of the mining, crushing and screening circuits were monitored and compared against the feed/product grades of the ore sorter and gravity processing plant. Based on these trials a top size of 500mm was chosen as optimum for operational efficiency without significantly impacting on tungsten recovery.

The results of the site trials were cross-referenced against detailed geological characterisation activities undertaken as part of a METS Ignited Project to ensure validity in the top size selection. In terms of cut-off grade, the 500mm top size equates to 0.074% WO_3, with only ~5% of the LGS (by mass) greater than 500mm top size.

Error! Reference source not found.12 in Section 7.2 details the available ore. The LGS resource comprises two blocks with ten sub-blocks, based on block number and toe RL value. Block 1 encompasses the south-eastern portion of the LGS and contains 44% of the mineable resource. Extraction commences in Block 1, where mining passes are taken from top to bottom, creating a void for later waste emplacement (Dump 3).

Block 2 contains approximately 5.57Mt, with extraction following the mining out of Block 1 in the same top-down approach.

Sufficient geological information was collected to generate an Indicated confidence level (JORC 2012) for the LGS resource. Conversion of the geological resource into a mineable resource is achieved through identification and analysis of any modifying factors (technical, environmental, cost and revenue drivers) that may impact on the economic extraction of the resource.

Modelling of the cost and revenue drivers was undertaken to determine financial sensitivities and resource viability. A positive return was generated when mining and processing all the LGS, hence the constraints on top size and mineable resource were technical, primarily focused on material handleability and equipment maintenance/reliability.

7.2. In-situ Open-cut Mineable Resource

The mineable resource model for the open cut is based on the 'Mt_Carbine_20210918.bmf'. This model contains a series of relatively thin (generally <2m wide) high-grade tungsten zones. As a result of the geological mineralising events, the contact between ore and waste material is generally very sharp, allowing for clear demarcation of the resource and a cut-off grade of 0.2% WO_3. Where possible, high-grade zones were aggregated in the resource model into a minimum mining thickness of 2m, incorporating intra-zone dilution.

Development of the insitu mineable resource was relatively straightforward due to the following primary attributes:

- The clear demarcation of ore and waste in the resource model;
- The resource model cut-off grade of 0.2% WO_3; and
- Successful ore-sorting of the LGS at 0.075% WO_3 grade demonstrating an effective process for insitu ore treatment and metal recovery.

Other factors included in the insitu mineable resource model were:

- Mobile fleet equipment matching to allow for a degree of selective mining. Whilst the dry processing plant and ore sorting unit negate the need for highly accurate selective mining, equipment matching was undertaken to ensure high productivity of both the mobile fleet and dry processing plant
- The ore zones include a component of dilution, as the thin, high-grade tungsten veins were artificially thickened to 2 metres (downhole width) and the WO_3% grade adjusted accordingly

No further dilution or loss factors were incorporated into the mineable resource (this was completed during the reserving stage).

The mineable resource model was developed using the following process:

1. The in-situ resource model and a series of revenue/cost inputs were passed through Maptek Vulcan Pit Optimiser software to deliver a number of cash-positive pit shell options; and
2. The preferred pit shell was then converted into a practical mining shell, by incorporating ramps, sufficient equipment working room and initial bench and batter designs.

Revenue, cost and mining parameter inputs for the pit shell optimiser process are detailed in Section 10. The insitu mineable resource is as per **Error! Reference source not found.**13.

During this process it was identified that an improved geological model would provide significant future upside. A number of the high-grade tungsten zones currently within the pit shell are laterally discontinuous, simply through insufficient drilling. Accordingly, an infill drilling program would likely increase the JORC resource and reserve base within the existing pit shell and immediate surrounds, improving overall Project economics.

Table 12 - LGS Mineable resource

Source (Triangulation Name)	Phase No.	Mining Seq.	Toe RL (m)	Volume (bcm)	Bulk Density	Ore Tonnage (t)	WO3 Ore Grade (%)	Tungsten Tonnes (t)
LGSP_RL_370_1_local.00t	1	1	370	56,707	1.60	90,731	0.075	68
LGSP_RL_374_1_local.00t	1	2	374	283,485	1.60	453,575	0.075	340
LGSP_RL_378_1_local.00t	1	3	378	401,238	1.60	641,981	0.075	481
LGSP_RL_382_1_local.00t	1	4	382	496,647	1.60	794,636	0.075	596
LGSP_RL_386_1_local.00t	1	5	386	372,966	1.60	596,746	0.075	448
LGSP_RL_390_1_local.00t	1	6	390	367,442	1.60	587,907	0.075	441
LGSP_RL_394_1_local.00t	1	7	394	350,423	1.60	560,676	0.075	421
LGSP_RL_398_1_local.00t	1	8	398	269,038	1.60	430,461	0.075	323
LGSP_RL_402_1_local.00t	1	9	402	165,861	1.60	265,377	0.075	199
LGSP_RL_406_1_local.00t	1	10	406	7,771	1.60	12,433	0.075	9
LGSP_RL_370_2_local.00t	2	11	370	110,794	1.60	177,270	0.075	133
LGSP_RL_374_2_local.00t	2	12	374	422,336	1.60	675,738	0.075	507
LGSP_RL_378_2_local.00t	2	13	378	566,492	1.60	906,387	0.075	680
LGSP_RL_382_2_local.00t	2	14	382	611,562	1.60	978,500	0.075	734
LGSP_RL_386_2_local.00t	2	15	386	500,781	1.60	801,250	0.075	601
LGSP_RL_390_2_local.00t	2	16	390	427,390	1.60	683,823	0.075	513
LGSP_RL_394_2_local.00t	2	17	394	339,085	1.60	542,536	0.075	407
LGSP_RL_398_2_local.00t	2	18	398	297,175	1.60	475,480	0.075	357
LGSP_RL_402_2_local.00t	2	19	402	190,462	1.60	304,740	0.075	229
LGSP_RL_406_2_local.00t	2	20	406	12,790	1.60	20,465	0.075	15
Total				**6,250,445**		**10,125,710**		**7,502**

Table 13 - Insitu mineable resource

Source (Triangulation Name)	Phase No.	Mining Seq.	Toe RL (m)	Volume (bcm)	Bulk Density	Ore Tonnage (t)	WO3 Ore Grade (%)	Tungsten Tonnes (t)
Pit17_6_Phase3_Solid.00t	3	1	300	33,370	2.74	91,433	0.94	864
Pit17_6_Phase3_Solid.00t	3	1	320	2,817	2.74	7,719	0.62	48
Pit17_6_Phase4_Solid.00t	4	2	220	18,388	2.74	50,384	0.81	406
Pit17_6_Phase4_Solid.00t	4	2	240	32,074	2.74	87,882	0.88	776
Pit17_6_Phase4_Solid.00t	4	2	260	71,862	2.74	196,901	0.91	1,796
Pit17_6_Phase4_Solid.00t	4	2	280	100,766	2.74	276,098	0.89	2,446
Pit17_6_Phase4_Solid.00t	4	2	300	69,019	2.74	189,111	0.80	1,512
Pit17_6_Phase4_Solid.00t	4	2	320	38,221	2.74	104,727	0.69	726
Pit17_6_Phase4_Solid.00t	4	2	340	20,029	2.74	54,878	0.59	322
Pit17_6_Phase4_Solid.00t	4	2	360	10,285	2.74	28,182	0.50	140
Pit17_6_Phase4_Solid.00t	4	2	380	3,657	2.74	10,021	0.59	59
Pit17_6_Phase4_Solid.00t	4	2	400	218	2.74	597	0.36	2
Total				**400,706**		**1,097,933**		**9,099**

8. Mine Production

8.1. LOM Mining Schedule

A LOM Mining Schedule was developed on the existing JORC Reserves from the LGS and insitu orebody. The considerable inferred resources in the in-situ orebody were excluded from the schedule.

Key drivers for mining schedule development were:

- Utilising all the gravity processing plant annual capacity to realise maximum revenue for the Project; and
- Optimising the volume and timing of high-grade ore from the in-situ orebody to the processing plant.

A number of scenarios with ranged input variables were analysed to deliver an optimised LOM mining schedule. Details of the scenario analysis are included in Section 10.

The LOM Schedule consists of three main components:

- One year of mining from the LGS, to allow for open cut regulatory approvals, infrastructure upgrades and mining contractor award and mobilisation.
- Three years of insitu open cut mining to deplete the current JORC Reserves, with supplementary feed from the LGS to maximise gravity processing plant throughput.
- Approximately eight years of mining to deplete the remaining LGS reserves.

Key physical metrics are included in **Error! Reference source not found.**14. The LOM Schedule with all p hysical metrics is detailed in Appendix C.

Due to the insitu orebody shape (tungsten grade and width increasing with depth), the initial mining benches contain minimal ore and a resultant high strip ratio. Strip ratio balancing was achieved by extraction of a high-grade ore zone (HGZ) immediately below the historic pit floor (325-300RL), in conjunction with the larger pit development at higher elevations.

Following early extraction of the HGZ, pit development adheres to a conventional top-down approach, with pit floor reached at 220RL.

Production scenarios were developed using Comet Strategy value optimisation software, with the main constraints being mobile fleet capacity, particle ore sorting capacity and gravity processing plant capacity. No constraint on the gravity processing plant head grade was applied.

As the open cut mining is to be a contract operation, particular emphasis was placed on delivering a schedule with consistent year-on-year physical metrics. Scenarios with varying mobile fleet capacities (4Mtpa, 5Mtpa, 6Mtpa) were scheduled and analysed against the following criteria:

- Overall Project cashflow;
- Volume and year on year consistency of ore flow to the gravity processing plant; and
- Year on year consistency of total material movement.

The base case for equipment selection was the existing EQR mobile fleet of 90t excavator and 50t articulated dump trucks for the following reasons:

- Small footprint and good performance in tighter working areas;
- Utilisation flexibility between the LGS and open cut; and
- Simplification of fleet maintenance (and associated infrastructure) requirements.

The equipment productivities required to achieve the Mining Schedule are detailed in Section 9.2.

Table 14: Key Physical Metrics

Variable	Unit	Annual Minimum	Annual Maximum	LOM
Total Mined Tonnes	t	786,000	5,999,000	25,201,000
Mined Ore Tonnes	t	783,000	1,003,000	11,382,000
Mined Waste Tonnes	t	3,634,000	5,129,000	13,819,000
Fines Ore Material	t	282,000	361,000	4,098,000
Fines Grade	%	0.11	0.96	0.22
Bypass Ore Material	t	1,030	3,750	6,410
Bypass Grade	%	2.74	2.90	2.83
Ore Sorter Feed	t	497,000	642,000	7,278,000
Ore Sorter Product	t	35,210	122,130	670,000
Ore Sorter Product Grade	%	0.69	1.61	1.00
Ore Sorter Waste	t	375,000	597,000	6,608,000
Gravity Processing Plant Feed	t	318,000	408,000	4,774,000
Gravity Processing Plant Head Grade	%	0.18	1.17	0.33
Produced Concentrate	t	950	8,020	26,680
HGZ OC Mined Tonnes	t	15,150	787,000	802,000
HGZ OC Ore Tonnes	t	4,060	109,900	113,900
HGZ OC Strip Ratio	-	3.7	7.2	7.0
HGZ OC Insitu Ore Grade	%	0.63	0.79	0.78
OC Mined Tonnes	t	14,400	5,534,000	14,273,000
OC Ore Tonnes	t	3,670	636,000	1,150,000
OC Strip Ratio	-	3.9	47.0	12.4
OC Insitu Ore Grade	%	0.48	0.77	0.71
LGS Mined/Ore Tonnes	t	150,000	1,000,000	10,125,000
LGS Insitu Ore Grade	%	0.075	0.075	0.075

The 6Mtpa mining scenario was chosen as it best met the assessment criteria of Project cashflow, consistency of material movement and ore flow to the gravity processing plant. For open cut mining a contract fleet of approximately 4.5Mtpa capacity is recommended, with the EQR mobile fleet utilised for LGS and remaining open cut material movement requirements.

Key outputs from Figure 45, Figure 56 and Figure 67 are as follows:

- Strip ratio balancing through early extraction of the HGZ delivers a 2023 OC strip ratio of 25:6 and positive operational cashflow;

- As ore mass and grade increases with OC depth, there is a commensurate reduction in LGS material. In 2025 the least volume of ore is mined (636kt OC and 150kt LGS) yet delivers the most concentrate tonnes (37% more than the next best year) due to the high grade of the ore; and
- Infill drilling both within and immediately adjacent to the pit shell would likely allow for multiple mining sequence options, facilitating further strip ratio and gravity processing plant head-grade balancing.

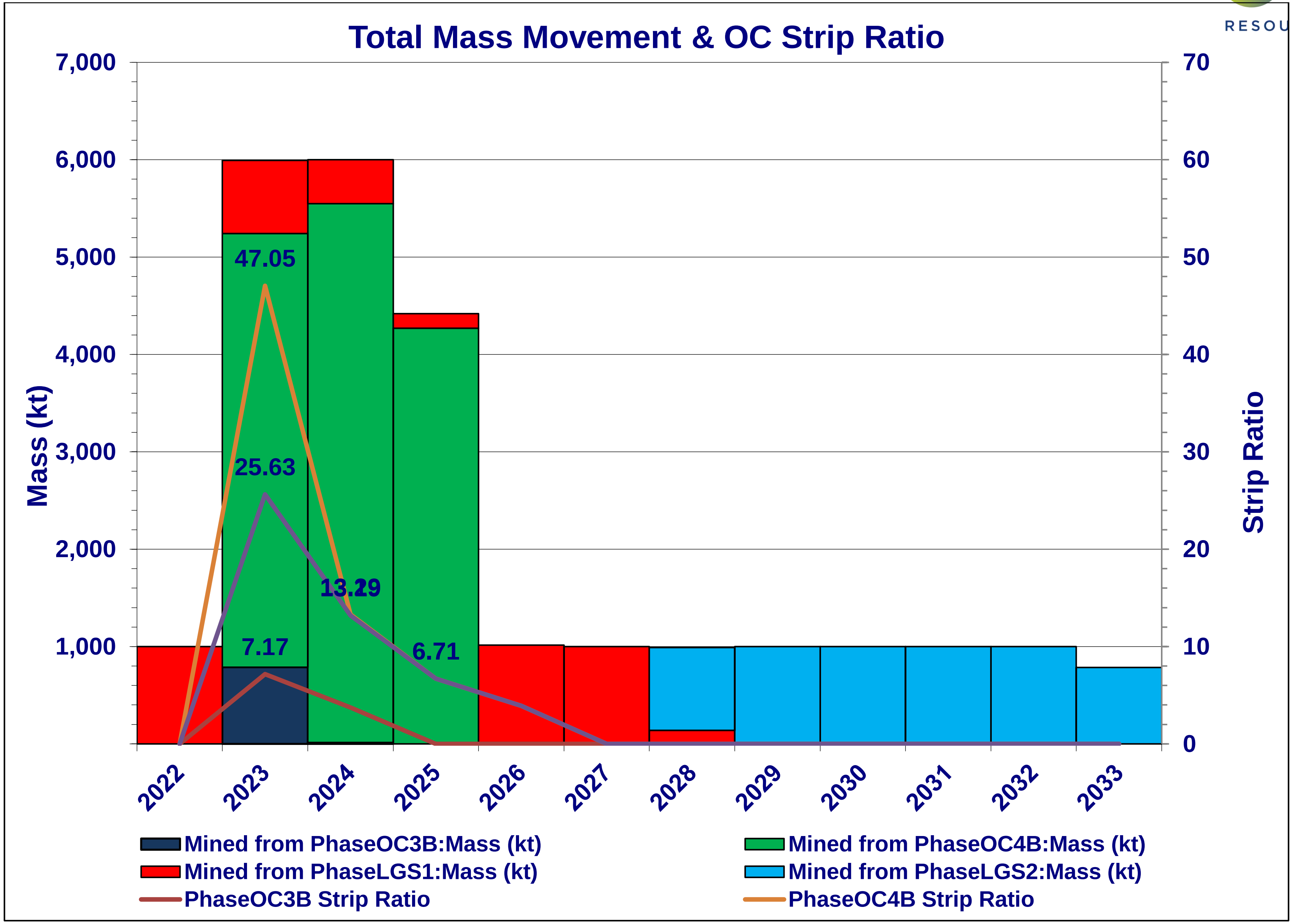

Figure 4: Total Mass Movement & OC Strip Ratio

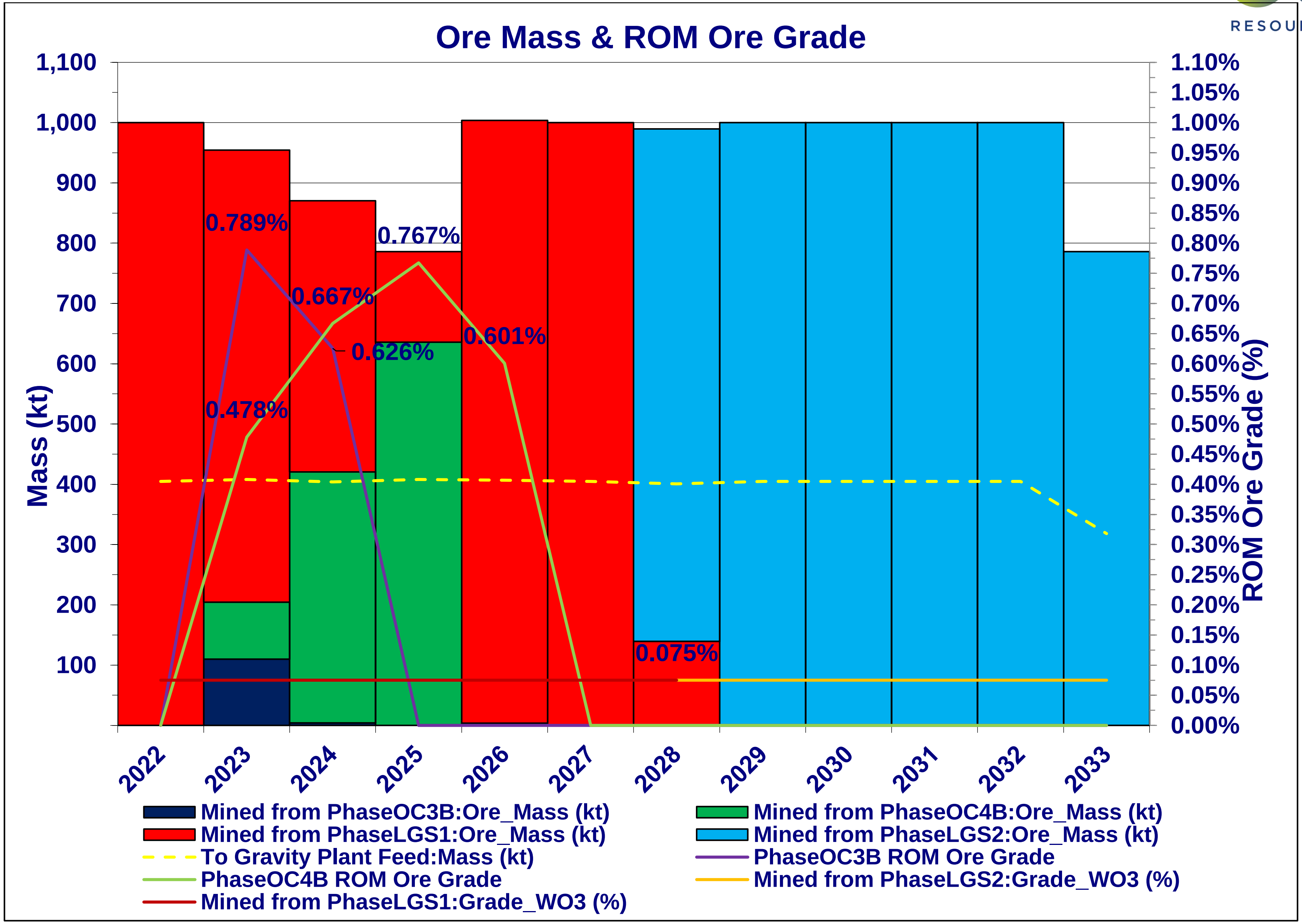

Figure 5: Ore Mass & ROM Ore Grade

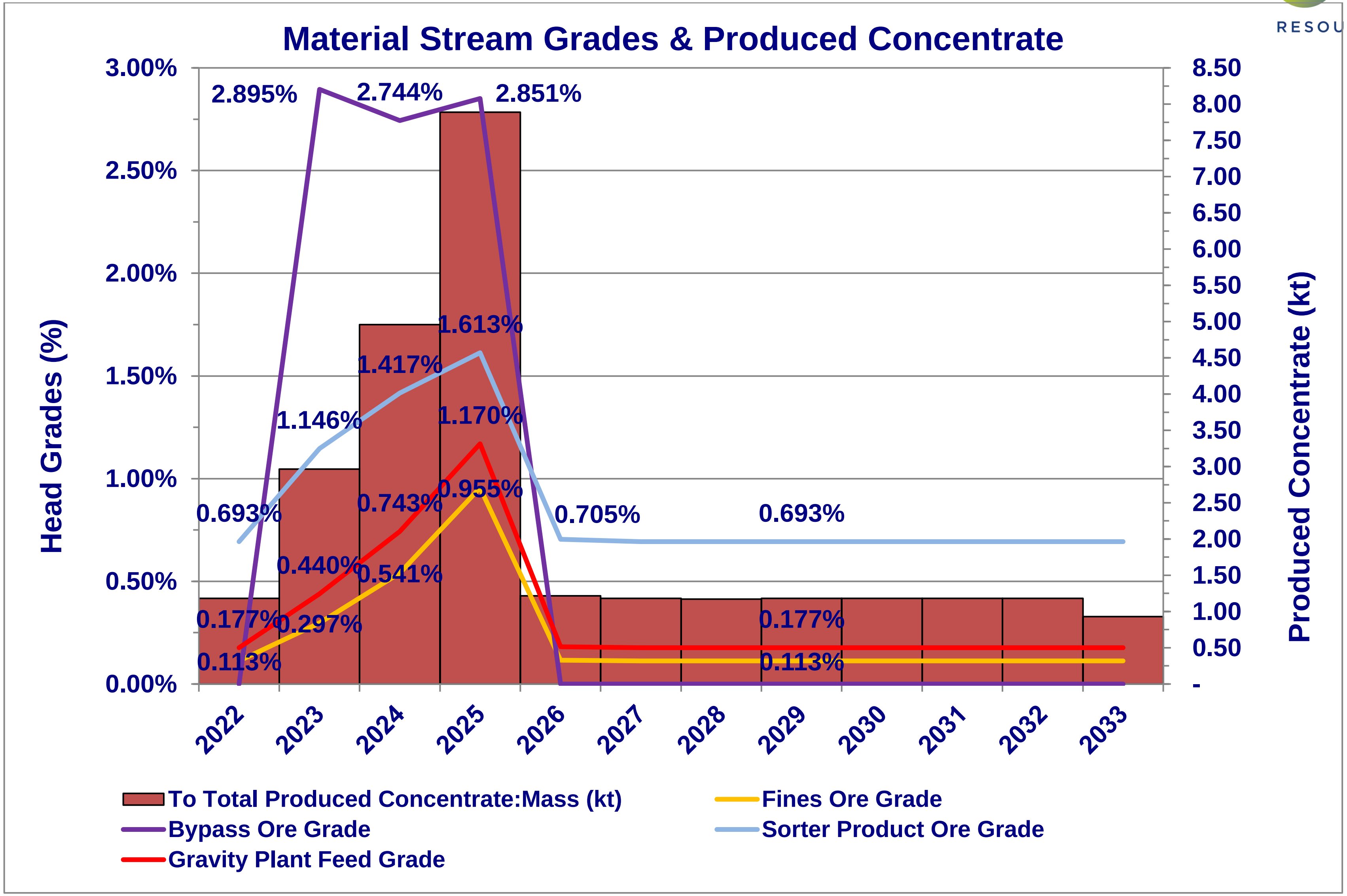

Figure 6: Material Stream Grades & Produced Concentrate

8.2. Equipment Productivities

To achieve nameplate capacity at the processing plant, the productivity rates stated in Table 15 are required.

Table 15: Equipment Productivities

Component	Productivity rate per hour (t)	Operating hours (hr)	Annual capacity (t)
LGS Mining Fleet	331	3024	1,000,000
OC Mining Fleet	824	6048	5,000,000
Crushing & Screening	200	6804	1,500,000
Ore Sorting	160	5100	816,000
Gravity Processing Plant	60	6804	408,000
Front End Loaders (2 units)	350	6048	2,110,000

Cycle time productivities for the two mining fleets are as per Sections 8.2.1 and 8.2.2.

8.2.1. Open Cut Mining

The open cut mining productivity assumptions are listed in Table 16 and Table 17.

Table 16: OC Excavator Cycle Time Productivity

Item	Parameter	Unit
Mt Carbine Excavator Selection Parameters		
Material Type	Rock - Well blasted	
Range	Upper	
Bucket Fill Factor	90.0	%
Estimated Cycle Time	0.60	min
Cycles per hour	100	
Operator skill factor	90.0	%
Machine availibility	85.0	%
General Opperational Efficiency	85.0	%
Effective cycles per hour	65	
Hourly Production Required	824	tph
Required payload	12.7	tonnes
Material loose density	1.9	tonne/m3
Bucket payload volume	6.7	m3
Nominal bucket size	7.4	m3

Table 17: OC ADT Cycle Time Productivity

Item	Parameter	Unit
Articulated Dump Truck Specifications		
Model	Bell B50E	
Payload	45	tonnes
Number of Excavator Passes	4	
Truck Cycle Times		
Truck Manoeuvre, Reverse and Spotting Time	30	sec
Excavator Time to Load Truck	144	sec
Truck Manoeuvre and Dump Time at the Dump	30	sec
Fixed Time Result (sec)	204	sec
Fixed Time Result (min)	3.40	min
Distance from Load Point to Dump	1840	m
Average Speed	20	km/hr
Travel Time	11.04	min
Total Truck Cycle Time	14.4	min
Cycles per Hour	4.16	
Production Capacity		
Production Truck Capacity Tonnes per Hour	187	tonnes
Total Required Truck Units	4.41	units

8.2.2. LGS Mining

The low-grade ore stockpile mining productivity assumptions are listed in Table 18.

Table 18: LGS ADT Cycle Time Productivity

Item	Parameter	Unit
Articulated Dump Truck Specifications		
Model	Bell B50E	
Payload	45	tonnes
Number of Excavator Passes	8	
Truck Cycle Times		
Truck Manoeuvre, Reverse and Spotting Time	30	sec
Excavator Time to Load Truck	252	sec
Truck Manoeuvre and Dump Time at the Dump x 2	60	sec

Item	Parameter	Unit
Fixed Time Result (sec)	342	sec
Fixed Time Result (min)	5.69	min
Distance from Load Point to Rock Screen	773	m
Average Speed	20	km/hr
Travel Time	2.319	min
Load Truck with FEL Time	2	min
Travel to Dump and Return to Initial Load Point Distance	1542	m
Travel to Dump and Return to Initial Load Point Time	4.6	min
Total Truck Cycle Time	14.6	min
Cycles per Hour	4	
Production Capacity		
Production Truck Capacity Tonnes per Hour	184	tonnes
Total Truck Capacity Tonnes per hour	553	tonnes
Annual Capacity	2,550,000	tonnes

9. Mining Method

Mining of both the LGS and open cut shall be performed using conventional excavator and truck operations. Similar sized fleets are utilised in both areas, providing flexibility for mine design, scheduling and operational execution.

9.1. Low Grade Stockpile

Extraction from the LGS is a straightforward load and haul process. Mining will be completed by a 90t excavator working on four metres benches, with a fleet of Bell 50t articulated dump trucks hauling material to the dry processing plant.

A crush and convey system was assessed against the excavator and truck option as part of a cost benefit analysis but eliminated for the following reasons:

- Higher capital cost option;
- Was specific to the LGS and could not be utilised for other site haulage requirements;
- Reduced mobility impacts flexibility in mine planning and operational decision-making;
- Multiple material movement systems onsite increase the complexity of maintenance and personnel training ; and
- Potential power supply constraints to the site.

Mining will be undertaken from top to bottom in the aforementioned 4m benches, commencing in the south-eastern section of the LGS and progressing to the north-west. As the LGS thickens, multiple benches will be excavated producing a conventional strip-mining arrangement.

Bench widths will nominally be a minimum 25m to allow sufficient working room (turning and reversing) for the articulated dump trucks.

Oversize material (+700mm) will be stockpiled separately and periodically transported to the waste dump. It is expected that in certain areas of the LGS there will be a higher frequency of oversize material (due to the original dumping methodology). In this situation a dozer or FEL may be utilised at the dig face to manage oversize material and ensure excavator productivity is not adversely impacted.

9.2. Open Cut

Similarly, open cut mining will be undertaken in a standard drill and blast, load and haul configuration.

9.2.1. Drill and Blast

Approximately 5.25 million tonnes of material require blasting in 2023, increasing to 5.55Mt in 2024, and then tapering off to 4.27 million tonnes in 2025. To maintain sufficient blasted inventory, a minimum 100kt of blasted material is required on a weekly basis. Mining blocks will be a minimum 20x20x30 metres in size, equating to 12,000bcm or 33,000 tonnes. Accordingly, at least 3 blocks will be blasted weekly to maintain the required inventory.

The tungsten bearing veins are hard but brittle and fracture easily, increasing the likelihood of preferential energy transfer should a blasthole intersect a tungsten vein. The detrimental outcome of this situation is blocky oversize waste material and fine ore with a reduced grade due to blasting loss. Whilst it will be practically impossible to avoid all tungsten bearing veins, operational mitigation controls such as ore mark-up and associated blast pattern modification will be required.

Drilling and blasting will be serviced by a total load contract, where all drilling, loading and blasting activities will be sub-contracted to deliver blasted inventory at a dollar per tonne rate. The bulk explosive volumes are approximately 30 tonnes per week, as such a drive-in, drive out service from Cairns is envisaged; no allowance for an on-site explosives compound has been made.

Table 19 and Table 20 detail indicative drill and blast design parameters and associated physical metrics per blast.

Table 19: Indicative Blast Quantities Per Blast

Avg. Blast Volume (m^3)	Avg. Blast Tonnes (t)	Avg. Blast Drill Holes	Avg. Blast Drill Meters (m)	Average Blast Bulk Explosive (kg)
36,496	100,000	174	3,699	29,723

Table 20 - Indicative Blast Design Parameters

Parameter	Quantity	Unit
Bench Height	20	m
Drill Angle	70	degree
Hole Length	21.2836	m
Horizontal	7.2794	m
Hole Diameter	102	mm
Bench Height	20	m
Bench Factor	196.08	(60f - 120f)
Burden	3	m
Burden Factor	29.41	(25f - 40f)
Spacing	3.50	m
Spacing Factor	1.17	(1.15B-1.5B)
Subdrill	0.7	m
Subdrill Factor	6.86	(6f - 12f)
Stemming	2.5	m
Stemming Factor	24.51	(20f - 26f)
Explosive Charge Weight per Metre	1.15	g/cc
Charge Length	18.20	m
Charge Weight	171	kg
Volume Blasted	210	Bcm
Powder Factor	0.81	

9.2.2. Excavation, Load and Haul

Loading of the ADT fleet will be undertaken by a single 100-120 tonne excavator with a bucket capacity of at least 7.4m^3 (in excavator configuration). An example of this excavator class is the Liebherr R9100. Technical specifications are included in Appendix D.

Bench geometry will be slightly smaller than the LGS, with a 4m height and minimum 20m width. Ore delineation and excavation will be undertaken as per the following process:

- Following blasting a visual on-foot inspection of the mining block surface will be undertaken and ore zones identified either through high-precision GPS survey, mark-up paint or both;
- Excavation of the initial mining bench will occur. Mining dilution will be impacted by blast fragmentation, operator proficiency and working conditions (primarily poor lighting and dust). Accordingly, a focus on operational excellence for both drill and blast and excavation practices will be required; and
- Following excavation of the bench, the mark-up process is repeated for the subsequent bench. Ideally, high-precision GPS would be utilised to facilitate the development of an operational model for machine guidance, operational geology and mine reconciliation purposes.

Detailed grade control practices are included in Chapter 3: Geology and Resources.

Haulage of waste and ore to the dump and dry processing plant respectively will be undertaken by five to six 45t class articulated dump trucks.

The 45t articulated dump truck was chosen for the following reasons:

- Superior operating performance with respect to ramp width and grade, working room and traffic management for the size of the deposit and the mining schedule; and
- The truck class is already on site, enabling mine planning flexibility and synergies for equipment maintenance and operator training

9.3. Future Opportunities

There exists a number of opportunities, that with additional activities, will improve the mining schedule and overall Project economics. These include:

- Infill drilling of the pit shell and adjacent areas; and
- Pre-processing of the ore fines component in the dry processing plant prior to transport to the gravity processing plant.

Infill drilling to define further zones of high-grade ore should either a) improve the strip ratio of the existing pit shell, or b) expand the pit shell, facilitating additional flexibility in the mining sequence for strip ratio and head-grade optimisation.

Although the fines are naturally upgraded in the crushing stage due to the mineralisation style, the fines material comprises between 68-89% of the gravity processing plant mass and is the lowest grade ore of the three material streams. It should also be noted that the gravity processing plant is the primary capacity constraint on Project economics.

Accordingly, any pre-processing system that removes non-tungsten bearing mass will have a significant impact on gravity processing plant capacity, head grade and associated concentrate production.

10. Mining Limits

Maptek Vulcan Pit Optimiser software was utilised to determine the optimal pit limits of the ore body based on a set of predefined parameters. This tool can be used with the Lerch & Grossmann 3D algorithm or the 3D Floating Cone algorithm. The tool allows for flexibility when applying costs to the optimisation, as values can be calculated manually or automatically calculated by the software. Pit Optimiser is run within an existing geological block model. The model used was 'Mt_Carbine_20210918.bmf' and several variables were added to the block model before running Pit Optimiser.

The minimum required model attribute variables are:

- At least one grade variable to define the desired product or products. Tungsten grade (WO_3%) was used as the primary grade variable. A secondary grade variable was created to filter and optimise the blocks by the indicated resource category. This attribute variable was created and called 'WO3_rescat'; and
- An attribute variable in which to store the resulting pits. An attribute variable was created for each scenario ie: pit_mc_oco_17 and saved within the block model for future pit generation.

Once the block model was configured, the below operational variables were input to create a range of individual scenarios:

- Revenue per tonne of product WO_3;
- Recoveries from insitu mining through to processing;
- Material densities;
- Overall pit slopes;
- Mining costs;
- Processing costs; and
- Rehabilitation costs.

Table 21: Maptek Pit Optimiser Variables and Ranges

Variable	Unit of Variable	Variable Minimum	Variable Maximum
Ore Resource Category	Tonnes	Indicated Resource	Indicated Resource
Revenue	AUD per tonne of tungsten	27,000	27,000
Mining Cost	AUD per tonne of mined material	4.00	6.44
Processing Cost	AUD per tonne including dry and gravity processing	14.50	14.50
Rehabilitation Cost	AUD per tonne of waste material	0.20	0.20

Table 22: Physical Parameters used in the Pit Optimizer

Variable	Bench Height (m)	Bench Batter Angle (^{0})	Berm Width (m)	Overall Slope Profile (^{0})
Pit Design	20	70	8	54

From each scenario a report was generated to show the following:

- Rock Best: The total amount of material to be mined within the optimised shell;
- Waste Best: The total amount of waste material to be mined within the optimised shell;
- Total Ore Best: The total amount of ore material to be mined within the optimised pit shell;
- Stripping Ratio Best: The ratio of waste to ore material to be mined within the optimised pit shell;
- Product Best: The amount of tungsten metal recovered from the entire mining and processing process;
- Mining Cost Best: The cost of mining per tonne to include drill and blast, load and haul, management overheads and mine rehabilitation;
- Processing Cost Best: The cost of processing per ore tonne to include crushing, screening, XRT sorting, wet processing, and tailings management;
- Revenue Best: The revenue generated from the product tonnes;
- Profit Margin: Total revenue – total costs;
- Insitu Tungsten: The amount of insitu tungsten within the optimised pit shell prior to mining and processing; and
- Ore Grade: The percentage of tungsten metal contained within the ore, within the optimised pit shell.

The mining scenario outputs from Pit Optimiser are shown in Table 23.

Table 23: Mining Scenario output from Pit Optimiser

Scenario Name	Scenario Variable	Resource Category	Gravity Plant Recovery (%)	Mining Cost (AUD)	Process Cost (AUD)	Rehab Cost (AUD)
MC_OCO_1	Base Case	Indicated	70	5.85	14.50	0.20
MC_OCO_2	Mining Cost +5%	Indicated	70	6.14	14.50	0.20
MC_OCO_3	Mining Cost +10%	Indicated	70	6.44	14.50	0.20
MC_OCO_4	Mining Cost - 5%	Indicated	70	5.56	14.50	0.20
MC_OCO_5	Mining Cost - 10%	Indicated	70	5.27	14.50	0.20
MC_OCO_6	Mining Cost - 15%	Indicated	70	4.97	14.50	0.20
MC_OCO_7	Mining Cost - 20%	Indicated	70	4.68	14.50	0.20
MC_OCO_8	Processing Cost +5%	Indicated	70	5.85	15.23	0.20
MC_OCO_9	Processing Cost +10%	Indicated	70	5.85	15.95	0.20
MC_OCO_10	Processing Cost -5%	Indicated	70	5.85	13.78	0.20
MC_OCO_11	Processing Cost -10%	Indicated	70	5.85	13.05	0.20

Scenario Name	Scenario Variable	Resource Category	Gravity Plant Recovery (%)	Mining Cost (AUD)	Process Cost (AUD)	Rehab Cost (AUD)
MC_OCO_12	Unconstrained Rescat	Indicated and Inferred	70	5.85	14.50	0.20
MC_OCO_13	Base Case 75% Recovery	Indicated	75	5.85	14.50	0.20
MC_OCO_14	75% Recovery Unconstrained Rescat	Indicated and Inferred	75	5.85	14.50	0.20
MC_OCO_15	75% Recovery Unconstrained Rescat	Indicated and Inferred	75	4.50	14.50	0.20
MC_OCO_16	75% Recovery Unconstrained Rescat	Indicated and Inferred	75	4.00	14.50	0.20
MC_OCO_17	Mining Cost AUD4.50 Recovery 75%	Indicated	75	4.50	14.50	0.20

From Table 23, Scenario 17 was selected as the most financially attractive and logical optimised economical pit shell to take forward for more detailed and practical mineable design. The following steps were executed to deliver the mine design:

1. Vulcan automated pit designer;
2. Setup spec file;
3. Batter Angle = 70 degree;
4. Berm width = 8m;
5. Contour existing block model (Mt_Carbine_20210918.bmf);
6. Generate pit shell based on block model variable (pit_mc_oco_17);
7. Benches – Min elevation = 220RL Max elevation = 400RL Bench height = 20m;
8. Create seed strings to develop final pit limits;
9. Create bench and batters;
10. Insert ramp;
11. Create triangulation;
12. Create mining solid; and
13. Run reserves editor.

Following the creation of a practical pit shell and ore reserves, Comet Strategy software was utilised to deliver a series of scenarios with a) a physical mining schedule and b) an associated Project NPV. In this way the Project drivers and constraints could be assessed against each other, and sensitivities identified. The comparison of the optimised pit shell to the practical pit shell is shown in Figure 7.

Scenario analysis of operational rosters and associated labour/cost components for the dry processing and gravity processing plants was undertaken during an earlier stage of the feasibility study. Table 24 details the Project variables and ranges included in the final scenario analysis.

Table 24: Variables and Ranges used in Comet Strategy Software

Variable	Unit	Range – Minimum	Range - Maximum
Tungsten Price Payable (50% Concentrate)	AUD	13,533	16,181
Mobile Fleet Capacity	Tonnes per annum	4,000,000	6,000,000
Gravity Plant Max. Head Grade	%	0.5	2
Roster for LGS Mining	hrs	3024	6048

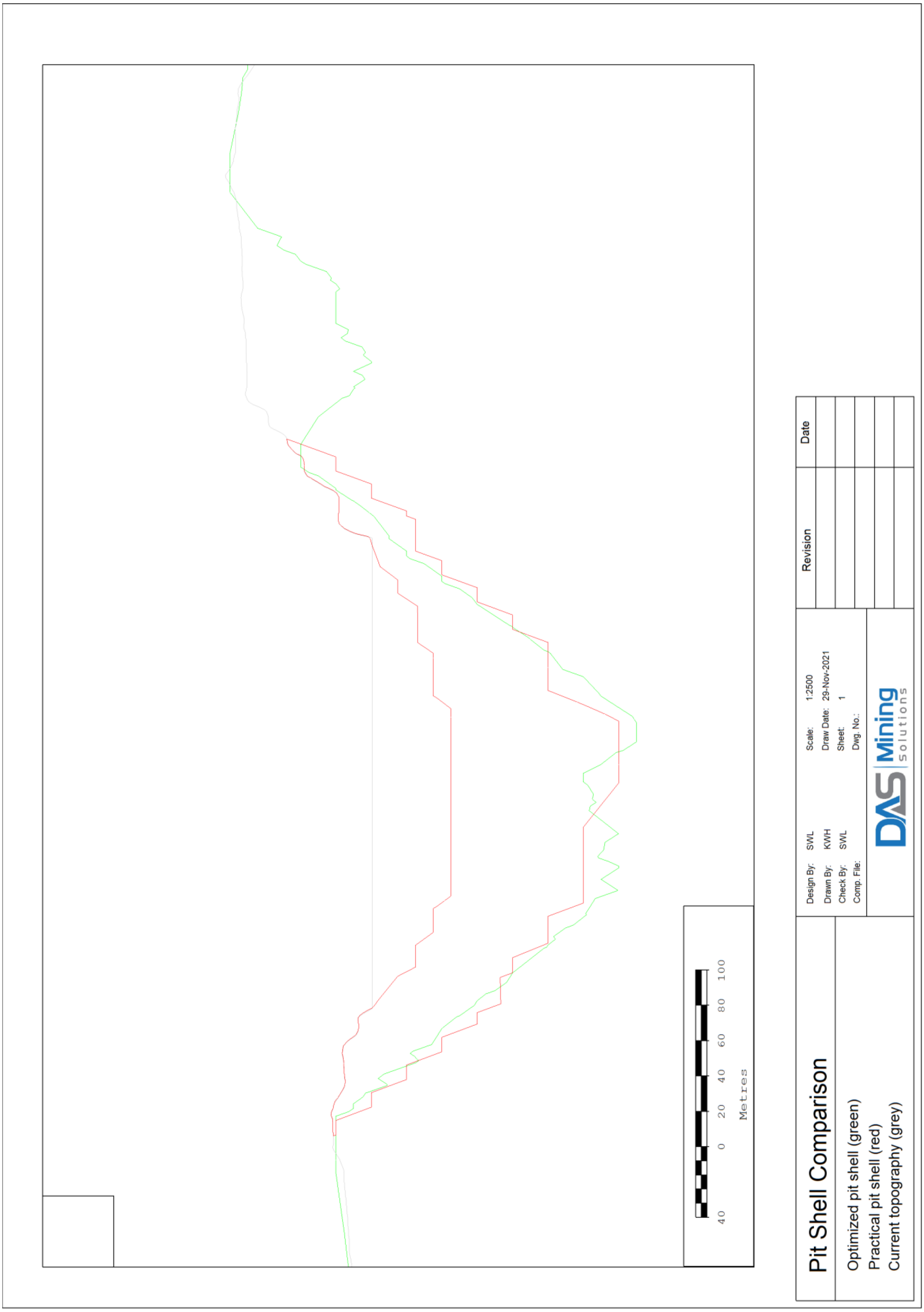

Figure 7: Cross Section Comparing Optimised and Practical Pit Shells

11. Mine Layout

11.1. Mine Layout

The Mt Carbine mining leases straddle the Mulligan Highway, which bisects the operation in an approximately east-west direction. Power and ancillary water services run adjacent to the highway with existing connections to the gravity processing plant and administration area.

To the north of the highway are:

- The insitu orebody;
- Low grade stockpile;
- Crushing screening and sorting plant (excluding ore sorter product sizing);
- Quarry and waste rock dumps; and
- Administration and maintenance facilities.

South of the highway are:

- The gravity processing plant;
- Secondary crushing plant for the ore sorter product material (VSI);
- Process plant reject material management system; and
- Historical tailings dam .

Figure 8 to Figure 12 illustrate the location of the various components of the mining operation.

The mining process is simple and is as follows:

- Material extracted from the LGS and open cut via excavator and trucked to the Grizzly/Rock Screen at the dry processing plant;
- Material fed to the Jaw Crusher via FEL and crushed to pass -100mm;
- Feed material fed to a double deck screen via conveyor;
- -6mm and -40+6mm material screened off and stockpiled;
- +40mm material crushed via cone crusher and recirculated back onto the double deck screen;
- FEL loads -6mm material into a hopper for slurry pumping to the gravity processing plant;
- -40+6mm material fed into the XRT Sorter via a reclaimer/conveyor system;
- XRT Sorter product and reject material stockpiled;
- FEL loads XRT Sorter reject material into ADTs for transport to the quarry or waste dump depending on quality of product; and
- FEL loads XRT Sorter product material into ADTs for transport to the gravity processing plant.

With the exception of the fines material, ADTs will manage transport of all material around site. Back-haulage practices will be employed, where ADTs will transport ore from the LGS or open cut the dry procesing plant, then from the dry processing plant to waste dump/quarry or down to the gravity processing plant, prior to returning to the LGS/open cut excavation faces.

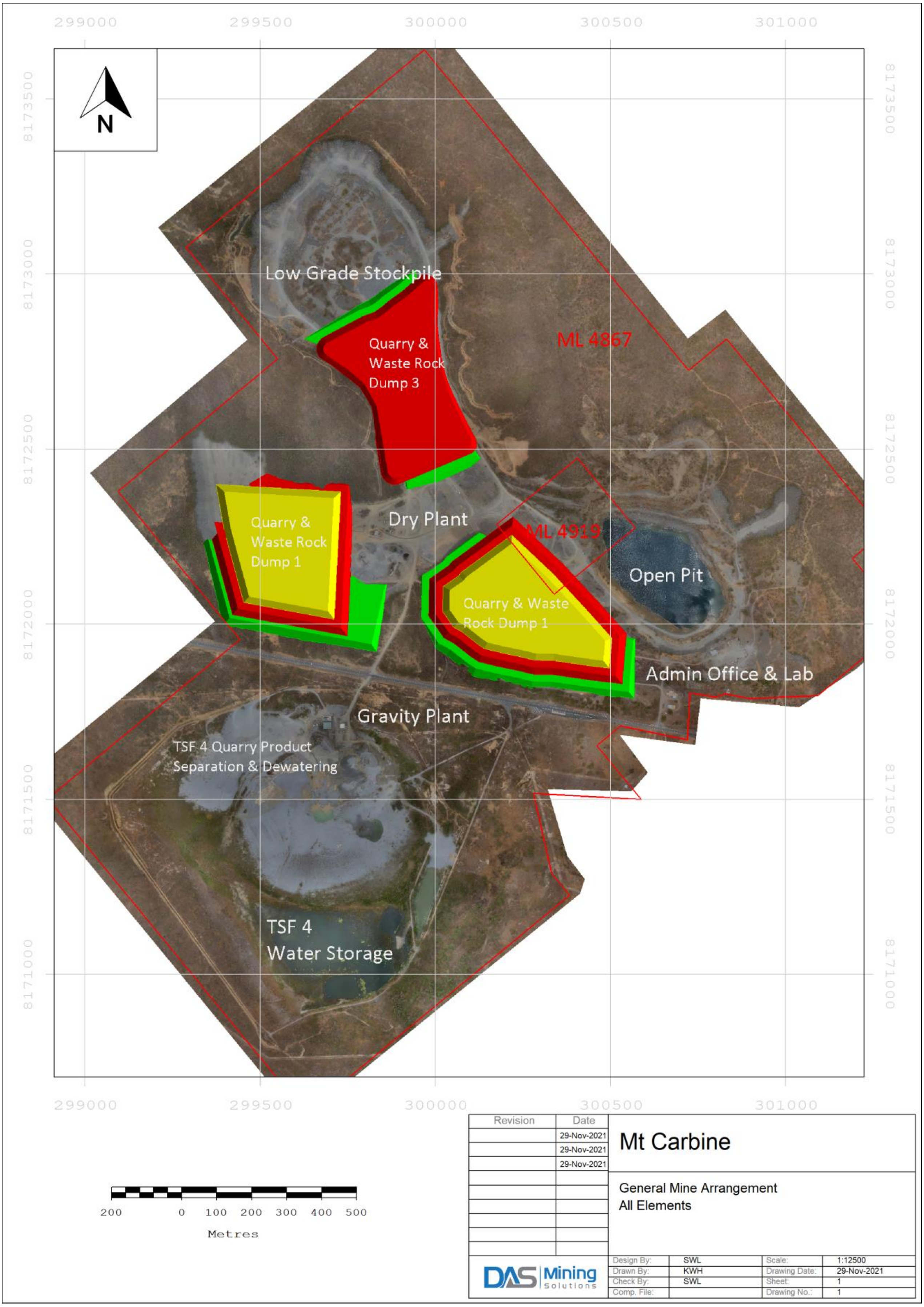

Figure 8: General Mine Arrangement

Figure 9: OC and LGS Mining Limits

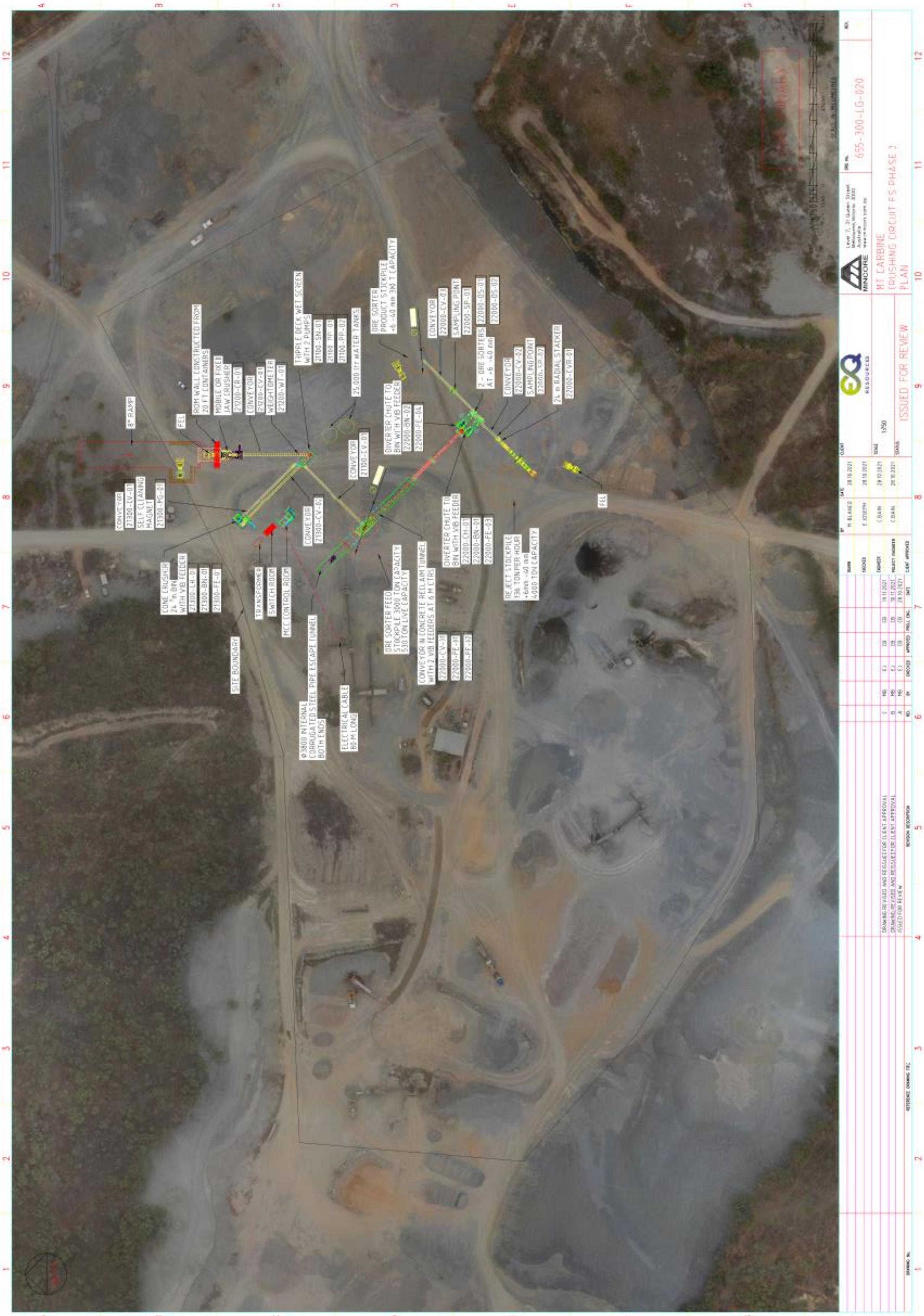

Figure 10: Crushing, Screening and Sorting Plant

Figure 11: Processing Plant and Tailings Dewatering System

Figure 12: Ore Sorter Product Rehandling Circuit

11.2. Technical Risks

The primary technical risks for the mining activities are summarized as follows:

- Open cut
  - Geotechnical instability due to the intersection of pit geometry with major faults/joints;
  - Poor blast fragmentation; and
  - Ore recovery.
- LGS
  - Higher frequency of oversize material; and
  - Bench instability due to poor excavation practices.
- Dry Processing Plant
  - Poor crushing & screening performance.

Table 25 details a relative risk matrix of the primary technical risks.

Table 25: Primary Technical Risks

Area	Risk	Consequence	Impact
Open Cut	Major Geotechnical Instability	Loss of production, potential loss of ore reserves.	Extreme
Open Cut	Poor Blast Fragmentation	Loss of production, reduced ore recovery, increased wear/damage to fleet and plant.	Major
Open Cut	Ore Recovery	Loss of ore, reduced operational profitability.	Major
LGS	Higher Frequency of Oversize Material	Loss of production, increased wear/damage to fleet and plant.	Minor
LGS	Bench Instability due to Poor Excavation {ractices	Loss of production.	Minor
Dry Processing Plant	Poor Crushing & Screening Performance	Loss of production, increased wear/damage to plant.	Moderate

Mitigation of the technical risks can be achieved through the following controls:

- Major geotechnical instability:
  - Detailed geotechnical assessment of the rockmass and associated structures;
  - Pit design modifications;
  - Ground stabilisation/support techniques; and
  - Operational best practice with respect to blasting, dig to design compliance and geotechnical input into mine planning processes.
- Poor blast fragmentation:
  - Incorporation of geological/geotechnical data (structures, rock strength) in blast design;
  - Drill to design compliance; and
  - Consumables (explosives, stemming) load to design.
- Ore recovery:
  - Adequate blast fragmentation;

    - Robust and consistent ore-markup practices;
    - Optimal working environment during extraction (light and dust); and
    - Ore spotter if required.
- Higher frequency of oversize material:
    - Ancillary equipment (dozer or loader) on standby to manage material; and
    - Separate stockpiling/removal of oversize material.
- Bench instability due to poor excavation practices:
    - Dig to design compliance – no undercutting of bench profiles.
- Poor crushing & screening performance:
    - Adequate blast fragmentation; ad
    - Consistent feed rate within equipment optimal performance envelope.

Table 26 details the risk matrix with controls applied.

Table 26: Primary Technical Risks with Controls Applied

Area	Risk	Revised Impact	Comment
Open Cut	Major Geotechnical Instability	Major	Continuous geotechnical control required during OC mining.
Open Cut	Poor Blast Fragmentation	Moderate	Has impacts on multiple downstream processes.
Open Cut	Ore Recovery	Moderate	Direct impact on profitability.
LGS	Higher Frequency of Oversize Material	Minor	
LGS	Bench Instability due to Poor Excavation Practices	Negligible	
Dry Processing Plant	Poor Crushing & Ccreening Performance	Minor	Adequate blast fragmentation a key sensitivity.

11.3. Mining Reserves

Loss and dilution values are included in the mining reserves and the detailed in Section 12.1.

Table 27 details the mining reserves for the open cut and

Table 28 details the mining reserves for the LGS.

Table 27: Open Cut Mining Reserves

Region	Bench RL	Resource Category	Material Type	Ore Grade (%)	WO3 Volume (bcm)	WO3 Mass (t)	WO3 Min (%)	WO3 Max (%)	Total Volume (bcm)	Total Mass (t)
Pit17_6_Phase3_Solid.00t	300	Target	Waste	0	-	-	0	0	237,263	650,102
Pit17_6_Phase3_Solid.00t	300	Indicated	Ore	0.94	33,370	91,433	0.19	5.09	33,370	91,433
Pit17_6_Phase3_Solid.00t	320	Target	Waste	0	-	-	0	0	20,960	57,431
Pit17_6_Phase3_Solid.00t	320	Indicated	Ore	0.62	2,817	7,719	0.22	5.09	2,817	7,719
Pit17_6_Phase4_Solid.00t	220	Target	Waste	0	-	-	0	0	76,261	208,955
Pit17_6_Phase4_Solid.00t	220	Indicated	Ore	0.81	18,388	50,384	0.26	1.95	18,388	50,384
Pit17_6_Phase4_Solid.00t	240	Target	Waste	0	-	-	0	0	172,818	473,521
Pit17_6_Phase4_Solid.00t	240	Indicated	Ore	0.88	32,074	87,882	0.19	5.09	32,074	87,882
Pit17_6_Phase4_Solid.00t	260	Target	Waste	0	-	-	0	0	491,246	1,346,013
Pit17_6_Phase4_Solid.00t	260	Indicated	Ore	0.91	71,862	196,901	0.19	5.09	71,862	196,901
Pit17_6_Phase4_Solid.00t	280	Target	Waste	0	-	-	0	0	752,221	2,061,084
Pit17_6_Phase4_Solid.00t	280	Indicated	Ore	0.89	100,766	276,098	0.19	5.09	100,766	276,098
Pit17_6_Phase4_Solid.00t	300	Target	Waste	0	-	-	0	0	767,437	2,102,777
Pit17_6_Phase4_Solid.00t	300	Indicated	Ore	0.8	69,019	189,111	0.14	5.09	69,019	189,111
Pit17_6_Phase4_Solid.00t	320	Target	Waste	0	-	-	0	0	838,345	2,297,066
Pit17_6_Phase4_Solid.00t	320	Indicated	Ore	0.69	38,221	104,727	0.14	1.88	38,221	104,727
Pit17_6_Phase4_Solid.00t	340	Target	Waste	0	-	-	0	0	823,520	2,256,444
Pit17_6_Phase4_Solid.00t	340	Indicated	Ore	0.59	20,029	54,878	0.14	1.63	20,029	54,878
Pit17_6_Phase4_Solid.00t	360	Target	Waste	0	0	0	0	0	740,548	2,029,101
Pit17_6_Phase4_Solid.00t	360	Indicated	Ore	0.5	10285	28182	0.17	1.63	10,285	28,182

Region	Bench RL	Resource Category	Material Type	Ore Grade (%)	WO3 Volume (bcm)	WO3 Mass (t)	WO3 Min (%)	WO3 Max (%)	Total Volume (bcm)	Total Mass (t)
Pit17_6_Phase4_Solid.00t	380	Target	Waste	0	0	0	0	0	185,499	508,267
Pit17_6_Phase4_Solid.00t	380	Indicated	Ore	0.59	3657	10021	0.17	1.57	3,657	10,021
Pit17_6_Phase4_Solid.00t	400	Target	Waste	0	0	0	0	0	2,816	7,716
Pit17_6_Phase4_Solid.00t	400	Indicated	Ore	0.36	218	597	0.22	0.79	218	597

Table 28: LGS Mining Reserves

Region	Resource Category	Material Type	Ore Grade (%)	WO3 Volume (bcm)	WO3 Mass (t)	WO3 Min (%)	WO3 Max (%)	Total Volume (bcm)	Total Mass (t)	Total Tungsten (t)
LGSP_RL_370_1_local.00t	Indicated	Ore	0.075	56,707	90,731	0.075	0.075	56,707	90,731	68
LGSP_RL_374_1_local.00t	Indicated	Ore	0.075	283,485	453,575	0.075	0.075	283,485	453,575	340
LGSP_RL_370_1_local.00t	Indicated	Ore	0.075	56,707	90,731	0.075	0.075	56,707	90,731	68
LGSP_RL_374_1_local.00t	Indicated	Ore	0.075	283,485	453,575	0.075	0.075	283,485	453,575	340
LGSP_RL_378_1_local.00t	Indicated	Ore	0.075	401,238	641,981	0.075	0.075	401,238	641,981	481
LGSP_RL_382_1_local.00t	Indicated	Ore	0.075	496,647	794,636	0.075	0.075	496,647	794,636	596
LGSP_RL_386_1_local.00t	Indicated	Ore	0.075	372,966	596,746	0.075	0.075	372,966	596,746	448
LGSP_RL_390_1_local.00t	Indicated	Ore	0.075	367,442	587,907	0.075	0.075	367,442	587,907	441
LGSP_RL_394_1_local.00t	Indicated	Ore	0.075	350,423	560,676	0.075	0.075	350,423	560,676	421
LGSP_RL_398_1_local.00t	Indicated	Ore	0.075	269,038	430,461	0.075	0.075	269,038	430,461	323
LGSP_RL_402_1_local.00t	Indicated	Ore	0.075	165,861	265,377	0.075	0.075	165,861	265,377	199
LGSP_RL_406_1_local.00t	Indicated	Ore	0.075	7,771	12,433	0.075	0.075	7,771	12,433	9
LGSP_RL_370_2_local.00t	Indicated	Ore	0.075	110,794	177,270	0.075	0.075	110,794	177,270	133

Region	Resource Category	Material Type	Ore Grade (%)	WO3 Volume (bcm)	WO3 Mass (t)	WO3 Min (%)	WO3 Max (%)	Total Volume (bcm)	Total Mass (t)	Total Tungsten (t)
LGSP_RL_374_2_local.00t	Indicated	Ore	0.075	422,336	675,738	0.075	0.075	422,336	675,738	507
LGSP_RL_378_2_local.00t	Indicated	Ore	0.075	566,492	906,387	0.075	0.075	566,492	906,387	680
LGSP_RL_382_2_local.00t	Indicated	Ore	0.075	611,562	978,500	0.075	0.075	611,562	978,500	734
LGSP_RL_386_2_local.00t	Indicated	Ore	0.075	500,781	801,250	0.075	0.075	500,781	801,250	601
LGSP_RL_390_2_local.00t	Indicated	Ore	0.075	427,390	683,823	0.075	0.075	427,390	683,823	513
LGSP_RL_394_2_local.00t	Indicated	Ore	0.075	339,085	542,536	0.075	0.075	339,085	542,536	407
LGSP_RL_398_2_local.00t	Indicated	Ore	0.075	297,175	475,480	0.075	0.075	297,175	475,480	357
LGSP_RL_402_2_local.00t	Indicated	Ore	0.075	190,462	304,740	0.075	0.075	190,462	304,740	229
LGSP_RL_406_2_local.00t	Indicated	Ore	0.075	12,790	20,465	0.075	0.075	12,790	20,465	15

12. Product and Waste

12.1. Mining Criteria for Product and Waste Determination

Due to the thin, vertical nature of the tungsten bearing veins, the following assumptions have been included in the mining criteria:

1. Mining dilution of 16% in addition to material added during the geological modelling process;
2. Minimum ore mining thickness of 1.0m;
3. Loss of 1% of ore (mainly attributable to drill and blast processes);
4. Bench height of 4m;
5. Excavator class of 100-120t for non-ore material with associated ADT fleet;
6. Excavator class of approximately 50t-90t for ore mining;
7. Tomra X-ray ore sorter:
    a. Metal recovery of 90% (-40+6mm ore size fraction); and
    b. Capacity of 816ktpa (120tph).
8. Processing plant:
    a. Fines (-6mm size fraction) metal recovery of 77%;
    b. Coarse (-40+6mm ore sorter product) metal recovery of 90%; and
    c. Capacity of 408ktpa (60tph).

The basis of the above input variables are as follows:

- Points 1 to 4 are based on experience in mining environments with similar production profiles and extraction requirements;
- Points 5 and 6 are based on a combination of current site production performance and previous mining experience;
- Point 7 is based on current site production data;
- Point 8b is based on current site production data; and
- Points 8a and 8c are based on current site production data with additional improvements due to plant upgrades (provided by Ausenco).

Dilution of the high-grade ore zones is a key component of the ore reserving process. This was completed in two steps:

- During geological modelling dilution was added to the tungsten veins to produce a 2m wide downhole intersection, with associated reduction in grade. When converted from downhole to horizontal width, this translates to between 1 and 1.50m (dependent on the angle of the drill hole); and
- Further dilution was added during the ore reserving process to reflect the D&B/load and haul extraction process.

12.2. XRT Sorter Mass Recovery Methodology

Metal recovery through the XRT Sorter is consistent at 90% or slightly higher, validated by a number of sampling tests where feed, product and waste samples were analysed for WO_3% grade.

However mass recovery is variable, linked as it is to XRT Sorter feed grade. The methodology for determining XRT Sorter mass recovery is as follows:

- Utilise the existing plant performance as a reliable source of information;
- Low grade analysis point is the LGS feed material – head grade is approximately 0.054% WO_3 (LGS global grade is 0.075%, but once the fines material is removed the grade drops to 0.054%):
  - Mass recovery is ~7%;
- High grade analysis point is ~1% - feed material was made up and passed through the XRT Sorter at a pilot scale:
  - Mass recovery is ~50%;
- Top grade cut-off of the XRT Sorter was set at 2%, based on metal recovery loss at higher grades becoming a material issue. All ore >2% grade bypasses the XRT Sorter and goes directly to the gravity processing plant;
- Based on the low and high grade points a linear trend and algorithm was generated; and
- The algorithm was then used to generate mass recovery for every mining block in the OC pit.

As further production or test data becomes available the algorithm will be refined.

Figure 13 details the linear algorithm developed for XRT Sorter mass recovery.

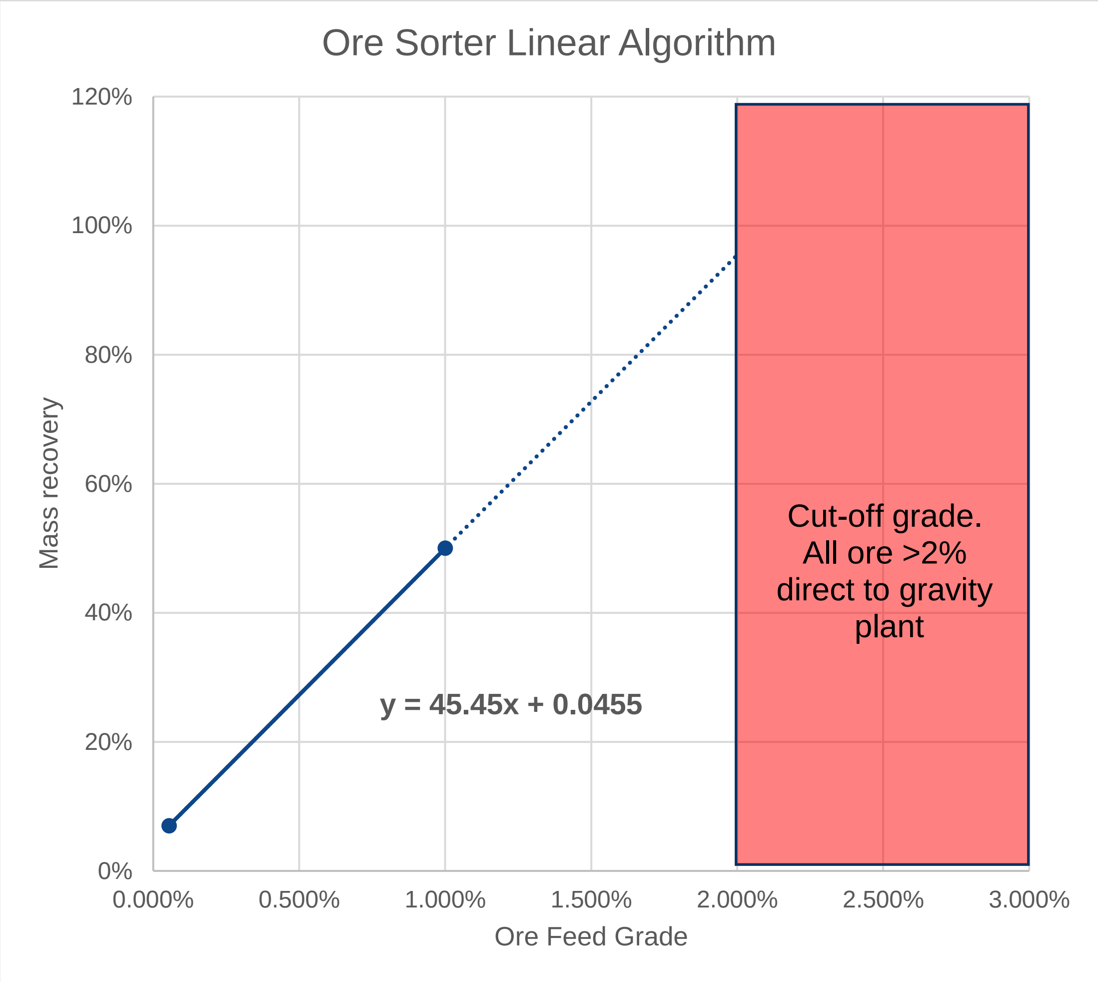

Figure 13: Algorithm for XRT Sorter Mass Recovery

12.3. Cut-off Grade

There are two main drivers of cut-off grade and associated product/waste determination:

- The use of a TOMRA XRT Sorter (XRT Sorter) to upgrade +6mm ore; and
- The presence of a LGS with a global grade of 0.075% WO_3.

The crushing characteristics of the tungsten bearing material result in the XRT Sorter processing 64% of all ore. Due to the relatively coarse nature of the predominantly wolframite mineralisation, ore and dilution material are efficiently separated with a high metal recovery of >90%. This process significantly lowers the insitu cut-off grade that can be economically mined.

Current operations are economically mining the LGS only, demonstrating a viable cut-off grade of 0.075% WO_3 on the existing cost base. The inclusion of a contract mining fleet and upgrades to both the dry and gravity processing plants deliver increased capacity, improved metal recoveries and associated scale economics to the cost base. Operational costs by component are detailed in Table 3.

The minimum block grade identified in the open cut ore reserves is 0.121% WO_3, well above the LGS global grade of 0.075% WO_3. Even with the increase in mining costs (when compared to the LGS), all open-cut reserves demonstrate positive cash-flows.

Accordingly, the site-wide cut-off grade is set at 0.075% WO_3 for all ore sources.

12.4. Grade Control System

The grade control system will have multiple processes, with the primary objectives of:

- Delineation of the insitu spatial distribution of tungsten ore and associated WO_3 % grade;
- Delineation of the spatial distribution of ore post-blasting for accurate selective mining; and
- Reconciliation of planned vs as-mined ore from a mass and grade perspective.

To achieve the primary objectives, the grade control system is comprised of grade control drilling and associated sampling at the on site laboratory, ore-markup post blasting and ore markup post mining (prior to drilling of the next bench).

Specific details of the grade control drilling and sampling are outlined in Chapter 3: Resources and Geology. Ore-markup will be undertaken by competent personnel, using high precision GPS and paint, preferably on night shift when the scheelite UV fluorescence will be most effective. The use of RTK GPS will assist in the creation of an accurate operational geological model that can be reconciled against the resource model.

13. Reserves Statement

Tony O'Connell of Optimal Mining Solutions was engaged as the Competent Person to produce a JORC compliant Reserves Statement for the Mt Carbine open cut and LGS. An excerpt of the Reserve Statement is provided below and the Reserve Report is included in Appendix E.

13.1. Overview

This statement provides a summary of the 2021 Ore Reserves Estimate for EQ Resources' Mt. Carbine Project. The full details of the Ore Reserve estimate can be found in the complete 2021 JORC Ore Reserve Estimate report completed by Measured Group in December 2021. The date of this statement is December 9 2021.

13.2. JORC Ore Reserve Estimate Statement Summary

Measured Group Pty Ltd (Measured Group) has been engaged by EQ Resources Pty Ltd (EQR) to prepare a Statement of the Ore Reserves for its fully owned Mt Carbine Tungsten Project (Mt. Carbine).

Mt Carbine is an operating tungsten mine and rock quarry located at the northern end of the Atherton Tableland approximately 130 km by sealed highway from the closest major centre of Cairns. EQR acquired the mine and associated quarry in June 2019 and has been operating the mine and quarry concurrently, with the mine currently processing tailings and low grade ore stockpiles located on the site that are remnant from previous operations on the site. The mine is well supported by existing services and infrastructure.

The current plan for Mt Carbine is to recommence the old open pit, which was shut in the late 1980's, whilst continuing to process the LGS. The site is currently permitted to process up to 100,000t per annum of ore, however an amendment to the current environmental authority has been submitted to allow up to 1,000,000t per annum of ore to be processed. The open pit is forecast to be developed at approximately 5mtpa with ore delivery to the plant fluctuating between 250ktpa and 600ktpa. Rehandling of the LGS will top up the total feed into the processing plant to approximately 1mtpa.

The processing plant generates a 50% WO3 concentrate which will be sold on the open market. Mt Carbine currently has off-take agreements for the W03 concentrate which it currently produces. The concentrate will be sold into a market with ongoing strong demand.

A Mineral Resource Statement compliant with the 2012 JORC Code has been prepared by Mr. Chris Grove, a full time employee of Measured Group. The Resources are split into two sections, one for the LGS and one for the open pit as summarised in the table below.

Table 29: Mt Carbine Mineral Resource – September 2021

Classification	Tonnes (million)	Grade (% WO_3)	WO_3 (mtu)
Low Grade Stockpile			
Indicated	12.00	0.075	900,000
In-Situ Hard Rock Resources			
Indicated	2.40	0.74	1,776,000
Inferred	6.81	0.59	4,017,900
Sub-Total	**9.21**	**0.63**	**5,793,900**
Total Mt Carbine Mineral Resource			
	21.21		**6,693,900**

The Ore Reserve was estimated as of 31st December 2021 by a team of mining experts from DAS Mining Solutions, Optimal Mining Solutions and Measured Group. The Competent Person for the Ore Reserve Estimate is Mr Tony O'Connell of Optimal Mining Solutions who contracts to Measured Group.

Open cut Ore Reserves have been estimated by applying modifying factors to the Mineral Resources. The modifying factors include practical pit limits which were based on the current economic limits, determined using indicative operating costs, metallurgical parameters, geotechnical constraints and projected revenue. Other modifying factors included mining losses, mining recovery and dilution factors. An economic evaluation of the mine plan and schedule was completed as part of the estimation process, with the project generating a positive net present value. It should be noted that no revenue was accounted for from the current quarrying services as part of the economic evaluation.

All the Reserves are classified into their respective category based on the level of detail completed in the mine plan and the level of confidence in the Resource estimate. In the categorisation of Reserves, all Indicated Resources have been classified as Probable Reserve. There are no Proven Ore Reserves. No Inferred Resources have been included in the Ore Reserve estimate.

A 0.2% W03 cut-off has been applied in the open pit resource model, however once loss and dilution is applied the minimum open pit ore grade mined is 0.12% WO_3. The average ROM grade of the open pit Ore Reserve is 0.713%. The LGS has been classified as a large homogenous orebody which contains an average of 0.075% WO_3.

The Ore Reserves for the low-grade stockpile and open pit are summarised in the tables below.

Table 30: Low Grade Stockpile Ore Reserve Estimate

Reserve Category	ROM Tonnes (mt)	WO_3 %
LGS - Proved	-	-
LGS - Probable	10.126	0.075%
LGS - Total	**10.126**	**0.075%**

Table 31: Open Pit Ore Reserve Estimate

Reserve Category	ROM Tonnes (mt)	WO_3 %
Open Pit - Proved	-	-
Open Pit - Probable	1.263	0.713%
Open Pit - Total	**1.263**	**0.713%**

The Resources are reported inclusive of the Ore Reserves. The Ore Reserves have been estimated using the same geological model as the Mineral Resource Statement.

The open pit Ore Reserves are accompanied by 14.0mt of waste which provides an overall ROM strip ratio of 11:1 t/t.

14. Waste Disposal

Waste material will be generated from both the insitu open cut and the dry processing plant. Material from the open cut will be transported directly from the dig face to the dump, whilst XRT Sorter reject material will be emplaced in either the quarry or dump depending on quality and quarry inventory.

Dump site selection was driven by two factors:

- Proximity to mining and processing activities to optimise mobile fleet efficiency. Backhauling of material was utilised where possible; and
- Where possible, remaining within the current disturbed footprint to minimise environmental and cultural issues.

As illustrated in Figure 14, there are three waste dumps planned. Dumps 1 and 2 are adjacent to the dry processing plant, will be up to 45m above original ground level with a combined capacity of 9.44Mbcm. Dump 3 will be created in the void created by LGS mining and has a capacity of 2.92Mbcm.

Between the three dumps there is an additional 15% capacity over what is required life of mine, with the mass balance detailed in Table 32 and Table 33.

All material will be emplaced using the ADT fleet. Due to the excellent material characteristics of the waste material – high strength, low slaking, no NAF/PAF – the dump design is straightforward and driven by operational factors. Dump lifts will be 15m in height, with a single lift slope angle of 37 degrees, and an overall dump angle of 27 degrees.

As all three dumps are predominantly located on disturbed land, dump preparation and topsoil stockpiling prior to emplacement is not required. Rehabilitation of the dumps will involve profiling of the overall slope to a long-term geotechnically stable position with subsequent top soiling, erosion control structures and seeding completed in accordance with EQR's Progressive Rehabilitation Plan.

Table 32: Waste Rock Dump Requirements

Source	Volume (m^3)	Swell Factor (%)	Loose Volume (m^3)	Material to Dump (%)	Total Volume to Dump
Insitu Open Cut - HGZ	252,434	20.0	302,920	100.0	302,920
Insitu Open Cut	4,796,450	20.0	5,755,740	100.0	5,755,740
Low Grade Stockpile	2,771,576	0.0	2,771,576	65.0	1,801,525
Low Grade Stockpile	3,478,868	0.0	3,478,868	65.0	2,261,264
Total	**11,299,328**		**4**		**10,121,449**

Table 33: Waste Rock Dump Capacities

Destination	Surface Area (m^2)	Volume (m^3)
FS_Dump_1_RL380.00t	359,258	1,467,350
FS_Dump_1_RL395.00t	343,948	2,284,711
FS_Dump_1_RL410.00t	223,885	1,437,936
FS_Dump_2_RL380.00t	221,849	1,176,120

Destination	Surface Area (m^2)	Volume (m^3)
FS_Dump_2_RL395.00t	292,844	1,629,505
FS_Dump_2_RL410.00t	224,247	1,443,008
FS_Dump_3_RL385.00t	242,405	1,050,478
FS_Dump_3_RL400.00t	296,271	1,872,108
Total	**2,204,707**	**12,361,216**

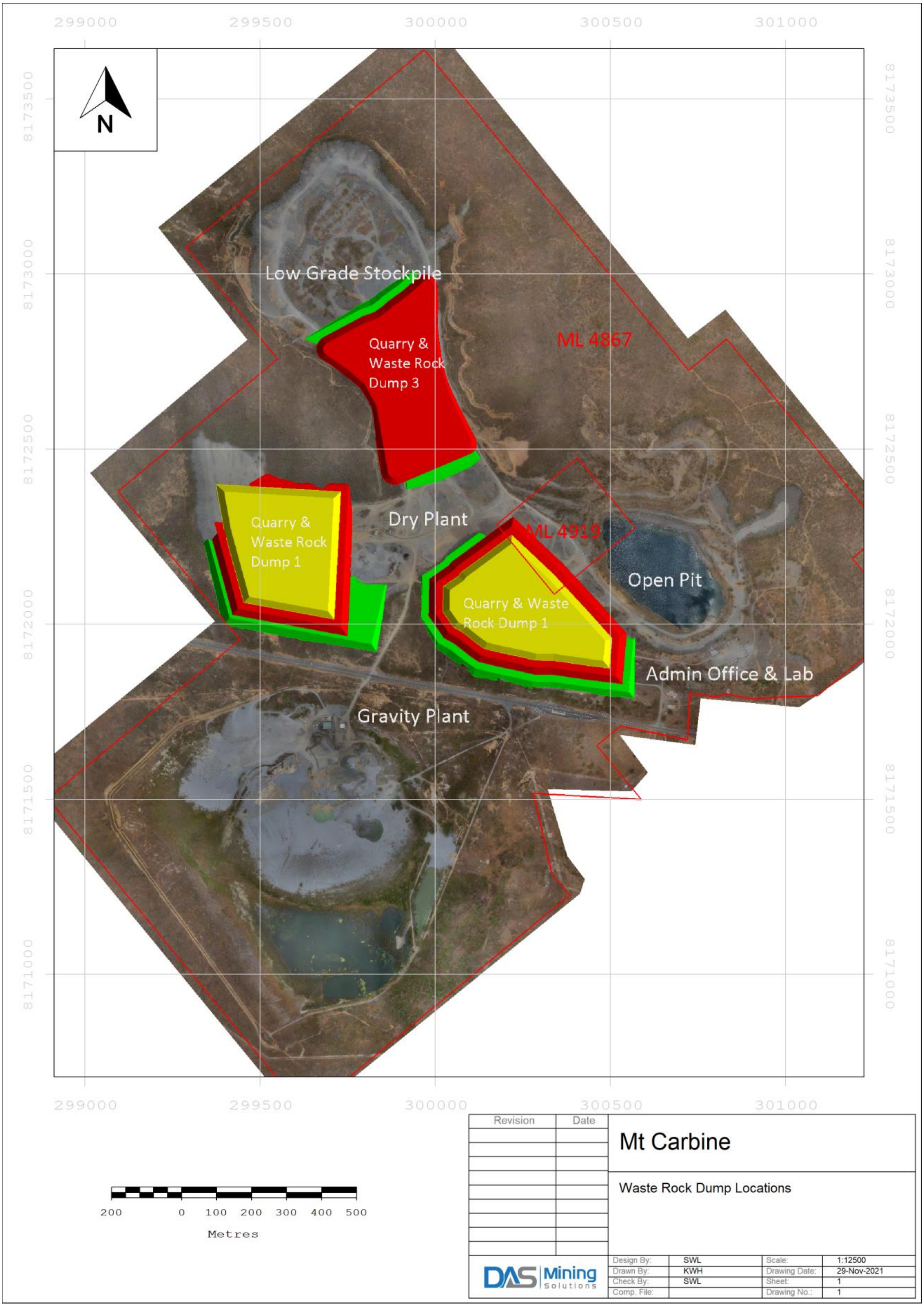

Figure 14: Waste Rock Dump Locations

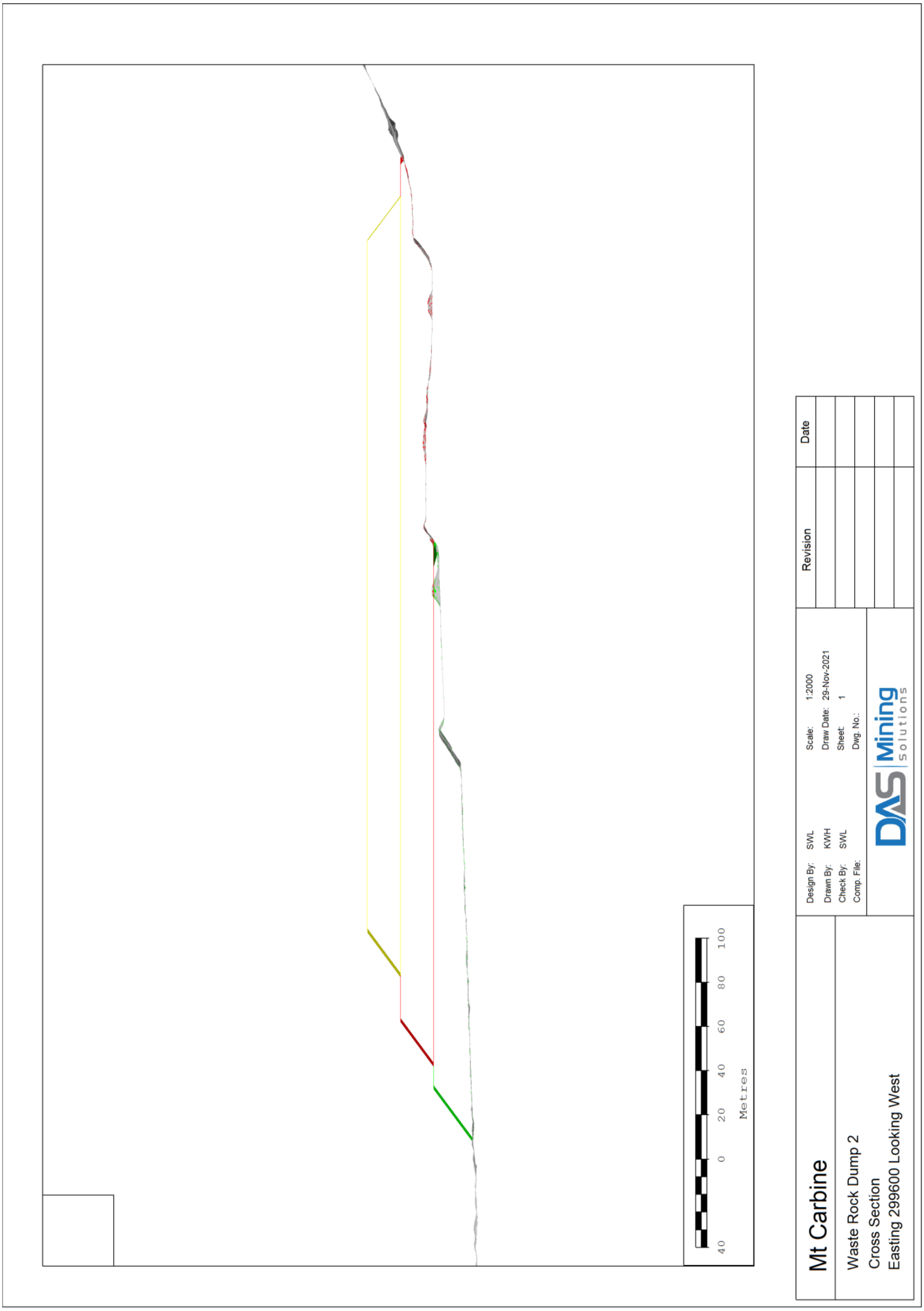

Figure 15: Waste Rock Dump 2: Cross Section

15. Equipment

15.1. Mobile Fleet

15.1.1. Mobile Fleet Equipment

Mobile fleet will be a combination of EQR owned and mining contractor equipment. Mining from both open cut and LGS will be undertaken by the mining contractor, utilising the EQR equipment as required. Table 34 details the mobile fleet to be used at Mt Carbine.

The primary excavation fleet will be a 100-120t class excavator, (such as a Liebherr 9100) and five or six 45t class ADTs. This fleet will focus predominantly on waste movement from the open cut. The EQR fleet will be utilised as the secondary fleet, primarily extracting ore from the OC or LGS.

Drilling and blasting activities will be sub-contracted out as a total load service, delivering blasted inventory on a cost per tonne basis, as part of the overall AUD4.50/t for open cut mining.

Table 34: Mobile Fleet Equipment

Equipment	Model	Capacity	Purchase Estimate (AUDk)	Project Capital Cost (AUDk)	Operational Cost (AUD/t (excl. labour)
EQ Resources Owned Equipment					
Excavator	Kobelco SK500 (2018)	50t	333	0 (leased through opex)	0.27
Articulated Dump Truck	3 x Bell B50E (2016)	45t	740	0 (leased through opex)	0.39
Front End Loader	Hyundai 980 (2016)	5.5m3	240	0 (leased through opex)	0.37
Front End Loader	Komatsu WA500	6.4m3	613	250 (remainder leased through opex)	0.37
Front End Loader	Hyundai 780A	5.1m3	-	-	0.35
Dozer	Caterpillar D6 LGP (2013)	5.8m3	160	0 (leased through opex)	0.31
Bobcat	Caterpillar 272	1.57t	50	0 (leased through opex)	-
Mining Contractor – Indicative equipment and class					
Excavator	Liebherr 9100	7.4m^3	-		All included in AUD4.50/t rate
Articulated Dump Truck	6 x Bell B50E	45t	-		
Grader	Caterpillar M120	-	-		

Equipment	Model	Capacity	Purchase Estimate (AUDk)	Project Capital Cost (AUDk)	Operational Cost (AUD/t (excl. labour)
Service Truck	Custom Heavy Rigid Truck	-	-		
Light Vehicles	3 x Toyota Landcruiser	-	-		
Water Truck	Isuzu or Similar	-	-		
Drill & Blast – Total Load Service					
Production Drill	2 x Epiroc THD (102mm hole dia.)	-	-		Included in AUD4.50/t rate
MMU	2 x Dyno Heavy Rigid Truck	-	-		

15.1.2. Mobile Fleet Productivities

To achieve the mining schedule material movement, the productivities stated in Table 35 are required.

Table 35: Mobile Fleet Hourly Productivities

Component	Productivity rate per hour (t)	Operating hours (hr)	Annual capacity (t)
Primary Mining Fleet	824	6048	5,000,000
Secondary Mining Fleet	331	3024	1,000,000
Front End Loaders (2 units)	350	6048	2,110,000

Annual operating hours for the mobile fleet are calculated as per Table 36. Maintenance will be undertaken at the OEM prescribed intervals by mining contractor and EQR qualified personnel.

Table 36: Mobile Fleet Annual Operating Hours Build-up

Variable	Value	Unit
Mine site operating days	350	Per annum
Mining operating hours	12 or 24	Per day
Equipment Availability	80	%
Equipment Utilisation	90	%
Mining effective production hours	8.64 or 17.28	Per day
Mining effective production hours	3024 or 6048	Per annum

15.2. Processing Plant

15.2.1. Processing Plant Equipment

Both the dry and gravity processing plants will be operated by EQR personnel on a 24hr, 7 day a week basis. Table 37 details the plant equipment that will be used at Mt Carbine.

Table 37: Processing Plant Hourly Productivities

Equipment	Model	Capacity (tph)	Annual Capacity (t)	Operational Cost (AUD/t (excl. labour)
Crushing, Screening and Sorting Plant (Dry Plant)				
Grizzly/Rock Screen	RSV 1400 Rockscreen	350	1,500,000	0.05
Crushing & Screening	Various	350	1,500,000	1.29
Ore Sorter	Tomra COM1200 XRT x 2 Modules	160	816,000	0.71
Processing Plant (Gravity Plant)				
Gravity Plant	-	60	408,000	7.73
Tailings Dewatering	-	60	408,000	0.34

Annual operating hours for the dry and gravity processing plants are calculated as per Table 38. Maintenance will be undertaken at the OEM prescribed intervals by EQR and third-party qualified personnel.

Table 38: Processing Plant Annual Operating Hours Build-up

Variable	Value	Unit
Mine Site Operating Days	350	Per annum
Processing Operating Hours	24	Per day
Equipment Availability	90	%
Equipment Utilisation	90	%
Processing Effective Production Hours	19.44	Per day
Processing Effective Production Hours	6804	Per annum

16. References

- Chapter 3: Geology and Resources
- Chapter 10: Environment and Approvals
- Chapter 12: Capital Cost Estimate
- Chapter 13: Operating Cost Estimate
- HD042 Slope Stability Analysis and Design of the Open Pit Slopes (Piteau 1982)
- Report on Carbine Tungsten Groundwater Study (Rob Lait and Associates 2012)

17. List of Abbreviations

Abbreviation	Description
ADT	Articulated dump truck
BCM	Bank cubic metre
EQR	EQ Resources Limited
FEL	Front end loader
HGZ	High grade ore zone
IDF	Iron Duke Fault
JORC	Australasian Joint Ore Reserves Committee
LGS	Low grade ore stockpile
LOM	Life of mine
NAF	Non-acid forming
OC	Open cut
PAF	Potentially acid forming
RL	Relative level
RQD	Rock quality designation
SWF	South Wall Fault
UCS	Unconfined compressive strength
VSI	Vertical impact shaft crusher

Appendix A Geotechnical Report

R.B. Mining Pty. Ltd.
Mount Carbine Mine
Queensland, Australia.

SLOPE STABILITY ANALYSIS AND DESIGN OF THE OPEN PIT SLOPES

April 1982

Piteau & Associates

(Copy)

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MOUNT CARBINE MINE

REPORT

ON

SLOPE STABILITY ANALYSIS

AND

DESIGN OF THE OPEN PIT SLOPES

Prepared by

Douglas R. Piteau
Dennis C. Martin
and
Eric J. Byres

Project 81-358 April 1982

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1. INTRODUCTION

The purpose of this report is to describe the results of studies which were carried out concerning slope stability analyses and related design for the Mount Carbine Mine. The scope and nature of the proposed geotechnical work were discussed with mine personnel during an initial visit to the mine by D.R. Piteau in late February, 1981. The terms of reference for this work are described in a proposal letter dated April 27, 1981 from D.R. Piteau to D. Wolff of R.B. Mining Pty. Limited.

The project was initiated on June 15, 1981 when D.C. Martin and E.J. Byres of Piteau & Associates visited the mine to organize the field mapping and mine office studies. Mr. E. Byres remained at the site until late November, 1981 to complete the geologic structural mapping, drilling studies, core logging, associated field work and preliminary analysis of the field data. Mr. D. Martin revisited the mine in early September, 1981 to review the program and to initiate a geotechnical drilling program. A rock strength testing program was also started at this time at James Cook University, Civil Engineering Department.

Significant findings of the study relating to engineering geology, analysis techniques, results and our preliminary slope design recommendations were discussed by D.R. Piteau, D.C. Martin and E.J. Byres with E. Brachmanski by telephone on January 13, 1982 and February 10, 1982. Draft copies of the report were prepared in late March, 1982. D.R. Piteau visited the mine in early April, 1982 at which time all aspects of the study and design recommendations were presented to mine personnel.

2. DESCRIPTION OF THE INVESTIGATION

2.1 FIELD DATA COLLECTION AND DATA COMPILATION

2.1.1 Geological Mapping

Geological mapping was carried out on all accessible benches in the open pit. This work involved engineering geology assessments relating to lithology, rock strength and geological structures (such as faults, shears, joints and geological contacts) to determine the physical and mechanical properties of the slope forming materials. Detailed line geological mapping techniques were used for the bench mapping and data was recorded on standard field sheets for computer processing and analysis.

Studies of rock competency, degree of fracturing, degree of weathering, rock hardness and bench face angles were conducted during the mapping to assist in evaluating the rock mass and related behaviour of slopes.

All major faults were carefully mapped and considered as thoroughly as possible. A review of historical geological information, level plans and geological sections was carried out to incorporate all relevant information in terms of defining the overall geologic model as accurately as possible. Other documented information concerning the geology, engineering properties and other aspects relevant to the problem were also reviewed.

The geological mapping information was used to develop a composite geology map and geological sections. This mapping showed the location and spatial relationship of major features, such as faults, geological contacts, etc. A realistic model was developed which could be used to determine both the location of the significant geologic structural features and the distribution of the various rock types on the proposed final pit wall. The structural geology and engineering geology of the open pit is shown in plan in Fig. 1 and on sections in Appendix A.

2.1.2 Geotechnical Drilling and Core Logging

A geotechnical drilling program, consisting of two diamond drillholes and eleven percussion drillholes, was carried out in the area of the south wall. Samples from these holes were logged and the results were compiled into sections showing lithology, weathering and rock strength profiles in the south wall. Drill logs are included in Appendix B. In addition, foliation angles were recorded in the diamond drillholes and were used to orient the core so that joints in the core could be analyzed statistically. The locations of the South Wall Fault at depth was also accurately determined from these drillholes.

Relogging of existing diamond drillholes was also carried out during the drilling program. Relevant sections of these holes were logged for lithology, weathering, breakage, foliation, RQD and joint frequency. The results were analyzed using the cumulative sums technique. Intersections of the South Wall Fault in existing drillholes was used to assist in locating the fault accurately and to develop a structure contour map of the fault.

2.1.3 Hydrogeological Investigations

In conjunction with the geotechnical investigation, a brief hydrogeological investigation and assessment was carried out. The purpose of this investigation was to determine if it would be necessary to depressurize the pit slopes, and hence improve pit slope stability. The study concentrated on the south wall since this is a particularly critical area of the pit.

Twenty piezometers and five standpipes were installed in the geotechnical drillholes to monitor water pressures in the south wall. Water levels were recorded by mine personnel throughout the rainy season and falling head tests were conducted in each piezometer to assess the permeability of the rock mass. In addition, a review of climatic data supplied by the mine was carried out.

2.2 STRENGTH TESTS AND BACK ANALYSIS OF FAILURES

2.2.1 Strength Tests

To obtain realistic strength data, four direct shear strength tests were performed on selected samples of open discontinuities. These tests were carried out to determine the relevant friction and cohesion properties of joints and foliation joints on the south wall area. The samples used in these tests were core samples obtained from diamond drilling.

A series of point load tests and unconfined compressive tests were performed to evaluate the intact rock strength of the schists and greywackes in the south wall. Both weathered and unweathered samples were collected and this allowed a realistic appraisal of the effect of weathering on rock strength to be made. The unconfined compressive strength tests were performed by Dr. H. Bock of James Cook University. Point load index tests were performed both at James Cook University as well as by Piteau & Associates.

2.2.2 Back Analysis of Failures

In order to obtain a better appreciation of the behaviour of the slope and the shear strength parameters of discontinuities involved in failures, particularly in the weathered schist and related rocks in the south wall, back analyses were carried out. Rotational failures in weathered rock on Bench 375 and a plane failure on Bench 365 were analyzed and strength parameters were developed. Road cuts along the Rex Range Highway near the Mount Carbine Mine were also documented and analyzed. The similarity of these road cuts, in terms of lithology, rock strength, weathering and structure, to the upper benches of the south wall of the pit make it possible to apply the results of this study to the design of the south wall.

2.3 GEOLOGIC STRUCTURAL ANALYSES

Geologic structural analyses and slope stability analyses were carried out using the geological data and test results. Geological mapping data was processed using a desktop computer at the mine to plot equal area projections. Statistical methods were used to define the basic geologic structural parameters and to develop an accurate appreciation of the nature and distribution of these features in the rock mass.

2.4 STABILITY ANALYSIS AND SLOPE DESIGN

Based on the markedly different engineering geology, character and constraints on slope design, slopes south of the South Wall Fault were considered separately from the remainder of the pit. For this reason, two separate sections on analysis and design are presented. The first section considers stability analysis and slope design in the bulk of the pit; the second section considers design and remedial measures for slopes south of the South Wall Fault.

2.4.1 Stability Analyses and Design Based on Design Sectors

Based on the results of the structural analysis, the pit was divided into areas where the rock type, strength and geological structure are similar in a statistical sense, i.e. structural domains. The final pit was divided into design sectors within which the orientation of the proposed ultimate slopes in the individual structural domains are similar. Separate slope stability analyses were carried out, and related design criteria were established for each design sector. Equal area projections were used to determine the kinematically possible failure modes which are likely to control slope design in each design sector. Mechanical stability analyses were carried out for possible failure modes. When a large number of possible failure modes exist in a particular design sector, statistical analyses were used to evaluate these potential failures in the process of determining optimum slope design.

Slope designs were developed with due consideration to the possibility of instability involving both benches and large sections of the overall slope. Consideration was also given to effects of plan radius of curvature, variation of geological structure, strength and groundwater conditions in the final open pit.

2.4.2 Analysis and Design of Slopes and Remedial Measures South of the South Wall Fault

Design of slopes and remedial measures in weathered schists and greywackes on the south wall above the South Wall Fault was carried out with due consideration of both deep seated rotational failures in the weathered rock as well as failure along discontinuities. The occurrence of a well developed weathering profile, weak rocks, unfavourably oriented discontinuities and critical nature of the slope was considered in preparing a slope design and remedial measures. Geological mapping, drilling, strength testing and back analyses results were used to prepare a rational design and remedial measures, with due consideration of the constraints imposed by the nature of the slope forming materials and location of the existing primary crusher and roads near the crest of the slope.

3. REGIONAL GEOLOGY

The Mount Carbine area is underlain by deep-water turbidite sequences of the Hodgkinson Formation, which consists mainly of metasediments of Middle Devonian to Lower Carboniferous age. These sediments were intruded by late-orogenic and post-orogenic granites and unconformably overlain by Permian and Triassic sediments. The region has undergone four stages of folding and low grade metamorphism (Amos, 1968).

At the mine, the Hodgkinson Formation consists of thinly bedded, intercalated units of greywacke, quartzitic greywacke, siltstone and shale, with minor chert and basic volcanics. Within the pit area, regional metamorphism as well as local and contact metamorphism from nearby intrusions have altered the sedimentary rocks to micaeous schists, siliceous greywackes, hornfels, phyllites and slates in addition to the original lithologies. Within the ore zone, extreme pneumatolytic tourmalinization and silicification has occurred, altering the engineering characteristics of the rock significantly.

Several large batholiths of Mareeba Granite can be found near Mount Carbine. The most dominant intrusion, which consists mainly of biotite-muscovite granite, forms the Mount Windsor and Mount Carbine tablelands. It is believed that this intrusion was responsible for the contact metamorphism which led to hydrothermal alteration, silicification and emplacement of the tungsten bearing quartz veins at Mount Carbine.

The quartz veins strike east-west and are distributed en-echelon along Mount Carbine Hill. The southern boundary of the quartz veins and significant silicification is defined by the South Wall Fault. This fault is the most significant structural feature in the pit.

According to Amos (1968), at least four stages of folding have been documented in the region. At Mount Carbine the first two phases of deformation have been completely masked by the third phase of deformation, which resulted in the development of a strong axial plane cleavage which obliterated the original bedding cleavage in the grewackes and siltstones. This axial plane cleavage now appears as generally steep dipping foliation throughout most of the mine. The fourth phase of deformation occurs as folding of the foliation cleavage.

4. LITHOLOGY

The basic engineering geology, lithology and structural geology are shown on plan in Fig. 1 and on geotechnical sections in Appendix A. A general description of the main rock types in the mine are given in the following.

4.1 HORNFELS

Hornfels is the predominant rock type in the pit (see Photo 1). Hornfels hosts the tungsten bearing quartz veins, giving rise to its common name of "Host Rock". However, hornfels does occur in areas without mineralization or quartz veins, and hence the term host rock is not used in this report.

The hornfels is a highly tourmalinized and silicified version of the surrounding greywackes and siltstones, which have undergone low grade metamorphism. This alteration has occurred throughout the rock mass without regard for original lithologic boundaries, and has masked most of the original lithologic differences. For purposes of this study, the hornfels has been divided into two main lithologic types based on present characteristics and estimated original composition as described in the following. This division is preliminary only, and careful petrographic investigation is needed before accurate divisions can be made.

4.1.1 Laminated Hornfels (LHF)

Laminated hornfels (LHF) is derived from the argillaceous and micaceous siltstone facies common in the region. This rock is dark grey, fine grained and contains up to 30 percent fine, light coloured quartz rich laminations. Laminations may be planar and uniform or highly twisted and deformed, giving the rock an irregular, gneissic texture.

PHOTO 1: View of hornfels and related rocks on the north wall of the open pit.

Unweathered, laminated hornfels is tough and has a hardness of between R4 and R5 (i.e. an unconfined compresive strength of greater than 8000 psi (56 MPa)).* Laminated hornfels tend to fracture along foliation when hit with a rock hammer, giving the rock a blocky appearance.

Weathered laminated hornfels has a hardness of R2 and R3, i.e. an unconfined compression strength of 1000 to 8000 psi (7 to 56 MPa), depending on the amount of siliceous banding and degree of weathering. Weathered rocks are a dark olive green colour. Extremely weathered rocks are bleached.

4.1.2 Banded Hornfels (BHF)

Banded hornfels (BHF) is derived from arenaceous greywackes that have undergone quartz segregation prior to tourmalinization and silicification. They are composed of 5mm to 20mm quartz rich bands separated by thin layers of dark argillaceous rock. These bands can comprise between 60 to 90 percent of the total volume of the rock. Bands are usually regular, but extreme deformation and boundinage is not uncommon.

Unweathered, banded hornfels is light blue-gray and has a glassy appearance. Banding may be obscured by silicification. Banded hornfels is generally less brittle than the laminated hornfels, and has an estimated hardness of R5 (unconfined compressive strength greater than 16,000 psi (112 MPa). There is no preferred fracture direction and banded hornfels rarely breaks along foliation.

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* A complete description of hardness as related to unconfined compressive strength is given in Fig. 9.

Weathered banded hornfels has a hardness of R3 (unconfined compressive strength of 4000 to 8000 psi (28-56 MPa). This rock has a rusty red brown colour and banding is highlighted by weathering.

4.2 IRON DUKE ROCKS

Iron Duke rocks are those units which have undergone low grade metamorphism, but have not been affected by the extreme tourmalinization and silicification noted in hornfels. Iron Duke rocks are formed from the same facies as the hornfels. Iron Duke rocks were noted on the south wall of the open pit above the South Wall Fault (see Photo 2). Iron Duke Duke rocks were also noted on the flanks of Mount Carbine hill outside the ore zone.

4.2.1 Siliceous Greywacke (GWK)

Siliceous greywacke (GWK) consists of bands, stringers, augens and clasts of quartz and feldpathic material in a variable matrix of silicified argillaceous material. Grain size is highly variable and large clasts of chert are common. Cherty zones and schistose zones several metres thick are common. Contacts between units of greywacke are gradational. The degree of silicification is variable and this effects the degree of weathering and the rock strength. Foliation is evident, but is often twisted and contorted. Such contortions may contribute in a substantial way to shear strength of the rock mass.

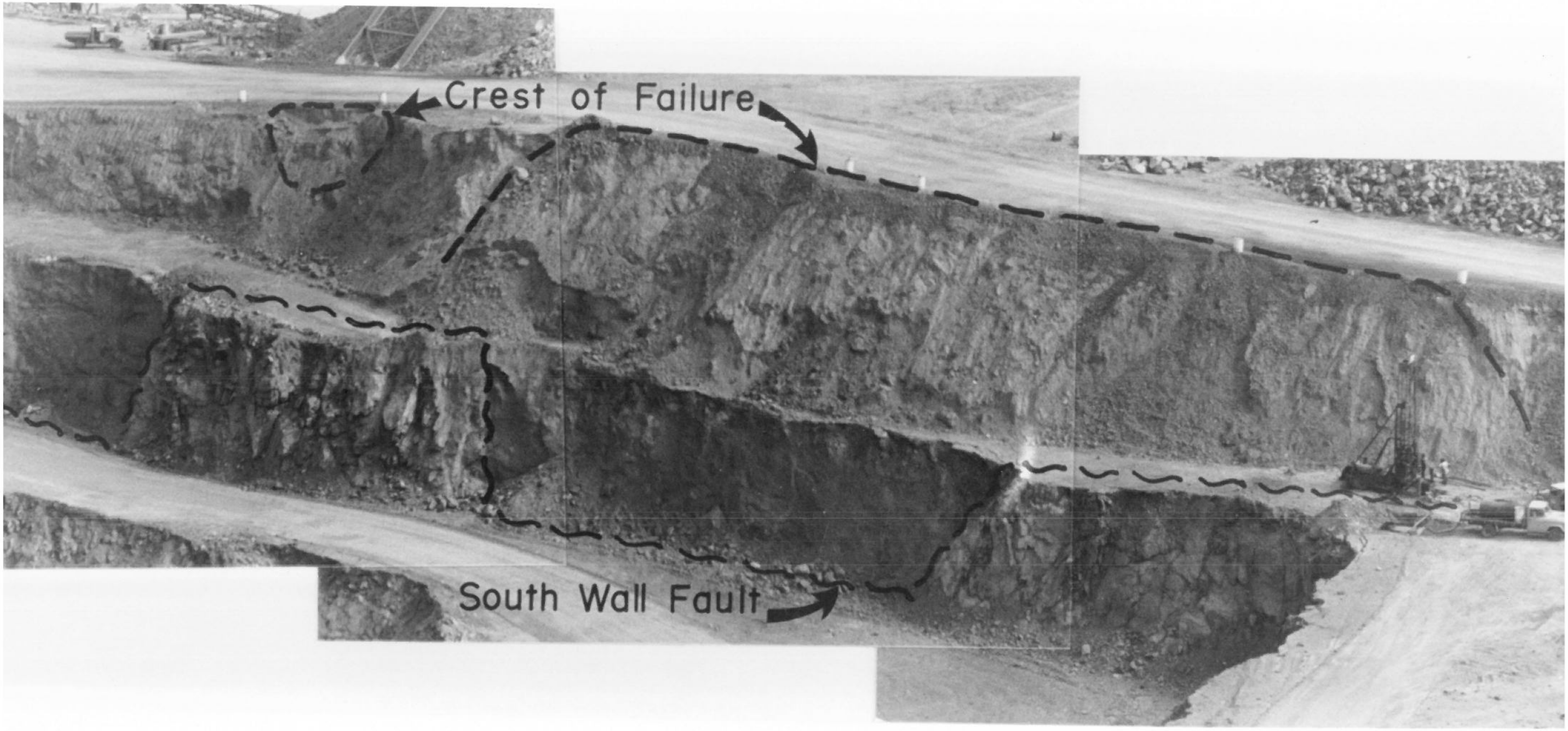

PHOTO 2: View of south wall showing location of South Wall Fault and highly weathered schists near the pit crest. Note rotational failures in highly weathered rock above 375m elevation.

Unweathered, siliceous greywacke is light or dark grey. Cherty zones have a slight green cast. Some of the mottling is probably caused by differences of grain size rather than mineralology, but the majority is caused by to the bands and stringers of quartz-feldspar material. The rock has a hardness of R3 to R4 (i.e. unconfined compressive strength of 4000 to ~~10000~~ 8000 psi (28 to 56 MPa).

Weathered silicious greywacke develops a rusty overall colour with well defined mottling and banding. The argillaceous matrix is considerably weakened by weathering, with the result that hardness drops to between R0 and R2 (unconfined compression strength of 100 to 4000 psi (0.7 to 28 MPa)).

Siliceous greywacke has been found in drillholes behind the south wall and in highly weathered outcrops on the east and west ends of the south wall. No unweathered surface outcrops have been located. This rock is expected to form part of the upper benches of the final pit, above the South Wall Fault.

4.2.2 Green Argillaceous Schist (GAS)

Green argillaceous schist (GAS) is found on the south wall of the pit where it forms a band or lense adjacent to the South Wall Fault (see Fig. 1 and Photo 1). The rock has a well developed undulating schistosity and is weakly silicified. Argillaceous schist grades into the siliceous greywacke, becoming more siliceous and containing occasional stringers and blobs of light coloured, quartz rich rock. Quartz stringers are oriented along laminations at approximately 40^{o} to the schistosity.

Unweathered argillaceous schist is dark green to dark grey and has a hardness of R3. Light green mottling, which is caused by slight changes in grain size, is common but has no affect on strength. Once weathered, argillaceous schist loses strength rapidly; slightly to moderately weathered schist has a hardness of R2 and heavily weathered schist has a hardness between S4 and R1. Fresh rock exposed to weathering is likely to deteriorate rapidly due to its low silica content.

The upper benches in the centre of the south wall are expected to be composed of argillaceous schist.

4.2.3 Black Slate (BST)

Black slate (BST) was noted in drillholes and outcrops behind the south wall. Black slate is very thinly laminated and has well developed slatey cleavage. It is uniform in both composition and degree of lamination and has a hardness of R3 when unweathered. Weathered samples have not been found, but very low strengths are likely.

Black slate is unlikely to appear in the walls of the final pit, but is mentioned here as it occurs close to the crest of the final open pit and is a major lithologic unit in the mine area.

4.3 INTRUSIVE AND VOLCANIC ROCKS

4.3.1 Felsite (FLS)

Felsite is a pale creamy coloured quartz-porphyry intrusive, which forms an approximately 5m wide dyke visible on the west side of the pit. It is coarse grained (except along the chill margins) and is acidic in composition. Unweathered, felsite has a hardness of R4. Felsite appears to weather rapidly and can have a hardness as low as R1. Weathering is most severe along chill margins. Stiff green clay, which probably occurs as a result of weathering, has been noted in the dyke at surface and in drillhole CB13, over 75m below the surface. Hardness of the clay is S5 (i.e. unconfined compression strength of 14 to 28 psi).

The felsite dyke is expected to occur on the west wall of the final pit, although its exact location is not clear due to the variable orientation of the dyke boundaries.

4.3.2 Andesite (ADS)

Several dykes up to 2m wide of purple grey, fine grained intermediate andesite (ADS) were mapped in the pit. These rocks have a hardness of R4. Moderate to severely weathered andesites have a hardness of R2 to R3. Andesites may form relatively small dykes on the east wall of the final pit.

4.4 SILICIFICATION

The most significant silicification and tourmalinization has occurred in the hornfels north of the South Wall Fault and west of Iron Duke Fault. Silicification has resulted in increased rock strength, increased resistance to weathering and general increase in competency of the rock mass. In the remaining areas of the pit, particularly in the schists and greywackes south of the South Wall Fault the rock is either slightly silicified or completely unsilicified.

The South Wall Fault and Iron Duke Fault appear to control the degree of silicification in the pit. East of the Iron Duke Fault the size and number of quartz veins and degree of silicification decreases gradually with the result that the rock becomes more susceptable to weathering. South of the South Wall Fault the rock is only slightly silicified. There are no mineralized quartz veins and the rock is extremely susceptible to weathering. This sharp contact of silicification may be related to the actual offset of the fault. Alternatively, it is noteworthy that the silicification in the hornfels increases as the South Wall Fault is approached. This suggests that the fault may have existed prior to silicification and acted as a dam to the migration of silica and tungsten bearing fluids. Detailed study of the relationship between the faults and ore emplacement is beyond the scope of this study.

4.5 WEATHERING

Rocks in the upper part of the open pit form a well developed weathering profile. This profile is particularly well developed in weaker rocks on the south wall. The weathering profile is clearly defined from surface mapping (see Fig. 2), in surface outcrops (see Photo 3) and from examination of diamond drill cores and percussion drillhole chip samples as shown in Photos 4 and 5.

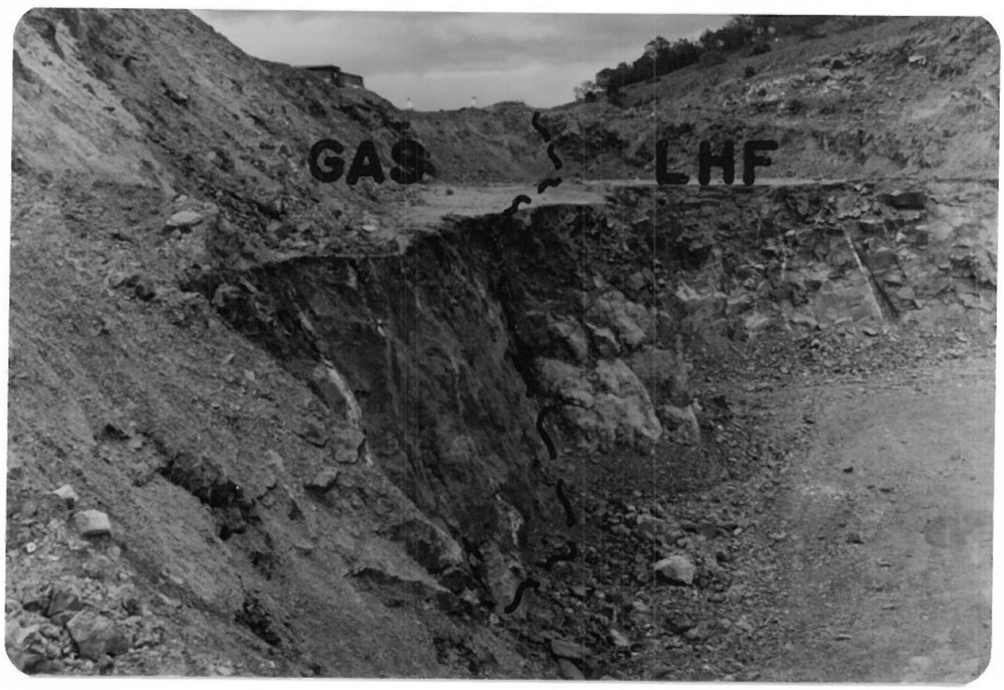

PHOTO 3: View of South Wall Fault in southwest corner of the pit. Note that silicified hornfels north of the fault is less deeply weathered than green argillaceous schist south of the fault.

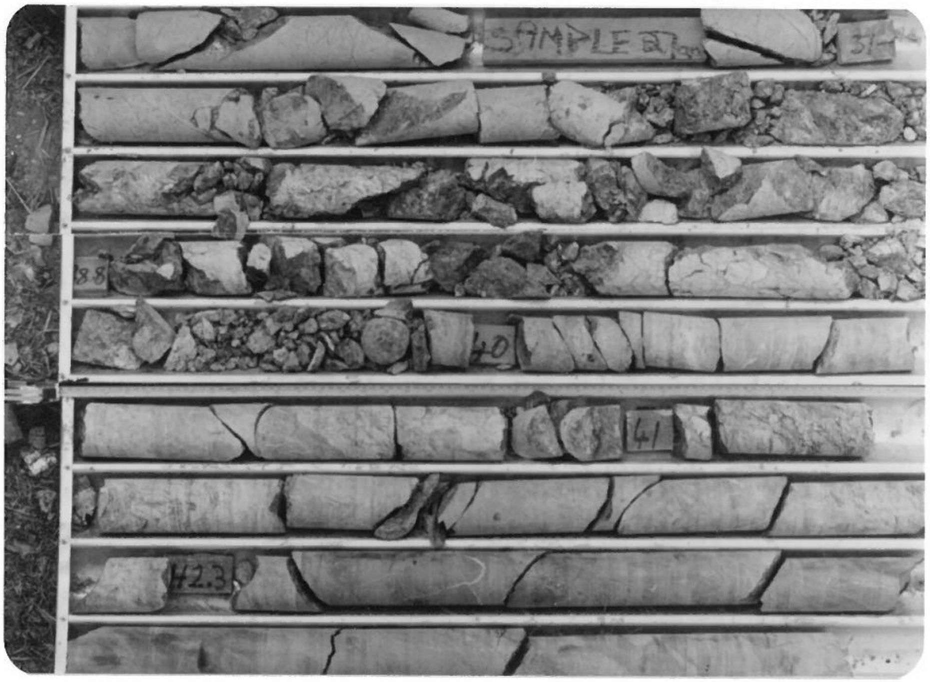

PHOTO 4: View of core in drillhole CB-20 showing marked decrease in weathering and increase in rock strength at a depth of about 40.1m.

PHOTO 5: Illustration of decrease in degree of weathering with depth in chip samples from percussion drillhole MCP4.

Residual soil and rocks within the weathering profile have been divided into five categories, based primarily on the degree of weathering and related rock strength. These categories range from extremely weathered residual soil (Category A) to hard unweathered rock (Category E). Definitions of each weathering category are given in the legend in Fig. 2.

The distribution of weathered zones in the open pit, based on geological mapping and core logging, is shown in Fig. 2 and on sections in Appendix A.

Assessment of information on weathering indicates that hornfels and related silicified rocks have a very limited weathering profile, whereas less silicified rocks south of the South Wall Fault have a well developed weathering profile. The contrast of depth of the weathering profile across the South Wall Fault is clearly illustrated in Photo 3.

Weathering has no significant effect on rock strength below the first bench in the silicified hornfels north of the South Wall Fault. In the less silicified rocks south of the South Wall Fault, a weathering profile up to 30m deep is developed as shown in Fig. 2 and Appendix A. The markedly different strength of the rocks within the weathering profile, and increase in strength with depth, indicates that a different slope design will be required for each succeeding bench in this area.

A summary of the weathering profile in rocks south of the South Wall Fault is given in Table I. Detailed discussion of the variation of rock strength within the weathering profile is given in Section 6.

5. STRUCTURAL GEOLOGY

Rational slope stability analysis and pit slope design requires that the pit be subdivided as best as possible into areas of approximately similar geologic structural characteristics. The engineering behaviour of the slope forming material can be expected to differ in different parts of the pit which have different geologic structural characteristics. Extrapolation of stability analysis results and slope design criteria is valid only within parts of the rock mass which have physical and mechanical properties which are similar in a statistical sense. Such areas with similar geological structure are called structural domains.

Attitude of geological structures is the most important consideration in determining whether geologic structural populations within a structural domain or between structural domains are similar or dissimilar. Other parameters, such as continuity (joint extent or size), joint infilling, waviness, etc., are also considered in evaluating the engineering properties and nature of joint sets, but are not used for designation of structural domains.

Structural domains at Mount Carbine were determined initially by assuming the boundaries of the structural domains were major faults, geological contacts and/or boundaries of areas with similar foliation. Attitudes of the geologic structural populations within different parts of each structural domain then were evaluated and compared. Six preliminary structural domains (i.e. Structural Domains 1 to 6) were chosen as shown in Fig. 1. In order to further compare populations within the structural domains, several structural sub-domains (i.e. Structural Domains 5A to 5D, 6A and 6B) were chosen based on variation in foliation dip.

Equal area projections were used to define the peak or average attitude and possible range of attitudes for the various sets of discontinuities in the individual structural domains. Four major joint sets, foliation, several minor joint sets, a vein set and two fault sets have been recognized in the open pit. The nature, distribution and possible genesis of these structural features are discussed below. Average orientation of discontinuity sets based on detailed line mapping are summarized in Table II.

5.1 FOLIATION

5.1.1 Nature and Distribution of Foliation

Foliation is a laminated structure developed parallel to "compositional layering" in the metasediments as shown in Photos 6 and 7. Foliation is well developed throughout the pit, although it is often difficult to recognize in unweathered hand specimens.

Equal area projections of foliation in structural domains are shown in Fig. 3 and summarized in Table II. These plots clearly show the variation of foliation orientation in the various structural domains. It can be seen that there are marked differences in orientation in argillaceous schist and greywackes south of the South Wall Fault. It also noteworthy that foliation is more variable in the area south of the South Wall Fault.

Foliation in hornfels dip steeply to the northeast or southwest at about 70° to 90°. Average dip directions vary from 041° to 056° and 221° to 239°. Iron Duke rocks south of the South Wall Fault have foliation that dips approximately due north at about 75°.

PHOTO 6: View of foliation and foliation joints in weathered banded hornfels in the southeast corner of the pit.

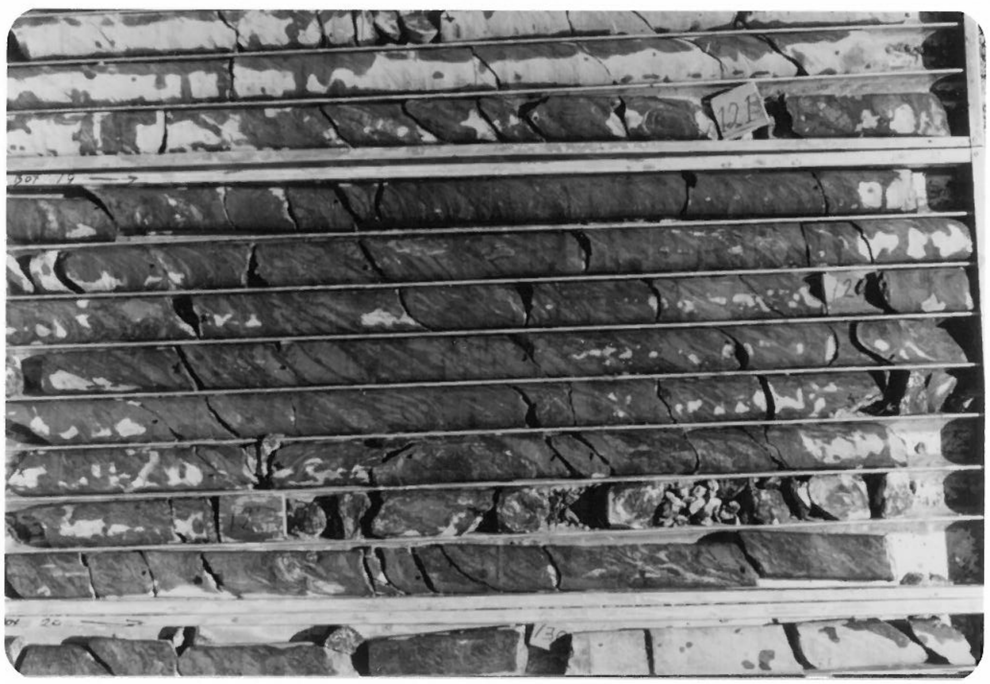

PHOTO 7: View of foliation in core from drillhole CB20.

Within individual structural domains there are marked variations in foliation dip due to the presence of minor folds. As a result, it is extremely difficult to predict variations in foliation dip or to delineate areas of consistent dip in the pit.

5.1.2 Assessment of Variation of Foliation Using the Cumulative Sums Technique

Foliation dip was assessed statistically using the cumulative sums (cusums) technique. This technique provides a rapid and precise method of determining major trends above or below a particular reference value (generally the mean), and of ascertaining both the magnitude and location of these variations. Complete details of the construction and interpretation of cumulative sums plots are given in Appendix C.

Data for the cusums analysis was collected at 3m intervals along all benches during the detailed line mapping. In addition, foliation angles were recorded at 3m intervals in drill core from selected drillholes.

The results of the cusums analysis of foliation dip were plotted on a current pit plan and on sections. Spot checks of foliation in critical areas and foliation measurements made by mine personnel during production face mapping were also plotted for comparison.

This information was used as a basis for defining zones with similar foliation dip and to prepare a three dimensional model of the distribution of foliation dip. The distribution of these zones on the current pit plan is shown in Fig. 3.

5.1.3 Selection of Structural Domains Based on Areas of Similar Foliation Dip

Distribution of zones of similar foliation dip direction and dip was used to assist with selection of structural domain boundaries as shown in Fig. 3. Within individual structural domains the foliation is expected to have a reasonably consistent dip direction and dip. In some cases the dip direction and dip of the foliation is variable over small areas. For engineering geology purposes, the foliation in these areas is assumed to have two or more dominant orientations at any one location. Areas of relatively flat dipping foliation, such as those mapped on the north wall or west wall, are included with steeper dipping foliation in each structural domain (see Fig. 3). In terms of slope stability analyses, these flatter foliation dips are considered in the slope design only in areas where they are considered to be statistically significant.

5.1.4 Origin of Foliation

Foliation in hornfels and related rocks appears to be related to the metamorphism and folding described by Amos (1968).

Within the schists and siliceous greywackes in Structural Domains 1 and 2 the foliation is variable, but generally oriented subparallel to the South Wall Fault. This suggests that foliation in these rocks may be, in part at least, related to the shearing and displacement along this fault. Foliation dips steeply to the northeast in hornfels in

Structural Domains 5D and 6A in the areas adjacent to the South Wall Fault. Confirmation of foliation origin and orientations in the argillaceous schists and siliceous greywacke is difficult due to the lack of good unweathered exposures of these rocks.

5.2 JOINTS

Lower hemisphere equal area projections were used to define the peak or average attitude, and possible range of attitudes, for each main joint set mapped in individual structural domains. The distribution of joints within the various structural domains (as determined from detail line mapping) is shown in Figs. 4 and 5. Average orientations of the main joint sets are summarized in Table II.

Equal area projections indicate that most joint sets occur as reasonably tight clusters, with moderate scatter in the data. In most cases, it is reasonable to represent each joint set by a single peak orientation. However, some joint sets are diffuse enough that they may be represented by two or three different peaks. In such cases the joint set is represented by a number of subsets, e.g. Joint Set B1, Joint Set B2, etc.

In addition to foliation joints (Joint Set A), three major joint sets and several minor and miscellaneous joint sets occur within the pit area. Joint Sets A and B occur consistently throughout the pit, while Joint Sets C and D occur in all domains north of the South Wall Fault. Minor and miscellaneous joint sets occur in specific domains, and only appear to be significant on a local basis only. In most cases, differences in attitude of joint sets are evident from one structural domain to another. The main joint sets are described in the following:

5.2.1 Joint Set A

Joints of Joint set A are foliation joints and their distribution is shown in Fig. 4. Average strike orientation of foliation joints is generally within 5° of peak orientation of foliation in any structural domain.

Foliation joints have an estimated average spacing of 0.5m and are generally less than five to ten metres long. Small rolls, undulations and waves are common. Foliation joints in hornfels and related rocks are moderately rough, whereas foliation joints in argillaceous schists and siliceous greywacke are generally rough and irregular.

Foliation joints are well developed in weathered areas, but become infrequent in unweathered rocks in the centre of the pit. Typical foliation joints are shown in Photos 6 and 7. Infilling of serpentine and/or chlorite are common on foliation joints in schists and greywacke. Foliation joints in hornfels are generally unfilled.

5.2.2 Joint Set B

Joints of Joint Set B are the most well developed in the entire pit. These joints form the host set for tungsten-bearing quartz veins. The similarity of peak orientations of vein sets with peak orientations of Joint Set B has been used as a selection criteria to designate Joint Set B from other joint sets.

Joints of Joint Set B are long and continuous, often extending over several benches. In most areas of the pit these joints have an average dip direction between 324^{o} and 360^{o}, and an average dip greater than 80^{o}. In structural domains adjacent to the South Wall Fault, average dip direction varies from 324^{o} to 020^{o} and average dips vary from 55^{o} to 76^{o}. Joint Set B characteristically occurs as two distinct subsets, with a 20^{o} to 30^{o} separation of average dip direction.

Joints of Joint Set B have a mean roughness of 2.7. Slickensides and other evidence of shearing are common. Typical joints of Joint Set B are shown in Photo 8.

The origin of Joint Set B is not definitely known. It is worth noting that the peak orientations of these joints is consistently 60^{o} from the peak orientation of foliation. Whittle (1969) states these joints are tension joints, based on the matching wall infilling of the quartz veins. If this is true, slickensides on joint would have developed during a later stage of deformation.

5.2.3 Joint Set C

Joints of Joint Set C are well developed in hornfels north of the South Wall Fault (see Photo 9). These joints dip shallowly to the east, with mean dips between 18^{o} and 36^{o} and mean dip directions between 060^{o} and 115^{o}. There is a possibility that the dip increases with depth, but this cannot be confirmed without further mapping as the pit is deepened.

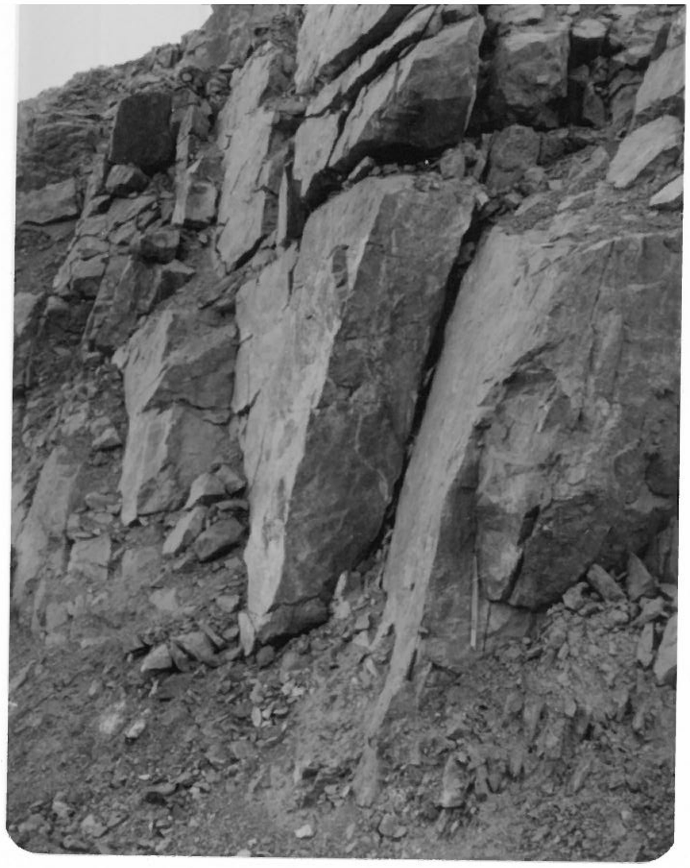

PHOTO 8: View of typical smooth continuous joints of Joint Set B.

PHOTO 9: View of flat lying joints of Joint Set C and steep dipping joints of Joint Set D in laminated hornfels.

Joints of Joint Set C are typically long, continuous and closely spaced, as shown Photo 9. These joints have a roughness between 3 and 5. Significant water flow has been observed along joints of Joint Set C during periods of heavy rain.

5.2.4 Joint Set D

Joints of Joint Set D are well developed in all hornfels north of the South Wall Fault (see Photo 9). The joints dip steeply to the west and northwest, with average dips of 80° to vertical. In Structural Domains 5A to 5D, average dip directions are between 275° and 299°. In Structural Domains 4, 6A and 6B, average dip direction is between 315° and 327°. This variation of dip direction was used to separate Structural Domains 5 and 6.

5.2.5 Joint Set E

Joints of Joint Set E are subparallel to foliation, but are distinct from foliation joints. These joints are weakly developed in Structural Domains 3 and 6B and are poorly developed in other Domains. Average dip direction is 050° and average dip is between 72° and 76°. Joints of Joint Set E are short and discontinuous, rarely extending over 5 metres. These joints have an average roughness of 3 or greater.

5.2.6 Joint Set G

Joint Set G is weakly developed in Structural Domains 1, 2, 5A, 5C and 6B. Average dip directions vary between 340° and 009° and dip varies between 43° and 50°. Although often widely spaced, joints of Joint Set G may be continuous, with trace lengths of 10 metres and greater. Roughness of less than 3 are not unusual for these joints. A plane failure along a joint of Joint Set G occurred in Green Argillaceous Schist on bench 375. A typical joint of Joint Set G in schist is shown in Photo 10.

5.2.7 Miscellaneous Joint Sets

In addition to the joint sets described above, several miscellaneous joint sets have been mapped in the open pit. These joint sets vary considerably and may be significant on a local basis only.

5.3 VEINS

Tungsten bearing quartz veins are found in hornfels north of the South Wall Fault. All tungsten ore mined in the pit is contained in these veins.

The quartz veins are sub-vertical, and occur subparallel to joints of Joint Set B or along pre-existing joints of joint set B. Two subsets are common in most areas, corresponding to the subsets in Joint Set B. Distribution of veins in the open pit is shown in Fig. 6 and summarized in Table II.

Thickness of quartz veins varies from a few centimetres to one to two metres. They are very continuous and can often be traced over several benches (see Photo 11).

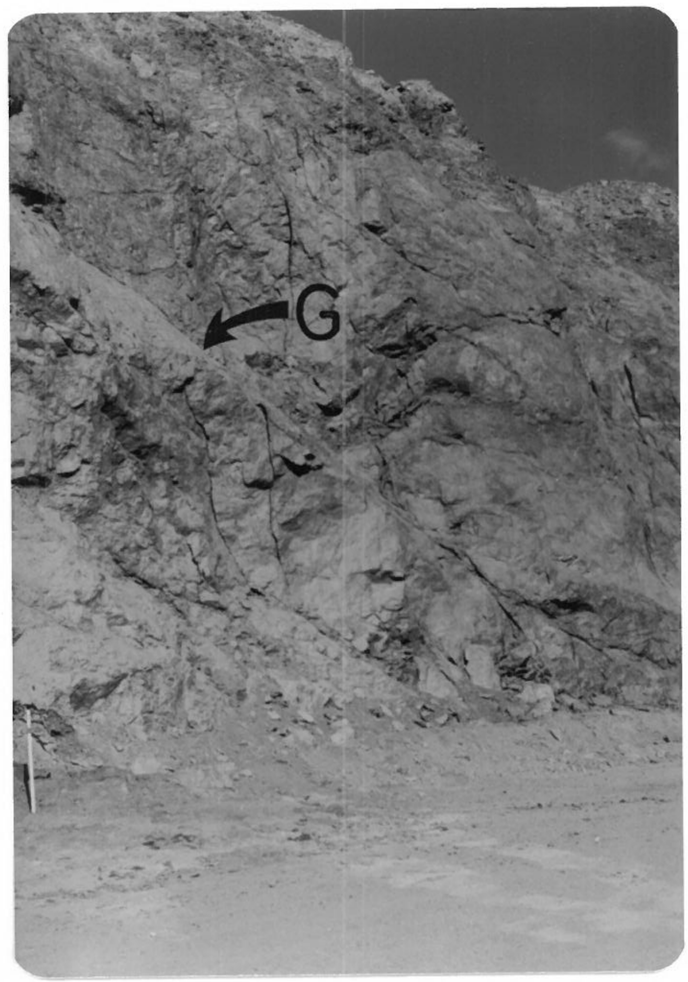

PHOTO 10: View of joint of Joint Set G in green argillite schist on bench 365 on the south wall. Note potential plane failure on this joint.

PHOTO 11: View of tungsten bearing quartz veins on the west wall of the pit.

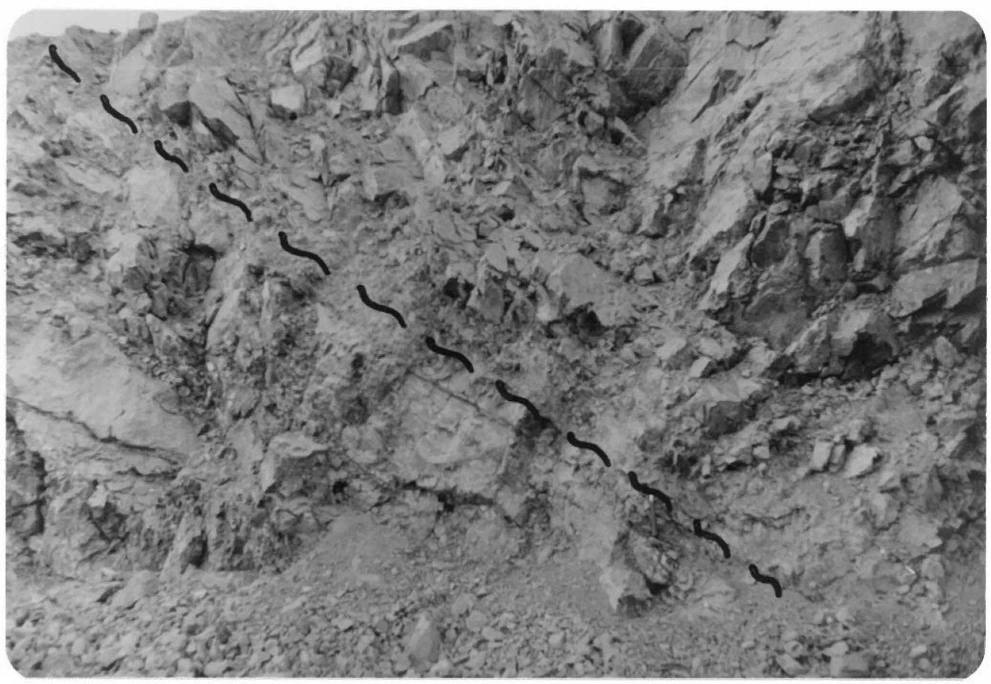

PHOTO 12: Flat lying shear of Shear Set SR1 on bench 385 on the north wall.

Quartz contained in veins is hard, brittle and often sheared and broken. The wall rock contact of veins is also often sheared and the quartz breaks away cleanly from this contact. These contacts are rough and irregular and may be healed or open.

5.4 FAULTS

Two major faults and a number of smaller faults and shears have been recognized in the pit. Their distribution throughout the pit is shown in Fig. 7.

5.4.1 South Wall Fault

The South Wall Fault is the most continuous and significant structural feature in the pit. It cuts across the south wall of the open pit, dips towards the north at 70^{o} and forms the southern limit of the ore zone (see Fig. 7). The structure contour plan of the fault given in Fig. 8 indicates the fault is planar and has a uniform dip and strike. Typical exposures of the South Wall Fault are shown in Photo 2 and 3.

The South Wall Fault consists of an approximately one metre wide zone of sandy clay silt which has been sheared and brecciated. Rocks within one or two metres of the fault are often sheared and brecciated.

Orientation of foliation in rocks adjacent to the fault is often significantly different than in other areas of the pit. Additional joint sets have been noted in the vicinity of the fault , and joint sets are often more diffuse in this area. As stated in Section 4, rocks south of the fault are only weakly silicified, resulting in increased susceptibility to weathering.

5.4.2 Iron Duke Fault

The Iron Duke Fault occurs in the northeast corner of the pit as shown in Fig. 7. It dips steeply to the north/northeast at about 85°. The fault zone is less than 0.5 metres wide and consists of oxidized clay gouge. To the northeast of the fault there is a noteable decrease in quartz veining and silicification, which results in an increased susceptibility to weathering. This increased susceptibility to weathering is not as pronounced as that south of the South Wall Fault, because the degree of silicification does not decrease as rapidly across the Iron Duke Fault.

5.4.3 Fault and Shear Sets

The distribution of all fault and shears mapped is shown in Fig. 7, and their orientations are summarized in Table III. Due to the relatively few observations available, Structural Domains 4, 5 and 6 were grouped together to form a more statistically meaningful population of faults and shears in the various areas of the open pit.

The results show two main fault and shear sets exist in Structural Domains 4 and 5. Both fault sets have dip directions of 035 and 045. Fault Set SR1 has an average dip of 35° to the northeast (see Photo 12). Fault Set SR2 has an average dip of 72° to the northeast. No consistent set of faults or shears are evident in Structural Domain 6. The South Wall Fault dominates Structural Domains 1, 2 and 3.

Faults are characterized by rough walled gouge zones filled with clay rich gouge and breccia (see Photo 12). The gouge zones are rarely wider than 10cm and are usually highly oxidized. Trace lengths are usually between 10 and 20 metres, but several 100 metre long faults are visible in the east wall.

6. STRENGTH PROPERTIES AND DOCUMENTATION OF SLOPE FAILURES

6.1 STRENGTH OF INTACT SAMPLES OF FRESH AND WEATHERED ROCK

6.1.1 Distribution of Rock Hardness in the Open Pit

The intact strengths of samples of fresh and weathered rock in the open pit have been determined primarily from simple field assessments, which consist of a set of simple mechanical tests based on the physical properties of the rock. Using this method, the hardness or the resistance of hand specimens to breaking or cutting can be related to the unconfined compressive strength using a simple classification system as summarized in the legend in Fig. 9. The distribution of rock hardness in the open pit, based on the detailed line mapping, is given in Fig. 9.

Fig. 9 indicates that most hornfels and related silicified rock in the open pit are uniformly hard, having a hardness greater than R4 (i.e. unconfined compressive strength greater than 8,000 psi (56 MPa). These rocks are extremely hard due to silicification. Additional rock mechanics testing of hornfels is not warranted at this time.

Schists, greywackes and other less silicified rocks south of the South Wall Fault have variable hardnesses, depending on the degree of weathering and silicification of the rock (see Figs. 1 and 2). Unweathered rocks in this area have hardnesses of R2 to R4 (i.e. unconfined compressive strength of 1000 psi to 16,000 psi). Weathered rocks have hardnesses of R0 to R3 (i.e. unconfined compressive strength of 100 psi to 8000 psi). Residual soils have hardnesses of S4 to R0 .

Earlier discussion in Section 4.5 indicate that there is a well defined variation of rock hardness with depth in the weathered rocks in this area (see Fig. 9 and the sections in Appendix A). These results are corroborated by examination of rock cores and percussion drill hole records.

6.1.2 Laboratory Tests

Detailed assessment of strength of weathered rocks and weaker schists in the open pit were estimated, based on field assessments and laboratory testing of selected core samples and block samples at James Cook University (as summarized in Table IV). These results are consistent with the distribution of hardness described above, and indicate that extremely weathered rock near surface are very weak and can be expected to behave as a soil. Hardness and strength increases with depth as the rock becomes less weathered.

6.1.3 Interrelationship of Hardness, Weathering and Unconfined Compressive Strength

Based on the distribution of hardness indicated in Fig. 9 and on the geotechnical section and the laboratory test results summarized in Table IV, the distribution of rock strength and degree of weathering has been defined for rocks south of the South Wall Fault. The distribution of rock strength and weathering with depth within these rocks is summarized in Table I.

North of the South Wall Fault the silicified hornfels and related rocks have a uniform hardness of R4 to R5 in all weathering zones, except the extremely weathered zone which occurs within a few metres of ground surface. In this area the upper 10m could be expected to be somewhat more weathered and broken than the rest of the rock mass.

6.2 SHEAR STRENGTH TESTING OF DISCONTINUITIES

Direct shear tests were carried out on selected core samples of four representative joints from the south wall area. Test results, summarized in Fig. 10, indicate that these joints have friction angles greater than 36^{o} and a relatively high cohesion. Cohesion on all joints tested was about 28 to 42 psi (193 to 290 kPa) as shown in Fig. 10. Subsequent back analyses of a plane failure along a shallow dipping discontinuity on the south wall indicate that a cohesion of 20 kPa to 51 kPa would be required along the joint plane for an acceptable factor of safety (see Section 6.4).

Based on the test results, and back analyses described in Section 6.4 it is felt justified to use a cohesion of at least 10 kPa for moderately rough discontinuities on the south wall. The typically planar discontinuities which occur in hornfels and related rocks are assumed to have a friction angle of 36^{o} and little or no cohesion.

Actual effective cohesion may be substantially decreased or increased, depending on the effects of blasting on the rock mass. Clearly, controlled blasting would help to preserve cohesion of the rock mass, and thus improve the stability of the slope.

6.3 DOCUMENTATION AND BACK ANALYSIS OF FAILURES IN WEATHERED ROCKS

6.3.1 Failures in Weathered Rock on the South Wall

Several failures were mapped in weathered rocks and soil on the south wall of the open pit. These failures appear to be typical soil type failures, with characteristic rotational failure surfaces (see Photo 2). Back analyses were carried out for the large failure below the primary crusher area, as this failure is the highest in the slope (being approximately nine metres high).

Back analyses were carried out using standard charts to determine likely shear strength properties for various possible groundwater conditions which may have existed prior to failure. If it is assumed that the friction angle, of the material is 24^{o}, the analyses indicates the cohesion (C') required for a factor of safety (F) of 1.0 varies from about 39 kPa to 45 kPa, depending on the groundwater conditions in the slope at the time of failure. These analyses indicate an average mass cohesion C' of about 40 kPa should be used for slope stability assessments in this area.

Analyses were carried out for a cohesion C' of 40 kPa and a friction angle Ø' of 24^{o} for 10m, 15m and 20m high slopes (bench faces). The results are summarized in Table V. These results indicate that slopes should be well drained to maintain reasonably steep slope angles.

6.3.2 Documentation of Highway Slopes in the Mt. Carbine Area

In order to assess shear strength properties of residual soils and weathered rock, slopes on the road between Mount Molloy and Port Douglas were documented. This road crosses through mountainous terrain and there are numerous cut slopes up to 10m high in weathered rock and residual soil. These slopes contain weathered schists and greywackes, which are similar to the rocks observed behind the South Wall Fault at Mt. Carbine. Also, the distribution of rocks of various strengths and degree of weathering is similar to that at Mount Carbine. It is noteworthy that these slopes appear to have been successfully excavated by ripping and extensive surface drainage ditches have been provided on these slopes (see Photo 14).

A plot of slope height vs slope angle for documented cut slopes showing stable, failed and partially failed slopes is given in Fig. 11. These results may be used to delineate the maximum safe slope angles for various slope heights, depending on rock strength as shown in Fig. 11. Because weathering decreases with depth and rock strength increases with depth, the relationship between slope height and slope angle is not consistent. Hence, an approximate line dividing stable and unstable slopes has been defined for extremely weathered to totally weathered rock (i.e. Weathering Category A and B) and highly weathered rock (i.e. Weathering Category C).

The results in Fig. 11 indicate that slope angles of 50^o would be acceptable for highly weathered rock and residual soil up to 10m high. Slope angles of 57^o would be acceptable for 10m high slopes in moderately weathered rock. Slopes up to 20m high would consist of residual soil in the

PHOTO 13: View of plane failure on joint of Joint Set G in green argillaceous schist on Bench 365 on the south wall.

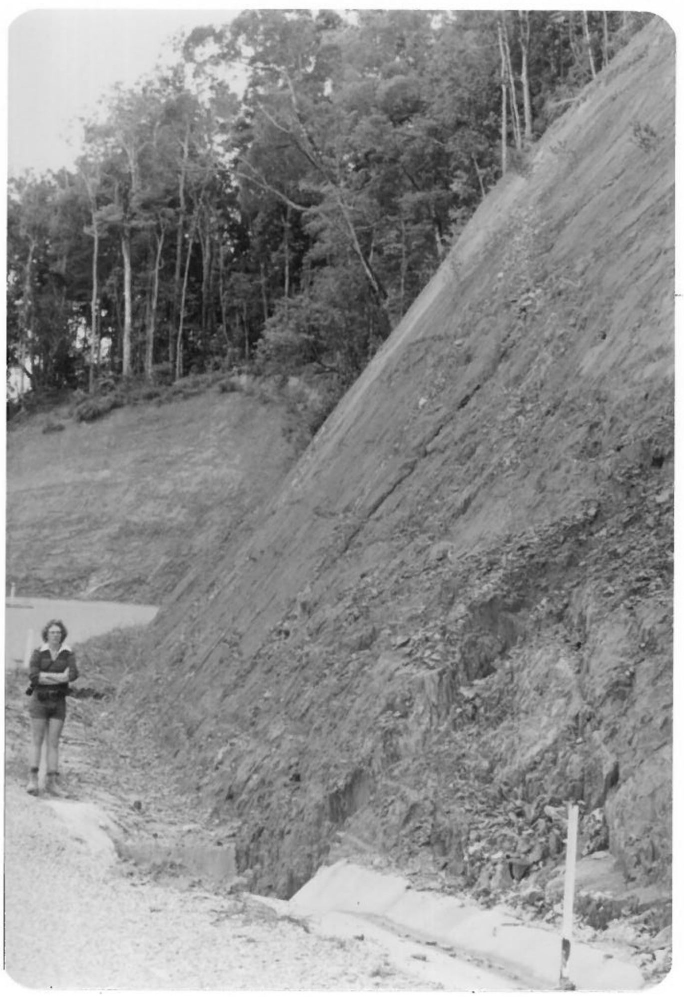

PHOTO 14: View of slopes in residual soil and weathered rock in the Rex Range near Mt. Carbine. These slopes were excavated by ripping and extensive surface drainage control measures have been installed.

upper portions and moderately weathered rock in the lower portions. Overall slope angles for slopes 20m high are indicated to be about 46^{o} (as shown in Fig. 11).

Although somewhat erratic and sparse, these results are consistent with the back analyses carried out using design charts (as described in Section 6.3.1). Hence, these results provide a reasonable estimte of shear strength of the weathered rock and residual soil for slope design in these materials.

6.4 BACK ANALYSES OF FAILURES ALONG DISCONTINUITIES

Numerous small wedges and plane failures, involving one bench in height, have been noted on the pit walls. Most of these failures are steep, and hence they have failed during or shortly after excavation. These failures provide little information concerning the shear strength of discontinuities, because their factor of safety is expected to be much less than 1.0.

Back analyses were carried out for one large plane failure on the 375m elevation near 22,925E (see Photo 13). This failure occurs primarily along a joint of Joint Set G, which has an average orientation of 330/47 and is typical of a possible failure which controls stability on the south wall.

Back analyses were carried out assuming a 10m high bench, a bench face angle of 80^{o}, a failure plane with a dip of 47^{o} and a friction angle of 36^{o}. Back analyses indicate that the cohesion required for a factor of safety of 1.0 to 1.5 varies from 20 kPa to 51 kPa for a slope with a tension crack (see Table VI). Additional analyses were carried out to determine the possible artificial support requirements, assuming that cohesion of 10 kPa could be mobilized along the joint.

Analyses results summarized in Table VI indicate the total anchor forces and number of anchors required to attain a factor of safety of 1.5 for this failure for a 10m high single bench. Results in Table VI indicate that the required anchor force decreases as the bench face angle is decreased. These results indicate that it is feasible to use grouted dowels or anchors to prevent small failures in the slope, provided dowels capable of maintaining sufficient loads can be installed in the slope. Results of trial dowel installations and pull out tests indicate that this is feasible.

6.5 DOCUMENTATION AND ASSESSMENT OF TRIAL INSTALLATIONS FOR POSSIBLE ARTIFICIAL SUPPORT

6.5.1 Background

In order to assess the feasibility of designing artificial support in the south wall, a series of trial dowel installations and pull out tests was recommended. The object of these tests was to install and test fully grouted dowels in representative weathered and unweathered rock types on the south wall. Details of dowel locations, installation details and test results are given in Table VII.

6.5.2 Installation and Test Results

Recommended locations and details of trial installations were summarized in a telex to the mine on October 18, 1981. Locations and test results for trial installations were transmitted by telex from the mine on February 8, 1982. These results are summarized in Table VII.

Cables tested had a nominal ultimate strength of 75 metric tonnes. Of ten dowels tested, nine tests were successful. The tenth test experienced jack failure and the jack could not be repaired on site. For all successful tests the cable strands failed at loads of 64 to 78 Tonnes. In no case did the anchor pull out for the test loads applied.

The average failure load was about 72 tonnes which indicate that a maximum design working load of 50 Tonnes (500 KN) would be reasonable and could be used for design of support systems for the dowel length specified.

7. HYDROGEOLOGY

Detailed assessment of hydrogeological conditions in the open pit is beyond the scope of this report. However some assessments of groundwater conditions in the open pit are justified, based on the obvious contribution of water to failures in weathered rock near the pit crest.

7.1 HYDROGEOLOGICAL CONDITIONS ON THE SOUTH WALL

Location of piezometer installations is shown on plan in Fig. 1 and on sections in Appendix A. A summary of all installations, maximum and minimum piezometric levels and hydraulic conductivities, based on falling head test results, is given in Table VIII.

Permeability test results indicate that rocks on the south wall have hydraulic conductivities of 3.8×10^{-7} to 2.7×10^{-10} m/sec. Hydraulic conductivities measured in percussion drillholes are generally about two orders of magnitude greater than those measured in diamond drillholes. Piezometric levels in individual piezometers and between adjacent piezometers indicate that groundwater on the south wall is related to local infiltration and flow in the immediate vicinity of the south wall. The South Wall Fault appears to form a barrier to groundwater flow, in that the water levels in some piezometers south of the fault are considerably higher than water levels north of the fault.

Regular weekly monitoring of piezometers indicates that there are reasonably large and rapid fluctuations in piezometric levels in response to precipitation. The moderate permeabilities and rapid fluctuation of piezometers indicates that the bulk of the groundwater in the south wall may be related to infiltration of surface water. It is likely that control of surface water, using sealed ditches and inclined benches, would limit the amount of groundwater in the slope. Because of the moderately permeable nature of the rocks, it is likely that the adverse effects of groundwater in the slope could be reduced by simple depressurization techniques, using subhorizontal upward inclined drainholes in critical areas.

7.2 HYDROGEOLOGICAL CONDITIONS ON THE NORTH, WEST AND EAST WALLS

Little is known concerning hydrogeological conditions in areas of the pit other than the south wall, because no installations were placed in these areas. Presence of water in the sump in the pit indicates some groundwater flow could occur towards the open pit as it is deepened. There will also be a significant contribution to water flow in the pit as a result of precipitation within the pit limits.

To date no significant effects of groundwater have been noted on the north, east or west walls.

7.3 GROUNDWATER CONTROL AND DEPRESSURIZATION

The importance of groundwater control depends on the volume of water, rate of flow through the rock mass and water pressure in the slope. The necessity for control of groundwater is expected to vary, depending on the sensitivity of the particular area of the slope to variation in groundwater pressure. Stability analyses discussed in Section 8 indicate that in many areas groundwater pressure is expected to have little effect on slope stability and slope design.

In areas where groundwater is expected to affect slope stability, groundwater control and depressurization measures, consisting of subhorizontal upward inclined drainholes, are recommended near bench toes. Drainholes are particularly important on the south wall to control water pressures in weak or weathered rocks south of the South Wall Fault. Drainholes are also recommended to control water pressures behind the South Wall Fault or other major faults. Drainholes may also be required in selected areas of the pit to control water pressures.

Location, length and spacing of drainholes is difficult to predict, based on the limited information available. However, optimum length and spacing of drainholes could be determined by installation of additional piezometers and monitoring the response when drainholes are installed near piezometers. In areas where drainholes are to be installed, it is suggested that piezometers should be installed and monitored prior to installing the drainholes. Drainholes should be installed initially at about 10m to 20m spacings. Spacing should be adjusted based on response of piezometers until an optimum spacing of drainholes has been achieved and the slope is depressurized.

Sealed piezometers should be installed on the south wall as the final pit is excavated. These piezometers should be monitored regularly to assess the effectiveness of drainholes, and to clearly determine the length and spacing of drainholes required to obtain effective depressurization of the slope.

7.4 SURFACE WATER CONTROL

Due to the unfavourable effects of groundwater on slope stability, particularly in the south wall of the open pit, it is recommended that a properly designed system of surface water control be constructed in the pit. Surface water control should consist of the following:

(i) A sealed drainage ditch behind the pit crest in all areas to intercept all water which could infiltrate into the rock mass.

(ii) The area around the entire pit crest should be properly graded to divert water into the drainage ditch or away from the pit.

(iii) Benches and haulroads on the south wall should be inclined and/or graded to divert water off the slope.

(iv) Drainage ditches or closed pipes are important to collect and control water from drainholes on the south wall.

(v) A sealed drainage ditch should be constructed on the haulroad to convey water off the slope and into sumps for pumping out of the open pit. Uncontrolled water flow should not be permitted in any area of the open pit.

(vi) Sealed drainage ditches should be constructed in areas of the pit other than the south wall only as required.

7.5 ADDITIONAL HYDROGEOLOGICAL INVESTIGATIONS

Other than specific areas as discussed above, serious groundwater problems have not occurred in the open pit. Hence, detailed investigation of hydrogeological conditions in the open pit is not considered warranted at this time.

However, it is recommended that sealed piezometers should be placed at strategic locations around the open pit to monitor groundwater pressures as the pit is developed. If monitoring indicates that adverse groundwater pressures are developing in certain areas, horizontal drainholes should be installed until the slope is sufficiently depressurized. Depending on the severity of groundwater problems encountered, additional study may be warranted at a later date.

8. SLOPE STABILITY ANALYSES AND SLOPE DESIGN

Slope stability analyses involved investigating all kinematically possible failure modes with respect to faults, joints and other discontinuities which could lead to shallow failure of individual benches and/or deep seated failure involving large sections of the interramp slopes between haulroads. Consideration was also given to the past performance of the slope with respect to evaluation of basic failure modes as well as general rock mass behaviour. Plan radius of curvature of the proposed final slopes was also considered in the design. Back analysis of failures in the pit was used to assist in evaluating the effective friction and cohesion of discontinuities and to complement strength testing results. Back analysis of failures which occurred in the weathered schist and residual soils in the upper portion of the south wall was used to prepare a slope design and remedial measures in this area.

8.1 BASIC SLOPE DESIGN CONSIDERATIONS

In rock slopes, instability occurs as a result of failure along structural discontinuities, such as foliation joints, geological contacts, faults, etc. Seldom is instability due to failure through intact material, unless the rock is very soft or weak such as in the weathered schists and greywackes near the crest of the south wall. Therefore, the most important single factor in the stability analysis and design of rock slopes is to evaluate and determine the orientation, geometry and spatial distribution of the discontinuities in the slope. Having done this, the next step is to evaluate these discontinuities with respect to the orientation and alternative possible angles of the proposed pit slope. It is following these basic principles that the slope stability analyses were carried out.

Slope control can be basically accomplished in two ways:

a) to design the slope so that no failures occur, or

b) to excavate the pit under controlled conditions and design the slope with adequate access so that failures can be caught on berms and removed if necessary.

The first solution is usually too conservative to be economically feasible. The second solution requires thorough consideration of slope geometry so that failures are contained on berms and safe access is available to the berms to allow removal of collected debris if required. This solution provides adequate safety at minimal cost, although special design or remedial measures may be required to insure the stability of haulroads or critical installations on benches.

The parameters which govern the geometry of a slope are shown in Fig. 12. These are bench height (H), berm width (ℓ), and bench face angle (β). These parameters are governed by the strength and nature of the slope, government mining regulations and equipment available. Bench height should be such as to provide a safe working slope as well as an optimum overall slope angle. It should be noted that without affecting the slope angle, higher benches will allow wider berms for better protection and more reliable and easier access, if this is desirable, although the size of possible failures may increase.

Berm width should be controlled to a certain extent by the amount of access required to the slope as well as the optimum width to accommodate failures. It must be accepted that, even with careful perimeter or controlled blasting techniques, a certain amount of breakback may occur. In general, slopes should have berms wide enough so that falling debris will be trapped and, equipment can have access to keep berms clean and effective as catchments if required.

By inclining the bench faces, blasting damage is reduced and high stresses are less likely to develop near the bench crests. Hence, tension cracks and overhangs are minimized. Avoiding these problems reduces the amount of rockfall and increases the safety of the slope.

8.2 DEEP SEATED FAILURE CONSIDERATIONS AND MAJOR FAILURES

Deep seated failure, involving several benches or even the whole slope, may involve major throughgoing faults or other major discontinuities. Deep seated failures may also develop through a complex interaction of minor and/or major structures to form a large, continuous, if somewhat irregular, failure surface(s). In some cases deep seated failure surfaces may in part develop by failure through intact rock bridges between discontinuities. Such failures are often precipitated by small slope readjustments and high stress concentrations at the toe of the slope, which tend to exceed the strength of the rock mass. This type of failure mechanism is particularly common in slopes where softer material at the toe of the slope, such as coal, potash, or schist bands, is squeezed out due to the load which is applied by the overlying strata.

8.2.1 Deep Seated Failure Involving Major Faults

As discussed in Section 5.4, major faults in the open pit are steep dipping. Hence, it is expected that even if undercut by mining instability would be confined to one or two benches at the most. In terms of the proposed design, some local design modifications and/or remedial measures may be required in areas of major faults. This is particularly the case for the South Wall Fault, wherein it is recommended that the benches be excavated to avoid undercutting this fault.

Major faults could result in significant damming of groundwater. Hence, drainholes may be required in the vicinity of major faults to control water pressures and stabilize slopes on a local basis.

8.2.2 Deep Seated Failures Involving Fault and Shear Sets

An equal area projection of planes representing the average orientation of faults and shears in the open pit is given in Fig. 13. The main orientations of fault sets and shear sets (summarized in Section 5.4) indicates that plane failures could occur along faults of either Set SR1 or Set SR2 in the south or southwest side of the pit where these faults dips to the northeast out of the slope. It is noteworthy that, based on geological mapping, these discontinuities are relatively poorly developed in Structural Domains 5D, 6A and 6B which occur on the south and southwest walls of the open pit.

Faults and shears of Set SR1 have an average dip of 35^o. Large potential areas of instability could occur along these shears, or along a combination of faults and shears of Set SR1 and SR2. Because the average dip of these faults is similar to the average friction angle of discontinuities described in Section 6.2, it is possible that failures could occur along these discontinuities, particularly, if high groundwater pressures develop in the slope. It is likely that shear strength along faults and shears would be less than that along joints. However, the surfaces of faults and shears of Set SR1 have been noted to be rough and irregular and may have some effective cohesion.

Because faults and shears of Set SR2 have an average dip of 72^o, it is expected that failures involving these discontinuities would only be developed over one or two benches. Such failures could form small areas of local instability, requiring remedial measures, if undercut by mining.

The possibility of occurrence of major faults of Sets SR1 or SR2 on the south and southwest walls of the final pit cannot be clearly stated at this time. Hence, it is not considered justified to flatten the slope in this area, based on the possible occurrence of a major fault. However, it is imperative that the slopes be carefully mapped and monitored. If a major failure develops, provision should be made for possible remedial measures such as artificial support, depressurization or possible slope flattening, if required.

8.2.3 Deep Seated Rotational Failures

For engineering purposes, the principal consideration in evaluating the compressive strength of the rock is to determine if there is any possibility of excessive deformation and ultimate rotational failure of the slope due to failure of the intact material, resulting from the weight of the overlying strata. In general, normal stresses up to 1000 psi (7 MPa) can be expected in deep, open pits. Thus, slopes which have part of the rocks with compressive strengths less than this could be subject to significant deformation and possibly failure if the crushing strengths are exceeded.

Based on the relatively high hardness and accordingly high compressive strength of most intact rocks in the pit, the likelihood of time dependent strain and subsequent failure of intact material in the slope appears to be small, the exception being the weathered or decomposed rocks on the upper benches of the south wall. Hence, the possibility of deep seated rotational failure, due to failure of intact material involving the whole slope, is low (at least within the range of overall slope angles which are geometrically feasible, and which are considered in this analysis).

Specific design and remedial measures to control possible rotational failures over one to two benches in the highly weathered rocks and residual soil near the crest of the south wall are discussed in Section 8.7.

8.3 STABILITY ANALYSES BASED ON GEOLOGIC STRUCTURAL ANALYSIS RESULTS

8.3.1 Description of the Design Sector Concept

Rational slope stability analysis and design requires prediction of the geologic structural conditions which occur on the final pit wall. Such a prediction includes determining the distribution and location of lithologic units, geological contacts, major faults and structural domain boundaries.

Not only must structural domains be considered for individual analysis, but also the overall orientation of the final pit wall must be considered as well. Different pit wall orientations require basically different design considerations. Hence, it is necessary to define zones which contain essentially one structural domain and one general pit slope orientation; these zones are designated design sectors.

If the orientation of the pit wall changes within a structural domain, depending on the number of different pit slope orientations, two or possibly more design sectors may be required within that structural domain. If, on the other hand, two structural domains occur where the orientation of the pit wall is constant, two design sectors will result.

All available geological mapping and core logging information was used to project the structural domain boundaries onto the proposed final pit wall. The boundaries of the structural domains and straight slope segments on the proposed final pit walls were used to determine the design sector boundaries, and thus delineate the design sectors, as shown in Fig. 14.

The proposed final pit was divided into 17 design sectors, designated Design Sectors 1-1, 6B-1, 6B-2 etc. In some cases, where small portions of the pit theoretically should comprise a separate design sector, these areas have been incorporated into larger design sectors. Information relating to pit wall orientation, structural areas, main rock types and related slope information for each design sector is given in Table IX.

8.3.2 Design Criteria Concerning Orientation of Geological Structure

Because there are significant variations in intensity, orientation and dip of each joint set and fault set in various areas of the pit, the effects of each joint set acting individually or in combination with other joint sets should be evaluated statistically. The statistical significance of one particular failure mode on stability may be low. However, the combined statistical significance of all possible failure modes, with respect to both the actual observed and predicted performance of the slope, should be evaluated to prepare a rational pit slope design.

Statistical analysis of joint sets is of key importance for bench design. In this regard, the volume of slope failure, which is considered to be statistically significant, is used to determine the bench design required to contain failed material on benches and provide a safe slope.

Within most rocks in the pit many joints are continuous over several benches. However, because of the generally steeply dipping nature of most structures in the pit, it is unlikely that bench failures, involving more than one or two benches, would develop along single sets of discontinuities or combinations of single sets of discontinuities. As discussed earlier, it is possible that large failures could develop involving occasional major discontinuities, which dip towards the pit and daylight in the wall. This is not seen to be a significant factor, however, in terms of the design criteria at Mt. Carbine. The location and extent of such shallowly dipping features cannot be predicted with any degree of confidence within practical limits on the final pit wall.

8.3.3 Determination of Kinematically Possible Modes of Failure

Equal area projections of planes representing the peak orientations of the various discontinuity sets (i.e. faults, shears, joints, veins and foliation in each design sector) were used to define the possible failure modes as shown in Fig. 14.

8.3.4 Mechanical Stability Analyses

Appropriate stability analyses, using a desktop computer, were carried out for each possible failure mode. Friction properties of the discontinuities used in the analyses were based on laboratory test results, as summarized in Section 6.

Stability analyses results were used to determine the kinematically possible failure mode(s) which controls slope stability. The failure mode(s) which has the shallowest average dip or plunge for a factor of safety of 1.2 and 2.0 were assumed to control stability for dry (or drained slopes) and saturated (underwatered slopes), respectively. In these cases the slope design would be based on the "worst case" assuming all possible failures would be controlled on each bench.

Because of numerous kinematically possible failures in the pit, stability analyses were also approached statistically, using simple limit equilibrium methods. The analysis technique assumed dry slopes to simplify the analysis. With regard to stability analyses, a factor of safety greater than or equal to 1.2 was considered adequate to insure stability of dry (i.e., drained or dewatered) slopes. A factor of safety greater than or equal to 2.0 (assuming the dry condition) was considered adequate for stability of slopes subject to adverse groundwater conditions (i.e. undrained or fully saturated slopes).

8.4 STATISTICAL ANALYSES OF POSSIBLE WEDGE FAILURES

In most areas, slope stability appears to be controlled by plane or wedge failures on benches. These failures are generally formed by a combination of joints, veins and/or faults. Thus, in terms of design of interramp slopes between haulroads, optimum bench geometry will control the overall slope design.

8.4.1 Determination of Possible Critical Failure Modes

The number and geometry of possible failure modes in each design sector is determined by evaluating the various combinations of discontinuity sets with respect to the proposed orientation of the final pit wall. Discontinuities are evaluated based on the peak or average orientations of each joint set. Kinematic plots of discontinuity sets in each design sector are presented in Fig. 14.

Because the analysis is developed for bench design, all kinematically possible failure modes which dip or plunge out of a slope face with an angle of up to 90^o and a factor of safety of less than 2.0, are considered in the analysis. These possible failures are designated "critical failure modes" or "critical wedges", in that they could lead to failure or breakback of individual bench crests.

In most design sectors there are a large number of possible critical failure modes, since several joint sets and fault sets occur in each design sector. Also, because vertical (90^o) bench face angles are considered for analysis purposes, a large number of critical failure modes has resulted. In such cases, statistical analysis was used to considerable advantage.

In some design sectors there are few joint sets, and consequently only a few critical failure modes. In these areas the dip or apparent plunge of the least favourable failure mode with a factor of safety less than 1.2 (dry slope) and 2.0 (undrained slope) is chosen for design. In these cases, the slope geometry is optimized to control possible failures on benches.

8.4.2 Statistical Analysis of Possible Critical Wedges

i) Orientation of Sets of Discontinuity

Sets of discontinuities which form critical failure modes are defined by the peak or average orientation of the discontinuity sets. Peak orientations of different discontinuity sets and subsets give an appreciation of the range of possible orientations of the discontinuity sets in the particular design sector.

ii) Determination of Number and Orientation of Possible Critical Failure Modes

Based on the critical failure modes identified, the total number of possible wedges was determined for all possible combinations of the peak orientation of each joint set. The number of possible wedges was determined by adding the number of discontinuities which correspond to the orientation of each discontinuity set involved in a particular possible wedge combination. Equal area projections of the joint sets were used to determine the average plunge of each possible wedge failure, as well as the apparent plunge relative to the proposed final slope. The apparent plunge was recorded for purposes of determining the actual size or volume of possible failures. All possible combinations were tested for type of failure, and a factor of safety was calculated using a standard wedge analysis method.

iii) Development of Cumulative Frequency Plots

Equal area projections and the stability analysis results summarize the orientation and apparent plunge of the possible critical failure modes examined. Unique statistical plots are used to assess the distribution and relative number of possible failure combinations. The cumulative frequency of possible failures are plotted aginst apparent plunge of the possible wedges in terms of those wedges which have a factor of safety less than 2.0 and less than 1.2, respectively. Cumulative frequency plots for all design sectors assessed by this method are given in Appendix D.

Cumulative frequency plots have been developed for wedges with a factor of safety less than 1.2 and 2.0, because it is considered that these values are required to prevent failure of a dry (or dewatered) slope or undewatered slope, respectively. In many cases, because of the steeply dipping nature of most discontinuity sets, there was little or no difference between the cumulative frequency plots for factor of safety less than 1.2 and 2.0.

iv) Interpretation of the Cumulative Frequency Plots

The statistical cumulative frequency plots described above were assessed by evaluating the apparent plunge of failures in terms of the cumulative percentages and factors of safety of the respective possible wedge failures in each design sector. The appropriate cumulative frequency plots in Appendix D, may be used to assess the significance of the cumulative frequency distribution for each design sector and the difference in apparent plunge for each factor of safety.

Most of the cumulative frequency plots have a somewhat characteristic, though crude, S shape, as shown in Appendix D. For any given design sector there is usually a point on the curves beyond which, for an increase in apparent plunge of wedges, there is a correspondingly sharp increase in the cumulative frequency of possible wedge failures. Cumulative percentages and corresponding apparent plunges for the break in curvature of the cumulative frequency distribution in each design sector are also presented in Tables IX.

8.4.3 Analysis of Wedge Failures When There Are Insufficient Wedges to Carry Out Statistical Analyses

In cases where there are not enough critical failure modes to carry out a statistical analysis, slope design is determined based on the least favourable failure mode with a factor of safety less than 2.0 and 1.2. The least favourable failure mode corresponds to a worst case. This failure mode is used to determine the berm widths, bench heights and bench face angles required to catch failed debris on individual benches. Design charts based on slope geometry and failure geometry are developed for these cases.

8.4.4 Effects of Plane Failures on Statistical Analysis Results

In cases where plane failure on shallow dipping joints dominates slope design, statistical analyses techniques are not valid. In these cases the slope design would be controlled by the "worst case" as described in Section 8.3.4.

At Mt. Carbine plane failures could occur primarily along joints, faults, shears or veins which dip towards the pit. In all design sectors where the dip of a possible failure plane is steeper than the angle of friction along these surfaces, planar sliding is assumed to be kinematically possible, and the slope should not be excavated at a steeper angle than these planes. This would be the case unless specifically required remedial measures such as artificial support or depressurization or both, are used. In general, the slope should be designed so that failures along single surfaces are caught on berms. Dips of failure surfaces for design sectors where plane failure controls slope design are summarized in the appropriate columns in Table IX. In several design sectors, shallow dipping planar failures are indicated, which involve poorly developed joint sets or fault sets. In these cases, it is not considered realistic to flatten the slope based on the questionable occurrence of these features. However, it is imperative that the slopes be carefully mapped and monitored. If failures are encountered, provisions should be made for possible remedial measures if required.

8.5 BENCH BREAKBACK ANALYSIS

An analysis of bench breakback was carried out to compare the current geometry in the pit with the failure geometry indicated by the stability analyses of possible failure modes. Bench breakback angle was obtained by calculating the existing bench breakback for 10m high benches from profiles taken normal to the pit walls in various areas of the current pit (October, 1981). Measurements taken for slopes of similar orientation in similar areas of the open pit were grouped together, and the average breakback was calculated (as summarized in Table X).

Bench breakback occurs in virtually all sections of the pit, due primarily to bench failures on steep dipping discontinuities, as well as the adverse effects of blasting. Typical breakback in the pit is shown in Photo 15. The analysis indicates that generally the average bench breakback is as steep or steeper than the geologic structures which control the potential failure modes. This is probably due to the favourable effects of asperities, intact rock bridges, cohesion, etc., which tend to add sufficient additional resisting forces to prevent complete failure of the potential sliding blocks. The presence of cohesion in the slope has already been indicated by the back analysis results for bench failures discussed previously.

The average breakback in various areas of the open pit (as summarized in Table X) ranges from 2.45m to 4.88m. It is noteworthy that in areas where controlled blasting has been used on the east wall, breakback is about 2.45m. Examination of the breakback in the pit indicates that over half the breakback on benches occurs near the bench crests and results primarily from blasting damage (see Photo 15). On the east wall, where controlled blasting has been used, bench breakback appears to be approximately half that compared to where controlled blasting was not used.

PHOTO 15: View of breakback of bench crests due to natural failures and blast damage on the west wall of the open pit. Note that over one half the breakback occurs near the bench crests.

Based on the analyses summarized in Table X, it is reasonable to assume that, if careful controlled blasting is used on the final wall, the average break back at the crest of vertical benches would be approximately 2.5m for every ten metres of bench height. This would result in breakback angles of 76^{o}. Therefore, 10, 20 and 30 metre high benches would be expected to have bench breakbacks of 2.5, 5.0 and 7.5m, respectively (provided that careful controlled blasting were used). It appears that if controlled blasting were not used, breakback would be significantly greater.

If bench faces are inclined, the breakback at bench crests would be accordingly reduced. Hence, bench faces at 76^{o} or less would be expected to have minimal breakback, in that the slopes would be excavated shallower than the anticipated breakback angle. Benches excavated at 80^{o} would be expected to have breakback of about 0.74m, 1.47m and 2.21m for 10, 20 and 30m bench heights, respectively.

8.6 DESIGN OF INTERRAMP SLOPES BASED ON STATISTICAL ANALYSES RESULTS

Statistical analysis of kinematically possible failure mechanisms was applied to design of benches and interramp slopes in all areas of the open pit. However, the stability of slopes in the weathered schists and greywackes south of the South Wall Fault is controlled largely by rotational failures, as well as failures along shallow dipping discontinuities. Because of the critical nature of this wall, and constraints on mining geometry as a result of the primary crusher and stock piles near the pit crest, the statistical analyses were not suitable by themselves for slope design in the area south of the South Wall Fault. Hence, other types of analysis had to be considered in this area. Detailed slope design and remedial measures for the area south of the South Wall Fault are discussed in Section 8.7.

8.6.1 Presentation and Evaluation of Analysis Results for Slope Design

The results of stability analyses for the various types of kinematically possible failure mechanisms are presented in Table IX. The average dip of planar failures or plunge of wedge failures for the worst case, or for design sectors where statistical analyses were not carried out, are recorded in the appropriate columnns. The results of the statistical analysis of wedges are also given for a cumulative frequency of 20% of possible unstable wedges, and for the break in the cumulative frequency plot for factors of safety less than 2.0 and 1.2, respectively.

In general, it is considered reasonable to design slopes to accommodate the bulk of the bench failures. Therefore, if the slope design is based on a cumulative percentages of 20%, this could result in 20% of the possible failure modes not being caught on berms or controlled on individual benches. Hence, it is possible that a failure, which is not controlled on a bench, would spill over and be caught on a lower bench. Consequently, safe reliable access on strategic or critical benches for clean up is desirable.

Based largely on engineering experience at other open pit mines, it is considered reasonable for preliminary slope design purposes to assume that a cumulative frequency of 20% of the possible wedge failures would not be retained on a single bench. In many cases, however, the break in the curve appears to be more realistic for design. Therefore, for purposes of the statistical analysis, the lesser of these two values is used to determine the apparent plunge angle of the possible wedge failures for design.

Based on the above discussion, the least favourable analysis results for each design sector are chosen for slope design. These values are circled in Table IX.

8.6.2 Determination of Slope Angle and Slope Geometry Based on Failure Geometry

The failure modes or apparent plunge of wedges, which are considered to control slope design in each design sector, are used to determine the possible failure geometry and slope design required to control these failures. Design charts have been developed for slope design, based on slope geometry and possible failure geometry, for 20m and 30m high benches, as shown in Figs. 15 and 16.

Bench heights of 20m to 30m are considered feasible in hornfels and related rock north of the South Wall Fault, due to the high compressive strength of the rock and steep dip of most joints. The analyses presented in this report are carried out for both 20m and 30m high benches to determine the optimum bench height to be used for each design sector.

The design charts may be used to determine the size of potential failures, the minimum berm width and the intermediate slope angle between haulroads required to control possible bench failures defined by structural discontinuities. These calculations are based on the dip or plunge of possible failures, β_W, which is indicated from the statistical analysis discussed above and circled in Table IX.

The volume of material in a bench failure is proportional to either the dip of a failure surface or plunge of wedge failure, β_W, bench face angle and bench height. The shallower the dip or plunge of a failure, the greater the volume of material which could fail onto the bench. Calculation of the cross-sectional area (A) of a possible bench failure for a particular bench height gives an estimate of the volume of failed material per unit width of berm (see Figs. 15(a) and 16(a)).

Cross-sectional area of a possible bench failure is used to calculate the minimum berm width (ℓ_{req}) which is required to catch all possible failed material (see Figs. 15(b) and 16 (b)). In this calculation, it is assumed that the failed material completely breaks down and comes to rest at the natural angle of repose (r) of the failed material, which is assumed to be 38^o. Minimum berm widths may be used to calculate the interramp slope angle between haulroads (θ), for the particular bench height and bench face angle which is assumed in the analysis (see Figs. 15(c) and 16(c)). A bulking factor due to an increase in volume of material during failure is not considered in the calculation of minimum berm width (ℓ_{req}). Results of the analyses with regard to the bulking factor are considered to be conservative.

It is recommended that the total berm width be calculated by adding the berm width required to catch failed material to the anticipated average breakback (ℓ_{bb}) at the bench crests. As discussed in Section 8.5, the average breakback is expected to be 5.0m and 7.5m for 20m and 30m high vertical benches, provided controlled blasting is used on final walls. If bench faces are inclined, the amount of breakback would be reduced accordingly. Total berm width recorded in Table IX is the sum of the calculated berm width, i.e.

$$\ell_{total} = \ell_{req} + \ell_{bb}$$

The total berm width is used to determine the interramp slope angle using the design charts in Figs. 15(c) and 16(c). These angles are assessed and a slope design prepared accordingly.

8.6.3 Determination of Optimum Bench Design

Analyses were carried out using the design charts described above for both 20m and 30m high benches to determine the optimum interramp slope angle. The optimum interramp slope angle is determined from the possible failure mode which is expected to control stability of slopes, using the design charts as described in the following and summarized for 20m and 30m high benches in Table IX.

(i) Selection of Optimum Bench Face Angle

For 20m or 30m high benches it is considered practical to excavate bench faces at 70^{o} to 80^{o} in order to provide better control on the final face and reduce high stresses near crests of individual benches.

The optimum bench face angle was selected based on the dip or apparent plunge of the failure mode. For 20m high benches the optimum bench face angle is 70° for failure modes which are less than 54°. The optimum bench face angle is 80° for failure modes which are steeper than 54°. For 30m high benches the optimum bench face angle is 70° for failure modes which are less than 67°, but the optimum bench face angle is 80° for failure modes which are steeper than 67°.

(ii) Calculation of Berm Width

The total berm width required to retain failed debris on the berm, with due consideration to the average expected breakback, was calculated for the optimum bench face angle.

For purposes of preliminary slope design, minimum allowable berm widths not including average breakback are assumed to be 9m and 10m for 20m and 30m high benches, respectively. These minimum berm widths are recommended to provide adequate safety, catchment, and access to the slope. Minimum berm widths could possibly be reduced for final slope designs, depending on the performance of interim slopes.

(iii) Calculation of Optimum Interramp Slope Angle

The optimum interramp slope angle was determined from the design charts based on the optimum bench face angle and total berm width. Calculated interramp slope angles for 20m and 30m high benches are summarized in Table IX and Fig. 14. Alternative slope angles were compared and the optimum bench height and slope design was selected.

8.6.4 Modification of Design Based on Effects of Plan Radius of Curvature at North and South Ends of Pit

It is considered justified that the relatively tightly curved slopes in the east and west ends of the open pit may be excavated at angles somewhat steeper than those indicated from the stability analyses, due to the effects of lateral constraint or arching caused by the relatively tight plan radius of curvature. Current theories of slope stability deal with the slope as a two-dimensional problem, i.e. the analysis is carried out on the basis of plane strain for a slice of unit thickness cut from within an infinitely long straight slope. As the radius of curvature decreases, horizontal tangential stresses increase and "arching" occurs which improves the stability of the slope. In effect, the tight curvature of a slope provides lateral constraint which tends to increase the cohesive forces and the competency of the rock mass as a whole.

Based on compilation by Coates and Sage (1974) of empirical data from field studies and mathematical modelling of radius of curvature effects in several open pits, a simple formula to recognize these effects in design has been developed. A correction increment (i) is proposed which represents the allowable increase in slope angle due to the effects of plan radius of curvature (R) and slope height (H) where

$$i = \frac{3}{2\frac{R}{H} - 1} \quad \text{for R/H greater than 0.5}$$

The plan radius of curvature and allowable increase in slope angle (i) are shown for the east and west wall in Fig. 14 and summarized in Table IX. These radius of curvature effects mainly apply to the lower reaches of the east and west wall. All other areas in the pit have a large radius of curvature and there is no significant effect of lateral constraint.

8.6.5 Recommended Slope Design

(i) Interramp Slope Design Between Haulroads in Design Sectors

The final recommended design for interramp slope for both dry, (i.e. drained or dewatered) conditions are summarized in the columns on the extreme right side in Table IX. These slope designs have been determined by adjusting the optimum interramp slope angle based on consideration of effects of plan radius of curvature and requirements for access and safety particularly on the south wall. These slope designs are also summarized on the proposed final pit plan in Fig. 17.

Based on the design criteria and procedures described above, the optimum interramp slope angle may be selected for the case of either 20m or 30m high benches (as summarized in Table IX). In general, 30m high benches result in interramp slopes as steep or steeper than those for 20m high benches, mainly because the required minimum berm width for 30m high benches is similar to that for 20m high benches. Hence, steeper slopes would generally be achieved if 30m high benches are used.

In certain areas of the open pit, particularly on the south wall, the proposed slope design and remedial measures require the use of 10m and 20m high benches (as described in Section 8.7). In nearly all areas it is important to maintain access to some benches. Access is particularly important on the south wall for maintenance of drainage ditches, remedial measures, monitoring etc.

Examination of the proposed pit plan indicates that, even though 30m high benches would be feasible in most areas north of the South Wall Fault, 20m high benches will be required in all areas between elevation 305 and 345m to insure access to benches on the south wall. Bench heights of 30m are feasible for interramp slopes above 345m and below 305m. Below 305m, the angle of the overall slope is considerably flattened as a result of haulroads. Hence, increased interramp slope angles and bench heights are considered justified. Above 345m, 30m high benches could be blended relatively easily to 10m high benches on the south wall with some loss of access. These bench heights are considered justified, based on the specific ground conditions at Mount Carbine. Distribution of the areas where 20m and 30m benches recommended are shown in Fig. 17.

(ii) Blending and Modification of Recommended Slope Designs for Pit Planning Purposes

Slope designs recommended are based on analysis of information within design sectors, but without due consideration of pit planning requirements and designs in adjacent design sectors. Design of the final pit slopes must include zones of transition between the slope designs of adjacent design sectors. In general, the recommended slope angles for a design sector should not be exceeded in transition zones. However, because of the additional berm width provided in the slope design to account for bench breakback, it is considered justified to steepen interramp slopes by up to three degrees over small sections of the slope and accept the possibility of increased failures spilling over benches. Therefore, if transition zones are not overly wide, it may be justified to increase slope angles up to three degrees near the boundaries of design sectors in order to enable blending of the slope designs between adjacent design sectors.

Results of blending slopes and related modification to slope design based on the probable mining geometry constraints are summarized in Fig. 18. Fig. 18 shows the recommended general slope designs, bench heights maximum and interramp slope angles in all areas of the pit except the areas south of the South Wall Fault (where specialized design and remedial measures have been prepared (see Section 8.7)).

(iii) Production Blasting and Final Wall Blasting

If 20m or 30m high benches and steep interramp slopes are used, it is imperative that breakback of the final wall should be minimized with careful perimeter blasting, using small diameter and lightly loaded holes. Blast design should be re-evaluated and trial blasting should be carried out to assess possible control of breakback on 20m and 30m high benches. Reduced breakback would indicate feasibility of steeper bench face angles, and hence steeper interramp slope angles.

(iv) Groundwater Control and Drainage

Little is known concerning groundwater conditions in the open pit. Horizontal drainholes may be required in selected areas to control groundwater pressures. Piezometers should be installed at appropriate locations to monitor groundwater if problems develop.

Control of surface drainage is extremely important. A carefully engineered surface drainage system with drainage ditches should be provided on all roads and selected benches to control water flow in the pit. Many of these drainage ditches should be lined in appropriate areas.

(v) Artificial Support

The steep slopes recommended herein could result in occasional failures which could result in loss of berms or loss of part of the haulroads. It is suggested that either predowelling or postdowelling with cables may be beneficial in some areas to prevent loss of the bench catchment immediately above haulroads or loss of part of the haulroad itself. Location and number of dowels would be determined based on detailed geologic structural mapping during excavation to identify possible failures so that appropriate remedial measures could be designed. It is noteworthy that, where any failures appear imminent or in a state of slow translation which could significantly affect the operation, predowelling may be the most effective approach.

(vi) Trial Slopes

Trial slopes, consisting of 30m high benches, steeper bench face angles and/or narrower berms, could be worked into the slope on a trial basis with due consideration of operational constraints and ramifications if the trials are not satisfactory. Trial slopes are particularly desireable in that the stability and feasibility of 20m and 30m high benches is essentially unknown. Results of monitoring of trial sections would be useful for preparing final slope designs in the open pit as mining proceeds.

(vii) Cleaning Berms and Scaling

Rockfall hazards can be minimized if berms are cleaned and all benches are carefully scaled during excavation. This is particularly important in the vicinity of haulroads or critical installations.

(viii) Geological Mapping and Monitoring

All slopes should be mapped during excavation. This is particularly important on the south wall where relatively little information is available concerning the geological structure below the South Wall Fault Mapping should be updated on an ongoing basis and assessed to predict changes in structural geology conditions. In this way justificiation for flattening or steepening can be considered on a rational basis in the lower reaches of the pit. Slopes should be carefully observed for signs of instability, ten sion cracking, movement, etc. If instability deve- lops, a survey monitoring network should be established to determine the magnitude and rates of movement.

8.7 DESIGN OF SLOPES ON THE SOUTH WALL ADJACENT TO AND SOUTH OF THE SOUTH WALL FAULT

Slopes in Design Sectors 1-1, 2-1 and 3-1 south of the South Wall Fault require special design considerations. This is due to the pre- sence of weaker rocks in the weathered zone, the strong structural control of the South Wall Fault as well as presence of relatively shallow kinematically possible failure modes. Kinematic stability analyses of structural discontinuities and back analyses of failures in weathered rock, as well as along discontinuities, were used to prepare a rational design for slopes in these design sectors. Because of the constraints on slope design which arise as a result of the location of the primary crusher and ore stock piles near the pit crest, specific remedial measures including, controlled excavation, artificial support and drainage are recommended to develop slopes at angles steeper than those which would normally be anticipated for the slope forming materials which exist in this area. Feasibility of using fully grouted cable dowels has been demonstrated by pull out tests of trial dowels. Pull out tests on dowels in ten selected locations indicated that acceptable loads were obtained in all dowels tested. It should be noted that, because kinematically possible failures could develop on the south wall slope, artificial support consisting of grouted dowels is necessary to maintain steep slope angles. Alternative methods of artificial support such as gabions, buttresses, shotcrete etc. would not be sufficient to prevent the failures along joints documented in these slopes.

Degree of weathering decreases and rock strength increases with depth in slopes south of the South Wall Fault (as summarized in Table I). Hence, slope design must be developed for individual benches based on the possibility of both rotational failure and failure on discontinuities. Back analyses results for both types of failure (see Section 6) and kinematic stability analysis results summarized in Table IX have been used to prepare a rational slope design for each bench and the entire slope. Details of the recommended design are shown in Fig. 19.

When designing artificial support, it is important that the strength provided by the dowels or anchors is fully utilized. In addition, it is imperative that an adequate distribution of pattern support be established across the failure surface. Hence, although it may be possible to develop high strength on individual dowels, it will be necessary to install a sufficient number of dowels to provide an even distribution of support in the slope.

The slope designs presented in this section pertain only to slopes south of the South Wall Fault and for the first bench below and north of the South Wall Fault. Design of slopes more than one bench north of the South Wall Fault is described in Section 8.6.

8.7.1 General Slope Design Recommendations

(i) Excavation Method

In weaker materials on the upper benches it is suggested that flatter bench face angles be adopted to control soil type failure. Steeper bench face should be used in lower benches where rock progressively becomes harder.

The slope should be excavated by ripping in the top three benches. Ripping is recommended on other benches wherever possible. If ripping is not used on lower benches or in the hornfels rock, preshearing or some other acceptable perimeter blasting technique using lightly loaded, closely spaced, small diameter holes should be used. Careful controlled blasting is required to maintain cohesion in the rock mass, since cohesion contributes significantly to stability of the south wall.

(ii) Artificial Support

Where required, use of vertical and inclined fully grouted dowels are recommended to provide artificial support of the slope to increase the stability of kinematically possible failures and enable steeper overall slopes to be mined on the south wall. Vertical dowels should consist of old drill steel, steel rail, beams, reinforcing bars or equivalent stiff members grouted into clean vertical holes using an acceptable cement grout. Inclined dowels would consist of degreased cables grouted into drillholes inclined at about -10^{o} (i.e. 10^{o} below horizontal).

Dowels should be placed in clean drillholes which have been thoroughly flushed to remove all loose debris, dust and drill cuttings. Cables should be clean and free from grease or deleterious substances. Dowels should be centred in the holes as best as possible. Grout should be an acceptable nonshrink cement grout capable of developing adequate bond strength. Grout should completely fill the drill holes and cables should be fully encased in grout. Preliminary lengths and spacing of dowels are recommended based on stability analyses results and necessity to obtain distribution of pattern support across possible failure surfaces.

If adequate distribution of support could be obtained with dowels it would be feasible to increase dowel spacing, although care should be taken to support all possible blocks in the slope. In the following a standard dowel spacing of 3m has been recommended for 200 kN (20 Tonne) dowels. In that dowel pullout tests indicate that it is possible to develop ultimate loads in excess of 700 kN (70 Tonnes) and design loads of 500 kN (50 Tonnes). In this regard it may be feasiblle to increase dowel spacings to about 5m on a trial basis on benches below the 365m elevation, provided that adequate distribution of support and sufficient restraint of surface blocks is obtained.

If additional support is required to retain small blocks or ravelling, it is suggested that bearing plates may be required for the dowels. Alternately, steel straps might be placed between the dowels for additional support. Also a sufficient length of dowel could be left at the face to enable the dowels to be laced together to provide additional support if required.

(iii) Drainage Ditches

A proper sealed drainage ditch should be placed at the pit crest to prevent infiltration of surface water. The area behind the pit crest should be sloped away from the pit and/or slightly cambered so that all surface water is directed into drains and carried away from the south wall. The existing topography in the south wall area is favourable for construction of a well engineered surface drain at the pit crest.

Drainage ditches should be placed on each bench to convey surface water or drainage water off the slope. Benches should be slightly inclined towards the drainage ditches. Benches should be given a sufficient grade across the slope to carry all water in surface drains off the slope and to a sump for removal from the pit. The haulroad possibly also should be inclined or cambered to control surface water, and a well engineered sealed ditch should be placed at the edge of the haulroad.

(iv) Sub Horizontal Drainholes

Drainholes, slightly inclined upwards at $+5^{0}$ to $+10^{0}$, approximately 30 metres long and spaced at 10 to 15 metre intervals should be installed on each bench. Spacing of drainholes may be increased on lower benches, depending on the volume of water encountered and how well surface water infiltration is controlled. Piezometer should be placed at selected locations on benches to monitor the effectiveness of drainholes and assist with determination of optimum length and spacing of drainholes.

(v) Piezometer Installations

Two to four piezometers should be installed on each bench on the south wall to assess effectiveness of drainholes and to monitor groundwater pressures in the slope.

(vi) Geological Mapping

The wall should be mapped as excavation proceeds and additional analyses carried out to determine dowel requirements in the lower reaches of the south wall. This is particularly important if dowel spacing is increased on a trial basis. It is mandatory that mapping and analysis be carried out during excavation to assess this design and apply and/or modify remedial measures as required.

(vii) Monitoring

Slopes should be monitored for movement on a regular basis, using relatively simple techniques. If movements are indicated, detailed monitoring should be initiated to determine the extent and nature of movement to enable rational design of remedial measures.

8.7.2 Design of the First Bench Above 375m

(i) One to two rows of vertical dowels should be installed at the crest of the slope. If two rows are used, they should be about 2m apart, and should be staggered with respect to the first row. Dowels should be 10m long and spaced at 3m along the crest. Dowels should consist of old drill steel, steel rail, H beams, or reinforcing bars grouted into blastholes drilled from the crest.

(ii) The slope should be ripped at a slope angle of 50^{o} in two lifts. The first lift should be ripped to 380m elevation and one row of vertical dowels should be installed as above. These dowels should be 7m long and spaced at 3m along the slope.

Where the pit crest extends above 385m, a row of vertical dowels should also be installed at 385m as described above (see Fig. 19).

(iii) The slope should continue to be ripped at an angle of 50^{o} to 375m elevation and a 5m wide berm should be established at 375m. Horizontal drainholes and a sealed ditch should be installed at the 375m elevation before commencing further excavation of the slope.

8.7.3 Design of the Second Bench (365 to 375m)

(i) The bench face should be ripped at 57° in two lifts. Firstly, rip the slope to 370m and install one row of 200 kN (20 Tonne) cable dowels at -10° at 3m spacing at elevation 373m. Install a second row of cable dowels at -10° at 3m spacing at 370m elevation. These dowels should be 6m long (see Fig. 19).

(ii) Continue to rip the slope at 57° to 365m elevation and establish a 6m wide berm.

(iii) Install horizontal drainholes and a sealed ditch on bench 365m before further excavation.

8.7.4 Design of the Third Bench (355 to 365m)

(i) The bench face should be ripped at an angle of 70° in 2 to 3 lifts down to elevation 355m. Install three rows of 200 kN (20 Tonne) cable dowels at -10° at elevations of 364m, 362m and 360m after each lift and as excavation proceeds. Dowels should be 8m long, spaced at 3m. These dowels should be placed prior to excavating more than 2m below each row. Dowel spacing could be increased to 5m on a trial basis if 500 kN (50 Tonne) cable dowels are used provided rock mass conditions would appear to be feasible for the wider dowel spacing.

(ii) A 6m wide berm should be established at elevation 355m.

(iii) Horizontal drainholes should be installed and a drainage ditch should be established at elevation 355m prior to further excavation below 355m elevation.

8.7.5 Design of the Fourth Bench (345 to 355m)

(i) Design is the same as for the third bench. If ripping is not possible control blasting should be used. Excavation lifts should not exceed 5m height in order that dowels can be placed prior to excavation of lower lifts. Three rows of 200 kN (20 Tonne) cable dowels should be installed at 354, 352 and 350m after each lift and as excavation proceeds. Dowels should be 8m long and spaced at 3m. As described above, dowel spacing could be increased to 5m on a trial basis if 500 kN (50 Tonne) dowels are used.

(ii) No subgrade drilling should be carried out.

8.7.6 Design of the Fifth Bench (325 to 345m)

(i) This bench should be 20m high (double bench) and the bench face should be ripped at 70^{o} in 5m high lifts. If ripping is not possible, adequate perimeter control blasting should be used. Lifts should be a maximum height of 5m in order that dowels may be placed prior to excavation of lower lifts. Install five rows of 500 kN (50 Tonne) dowels inclined at

-10^{o} at elevations 343m, 340m, 337m, 334m and 331m with specific dowel lengths of 12m, 10m, 9m, 8m and 6m, respectively. Dowels should be spaced at 3m. Because the bench height is 20m, a greater number of dowels is required than for the 10m high bench. Hence, 500 kN (50 Tonne) dowels spaced at 3m are required (see Fig. 19).

(ii) Establish a 10m wide berm at elevation 325m.

(iii) Drainholes and a drainage ditch should be placed at the toe of the bench face.

(iv) No subgrade drilling should be carried out.

(v) The need for the fifth bench only exists for a distance of about 60m on the south side of the South Wall Fault.

(vi) Additional benches below 325m south of the South Wall Fault would be developed using the same procedure as described for the fifth bench.

8.7.7 Slope Design Immediately North of the South Wall Fault

(i) It is recommended that 20m high benches should be used north of the South Wall Fault. Details of slope design in design sectors north of the South Wall Fault are given in section 8.6. Because of the geometry of South Wall Fault, this fault could lead to local instability if undercut. Therefore, it is important to design the first bench north of the fault in such a manner that the fault is not undercut by mining. In this regard a bench face angle of 70^{o} is recommended on the first bench immediately north of the South Wall Fault.

(ii) Drainholes are essential to drain or depressurize the fault for one to two benches north of the fault.

(iii) Benches north of the South Wall Fault should be excavated using preshearing, cushion blasting or some other acceptable perimeter blasting techniques, using lightly loaded and closely spaced steeply inclined drillholes. No subgrade drilling should be carried out on benches.

(iv) Dowel requirements north of the South Wall Fault are difficult to estimate due to limited information available in this area. Preliminary assessment of data indicates joints exist which dip at 44^{o} to 50^{o} in the rock in this area. Continuity of these joints is difficult to assess and dowelling may or may not be required.

(v) For the support of the haulroad it is advisable to anticipate the use of grouted dowels similar to those required for the 20m bench south of the fault (as described in Section 8.7.6 above). On lower benches dowelling requirements probably will be reduced and should be used primarily to contend with individual failures as required.

9. SUMMARY AND RECOMMENDATIONS

A detailed engineering geology and rock mechanics data collection and analysis program was carried out at the Mount Carbine Mine for the purpose of conducting slope stability analyses and preparing an optimum design of final slopes in the open pit. Particular emphasis was placed on design of slopes and remedial measures in weak and weathered rocks on the south wall of the open pit.

9.1 DESCRIPTION OF THE INVESTIGATION

Detailed geological mapping was carried out on all accessible benches and outcrops in the open pit to determine the physical and mechanical properties of slope forming materials. Two diamond drillholes and eleven percussion drillholes were used to define the nature of the rock and distribution of rock strength behind the south wall. Core orientation and core logging were carried out for selected existing diamond drillholes in the pit to assist with correlation of geological structures, such as foliation, to delineate the South Wall Fault and to develop a three dimensional rock mass model.

Core samples of intact rock and discontinuties were tested to obtain basic unconfined compressive strength and shear strength properties for design. Sealed piezometer and standpipes were installed in drillholes on the south wall to obtain an appreciation of groundwater conditions in this area of the open pit.

Lower hemisphere equal area projections were used to determine the attitude, general characteristics and behaviour of the various sets of discontinuities in each area of the pit and to define structural domains. Based on lithology, distribution of structural domains and orientation of proposed final pit walls, the pit was divided into design sectors. For each design sector, kinematic analyses and the appropriate mechanical stability analyses were carried out for each possible failure mode to determine the optimum design of interramp slopes between haulroads. Effects of bench breakback, plan radius of curvature, blasting methods, groundwater distribution, as well as constraints of mining geometry were also considered in the design.

9.2 ENGINEERING GEOLOGY

9.2.1 Lithology and Rock Strength

The engineering properties of rocks in the open pit appear to be controlled primarily by processes of silicification and weathering. The bulk of the rocks in the open pit consist of fine grained metasediments of the Hodgkinson Formation which is of middle Devonian to Lower Carboniferous age. To the north of the South Wall Fault moderate to intense silicification of the metasediments has resulted in the formation of tough hard hornfels. This rock appears to be very resistant to surface weathering. Laminated hornfels and banded hornfels have been mapped within the open pit. South of the South Wall Fault the rocks consist of unsilicified or weakly silicified green argillaceous schist and siliceous greywackes, which are designated as Iron Duke rocks. Moderately silicified hornfels also occur south of the South Wall Fault on the east side of the open pit. Unsilicified and weakly silicified rocks have low to moderate strength and are extremely susceptible to weathering. As a result, rocks south of the South Wall Fault have formed a weathering profile up to 30m deep which varies from residual soil at surface and changes gradually to highly weathered, moderately weathered and slightly weathered rocks with increasing depth. As the degree of weathering decreases the rock strength and competency increases. Consequently, the slope design within the weathered profile south of the South Wall Fault is different on each succeeding bench as the rock strength and competency increases.

9.2.2 Structural Geology

Rational slope stability analysis and pit slope design requires that the pit be subdivided as best as possible into areas of approximately similar geologic structural characteristics. These areas are designated structural domains.

Structural domains at Mount Carbine were determined initially by assuming the boundaries of the structural domains were major faults, geological contacts and/or boundaries of areas with similar foliation. Attitudes of the geologic structural populations within different parts of each structural domain were then evaluated and compared. The distribution of major structural features, main rock types and structural domains is shown in Fig. 1. A brief description of the main geologic structural features in the open pit is given in the following, based on detailed statistical analyses of available data using appropriate techniques.

(i) Foliation

Foliation is a well developed laminated structure developed parallel to "compositional layering" in the metasediments. Foliation in hornfels dips steeply to the northeast or southwest at about 70^o to 90^o. Average dip directions vary from 041^o to 056^o and 221^o to 239^o. The Iron Duke rocks south of the South Wall Fault have foliation that dips due north at approximately 75^o.

(ii) Joints

In addition to foliation joints (Joint Set A), three joint sets and several minor and miscellaneous joint sets occur within the pit area. Joint Sets A and B occur consistently throughout the pit, while Joint Sets C and D occur in all structural domains north of the South Wall Fault. Joint Sets E and G and other miscellaneous joints only occur in specific structural domains, and appear to be significant on a local basis only. Differences in orientation and geotechnical properties of joint sets have been documented in individual structural domains in the pit.

(iii) Veins

Tungsten bearing quartz veins and barren quartz veins are sub-vertical and occur subparallel to joints of Joint Set B or along pre-existing joints of Joint Set B. Two subsets are common in most areas, corresponding to the subsets of Joint Set B. Veins are very continuous and can often be traced over several benches.

(iv) Faults and Shears

Two major faults and a number of small faults and shears have been recognized in the pit. The South Wall Fault is the most continuous and significant structural feature in the pit. It has a uniform strike and dips towards the north at 70^{o}. This fault forms the southern limit of the zone of extensive silicification and delineates the boundary of the ore zone. The Iron Duke Fault occurs in the northeast corner of the pit and dips steeply to the north/northeast at about 85^{o}.

Two main fault and shear sets exist in the hornfels rocks. These faults have dip directions of 035 and 045 and dips of 72^{o} and 35^{o}, respectively. These faults could affect stability in the south and south-west sides of the open pit, in that they may daylight due to mining.

9.2.3 Strength Properties

Intact strengths of samples of fresh and weathered rock in the open pit have been determined primarily from simple field assessments of rock hardness. A limited number of laboratory tests consisting of uniaxial compressive tests and point load index tests were carried out to obtain more accurate estimates of strength parameters in weathered rocks.

Direct shear tests were carried out on selected core samples of four representative joints from the south wall area. Test results indicate that these joints have friction angles greater than 36^o and a relatively high cohesion.

9.2.4 Back Analysis and Documentation of Failures

Back analyses were carried out for a large rotational type failure in residual soil and weathered rock below the primary crusher using standard design charts. Analyses indicate that appropriate values for cohesion and friction angle for these materials are about 40 kPa and 24^o respectively. Comparison of slopes in the pit with road cuts in similar weathered rocks and residual soil indicates that the design parameters discussed above are reasonable. The comparative analysis also indicates that these slopes should be well drained to maintain reasonably steep slopes.

Back analyses were carried out for one large plane failure which occurred on a well defined joint in the south wall. Back analyses results indicate that the cohesion required for a factor of safety of 1.0 to 1.5 varies from 20 kPa to 51 kPa for a slope with a tension crack (see Table VI). Additional analyses were carried out to determine the possible artificial support requirements, assuming that a cohesion of 10 kPa could be mobilized along the joint.

Back analyses results and results of trial installations based on dowel pull out tests indicate that it is feasible to achieve relatively steep slopes on the south wall using artificial support members. Fully grouted vertical or inclined dowels appear to provide ideal support for the slope in question. The number, length and design load of dowels accordingly have been determined from the data collection, back analyses, laboratory tests and dowel pull out test results, from which the relevant design parameters have been obtained.

9.3 SLOPES STABILITY ANALYSIS AND DESIGN

9.3.1 Deep Seated Failure Considerations

Analyses of geological mapping data, core logs and strength testing results indicate that, within the range of overall slope angles which are geometrically feasible, the possibility of deep seated failure of the whole slope along a single major discontinuities is small. Plane failures could possibly occur along shallow dipping faults and shears of Set SR1 on the south wall and/or west wall of the open pit. However, the possibility of occurrence of major faults or shears cannot be clearly stated at this time. Hence, it is not considered justified to flatten the slope in this area based only on possible occurrence of a major fault. However, it is imperative that the slopes be carefully mapped and monitored. If a major failure develops provisions should be made for possible remedial measures such as artificial support, depressurization or possibly slope flattening if required.

Based on the relatively high hardness, and accordingly, high compressive strength of most intact rocks in the pit, the likelihood of time dependent strain and subsequent failure of intact material in the slope appears to be small except within the weathered or weaker rocks on the upper benches of the south wall. Hence, the possibility of deep-seated rotational failure due to failure of intact material involving the whole slope is low (at least within the range of overall slope angles which are geometrically feasible, and which are considered in this analysis).

Specific design and remedial measures to control possible rotational failures in the weathered rocks and residual soil near the crest of the south wall are discussed in Section 8.7.

9.3.2 Stability Analyses and Design North of the South Wall Fault

Within the pit area local failures involving one or two benches are possible in most areas. Consequently, the bulk of the stability analyses were carried out to assess plane and wedge failures on benches, using the structural geologic information available, and were based on valid statistical techniques. This statistical approach in designing the slope was not used for the areas south of the South Wall Fault. In these areas slope design and remedial measures have been prepared for individual benches to enable design of relatively steep slopes in weak, deeply weathered rock. Details of design in this area of the pit is discussed separately below.

Not only must structural domains be considered for individual stability analysis, but another overriding consideration is the orientation of the final pit wall. Different pit wall orientations require basically different design considerations. Hence, it is necessary to define zones of the proposed slope which contain one structural domain and one general pit slope orientation; these zones are designated design sectors.

Within each design sector the main modes of failure were determined and stability analyses carried out for both interramp slopes between haulroads and benches. Due to the considerable number of possible wedge failures of various orientations in most design sectors, limit equilibrium analysis results and wedge distributions were treated statistically. A recently developed statistical analyses technique was used, based on assessing the cumulative percentage frequency of possible unstable wedge failures with respect to the apparent plunge angle and factor of safety of the possible wedges. Slope angles were determined assuming that 20 percent of the possible wedges could spill over berms if they failed. Design charts were used to develop slope designs for interramp slopes based on the mode of failure which was considered to control slope stability. Due consideration also was given to bench breakback, effects of plan radius of curvature and groundwater conditions.

Slope stability analyses were carried out for both 20m high and 30m high benches. In most areas of hard unweathered silicified hornfels, 30m high benches appear to be feasible provided undue breakback does not occur due to uncontrolled blasting or bench failures. 30m high benches would result in wider berms, giving more reliable access and rockfall protection.

Contingent on successful trials, 30m high benches are recommended in the hornfels rocks above 345m elevation and below 305m elevation. Between 305m elevation and 345m elevation 20m high benches are recommended to insure reliable access to the south wall for maintenance of remedial measures and slope protection in the vicinity of the haulroad.

Information on individual design sectors, analyses results and specific slope design recommendations are presented in Table IX and Fig. 14. Modification to the design, based on recommended bench height, effects of breakback, plan radius of curvature and other relevant considerations for individual design sectors, are summarized in Fig. 17. In order to assist with mine planning and requirements of blending slope designs between individual design sectors, the general distribution of recommended interramp slope angles in each area of the open pit is shown in Fig. 18*.

Slope designs shown in Fig. 18 assume dry (i.e. drained or dewatered slopes). In areas where groundwater pressures are anticipated, slope depressurization using subhorizontal drainholes may be required.

9.3.3 Design of Slopes and Related Remedial Measures Adjacent to and South of the South Wall Fault

Slopes in Design Sectors 1-1, 2-1 and 3-1 south of the South Wall Fault require special design considerations. As discussed earlier, this is due to the presence of weaker rocks in the weathered zone, the strong structural control of

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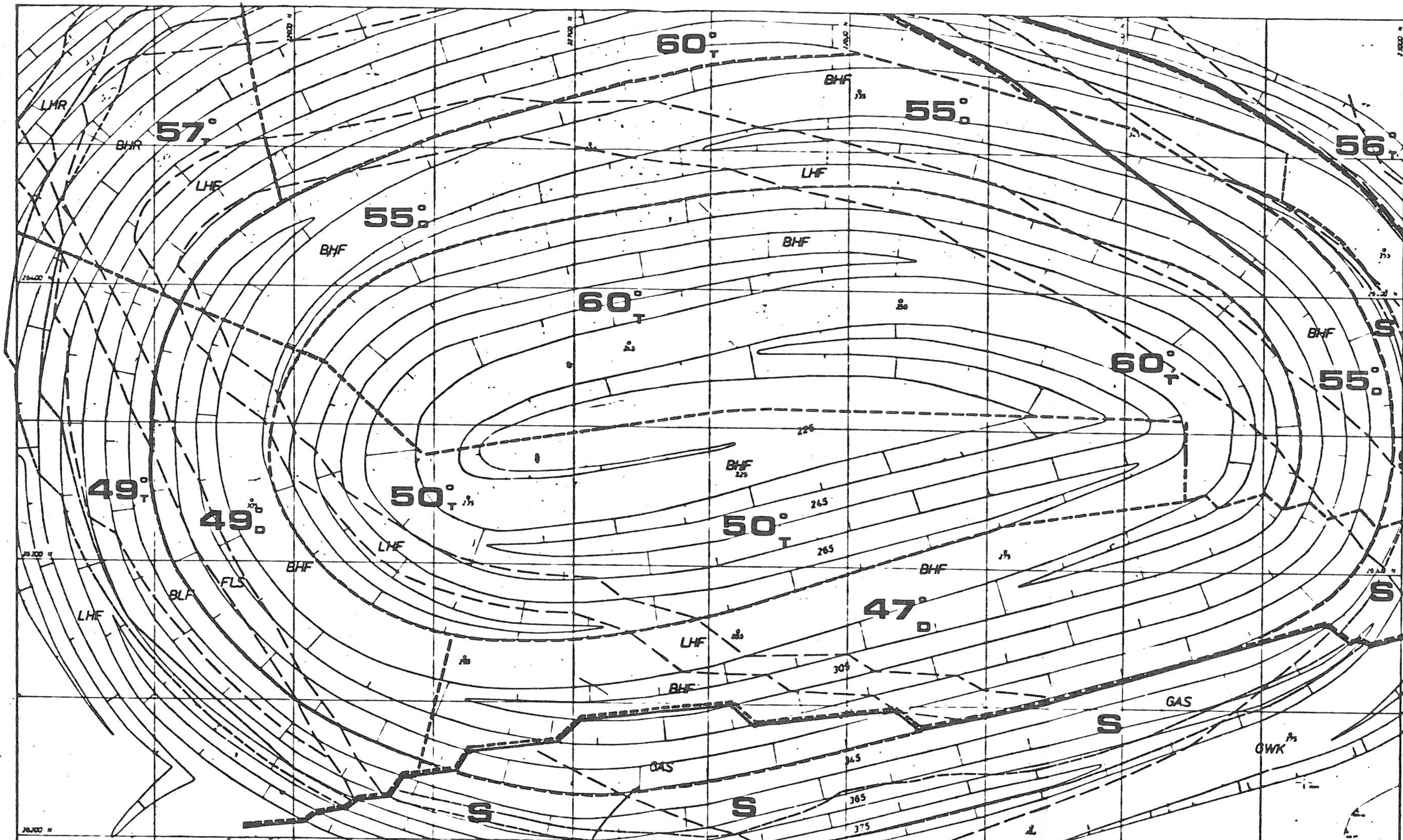
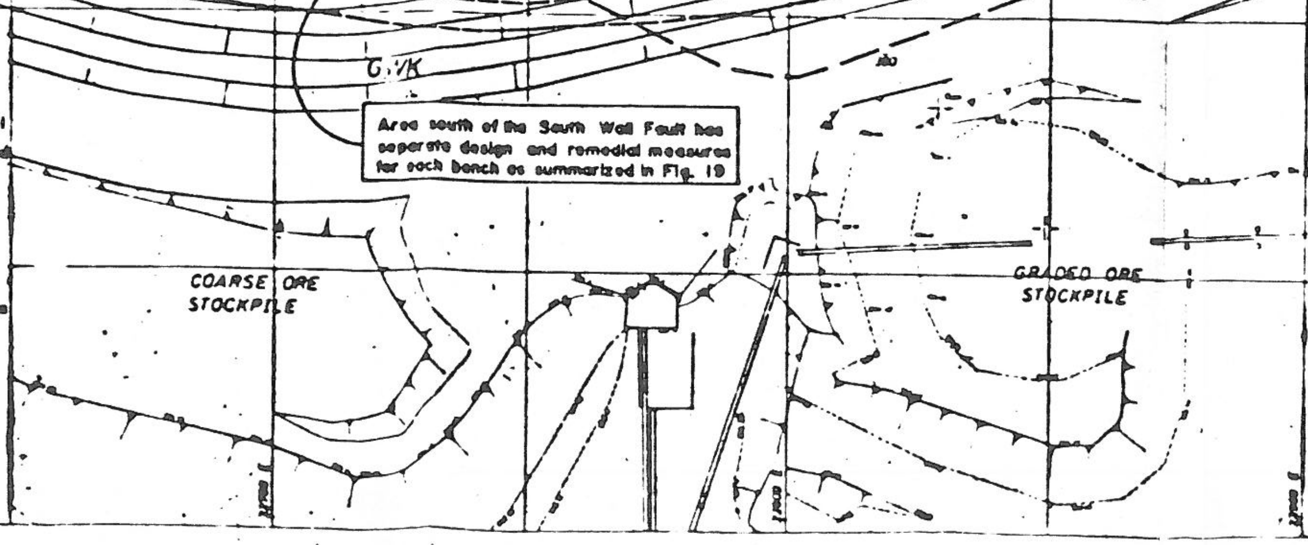
* Fig. 18 is included in the text (following page) as well as at the end of the report for convenience.





NOTES

1. The slope angles recommended are for interramp slopes. Recommended slope geometry and remedial measures for each design sector are summarized in Table IX.
2. Slope angles recommended are for dry slopes.
3. Interramp slope may be steepened on a trial basis in the north wall provided adequate access and safety is maintained and slopes are monitored.

SYMBOLS

Boundary of area with equal recommended interramp slope angle ....

Recommended maximum interramp slope angle between haulroads attainable with adequate groundwater depressurization (D = 20m high double benches; T = 30m high triple benches)............ 47° D

Area where single 10 m high benches are recommended for access to the south wall............ S

SCALE

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R & MINING PTY LTD

JULY 1980 PROPOSED FINAL PIT OUTLINE

GENERAL DISTRIBUTION OF RECOMMENDED INTERRAMP SLOPE ANGLES FOR THE PROPOSED FINAL PIT PLAN

MARCH 82

FIG. 18

the South Wall Fault and the presence of relatively shallow kinematically possible failure modes in the area. Kinematic stability analyses of structural discontinuities, back analyses of failures in weathered rock and artificial support considerations were used to prepare a rational design for slopes in these design sectors. Because of the constraints on slope design, which arise as a result of the location of the primary crusher and ore stock piles near the pit crest, particular emphasis had to be given to stabilization, protection and monitoring measures. Specific remedial measures, including controlled excavation without blasting, application of artificial support, drainage and careful slope monitoring are recommended if these slopes are to be developed at angles steeper than those which would normally be anticipated for the slope forming materials which exist in this area. Feasibility of using fully grouted cable dowels has been demonstrated by pull out tests of trial dowels. Pull out tests on dowels in ten selected locations indicate that acceptable loads were obtained in all dowels tested. It should be noted that, because kinematically possible failures could develop on the south wall slope, artificial support consisting of grouted dowels is necessary to maintain steep slope angles. Alternative methods of artificial support, such as gabions, buttresses, shotcrete, etc. without some form of deep artifical support, would not be sufficient to prevent the failures along joints which exist in these slopes.

Degree of weathering decreases and rock strength increases with depth in slopes south of the South Wall Fault. Hence, different slope designs are recommended for individual benches, based on the possibility of both rotational failure and failure on discontinuities. Details of the recommended design of this slope are shown in Fig. 19*.

9.4 EXCAVATION METHOD AND BLASTING

9.4.1 Excavation Method

Slopes in weak or weathered rock should be excavated by ripping or some other accepted non-explosive method. Slopes should be excavated in lifts of limited depth to permit installation of remedial measures as required. In this regard, excavation lifts of 2 to 5m could be used to considerable advantage in certain areas of the south wall.

9.4.2 Controlled Blasting

In areas where ripping is not possible, all final walls should be developed by preshearing or some other acceptable form of perimeter or controlled blasting techniques. If proper controlled blasting is not used on the final wall, it may not be possible to achieve the steep slope angles recommended. Considerations should be given to using closely spaced, lightly loaded, small diameter blastholes which can be used to develop clean unbroken walls, maintain maximum cohesion in the rock mass and reduce breakback of bench crests. To maximize the effects of the perimeter blasting program, trial blasts should be carried out to determine the optimum blasting method on final walls.

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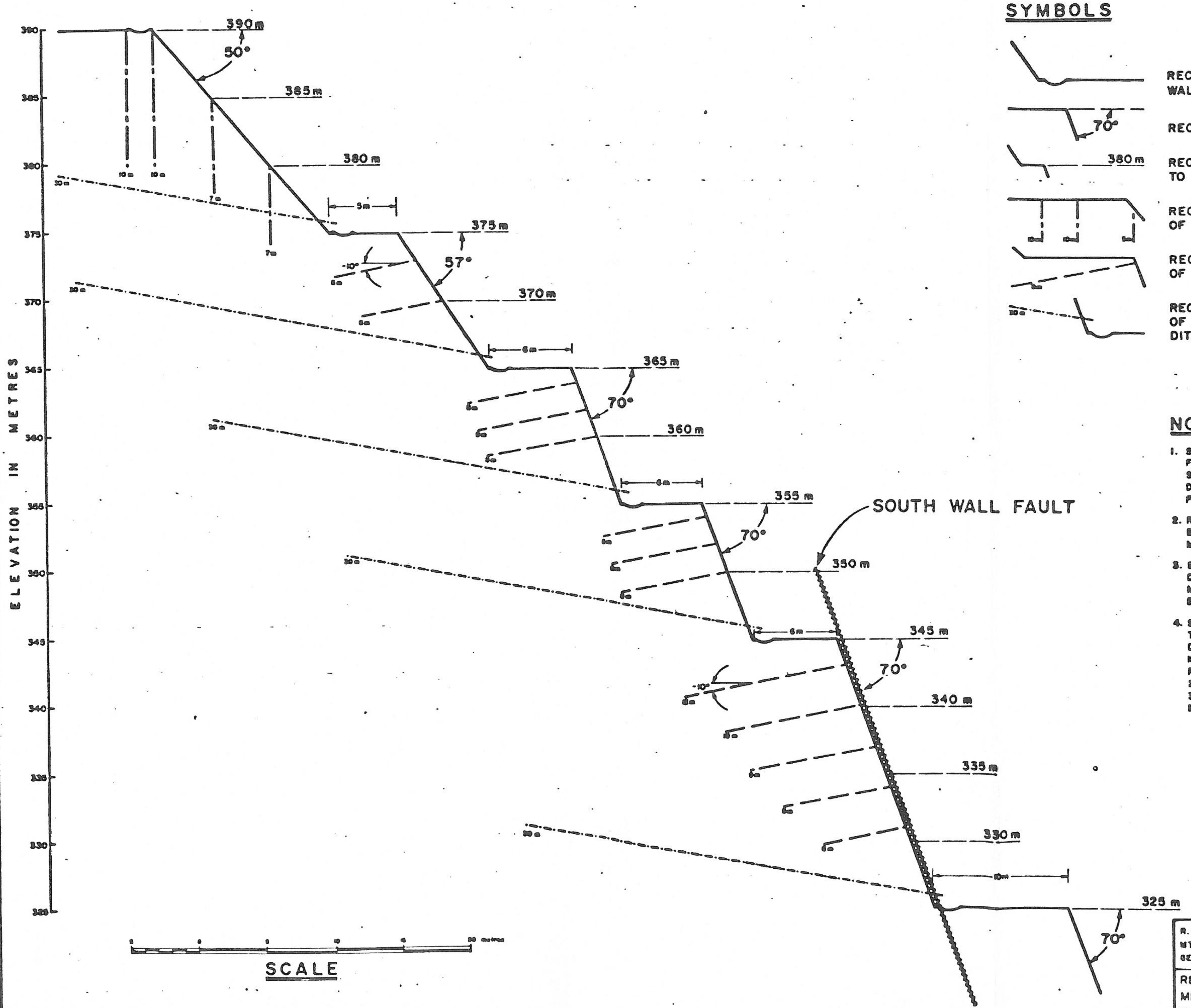
* Fig. 19 is included in the text (following page) as well as at the end of the report for convenience.



SYMBOLS

- RECOMMENDED SLOPE GEOMETRY SOUTH OF SOUTH WALL FAULT
- RECOMMENDED BENCH FACE ANGLE
- RECOMMENDED LEVEL OF EACH EXCAVATION LIFT TO BE USED FOR INSTALLING REMEDIAL MEASURES
- RECOMMENDED LOCATION AND LENGTH IN METRES OF FULLY GROUTED VERTICAL DOWELS
- RECOMMENDED LOCATION AND LENGTH IN METRES OF FULLY GROUTED INCLINED CABLE DOWELS
- RECOMMENDED LOCATION AND LENGTH IN METRES OF SUBHORIZONTAL DRAINHOLES AND DRAINAGE DITCH

NOTES

1. SLOPE DESIGN IS FOR AREA SOUTH OF SOUTH WALL FAULT ON SECTION 22,700 E. RECOMMENDED DESIGN SHOULD BE APPLIED SOUTH OF FAULT ONLY. SLOPE DESIGN FOR AREA NORTH OF FAULT IS GIVEN IN FIG. 18.
2. RECOMMENDED DESIGN ASSUMED SLOPES ARE EXCAVATED BY RIPPING OR OTHER NON EXPLOSIVE EXCAVATION METHODS TO MAINTAIN COHESION IN THE ROCK MASS.
3. SLOPES SHOULD BE CAREFULLY MAPPED AND MONITORED DURING EXCAVATION. EXTENT OF REMEDIAL MEASURES MAY BE VARIED DEPENDING ON ACTUAL CONDITIONS ENCOUNTERED DURING EXCAVATION.
4. SPACING OF ALL DOWELS IS ASSUMED TO BE 3m TO INSURE ADEQUATE SUPPORT AND ACCEPTABLE DISTRIBUTION OF SUPPORT IN THE SLOPE. INCLINED DOWELS ABOVE 345m ELEVATION ARE REQUIRED TO HAVE A MINIMUM DESIGN LOAD OF 200 kN (20 TONNE). INCLINED DOWELS BELOW 345m ELEVATION ARE REQUIRED TO HAVE A MINIMUM DESIGN LOAD OF 500 kN (50 TONNE).

FIG. 19

R. B. MINING PTY. LTD.
MT. CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

RECOMMENDED SLOPE DESIGN AND REMEDIAL MEASURES FOR SLOPES SOUTH OF THE SOUTH WALL FAULT

FEB 82
81-390

Regardless of location, to protect haulroads it is recommended that a more sophisticated or higher level of control blasting be carried out on the bench immediately above and below haulroads. Control blasting could also be used in the vicinity of other important installations in the pit. If control blasting is not sufficient to maintain the integrity of the slopes, berms may be reinforced by grouted dowels in site specific areas.

9.4.3 Production Blasting

Blast patterns, most particularly those near the final slope, should be designed so that blasting damage to the pit wall is minimized. With regard to benches, for example, this can be done by reducing subgrade drilling and having blastholes span the bench crests. Accurate location of all blastholes by surveying is essential. Optimum results can only be obtained by varying blasting techniques and correctly supervising and designing field trials where loads, spacing, burden, delays, etc. are varied to obtain the best possible results. It is sound engineering to make special provision for a series of test blasts to ensure optimum results for existing operating conditions.

9.5 GROUNDWATER CONTROL AND DRAINAGE

Stability analyses indicate that in certain design sectors groundwater control or depressurization could be used to advantage. The slope design for these cases is shown in brackets for the appropriate design sectors in Table IX and Fig. 14. Control of groundwater and surface water is essential in the south wall area and behind the South Wall Fault, as well as in some design sectors in other areas of the open pit.

In areas where groundwater is expected to affect slope stability, groundwater control and depressurization measures, consisting of subhorizontal upward inclined drainholes, are recommended near bench toes. Drainholes are particularly important on the south wall to control water pressures in weak or weathered rocks south of the South Wall Fault. Drainholes are also recommended to control water pressures behind the South Wall Fault or other major faults. Drainholes may also be required in selected areas of the pit to control water pressures which could precipitate local failures.

Sealed piezometers should be installed on the south wall as the final pit is excavated. These piezometers should be monitored regularly to assess the effectiveness of drainholes, and to clearly determine the length and spacing of drainholes required to obtain effective depressurization of the slope.

9.6 SURFACE WATER CONTROL

Due to the unfavourable effects of groundwater on slope stability, particularly in the south wall of the open pit, it is recommended that a properly designed system of surface water control be constructed in the pit. Surface water control should consist of the following:

(i) a sealed drainage ditch behind the pit crest in all areas to intercept all water which could infiltrate into the rock mass

(ii) the area around the entire pit crest should be properly graded to divert water into the drainage ditch or away from the pit

(iii) benches and haulroads on the south wall should be inclined and/or graded to divert water off the slope

(iv) drainage ditches or closed pipes are important to collect and control water from drainholes on the south wall

(v) a sealed drainage ditch should be constructed on the haulroad to convey water off the slope and into sumps for pumping out of the open pit. Uncontrolled water flow should not be permitted in any area of the open pit.

(vi) sealed drainage ditches should be constructed in areas of the pit other than the south wall as required.

9.7 TRIAL SLOPES

In addition to the slopes recommended, other slope geometries could be considered on a trial basis, on small sections of the wall. As stated earlier, trial slopes would be particularly useful for assessing the feasibility of using 30m high benches on the final slope. The performance of trial slopes will be helpful in modifying the slope for succeeding mining phases.

Trial slopes might include steeper bench faces, higher benches or narrower berms. Such trial slopes should be developed in areas where failures can be allowed to occur without affecting the efficiency and safety of the operation. If failures do occur on a particular trial section, provision should be made so that the failure can be controlled, and the mining geometry modified accordingly, without jeopardizing the safety or efficiency of the mine operation.

All significant failures should be completely and accurately documented so that back analyses can be carried out to help assess the rock mass strength for design.

9.8 CLEANING BERMS AND SCALING

With due considerations of efficiency and safety, benches should be adequately scaled to minimize rockfalls. Debris buildups may require cleanup at a later stage in certain areas. Appropriate berms should be kept relatively free of excessive buildup of rockfall, ravelling and slide material to maintain adequate catchments. Such berms, if possible, should be accessible from both ends so that access will not be lost if a failure occurs.

9.9 SUMMARY OF RECOMMENDATIONS FOR ONGOING WORK RELATED TO PIT DESIGN

The following summarizes many of the aspects discussed earlier pertaining to ongoing work that should be considered by the mine in terms of stability and design of slopes in the pit.

(i) Geologic Structural Mapping

Because of the complex geology and importance of the geological structure to slope design, detailed geologic structural mapping should be carried out on a reasonably continuous basis as mining proceeds. This information should be compared to the existing information and should be used to determine if any slope design modifications are required as mining proceeds. All geological structures, most particularly major faults and shears, should be carefully mapped and described. This would help to correlate major geological structures and to update important information on faults which have already been mapped and described. The ultimate objective should be to establish an appropriate engineering geologic data storage and retrieval system such that local design problems can be analyzed on a rational basis. The computer programs available on the mine should assist greatly in this regard.

(ii) Documentation and Back Analysis of Slope Failures

Documentation and back analysis of slope failures should be carried out to obtain additional information on rock mass shear strength, shear strength of discontinuities and general behaviour of slope forming materials.

(iii) Trial Slopes and Trial Blasts

A program of trial slopes and trial blasts should be considered to evaluate the proposed bench and overall slope geometry as well as the blasting procedures to be used in the pit. In conjunction with the above work, an analysis of bench breakback could be carried out to compare the designed and "as constructed" slope geometry, thus evaluating the design and effects of control blasting and other remedial measures.

(iv) Monitoring Slopes

Slopes should be monitored for horizontal and vertical movement at appropriate intervals, depending on the degree and nature of the movement. Carefully installed ground control points would probably suffice. Also, a periodic visual inspection of the pit crest should be carried out to determine visible signs of movement.

Consideration could also be given to utilizing the dowels (or possibly rockbolts) as movement monitoring devices. If some of these artificial support members were ungrouted and installed sufficiently deep, movement could be measured directly or change in tensioning could be observed. In either case, evidence of movement would be known.

(v) Piezometer Installations and Monitoring

As described above, piezometers should be installed in strategic locations to monitor groundwater pressures and to assess the need for groundwater depressurization or additional hydrogeological studies.

9.10 PERSONNEL FOR CONTROL OF SLOPE DESIGN AND REMEDIAL MEASURES

The stability and integrity of the south wall depends on careful excavation and installation of remedial measures as discussed above. In order to insure a high degree of accuracy and acceptable results from the remedial measures the work should be carefully monitored. It is suggested that a qualified technician or engineer be utilized to organize and control all remedial work. His duties would be as follows:

(i) supervise installation of grouted dowels and dowel pullout tests, if required
(ii) supervise installation of drainholes, piezometers and drainage ditches
(iii) plan and evaluate trial blasts and trial slopes
(iv) carry out detailed geological mapping, document slope failures, back analyze slope failures and carry out slope monitoring, as required

(v) establish a movement monitoring system utilizing dowels if required

(vi) evaluate design on a regular basis and carry out appropriate modifications based on results of field performance of slopes

10. ACKNOWLEDGEMENTS

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for D.R. PITEAU & ASSOCIATES LIMITED

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April 30, 1982

Distribution of Final Report:

R.B. Mining Pty Ltd. - 3 copies
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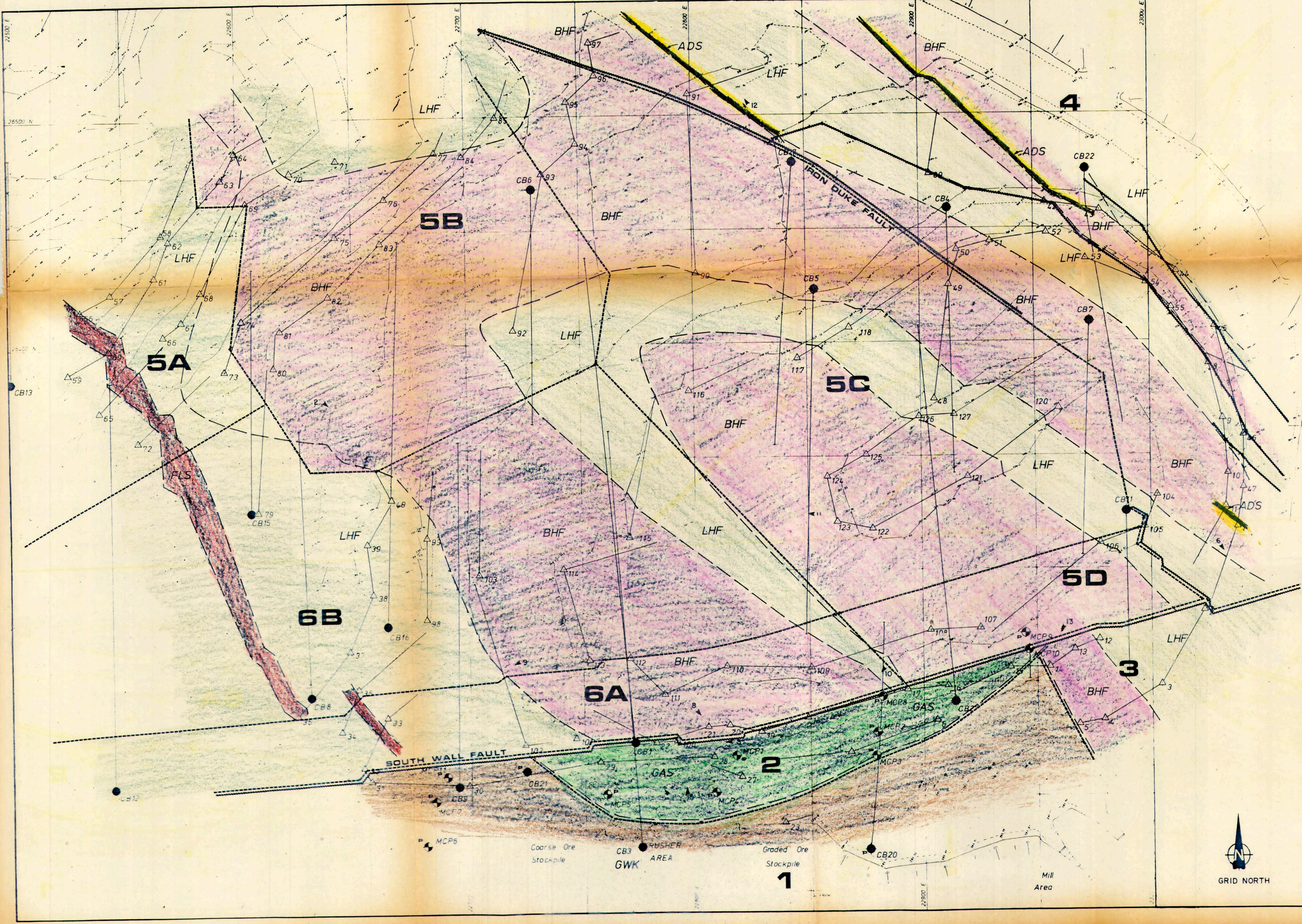

LEGEND

FLS	FELSITE and related coarse grained quartz – porphyry intrusive dykes
ADS	ANDESITE and related intermediate fine grained intrusive rocks which occur as small dykes
LHF	LAMINATED HORNFELS and related hard, grey, highly silicified, laminated rocks derived from micaceous siltstones
BHF	BANDED HORNFELS and related hard, highly silicified rocks with quartz banding, augens, and related features
GWK	SILICEOUS GREYWACKE and related moderately silicified argillaceous rocks with chert and quartz stringers
GAS	GREEN ARGILLACEOUS SCHIST and related non-siliceous, argillaceous rocks
BST	BLACK SLATE and related thinly laminated, well developed slates

SYMBOLS

Geological Contact	
Fault	
Structural Domain boundary	
Structural Domain number	6A
Crest and toe of bench with elevation	
Detailed line mapping traverse: number and location	47
Number and location of percussion drillhole	MCP7
Number and location of diamond drillhole	CB21
Drillhole with piezometers installed	P
Number, location, and direction of photographs	6
Label for Joint Set on lower hemisphere projection	A

NOTE

Contour values on lower hemisphere equal area projections are expressed as percent of the total weight occurring in a 1.0 percent area of the plot.

10 0 10 20 30 40 50 60 70 80 90 100 metres

SCALE

FIG. 1

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MT. CARBINE MINE
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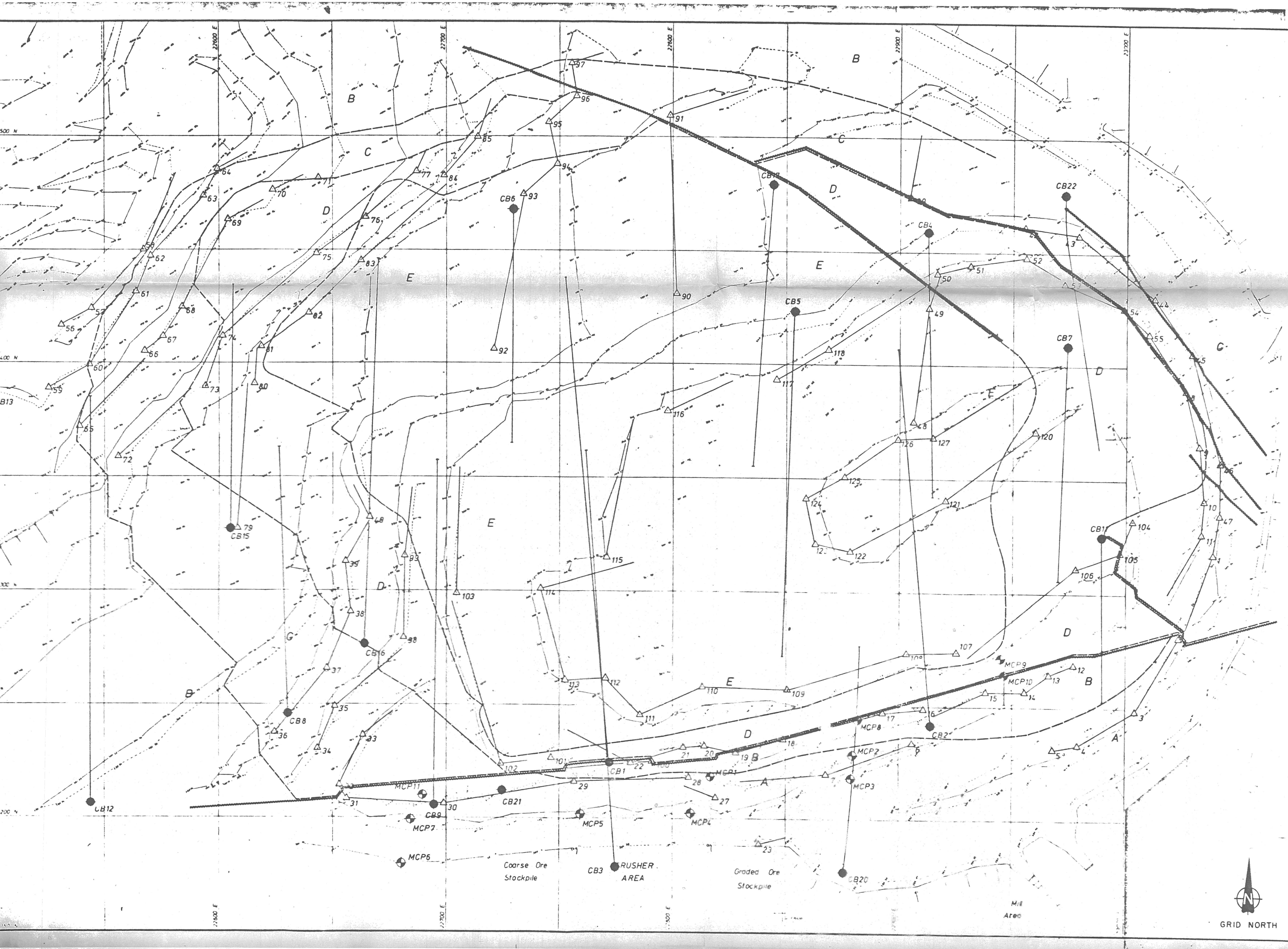
VANCOUVER CALGARY

STRUCTURAL GEOLOGY AND ENGINEERING GEOLOGY IN THE OPEN PIT

BY	DATE
EB DCM	MAR 82
APPROVED	JOB NO.
	81-358



LEGEND

A	EXTREMELY WEATHERED ROCK : Weathering and oxidation has reduced rock to soil with relic structures
B	HEAVILY WEATHERED ROCK : Weathering throughout rock with a significant reduction in rock strength
C	MODERATELY WEATHERED ROCK : Weathering and oxidation along joints and through intact rock with little reduction in rock strength
D	SLIGHTLY WEATHERED ROCK : Weathering and oxidation along joints only with no effect on intact rock strength
E	UNWEATHERED ROCK

SYMBOLS

Estimated boundary of weathered zone

Fault

Structural Domain boundary

Structural Domain number 6A

Crest and toe of bench with elevation

Detailed line mapping traverse : number and location 47

Number and location of percussion drillhole MCP7

Number and location of diamond drillhole CB21

Drillhole with piezometers installed .... P

Number, location, and direction of photographs 6

10 0 10 20 30 40 50 60 70 80 90 100 metres

SCALE

FIG. 2

R. B. MINING PTY. LTD.
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GEOTECHNICAL STUDIES

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DISTRIBUTION OF WEATHERING

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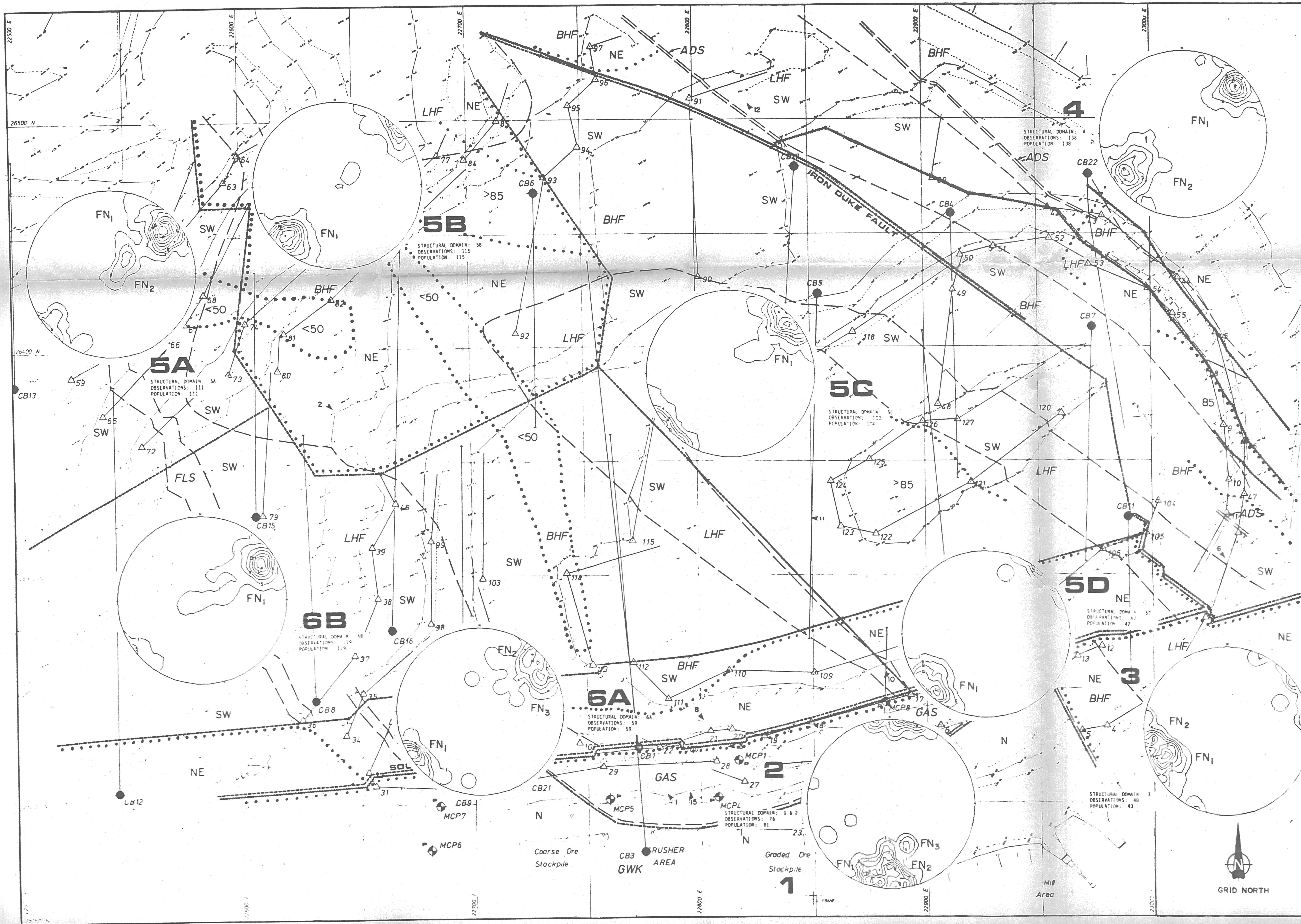

LEGEND

NE	Foliation dip 50 NE - 85 NE
>85	Foliation dip 85 NE - 85 SW
SW	Foliation dip 50 SW - 85 SW
<50	Foliation dip less than 50 SW or NE
N	Foliation dip 45 N - 90 N

SYMBOLS

Boundary of area with similar foliation dip

Geological Contact

Fault

Structural Domain boundary

Structural Domain number ... 6A

Crest and toe of bench with elevation

Detailed line mapping traverse: number and location ... 47

Number and location of percussion drillhole ... MCP7

Number and location of diamond drillhole ... CB21

Drillhole with piezometers installed

Number, location, and direction of photographs ... 6

Label for Joint Set on lower hemisphere projection ... A

NOTE

Contour values on lower hemisphere equal area projections are expressed as percent of the total weight occurring in a 1.0 percent area of the plot.

0 10 20 30 40 50 60 70 80 90 100 metres

SCALE

FIG. 3

R. B. MINING PTY. LTD. MT. CARBINE MINE GEOTECHNICAL STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY	
DISTRIBUTION OF FOLIATION IN THE OPEN PIT	BY: EB DCM APPROVED:	DATE: MAR 82 JOB NO: 81-358

FIG. 4

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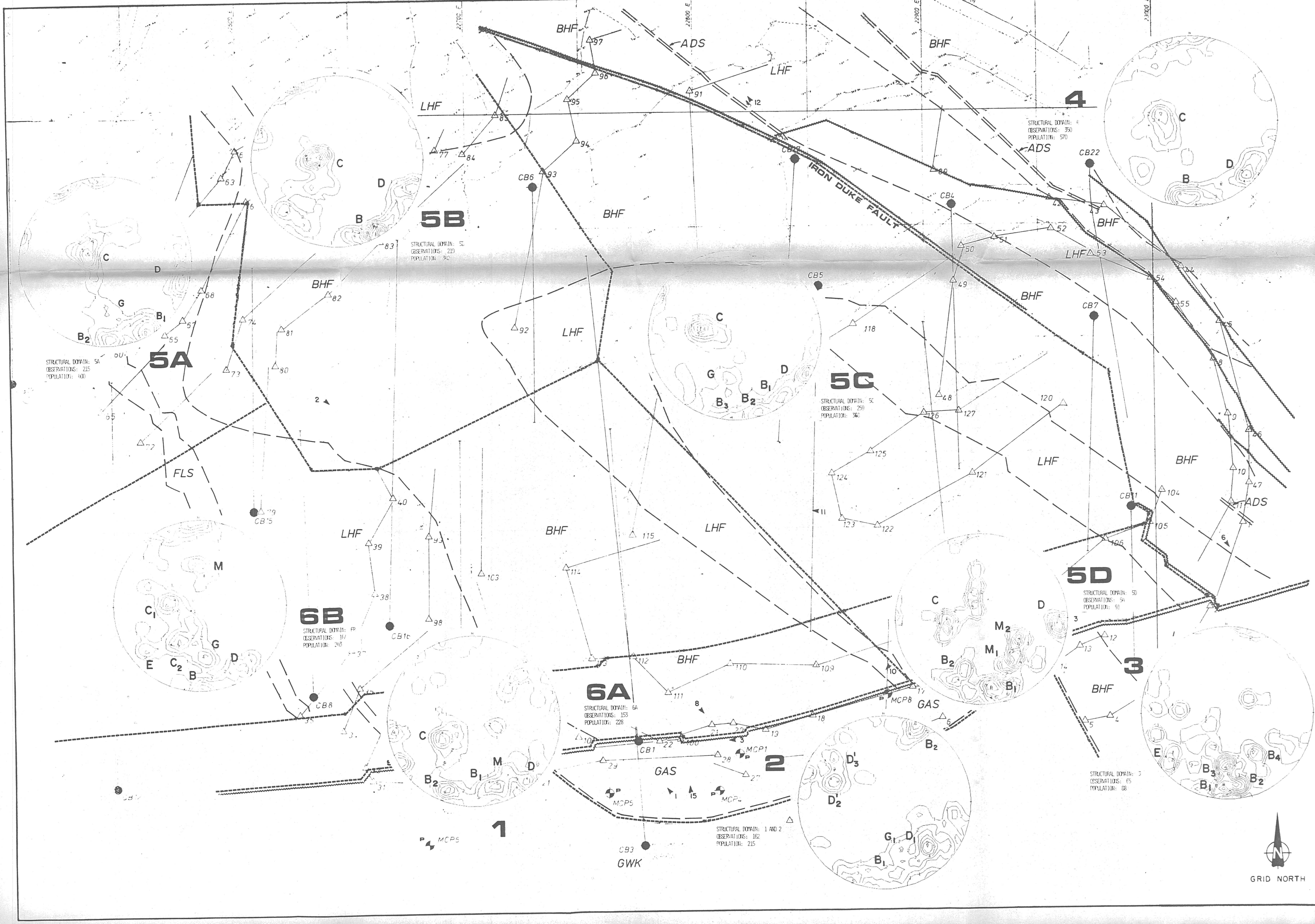
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

DISTRIBUTION OF FOLIATION JOINTS IN THE OPEN PIT

BY: EB DCM | DATE: MAR 82 | JOB NO: 81-358



LEGEND

FLS	FELSITE and related coarse grained quartz – porphyry intrusive dykes
ADS	ANDESITE and related intermediate fine grained intrusive rocks which occur as small dykes
LHF	LAMINATED HORNFELS and related hard, grey, highly silicified, laminated rocks derived from micaceous siltstones
BHF	BANDED HORNFELS and related hard, highly silicified rocks with quartz banding, augens, and related features
GWK	SILICEOUS GREYWACKE and related moderately silicified argillaceous rocks with chert and quartz stringers
GAS	GREEN ARGILLACEOUS SCHIST and related non-siliceous, argillaceous rocks
BST	BLACK SLATE and related thinly laminated, well developed slates

SYMBOLS

Geological Contact

Fault

Structural Domain boundary

Structural Domain number 6A

Crest and toe of bench with elevation

Detailed line mapping traverse: number and location 47

Number and location of percussion drillhole MCP7

Number and location of diamond drillhole CB21

Drillhole with piezometers installed

Number, location, and direction of photographs 6

Label for Joint Set on lower hemisphere projection .... A

NOTE

Contour values on lower hemisphere equal area projections are expressed as percent of the total weight occurring in a 1.0 percent area of the plot.

10 0 10 20 30 40 50 60 70 80 90 100 metres

SCALE

FIG. 5

R. B. MINING PTY. LTD.
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GEOTECHNICAL STUDIES

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DISTRIBUTION OF JOINTS IN THE OPEN PIT

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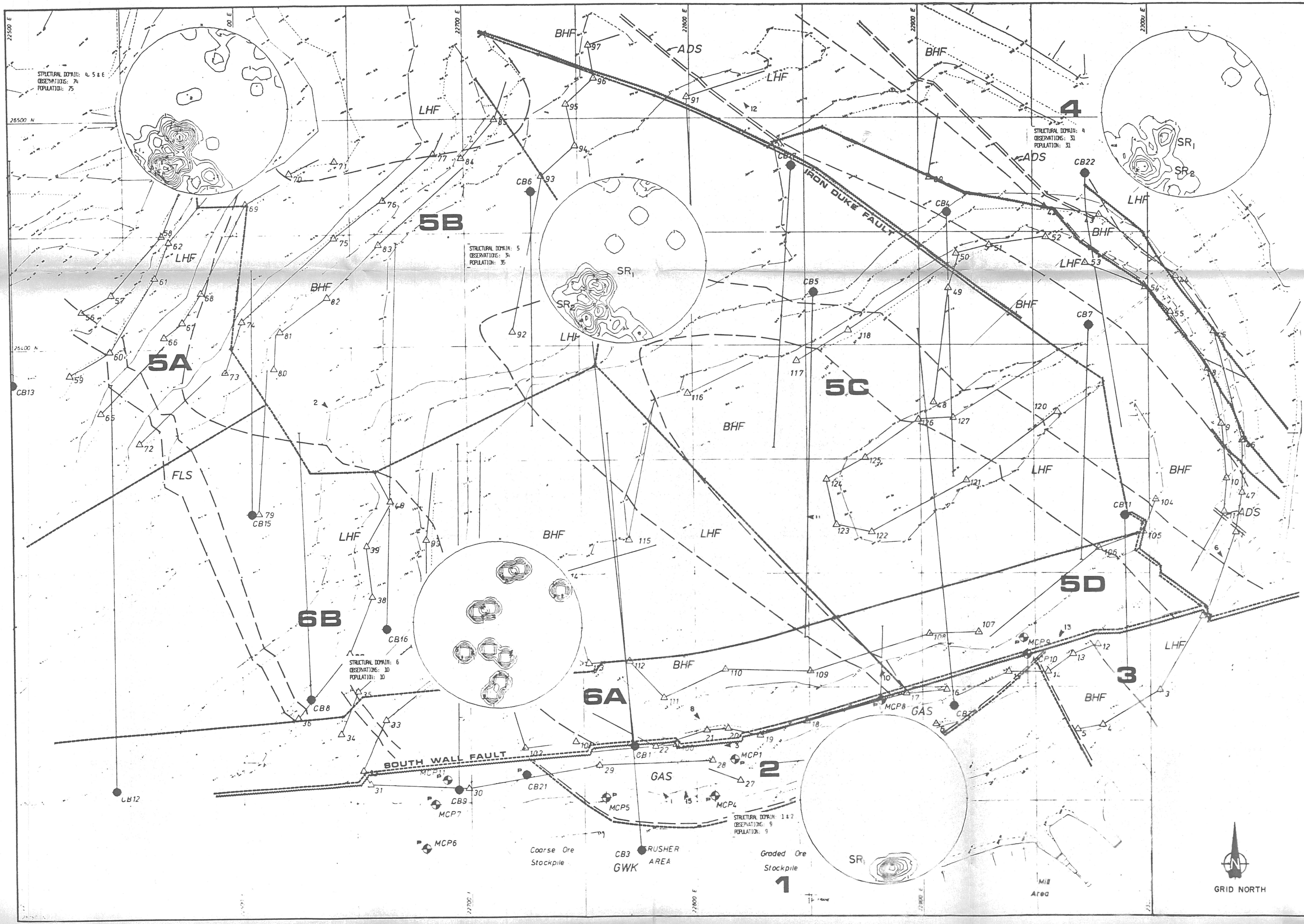
FIG. 6

DISTRIBUTION OF VEINS IN THE OPEN PIT



LEGEND

FLS	FELSITE and related coarse grained quartz – porphyry intrusive dykes
ADS	ANDESITE and related intermediate fine grained intrusive rocks which occur as small dykes
LHF	LAMINATED HORNFELS and related hard, grey, highly silicified, laminated rocks derived from micaceous siltstones
BHF	BANDED HORNFELS and related hard, highly silicified rocks with quartz banding, augens, and related features
GWK	SILICEOUS GREYWACKE and related moderately silicified argillaceous rocks with chert and quartz stringers
GAS	GREEN ARGILLACEOUS SCHIST and related non-siliceous, argillaceous rocks
BST	BLACK SLATE and related thinly laminated, well developed slates

SYMBOLS

Geological Contact
Fault
Structural Domain boundary
Structural Domain number — 6A
Crest and toe of bench with elevation
Detailed line mapping traverse: number and location — 47
Number and location of percussion drillhole — MCP7
Number and location of diamond drillhole — CB21
Drillhole with piezometers installed — P
Number, location, and direction of photographs — 6
Label for Joint Set on lower hemisphere projection — A

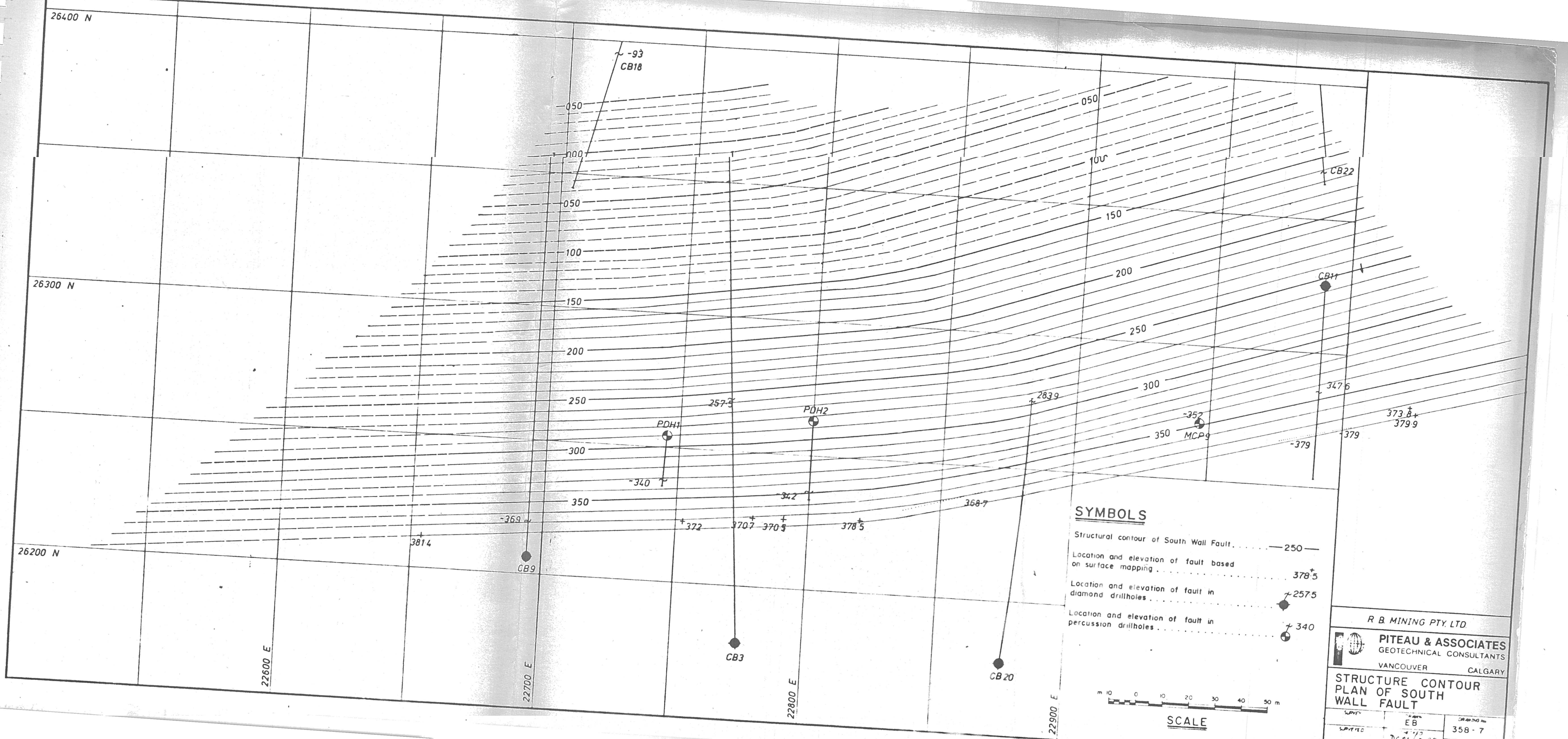
NOTE

Contour values on lower hemisphere equal area projections are expressed as percent of the total weight occurring in a 1.0 percent area of the plot.

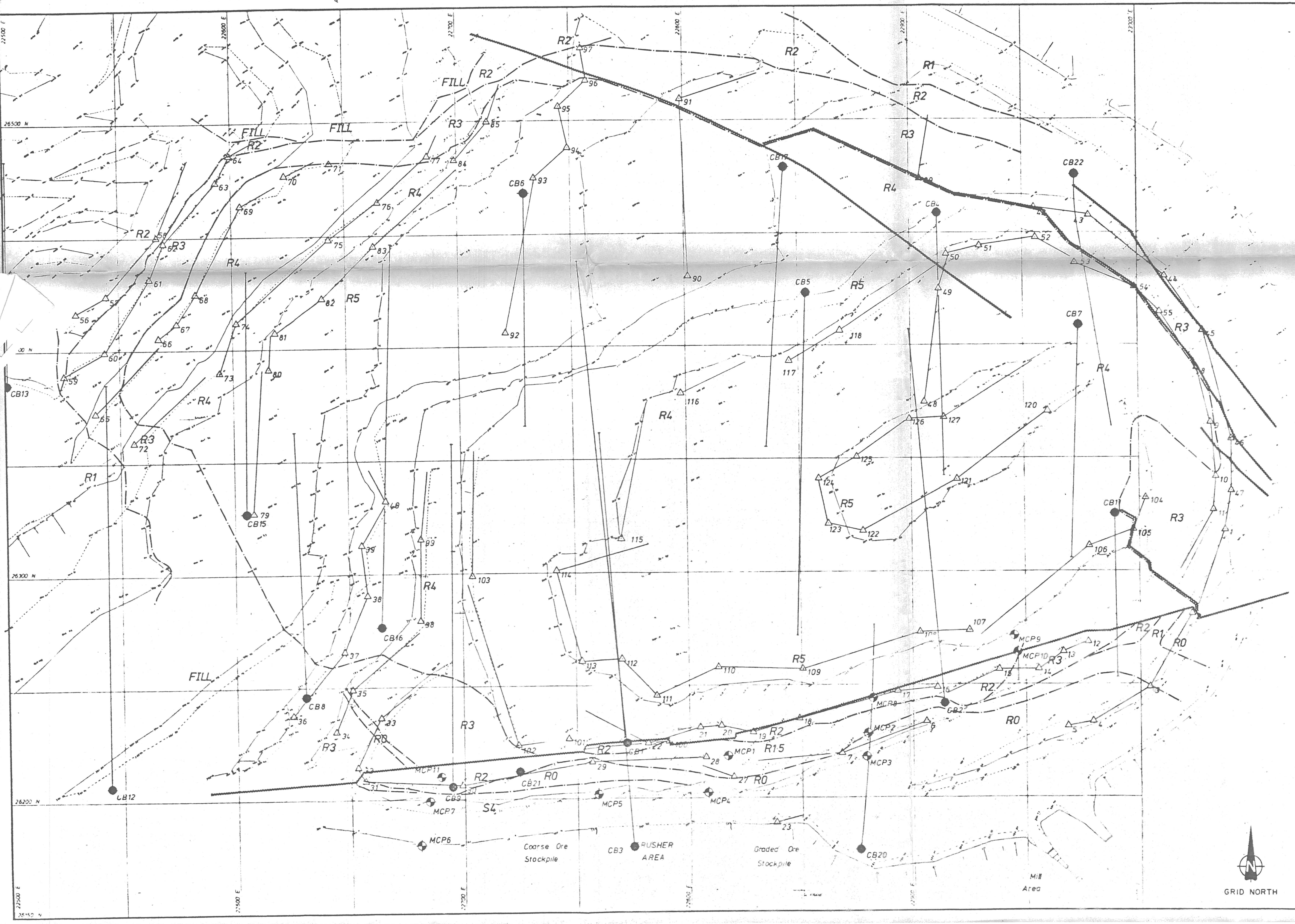
10 0 10 20 30 40 50 60 70 80 90 100 metres
SCALE

FIG. 7

R. B. MINING PTY. LTD. MT. CARBINE MINE GEOTECHNICAL STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY		
DISTRIBUTION OF FAULTS AND SHEARS IN THE OPEN PIT	BY: EB DCM	DATE: MAR 82	
	APPROVED:	JOB No: 81-358	



26400 N
26300 N
26200 N
22600 E
22700 E
22800 E
22900 E
CB18
CB22
CB11
CB9
CB3
CB20
PDH1
PDH2
MCP9
050
100
150
200
250
300
350
SYMBOLS
Structural contour of South Wall Fault . . . . . . —250—
Location and elevation of fault based on surface mapping 378.5
Location and elevation of fault in diamond drillholes 257.5
Location and elevation of fault in percussion drillholes 340
SCALE
R.B. MINING PTY. LTD.
PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER
CALGARY
STRUCTURE CONTOUR PLAN OF SOUTH WALL FAULT
EB
358 - 7



LEGEND

		ESTIMATED UNCONFINED COMPRESSIVE STRENGTH kg/cm^2	psi
S1	VERY SOFT: EASILY PENETRATED SEVERAL INCHES BY FIST	<0.25	<3.5
S2	SOFT: EASILY PENETRATED SEVERAL INCHES BY THUMB	0.25-0.5	3.5-7
S3	FIRM: CAN BE PENETRATED SEVERAL INCHES BY THUMB WITH MODERATE EFFORT	0.5-1.0	7-14
S4	STIFF: READILY INDENTED BY THUMB BUT PENETRATED ONLY WITH GREAT EFFORT	1.0-2.0	14-28
S5	VERY STIFF: READILY INDENTED BY THUMBNAIL	2.0-4.0	28-56
S6	HARD: INDENTED WITH DIFFICULTY BY THUMBNAIL	>4.0	>56
R0	EXTREMELY SOFT ROCK: INDENTED BY THUMBNAIL	2.0-7.0	28-100
R1	VERY SOFT ROCK: CRUMBLES UNDER FIRM BLOW WITH POINT OF GEOLOGICAL PICK, CAN BE PEELED BY POCKET KNIFE	7.0-70	100-1000
R2	SOFT ROCK: CAN BE PEELED BY POCKET KNIFE WITH DIFFICULTY, SHALLOW INDENTATIONS MADE BY FIRM BLOW OF GEOLOGIC PICK	70-280	1000-4000
R3	AVERAGE ROCK: CANNOT BE SCRAPED OR PEELED WITH POCKET KNIFE, SPECIMEN CAN BE FRACTURED WITH SINGLE FIRM BLOW OF HAMMER END OF GEOLOGICAL PICK	280-560	4000-8000
R4	HARD ROCK: SPECIMEN REQUIRES MORE THAN ONE BLOW WITH HAMMER END OF GEOLOGICAL PICK TO FRACTURE IT	560-1,120	8000-16,000
R5	VERY HARD ROCK: SPECIMEN REQUIRES MANY BLOWS OF HAMMER END OF GEOLOGICAL PICK TO FRACTURE IT	1,120-2240	16,000-32,000

SYMBOLS

Estimated boundary between rock hardness zones ....

Fault ....

Structural Domain boundary ....

Structural Domain number 6A

Crest and toe of bench with elevation ....

Detailed line mapping traverse: number and location 47

Number and location of percussion drillhole MCP7

Number and location of diamond drillhole CB21

Drillhole with piezometers installed .... P

Number, location, and direction of photographs 6

SCALE

10 0 10 20 30 40 50 60 70 80 90 100 metres

GRID NORTH

FIG. 9

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DISTRIBUTION OF ROCK HARDNESS

BY DCM EB | DATE MAR 82 | APPROVED | JOB NO. 81-358

JOB NUMBER

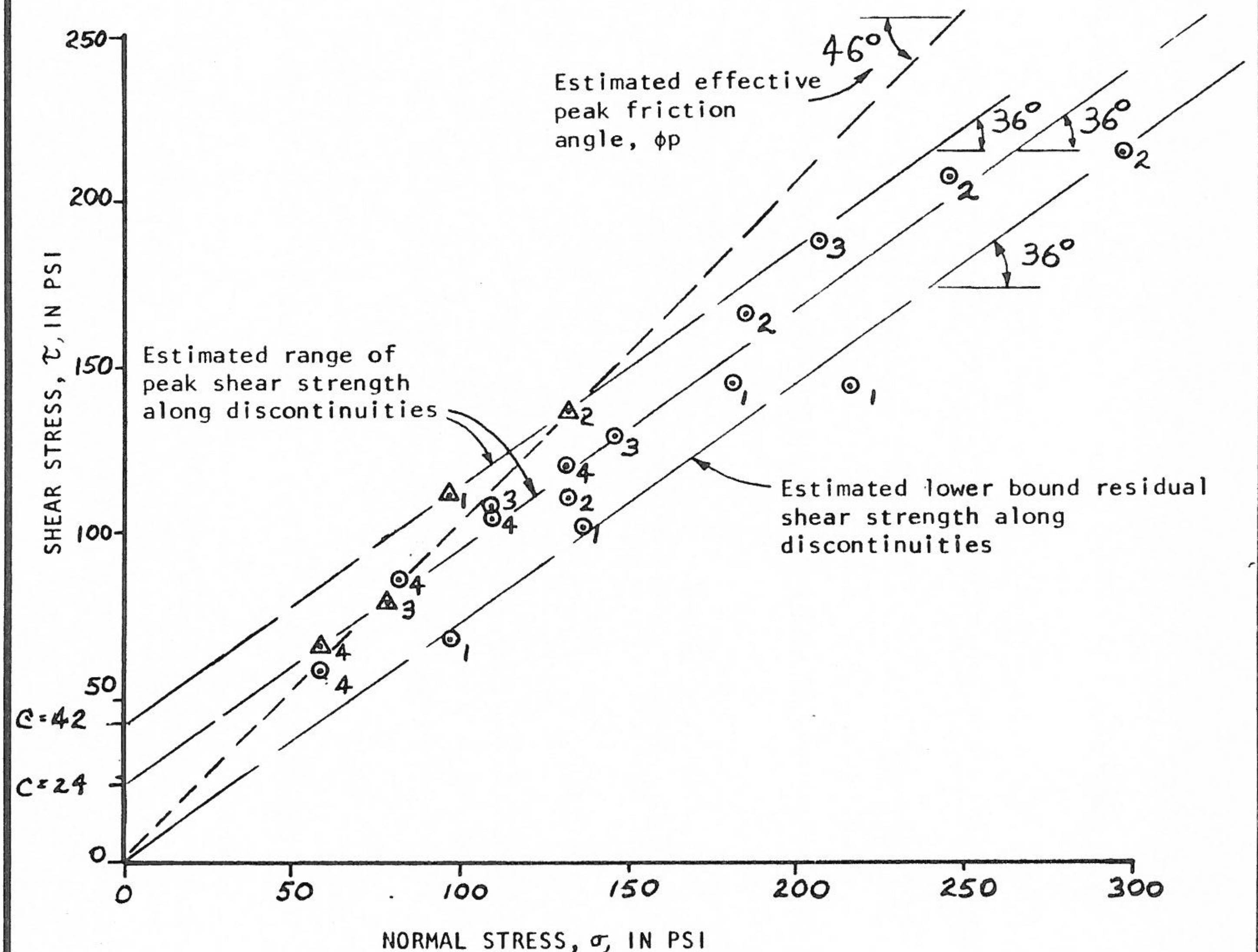
SAMPLE DESCRIPTION AND LOCATION

1 Natural joint: CB20 (64m)
2 Rough foliation joint: CB20 (104m)
3 Foliation joint: CB21 (38m)
4 Weathered joint: CB21 (4m)

SYMBOLS

Peak shear strength for initial test run △ 2

Residual shear strength for test run ⊙ 2



NOTE: Shear tests were carried out with a direct shear testing machine using standard testing techniques

FIG. 10

R.B. MINING PTY LIMITED
MT. CARBINE MINE
SLOPE STABILITY STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

SHEAR STRENGTH OF DISCONTINUITIES

BY:	DATE:
D.C.M.	MAR/82
APPROVED: D. Piteau	DWG:

JOB NUMBER 81-358

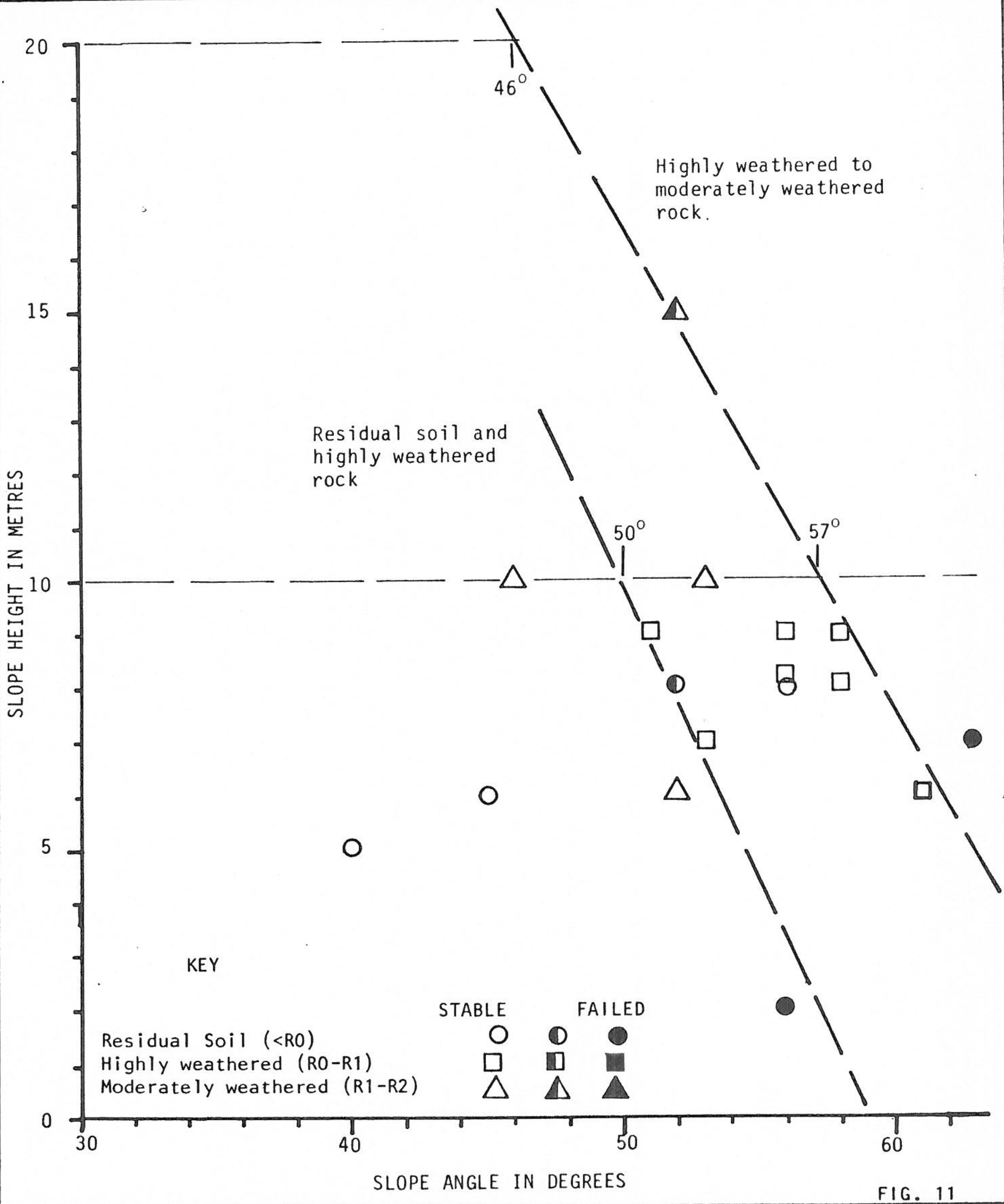


FIG. 11

R.B. MINING PTY LIMITED MT. CARBINE MINE SLOPE STABILITY STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY

DOCUMENTATION OF ROAD CUT SLOPES IN THE MT. CARBINE AREA	BY: DCM	DATE: MAR/82
	APPROVED:	DWG: 358-10

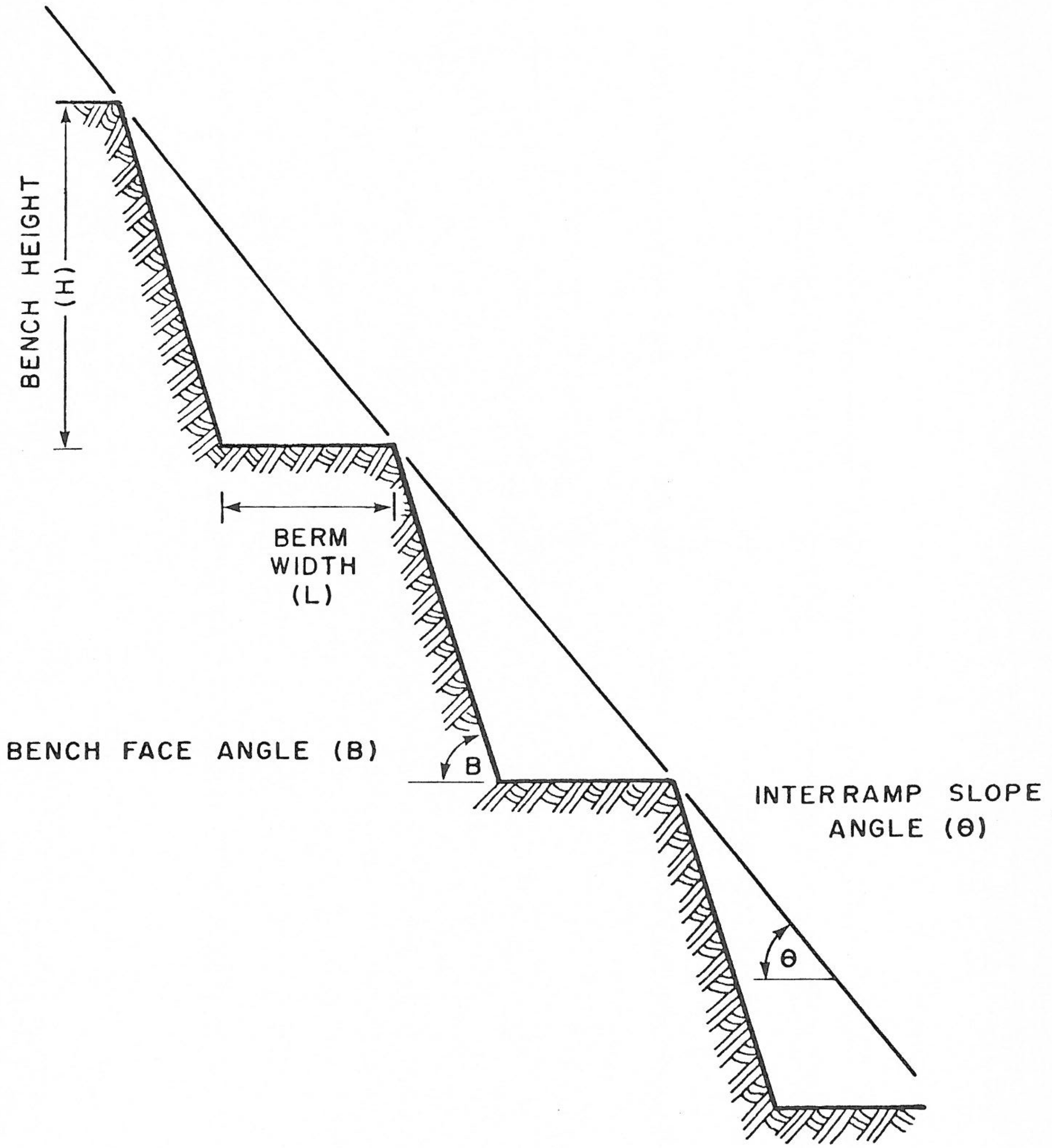


FIG. 12

R.B. MINING PTY LTD
MT. CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

SLOPE GEOMETRY PARAMETERS USED IN SLOPE DESIGN

BY: A.F.S. / V.G.	DATE: MAR/82
APPROVED: D. Piteau	DWG: 358-12

JOB NUMBER 81-358

GEOPLN - LOWER HEMISPHERE EQUAL AREA PROJECTION OF PLANES

PROJECT: MOUNT CARBINE
DATE: MARCH 82
STRUCTURAL DOMAIN: ALL PIT

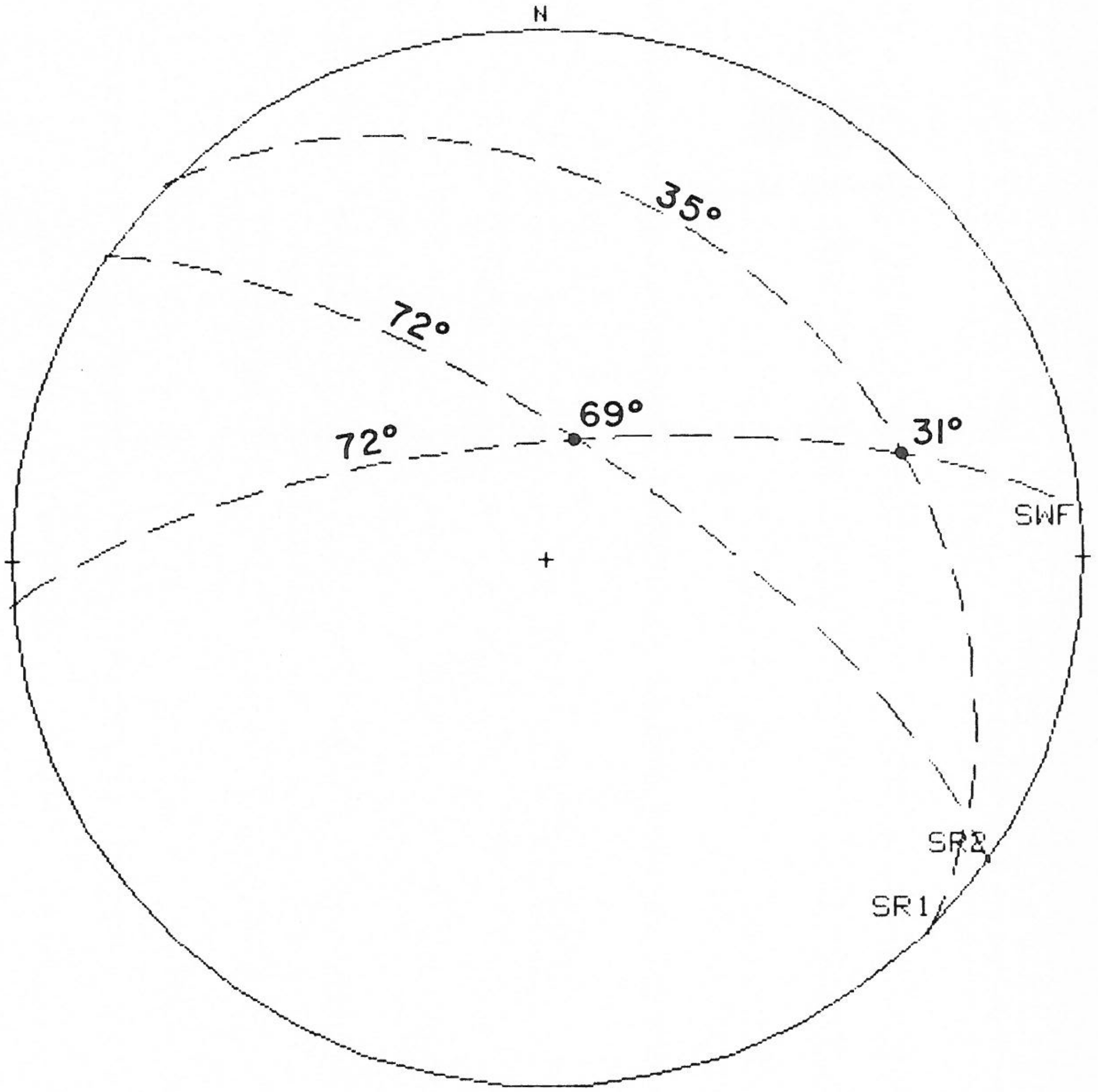


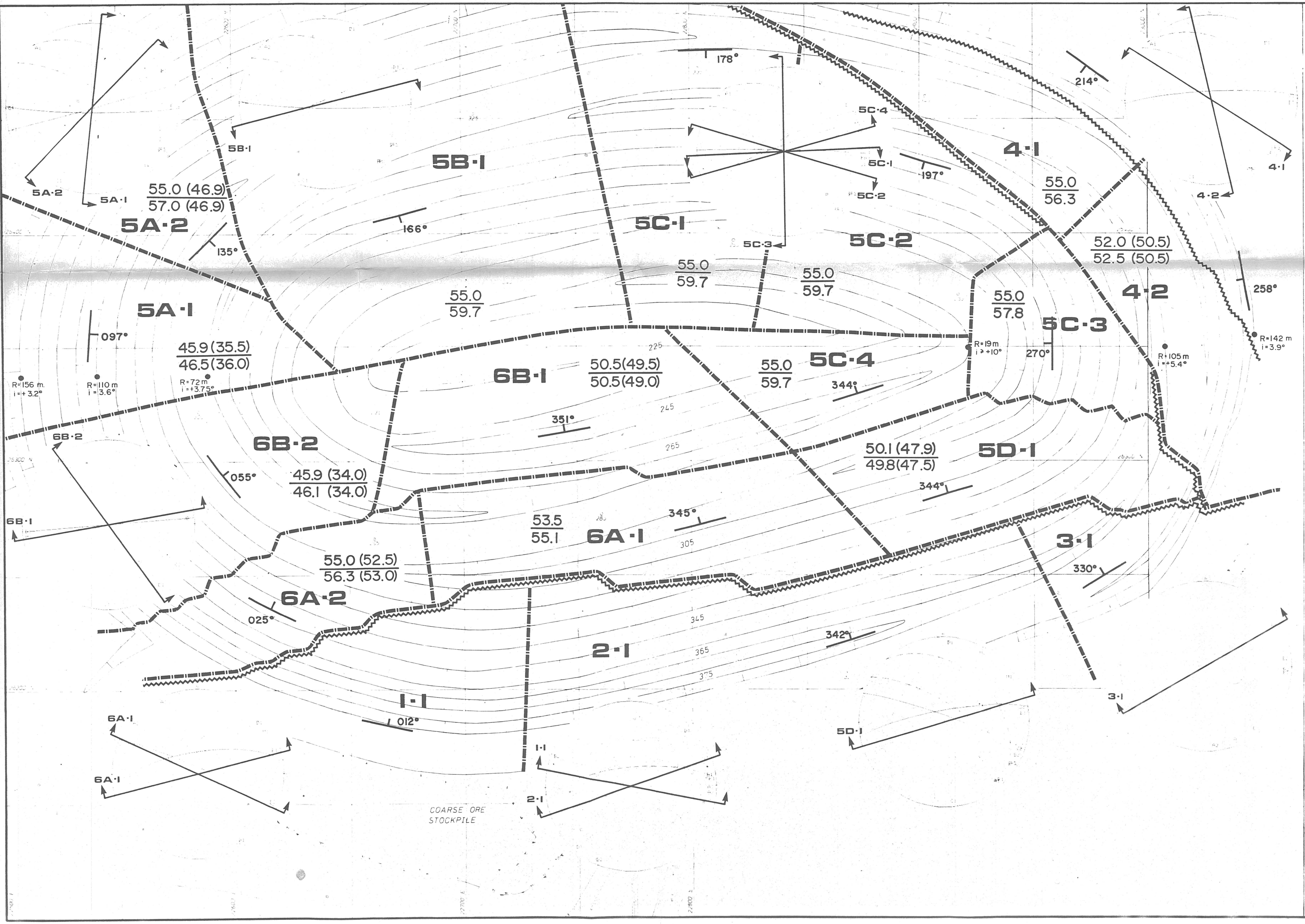
FIG. 13

R.B. MINING PTY LTD
MT. CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

LOWER HEMISPHERE EQUAL AREA PROJECTIONS OF PLANES REPRESENTING PEAK ORIENTATION OF FAULT AND SHEAR SETS IN THE OPEN PIT

BY: D.C.M.	DATE: MAR/82
APPROVED: D. Piteau	DWG: 358-B



SYMBOLS

Design Sector boundary	
Design Sector number	6B·1
Orientation and dip direction of final pit wall	175°
Lower hemisphere equal area projection of peak attitude of discontinuity set used in the stability analysis	
Approximate trend and dip direction of proposed final pit wall on equal area projection for Design Sector considered	6A·I
Optimum interramp slope angle for Design Sectors with and without adequate groundwater depressurization — Bench height = 20m	55.0 (46.9)
Bench height = 30m	57.0 (47.2)
Location of measurement of plan radius of curvature (R) and radius of curvature increment (i)	R = 105m i = +5.4°
Crest, toe, and elevation of benches on the proposed final pit (Plan 5A; July 1980)	265

NOTES

1. Design sector boundaries are based on structural domain boundaries and boundaries of straight slope segments
2. Results of kinematic statistical analyses were not used for design of slopes in Design Sectors 1-1, 2-1 and 3-1. Separate design of slopes and remedial measures for individual benches south of the South Wall Fault are discussed in Section 8.7.

0 10 20 30 40 50 60 70 80 90 100 metres

SCALE

GRID NORTH

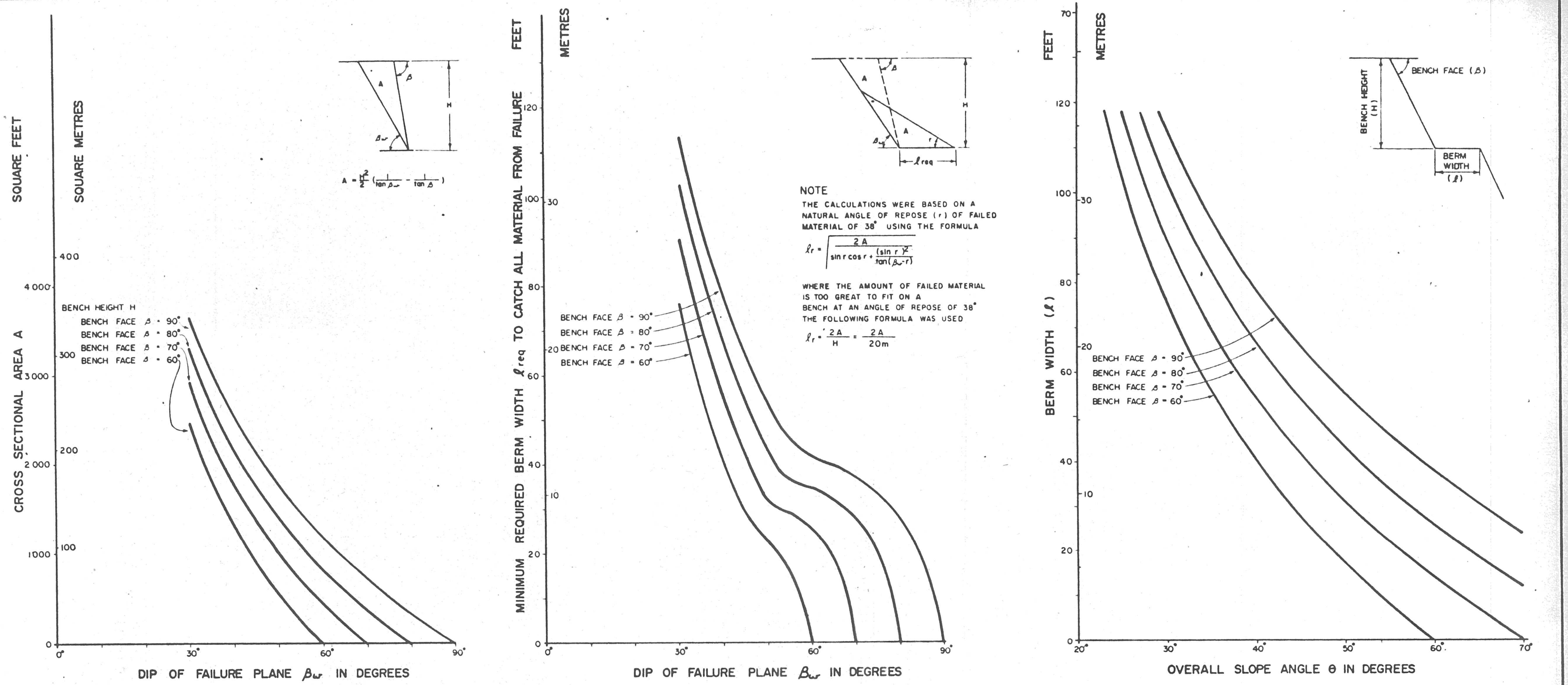
FIG. 14

R.B MINING PTY LIMITED
MT CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

DISTRIBUTION OF DESIGN SECTORS AND KINEMATICALLY POSSIBLE FAILURE MODES

BY EB DCM | DATE MAR 82 | APPROVED | DWG 358-14



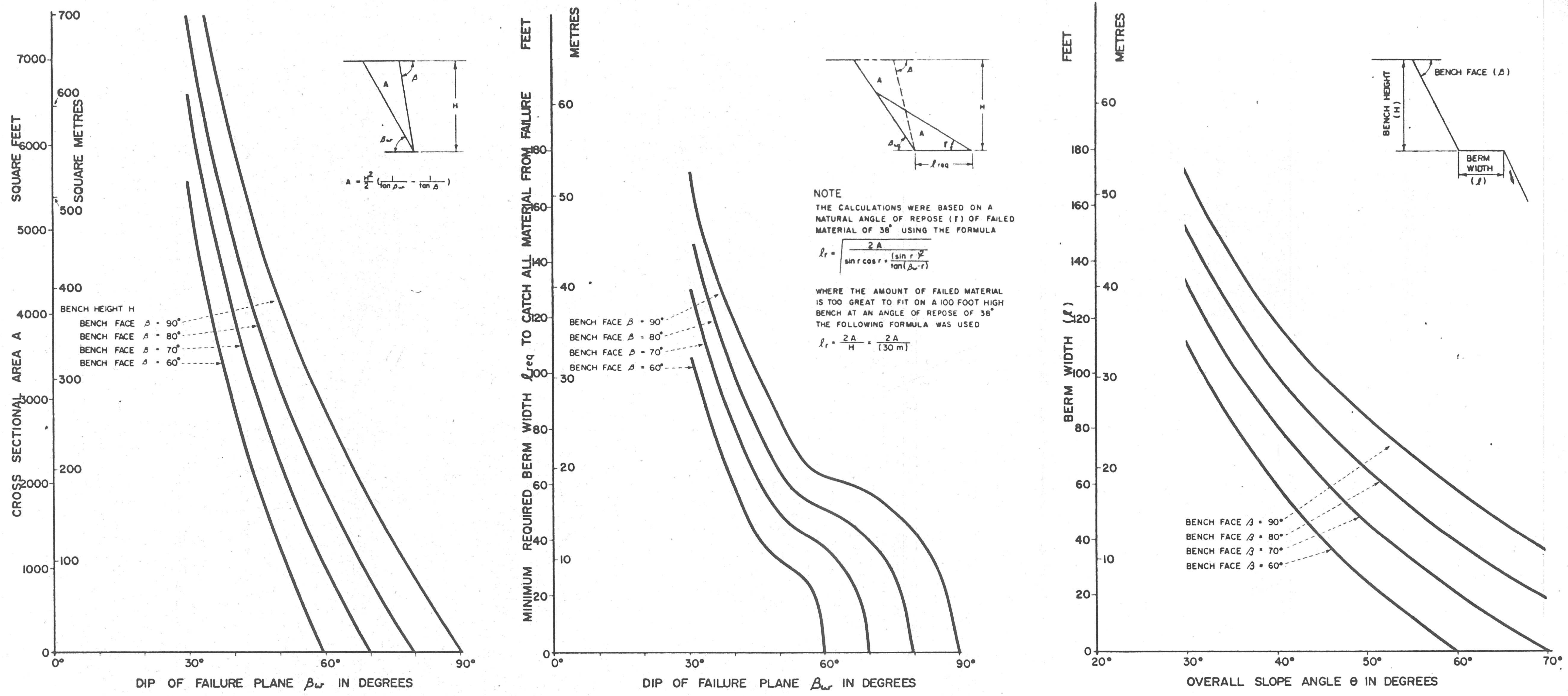
a) VARIATION OF CROSS SECTIONAL AREA WITH DIP OF FAILURE PLANE FOR FAILURES INVOLVING A SINGLE 20m (65.6 ft.) HIGH BENCH

b) VARIATION OF BERM WIDTH REQUIRED TO CATCH FAILED MATERIAL WITH DIP OF FAILURE PLANE FOR FAILURES INVOLVING A SINGLE 20m (65.6 ft.) HIGH BENCH.

c) GEOMETRIC RELATIONSHIP OF OVERALL SLOPE ANGLE, BERM WIDTH AND BENCH FACE FOR SINGLE 20m (65.6 ft.) HIGH BENCHES

FIG. 15

R.B MINING PTY LIMITED MT. CARBINE MINE GEOTECHNICAL STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY	
INTERRELATIONSHIP OF SLOPE GEOMETRY AND FAILURE GEOMETRY FOR 20m (65.6 ft.) HIGH BENCHES	BY BST APP	DATE 04-82 DWG. 358-15



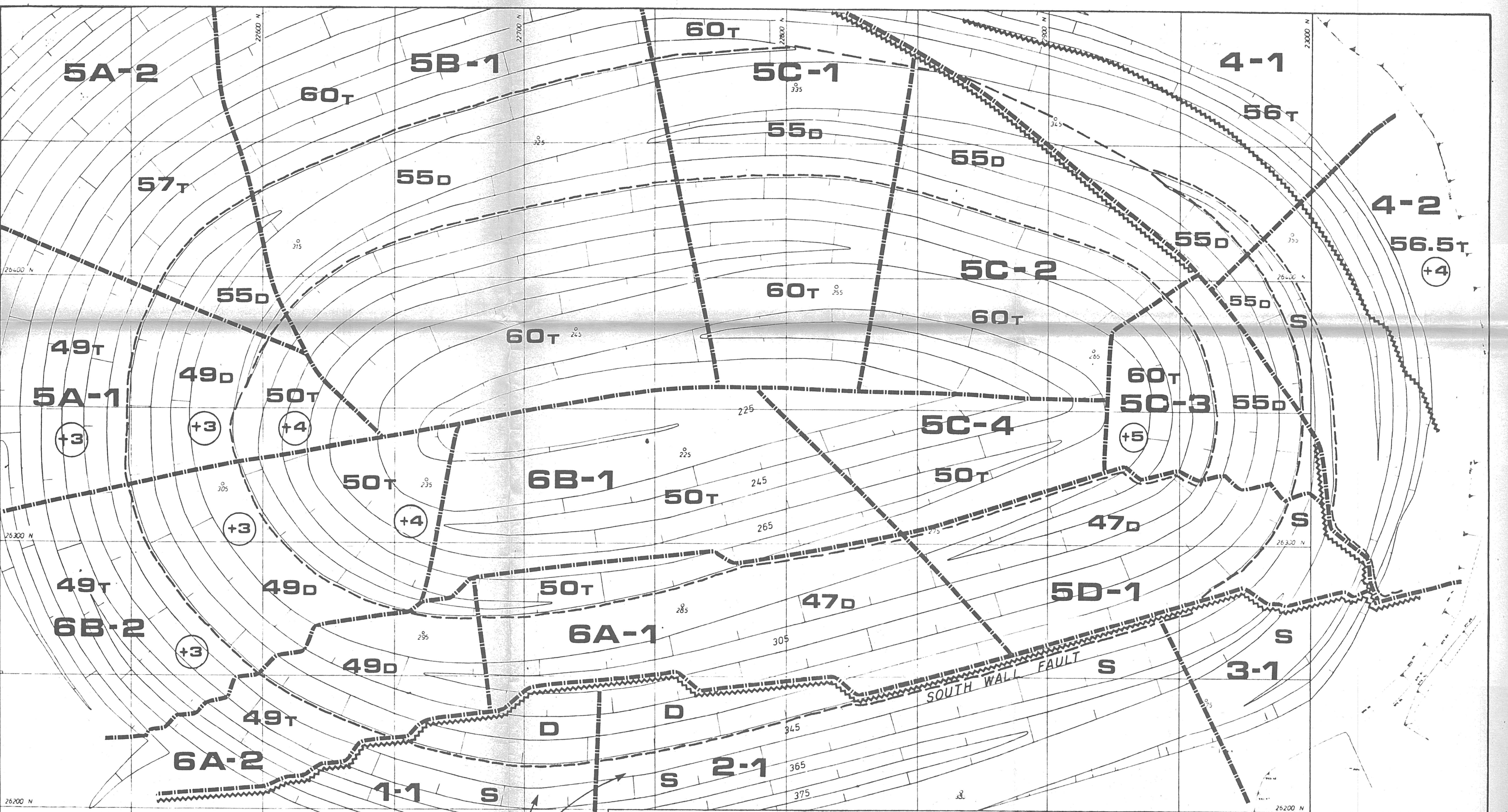
a) VARIATION OF CROSS SECTIONAL AREA WITH DIP OF FAILURE PLANE FOR FAILURES INVOLVING A SINGLE 30 METRE HIGH BENCH

b) VARIATION OF BERM WIDTH REQUIRED TO CATCH FAILED MATERIAL WITH DIP OF FAILURE PLANE FOR FAILURES INVOLVING A SINGLE 30 METRE HIGH BENCH

c) GEOMETRIC RELATIONSHIP OF OVERALL SLOPE ANGLE, BERM WIDTH AND BENCH FACE FOR 30 METRE HIGH BENCHES

FIG. 16

R B MINING PTY LIMITED MT CARBINE MINE GEOTECHNICAL STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY	
INTERRELATIONSHIP OF SLOPE GEOMETRY AND FAILURE GEOMETRY FOR 30 METRE HIGH BENCHES	BY BST CHK'D DEM APP D.P.	DATE 04·82 DWG 358·16



SYMBOLS

Boundary of Design Sector

Design Sector Number 6B-1

Boundary of area with similar recommended bench height

Recommended interramp slope angle and bench height for slopes with adequate groundwater depressurization (D= 20m high benches ; T= 30m high benches.) 60T

Area where single 10m high benches are recommended for access to the South Wall .... S

Recommended increase in slope angle included due to effects of plan radius of curvature .... +5

NOTE

Slope angles are recommended for interramp slopes north of the South Wall Fault. The recommended slope geometry and remedial measures for each Design Sector are summarized in Table IX.

10 0 10 20 30 40 50 60 70 80 90 100 metres

SCALE

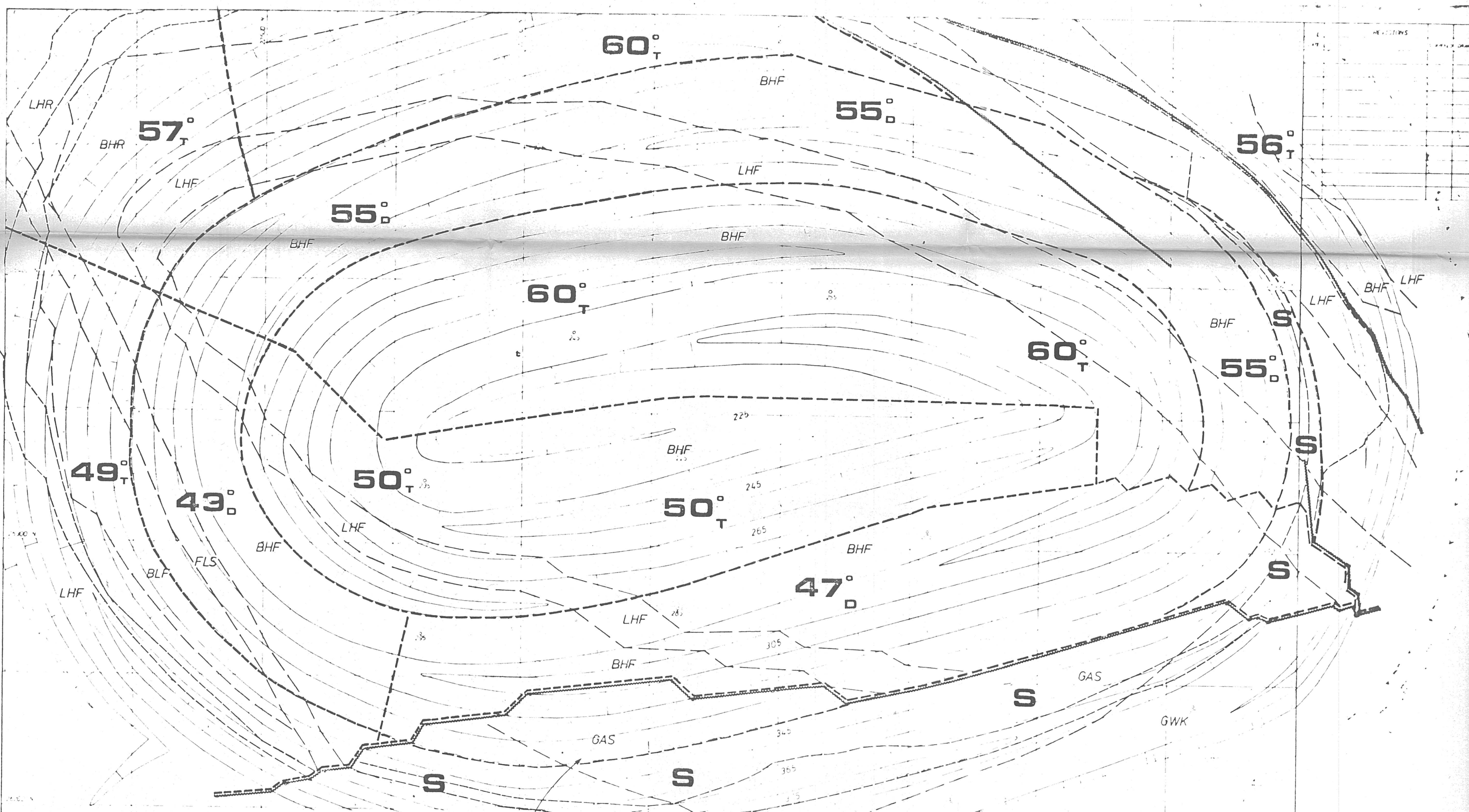
FIG. 17

R.B. MINING PTY LIMITED
MT CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

RECOMMENDED INTERRAMP SLOPE DESIGN IN INDIVIDUAL DESIGN SECTORS

BY	DATE
DCM	MAR 82
APPROVED	DWG
O. Piteau	358-18



Area south of the South Wall Fault has separate design and remedial measures for each bench as summarized in Fig. 19

NOTES

1. The slope angles recommended are for interramp slopes. Recommended slope geometry and remedial measures for each design sector are summarized in Table IX.
2. Slope angles recommended are for dry slopes.
3. Interramp slope may be steepened on a trial basis in the north wall provided adequate access and safety is maintained and slopes are monitored.

SYMBOLS

Boundary of area with equal recommended interramp slope angle — — —

Recommended maximum interramp slope angle between haulroads attainable with adequate groundwater depressurization (D = 20m high double benches, T = 30m high triple benches) 47°D

Area where single 10m high benches are recommended for access to the south wall S

SCALE

10 0 10 20 30 40 50 metres

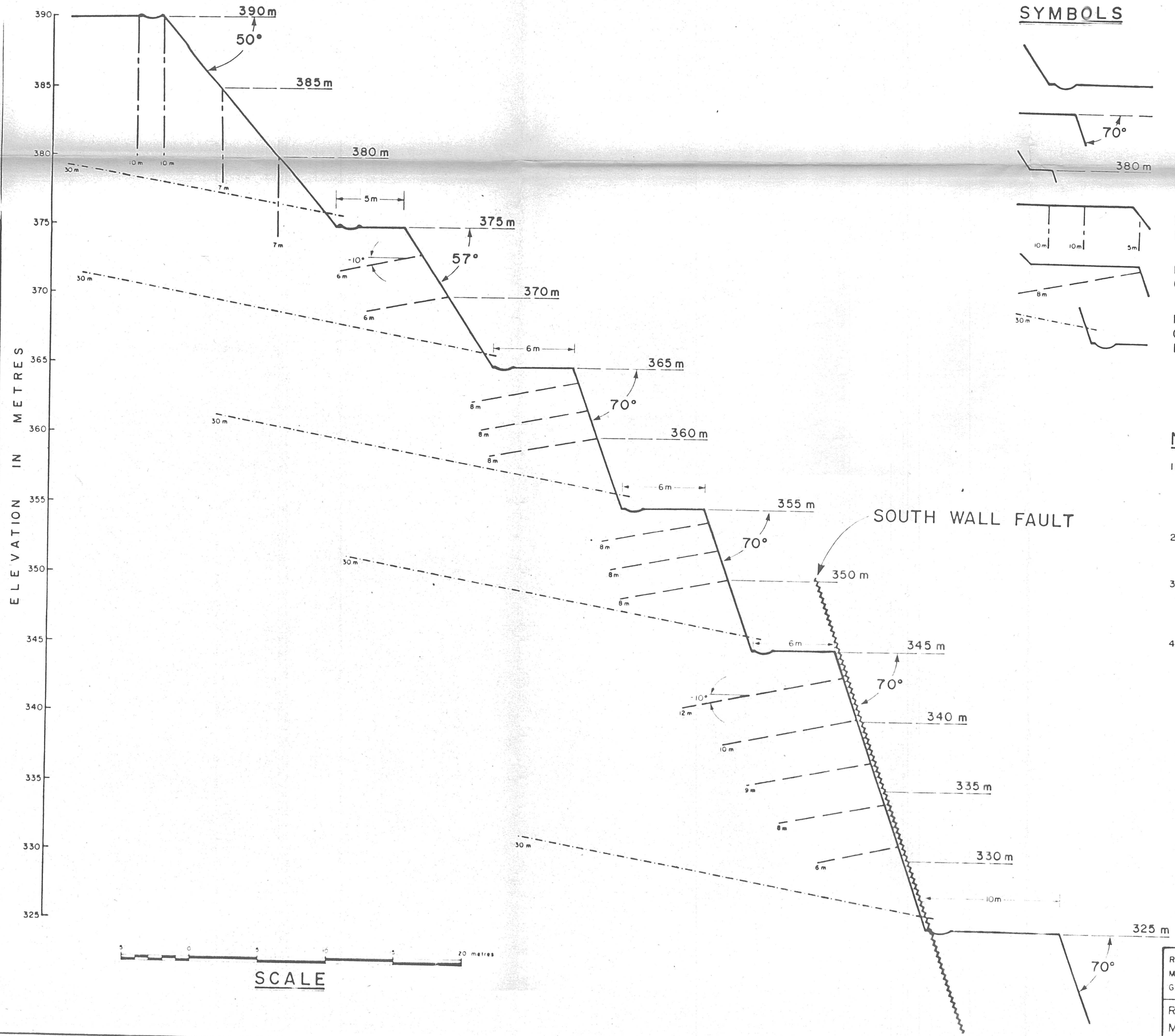
PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

R B MINING PTY LTD

GENERAL DISTRIBUTION OF RECOMMENDED INTERRAMP SLOPE ANGLES FOR THE PROPOSED FINAL PIT PLAN

MARCH, 82

FIG. 18



SYMBOLS

RECOMMENDED SLOPE GEOMETRY SOUTH OF SOUTH WALL FAULT

RECOMMENDED BENCH FACE ANGLE

RECOMMENDED LEVEL OF EACH EXCAVATION LIFT TO BE USED FOR INSTALLING REMEDIAL MEASURES

RECOMMENDED LOCATION AND LENGTH IN METRES OF FULLY GROUTED VERTICAL DOWELS

RECOMMENDED LOCATION AND LENGTH IN METRES OF FULLY GROUTED INCLINED CABLE DOWELS

RECOMMENDED LOCATION AND LENGTH IN METRES OF SUBHORIZONTAL DRAINHOLES AND DRAINAGE DITCH

NOTES

1. SLOPE DESIGN IS FOR AREA SOUTH OF SOUTH WALL FAULT ON SECTION 22,700 E. RECOMMENDED DESIGN SHOULD BE APPLIED SOUTH OF FAULT ONLY. SLOPE DESIGN FOR AREA NORTH OF FAULT IS GIVEN IN FIG. 18.
2. RECOMMENDED DESIGN ASSUMED SLOPES ARE EXCAVATED BY RIPPING OR OTHER NON EXPLOSIVE EXCAVATION METHODS TO MAINTAIN COHESION IN THE ROCK MASS.
3. SLOPES SHOULD BE CAREFULLY MAPPED AND MONITORED DURING EXCAVATION. EXTENT OF REMEDIAL MEASURES MAY BE VARIED DEPENDING ON ACTUAL CONDITIONS ENCOUNTERED DURING EXCAVATION.
4. SPACING OF ALL DOWELS IS ASSUMED TO BE 3m TO INSURE ADEQUATE SUPPORT AND ACCEPTABLE DISTRIBUTION OF SUPPORT IN THE SLOPE. INCLINED DOWELS ABOVE 345m ELEVATION ARE REQUIRED TO HAVE A MINIMUM DESIGN LOAD OF 200 kN (20 TONNE). INCLINED DOWELS BELOW 345m ELEVATION ARE REQUIRED TO HAVE A MINIMUM DESIGN LOAD OF 500 kN (50 TONNE).

FIG. 19

R B MINING PTY LTD MT CARBINE MINE GEOTECHNICAL STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY	
RECOMMENDED SLOPE DESIGN AND REMEDIAL MEASURES FOR SLOPES SOUTH OF THE SOUTH WALL FAULT	DCM E B	FEB 82 81-358

TABLE I

SUMMARY OF CHARACTERISTICS OF WEATHERED ROCKS

SOUTH OF THE SOUTH WALL FAULT

ASSUMED ELEVATION FOR DESIGN (m)	DESCRIPTION	DEGREE OF WEATHERING*	AVERAGE ROCK HARDNESS*	ESTIMATED AVERAGE UNCONFINED COMPRESSIVE STREGHTH (MPa)	AVERAGE DEPTH (m)
above 375	Fill and extremely weathered residual soil	A	S4 to R0	0.19 to 0.69	10.5
365 to 375	Highly weathered schists and greywackes	B	R1	5.6	7.0
355 to 365	Moderately weathered schists and greywackes	C	R2	6.9	9.5
below 355	Slightly weathered to unweathered schists and greywackes	D and E	$>$R3	27.5	-

* Note: A description of Degree of Weathering and Rock Hardness is given in Figs. 2 and 9.

TABLE III

ORIENTATION OF FAULT AND SHEAR SETS

IN THE OPEN PIT

STRUCTURAL DOMAIN	TOTAL POP.	SHEAR SET SR1			SHEAR SET SR2		
		DIP DIRECTION	DIP	%	DIP DIRECTION	DIP	%
1, 2 & 3	9	355	72	>35			
4	31	043	33	11	038	70	19
5	35	047	37	14	032	73	13
6	10	-	-	-	-	-	-
4, 5 & 6	75	045	35	11	035	72	12

NOTES: 1. Orientation is given in terms of dip direction and dip of the peak or average orientation of the discontinuity set.

2. Percent refers to the percent concentration with one percent area of the lower hemisphere for the peak or average orientation of the population.

TABLE IV

SUMMARY OF UNCONFINED COMPRESSIVE STRENGTH[1]

ROCK TYPE[2]	ESTIMATED AVERAGE HARDNESS	DRILLHOLE	DEPTH (m)	UNIAXIAL COMPRESSION TESTS[3]: UCS (MPa)	UNIAXIAL COMPRESSION TESTS[3]: AVERAGE UCS (MPa)	POINT LOAD INDEX TESTS: UCS PARALLEL SCHISTOSITY (MPa)	POINT LOAD INDEX TESTS: UCS NORMAL TO SCHISTOSITY (MPa)	
Fill and residual soil (A)	S4 to R0	-	- -	-	-	-		
Highly weathered schist and greywacke (B)	R1	CB20	24.0	7.04	5.6	-	-	
			26.1	4.60				
			28.5	4.53				
			28.7	6.01				
		CB3	16.8	-	-	5.76	-	*
Moderately weathered schist and greywacke (C)	R2	CB20	-	6.48	10.44	-	-	
		CB21	4.0	14.4				
Unweathered greywacke (D to E)	R3 to R4	CB20	51.0 to 56.0	-	-	-	93.4	
		CB20	77.0 to 81.0	-	-	29.0	59.8	
		CB3	54.3 to 60.4	-	-	-	54.0	*
Unweathered schistose greywacke (D to E)	R3 to R4	CB20	122.0 to 127.0	-	-	28.1	73.4	
		CB20	131.0 to 137.0	-	-	3.7	91.2	*
Unweathered hornfels (D to E)	R4 to R5	CB3	71.0 to 77.13	-	-	-	62.4	

NOTES:

1. Laboratory tests were carried out in November, 1981 at James Cook University, Townsville, under the supervision of Dr. H. Bock. Test results indicated by an asterisk (*) were carried out by Piteau & Associates.
2. Degree of Weathering is classified by letters A to E as summarized in Fig. 2.
3. Unconfined compressive strength is abbreviated as UCS

TABLE V

SUMMARY OF BACK ANALYSIS RESULTS FOR SLOPES

IN WEATHERED ROCK AND RESIDUAL SOIL

SOUTH OF THE SOUTH WALL FAULT

SLOPE HEIGHT (m)	DRY SLOPE	MAXIMUM SLOPE ANGLE FOR F = 1.2	
		MODERATE GROUNDWATER	HIGH GROUNDWATER
10	60	58	47
15	48	40	31
20	41	29	25

NOTES:

1. Analyses were carried out using design charts.

2. Analyses results are based on a factor angle of 24° and a mass cohesion C' of 40 kPa.

3. It is anticipated that slopes 20m in height would contain significantly stronger rock in the toe area. Hence, somewhat steeper slope angles could be appropriate for 20m high slopes.

TABLE VI

SUMMARY OF
BACK ANALYSIS RESULTS
FOR PLANE FAILURE IN ARGILLACEOUS SCHIST
ON THE SOUTH WALL

---

1. SLOPE GEOMETRY AND STRENGTH PARAMETERS

- Slope Height H = 10m
- Slope Angle B = 80^o
- Friction Angle = 36^o
- Failure Surface Dip, $_p$ = 47^o

2. CALCULATION OF REQUIRED COHESION FOR SLOPE ANGLE OF 80^o

Factor of Safety	Cohesion Required, kPa: Without Tension Crack	Cohesion Required, kPa: With Tension Crack
1.0	16	20
1.2	26	33
1.5	42	51

3. CALCULATION OF ANCHOR FORCES REQUIRED FOR VARIOUS SLOPE ANGLES ON A 10m HIGH BENCH FOR COHESION OF 10 kPa

Bench Face Angle (o)	Total Anchor Force Required kN/m	Equivalent Number of Rows of Grouted Dowels: 200 kN (20 Tonne) Dowels at 3m Spacing	Equivalent Number of Rows of Grouted Dowels: 500 kN (50 Tonne) Dowels at 5m Spacing
50	0	-	-
60	81	2	1
70	186	3	2
80	262	4	3

NOTES: 1. Calculations are for horizontal dowels. Dowels inclined at -10^o would require higher anchor forces.

2. Sufficient number of rows of dowels are required to obtain acceptable distribution of support in the slope.

TABLE VII

SUMMARY OF GROUTED DOWELS AND PULLOUT TEST RESULTS

DOWEL NO.	LOCATION			ORIENTATION		LENGTH (m)	GROUTED LENGTH (m)	ESTIMATED ROCK HARDNESS[1]	LOCATION IN FINAL WALL	JACK FORCE AT FAILURE (TONNES)[2]
	NORTHING (m)	EASTING (m)	ELEVATION (m)	DIP DIRECTION (°)	INCLINATION (°)					
1	26226.6	22887.1	380.8	160	-30	10	10	R0	1st Bench	67
2	26202.8	22804.3	389.7	183	-30	10	10	S5	Top	48*
3	26212.8	22862.3	384.8	172	-30	20	20	R0	Top	64
4	26216.6	22856.2	381.2	171	-30	20	20	1.5	1st Bench	70
5	26212.9	22837.2	381.5	171	-30	10	10	R1	1st Bench	77
6	26215.3	22807.8	380.5	189	-30	10	10	R1	1st Bench	71
7	26228.4	22958.4	379.5	159	-30	20	10	R0	1st Bench	76
8	26241.1	22874.2	371.0	174	-30	10	10	R2	2nd Bench	71
9	26246.1	22899.1	370.7	181	-30	20	20	R2	2nd Bench	76
10	26255.9	22954.6	370.9	173	-30	10	10	R3	2nd Bench	78

NOTES:
1. An explanation of rock hardness provided on the legend Fig. 2.
2. In each test failure was due to cable failure and not ground failure.
3. Anchor did not fail: Jack Failure prevented completion of test on this anchor.

TABLE VIII

SUMMARY OF HYDROGEOLOGICAL INFORMATION

DRILLHOLE NUMBER[1]	LOCATION	COLLAR ELEVATION (m)	INCLINATION (o)	INSTALLATION[2]	ROCK TYPE[3]	DEGREE OF WEATHERING	POCKET LENGTH (m)	TIP ELEVATION (m)	ELEVATION OF RECORDED STATIC WATER LEVELS (m): LOW[4] (DRY SEASON)	HIGH[5] (WET SEASON)	HYDRAULIC CONDUCTIVITY BASED ON FALLING HEAD TESTS (m/sec)	COMMENTS
MCP1	26218.7N 22817.5E	378.8	90	P1	GAS	E	3.0	346.8	361.34	365.85	-	Seals may be inadequate.
				P2	GAS	E	2.0	357.3	361.37	365.86	1.0×10^{-7}	Seals may be inadequate.
				S1	GAS	C	-	367.3	dry	dry	-	
MCP2	26228.5N 22880.0E	379.3	90	P1	GAS	E	3.0	347.3	359.95	364.94	5.5×10^{-10}	
				P2	GAS	E	2.0	358.3	360.73	365.01	3.4×10^{-8}	
				S1	GAS	C	-	368.3	dry	dry	-	
MCP3	26218.3N 22879.3E	383.4	90	P1	GWK	E	3.0	349.4	360.07	364.87	3.8×10^{-7}	Seals may be inadequate.
				P2	GWK	E	3.0	355.4	360.04	364.69	1.8×10^{-7}	Seals may be inadequate.
				S1	GWK	B & C	-	371.9	dry	dry	-	
MCP4	26202.7N 22808.0E	389.6	90	P1	GWK	E	4.6	352.9	361.09	364.73	-	Seals may be inadequate
				S1	GAS	C	-	370.1	dry	dry	-	
MCP5	26202.4N 22759.8E	389.3	90	P1	GWK	E	3.0	355.8	360.60	363.67	1.6×10^{-8}	
MCP6	26180.3N 22680.1E	385.3	90	P1	GWK	D & E	3.0	348.3	359.92	361.73	2.6×10^{-7}	Seals may be inadequate.
MCP7	26199.3N 22684.2E	385.7	90	P1	GWK	E	3.0	350.9	359.77	361.98	4.9×10^{-8}	
MCP8	26244.7N 22882.4E	368.9	90	P1	GAS	E	2.0	347.9	360.22	364.98	3.0×10^{-7}	Test results may have been affected by caving.
MCP9	26271.4N 22945.6E	369.2	90	P1	GWK	E	3.1	334.5	359.49	364.58	1.0×10^{-8}	
				P2	GAS	E	2.0	345.6	354.75	362.05	5.9×10^{-8}	
				S1	LHF	E	-	354.2	dry	358.17	-	
MCP11	26210.3N 22689.6E	378.8	90	P1	GWK	E	3.0	343.8	359.58	362.62	2.3×10^{-9}	
				P2	GWK	C & D	2.2	357.1	360.95	363.96	4.1×10^{-8}	
CB20	26176.3N 22875.9E	382.8	45	P1	Fault Gouge	E	7.0	282.6	348.93	346.74	4.1×10^{-10}	Test results may have been affected by drilling mud.
				P2	GWK	B & D	6.0	352.1	358.65	360.43	4.8×10^{-9}	Test results may have been affected by drilling mud.
				P3	GWK	A	6.0	366.5	dry	dry	-	
CB21	26210.3N 22689.6E	378.8	90	P1	GWK	E	4.0	310.8	360.05	363.63	7.1×10^{-10}	Test results may have been affected by drilling mud.
				P2	GWK	D & E	4.6	356.0	359.75	362.02	2.7×10^{-10}	Test results may have been affected by drilling mud.

NOTES.
1. MCP1 to MCP11 refers to 165mm diameter percussion drillholes and CB20 and CB21 refers to 76mm to 96mm diameter diamond drillholes.
2. Installations are given as type and number where P = sealed piezometer and S = open standpipe.
3. A description of the symbols for the various rock types is given in Fig. 1. Explanation of weathering categories is given in Fig. 2.
4. Low static water levels listed were recorded on November 6, 1981 for all drillholes, except MCP4, MCP5, MCP6, MCP7, MCP11 and CB20, which were recorded on December 3, 1981.
5. High static water levels listed were recorded on February 9, 1982 for all drillholes

TABLE X

AVERAGE BREAKBACK OF BENCH CRESTS IN HORNFELS ROCK

AREA OF THE OPEN PIT	STRUCTURAL DOMAIN	AVERAGE BREAKBACK AT CREST OF 10m HIGH BENCH (m)	EQUIVALENT BREAKBACK ANGLE (°)	COMMENTS
North Wall	5B, 5C	3.42	71	
West Wall	5A, 6B	4.50	66	
South Wall (north of South Wall Fault)	5D, 6A	4.88	64	
East Wall	4	2.45	76	Slopes in this area were excavated by controlled blasting techniques

NOTE: The average breakback was calculated from measurement of distance between bench toes and crests for selected locations on the current mining plan (October 1981)

STRUCTURAL DOMAIN	FOLIATION					FOLIATION JOINTS JOINT SET A					JOINTS																	VEINS				
												JOINT SET B				JOINT SET C				JOINT SET D				MISCELLANEOUS JOINT SETS								
	POPULATION	SET	DIP DIRECTION	DIP	%	POPULATION	SET	DIP DIRECTION	DIP	%	POPULATION	SET	DIP DIRECTION	DIP	%	SET	DIP DIRECTION	DIP	%	SET	DIP DIRECTION	DIP	%	SET	DIP DIRECTION	DIP	%	POPULATION	SET	DIP DIRECTION	DIP	%
1 & 2	81	FN1 FN2 FN3	010 002 342	85 63 44	19 15 6	15	A1 A2	001 005	62 45	>35 10	215	B1 B2	352 203	74 88	4 7	–	–	–		D1' D2' D3'	316 114 131	59 51 67	7 5 4	G1	340	48	3	-	-	-	-	-
3	43	FN1 FN2	067 083	67 77	21 18	43	A1 A2	067 083	67 77	21 18	88	B1 B2 B3 B4	004 354 002 328	78 74 52 45	10 9 10 6	–	–	–		–	–	–	–	E1	051	76	5	-	-	-	-	-
4	138	FN1 FN2	228 056	72 75	16 12	63	A1 A2 A3	225 060 062	67 69 46	16 16 17	570	B1	003	80	7	C1	105	30	7	D1	315	80	6	-	-	-	-	73	V1	358	82	35
5A	111	FN1 FN2	230 221	71 21	17 6	82	A1	229	66	33	400	B1 B2	333 359	86 83	7 6	C1	115	19	6	D1	286	80	6	G1	000	50	2	48	V1 V2	165 181	89 85	22 19
5B	115	FN1	047	77	22	63	A1 A2	047 225	69 16	36 6	342	B1	333	86	12	C1	105	18	9	D1	290	82	6	-	-	-	-	69	V1 V2	336 019	87 83	27 10
5C	104	FN1	229	80	21	16	A1	277	76	27	368	B1 B2 B3	324 344 004	80 85 89	5 6 4	C1	100	32	7	D1	299	86	6	G1	009	43	3	50	V1 V2 V3	331 339 001	79 84 88	17 17 15
5D	42	FN1	041	80	26	12	A1	049	72	>35	91	B1 B2	354 025	72 67	8 5	C1	084	33	7	D1	275	78	8	M1 M2	326 300	50 41	9 7	13	V1 V2	350 020	79 84	>35 31
6A	59	FN1 FN2 FN3	056 823 239	85 83 73	19 13 14	35	A1 A2 A3	057 220 245	83 65 70	18 14 24	228	B1 B2	343 021	70 76	6 6	C1	060	36	5	D1	320	82	4	M1	328	62	4	4	V1 V2	351 024	69 84	6 28
6B	119	FN1	239	71	29	40	A1	235	73	32	243	B1	176	88	6	C1 C2	092 031	32 41	5 6	D1	327	77	4	G1 E1 M1	356 050 172	50 72 40	4 3 3	134	V1	356	90	78

NOTES

1. Orientation is given in terms of dip direction and dip of the peak or average orientation of the discontinuity set.

2. Percent refers to the percent concentration in a one percent area of the lower hemisphere for the peak or average orientation of the population.

TABLE II

R.B. MINING PTY LIMITED
MT. CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

ORIENTATION OF DISCONTINUITY SETS IN THE OPEN PIT

BY: DCM

DATE: MAR 82

APPROVED: [signature]

DWG: 358-T2

DESCRIPTION OF DESIGN SECTORS					MAIN MODES OF SLOPE FAILURE WHICH CONTROL SLOPE STABILITY	STABILITY ANALYSIS RESULTS FOR KINEMATICALLY POSSIBLE FAILURE MODES[1,2]							
						AVERAGE DIP OR APPARENT PLUNGE OF FAILURE FOR WORST CASE WHERE STATISTICAL ANALYSIS NOT USED		APPARENT PLUNGE OF WEDGES CORRESPONDING TO A CUMULATIVE PERCENTAGE OF 20%		APPARENT PLUNGE OF WEDGES CORRESPONDING TO THE BREAK IN THE CUMULATIVE FREQUENCY PLOT			
										F<2.0		F<1.2	
DESIGN SECTOR	STRUCTURAL DOMAIN	SLOPE DIP DIRECTION (°)	MAXIMUM SLOPE HEIGHT (m)	NUMBER OF HAULROADS		F<2.0	F<1.2	F<2.0	F<1.2	CUMUL. PERCENT	APPARENT PLUNGE	CUMUL. PERCENT	APPARENT PLUNGE
1-1	1 and 2	012	85	-	Rotational failure in weathered rocks at pit crest. Wedge failures formed by combination of joints of Joint Set A and Joint Set D.	40°	44°	58°	54°	15	53°	10	56°
2-1	1 and 2	342	85	-	Rotational failure in weathered rocks at pit crest. Plane failures along joints of Joint Set A and Joint Set G.	47	47	50	50	-	-	-	-
3-1	3	330	30	1	Rotational failure in weathered rock at pit crest. Plane failure along joints of Joint Set B.	44	44	44	44	6	44	6	44
4-1	4	214	60	1	Plane failure along joints of Joint SetA. Wedge failures formed by combination of joints of Joint Set A and Joint Set D.	69	69	69	69	-	-	-	-
4-2	4	258	60	1	Wedge failure formed by combination of joints of Joint Set A and Joint Sets B.	52	56	52	58	-	-	-	-
5A-1	5A	097	170	2	Wedge failures formed by combination of joints, faults, shears and veins.	27	42	36	46	-	-	22	66
5A-2	5A	135	170	1	Wedge failures formed by combination of joints, faults, shears and veins.	34	70	47	70	-	-	-	-
5B-1	5B	166	260	2	Wedge failures formed by combination of joints, faults, shears and veins.	59	76	75	76	12	74	-	-
5C-1	5C	178	260	2	Wedge failures formed by combination of joints, faults, shears and veins.	57	80	79	80	23	79	-	-
5C-2	5C	197	120	2	Wedge failures formed by combination of joints, faults, shears and veins.	77	77	84	84	-	-	-	-
5C-3	5C	270	100	1	Wedge failure formed by combination of joints, faults, shears and veins.	68	68	71	71	-	-	-	-
5C-4	5C	344	50	1	Wedge failures formed by combination of joints, faults, shears and veins.	43	43	74	74	18	74	17	74
5D-1	5D	344	100	2	Wedge failures formed by combination of joints, faults, shears and veins.	26	40	48	51	-	-	-	-
6A-1	6A	345	60	1	Wedge failures formed by combination of joints, faults, shears and veins.	47	53	64	64	20	65	20	66
6A-2	6A	025	120	1	Wedge failures formed by combination of joints, faults, shears and veins.	34	36	59	69	25	70	16	68
6B-1	6B	351	50	1	Plane failure along joints of Joint Set G1. Wedge failures formed by combination of joints, faults, shears and veins.	47	49	50	52	-	-	-	-
6B-2	6B	055	170	2	Wedge failures formed by combination of joints, faults, shears and veins.	27	40	34	46	-	-	-	-

	OPTIMUM INTERRAMP SLOPE DESIGN FOR DRY SLOPES BASED ON KINEMATIC ANALYSES RESULTS (SLOPE DESIGN FOR SLOPES WITH HIGH GROUNDWATER PRESSURES ARE GIVEN IN BRACKETS WHERE DIFFERENT)						POSSIBLE INCREASE IN SLOPE ANGLE DUE TO PLAN RADIUS OF CURVEATURE (°)	RECOMMENDED INTERRAMP SLOPE DESIGN FOR DRY OR DRAINED SLOPES OR SLOPES WITH ADEQUATE GROUNDWATER DEPRESSURIZATION[4]						
	BENCH HEIGHT = 20m			BENCH HEIGHT = 30m				BENCH HEIGHT = 20m BETWEEN 305m AND 345m ELEVATION			BENCH HEIGHT = 30 m ABOVE 345m AND BELOW 305m ELEVATION			REMEDIAL MEASURES AND COMMENTS[5,6,7]
DESIGN SECTOR	OPTIMUM BENCH FACE ANGLE (°)	TOTAL BERM WIDTH[3] (m)	INTERRAMP SLOPE ANGLE (°)	OPTIMUM BENCH FACE ANGLE (°)	TOTAL BERM WIDTH[3] (m)	INTERRAMP SLOPE ANGLE (°)		BENCH FACE ANGLE (°)	BERM WIDTH (m)	INTERRAMP SLOPE ANGLE (°)	BENCH FACE ANGLE (°)	BERM WIDTH (m)	INTERRAMP SLOPE ANGLE (°)	
1-1	-	-	-	-	-	-	-	-	-	-	-	-	-	Statistical analysis and design techniques were not used in Design Sectors 1-1, 2-1 and 3-1 because slope stability is controlled by rotational failure in weathered rocks as well as kinematically possible failure modes. Slope design and remedial measures in this area are discussed in section 8.7.
2-1	-	-	-	-	-	-	-	-	-	-	-	-	-	
3-1	-	-	-	-	-	-	-	-	-	-	-	-	-	
4-1	80	10.5	55.0	80	14.7	56.3	-	80	10.5	55	80	14.7	56	
4-2	80 (70)	12.1 (16.5)	52.1 (50.5)	70 (70)	23.0 (24.7)	52.5 (50.5)	+4°	80	10.5	55	80	14.4	56.5	Slope angle of 55° is recommended for 20m high benches to provide access to the south wall. Drainholes may be required to control groundwater pressures.
5A-1	70 (70)	19.1 (28.0)	45.9 (35.5)	70 (70)	28.9 (41.4)	46.1 (36.0)	+3°	70	17.4	49	70	26.1	49	Slopes below 305m elevation could be increased to 50° based on increased effects of plan radius of curvature. Horizontal drainholes may be required.
5A-2	80 (70)	10.5 (18.7)	55.0 (46.9)	80 (70)	14.2 (28.1)	57.0 (46.9)	-	80	10.5	55	80	19.5	57	Horizontal drainholes may be required in specific areas on benches to control groundwater pressures in the slope.
5B-1	80	10.5	55.0	80	12.2	59.7	-	80	10.5	55	80	12.0	60	
5C-1	80	10.5	55.0	80	12.2	59.7	-	80	10.5	55	80	12.0	60	
5C-2	80	10.5	55.0	80	12.2	59.7	-	80	10.5	55	80	12.0	60	
5C-3	80	10.5	55.0	80	13.6	57.8	+5	80	10.5	55	80	12.0	60	Maximum slope angles of 55° are recommended for 20m high benches in this design sector to insure reliable access to the south wall.
5C-4	80	10.5	55.0	80	12.2	59.7	-	70	16.7	50	70	25.3	50	Maximum interramp slopes of 50° are recommended below 305m elevation due to sensitive nature of slope in this area and possible requirements for remedial measures.
5D-1	70 (70)	16.7 (18.1)	50.1 (47.9)	70 (70)	25.4 (27.5)	49.8 (47.5)	-	70	18.7	47	N/A	N/A	N/A	20m high benches and maximum interramp slope angles of 47° are recommended above the haulroad (300m elevation) in Design Sectors 5D-1 and 6A-1 due to the sensitive nature of the slope and possible requirements for remedial measures. Horizontal drainholes may be required.
6A-1	80	11.3	53.5	70	20.9	55.1	-	70	18.7	47	70	25.3	50	
6A-2	80 (80)	10.5 (12.0)	55.0 (52.2)	80 (70)	14.7 (22.6)	56.3 (53.0)	-	70	17.4	49	70	26.1	49	Interramp slope angles of 49° are recommended in this design sector for blending of slope designs between adjacent design sectors. Horizontal drainholes may be required.
6B-1	70 (70)	16.5 (17.1)	50.5 (49.5)	70 (70)	24.7 (26.1)	50.5 (49.0)	-	N/A	N/A	N/A	70	25.3	50	Same as comments for Design Sector 5C-4. Horizontal drainholes may be required.
6B-2	70 (70)	19.1 (30.7)	45.9 (34.0)	70 (70)	28.9 (44.4)	46.1 (34.0)	+3°	70	17.4	49	70	26.1	49	Slopes below 305m elevation could be increased to 50° based on increased effects of plan radius of curvature. Horizontal drainholes may be required.

NOTES

1. Analyses results are presented for F<2.0 and F<1.2 which relate to undewatered and dry, drained or dewatered slopes respectively.
2. The dip or apparent plunge which is considered to control slope stability is circled for both undewatered (F<2.0) and drained (F<1.2) conditions in each design sector.
3. Total berm width is calculated based on berm width required to catch failed material with consideration of anticipated breakback of bench crests.
4. Feasibility of using 30m high benches should be verified by assessment of trial slopes.
5. Control blasting is recommended in all areas of the final slope to ensure minimal disturbance to the rock and maintain cohesion in the slope.
6. If control blasting is not adequate or undue failures occur in certain areas, grouted dowels may be required to ensure the integrity of haulroads or other critical installation.
7. Remedial measures are recommended to control groundwater and surface water. Sealed drainage ditches graded berms and horizontal drainholes are recommended in critical areas as required. If adequate groundwater control is not obtained using drainholes additional hydrogeological studies may be required.

TABLE IX

R. B. MINING PTY. LIMITED
MT. CARBINE MINE
GEOTECHNICAL STUDIES

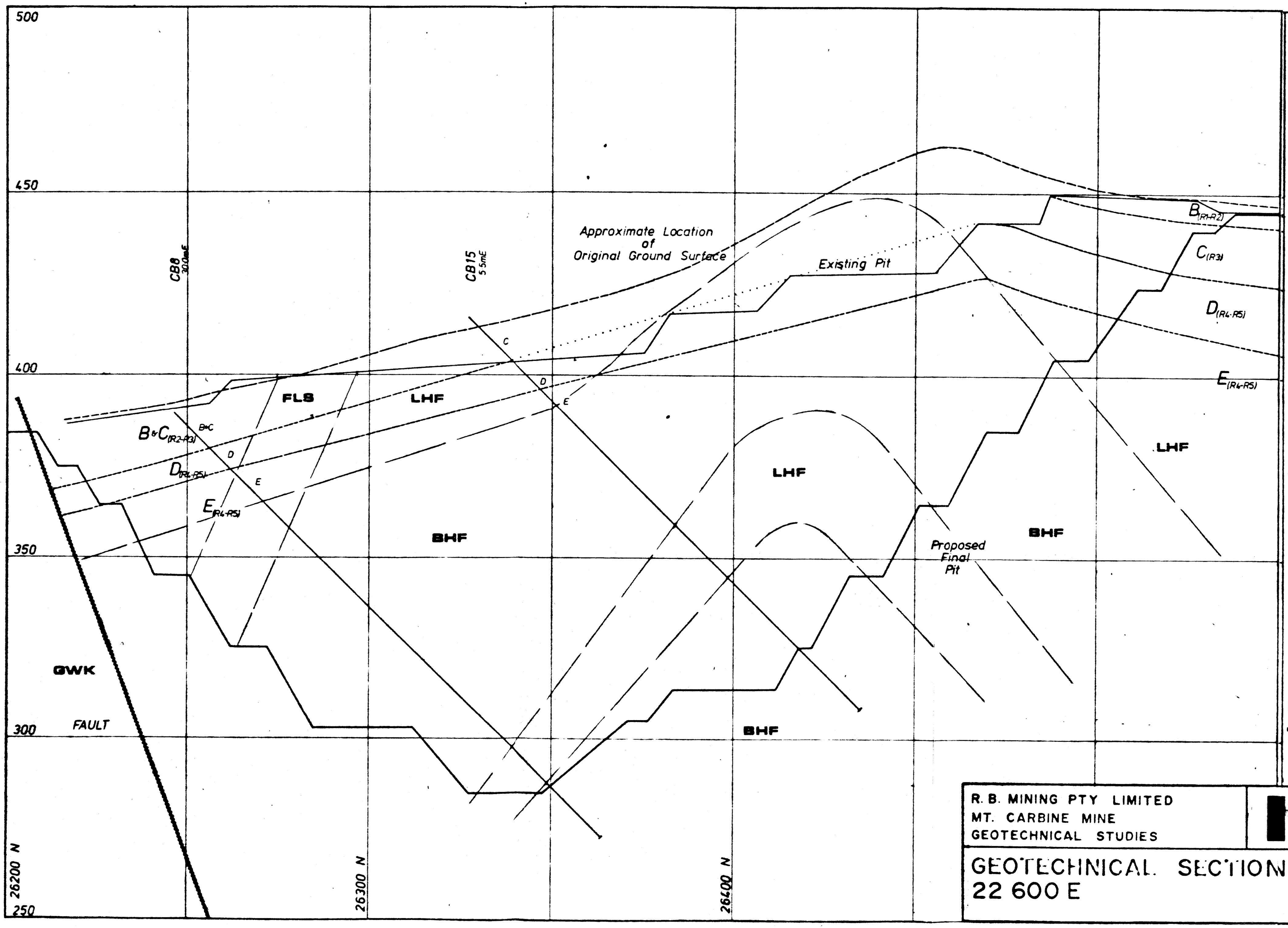
PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

SUMMARY OF SLOPE INFORMATION, STABILITY ANALYSIS RESULTS AND SLOPE DESIGN RECOMMENDATIONS FOR DESIGN SECTORS

BY	DATE
DCM	MAR 82
APPROVED	DWG.
D. Piteau	358-20

APPENDIX A

GEOTECHNICAL SECTIONS



SYMBOLS

Geological contact

Boundary of weathered zones
Defined & assumed

Number & location of drillholes showing piezometer installations (•), standpipe installations (▪), and piezometric levels for dry & wet seasons

CB21
S1
P1

Proposed location of final pit wall

Estimated piezometric surface

NOTE:

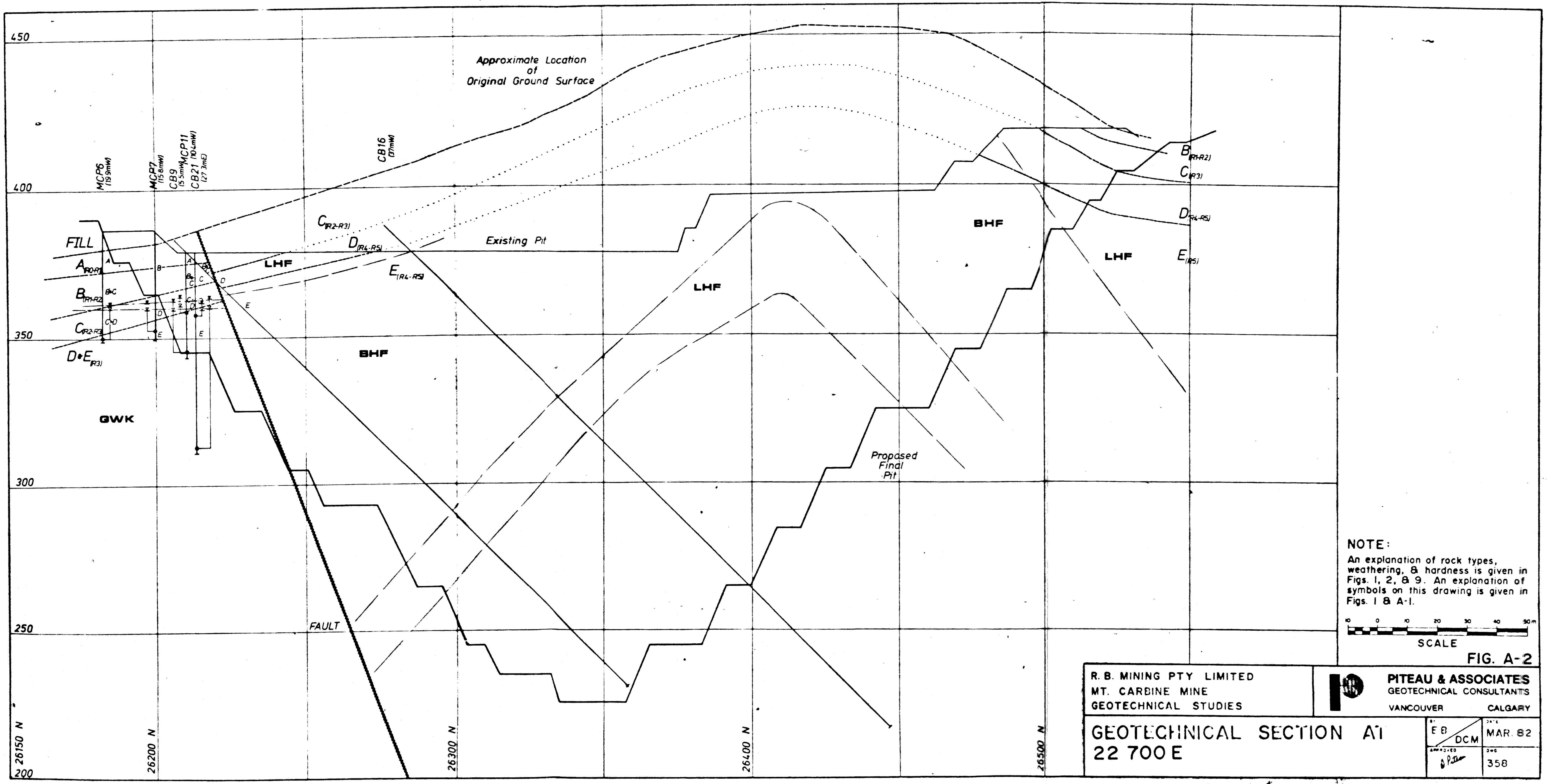
An explanation of rock types, weathering, & hardness is given in Figs. 1, 2, & 9. An explanation of symbols on this drawing is given in Figs. 1 & A-1.

10 0 10 20 30 40 50 m

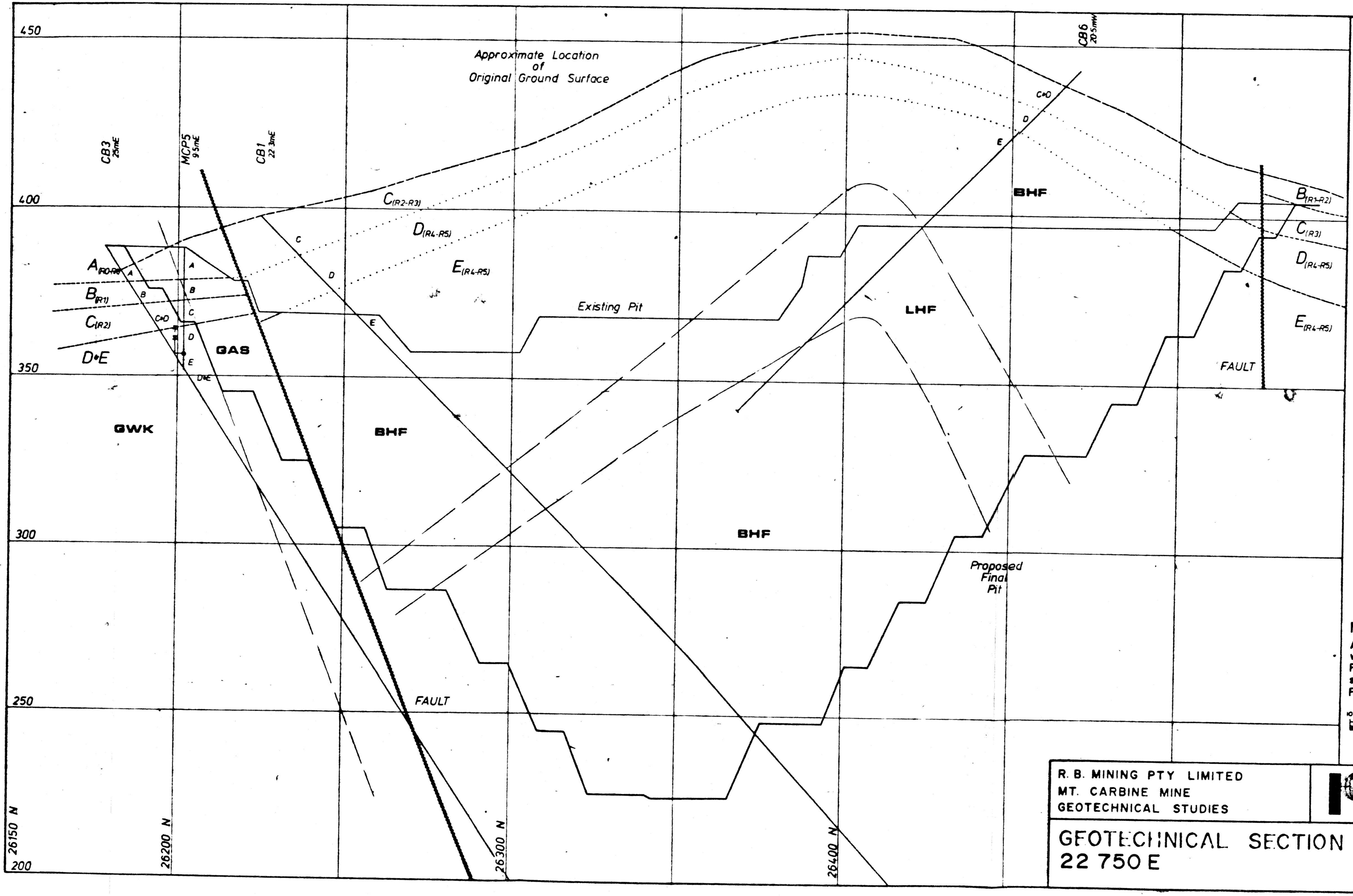
SCALE

FIG. A-1

R. B. MINING PTY LIMITED MT. CARBINE MINE GEOTECHNICAL STUDIES		PITEAU & ASSOCIATES. GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY
GEOTECHNICAL SECTION AT 22 600 E	EB / DCM	MAR. 82
		358



Approximate Location of Original Ground Surface
Existing Pit
Proposed Final Pit
FAULT
FILL
GWK
LHF
BHF
MCP6 (19.9mW)
MCP7 (15.8mW)
CB9 (5.5mW)
MCP11 (10.4mW)
CB21 (27.3mE)
CB16 (37mW)
450
400
350
300
250
200
26150 N
26200 N
26300 N
26400 N
26500 N
NOTE:
An explanation of rock types, weathering, & hardness is given in Figs. 1, 2, & 9. An explanation of symbols on this drawing is given in Figs. 1 & A-1.
SCALE
FIG. A-2
R. B. MINING PTY LIMITED
MT. CARBINE MINE
GEOTECHNICAL STUDIES
PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER
CALGARY
GEOTECHNICAL SECTION AT 22 700 E
EB
DCM
MAR. 82
358



NOTE:
An explanation of rock types, weathering, & hardness is given in Figs. 1, 2, & 9. An explanation of symbols on this drawing is given in Figs. 1 & A-1.

10 0 10 20 30 40 50 m
SCALE

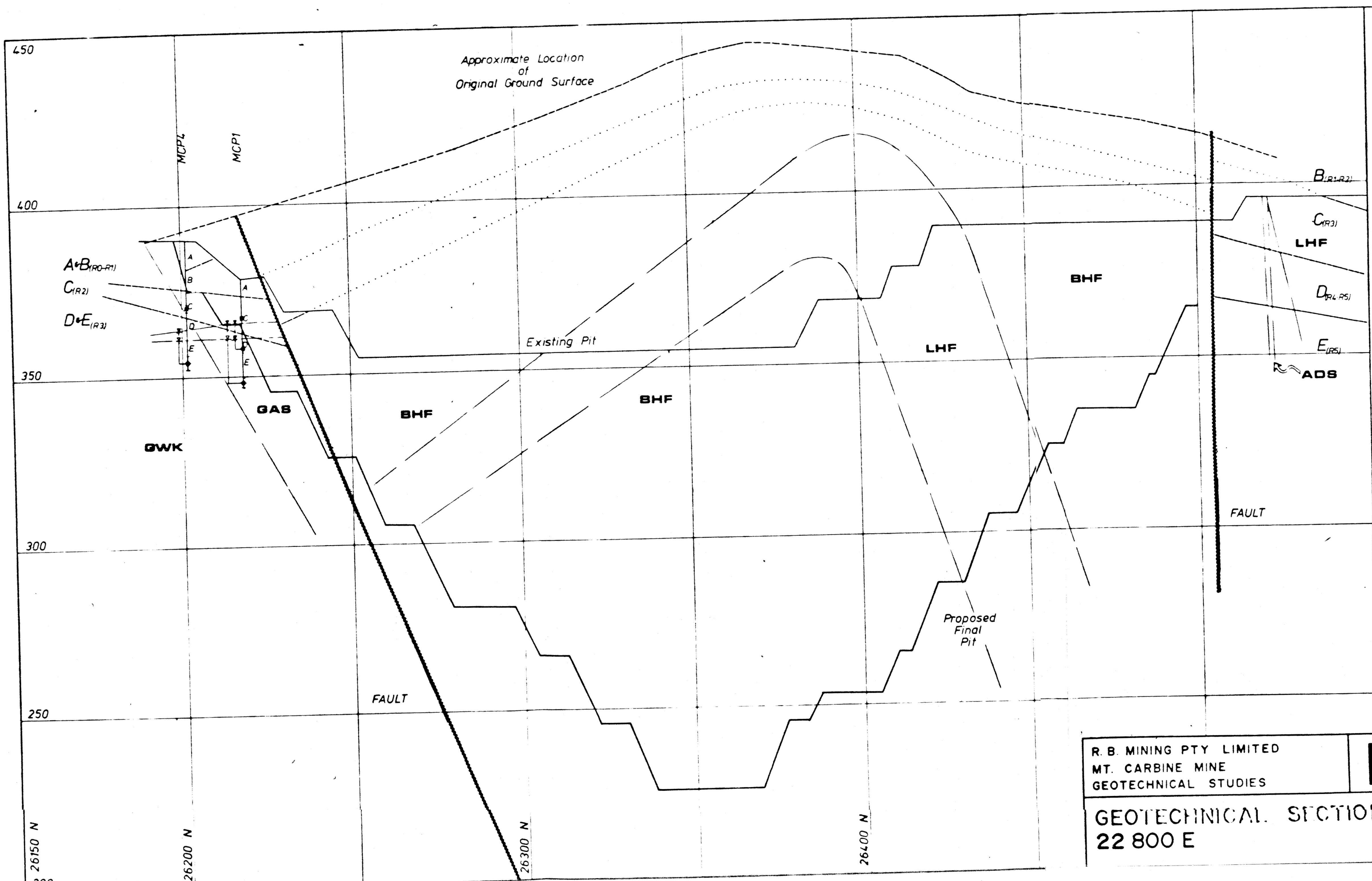
FIG. A-3

R. B. MINING PTY LIMITED
MT. CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

GEOTECHNICAL SECTION AT 22 750 E

EB	DCM	MAR. 82
		358

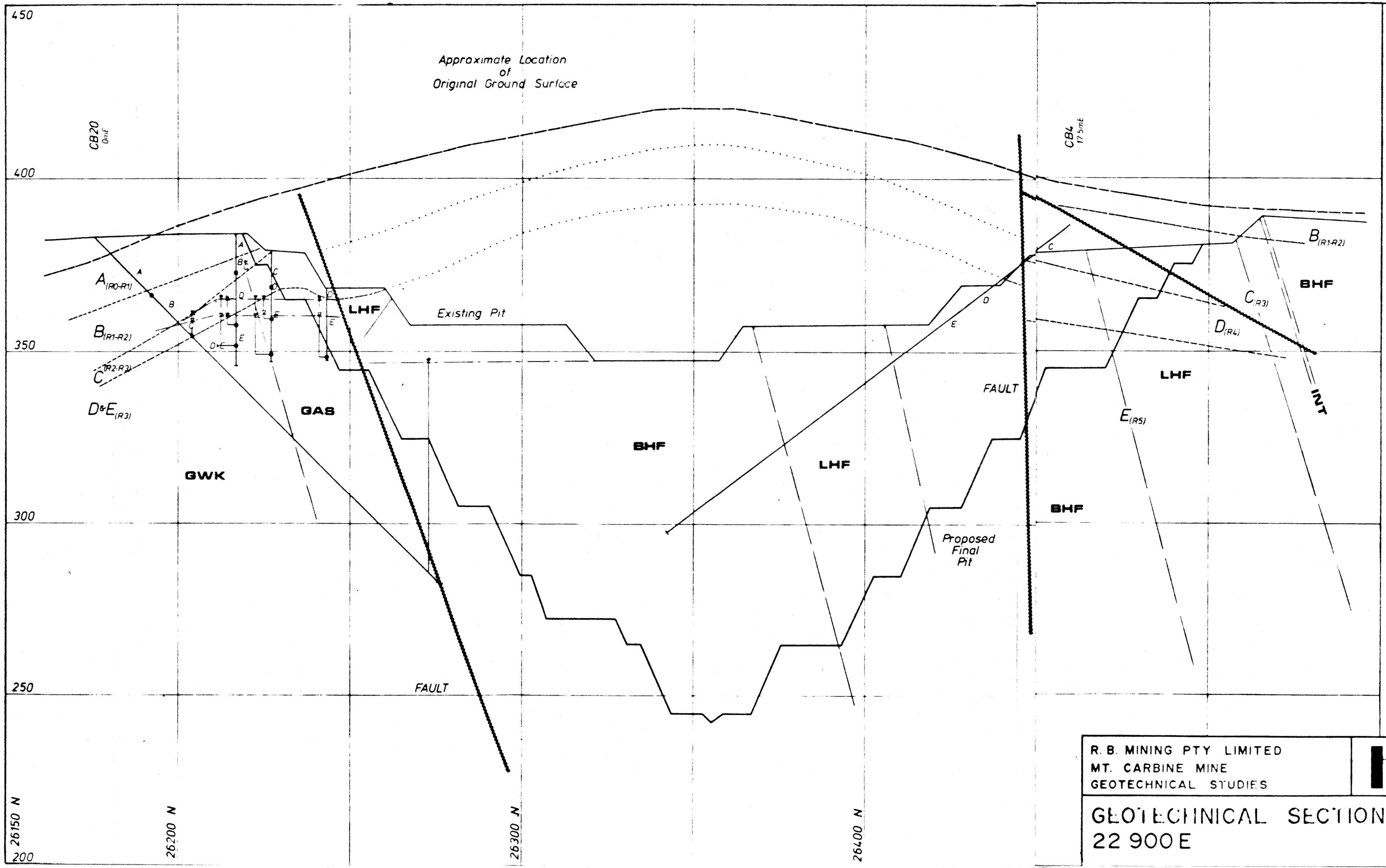


NOTE:
An explanation of rock types, weathering, & hardness is given in Figs. 1, 2, & 9. An explanation of symbols on this drawing is given in Figs. 1 & A-1.

SCALE

FIG. A-4

R. B. MINING PTY LIMITED MT. CARBINE MINE GEOTECHNICAL STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY		
GEOTECHNICAL SECTION AT 22 800 E	EB	DCM	MAR 82
			358



NOTE:
An explanation of rock types, weathering, & hardness is given in Figs. 1, 2, & 9. An explanation of symbols on this drawing is given in Figs. 1 & A-1.

10 0 10 20 30 40 50 m
SCALE

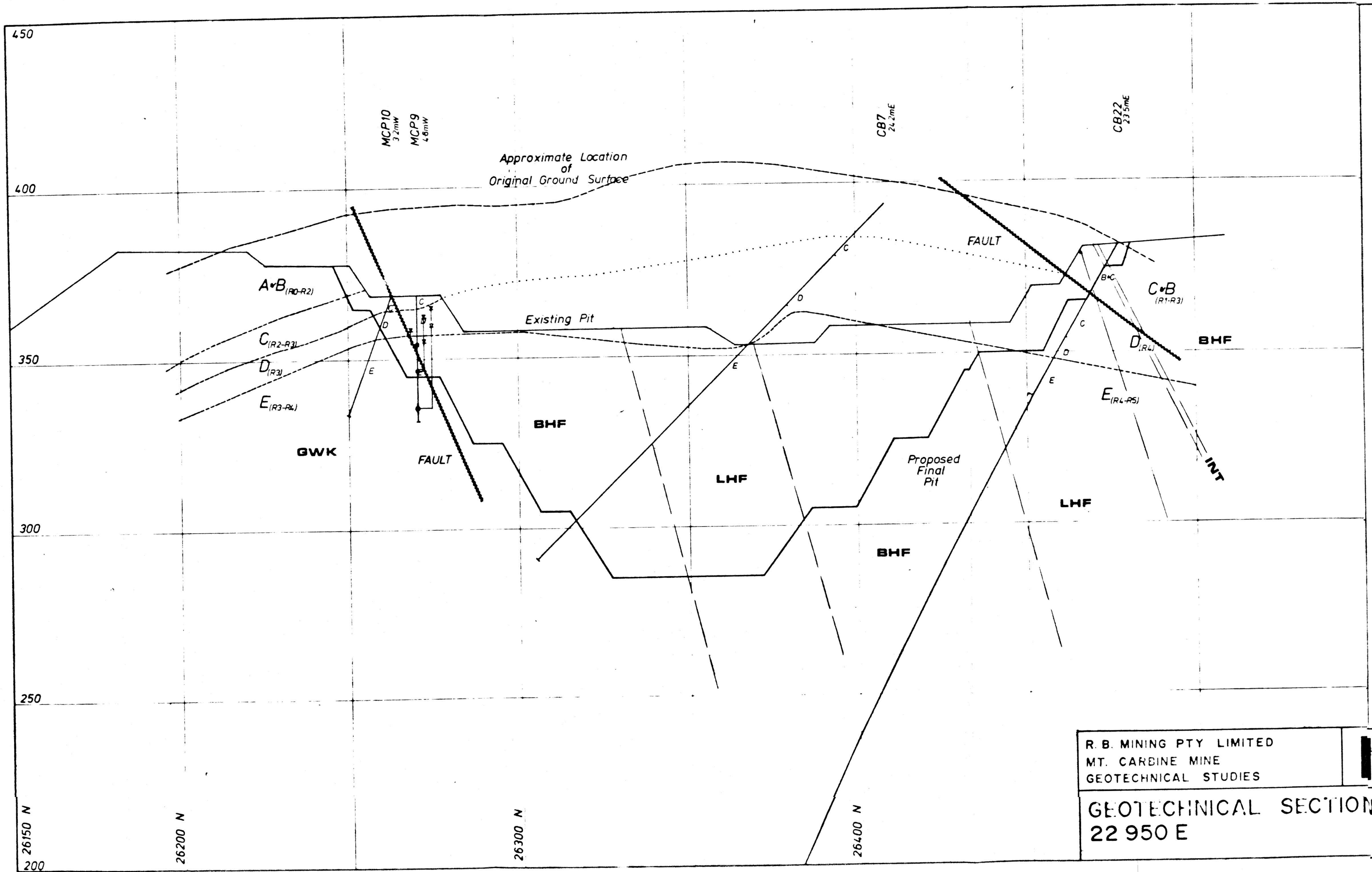
FIG. A-5

R. B. MINING PTY LIMITED
MT. CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

GEOTECHNICAL SECTION AT 22 900 E

EB / DCM	MAR 82
	358



NOTE:
An explanation of rock types, weathering, & hardness is given in Figs. 1, 2, & 9. An explanation of symbols on this drawing is given in Figs. 1 & A-1.

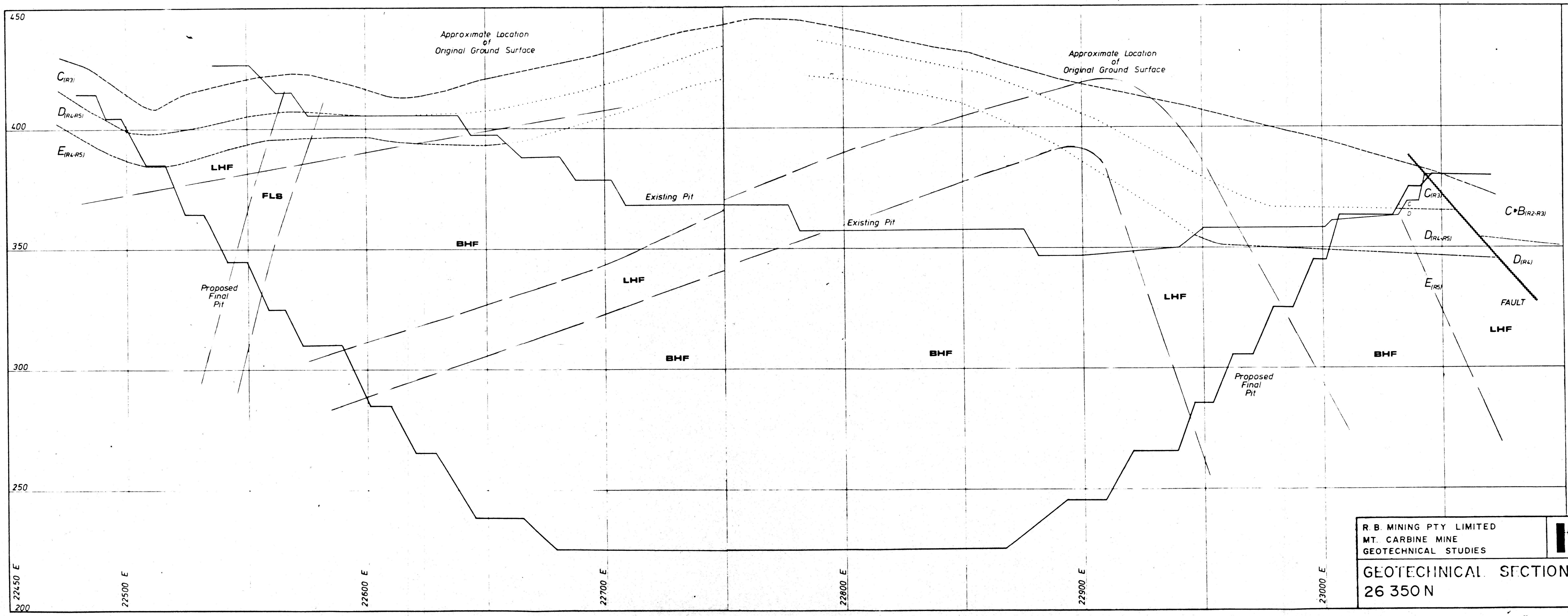
FIG. A-6

R. B. MINING PTY LIMITED
MT. CARBINE MINE
GEOTECHNICAL STUDIES

PITEAU & ASSOCIATES
GEOTECHNICAL CONSULTANTS
VANCOUVER CALGARY

GEOTECHNICAL SECTION AT 22 950 E

EB DCM MAR 82 358



NOTE:
An explanation of rock types, weathering, & hardness is given in Figs. 1, 2, & 9. An explanation of symbols on this drawing is given in Figs. 1 & A-1.

10 0 10 20 30 40 50 m
SCALE

FIG. A-7

R. B. MINING PTY LIMITED MT. CARBINE MINE GEOTECHNICAL STUDIES	PITEAU & ASSOCIATES GEOTECHNICAL CONSULTANTS VANCOUVER CALGARY		
GEOTECHNICAL SECTION AT 26 350 N	EB / DCM	MAR 82	358

APPENDIX B

PERCUSSION DRILL LOGS

Sheet 1 of 1

DRILLING REPORT

DRILL ROTATION 24 RPM	CO-ORDINATES 26218.7 N 22817.5 E 378.8 RL	HOLE No. MCP - 1 PROJECT MT. CARBINE LOCATION SOUTH 375 JOB No. 81 - 358 DATE 12.09.81

DRILLING DATA

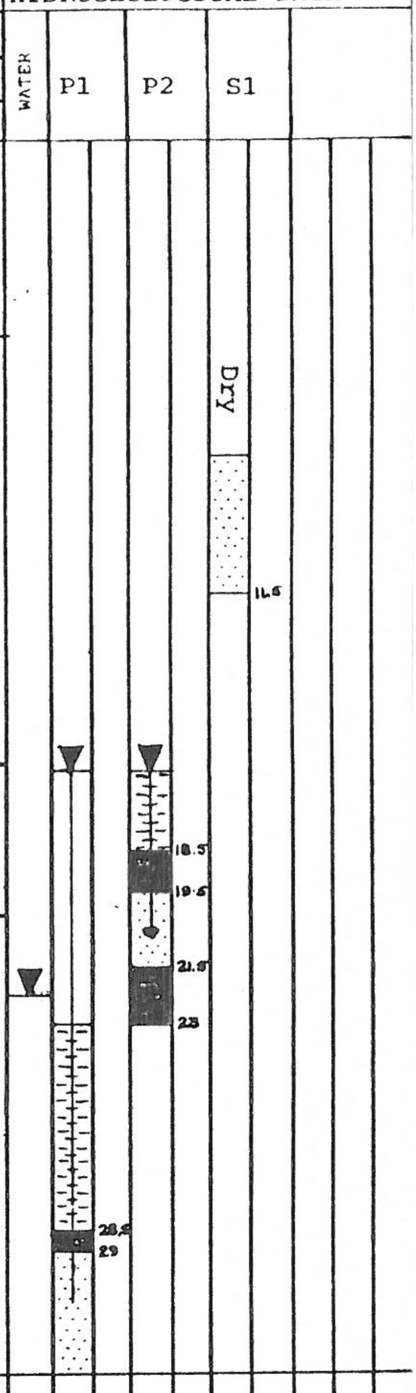
CLIENT

Elevation Ground/Collar

Water Level

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQD	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
0–2	100	-			A	On to 5m - Moderately to highly weathered schistose chips
2–4		130		X		
4–6		95			C	5m to 16m - Moderately weathered schistose chips - Chip surfaces weathered but rock unaltered
6–8		135		X		
8–10		140				
10–12		160		X		
12–14		150				
14–16		180		X		
16–18		120			D	16m to 20m - Slightly weathered laminated schist
18–20		140		X		
20–22		195			E	20m to 32m - Unweathered laminated schist with quartz fragments
22–24		190		X		
24–26	O	170				
26–28		190		X		
28–30		220				
30–32		210		X		END OF HOLE
32–34						
34–36						
36–38						
38–40						

HYDROGEOLOGICAL DATA



Sheet 1 of 1

DRILLING REPORT

DRILL ROTATION	CO-ORDINATES		
24 RPM	26228.5 N 22880.0 E 379.3 RL	HOLE No.	MCP - 2
		PROJECT	MT. CARBINE
		LOCATION	SOUTH 375
		JOB No.	81 - 358
		DATE	12.09.81

DRILLING DATA

CLIENT

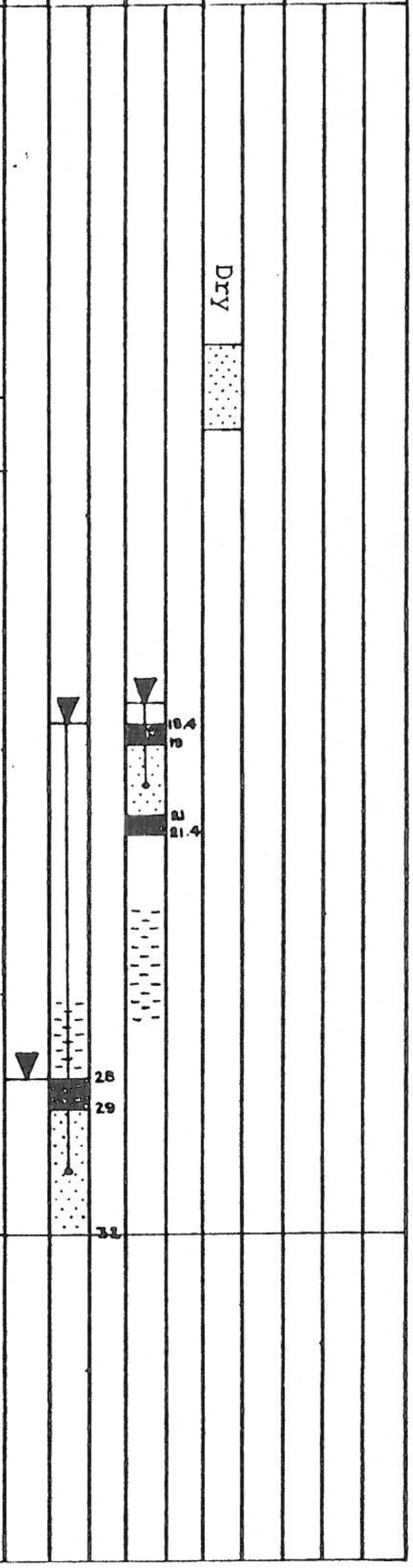
Elevation Ground/Collar

Water Level

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	100	90				0m - 10m - Moderately weathered laminated schist. - Strength Affected.
4		105		X	C	
6		90				
8		100		X		
10		90				
12		95		X	D	10m to 12m - Slightly weathered laminated schist
14		102				12m to 32m - Unweathered grey laminated schist. - Unsilicified
16		145		X		
18		153				
20	500	146		X		
22		195			E	
24		180		X		
26	O	170				
28		192		X		
30		218				
32		200		X		END OF HOLE
34						
36						
38						
40						

HYDROGEOLOGICAL DATA

WATER	P1	P2	S1



DRILLING REPORT

DRILL ROTATION	CO-ORDINATES	
24 RPM	26218.3 N 22879.3 E 383.4 RL	HOLE No. MCP - 3 PROJECT MT. CARBINE LOCATION SOUTH 385 JOB No. 81 - 358 DATE 12.09.81

DRILLING DATA

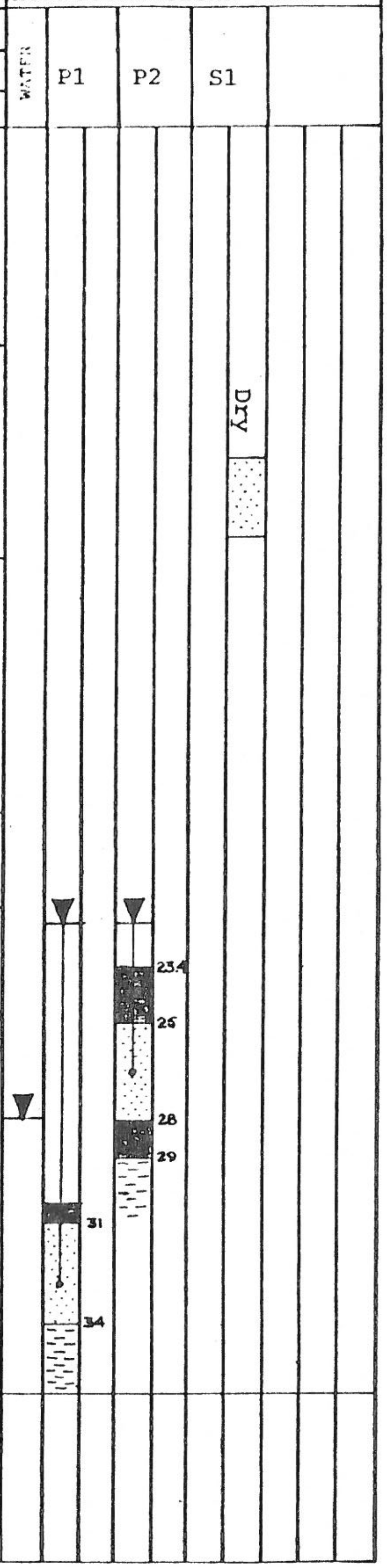
CLIENT

Elevation Ground/Collar

Water Level

HYDROGEOLOGICAL DATA

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	100	90				0m to 6m - Yellow brown fine grained powder
4		130		X	A	- A few chips - Extremely weathered
6		93				
8		157		X X	B	6m to 12m - Moderately weathered siliceous laminated rock chips
10		124		X	+ C	- Green
12		105		X		
14	500	105				12m to 22m - As above but slightly weathered
16		104		X		
18		147			D	19m - Moderately weathered seam, 0.2m wide
20		137		X		
22	100	225				
24		214		X		22m to 36m - Slightly weathered to unweathered finely laminated chips of green siliceous rock
26		250				
28		262		X		
30		223			E	
32		250		X		
34		220				
36		235		X		END OF HOLE
38						
40						



DRILLING REPORT

DRILL ROTATION 24 RPM	CO-ORDINATES 26202.7 N 22808.0 E 389.6 RL	HOLE No. MCP - 4 PROJECT MT. CARBINE LOCATION SOUTH 385 CREST JOB No. 81 - 358 DATE 12.09.81

DRILLING DATA

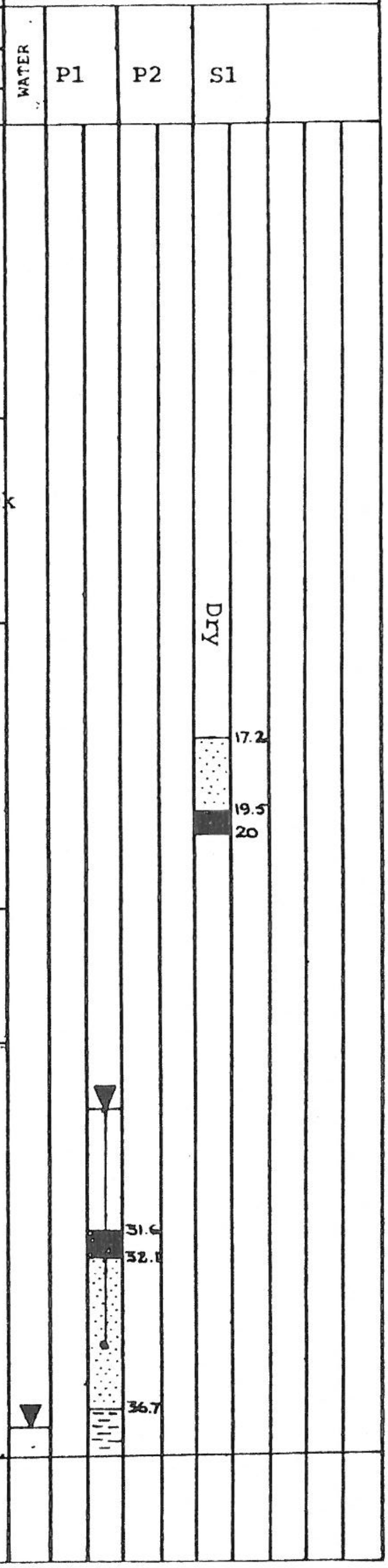
CLIENT

Elevation Ground/Collar

Water Level

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	100	75			A + B	0m to 8m - Extreme to highly weathered schistose rock - Yellow-brown - Areas of both siliceous and nonsiliceous rock
4		64		X		
6		67				
8		61		X		
10		90			B	8m to 14m - Highly weathered siliceous and non-siliceous schistose rock
12		104		X		
14		113				
16		104		X	C	14m to 22m - Moderate to slightly weathered schistose rock - Varying degrees of silicification
18		116				
20		115		X		
22	0	181				
24		184		X	D	22m to 26m - Grey laminated cherty rock - Similar to LHR
26		180				
28		345		X	D + E	26m to 38m - Unweathered green slightly laminated cherty rock with quartz chips - Similar to MCP-3 and GHR
30		266				
32		230		X		
34		180				
36		205				
38		180				END OF HOLE
40						

HYDROGEOLOGICAL DATA



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DRILLING REPORT

DRILL ROTATION 24 RPM	CO-ORDINATES 26202.4 N 22759.8 E 389.3 RL	HOLE No.	MCP-5
		PROJECT	MT. CARBINE
		LOCATION	SOUTH 385 CREST
		JOB No.	81-358
		DATE	12.9.81

DRILLING DATA

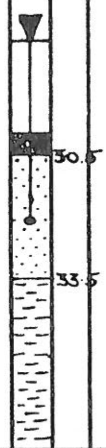
CLIENT

Elevation Ground/Collar

Water Level

HYDROGEOLOGICAL DATA: WATER | P1 | P2 | S1

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	100	81			A	0m to 10m - Extremely weathered reddish brown chips. - Not identifiable
4		105		X	A	
6		71			A	
8		155		X	A	
10		165			A	
12		95		X	B	10m to 16m - Highly weathered schistose rock. - Not very siliceous
14		107			B	
16		119		X	B	
18		117			C	16m to 24m - Moderately to highly weathered siliceous rock chips
20		119		X	C	
22		102			C	
24		116		X	C	
26		105			D	24m to 30m - Moderate to slight weathered grey green siliceous chips
28		117		X	D	
30		118			D	
32	0	177		X	E	30m to 38m - Unweathered grey green siliceous rock chips - Less silicified than other rocks
34		160			E	
36		151		X	E	
38		142			E	
						END OF HOLE
40						



DRILLING REPORT

DRILL ROTATION	CO-ORDINATES		
24 RPM	26180.3 N 22680.1 E 385.3 RL	HOLE No.	MCP-6
		PROJECT	MT. CARBINE
		LOCATION	COARSE ORE STOCKPILE
		JOB No.	81-358
		DATE	12.9.81

CLIENT

Elevation Ground/Collar

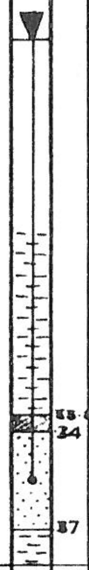
Water Level

DRILLING DATA

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2						0m to 5m – Rock fill – Clay used to stabilize hole
4						
6						5m to 14m – Highly weathered schist mixed with chips of host rock. – Sample probably erroneous due to fill
8	100	77				
		75			A	
10				X		
		85				
12						
		70				
14				X		
		88				14m to 26m – Moderatley to strongly weathered green argillite schist
16				X		
		77				
18					B	
		97			+	
20				X	C	
		82				
22						
		92				
24				X		
		115				
26						26m to 35.5m – Moderately to slightly laminated schistose rock. – Only slightly siliceous
		89				
28				X		
		?			D	
30						
		104				
32				X		
		95				
34						
		105				
36					D + E	35.5m to 37.5m – Slightly weathered siliceous laminted chips.
		90				END OF HOLE
38						
40						

HYDROGEOLOGICAL DATA

WATER	P1	P2	S1



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DRILLING REPORT

DRILL ROTATION	CO-ORDINATES		
20 RPM	26199.3 N 22684.2 E 385.7 RL	HOLE No.	MCP-7
		PROJECT	MT. CARBINE
		LOCATION	SOUTH WEST 385 CREST
		JOB No.	81-358
		DATE	13.9.81

DRILLING DATA

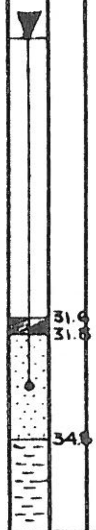
CLIENT

Elevation Ground/Collar

Water Level

HYDROGEOLOGICAL DATA: WATER | P1 | P2 | S1

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	0	78				0m to 5m - Rock Fill
4		75				
6		100				5m to 16m - Highly weathered siliceous rock in very small fragments
8		95		X		
10		64			B	
12		94		X		
14		90				
16		70		X		
18		89				16m to 25m - Moderately weathered siliceous rock. - 5% white quartz
20		113				
22		105			C	
24		107		X		
26	200	105				25m to 30m - Slightly weathered laminated siliceous - A few quartz fragments with sulfide min.
28		135		X	D	
30		213				
32		198		X		30m to 37m - Unweathered siliceous cherty rock - Laminated blue-grey. - 5% quartz with sulfides
34		195			E	
36		203		X		
						END OF HOLE
38						
40						



DRILLING REPORT

DRILL ROTATION	CO-ORDINATES		
	26244.7 N 22882.4 E 368.9 RL	HOLE No.	MCP - 8
		PROJECT	MT. CARBINE
		LOCATION	SOUTH 365
		JOB No.	81-358
		DATE	13.9.81

CLIENT

Elevation Ground/Collar

Water Level

DRILLING DATA

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	200	83			C + D	0m to 4m - Slightly to Moderately grey schist
4		90		X		
6		104				4m to 21.5m - Unweathered grey to schistose chips with pyrite. - Slightly siliceous
8		130		X		
10		100				
12		105		X		
14		146			E	
16		145		X		
18		140				
20		135		X		
22		140				END OF HOLE
24						
26						
28						
30						
32						
34						
36						
38						
40						

HYDROGEOLOGICAL DATA

WATER	P1	P2	S1
2.5 gpm	18.5 19 21		

Sheet 1 of 1

DRILLING REPORT

DRILL ROTATION	CO-ORDINATES		
	26271.4 N 22945.6 E 369.2 RL	HOLE No.	MCP - 9
		PROJECT	MT. CARBINE
		LOCATION	SOUTH 365 -
		JOB No.	81-358
		DATE	13-9-81

DRILLING DATA

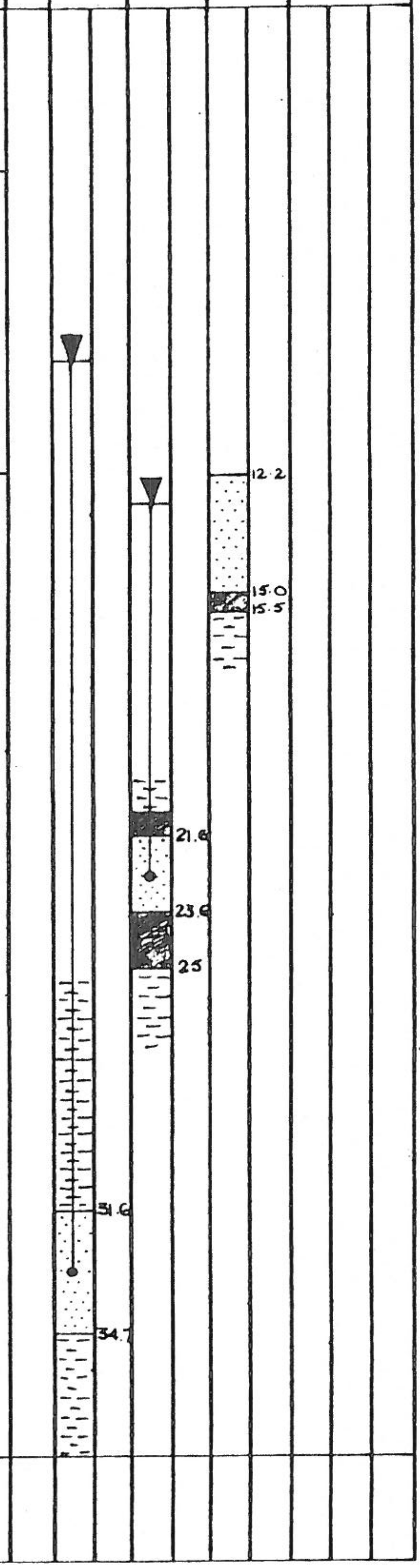
CLIENT

Elevation Ground/Collar

Water Level

HYDROGEOLOGICAL DATA (WATER, P1, P2, S1)

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	200	-			C	0m to 4m - Moderately weathered laminated Host rock
4		180		X		
6		174				4m to 12m - Slightly weathered laminated to massive greenish Host rock
8		194		X	D	
10		173				
12	100	160		X		
14		150				12m to 38m - Hard grey siliceous horfels rock
16		185		X		
18		153				17.5m - Rods drop 0.1m - Fault?
20		153		X		
22		180				
24		170		X	E	
26		164				
28		196		X		
30		180				
32		185		X		
34		198				
36		190				
38		100		X		END OF HOLE
40						



DRILLING REPORT

DRILL ROTATION 24 - 26 RPM BEARING: 180° INCL: 70°	CO-ORDINATES 26264.1 N 22946.9 E 368.9 RL	HOLE No. MCP-10 PROJECT MT. CARBINE LOCATION SOUTH ~~365~~ JOB No. 81-358 DATE 13-9-81

CLIENT

Elevation Ground/Collar

Water Level

HYDROGEOLOGICAL DATA: WATER, P1, P2, S1

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING	DESCRIPTION OF MATERIAL
2	200	120				0m to 5m - Moderately weathered siliceous rock
4		155		X	C	
6		133				5m to 14m - Grey to black siliceous Host rock. - Unweathered
8		134		X		
10		–			E	
12		148		X		
14		148				
16		138		X		14m to 36m - Grey to black schistose rock, less cherty
18		161				
20		161		X		
22		170				
24	150	167		X		
26		170			E	
28		198		X		
30		192				
32		191		X		
34		170				
36		142		X		END OF HOLE
38						
40						

DRILLING REPORT

DRILL ROTATION	CO-ORDINATES		
	26210.3 N 22689.6 E 378.8 RL	HOLE No.	MCP-11
		PROJECT	MT. CARBINE
		LOCATION	SOUTH WEST 375
		JOB No.	81-358
		DATE	13-9-81

CLIENT

Elevation Ground/Collar

Water Level

DRILLING DATA

DEPTH IN METERS	DOWN PRESSURE	TIME FOR 2m RUN	RQI	SAMPLE LOCATION	WEATHERING
2	200	58			A
4		73		X	
6		77			
8		74		X	B + C
10		84			
12		88		X	
14		93			
16		125		X	
18		91			C + D
20		132		X	
22		155			
24		170		X	
26		157			
28		135		X	E
30		128			
32		124		X	
34		119			
36		148		X	
38					
40					

DESCRIPTION OF MATERIAL

0m to 4m - Highly weathered siliceous rock

4m to 16m - Moderate to highly weathered siliceous rock

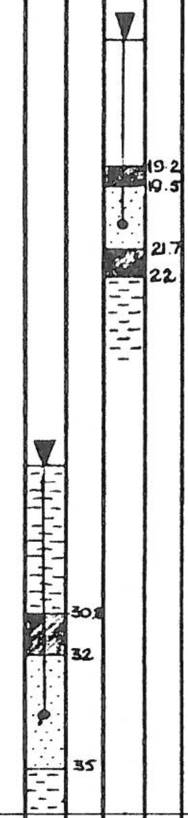
16m to 20m - Slightly to moderately weathered siliceous rock

20m to 26m - Unweathered laminated siliceous rock

END OF HOLE

HYDROGEOLOGICAL DATA

WATER	P1	P2	S1



APPENDIX C

DESCRIPTION OF THE

CUSUMS TECHNIQUE

Reprinted from STABILITY OF ROCK SLOPES
Thirteenth Symposium on Rock Mechanics
ASCE / Urbana, Illinois / August 30–September 1, 1971

CUMULATIVE SUMS TECHNIQUE: A NEW APPROACH TO ANALYZING JOINTS IN ROCK

By Douglas R. Piteau* and Lindsay Russell**

SYNOPSIS

The cumulative sums technique for analyzing joints in rock was developed as part of an extensive slope stability study of Nchanga pit. It was used successfully to determine the joint orientation trends, the pattern of their behavior and whether the joint information could be extrapolated to other areas in which slopes are proposed. This technique is illustrated with reference to the Nchanga study.

INTRODUCTION

The cumulative sums technique for analyzing joints in rock was developed as part of an extensive slope stability analysis of the hanging-wall of the Nchanga open pit in Zambia. This technique was used successfully to define the characteristic features of the joints, most particularly to determine the pattern of their behavior from one part of the Nchanga syncline, where the pit is situated, to the other. A description of cumulative sums technique is

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** Senior Operations Research Officer, Nchanga Consolidated Copper Mines Ltd., Chingola, Zambia.

given with particular reference to the joint analysis of the Nchanga pit.

Basically, the cumulative sums technique was developed to analyze the joint trends in a more definitive manner. This technique led to a better understanding of the genetic relationship of the joints occurring within the overall synclinal structure. Ultimately, predictions were made as to whether the joint data acquired from the existing hanging-wall pit face could be extrapolated (a) to an area some 300 ft behind the existing hanging-wall face, where the final slope is to be located and (b) to areas east of the existing face, where the pit is to be extended another mile.

The Nchanga syncline is approximately one and one quarter miles wide and seven miles long. The Nchanga open pit is located in the southern half of the western limits of the syncline, which consists of a clearly defined sequence of mainly sedimentary rocks (i.e. argillite, siltstones, shale, sandstone, etc). The sediments strike roughly east-west. The south limb dips between 20° N to 35° N, and the north limb dips steeply to the south, forming an asymmetrical synclinal structure with an axial plane dipping steeply north. The syncline plunges between 5° and 15° to the west.

The overall approach to the structural analysis of the hanging-wall slopes was basically straightforward [Piteau (6)]. Discontinuities in the rock were systematically measured along over three miles of benches, using the continuous detail line survey method as described by Piteau (5). The joint data were statistically analyzed, initially using rectangular, histogram, cumulative sums and other analysis methods to determine their nature and distribution. For purposes of this discussion "joint" is meant to include any naturally-occurring structural discontinuity in the rock mass.

CONSIDERATION OF THE ROCK MASS JOINT MODEL

An objective of the joint analyses is to obtain a schematized concept or model of the joints in the rock mass and to establish certain criteria which indicate where this model changes. Also, one seeks to establish confidence limits in areas where the model is considered to apply, regardless of whether it is in areas of extensive, limited or no sampling.

When designing open pit slopes on a rational basis, an important, if not the most important, consideration in most geological environments is the determination of the attitude, geometry and spatial distribution of the joints in the boundary of the proposed excavation. Thus, for purposes of rationally analyzing a rock slope, such a study must be dependent upon assessing three main factors, namely (1) the nature and structural arrangement of the joints; (2) the strength parameters of the joints; and (3) their relationships to possible failure surfaces. Based on this approach, the geological factors and certain geological premises are given by Piteau (4) and (5), methods of structural interpretation by Robertson (8) and mathematical theories for stability calculations by Jennings (3). This discussion deals exclusively with assessment of factor (1).

Of the three factors listed above, the first is the most important, as the two others are of little consequence if the structural interpretation, and hence the jointing model of the rock mass, does not represent the actual situation in a statistical sense. The first requirement of the model, whether it is of a physical, graphical or mathematical nature, is that it be true, and that a statistical sampling of any property will give a representative picture of the whole situation. The second is that any calculations made for a representative portion or section of the model apply to the model as a whole.

Thus, on the basis of the jointing model, and with due consideration of the strength parameters and kinematically possible failure modes for that particular structural situation, the stability of the slope can be theoretically determined /Jennings (3)/.

CONSIDERATIONS AT NCHANGA LEADING TO THE DEVELOPMENT OF THE CUMULATIVE SUMS TECHNIQUE

The present dimensions of the pit are 9,600 ft along strike, but will extend, eventually, along strike for three miles, after the extension of the pit eastwards is completed. It is presently approximately 2,500 ft wide at its present depth of 750 ft. However, it is planned to go to 1,000 ft, and possibly even to 1,200 ft depth, the result being a final width of about 3,000 ft.

The structural mapping was conducted on the hanging-wall face of the pit. The problem involved trying to determine whether the same or a different structural situation can be expected to exist in the hanging-wall slope when the final depth of 1,000 ft is achieved. The final hanging-wall slope will be at least 300 ft farther in from the existing face as the pit is advanced northwards. The existing and approximate final locations of the hanging-wall, along with some salient geological features, are shown in Fig. 1.

A print-out of the raw joint data representing greater than 3,000 joints from the hanging-wall is shown in a rectangular plot in Fig. 2. Horizontal rows indicate similar angle of dip, whereas vertical rows on the upper half and lower half of the plot indicate joints with similar direc-

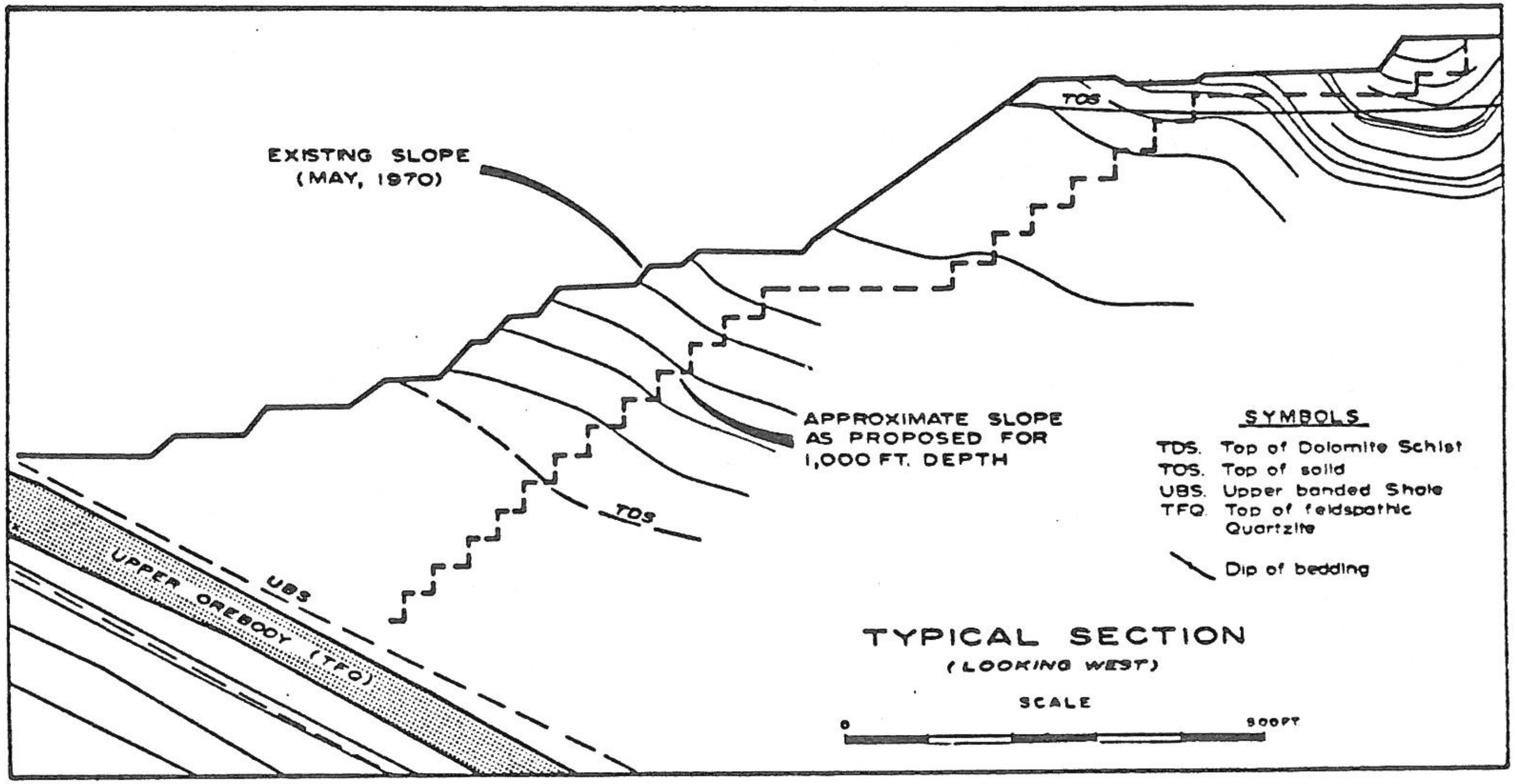

Fig. 1 Typical geological section (looking west), showing the existing and approximate final location of the Nchanga pit hanging-wall.

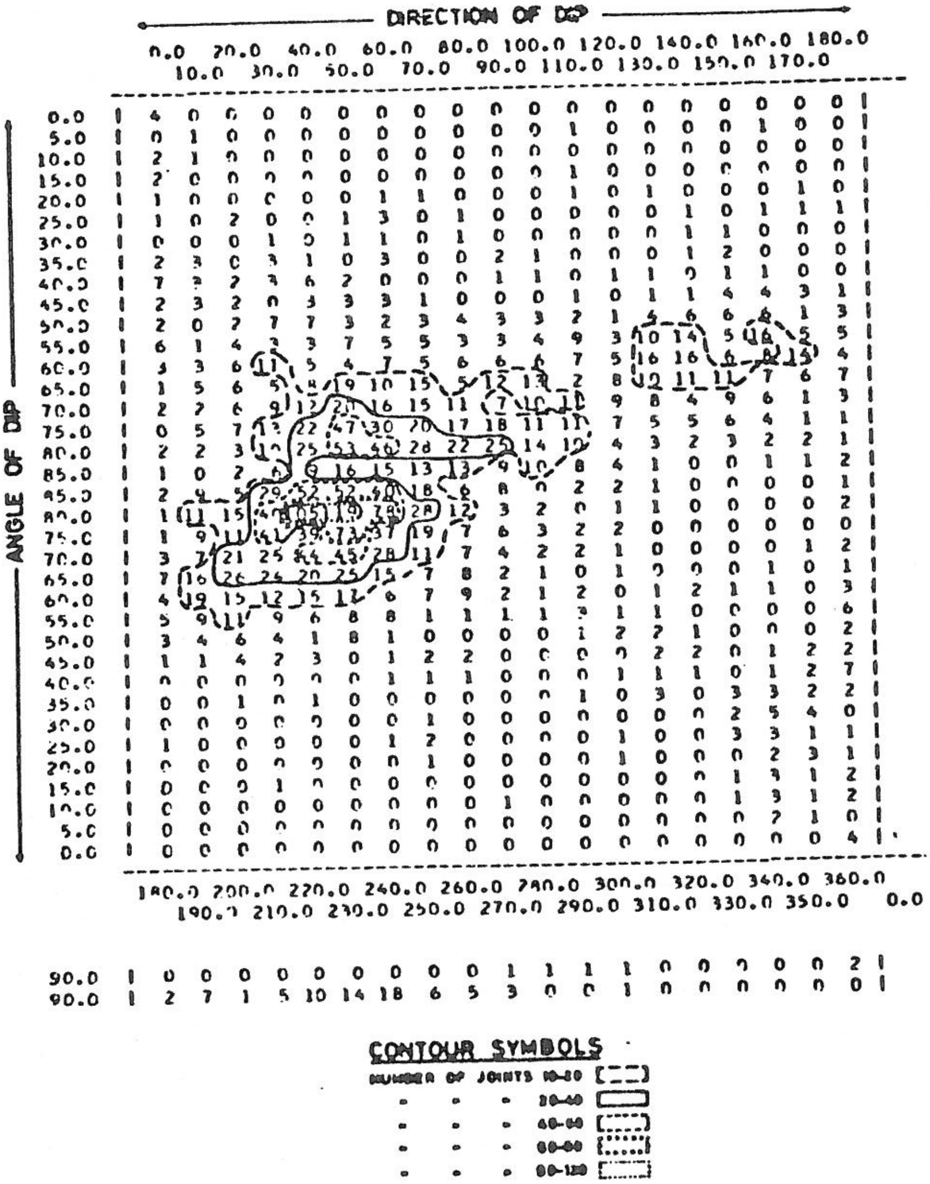

Fig. 2 Rectangular plot showing the distribution of raw joint data from Nchanga north face

tion of dip* /Robertson (8)/. It can be seen that about 70 percent of the joints are highly concentrated, occurring within a direction of dip interval of 50° (i.e. between 20° and 70°, and 200° and 250° in the upper and lower halves, respectively), and within an angle of dip interval of 55° (i.e. between 70° and 90°, and 65° and 90° in the upper and lower halves, respectively). That is, the peak concentration of these joints is centred about a strike of approximately 310°, and they dip steeply both to the northeast and southwest.

Detailed examinations of drag, monoclinal and major recumbent folds, both locally and at other points around the syncline, revealed that their axial planes were in fact striking about 300° to 310°, and not east-west, as might be expected from the approximately east-west orientation of the Nchanga syncline proper. From the results in Fig. 2 it can be seen that the peak concentration of the joints approximately parallel this tectonic fold axis.

Further study indicated that the topography of underlying basement granite dome structures to a large extent controlled the overall synclinal shape and did not control the tectonic fold process proper. For purposes of extrapolation and, ultimately, for assessing the significance of the joints with respect to slopes developed at different locations in the syncline, a more definitive knowledge of the joint behavior in regards to both dip and strike trends was required.

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* Direction of dip of a joint is the strike plus or minus 90°, depending upon whether the joint dips in a clockwise or counter-clockwise direction.

THE CUMULATIVE SUMS TECHNIQUE

GENERAL FEATURES

Cumulative sums, or "cusums" as they are also called, have been used extensively in industrial quality control [Woodward and Goldsmith (9)]. They have also been used for studying long-term trends in natural phenomena, such as river volume flows and silt deposition [Hurst et al (2)]. As far as is known, these techniques have been applied only to series of events equally spaced in time. In the analysis of joints, however, we have used these methods to study events occurring, not in time, but in an irregular sequence in space. This analysis is sequential in that the dip or strike values of the joints are considered in the order in which they are derived along the survey line.

The cumulative sums technique provides a rapid and a precise method of determining major trends above or below a particular reference value which is selected, and for ascertaining both the magnitude and location of these variations. The main uses of such an analysis method can be summarized briefly as follows:

(a) To detect general changes in joint orientation above and below the mean level of the joint orientation data;

(b) To determine where changes in joint orientation take place in the rock mass;

(c) To determine a reliable estimate of the mean orientation of the joints at any point along the surveyed pit face;

(d) To predict the average orientation of a particular joint set, or group of joints, in other parts of the mass where information is not available.

METHOD OF COMPUTATION

Basically, the approach is simple, consisting merely of subtracting a constant quantity, which at Nchanga was taken to be the mean value of either the strike or dip, from each value of strike or dip in the series, and accumulating the differences as each additional value is introduced. Successive accumulated differences are designated the "cumulative sums" of the original sequence of joint orientation values. The resulting graph of these sums is designated the "cumulative sum joint orientation plot".

When large numbers of joints are to be analysed, it is convenient to create cusum plots by computer methods, methods to which the analysis is ideally suited. Plots on the line printer, using a width of 100 characters, have proven to be an excellent medium for this method of analysis. In order to make the plots comparable, however, it is necessary to use the same cusum range and mean for all plots.

Let us suppose that we have a series of joint strike values acquired from a continuous detail joint survey. We will denote these values by $X_1, X_2, \ldots X_r$, recorded in that order along the pit face. From each X we subtract a reference value K, the mean strike of the joints. We then add these deviations to form a series of partial sums:

$$S_1 = X_1 - K$$

$$S_2 = (X_1 - K) + (X_2 - K) = S_1 + (X_2 - K)$$

$$S_3 = S_2 + (X_3 - K)$$

The general equation for the cumulative sums can thus be written as follows:

$$S_r = S_{r-1} + (X_r - K) = X_1 + X_2 + \ldots\ldots X_r - rK$$

$S_1, S_2, S_3, \ldots S_r$ is the cumulative sum series (or cusum) of the joint strike series. The plot of S against position in the sequence (S_r vs r) is the cumulative sum joint orientation plot.

The random spacing of the joints presents no problem, so long as the position, not the distance, in the sequence is used. It does not matter if the interval between observations changes. In this computation, strike or direction of dip data must be converted so that only values from either the 0° to 180° or the 180° to 360° intervals are calculated in the same analysis.

METHOD OF INTERPRETATION

If there is no trend in the strike of the joints, some of the difference terms (X_r - K) will be positive and others negative, with the result being that the cusum will be basically constant. But, if the current or local mean strike value is slightly greater than K (the overall mean), more of the differences will be positive, and the cusum will then be a straight line or curve sloping upwards. The reverse will occur if the current mean is less than K.

The actual distance of the plotted cusum curve from the horizontal is irrelevant; the interpretation is based exclusively on the average slope of the curve. The steeper the curve, the further the mean strike of the joints within any particular location is from the mean value K. The slope of the line (and hence the amount of deviation of the current mean strike from the overall mean value) can be easily calculated. The slope of the plotted line joining, let us say, the mth point and an nth point further along in the series indicates the average difference from the reference value of all the results from X_{m+1} to X_n inclusive. The mean strike ($\bar{X}$) over any interval of the cumulative sum joint orientation plot is given by

$$\bar{X} = K + \frac{\text{change in cumulative sum}}{\text{change in } n}$$

When conducting this type of analysis considerable care should be taken in selecting a suitable reference value (K). One important feature of this analysis method is that relatively small changes, say in the current mean value of the joint strike, will appear as clearly different slopes. However, changes from one positive value to another in the slope of the cusum plot are not nearly so discernable as a reversal of the sign of the slope, i.e. a change from a situation in which the mean strike of the joints is above the reference value, to one in which it is below. The reference value K should be chosen as a reasonable target from which the results are expected to vary. Also, erratic variations or "noise" in the data are smoothed out. This is a significant factor when looking for trends and patterns, particularly when analyzing data from natural phenomena such as joints.

This technique is best used to determine long-term trends. Interpretation becomes difficult if attempts are made to include short-duration effects.

COMPARISON WITH TIME TREND ANALYSES

Several techniques, adapted from time series analysis, have been used extensively to analyze sets of geological data which are arranged as a series in space [Harbough and Merriam (1)]. They include moving average methods, harmonic analysis, spectral analysis and auto-correlation. All but the first of these are concerned with acquiring information from rapid fluctuations present in all data.

The moving average techniques (including polynomial trend analysis) tackle a problem similar to that discussed here. However, they assume that the underlying variations sought are continuous functions and will smooth out any sudden breaks. The analyst is presented with a plethora of results which are difficult to interpret.

In contrast, cusums are best used to highlight step changes in the underlying function, and are excellent for displaying slow cyclic variations. A comparison of cusums with other techniques used to detect slow variations is given by Hurst et al (2). It is interesting to note that the cusum of a series of equally spaced events is a convenient aid in calculating the simple moving average, particularly when a number of base lengths are to be examined.

APPLICATION OF THE CUMULATIVE SUMS ANALYSIS AT NCHANGA

METHOD OF APPROACH

All joints occurring within 30° of either side of the tectonic fold axis (which for analysis purposes was taken to be 300°) were considered in the analysis (i.e. joints with a direction of dip of 0° to 60° and 180° to 240°). In order that the joint data be representative of different parts of the pit slope, the hanging-wall was sub-divided into 14 arbitrary areas of approximately similar size going from west to east.

Analysis Using One Mean

One cusums technique consisted of analyzing the direction of dip data of all joints within the limits defined. Those joints with direction of dip of 0° to 60° were converted to 180° to 240° by adding 180° to their respective values. Thus, all joints could be analyzed together in the 180° to 240° range.

An example of the method of interpretation of the cumulative sums is shown in Fig. 3. The actual direction of dip orientations, as calculated from strike measurements in the field, are shown in Fig. 3(a). The resulting cumulative sums joint orientation plot of the raw data in Fig. 3(a) is shown in Fig. 3(b). Fig. 3(c) shows a Manhattan diagram, depicting the degree of deviation of the current mean strike above or below the overall mean strike K, according to the curves plotted in Fig. 3(b). See Fig. 4 for details of Fig. 3.

In Fig. 5, Manhattan diagrams of the cumulative sums of the analyses of the entire hanging-wall area that was surveyed, are shown. The various bench levels and subdivided areas of the hanging-wall (i.e. 1 to 14) are denoted accordingly. The bottom Manhattan diagram in Fig. 5 gives, for each of the 14 areas, the current mean deviation of the strike* of the joints about the mean. This is determined by calculating the mean deviation for all the benches occurring in a particular area.

Analysis Using Four Means

This analysis technique consisted of determining the cusums of both the direction of dip and angle of dip of the joints dipping to the northeast and of those dipping to the southwest. The respective mean direction of dip and mean angle of dip for each group were used. Hence, four K values are required, giving four cusum plots. The four K values applying to the Nchanga data are as follows:

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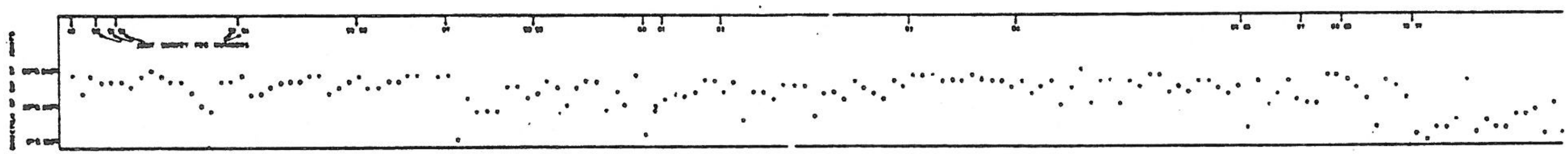
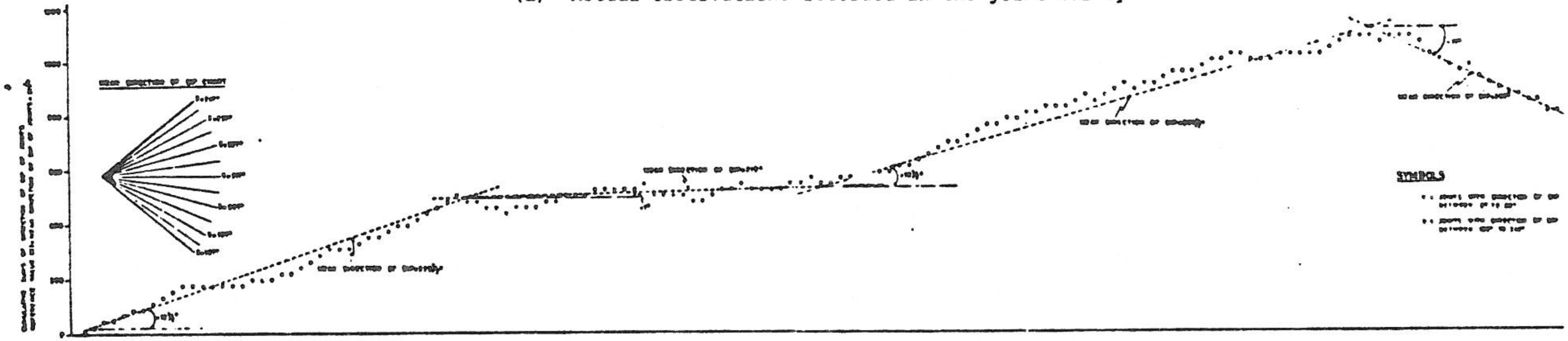
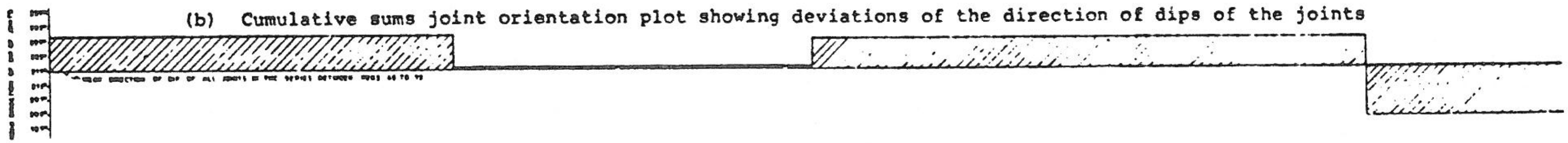
* The cumulative sums plots in Fig. 3, of direction of dip data, is converted to actual strike values in Fig. 5 for purposes of clarity.



(a) Actual observations recorded in the joint survey



(b) Cumulative sums joint orientation plot showing deviations of the direction of dips of the joints



(c) Manhattan diagram showing the current mean direction of dip of the joints at progressive positions along the survey line

Fig 3 Illustration of cumulative sums technique for analysing joint direction of dip trends

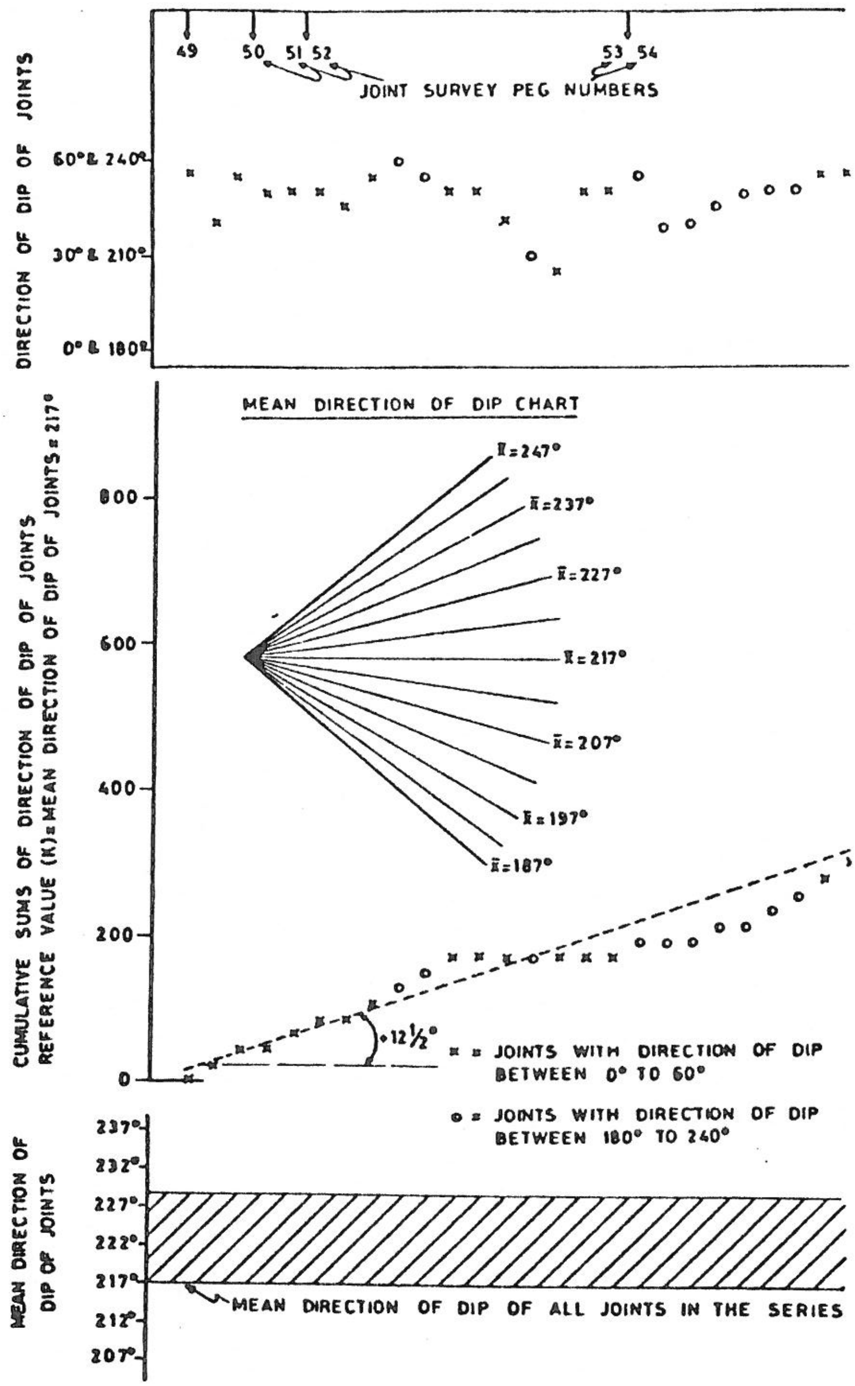


Fig. 4 Enlarged portion of the left side of Fig. 3 showing detail clearer

Joints Considered	Orientation Data	K Value
Direction of Dip 0° - 60°	Strike	310°
" " " 0° - 60°	Angle of Dip	74°
" " " 180° - 240°	Strike	306°
" " " 180° - 240°	Angle of Dip	78°

The general results are similar to those illustrated in Fig. 3, except that four cusum joint orientation plots are produced. Two plots apply to strike and two to angle of dip, although only one strike chart and one angle of dip chart are required. Four Manhattan diagrams must be considered in the same manner. The Manhattan diagrams of this analysis are given in Fig. 6. See Fig. 7 for some details of Fig. 6.

Applying this general form of cumulative sums analysis, boundaries to structural regions (i.e. areas of similar jointing characteristics in a statistical sense) were also determined. Cusum techniques used for this purpose will be published elsewhere.

DISCUSSION OF THE RESULTS

Counter-clockwise Rotation

In Fig. 5 it can be seen that the mean strike of all the joints is 307°. There is, however, a counter-clockwise rotation in the current mean strike, going from west to the east side of the hanging-wall. It rotates from about 317° in areas 1 and 2 to about 297° in areas 9 to 14. Around areas 5 and 6 the current mean strike is about the same as K.

Effects of Major Fault

As shown in Fig. 5, the rate of change of this rotation is greatest in areas 3 to 7. In Fig. 6, where the northeast and southwest dipping joints are analyzed separately, it can be seen that this phenomenon is due largely to the rotation

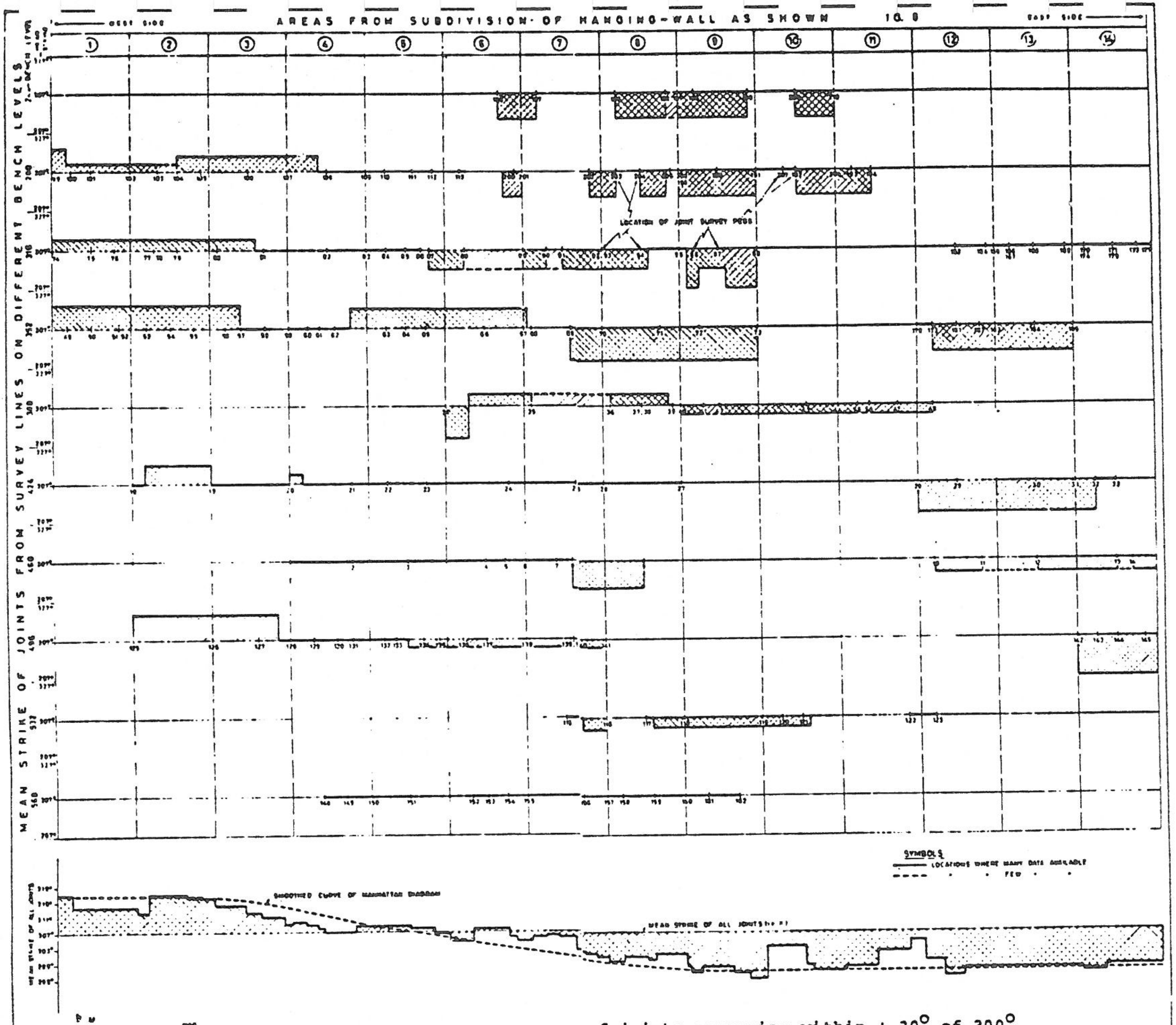


Fig. 5 Manhattan diagrams of strike trends of joints occurring within $\pm$ 30° of 300°

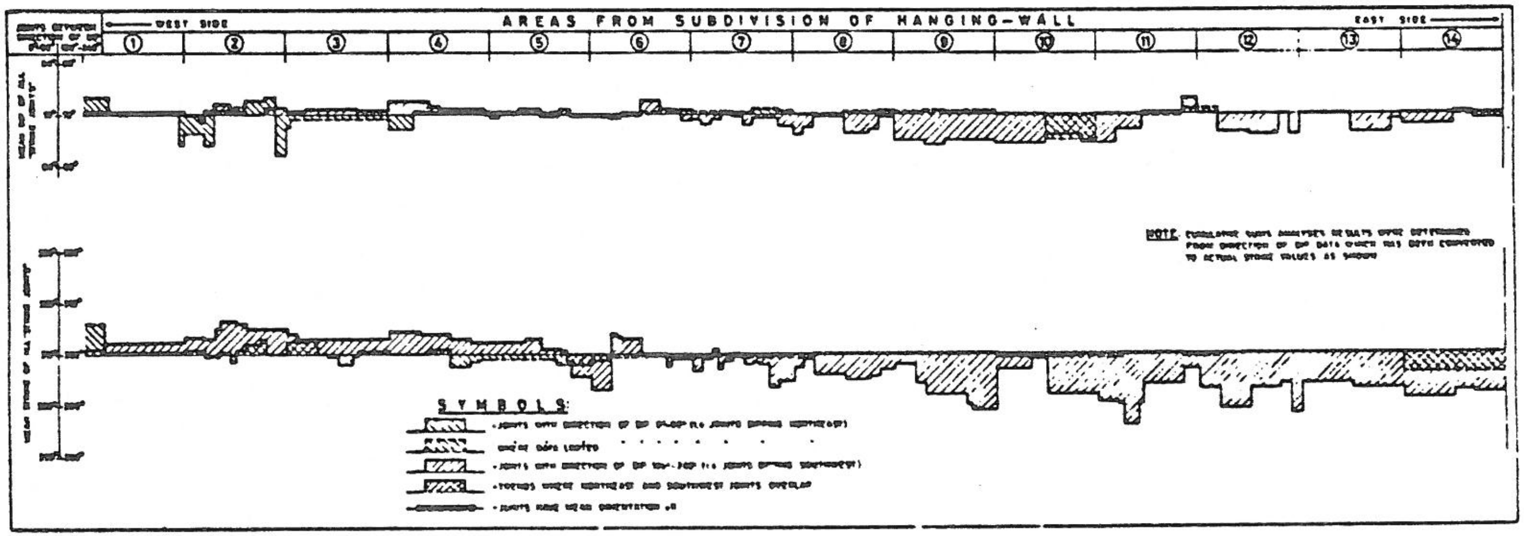


Fig. 6 Manhattan diagrams of the mean values of the cumulative sums analysis of strike and dip trends of "strike joints" dipping northeast and southwest of the estimated regional tectonic structural axis (300°) at Nchanga

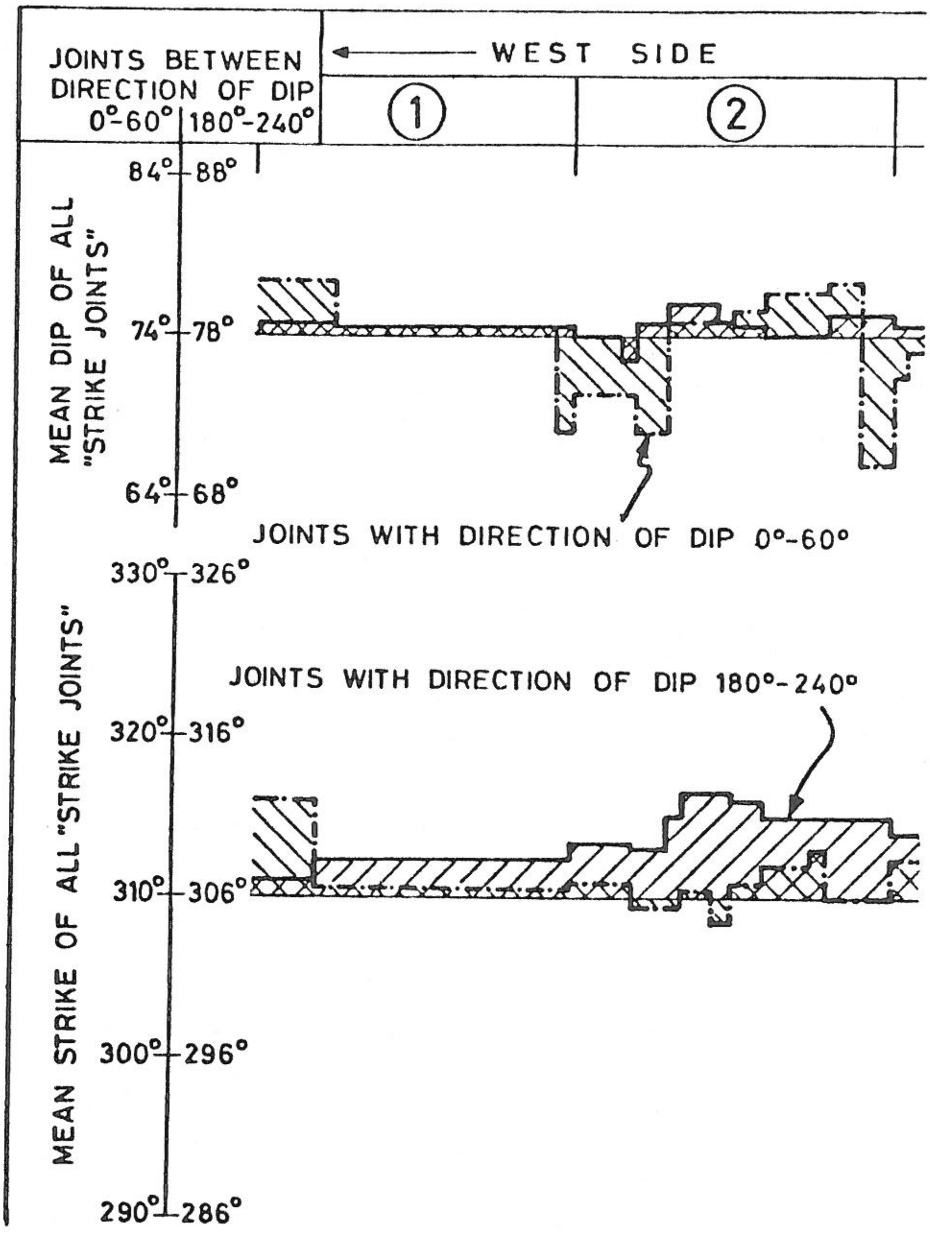


Fig. 7 Enlarged portion of the left side of Fig. 6 showing detail clearer

of the southwest dipping joints. This rotation is due to a major fault, the only major fault occurring in the area considered. This fault strikes 320^{o} and dips 80^{o} to 85^{o} SW. The vertical component of net slip is about 80 ft, the downthrow being to the southwest.

In Fig. 6 the northeast dipping joints, with respect to both strike and dip, vary only slightly about the mean. Also, they decrease in frequency going eastwards, becoming negligible beyond area 7. This indicates, along with their angular relationship to the fault, that these joints are probably feather fractures which have developed sympathetic to the fault.

The southwest dipping joints, on the other hand, are significantly above the mean on the west side of the pit. Here, sympathetic fracturing parallel to the fault has swung the current mean strike slightly towards that of the fault. Further east, however, the southwest dipping joints rotate counter-clockwise past the mean. Beyond area 9, where the influence of the fault is negligible, and where only tectonic forces appear to have been significant in causing the existing joints, they maintain a remarkably consistent current mean strike of about 295^{o}.

Plots in Fig. 8 of both (1) the percentage and (2) the number of joints per foot (i,e, joint intensity) of northeast and southwest dipping joints occurring in each of the areas 1 to 14, provide convincing additional evidence of the conclusions above. Fig. 8(a) indicates that the percentage of northeast dipping joints is considerably greater on the west side of the pit and decreases rapidly, becoming negligible east of area 7. The opposite is true for the southwest dipping joints. Area 4 is the changing point where one or the other dominates. In Fig. 8(b) the influence of the fault can be seen clearly. The intensity of the northeast dipping joints is excessively high in areas 1 to 4,

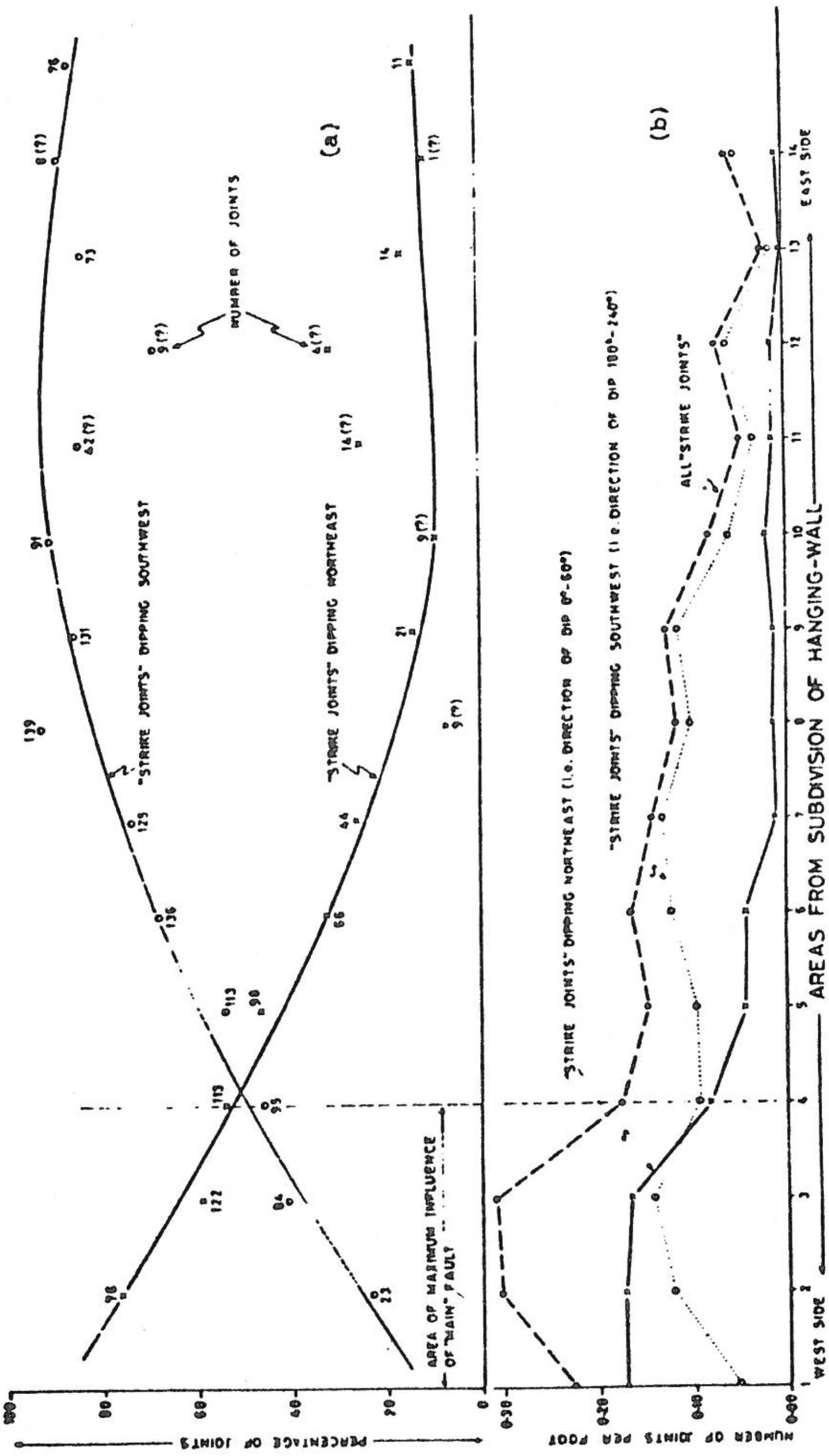


Fig. 8 Plot of the percentage (a) and frequency (b) of "strike joints" dipping southwest and northeast of the synclinal axis for different areas across the hanging-wall

but negligible beyond area 7. The southwest dipping joints, however, increase in frequency only slightly across the fault-affected area and maintain a fairly constant, though slightly decreasing, frequency going from west to east beyond this area.

Regional Joint Pattern

East of area 7 additional joints of an anomalous nature, (i.e. joints other than those originating through regional tectonic processes) are not evident. Thus, it must be assumed that those remaining, namely the joints occurring outside of the limits of the fault influence, are of the regional joint pattern. These are exclusively the southwest dipping joints.

Not only are these joints part of the regional pattern, but they represent greater than 80 per cent of the regional pattern (see Fig. 2 and 8). Greater than 80 per cent of the regional joint pattern, therefore, can be defined, approximately, as having an average strike of 295° and an average dip of 72° SW to 76° SW.

Genesis of Jointing and the Tectonic Process at Nchanga

With the general joint distributions in Fig. 2 and other structural relationships, the genesis of this dominant regional set, and the conditions during which both folding and this jointing took place, can be postulated.

Since definite sets of either one or both conjugate shear joints are not evident, and the intermediate tension joint set is essentially absent, the sedimentary rocks in the area appear to have yielded, at least initially, by plastic deformation or flowage and recrystallization in contrast to brittle fracture. The first and major form of brittle fracture (i.e. southwest dipping joints) appears to have developed as a result of elastic rebound of the originally highly compressed materials after both the temperature and pressure subsided. The southwest dipping joints, therefore, appear to be tension joints, having developed due to elastic rebound after the principal tectonic force had terminated. The principal form of deformation was that of crustal shortening. The type of folding was related primarily to those of horizontal tectonics, i.e. to processes of deformation wherein the maximum principal stress (tectonic stress) acted horizontally.

EXTRAPOLATION OF JOINT DATA

For purposes of making slope stability evaluations for the pit faces advancing both northwards and eastwards at Nchanga, the question of the reliability of applying information acquired from the existing hanging-wall slope to other parts of the mass where information is not available and where the advancing and final pit faces are to be located, is an important consideration. If any degree of confidence is to be achieved in proposing slope designs based to a large extent on these results, it must be shown whether the joint characteristics can be expected to be the same or to differ, and in what way to differ, in other parts of the mass where information is not available. The question of the extrapolation of joint properties when designing engineering structures in rock, and basic considerations relating to this problem, are discussed by Piteau (7).

Results in Fig. 6 show that both the current mean strike and current mean angle of dip are, statistically speaking, remarkably consistent east of area 8. This is particularly so with respect to the current mean strike. In either case, the deviation about the overall mean strike and mean dip orientation in this area is plus or minus three degrees. The history of folding in the majority of the syncline, and at least within the confines of the proposed final pit limits, is expected to be reasonably similar. Since the joints in question are genetically related to this folding process (in that they are rebound features which developed normal to the principal tectonic stress), based on the results of the cumulative sums analysis described above (see Fig. 6), there is good reason to believe that southwest dipping joints with similar orientations will exist in the proposed eastern extension areas of the pit.

For comparative purposes it is fortunate that at Nchanga an extensive joint survey had been conducted on the hanging-wall of the pit in 1966. The pit face at the time was 250 ft to the south of its present location, but the joint survey was conducted at approximately the same elevation and same relative location as that of the present survey. Hence, an ideal situation exists for determining whether the joint patterns are similar between the two survey lines and, accordingly, whether extrapolation of such structures is reasonable over this same distance in the opposite direction.

The 1966 survey results were available on stereographic projections, hence the peak concentration of the southwest dipping joints was easily measured. This information was compared directly to the cusum results for the respective areas across the pit. Except for minor variations, in general a remarkable similarity was found between the two separate survey results. Since the history of deformation is expected to be similar within the final pit limits, this indicated that the results from the present analysis would probably apply also behind the existing face in areas where the advancing pit faces are to be eventually located. These results also confirmed the conviction that the joint trends will be maintained in areas further east of the pit in which the extension is proposed.

CONCLUSIONS

The cumulative sums technique, illustrated with particular reference to an extensive joint analysis of the Nchanga open pit hanging-wall, provides an efficient and definitive method of examining joint dip and/or joint strike data in the order in which the joints are derived along the survey line. Unlike most joint analysis methods, this technique smooths out "noise" effectively. Also, both step changes in the underlying function and slow cyclic variations are readily displayed.

Basically, it is used to determine:

(a) the current deviation of either the joint dip or strike above or below some level of the orientation data or reference value (K) (i.e. in the Nchanga analysis K was taken to be the mean of the orientation data used);

(b) where these particular changes take place along the pit face; and

(c) the current mean orientation or simple moving average at any point in the consecutive sequence of the joints.

The behavior of a particular group of joints can be ascertained with respect to such characteristics as imperceptible rotation, both in the horizontal (i.e. strike) and vertical (i.e. dip) planes. In that the plots depicting this behavior are statistically significant, they can assist in predicting whether the information from the exposed pit face can be extrapolated with confidence to other parts of the mass where information is limited, but where pit slopes are to be eventually located. In this respect a knowledge of the geological history and the genetic relationship of the joints to the regional structure is important.

ACKNOWLEDGEMENTS

The authors extend their thanks to Nchanga Consolidated Copper Mines Limited - Chingola Division, for the co-operation which was given throughout the course of this work. The authors particularly wish to express their appreciation to C. Kovacevic, cartographer, for drawing not only these figures but numerous others in the Nchanga report; to A. Carbray, for helping to process the data; to F. Chisela, for supervising the survey; and to J. Hellings, for his general co-operation. Finally, the authors acknowledge with grateful thanks J. E. Jennings, O. K. H. Steffen and A. MacG. Robertson, with whom the senior author worked closely in recent years and who, although not directly involved with the development of this new technique, contributed significantly to important earlier development work, of which the results in this paper are an outgrowth.

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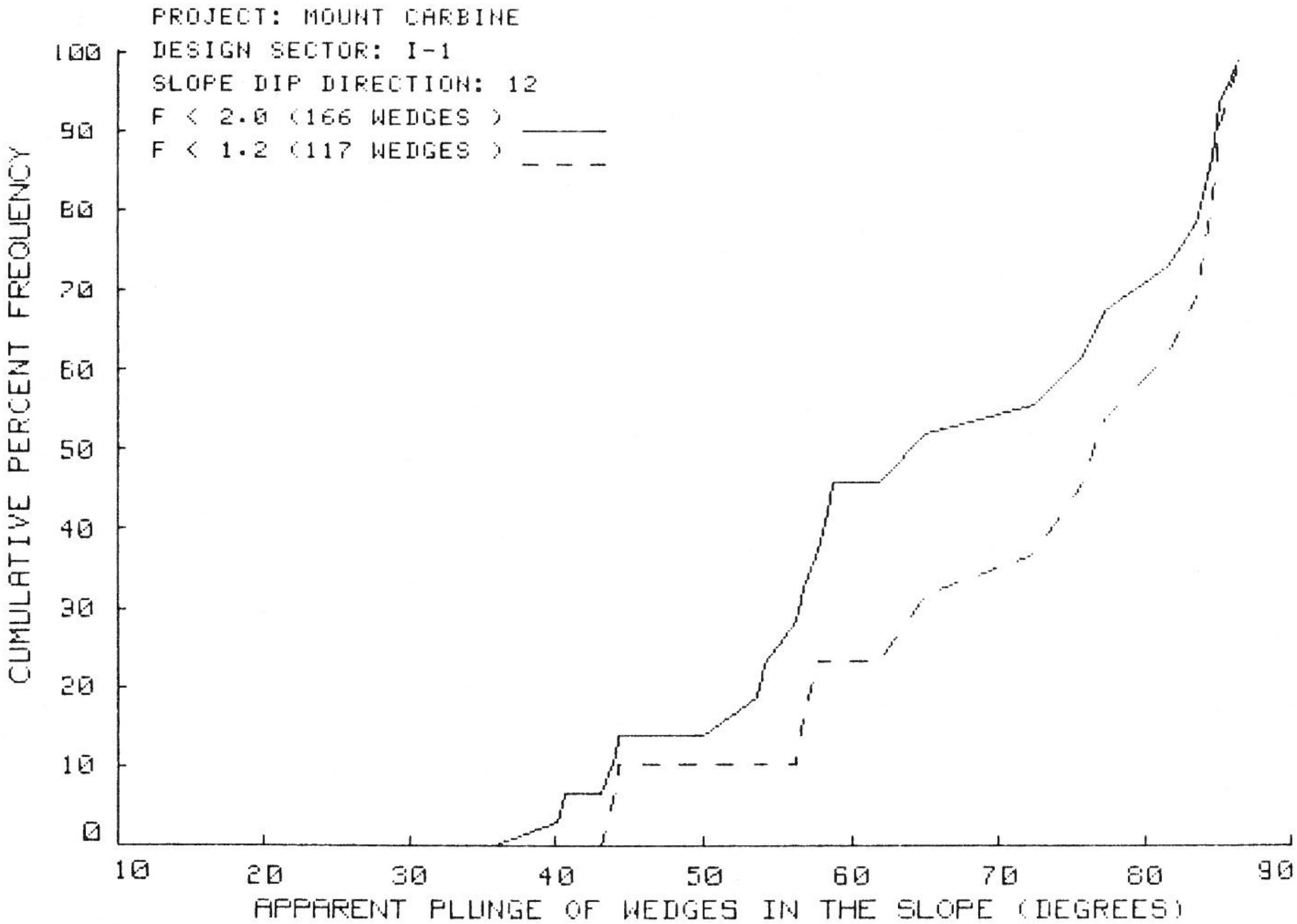
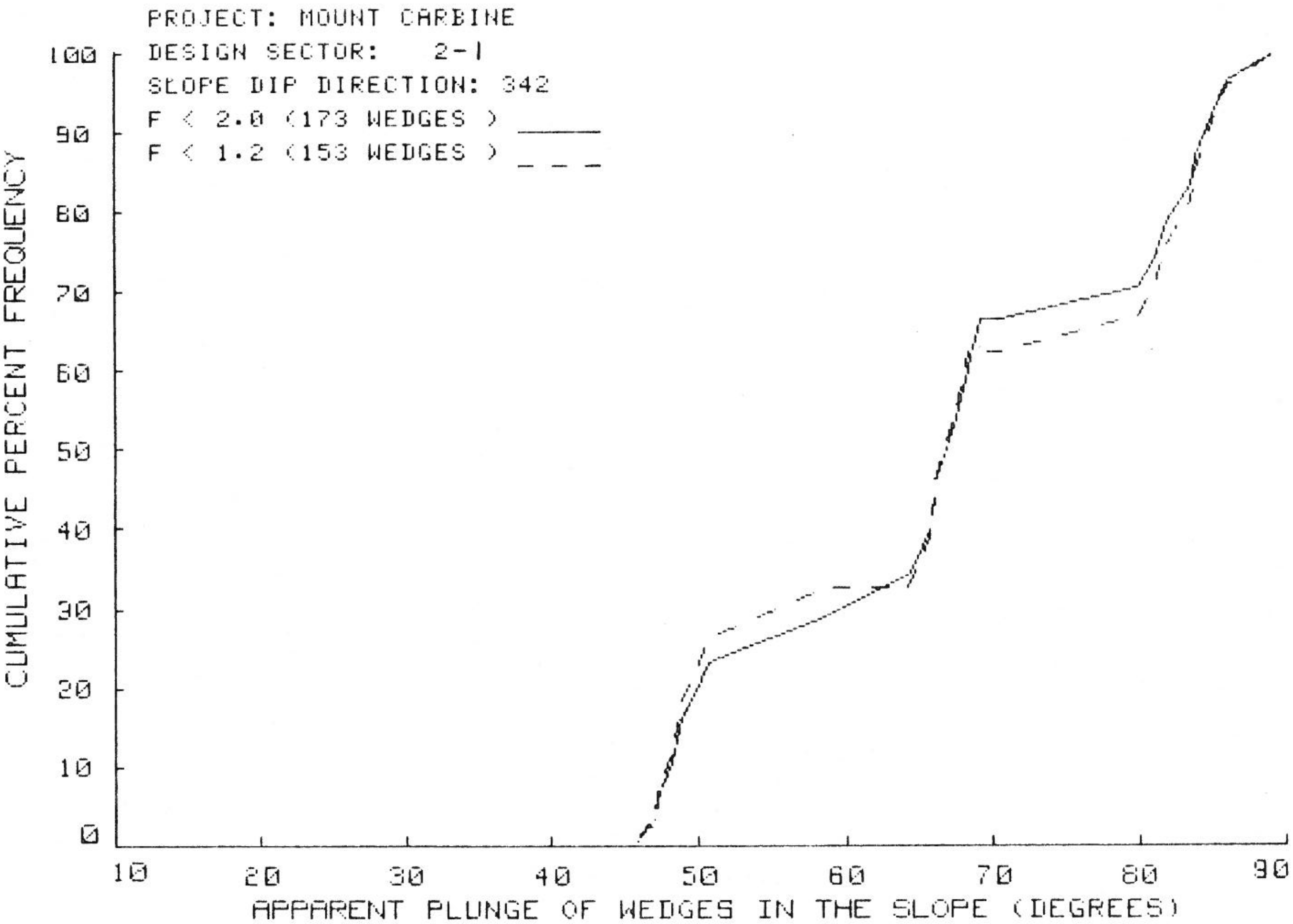
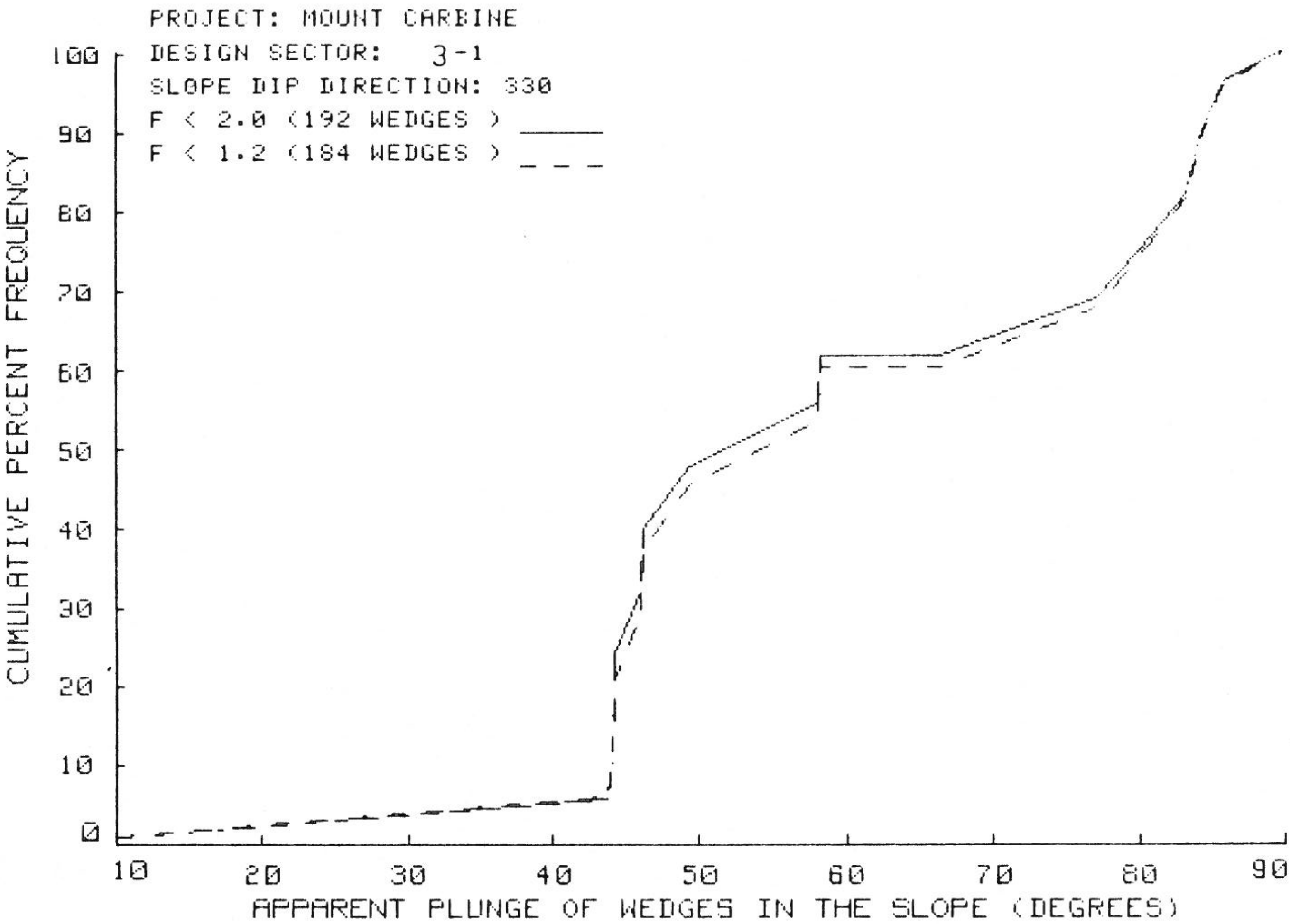
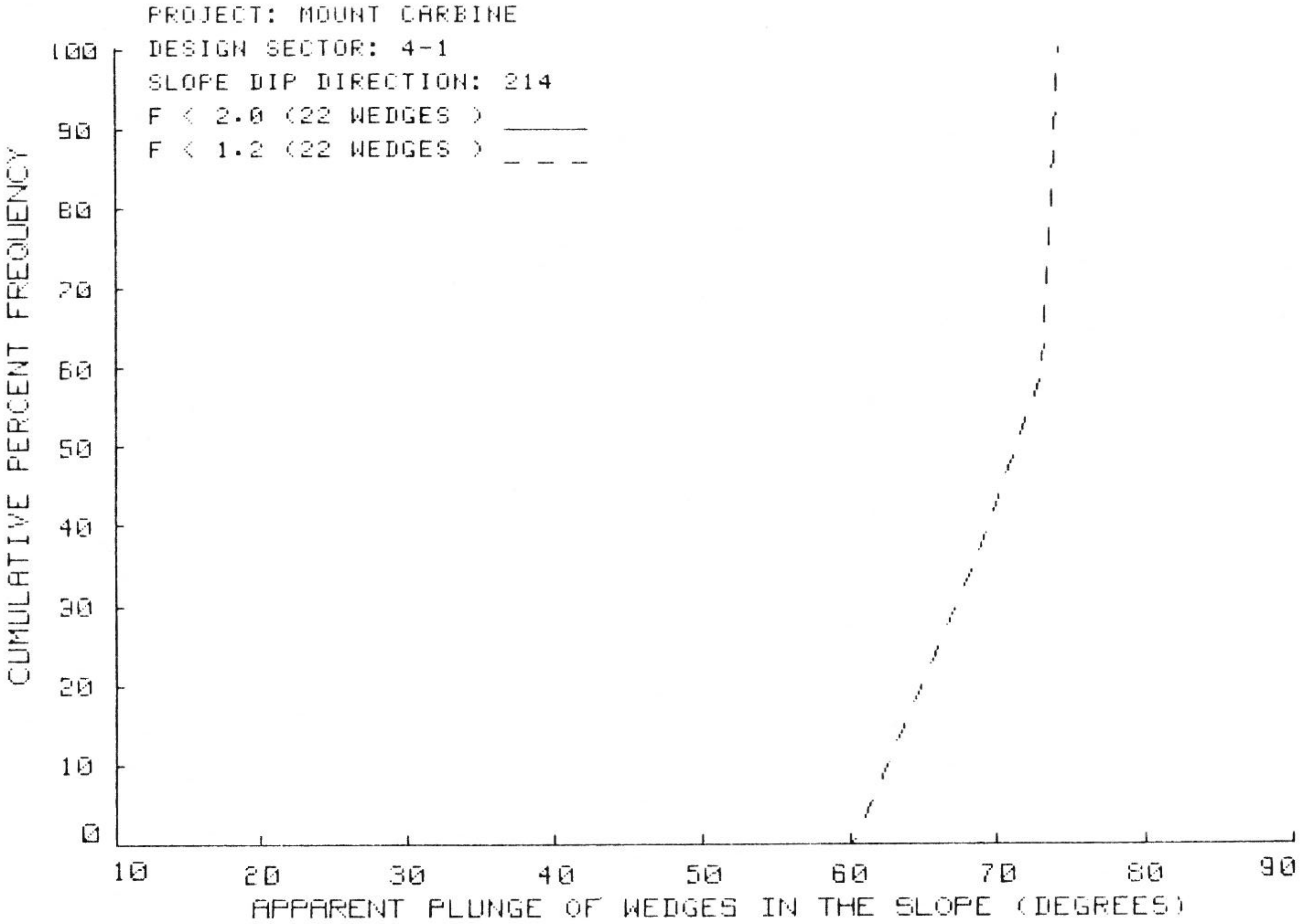
APPENDIX D

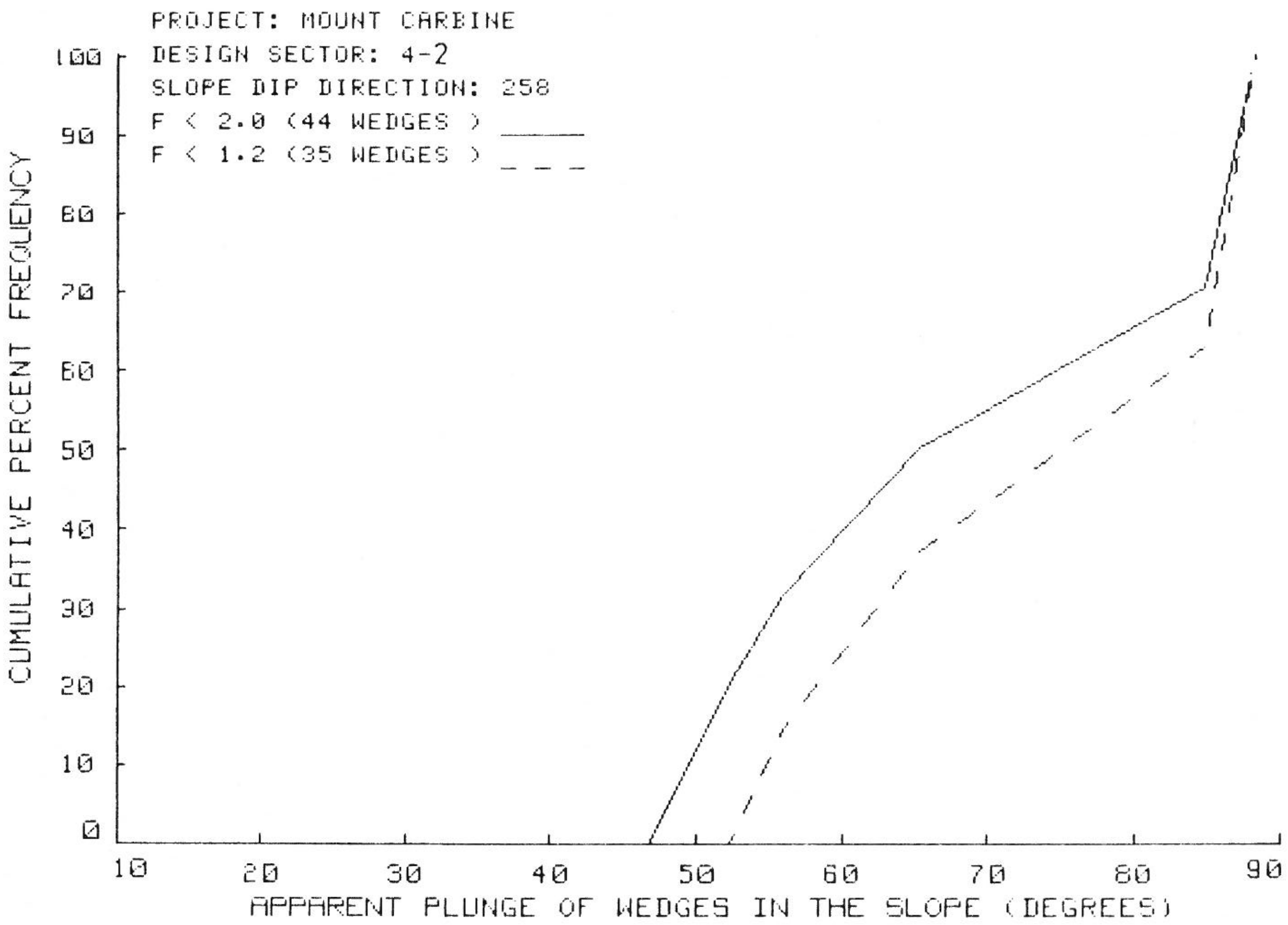
CUMULATIVE FREQUENCY PLOTS
OF KINEMATICALLY PROBABLE
FAILURES FOR DESIGN SECTORS
IN THE OPEN PIT

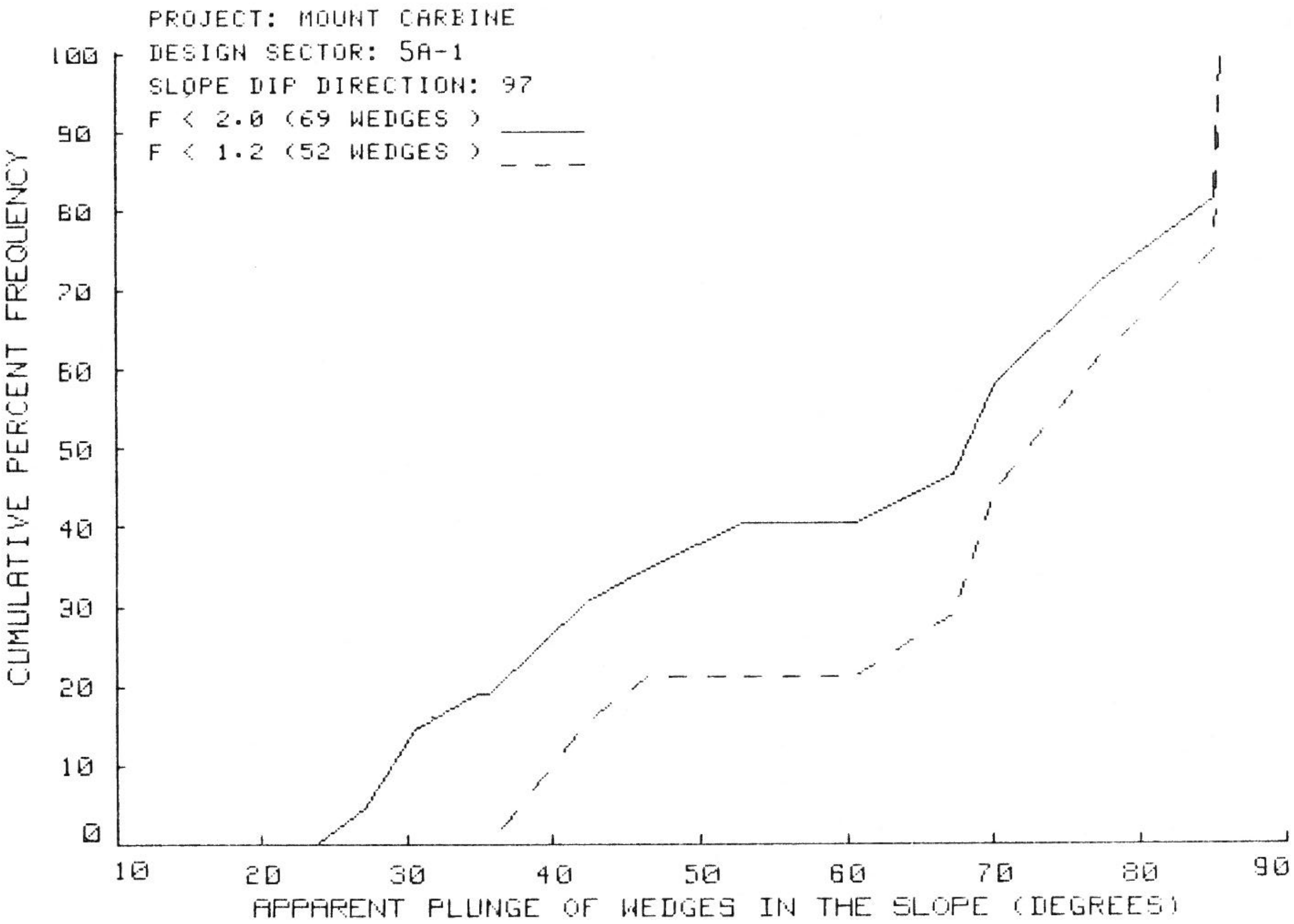

PROJECT: MOUNT CARBINE
DESIGN SECTOR: I-1
SLOPE DIP DIRECTION: 12
F < 2.0 (166 WEDGES)
F < 1.2 (117 WEDGES)
CUMULATIVE PERCENT FREQUENCY
100
90
80
70
60
50
40
30
20
10
0
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

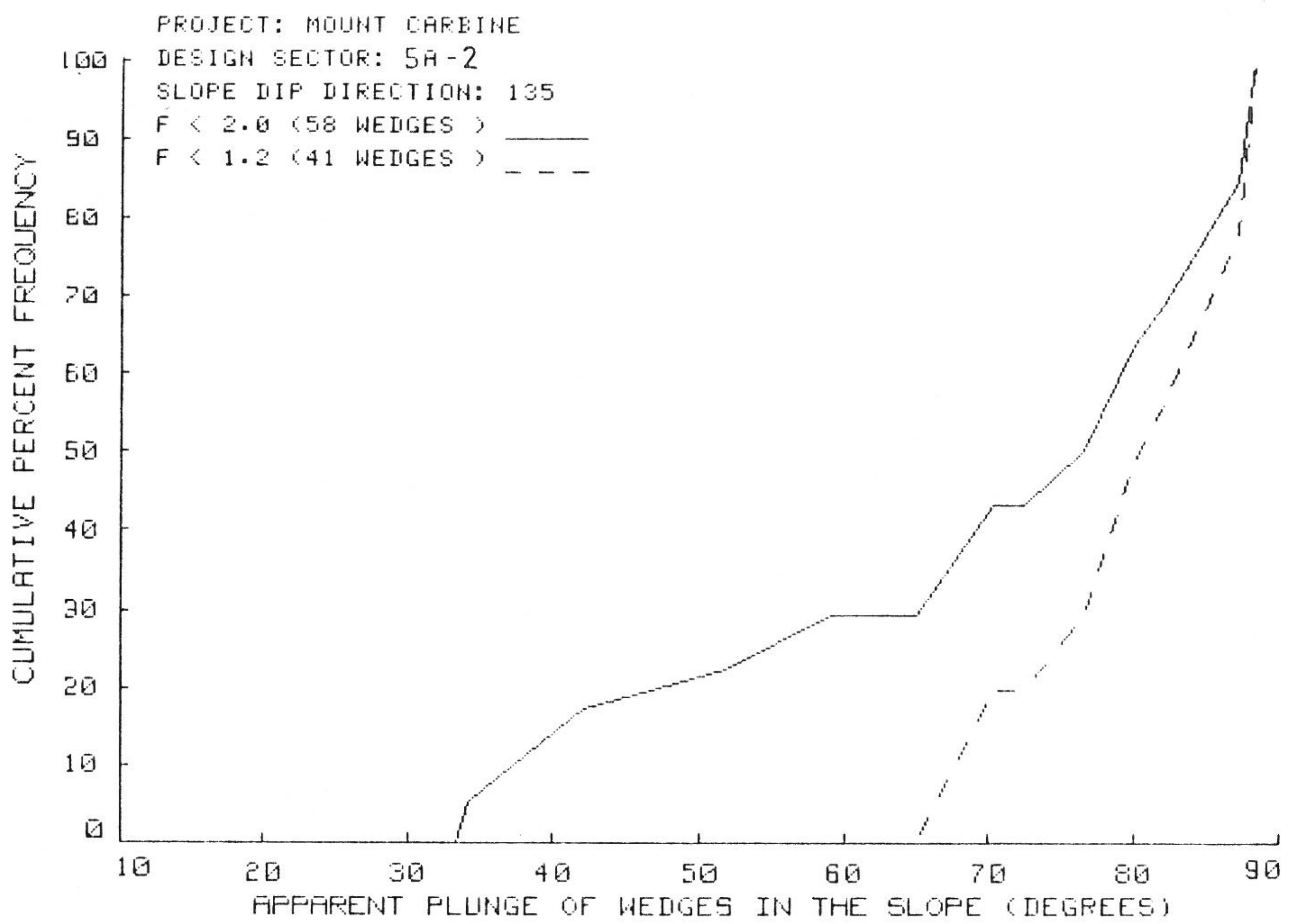

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 2-1
SLOPE DIP DIRECTION: 342
F < 2.0 (173 WEDGES)
F < 1.2 (153 WEDGES)
CUMULATIVE PERCENT FREQUENCY
100
90
80
70
60
50
40
30
20
10
0
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

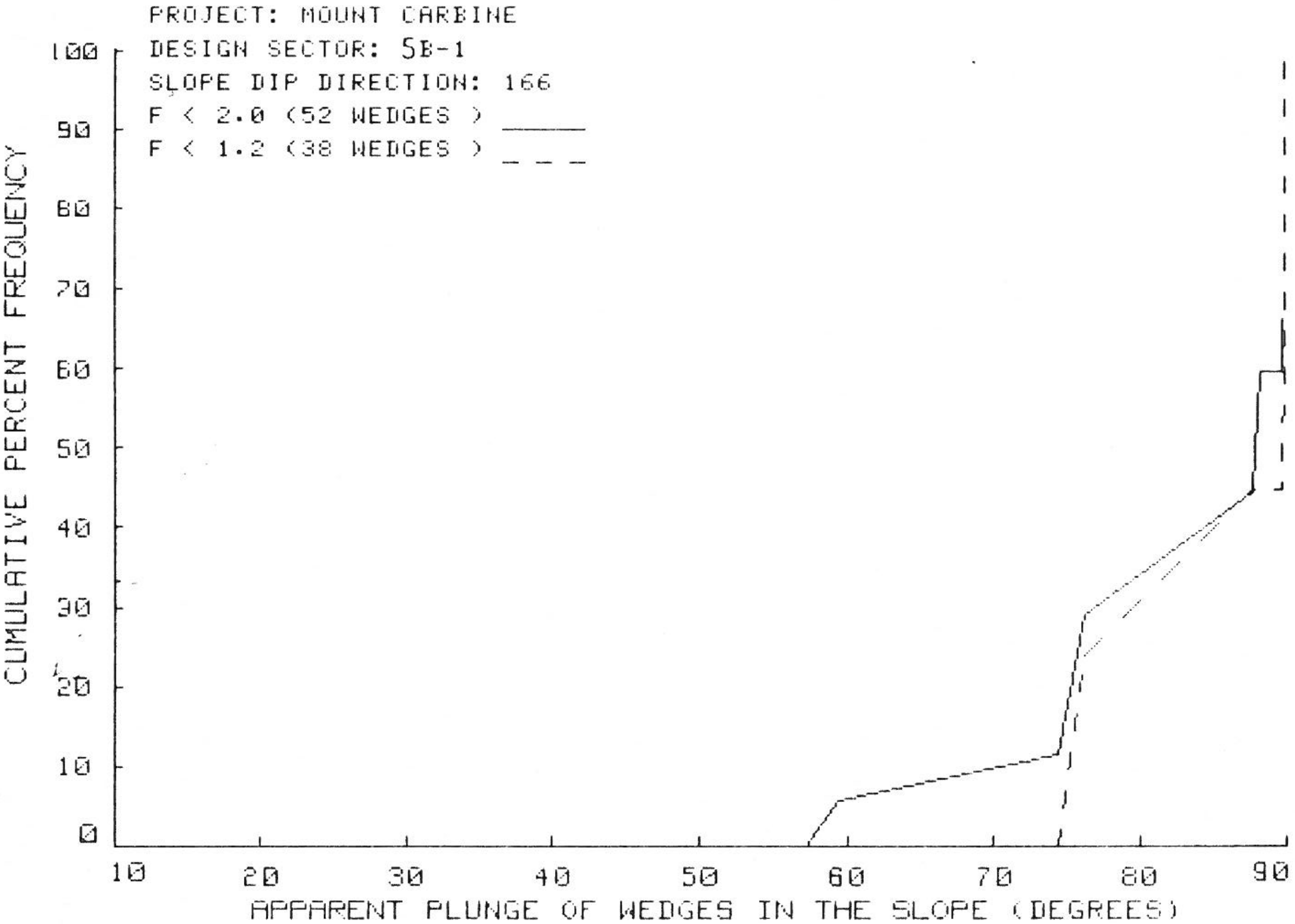

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 3-1
SLOPE DIP DIRECTION: 330
F < 2.0 (192 WEDGES)
F < 1.2 (184 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

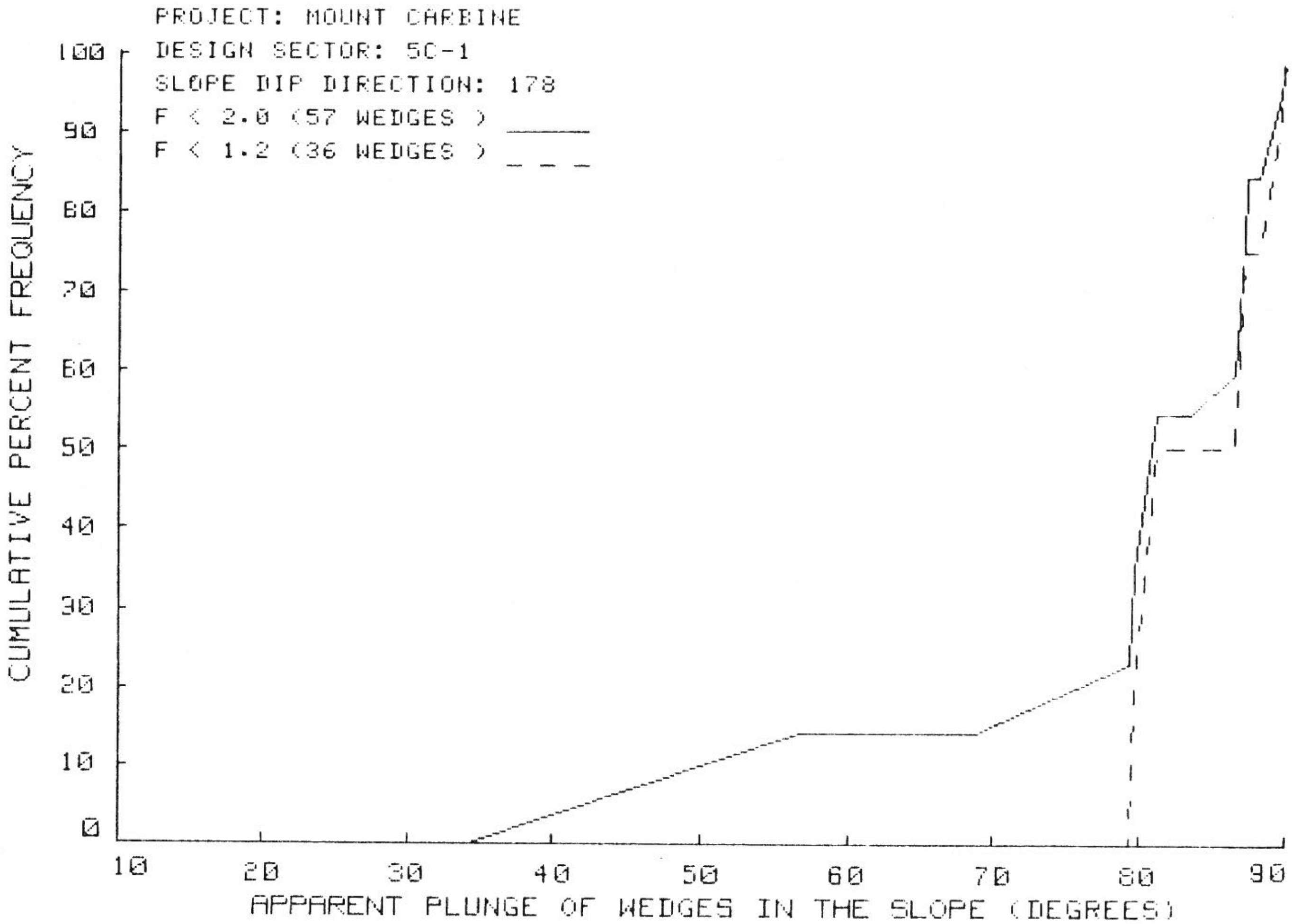

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 4-1
SLOPE DIP DIRECTION: 214
F < 2.0 (22 WEDGES)
F < 1.2 (22 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

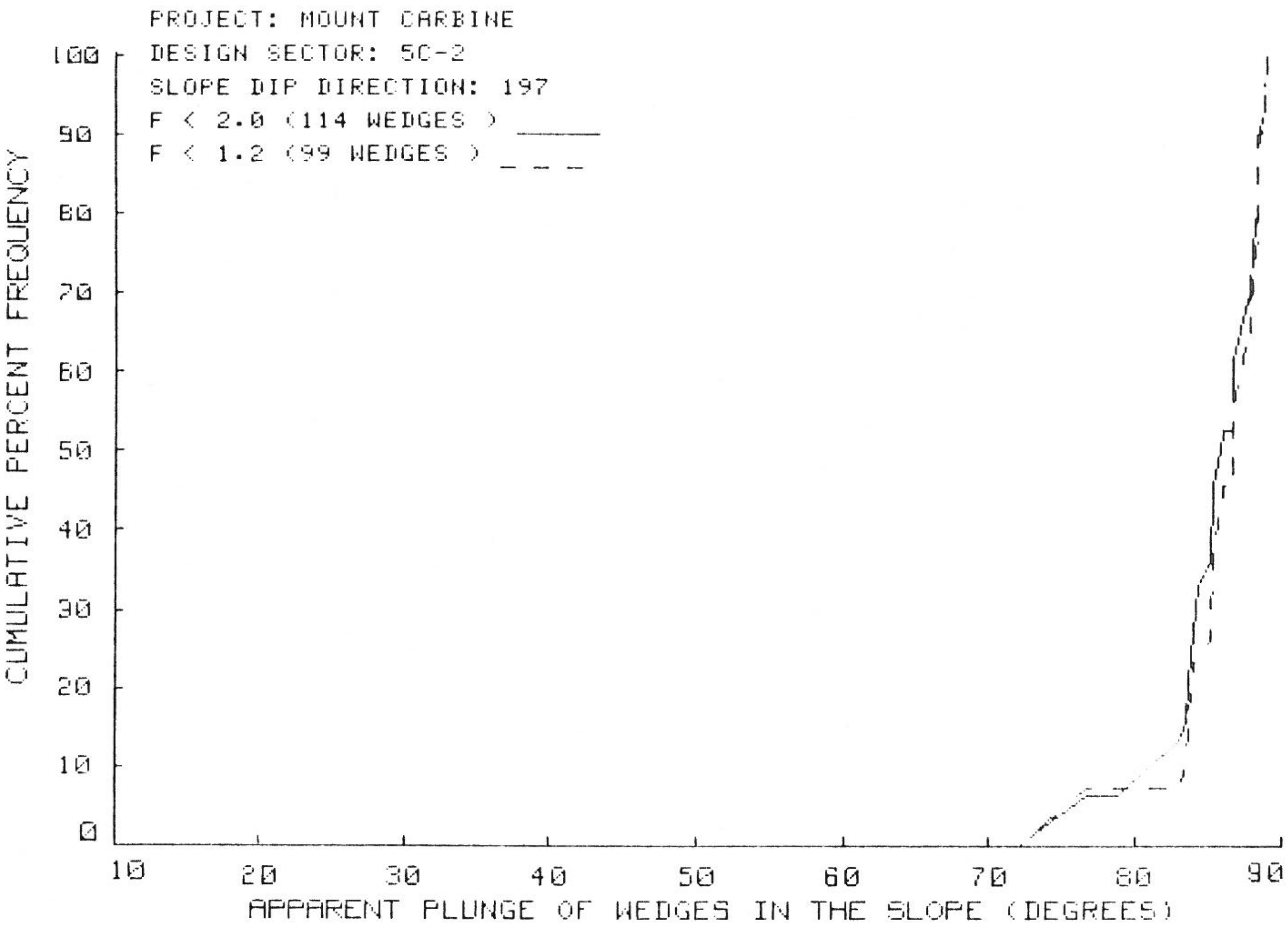

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 4-2
SLOPE DIP DIRECTION: 258
F < 2.0 (44 WEDGES)
F < 1.2 (35 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

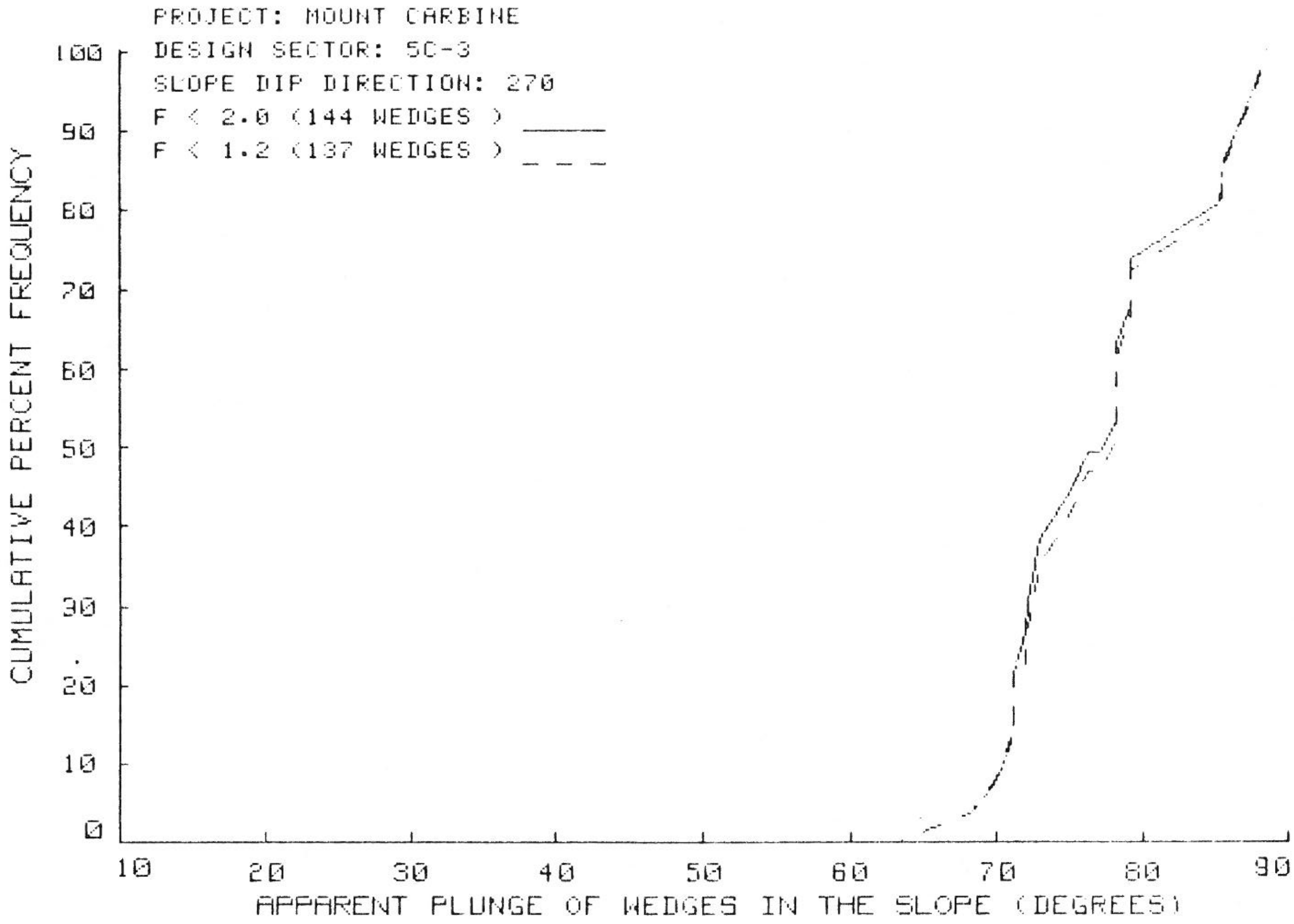

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 5A-1
SLOPE DIP DIRECTION: 97
F < 2.0 (69 WEDGES)
F < 1.2 (52 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)


PROJECT: MOUNT CARBINE
DESIGN SECTOR: 5A-2
SLOPE DIP DIRECTION: 135
F < 2.0 (58 WEDGES)
F < 1.2 (41 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

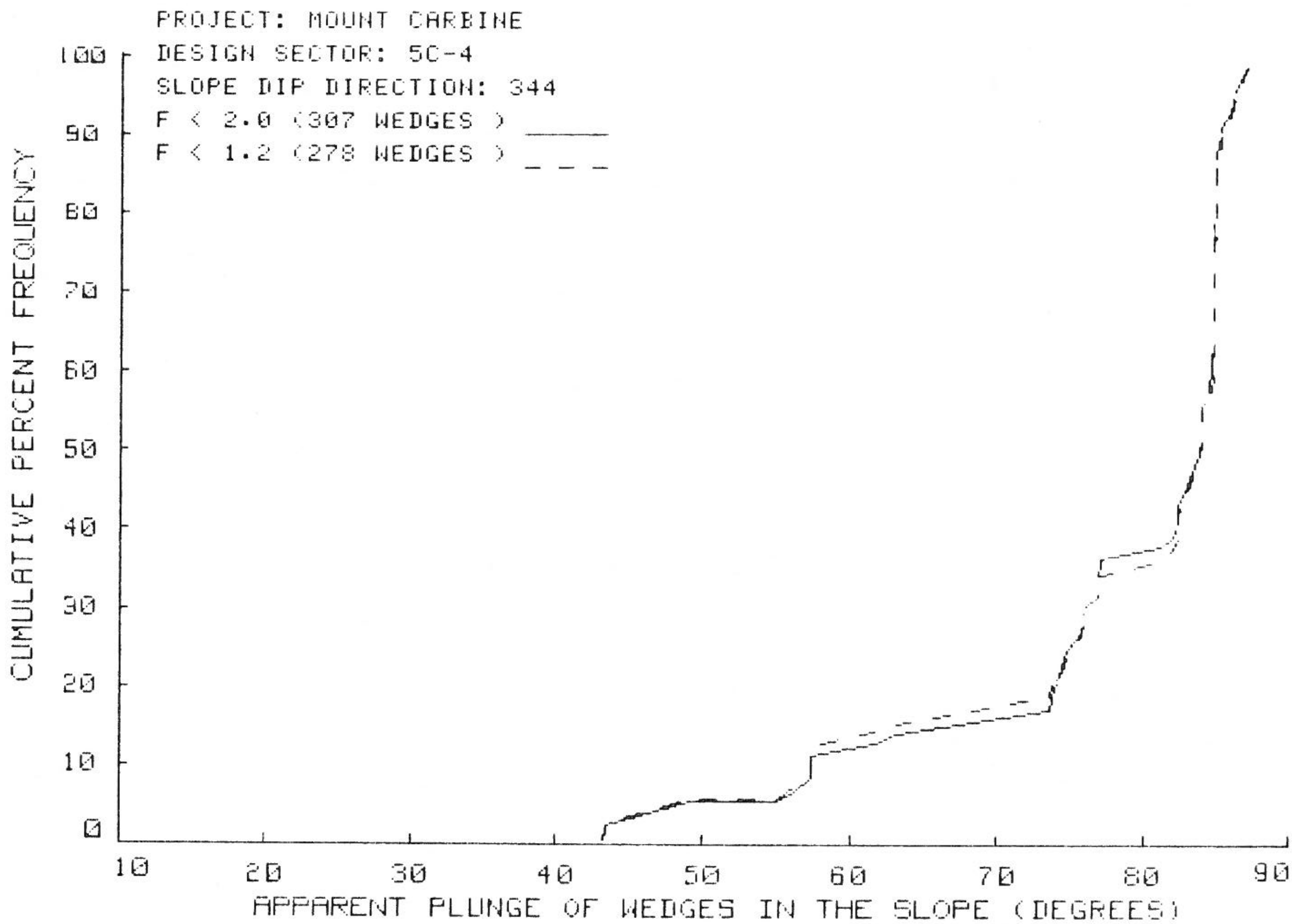

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 5B-1
SLOPE DIP DIRECTION: 166
F < 2.0 (52 WEDGES)
F < 1.2 (38 WEDGES)
CUMULATIVE PERCENT FREQUENCY
100
90
80
70
60
50
40
30
20
10
0
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

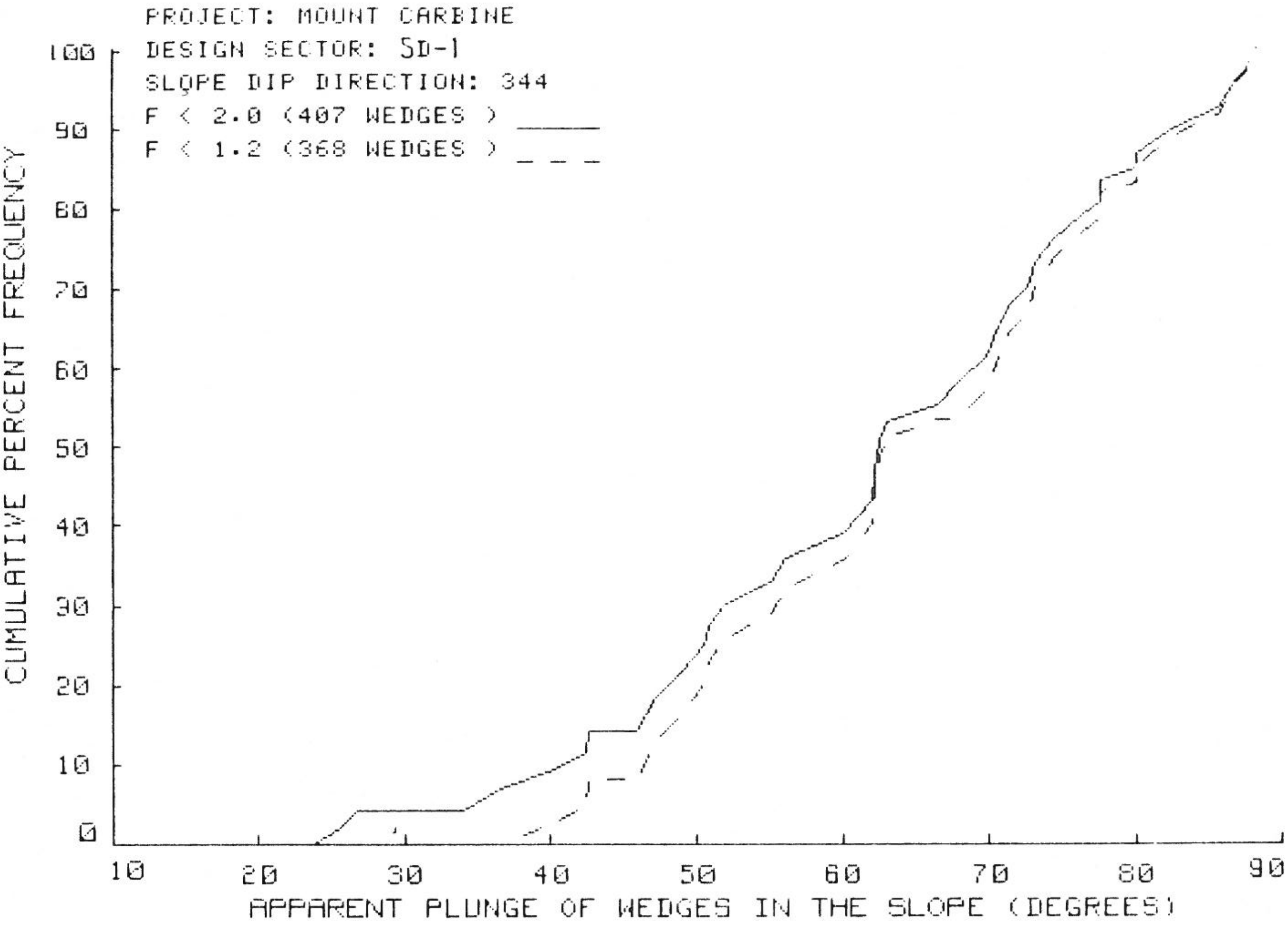

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 5C-1
SLOPE DIP DIRECTION: 178
F < 2.0 (57 WEDGES)
F < 1.2 (36 WEDGES)
CUMULATIVE PERCENT FREQUENCY
100
90
80
70
60
50
40
30
20
10
0
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)


PROJECT: MOUNT CARBINE
DESIGN SECTOR: 5C-2
SLOPE DIP DIRECTION: 197
F < 2.0 (114 WEDGES)
F < 1.2 (99 WEDGES)
CUMULATIVE PERCENT FREQUENCY
100
90
80
70
60
50
40
30
20
10
0
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

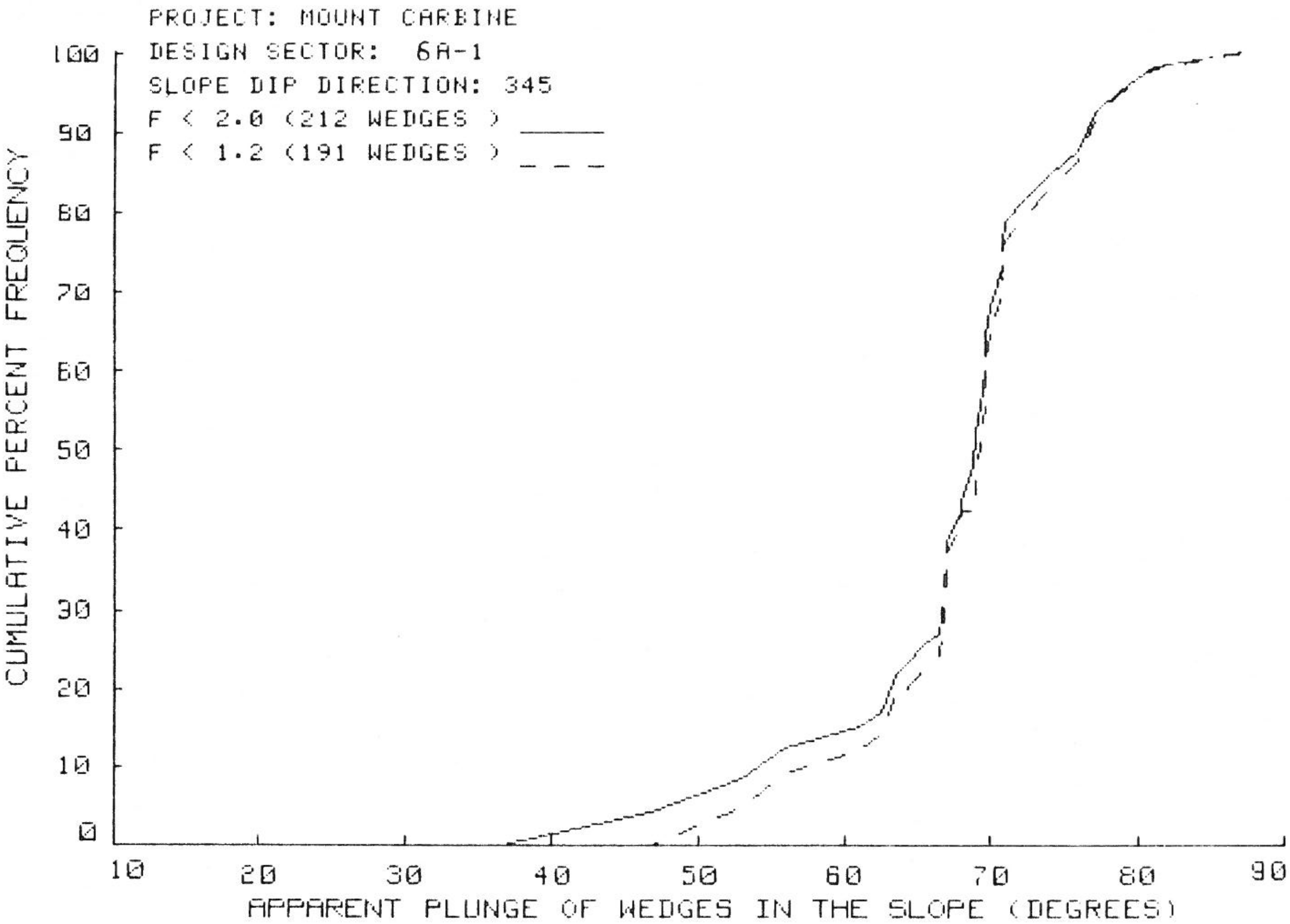


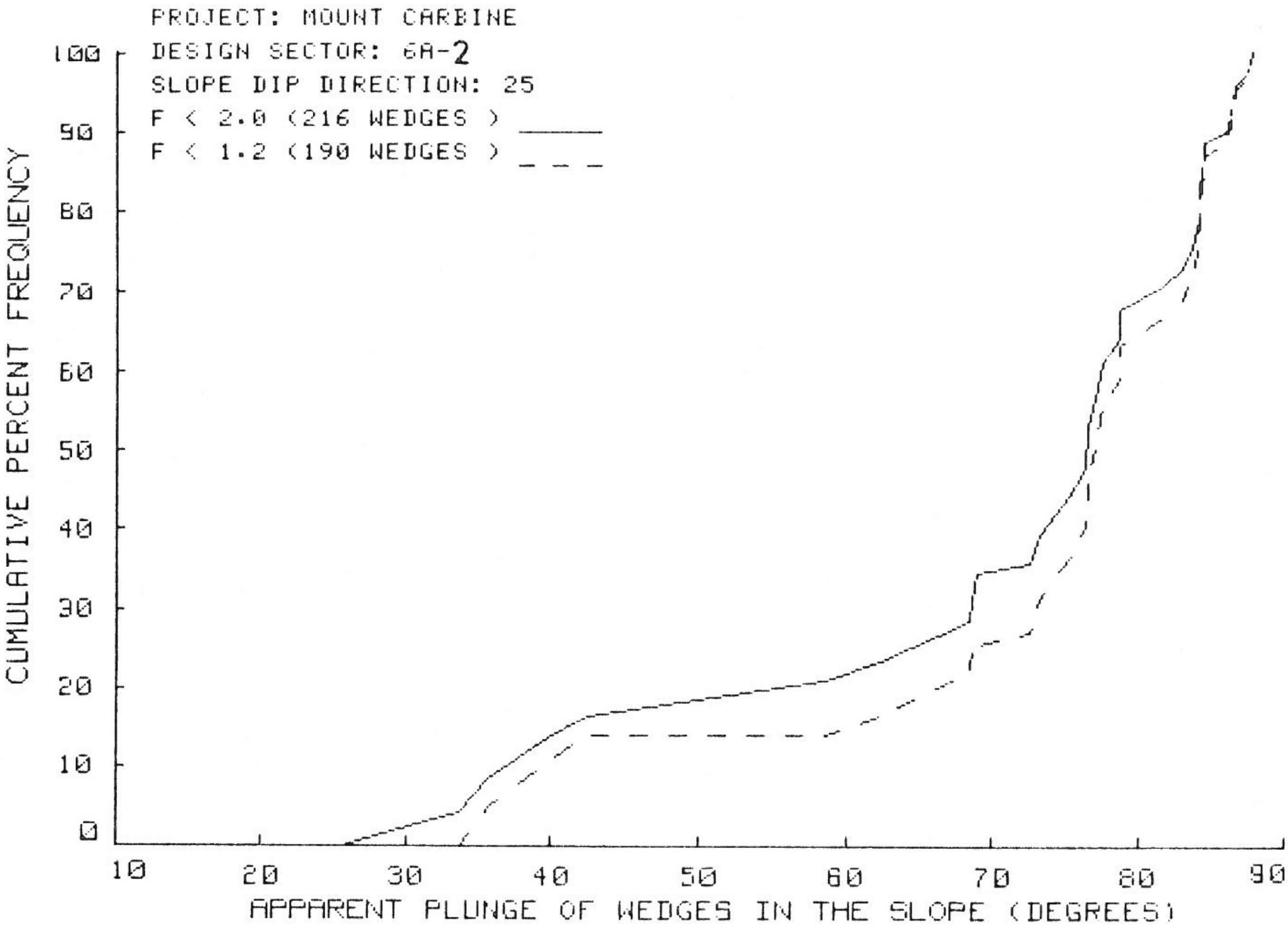
PROJECT: MOUNT CARBINE
DESIGN SECTOR: 5C-3
SLOPE DIP DIRECTION: 270
F < 2.0 (144 WEDGES)
F < 1.2 (137 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

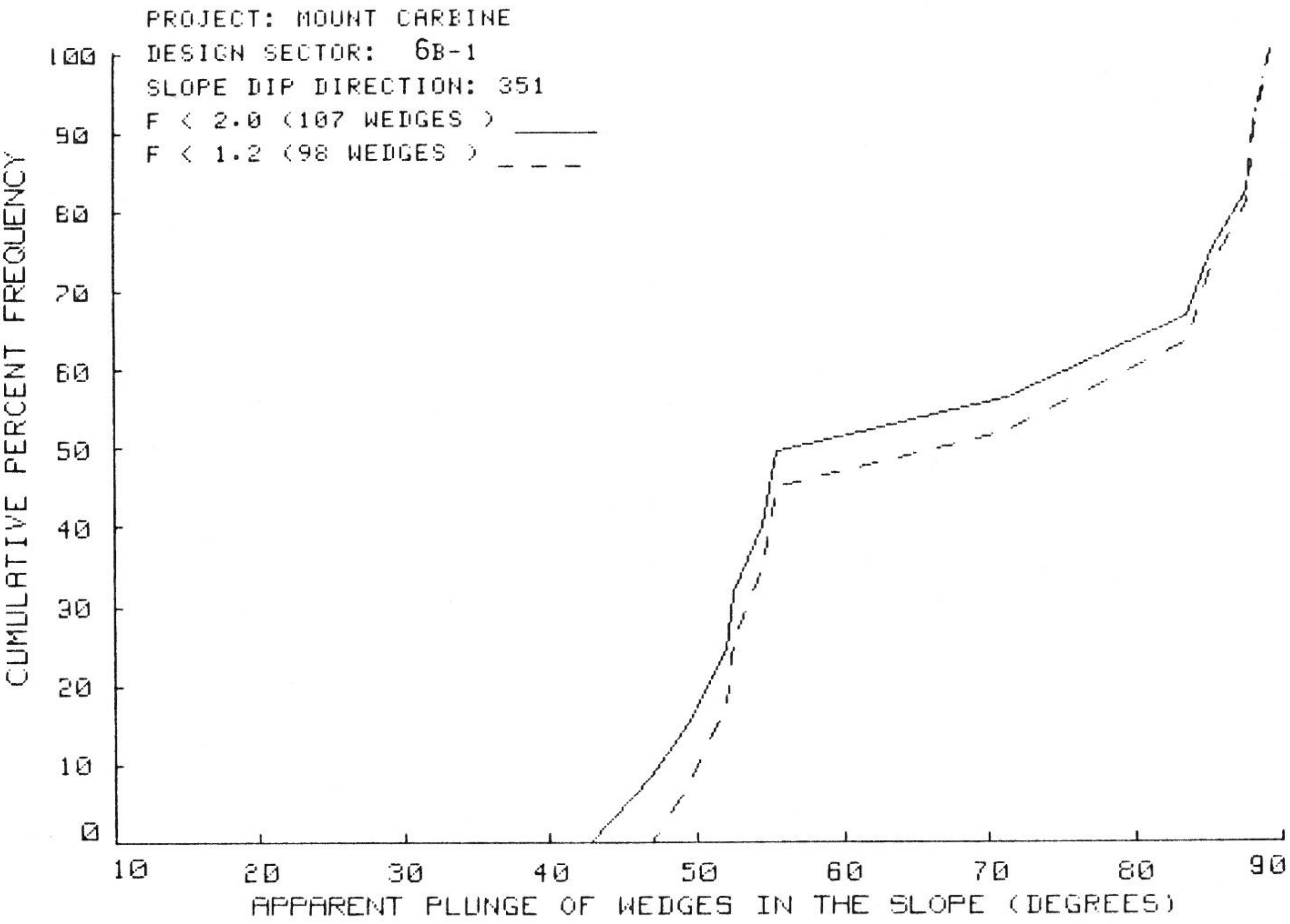

PROJECT: MOUNT CARBINE
DESIGN SECTOR: 5C-4
SLOPE DIP DIRECTION: 344
F < 2.0 (307 WEDGES)
F < 1.2 (278 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)



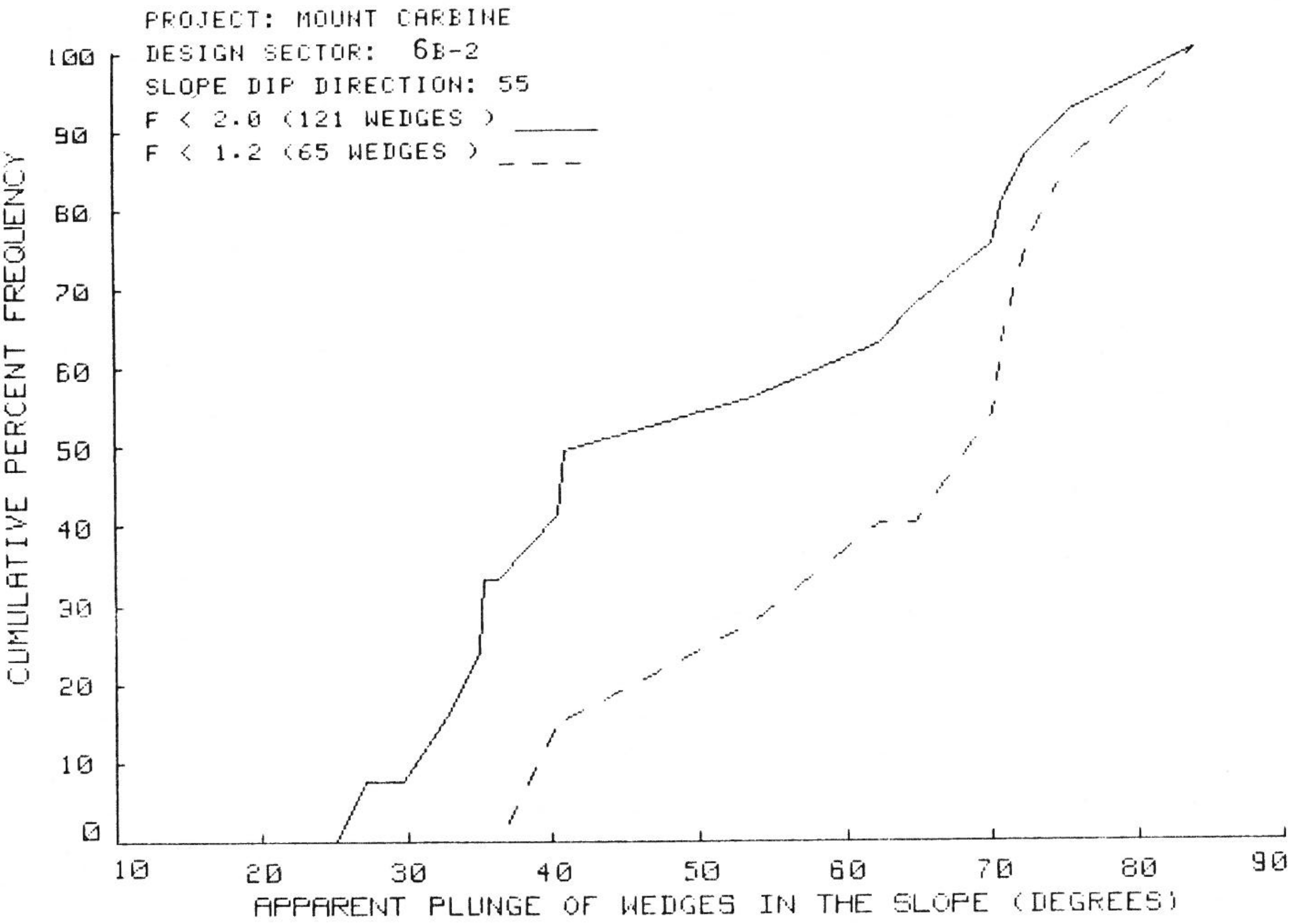
PROJECT: MOUNT CARBINE
DESIGN SECTOR: SD-1
SLOPE DIP DIRECTION: 344
F < 2.0 (407 WEDGES)
F < 1.2 (368 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)


PROJECT: MOUNT CARBINE
DESIGN SECTOR: 6A-1
SLOPE DIP DIRECTION: 345
F < 2.0 (212 WEDGES)
F < 1.2 (191 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)


PROJECT: MOUNT CARBINE
DESIGN SECTOR: 6A-2
SLOPE DIP DIRECTION: 25
F < 2.0 (216 WEDGES)
F < 1.2 (190 WEDGES)
CUMULATIVE PERCENT FREQUENCY
0
10
20
30
40
50
60
70
80
90
100
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)



PROJECT: MOUNT CARBINE
DESIGN SECTOR: 6B-1
SLOPE DIP DIRECTION: 351
F < 2.0 (107 WEDGES)
F < 1.2 (98 WEDGES)
CUMULATIVE PERCENT FREQUENCY
100
90
80
70
60
50
40
30
20
10
0
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)


PROJECT: MOUNT CARBINE
DESIGN SECTOR: 6B-2
SLOPE DIP DIRECTION: 55
F < 2.0 (121 WEDGES)
F < 1.2 (65 WEDGES)
CUMULATIVE PERCENT FREQUENCY
100
90
80
70
60
50
40
30
20
10
0
10
20
30
40
50
60
70
80
90
APPARENT PLUNGE OF WEDGES IN THE SLOPE (DEGREES)

Appendix B Hydrogeology Report

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REPORT ON

CARBINE TUNGSTEN

GROUNDWATER STUDY

Prepared for

Carbine Tungsten Pty Ltd

December 2012

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Project No. 222
17th December 2012

CARBINE TUNGSTEN

GROUNDWATER STUDY

Table of Contents

1.0 INTRODUCTION

Carbine Tungsten propose reprocessing of old tailings and low grade ore and expansion of the open pit at their Mount Carbine site. This development will occur in a staged manner.

Rob Lait and Associates Pty Ltd was requested to review existing groundwater data (bore logs, geological data and any Department of Natural Resources and Mines (NRM) groundwater data that is pertinent to the site, conduct of a site visit, identify environmental values of groundwater, describe groundwater conditions and potential risks and evaluate monitoring requirements for all stages of the proposed project.

2.0 SCOPE OF WORK

The scope of the work undertaken included assessment of the hydrogeological regime that would normally required for an environmental impact assessment, as follows:

- Description of aquifers - their lateral and vertical occurrence;
- Discussion of recharge and natural groundwater discharge;
- Discussion of groundwater dependent ecosystems;
- Discussion of groundwater level trends;
- Assessment of groundwater quality, including ionic speciation if major cation and anion data exist;
- Identification of groundwater environmental values;
- A risk assessment of potential threats to the groundwater environmental values of the area, including any identifiable impacts on surface water;
- Discussion of the groundwater monitoring bore strategy that is in place; and
- Recommendations for additions to the groundwater monitoring bore suite, should that prove necessary.

3.0 PHYSICAL SETTING

3.1 Site Features

The Carbine Tungsten mining operation is located about 70km by road to the north of Mareeba in North Queensland. The operation abuts the northern outskirts of the township of Mount Carbine itself.

Figure 3-1 shows the essential elements of site infrastructure and their layout.

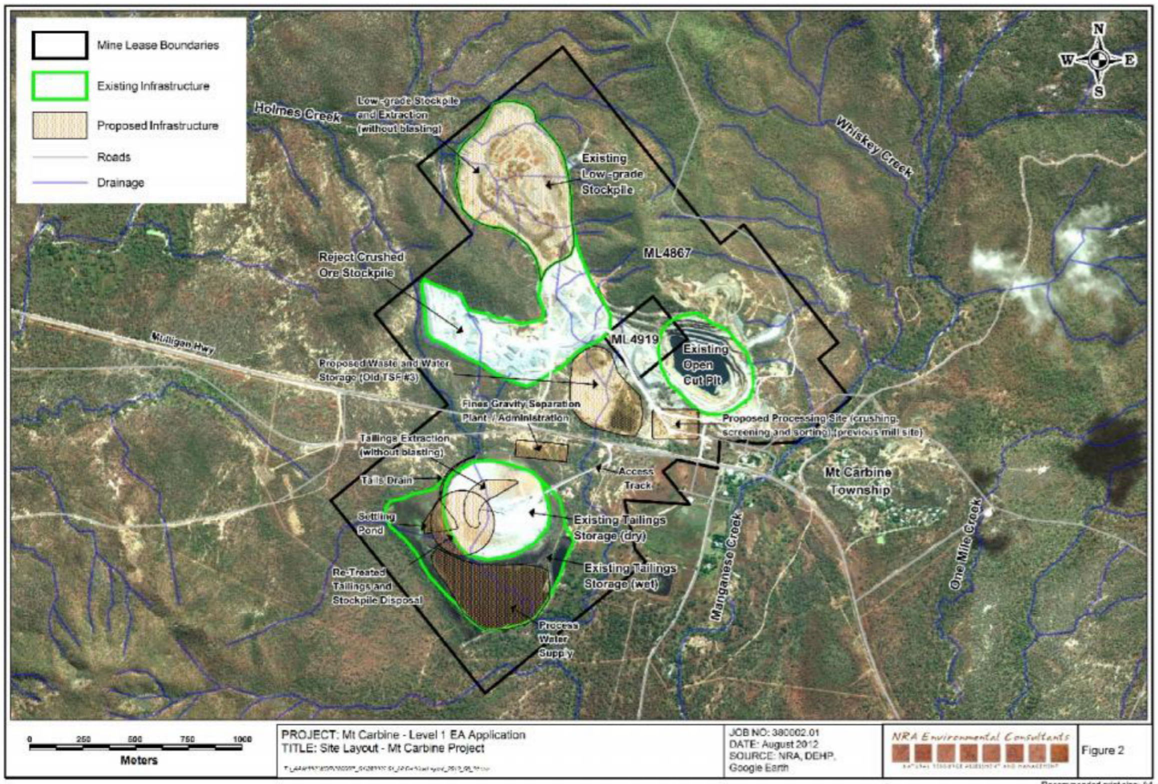


Figure 3-1: Carbine Tungsten Site Infrastructure Features and Layout (figure courtesy of Natural Resource Assessments Pty Ltd)

3.2 Surface Water Drainage

The Carbine Tungsten mining operation is situated in the Mitchell River catchment, in the foothills of the range which separates the westward draining Mitchell River catchment from the eastward draining Daintree River catchment. The elevation of the site is approximately 500m above sea level.

Local surface water drainage around the open pit is towards Manganese Creek, and around the waste rock dump is towards Holmes Creek, both of which are tributaries of the Mitchell River (see

Figure 3-2). The coordinates shown on Figure 3-2, and all subsequent figures in this report are eastings and northings according to the MGA94 mapping grid.

The most obvious features of the topography in the area are the relatively flat and somewhat featureless floodplains of the Mitchell river and sharp, upstanding ridges that rise to 150m above the surrounding floodplains in the foothills to the north of the mining operation. The drainage patterns of Manganese Creek and the ephemeral gullies around the site suggest that they are structurally controlled by a system of conjugate joints in the underlying Hodgkinson Formation bedrock.

Typically faulting and fracture zones control the development of such drainage. A dendritic drainage pattern is evident adjacent to the ridges. This pattern rapidly loses definition at the base of the slopes.

An open pit has existed at the site for over 25 years and currently contains a pit lake. Drainage around the open pit is locally diverted to the pit but is not otherwise disrupted by the pit.

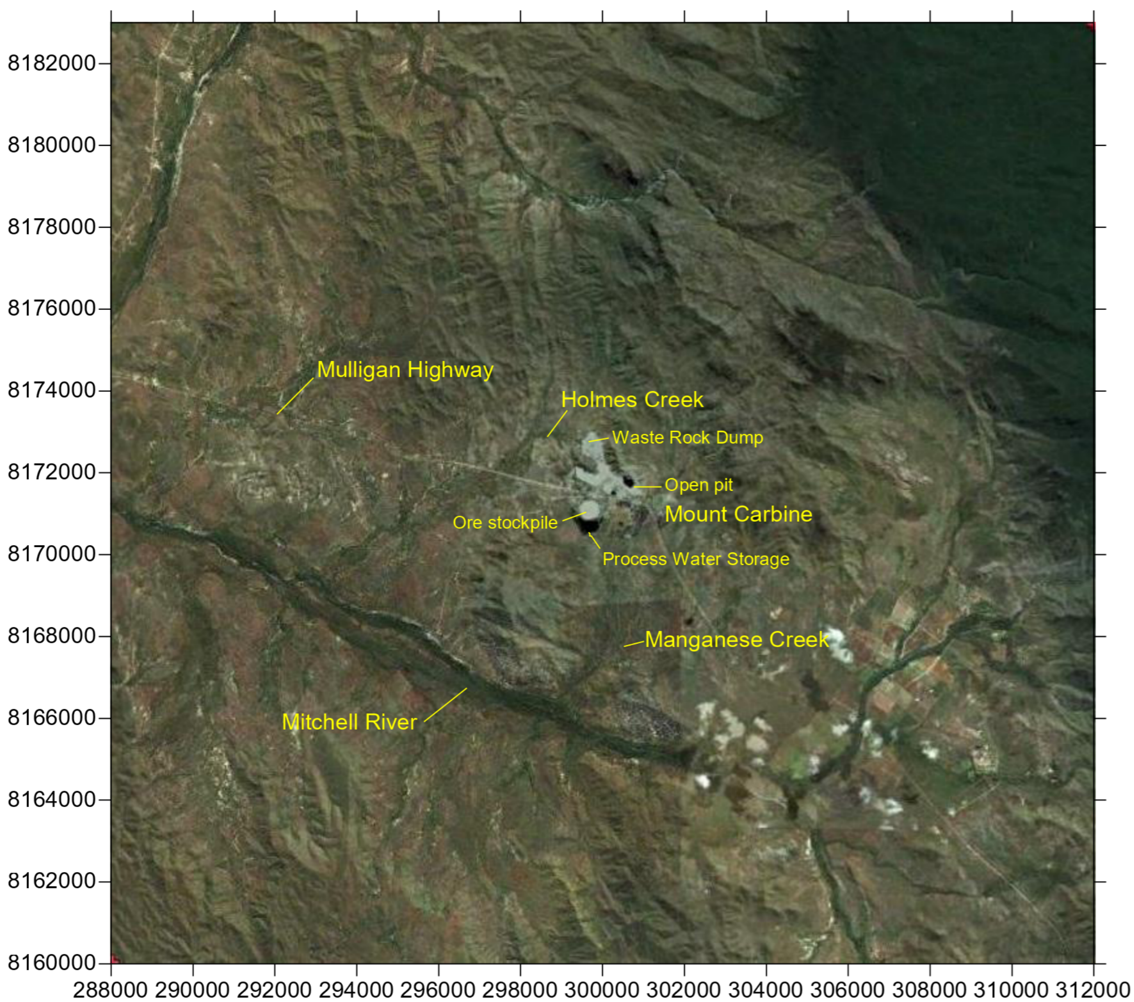


Figure 3-2: Aerial Photograph of Mount Carbine Area and Principal Site Drainage

3.3 Geology

Mount Carbine is underlain by the Silurian to Devonian Hodgkinson Formation. This is an extensive sequence of marine silici-clastic rocks, dominated by deep water turbidite sequences[1]. These rocks are most commonly described as 'shale' by groundwater drilling contractors. Subordinate chert layers and metabasalt lenses occur locally, and muddy sandstone (termed 'greywacke') is common.

The region has experienced multiple deformations, and been extensively intruded by granite. In the Mount Carbine area, the main rock type is a metamorphosed shale with a strong, near vertical cleavage.

A slaty cleavage is well developed within the turbidite sequences. Cleavage generally dips towards the southwest at angles of between 70^{o} and 90^{o} and strikes in a northwesterly-southeasterly direction.

Fractures resulting from deformation events are generally infilled with broken quartz associated with quartz veins emplaced during the later intrusion of granite. There are numerous local cross-cutting fractures.

Silt and clay have been derived from in situ weathering of fine grained rocks of the Hodgkinson Formation on the valley floor and from mass wasting from elevated areas. The Mitchell River and its tributaries has deposited sandy sediments along their natural levee banks.

Geological Structure

Structural features have a major influence on the geology of the Mount Carbine area. The structural features of importance in any discussion of the likely occurrence of groundwater are geological fractures. A geological fracture is defined as the discontinuity that disrupts rocks along cracks, fissures and joints. The scale of disruption determines whether the discontinuities are known as joints (insignificant disruption) or faults (major disruption). Within the area of interest fractures and subsidiary joints intersect the major lineaments at angles of about 60^{o}.

There is little evidence of faulting on the surface of the floodplain as the soils obscure any geological features.

[1] Bultitude, R.J., Donchak, P.J., Domagala, J., Fordham, B.G. and Champion, D.C., 1990. *Geology and Tectonics of the Hodgkinson Province, North Queensland*. Pacific Rim Congress, 1990 - Australian Institute of Mining and Metallurgy. Pages 75-81.

4.0 EXISTING GROUNDWATER INFORMATION

4.1 NRM Groundwater Database

A search of the NRM groundwater database was undertaken for all available groundwater data within a radius of 5km from the Carbine Tungsten mining operation. The bores for which groundwater data are available are assigned a registered number (RN) by NRM. Essential details of the private bores within the search radius are shown in Table 1.

TABLE 1: ESSENTIAL DETAILS OF PRIVATELY OWNED GROUNDWATER BORES IN SEARCH AREA

Registered Number	*Bore Name*	*Status*	*Easting*	*Northing*	*Depth drilled (m)*	*Perforated Interval (m from - m to)*	*Airlift Yield (L/s)*
78202	3 Mile	Abandoned and de-commissioned	302665	8168413	46	Not cased - too salty to use	-
78259	Village Bore	Existing	301865	8173079	38	26-38	6.8
78407		Existing	300806	8171700	30.5	19-30.5	3.2

The search area and the bores within the search zone are shown on Figure 4-1.

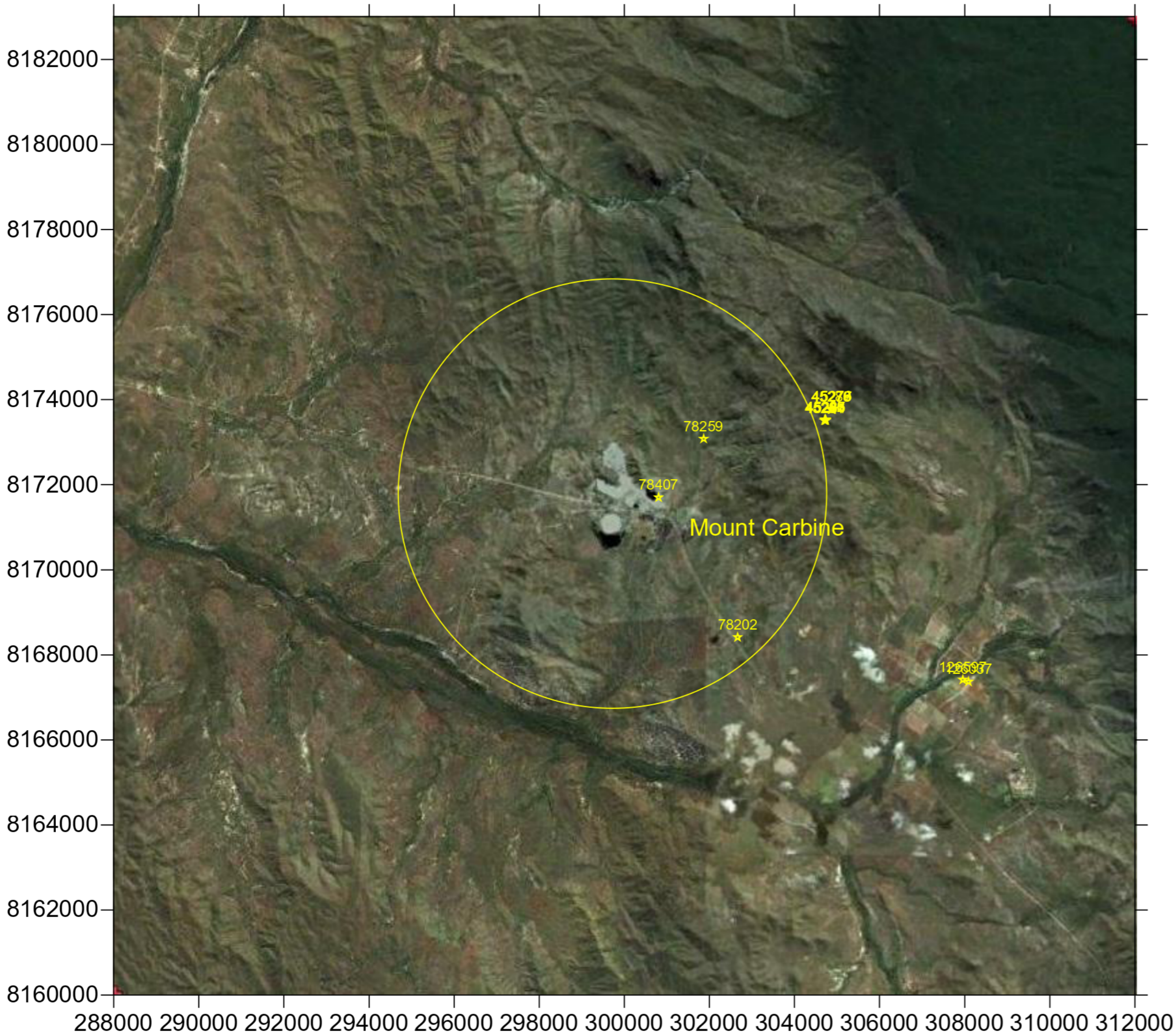


Figure 4-1: Search Area for NRM Groundwater Database Information

4.2 Dedicated Groundwater monitoring bores

A suite of groundwater monitoring bores was installed by Carbine Tungsten in November 2011. Table 2 shows essential details of that groundwater monitoring bore suite. Note that MB2 was not required and consequently was never drilled. The Rural Fire Brigade bore is also used by Carbine Tungsten as an additional source of groundwater samples for groundwater chemistry data.

TABLE 2: ESSENTIAL DETAILS OF DEDICATED CARBINE TUNGSTEN GROUNDWATER MONITORING BORES						
Bore	***Easting***	***Northing***	***Depth drilled (m)***	***Perforated Interval (m from - m to)***	***Airlift Yield (L/s)***	***Reference Point Elevation (m AHD)***
MB1A	299177	8171929	36.5	Not reported	Not reported	364.54
MB1B	299177	8171929	74	44-74	5.5	364.91
MB3A	301004	8172643	60	30-60	0.5	376.50
MB3B	301004	8172643	90	72-90	0.4	376.89
MB4A	299994	8170776	30	18-30	5.8	353.49
MB4B	299994	8170776	61	37-61	25.0	353.30
MB5A	298226	8172597	30	18-30	3.0	357.30
MB5B	298226	8172597	60	42-60	Not reported	357.41
MB6A	300597	8171181	31	19-31	1.7	357.19
MB6B	300597	8171181	62.5	44.5-62.5	2.0	357.28
Rural Fire Board	301144	8171764	Not reported	Not reported	Not reported	360.87

Table 2 shows that two aquifer intervals were targeted by the dedicated groundwater monitoring bores - one at a depth of about 30m and a deeper interval at a depth of 60 to 90m.

Figure 4-2 is an aerial photo showing the locations of the dedicated groundwater monitoring bore suite at Carbine Tungsten.

Carbine Tungsten monitor groundwater in the Rural Fire Board at Mount Carbine. This is a prudent choice as the bore is directly along strike of cleavage and potentially down the groundwater flow direction and groundwater flow gradient from the open pit. The abbreviation RFB is used for the Rural Fire Brigade bore.

The data from the privately owned groundwater bores and the dedicated groundwater monitoring bores illustrates the variability of the fractured Hodgkinson Formation aquifer in the Carbine Tungsten vicinity. The airlift yields are highly variable (0.4 to 25L/s) and there is no definite relationship between the depth at which the aquifer intervals were encountered (these being indicated by the perforated intervals). The airlift yield from MB4B is exceptional.

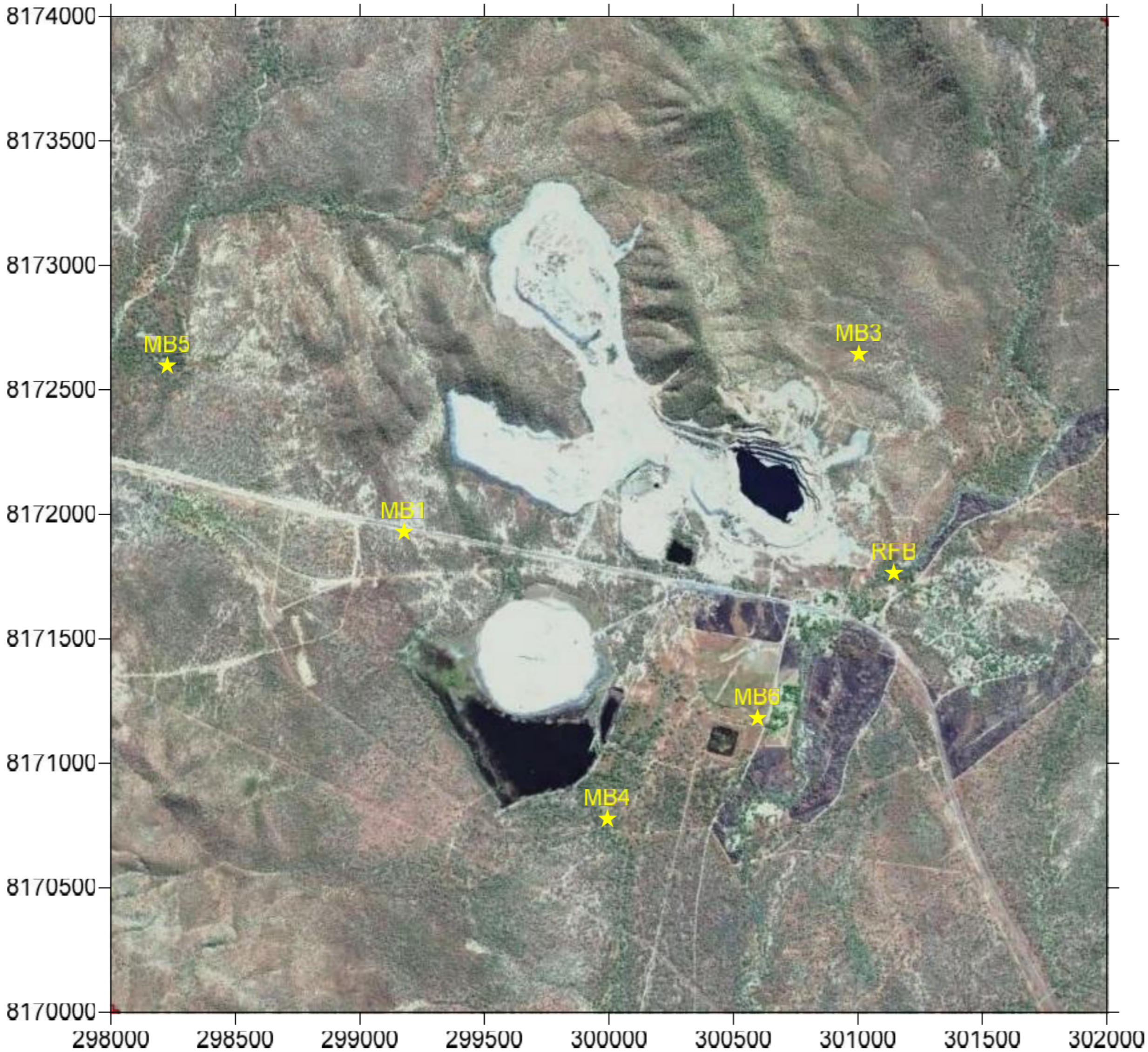


Figure 4-2: Carbine Tungsten Dedicated Groundwater Monitoring Bores

5.0 HYDROGEOLOGY

5.1 Aquifer Units

Hodgkinson Formation

The greatest potential for groundwater at Mount Carbine occurs within the Hodgkinson Formation. Fractured rocks within the Hodgkinson Formation contain aquifers that generally have secondary porosity, may be hydraulically disjointed and can be difficult to analyse because of these characteristics.

Photograph 1 shows the almost vertical attitude of cleavage within fine-grained rocks of the Hodgkinson Formation. These rocks are situated in Manganese Creek just to the east of the existing open pit . Open cleavage partings such as those shown in Photograph 1 offer the greatest opportunity for recharging rainfall to penetrate fractured rock aquifers, and accumulate as groundwater at depth.

Groundwater primarily resides along the cleavage partings and along the traces of faults.

Photograph 1: Cleavage Partings in fine-grained Hodgkinson Formation Rocks (near existing Open Pit)

Alluvium

There is little or no development of sandy alluvial material or levee development within the Carbine Tungsten lease. Based on previous experience it is concluded, therefore, that alluvium is not hydrogeologically significant.

5.2 Groundwater Recharge

The primary mechanism for recharge to the aquifers in the area is the direct infiltration of rainfall. Recharge to fractured zones in the Hodgkinson Formation will be rapid in areas where open cleavage partings are both exposed at the surface and persist to depth. There are significant areas to the north of the Mulligan Highway where such conditions exist.

A secondary mechanism for recharge to the aquifers at Mount Carbine is *episodic* or *flood recharge*. During a major rainfall event, 80% of water not lost by evaporation may discharge as surface runoff. The maximum opportunity for recharge to these aquifers will occur where cleavage partings are exposed in ephemeral gullies that traverse the site and flow during the wet season.

There is little or no evidence of exposed Hodgkinson Formation to the south of the Mulligan Highway.

5.3 Groundwater Levels

Groundwater levels have been measured within the dedicated groundwater monitoring bores since they were installed in November 2011.

Table 3 shows the groundwater level measurements in the dedicated groundwater monitoring bores for 2012.

TABLE 3: 2012 GROUNDWATER LEVELS IN CARBINE TUNGSTEN DEDICATED GROUNDWATER MONITORING BORES
(all groundwater levels in metres below measurement reference point)

Date	***5/01/12***	***30/05/12***	***27/06/12***	***31/07/12***	***24/08/12***	***5/11/12***
MB1A	16.6	16.41	16.3	16.2	16.15	15.62
MB1B	17	16.84	16.3	16.55	16.55	16.72
MB3A	24.77	25.6	24.5	24.25	24.4	25.04
MB3B	24.91	25.15	25	24.7	24.7	25.43
MB4A	5.4	5.02	4.5	4.8	4.8	5.11
MB4B	5.08	4.81	5.08	4.65	4.7	4.94
MB5A	11.75	11.56	11.3	11.35	11.35	11.93
MB5B	11.48	11.69	11.3	11.55	11.5	11.57
MB6A	11.65	11.16	11	10.95	11	11.32
MB6B	12.16	11.56	11	11.3	11.4	11.71

Figure 5-1 shows a chart of the 2012 groundwater levels in the dedicated groundwater monitoring bores.

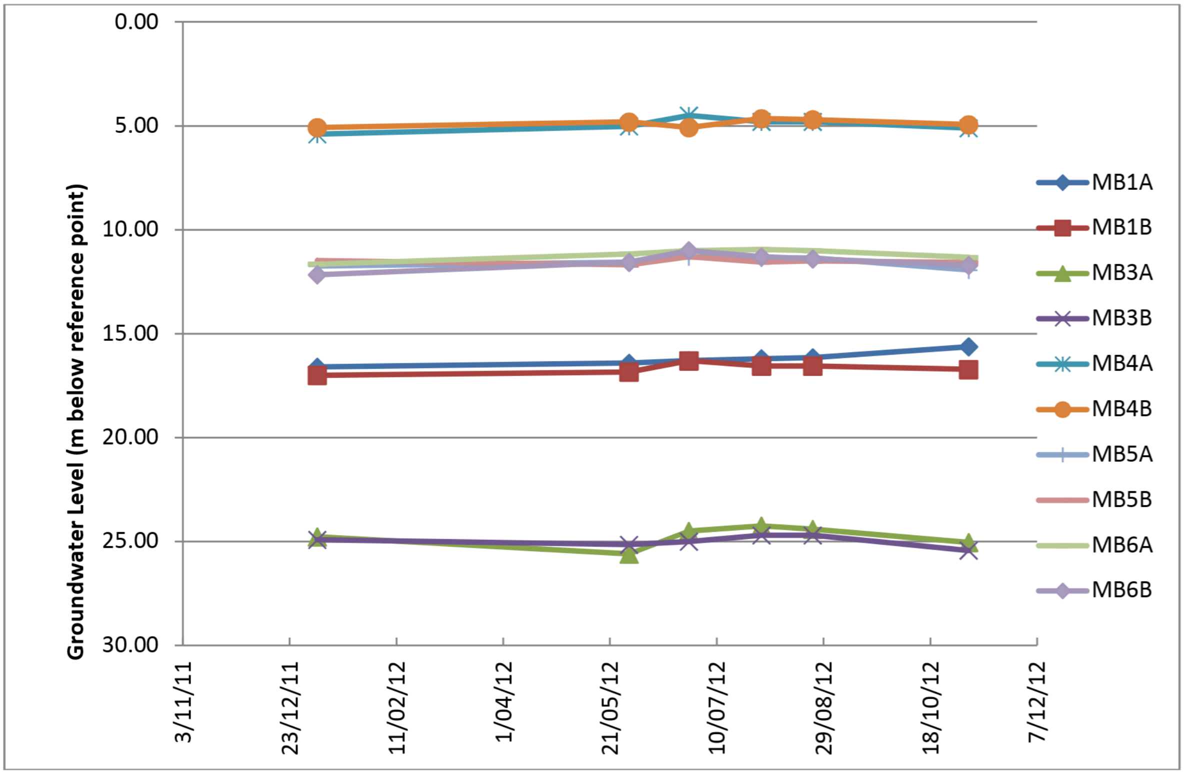


Figure 5-1: Chart of 2012 Groundwater levels in Carbine Tungsten Dedicated Groundwater Monitoring Bores

The groundwater levels in MB4A and MB4B are relatively higher that the groundwater levels in the rest of the groundwater monitoring bore suite. The groundwater levels in MB4A and MB4B are probably influenced by the standing water in the processing pond, which has been a more or less permanent water storage for a number of years.

5.4 Groundwater Flow

The elevations of the groundwater in the monitoring bores were calculated from the groundwater level measurements for 5th November 2012, and the elevations of the measurement reference points. The elevation of the water level in the open pit is also know. These data were then contoured using the Surfer proprietary software package. The water level elevation data were interpolated by kriging.

The water level elevation contours are considered to be valid based on the following assumptions:

1. There is generally sufficient hydraulic connectivity within bedding, cleavage and fractures of the Hodgkinson Formation to permit groundwater flow;

2. There is generally sufficient hydraulic conductivity within the Hodgkinson Formation to permit groundwater flow; and

3. Groundwater will flow from more elevated areas to less elevated areas.

Figure 5-2 shows the contours of the groundwater level elevation in the 'A' (shallow aquifer zone) pipes and at OW1 (the open pit) for 5th November 2012, in metres (Australian Height Datum).

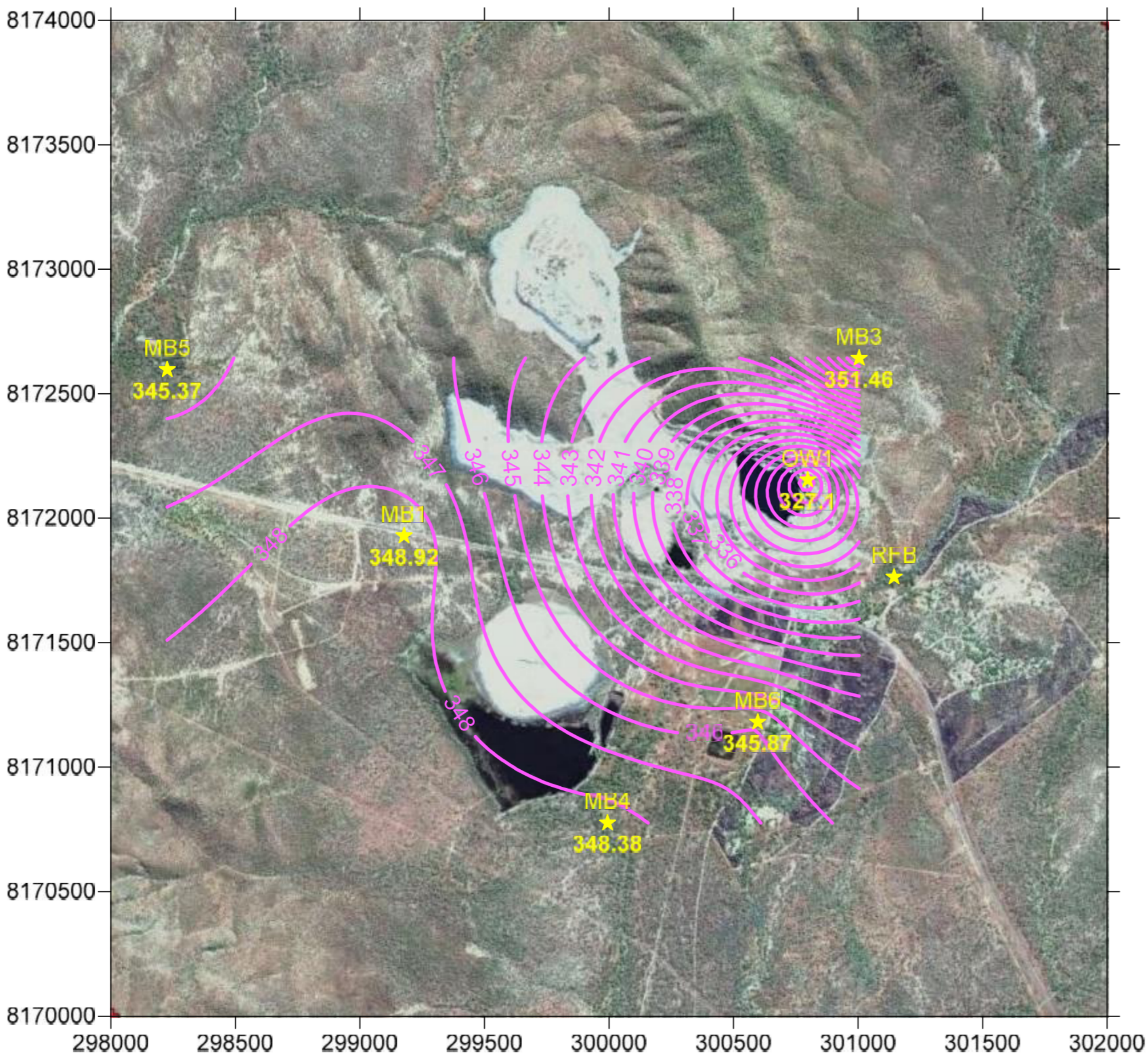


Figure 5-2: Contours of Groundwater and Open Pit Water level Elevation (m AHD) - 5th November 2012

(Water level elevations shown below monitoring site IDs)

The contours show that the open pit is a significant sink for groundwater flow with groundwater travelling towards the pit radially (i.e. from every direction). This will be more so if the open pit is expanded, dewatered and deepened as intended.

This is a hydrogeologically advantageous scenario as groundwater is not flowing towards either Manganese or Holmes Creeks. Although groundwater contamination measures will be implemented by Carbine Tungsten, in the event that a contamination event does occur, the contaminated groundwater will flow towards and be captured within the open pit. Contaminated water in the open pit is considered to be manageable.

5.5 Natural Groundwater Discharge

From the water level elevation contours presented above it is obvious that the open pit is the controlling hydrological influence at Carbine Tungsten. Groundwater flow towards the open pit will preclude any possibility of natural groundwater discharge from the mining infrastructure to Manganese and Holmes Creeks, or to the Mitchell River

There was no evidence of springs or seeps in the ephemeral gullies at the time of the site inspection. There is no report of springs or seeps in the immediate vicinity of Mount Carbine either in the NRM groundwater database or anecdotally.

There are no recognisable or known groundwater dependent ecosystems within the Carbine Tungsten operation area.

6.0 GROUNDWATER CHEMISTRY

Water chemistry is monitored at the carbine Tungsten site at a number of surface water sites as well as in the dedicated groundwater monitoring bores.

Figure 6-1 shows the sampling sites used by Carbine Tungsten (courtesy of Natural Resource Assessments Pty Ltd). For comparison purposes the water chemistry from OW1, OW4 and SW3 will be compared to the groundwater chemistry.

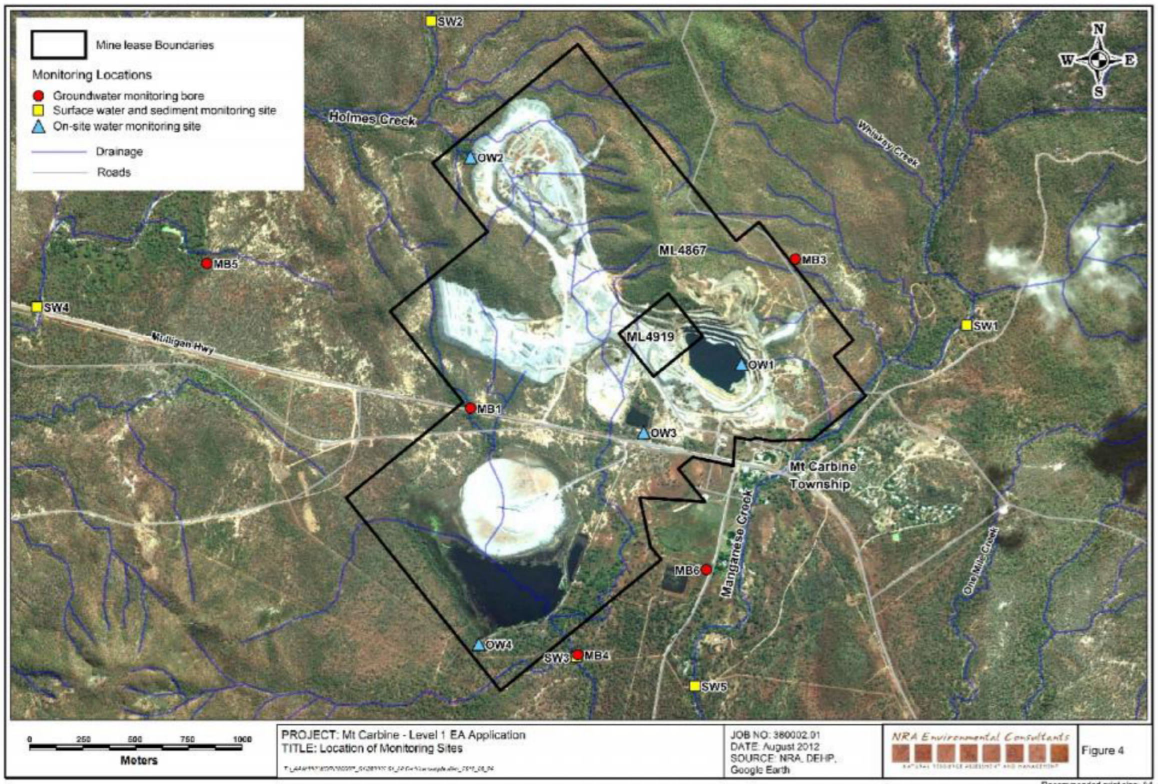


Figure 6-1: Carbine Tungsten Environmental Sampling Sites (courtesy of Natural Resource Assessments Pty Ltd)

Groundwater chemistry samples have been retrieved from the dedicated groundwater monitoring bores on five occasions since the bores were installed in late 2011 (5/01/12, 30/05/12, 27/06/12, 31/07/12 and 24/08/12).

Median values of the analyte concentrations were calculated using the data from those five sampling events. These values are presented in Table 4.

TABLE 4: GROUNDWATER AND OPEN PIT MAJOR ION CONCENTRATIONS MEDIAN VALUES								
	MB1A	***MB1B***	***MB3A***	***MB3B***	***MB4A***	***MB4B***	***MB5A***	***MB5B***
pH	7.4	8.6	7	9.1	7.6	7.8	7.2	7.5
EC	1800	1200	730	640	1700	2500	2200	2900
$HC0_3$	610	200	270	88.5	580	620	720	695
Cl	240	225	60	66	92.5	315	300	555
Ca	110	20.5	52	4.35	34	62.5	87.5	125
K	4.8	13.5	2.55	16	4.05	8.3	2.1	9.55
SO_4	29	48	53	120	270	330	42	13
Na	200	210	76.5	110	330	450	310	360
Mg	38.5	5	20	1.05	25	47	56.5	78.5
Mn	0.135	0.017	0.2	0.0055	0.05	0.05	0.02	0.99

TABLE 4: GROUNDWATER AND OPEN PIT MAJOR ION CONCENTRATIONS MEDIAN VALUES								
Zn	0.0085	0.005	0.009	0.005	0.005	0.0075	0.008	0.005
S	10	17	19	39	89	120	15	4.65
	MB6A	***MB6B***	***Open Pit***	***RFB***				
pH	7.1	7.2	8.1	7.3				
EC	5500	5700	820	1400				
$HC0_3$	840	710	54	310				
Cl	1350	1450	61	230				
Ca	200	215	100	85				
K	11.5	17	2.8	2.6				
SO_4	88	84	290	110				
Na	680	675	45	130				
Mg	180	165	27	45				
Mn	0.028	4.5	0.005	0.35				
Zn	0.009	0.0065	0.014	0.011				
S	30	29	96	35				

(Notes: pH in pH units; electrical conductivity in µS/cm, all other analytes in mg/L)

The data in Table 4 show that:

- The groundwater in the area is moderately alkaline;

- The shallower groundwater (i.e. from the "A" pipes) is generally of lower electrical conductivity than the deeper groundwater (i.e. from the "B" pipes). As electrical conductivity is one measure of water salinity the deeper groundwater is more saline than the shallower groundwater. Higher salinity groundwater is generally an indication of longer residence time within the aquifer zone. Therefore it is likely that the hydraulic conductivity of the deeper aquifer zone is less than the hydraulic conductivity of the shallower aquifer zone, with consequent lower groundwater flow velocity in the deeper aquifer zone;

- The electrical conductivity of the groundwater indicates that the water is naturally brackish and not suitable for human consumption. This observation is reinforced particularly by the electrical conductivity values from MB5A and MB5B (both in excess of 2,000µS/cm) and by the fact that RN 78202 was abandoned and decommissioned as the groundwater encountered in it was too salty. MB5 is located in a position where it is unlikely to be impacted by any of the mine infrastructure. Similarly, RN 78202 is considered to be too remote from the mine infrastructure to be impacted by the mine;

- The sulphate concentrations in all the groundwater around the mine infrastructure is not excessive and is certainly less that the stockwatering guideline value of 1,000 mg/L;

- The water from the open pit is of considerably better quality than the groundwater in the area

6.1 Ionic Composition

The major anions and cations data were used to assess the ionic speciation of the groundwater and the water from the open pit. A Piper Diagram was used to compare and represent the ionic speciation of these waters (Figure 6-2).

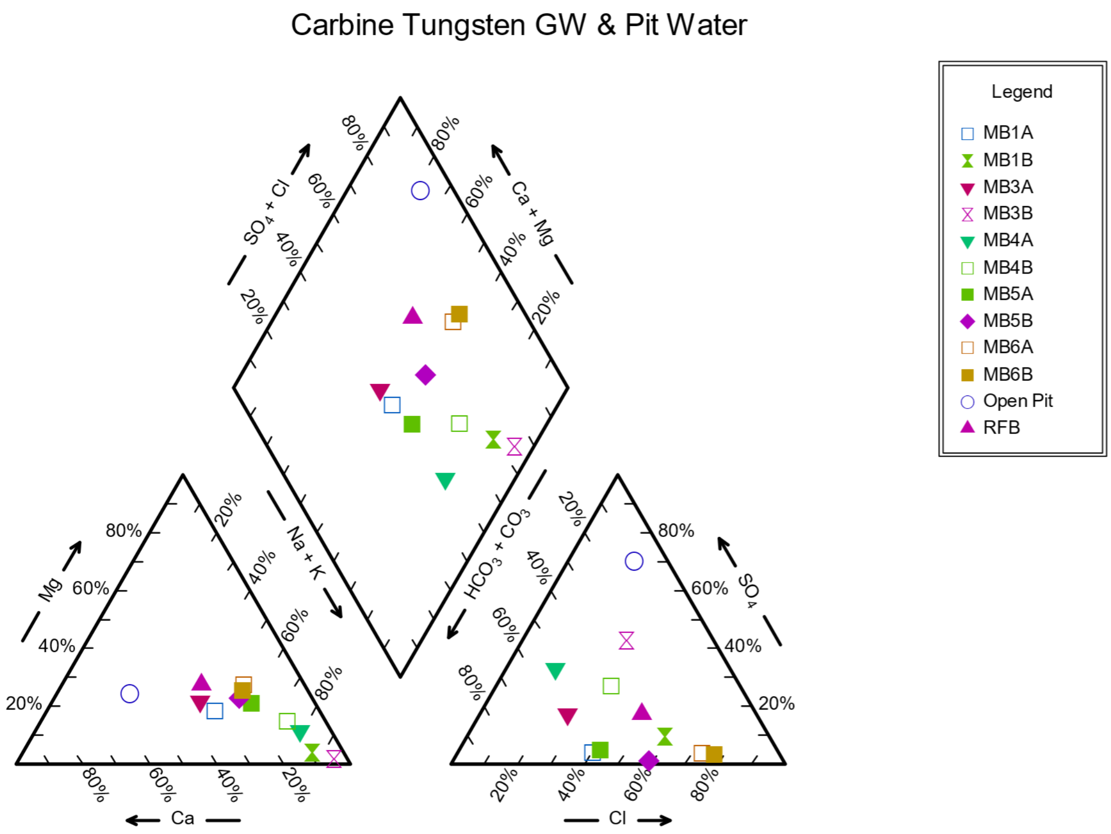


Figure 6-2: Piper Diagram of Carbine Tungsten Groundwater and Open Pit Water

Table 5 shows the ionic type of the groundwater and open pit water at Carbine Tungsten.

TABLE 6: IONIC WATER TYPE OF GROUNDWATER AND OPEN PIT	
Sample-Site	***Ionic water type***
MB1A	$NaHCO_3$
MB1B	NaCl
MB3A	$NaHCO_3$
MB3B	$NaSO_4$
MB4A	$NaHCO_3$
MB4B	$NaHCO_3$
MB5A	$NaHCO_3$
MB5B	NaCl
MB6A	NaCl
MB6B	NaCl
RFB	NaCl
Open Pit	$CaSO_4$

It is evident from Table 6 and the position of the symbol representing the open pit water in the quadrilateral in Figure 6-2 that the open pit water has a significantly different ionic composition than the groundwater from all other dedicated groundwater monitoring bores.

There is no conclusive ionic evidence that the water in the open pit is currently impacting on the groundwater in the surrounding groundwater monitoring bores.

6.2 Comparison Between Open Pit Water and RFB Groundwater

The analyte values for pH, electrical conductivity and sulphate were used for comparison of the water from the open pit and the RFB bore. Charts of these comparisons are presented in Figures 6-3, 6-4 and 6-5.

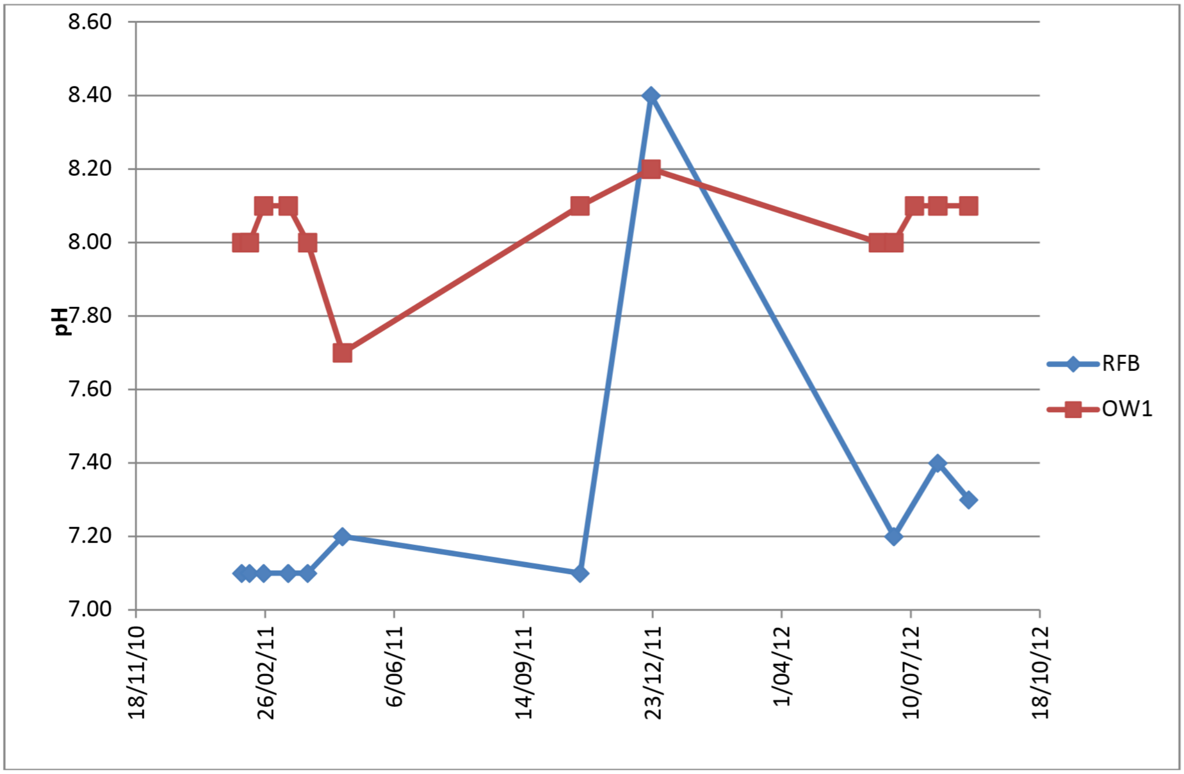


Figure 6-3: Chart of pH Comparison of Open Pit Water (OW1) and RFB Groundwater

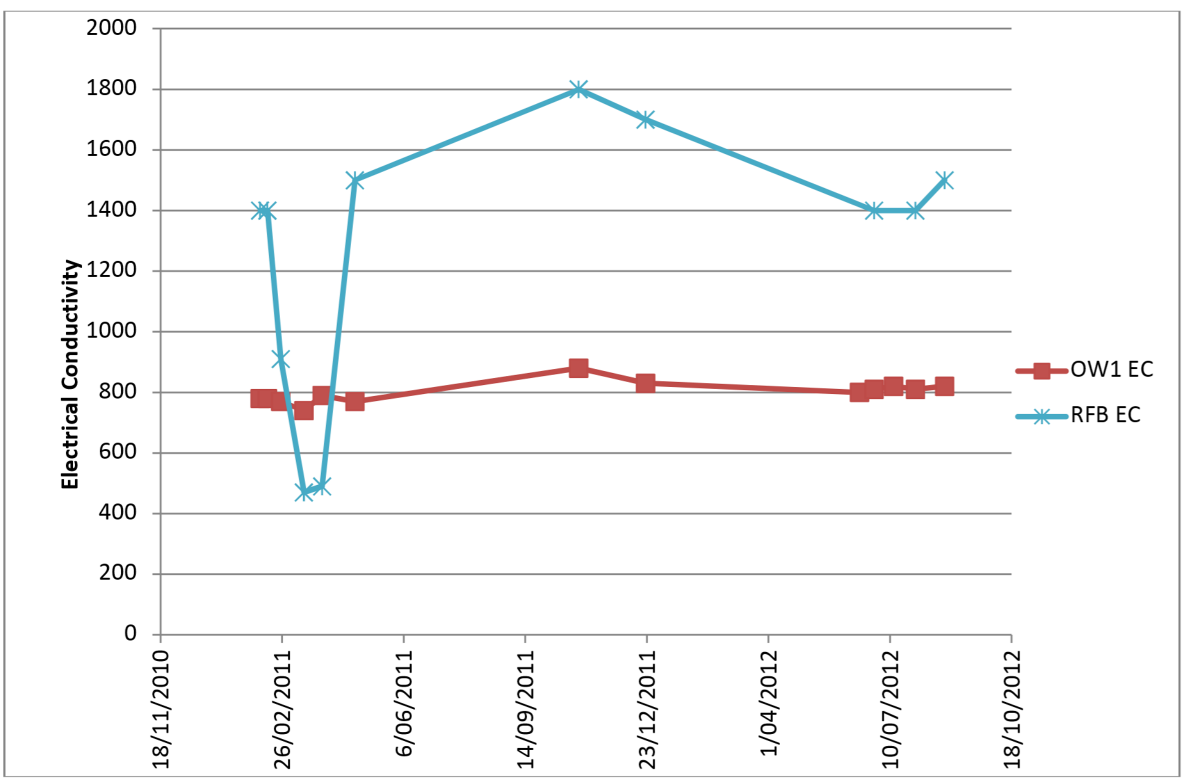


Figure 6-4: Chart of Electrical Conductivity Comparison of Open Pit Water (OW1) and RFB Groundwater

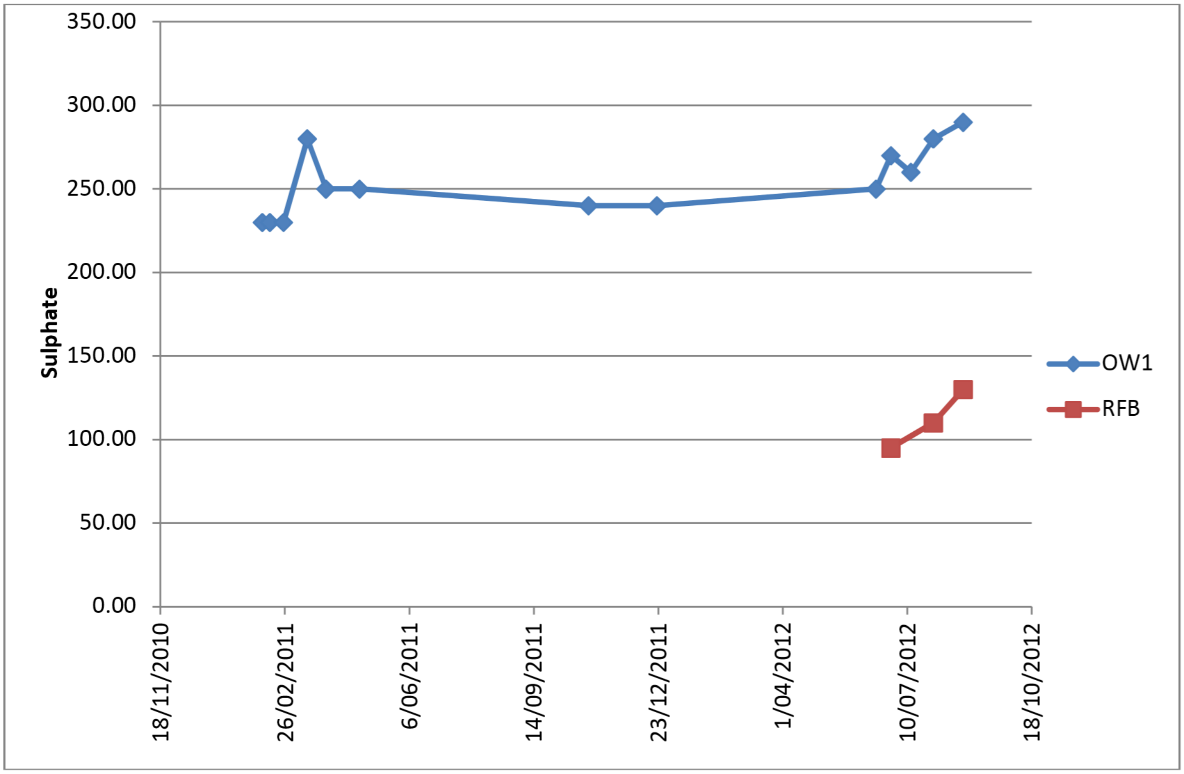


Figure 6-5: Chart of Sulphate Comparison of Open Pit Water (OW1) and RFB Groundwater

It is assessed from these comparisons that:

- No definitive correlation exists between the open pit water and groundwater pH;

- There is currently no correlation between surface water and groundwater electrical conductivity; and

- A correlation of increasing sulphate concentration seems to be emerging between the open pit water and the groundwater in the RFB bore, despite the water level elevation contours indicating that groundwater flow ifs towards the open pit from the RFB bore. Regular monitoring of sulphate at these sites should be maintained to assess whether this trend continues throughout and following the forthcoming wet season.

6.3 Comparison Between SW3 and MB4 Groundwater

The analyte values for pH, electrical conductivity and sulphate were used for comparison of the water from the creek that drains the process water pond (sampling site SW3) and the groundwater from the MB4 bores. Charts of these comparisons are presented in Figures 6-6, 6-7 and 6-8.

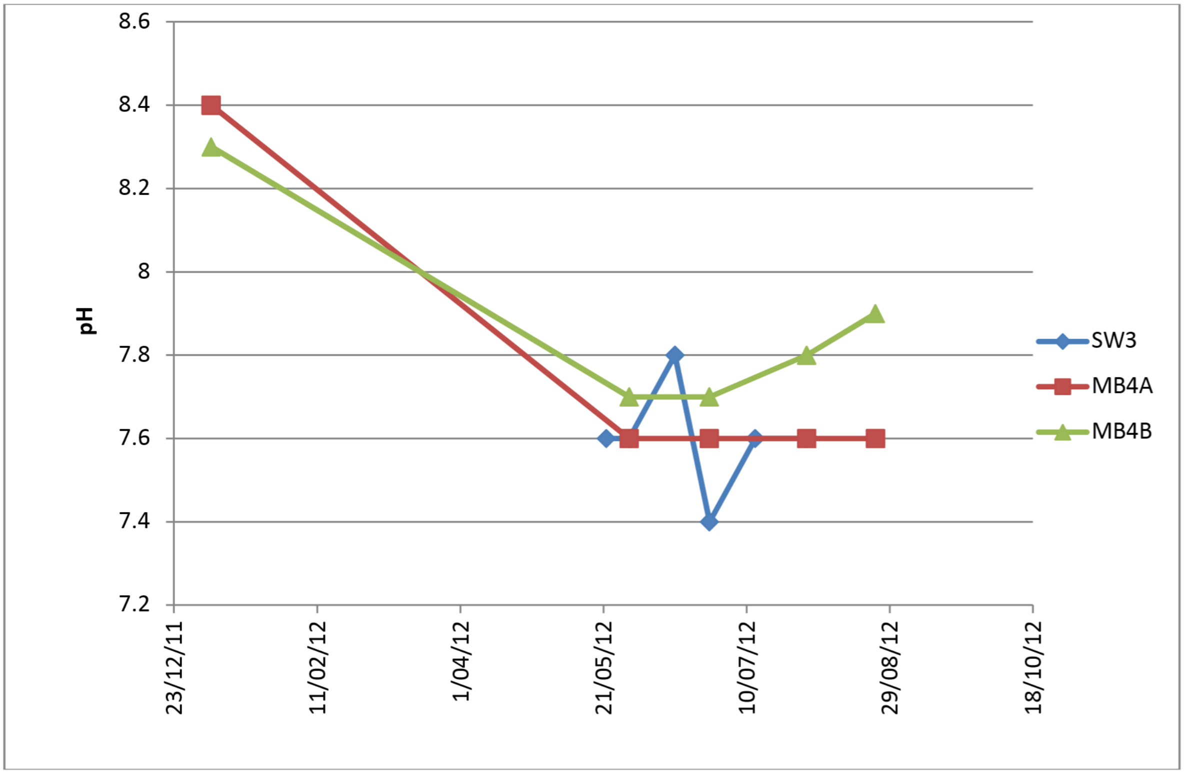


Figure 6-6: Chart of pH Comparison of SW3 and MB4 Groundwater

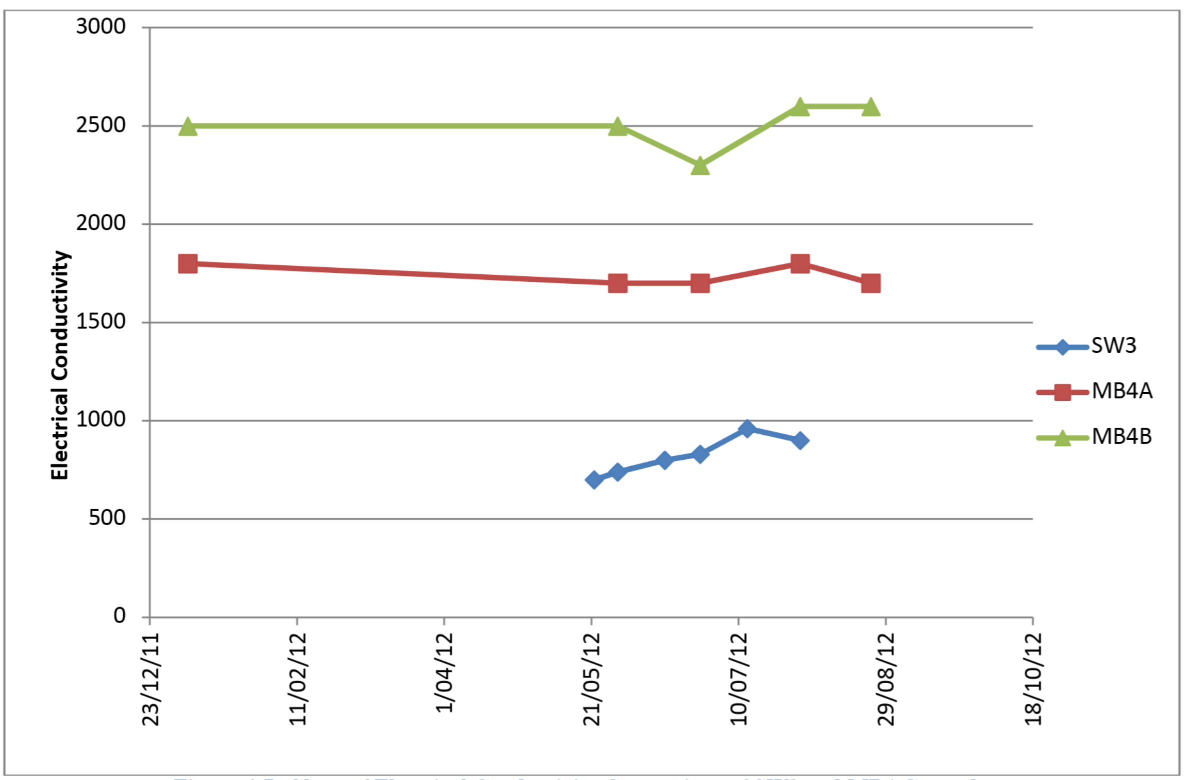


Figure 6-7: Chart of Electrical Conductivity Comparison of SW3 and MB4 Groundwater

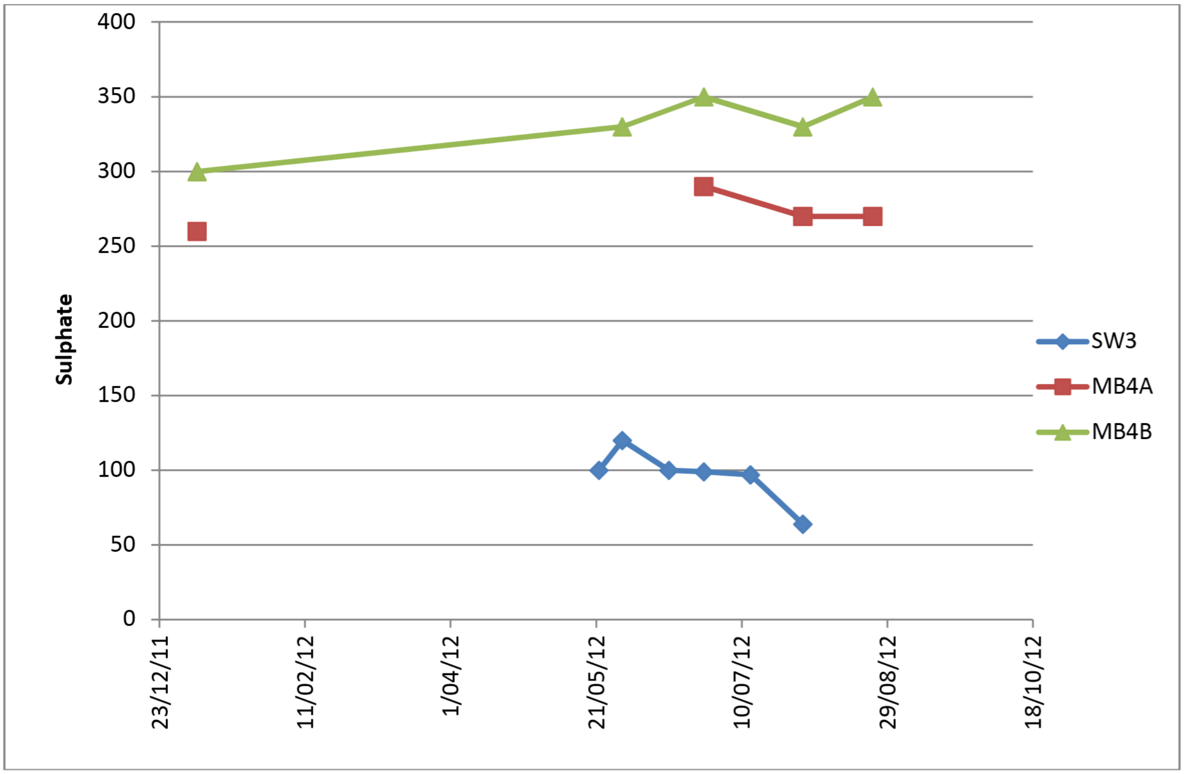


Figure 6-8: Chart of Sulphate Comparison of SW3 and MB4 Groundwater

It is assessed from these comparisons that:

- No correlation exists between SW3 and groundwater pH;
- There is currently no correlation between SW3 and groundwater electrical conductivity; and
- No correlation currently exists between SW3 and groundwater sulphate.

7.0 GROUNDWATER - SURFACE WATER INTERACTION

Based on the discussion of surface drainage, groundwater levels and groundwater chemistry presented above it is assessed that there is minimal interaction between the surface water and the groundwater at the Carbine Tungsten site. The chemical signatures of the surface water and groundwater, in particular, are very different.

Some surface water - groundwater interaction may occur for a short period during the wet season but it is expected to be short-lived. Monitoring of groundwater levels will assist with this assessment.

8.0 GROUNDWATER RISK ASSESSMENT

The risks of pollution of the hydrogeological regime by proposed increased mining activity in the future are considered to be low because of:

- Low hydraulic conductivity of the Hodgkinson Formation aquifers;
- Depth to the groundwater;
- Groundwater flow towards the open pit from every direction;
- No conclusive links between groundwater and surface water chemistry at the site;
- Potentially low ’use’ of groundwater by Carbine Tungsten for future mining operations.

Provided that the usual safeguards are put in place (adoption of sound chemical handling, storage and spill clean-up procedures, bund wall diversions and construction of mine infrastructure away from watercourses) there is little risk of pollution of the groundwater or of contaminant travel from the site.

Large-scale extraction of groundwater for mining purposes is not proposed so the risk of regional lowering of groundwater levels (depressurisation) is also low. There is little known use made of groundwater in the Mount Carbine area.

9.0 GROUNDWATER ENVIRONMENTAL VALUES

Table 7 identifies the groundwater values, potential threats to those values and management needs associated with the Carbine Tungsten project.

TABLE 21: GROUNDWATER VALUES – CARBINE TUNGSTEN PROJECT

Value	*Description*	*Potential Threats*	*Management Requirements*
Economic	• Bores in the region are used for stock-watering	• Groundwater contamination from chemical spills and tailings storage dam	• Adoption of sound chemical handling, storage and spill clean up procedures will minimise impact on groundwater quality
	• Groundwater use for the project is expected to be minor	• Reduction in available groundwater resource for stock-watering	• Monitoring of groundwater levels and groundwater quality in dedicated groundwater monitoring bore suite. • Maintain minimum use of groundwater for the project
	• Private groundwater use is minor - groundwater is not naturally potable and is used for fire control	• Reduction in available groundwater resource for fire fighting	• Monitoring of groundwater levels and groundwater quality in dedicated groundwater monitoring bore suite. • Maintain minimum use of groundwater for the project
Hydrological	• Groundwater in Corella Formation may contribute to flow of Manganese Creek and Holmes Creek in wet season	• Assessed to be minimal owing to location of water table and incision depths of streams.	• Maintain minimum use of groundwater for the project • Monitoring of groundwater levels and groundwater quality in dedicated groundwater monitoring bore suite.
	• Dewatering during mining unlikely to impact on groundwater levels	• Only minor dewatering of the ore zone is likely to be required, owing to expected very low hydraulic conductivity in Hodgkinson Formation	• Groundwater inflow to expanded open pit expected to be minimal based on low indicated hydraulic conductivity. Measure and record inflows as they are encountered.

TABLE 21: GROUNDWATER VALUES – CARBINE TUNGSTEN PROJECT			
Value	***Description***	***Potential Threats***	***Management Requirements***
	• Leachate from open pit may impact on groundwater in the upper fractured zone of the Hodgkinson Formation		• Continue monitoring of groundwater levels around site as mining proceeds. • Monthly monitoring of pH, electrical conductivity and sulphate in open pit and RFB bore • Review results of groundwater level and groundwater quality monitoring on an annual basis, in the form of a borefield performance review report.
Environmental	• Groundwater – surface water exchange	• Contaminated groundwater discharge may impact on surface water	• Assessed that groundwater – surface water exchange is only likely to occur during wet season • Do not construct infrastructure that can cause ground water contamination within 50m of Manganese Creek and Holmes Creek. • Implement bund wall diversions • Continue groundwater quality monitoring in dedicated groundwater monitoring bore suite.

10.0 GROUNDWATER MONITORING STRATEGY

10.1 Groundwater Monitoring Network

The current groundwater monitoring bore network is in good condition and is well spaced and located to detect changes in groundwater conditions from existing and proposed mining activities. Additionally it monitors groundwater levels and groundwater quality in both shallow and deep aquifer sequences of the prime aquifer.

It is considered that the groundwater monitoring bore suite is both appropriate and adequate for its intended purpose.

It would be of significant advantage if the groundwater level in the Rural Fire Board bore could be measured at the same times as the remainder of the dedicated groundwater monitoring bores.

10.2 Groundwater Monitoring Frequency

It is recommended that the frequency of groundwater level measurements, and samples for full chemical analysis should be increased to monthly from December 2012 until May 2013 to assess the impacts of rainfall during the forthcoming wet season on groundwater levels and groundwater quality.

After May 2013 groundwater levels should be measured monthly and samples for full chemical analysis should be collected quarterly.

The data that are collected will provide the basis for statistical analysis and groundwater contaminant trigger limits for an Environmental Authority for the Carbine Tungsten operation.

11.0 RECOMMENDATIONS

It is recommended that:

1. The frequency of groundwater level and groundwater quality sampling should be increased to monthly from December 2012 until May 2013.

2. Groundwater levels should be measured monthly from now on.

3. Samples for full chemical analysis should be collected quarterly after May 2013.

4. All groundwater data should be reviewed and an annual groundwater review report should be compiled in about October 2013.

Rob Lait and Associates Pty Ltd

ROB LAIT
Principal Hydrogeologist

LIMITATIONS OF REPORT

Rob Lait and Associates Pty Ltd (RLA) has prepared this report for the use of Carbine Tungsten Pty Ltd in accordance with the usual care and thoroughness of the consulting profession. It is based on generally accepted practices and standards at the time it was prepared. No other warranty, expressed or implied, is made as to the professional advice included in this report.

The methodology adopted and sources of information used by RLA are outlined in this report. RLA has made no independent verification of this information beyond the agreed scope of works and RLA assumes no responsibility for any inaccuracies or omissions. No indications were found during our investigations that information contained in this report as provided to RLA was false.

This study was undertaken between 5th November and 17th December 2012 and is based on the conditions encountered and the information available at the time of preparation of the report. RLA disclaims responsibility for any changes that may occur after this time.

This report should be read in full. No responsibility is accepted for use of any part of this report in any other context or for any other purpose or by third parties. It may not contain sufficient information for the purposes of other parties or other users. This report does not purport to give legal advice. Legal advice can only be given by qualified legal practitioners.

This report contains information obtained by inspection, sampling, testing and other means of investigation. This information is directly relevant only to the points in the ground where they were obtained at the time of the assessment. Where borehole logs are provided they indicate the inferred ground conditions only at the specific locations tested. The precision with which conditions are indicated depends largely on the frequency and method of sampling, and the uniformity of the site, as constrained by the project budget limitations. The behaviour of groundwater is complex.

Our conclusions are based upon the analytical data presented in this report and our experience.

Where conditions encountered at the site are subsequently found to differ significantly from those anticipated in this report, RLA must be notified of any such findings and be provided with an opportunity to review the recommendations of this report.

Whilst to the best of our knowledge, information contained in this report is accurate at the date of issue, subsurface conditions, including groundwater levels can change in a limited time. Therefore this document and the information contained herein should only be regarded as valid at the time of the investigation unless otherwise explicitly stated in this report.

Appendix C Life of Mine Schedule

Period Schedule

2% HG, 6Mtpa extraction limit, $31,500 WO3 price													
Period Name	Total	2022.00	2023.00	2024.00	2025.00	2026.00	2027.00	2028.00	2029.00	2030.00	2031.00	2032.00	2033.00
Duration (years)	11.79	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	0.79
Cash Flow ($M)	206.9	6.4	8.1	38.6	95.4	7.7	7.5	7.4	7.5	7.5	7.5	7.5	5.9
Policies													
Cut Off Grade: Cut-off grade	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%
Key Material Properties													
Total Period:Mass (kt)	25,201.45	1,000.00	5,993.19	5,999.79	4,418.43	1,014.37	1,000.00	989.58	1,000.00	1,000.00	1,000.00	1,000.00	786.09
To Ore:Mass (kt)	11,382.26	1,000.00	948.62	870.64	783.67	1,003.67	1,000.00	989.58	1,000.00	1,000.00	1,000.00	1,000.00	786.09
To Ore:Grade_WO3 (%)	0.146%	0.075%	0.198%	0.361%	0.637%	0.077%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%
To Ore:TotalTungsten (kt)	16.61	0.75	1.88	3.14	4.99	0.77	0.75	0.74	0.75	0.75	0.75	0.75	0.59
ROM Ore Grade	0.146%	0.075%	0.198%	0.361%	0.637%	0.077%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%	0.075%
To Waste:Mass (kt)	13,819.19		5,044.57	5,129.15	3,634.76	10.70							
To Waste:Ore_Mass (kt)	8.23		5.95		2.28								
To Waste:TotalTungsten (kt)	0.01		0.00		0.00								
Waste Ore Grade	0.075%		0.075%		0.075%								
To Fines:Mass (kt)	4,097.61	360.00	341.50	313.43	282.12	361.32	360.00	356.25	360.00	360.00	360.00	360.00	282.99
To Fines:Fines_Tungsten (kt)	8.97	0.41	1.01	1.70	2.70	0.42	0.41	0.40	0.41	0.41	0.41	0.41	0.32
Fines Ore Grade	0.219%	0.113%	0.297%	0.541%	0.955%	0.115%	0.113%	0.113%	0.113%	0.113%	0.113%	0.113%	0.113%
To Bypass:Mass (kt)	6.41		1.03	1.62	3.75								
To Bypass:Bypass_Tungsten (kt)	0.18		0.03	0.04	0.11								
Bypass Ore Grade	2.831%		2.895%	2.744%	2.851%								
To Ore Sorter Feed:Sorter_Feed_Mass (kt)	7,278.24	640.00	606.08	555.59	497.79	642.35	640.00	633.33	640.00	640.00	640.00	640.00	503.10
To Ore Sorter Product:Mass (kt)	670.15	44.79	65.47	88.90	122.13	45.36	44.79	44.33	44.79	44.79	44.79	44.79	35.21
To Ore Sorter Product:Sorter_Tungsten (kt)	6.71	0.31	0.75	1.26	1.97	0.32	0.31	0.31	0.31	0.31	0.31	0.31	0.24
Sorter Product Ore Grade	1.002%	0.693%	1.146%	1.417%	1.613%	0.705%	0.693%	0.693%	0.693%	0.693%	0.693%	0.693%	0.693%
To Gravity Plant Feed - Fines Mass:Mass (kt)	4,097.61	360.00	341.50	313.43	282.12	361.32	360.00	356.25	360.00	360.00	360.00	360.00	282.99
To Gravity Plant Feed - Coarse Mass:Mass (kt)	676.56	44.79	66.50	90.53	125.88	45.36	44.79	44.33	44.79	44.79	44.79	44.79	35.21
To Gravity Plant Feed:Mass (kt)	4,774.18	404.79	408.00	403.96	408.00	406.68	404.79	400.58	404.79	404.79	404.79	404.79	318.20
To Gravity Plant Feed - Tungsten Fines:Mass (kt)	8.97	0.41	1.01	1.70	2.70	0.42	0.41	0.40	0.41	0.41	0.41	0.41	0.32
To Gravity Plant Feed - Tungsten Coarse:Mass (kt)	6.90	0.31	0.78	1.30	2.08	0.32	0.31	0.31	0.31	0.31	0.31	0.31	0.24
Gravity Plant Feed Grade	0.332%	0.177%	0.440%	0.743%	1.170%	0.181%	0.177%	0.177%	0.177%	0.177%	0.177%	0.177%	0.177%
To Produced Concentrate - Fines:Mass (kt)	14.26	0.64	1.61	2.70	4.29	0.66	0.64	0.64	0.64	0.64	0.64	0.64	0.51
To Produced Concentrate - Coarse:Mass (kt)	12.41	0.56	1.40	2.35	3.74	0.58	0.56	0.55	0.56	0.56	0.56	0.56	0.44
To Total Produced Concentrate:Mass (kt)	26.68	1.20	3.02	5.04	8.02	1.24	1.20	1.19	1.20	1.20	1.20	1.20	0.95
To Gravity Plant Tailings:Mass (kt)	4,747.50	403.59	404.98	398.91	399.98	405.44	403.59	399.39	403.59	403.59	403.59	403.59	317.26
Source Material Properties													
To OC Mining:Mass (kt)	15,075.78		5,243.19	5,549.79	4,268.43	14.37							
Mined from PhaseOC3B:Mass (kt)	802.47		787.32	15.15									
Mined from PhaseOC3B:Ore_Mass (kt)	113.93		109.87	4.06									
PhaseOC3B Strip Ratio	7.04		7.17	3.73									
Mined from PhaseOC3B:TotalTungsten (kt)	0.89		0.87	0.03									
PhaseOC3B ROM Ore Grade	0.783%		0.789%	0.626%									
Mined from PhaseOC3B:Total_Product_Mass (kt)	62.61		60.50	2.11									
Mined from PhaseOC3B:Total_Product_Tungsten (kt)	0.85		0.83	0.02									
Mined from PhaseOC3B: Waste (kt)	688.54	0.00	677.45	11.09	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PhaseOC3B Product Ore Grade	1.364%		1.372%	1.148%									
Mined from PhaseOC4B:Mass (kt)	14,273.31		4,455.87	5,534.64	4,268.43	14.37							
Mined from PhaseOC4B:Ore_Mass (kt)	1,150.89		94.70	416.58	635.95	3.67							

	Total	2022.00	2023.00	2024.00	2025.00	2026.00	2027.00	2028.00	2029.00	2030.00	2031.00	2032.00	2033.00
PhaseOC4B Strip Ratio	**12.40**		47.05	13.29	6.71	3.92							
Mined from PhaseOC4B:TotalTungsten (kt)	**8.13**		0.45	2.78	4.88	0.02							
PhaseOC4B ROM Ore Grade	**0.707%**		0.478%	0.667%	0.767%	0.601%							
Mined from PhaseOC4B:Total_Product_Mass (kt)	**616.10**		46.32	219.69	348.20	1.89							
Mined from PhaseOC4B:Total_Product_Tungsten (kt)	**7.77**		0.43	2.65	4.67	0.02							
Mined from PhaseOC4B: Waste (kt)	**13122.42**	0.00	4361.17	5118.07	3632.48	10.70	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PhaseOC4B Product Ore Grade	**1.262%**		0.933%	1.208%	1.340%	1.114%							
Overall Strip Ratio	**11.92**		25.63	13.19	6.71	3.92							
Mined from PhaseLGS1:Mass (kt)	**4,489.58**	1,000.00	750.00	450.00	150.00	1,000.00	1,000.00	139.58					
Mined from PhaseLGS1:Ore_Mass (kt)	**4,489.58**	1,000.00	750.00	450.00	150.00	1,000.00	1,000.00	139.58					
Mined from PhaseLGS1:Total_Product_Mass (kt)	**1,817.35**	404.79	303.59	182.16	60.72	404.79	404.79	56.50					
Mined from PhaseLGS1:Total_Product_Tungsten (kt)	**3.21**	0.72	0.54	0.32	0.11	0.72	0.72	0.10					
Mined from PhaseLGS1: Waste (kt)	**0.00**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PhaseLGS1 Ore Grade	**0.177%**	0.177%	0.177%	0.177%	0.177%	0.177%	0.177%	0.177%					
Mined from PhaseLGS2:Mass (kt)	**5,636.09**							850.00	1,000.00	1,000.00	1,000.00	1,000.00	786.09
Mined from PhaseLGS2:Ore_Mass (kt)	**5,636.09**							850.00	1,000.00	1,000.00	1,000.00	1,000.00	786.09
Mined from PhaseLGS2:Total_Product_Mass (kt)	**2,281.45**							344.07	404.79	404.79	404.79	404.79	318.20
Mined from PhaseLGS2:Total_Product_Tungsten (kt)	**4.03**							0.61	0.72	0.72	0.72	0.72	0.56
Mined from PhaseLGS2: Waste (kt)	**0.00**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PhaseLGS2 Ore Grade	**0.177%**							0.177%	0.177%	0.177%	0.177%	0.177%	0.177%
To FEL Material Movement:Mass (kt)	**17,990.35**	1,595.21	1,489.23	1,337.32	1,159.34	1,600.66	1,595.21	1,578.59	1,595.21	1,595.21	1,595.21	1,595.21	1,253.97
Cash Flow Breakdown													
Terminal Value ($M)													
Period Time Value ($M)	**(30.0)**	(2.0)	(3.8)	(3.5)	(3.6)	(2.4)	(2.2)	(2.2)	(2.2)	(2.2)	(2.2)	(2.2)	(1.7)
OC Mining cost - Phase 3B: Cash Flow Total ($M)	**(3.6)**		(3.5)	(0.1)									
OC Mining cost - Phase 4B: Cash Flow Total ($M)	**(64.2)**		(20.1)	(24.9)	(19.2)	(0.1)							
LGS Mining cost - LGS1: Cash Flow Total ($M)	**(7.5)**	(1.7)	(1.3)	(0.8)	(0.3)	(1.7)	(1.7)	(0.2)					
LGS Mining cost - LGS2: Cash Flow Total ($M)	**(9.5)**							(1.4)	(1.7)	(1.7)	(1.7)	(1.7)	(1.3)
Dry Processing cost: Cash Flow Total ($M)	**(22.8)**	(2.0)	(1.9)	(1.7)	(1.6)	(2.0)	(2.0)	(2.0)	(2.0)	(2.0)	(2.0)	(2.0)	(1.6)
Ore Sorting cost: Cash Flow Total ($M)	**(7.7)**	(0.7)	(0.6)	(0.6)	(0.5)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.5)
Gravity Plant cost: Cash Flow Total ($M)	**(52.5)**	(4.5)	(4.5)	(4.4)	(4.5)	(4.5)	(4.5)	(4.4)	(4.5)	(4.5)	(4.5)	(4.5)	(3.5)
Tailings cost: Cash Flow Total ($M)	**(1.6)**	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)
Rehabiliation & Closure cost: Cash Flow Total ($M)	**(3.0)**		(1.0)	(1.1)	(0.9)	(0.0)							
FEL cost: Cash Flow Total ($M)	**(15.1)**	(1.3)	(1.2)	(1.1)	(1.0)	(1.3)	(1.3)	(1.3)	(1.3)	(1.3)	(1.3)	(1.3)	(1.1)
Ancillary cost: Cash Flow Total ($M)	**(0.9)**		(0.3)	(0.3)	(0.3)	(0.0)							
State Govt Royalty cost: Cash Flow Total ($M)	**(11.5)**	(0.5)	(1.3)	(2.1)	(3.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.4)
Tungsten revenue: Cash Flow Total ($M)	**425.1**	18.2	46.9	78.4	129.8	20.0	19.5	19.3	19.5	19.5	19.5	19.5	15.3
Quarry Product revenue: Cash Flow Total ($M)	**11.7**	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.8
Other Material Properties													
To Ore Sorter Waste:Mass (kt)	**6,608.09**	595.21	540.62	466.68	375.67	596.99	595.21	589.01	595.21	595.21	595.21	595.21	467.88
To ADT Material Movement:Mass (kt)	**31,809.54**	1,595.21	6,533.80	6,466.47	4,794.10	1,611.36	1,595.21	1,578.59	1,595.21	1,595.21	1,595.21	1,595.21	1,253.97

Appendix D Liebherr R9100 Excavator Product Sheet

Mining Excavator

R 9100

Generation 6

Operating Weight
Backhoe Configuration
113 tonnes / 125 tons

Face Shovel Configuration
116 tonnes / 128 tons

Engine Power
565 kW / 757 HP

Standard Bucket
Backhoe Configuration
7.0 – 7.5 m^3 / 9.2 – 9.8 yd^3

Face Shovel Configuration
7.3 m^3 / 9.6 yd^3

LIEBHERR

Productivity
Working Harder and Faster

Efficiency
Moving More for Less

Reliability
Ready to Work
When You Need It

Operating Weight
Backhoe Configuration
113 tonnes / 125 tons

Face Shovel Configuration
116 tonnes / 128 tons

Engine Power
565 kW / 757 HP

Standard Bucket
Backhoe Configuration
7.0 – 7.5 m^3 / 9.2 – 9.8 yd^3

Face Shovel Configuration
7.3 m^3 / 9.6 yd^3

Customer Service
World-Class Support,
Everywhere, Every Day

Safety
Protecting Your Most
Important Assets

Environment
Mining Responsibly

Productivity

Working Harder and Faster

Efficient and effective by design, the R 9100 B sets a new standard in job performance and functions as the optimal tool for loading 50 t up to 100 t off-highway trucks.

Fast and Precise Movement

Liebherr Engine V12
The R 9100 B is equipped with the long-lasting and proven Liebherr V12 diesel engine specifically designed to withstand extreme outside temperatures and high altitudes with low atmospheric pressure. Integrating the latest engine management system, the R 9100 B is built for extreme conditions.

Fast Cycle Time
Like all other Liebherr mining excavators, the R 9100 B uses a closed-loop swing circuit. The main hydraulic circuit comprises a combination of three independent main valves fed by three working pumps, providing unrivaled flexibility of attachment control and force distribution, while allowing full oil flow integration for fast movement and lowest cycle times.

Precise Machine Motions
The R 9100 B's hydraulic control system is optimized in order to improve combined machine motions. The ergonomically mounted joysticks on the suspended seat armrests allow the operator to precisely position the machine.

High Digging and Lifting Capabilities

High Digging Forces
Designed for the best mechanical force distribution, the production-tailored attachment delivers tough digging and lifting forces. Integrating Liebherr-made cylinders and a wide range of buckets with mining optimized GET, the R 9100 B's attachment ensures the highest forces, easy bucket penetration and high fill factor to perform even in the most demanding conditions.

Power-Oriented Energy Management
The R 9100 B's attachment is equipped with the pressureless boom-down function to enable fast cylinder retraction without the need for pump energy. Intelligent energy management diverts the pump flow during boom lowering, allowing other cylinder motions to operate unimpeded.

Liebherr Diesel Engine
- V12 by Liebherr
- US EPA Tier 2, US EPA Tier 4f / EU Stage V compliant
- Automatic idle control
- Max. altitude without derating: 3,600 m
- Eco-Mode selector

Liebherr Site-Specific Bucket
- 4 to 5 passes to load a 50 t off-highway truck
- 3 types of wear package
- Maximal bucket fill factor
- Integrated approach on machine capabilities
- Customized solutions according to customer application

Exclusive EVO Bucket Solution
- Liebherr patented EVO design to maximize the loading capacity
- Optimized Liebherr GET and wear package according to customer application
- Ensures optimal penetration efficiency
- Single GET hammerless locking system for safe and easy maintenance
- Fully patented GET system design for optimal penetration / lifetime
- 4 tooth profiles available for various range of applications

Efficiency

Moving More for Less

The R 9100 B follows the Liebherr design philosophy of maximizing a machine's performance by improving the efficiency of all individual subsystems. Engineered for easy serviceability, the machine is designed to ensure maximum uptime. The R 9100 B's modern cab creates a comfortable working environment, ensuring peak operator performance at every shift.

Built for Maximum Profitability

Hydraulic System Efficiency
Liebherr advanced hydraulic technology contributes to the R 9100 B's energy optimization. The high-pressure hydraulic system and the optimized pipe and hose layout maximize usable power transmission. The hydraulic pumps are managed to provide optimal pressure compensation and oil flow management. The hydraulic system is independently regulated over the engine circuit for the best operational efficiency.

Closed Loop Swing Circuit
The Liebherr Mining excavators are all equipped with a closed loop swing circuit. The kinematic energy is recovered when the swing motion is used during deceleration, to drive the main and auxiliary pumps, reducing fuel consumption and allowing faster boom lift motion.

Independent Cooling System
Oil and water cooling fans are independent and electronically managed. The on-demand cooling control enables to maximize available power for the working process. This technology contributes to maintain sustainable temperature of all the hydraulic components extending their life.

Comfortable Cab for Efficient Work

Superior Operator Comfort
The modern large cab provides ideal working conditions and optimal operator's comfort. Mounted on silent blocks, the R 9100 B's cab design reduces vibrations. The new headliner limits noise pollution to provide a quiet working environment.

Extended Components Lifetime

The R 9100 B's hydraulic oil filtration systems remove fluid contaminants to offer the highest rate of hydraulic components durability. To maintain oil quality, all return hydraulic oil flow goes through a fine filtration system (15/5 µm) and oil tank is sized to considerably extend the time between service intervals.

Advanced Machine Monitoring
- 10.5" LCD color screen
- Information interface to operator
- On-board diagnostics to service staff
- Real text information
- Long term data storage for maintenance

First-Class Service Arrangements
Service friendly design allows easy and fast maintenance for maximum uptime:
- Service from one-side
- Large catwalks and walkways
- Refillable grease tanks instead of drums to be changed
- Centralized lubrication system
- Enhanced single-line lubrication system

Comfort-Oriented Cab Design
- Tinted laminated safety glass
- Armored front window
- Adjustable air suspended seat
- A/C with dust filter in fresh/recirculated air
- Pressurization to prevent dust penetration (optional)
- Optional Operator Comfort Kit: sun blinds, bottle cooler, reading light, premium seat with cooling/airing system, electronic weight adjustment
- Pre-heating system (optional)

Reliability

Ready to Work When You Need It

With over 50 years of innovative thinking, engineering and manufacturing excellence, Liebherr sets the industry standard for advanced equipment design and technology tools to provide the most up-to-date product, responding to the requirements of mining customers.

Quality: the Liebherr Trademark

Structure Made Exclusively for Mining
Liebherr mining excavators are conceptualised, designed and dedicated to the mining industry. The engineering department uses specific 3D solution in order to meet possible requirements, such as Finite Element and Fatigue Life Analysis. In combination, the manufacturing department uses advanced welding techniques to strategically reinforce the structure. The synergy of our skills allows to obtain maximal machine availability.

Reinforced Undercarriage Structure
Specifically designed for extreme mining conditions, the rugged R 9100 B undercarriage represents the basis for the stability of the machine. Developed and built for both shovel and backhoe configurations, the enlarged undercarriage offers an efficient ground bearing pressure management providing the necessary stability and reliability. The access to the travel motors and brakes has been designed to provide maximum protection to the components, while providing easy and fast service access.

Long-lasting Job Performances

Maximized Components Lifetime
The R 9100 B is equipped with an automatic single line lubrication system for the entire attachment and swing ring. All greasing points are suitably protected against external damages, extending component life and ensuring constant performance over the excavator's operational life.

Liebherr Components Integration
As an OEM, Liebherr has built a solid reputation for its development and production of high quality strategic mining components. The R 9100 B integrates robust and reliable mining optimized components that are developed, manufactured and controlled by Liebherr, ensuring reliability and high performance for the entire machine.

Liebherr Component Integration
- Diesel engine
- Hydraulic pumps and motors
- Electronic and control technology
- Swing and travel drives
- Hydraulic cylinders
- Splitter box
- Swing ring
- GET

Quality Commitment
- Liebherr-Mining Equipment Colmar, France, ISO 9001 certified
- Compliance of materials tested in laboratory
- Quality control during the stages of production
- Vertical integration practice

Arctic Package (optional)
Designed for reliability in regions with temperatures of down to – 50 °C / – 58 °F:
- Integrated into machine structure
- Start up easily even at very low temperatures
- Increases machine availability and components lifetime
- Optimum operator comfort even in harsh temperature conditions
- Facilitate machine servicing

Customer Service

World-Class Support, Everywhere, Every Day

By partnering with our customers, Liebherr implements tailored solutions from technical support, spare parts and logistics solutions to global maintenance for all types of equipment, all over the world.

Customer Support

International Service Organization
The Liebherr Service Support has always been an important focus for the company. Complete service during all operating phases from machinery installation to problem solving, spare parts inventory and technical service. Our service team is close to our customers, delivering the best specific maintenance solution to reduce both equipment downtime and repair costs.

Complete Training Programs
The Liebherr Mining Training System provides blended training sessions for operator and maintenance staff to encourage productive, cost-effective and safe mining operation. The Liebherr Mining Training System employs online learning programs, factory and on-site sessions and simulator training.

Troubleshoot Advisor Platform

- Unique maintenance system to help you identify problems
- Easy and friendly-user interface
- Compatible with mobile, tablet or laptop
- Regular updating of the database
- Procedures described by specialist with images and videos

Remanufacturing

Reduced Costs and Investment
Over the course of a mining machine's lifetime, major components must be replaced to ensure continued safety, productivity and reliability. The Liebherr Mining Remanufacturing Program offers customers an OEM alternative to purchasing brand new replacement components. Enabling customers to achieve lowest possible equipment lifecycle costs without compromising quality, performance or reliability.

Fast Availability
A international service network and component facilities worldwide means that component repair services and exchange components are available to customers regardless of their location.

Connectivity Kit

- Machine is serially equipped with GSM data transmission functionalities
- Collection of operating parameters + error codes / machine faults
- Data access through the Liebherr-Mining Data platform (LMD)
- Customized reports accessible on LMD to track & analyze machine data
- Monitor & follow your fleet
- Maintenance prediction, machine troubleshooting and uptime optimization

Genuine Parts

Performance
Using genuine Liebherr components ensures the best interaction within your machine, encouraging optimal performance and most effective machine operation. For all major components, Liebherr relies on its Liebherr Maintenance Management System to follow and monitor service life while predicting maintenance activities.

Partnership
Liebherr regularly reviews requirements for parts and components for individual machines, based on operating hours, consumption and planned maintenance, resulting in minimized down time for customers. With access to the Global stock via all Liebherr Mining Warehouses, you will improve productivity by having the part you need, when you need it.

MyLiebherr Customer Portal

- Easy access parts online
- Available any time anywhere
- User friendly interface
- Online ordering
- Save time and money

Safety

Protecting Your Most Important Assets

The Liebherr R 9100 B provides uncompromising safety for operators and maintenance crew. As it is designed to be serviced from one side, the R 9100 B allows effortless access facilities to the major service points for quick and safe maintenance. The R 9100 B offers numerous features for operator safety.

Safety-First Working Conditions

Safe Service Access
The R 9100 B is fitted with ergonomic access for fast and safe maintenance. All service points are within reach from one side and at machine level. The R 9100 B's upperstructure is accessible via a robust fixed ladder and integrates one large central platform equipped with slip resistant surfaces. The wide catwalks facilitate maintenance and ensure comfort during all the operations.

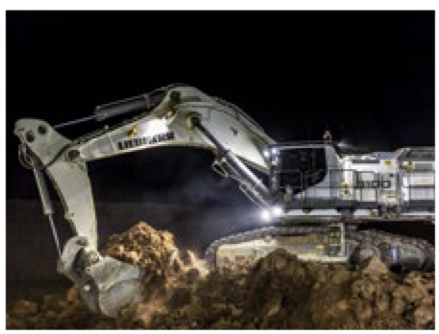
Secure Maintenance
All components have been located to allow for effortless inspection and replacement. Numerous service lights are perfectly located in the service areas to guaranty suitable maintenance conditions, day or night. Emergency stops have been strategically placed in the cab and engine compartment (at ground level in option). The R 9100 B eliminates hazards to ensure a safe environment for the service staff during maintenance.

Efficient Machine Protection

Protection Against Fire Ignition
The engine compartment integrates a bulkhead wall that separates the engine from the hydraulic pumps. This reduces the risk of hydraulic oil entering the engine compartment. The turbochargers and exhaust systems are heat shielded, and all the hydraulic hoses are made from a fire resistant material.

Automatic Fire Suppression System
The R 9100 B can be equipped with a fully integrated fire suppression, employing a dual agent solution to prevent and protect the machine. The fire suppression system has both automatic and manual release capabilities, E-stops devices are strategically located in the cab and over the machine to be easily accessible in any case by the operator or maintenance.

User Friendly Maintenance
- Wide walkways with slip-resistant surfaces
- Emergency ladder available outside the cab
- Wide open service access
- Reflective stripes on counterweight
- 45° hydraulic driven access stair (optional)



Working Environment Control
- Rear and side camera system
- LCD color screen to display cameras view
- 9 long-range working LED lights located on attachment and upperstructure

Commitment to Employees Safety
- Safe and protected access to the components
- Major components centralized to be easily accessible
- Optional ground-level fluid maintenance hub
- E-stops located for the operator and maintenance staff
- FOPS: Falling Object Protective Structure (optionnal)

Environment

Mining Responsibly

Liebherr considers the conservation and preservation of the environment as a major challenge for the present and future. Liebherr are considerate of environmental issues in designing, manufacturing and managing machine structures, providing solutions that allow customers to balance performance with environmental consciousness.

Minimized Impact on Life

Optimized Energy Consumption, Fewer Emissions
Constant power regulation of the hydraulic system and engine output optimize equipment fuel efficiency, depending on the application. In “Eco-Mode” setting, the machine is set up to reduce engine load, significantly improve fuel consumption and reduce emissions.

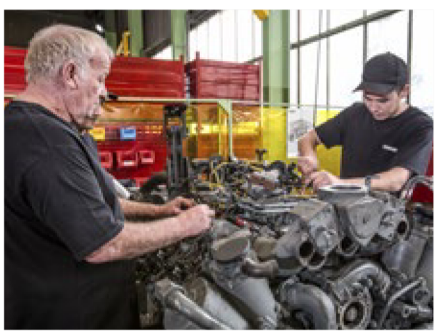
Controlled Emission Rejection
The R 9100 B is powered by a high horsepower diesel engine which complies with the US EPA Tier 2 or US EPA Tier 4f/EU Stage V compliant emission limits. This power drive makes the R 9100 B cost effective without compromising productivity and reduces the machines impact on the environment.

Sustainable Design and Manufacturing Process

Certified Environment Management Systems
Subject to the stringent European program for the regulation of the use of chemical substances in the manufacturing process REACH*, Liebherr undertakes a global evaluation to minimize the impacts of hazardous material, pollution control, water conservation, energy and environmental campaigns.

Extended Components and Fluids Lifetime
Liebherr is constantly working on ways to extend component life. Through the Liebherr-Mining Remanufacturing Program, superior lubrication systems and the reinforcement of parts under stress, Liebherr can reduce frequency of part replacement. The result minimizes environmental impact and lowers the overall total cost of ownership.

*REACH is the European Community Regulation on chemicals and their safe use (EC 1907/2006)
It deals with the Registration, Evaluation, Authorization and Restriction of Chemical Substances.



The Liebherr-Mining Remanufacturing Program
- Reduced environmental impact
- Second life for your components
- Reduced costs and investment
- Liebherr certified workshops
- Alternative to purchase brand-new replacement components

Eco-Mode
The Eco-Mode can be manually selected by the operator when maximal power is not required according to job need for:
- An improved fuel efficiency
- Less load on the engine
- Less noise pollution
- Less dioxide carbon emissions

Automatic Idle Control
Electronic idle control of the engine results in:
- Less fuel consumption
- Less load on the engine
- Reduced emissions
- More comfort to the operator (reduced noise pollution)

Technical Data

Engine

1 Liebherr diesel engine	
Rating per ISO 9249	565 kW (757 HP) at 1,800 rpm
Model	Liebherr D9512 (US EPA Tier 2, US EPA Tier 4f/EU Stage V compliant)
Type	V12 cylinder engine
Bore/Stroke	128/157 mm / 5.04/6.18 in
Displacement	24.24 l/1,479 in³
Engine operation	4-stroke diesel common-rail direct injection turbo-charged
Cooling	water-cooled, hydrostatic fan drive
Air cleaner	dry-type air cleaner with pre-cleaner, primary and safety elements, automatic dust discharge
Fuel tank capacity	1,478 l/390 gal (2,580 l/682 gal optional)
Engine idling	electronically controlled
Electrical system	
Voltage	24 V
Batteries	4 x 75 Ah/12 V
Starter	24 V/2 x 8.4 kW
Alternator	24 V/140 A
RPM adjustment	brushless adjustment of engine output via rpm selector

Hydraulic Controls

Power distribution	via monoblock control valves with integrated primary and secondary relief valves
Flow summation	to attachment and travel drive
Closed-loop circuit	for uppercarriage swing drive
Servo circuit	
Attachment and swing	proportional via hydraulic joystick levers
Travel	proportional via hydraulic pedals or removable hand levers
Shovel flap functions	proportional via hydraulic pedals

Swing Drive

Hydraulic motor	2 Liebherr axial piston motors
Swing gear	2 Liebherr planetary reduction gears
Swing ring	Liebherr, sealed single race ball bearing swing ring, internal teeth
Swing speed	0 – 6.0 rpm
Swing-holding brake	wet multi-disc brakes, spring applied, hydraulically released

Hydraulic System

Hydraulic pump	
for attachment and travel drive	3 Liebherr variable flow axial piston pumps
Max. flow	3 x 435 l/min./3 x 115 gpm
Max. pressure	350 bar/5,076 psi
for swing drive	1 Liebherr reversible swashplate pump, closed-loop circuit
Max. flow	420 l/min./111 gpm
Max. pressure	350 bar/5,076 psi
Pump management	electronically controlled pressure and flow management with oil flow optimisation
Hydraulic tank capacity	1,000 l/264 gal
Hydraulic system capacity	1,500 l/396 gal
Hydraulic oil filter	1 high pressure safety filter after each high pressure pump + extra-fine filtration of entire return flow with integrated by-pass filtration (15/5 µm) + dedicated leak-oil filtration
Hydraulic oil cooler	1 separated cooler, temperature controlled fan driven via 1 hydraulic piston motor
MODE selection	adjustment of machine performance and the hydraulics via a mode selector to match application
ECO	for economical operation (can be combined with fuel optimized setting)
POWER	for maximum digging power and heavy duty jobs

Electric System

Electric isolation	easy accessible battery isolators
Working lights	high brightness LED lights: – 2 on working attachment – 2 on cabin – 2 on RHS of uppercarriage – 3 on LHS of uppercarriage
Emergency stop switches	in the cab / in engine compartment
Electrical wiring	heavy duty execution in IP 65 standard for operating conditions of –50 °C to 100 °C / –58 °F to 212 °F

Uppercarriage

Design	torque resistant modular design upper frame
Attachment mounting	parallel length girders
Catwalks	large catwalk on the left-hand side

Operator's Cab

Design	sound insulated, tinted windows, front window armored glass, door with sliding window
Operator's seat	air suspended, body-contoured with shock absorber, adjustable to operator's weight
Joysticks	joystick levers integrated into armrest of seat, armrest adjusted to seat position
Condition monitoring	machine condition monitoring system with error reporting and operational information
Display	color LCD-display with low and high brightness settings, 1 additional fixation for supplementary customer device
Vision system	camera installation on counterweight and right-hand side of the uppercarriage, displayed over the LCD-display
Heating system / Air conditioning	standard automatic air conditioning, contains fluorinated greenhouse gases HFC 134a with a Global Warming Potential (GWP) of 1430, the AC circuit contains 1.7 kg / 3.8 lb of HFC-134 representing an equivalent of 2.4 tonnes / 2.7 tons of CO_2, combined cooler / heater, additional dust filter in fresh air / recirculated
Noise level (ISO 6396)	L_{pA} (inside cab) = 76 dB(A)

Undercarriage

Version HD	heavy duty
Drive	Liebherr swashplate motors
Travel gear	Liebherr planetary reduction gears
Travel speed	0 – 3.5 km/h / 0 – 2.17 mph
Track components	track pitch 280 mm / 11.02 in, maintenance-free
Track rollers / Carrier rollers	8 / 2 per side frame
Track pads	double grouser
Track tensioner	spring with grease tensioner
Parking brake	wet multi-discs (spring applied, pressure released)
Brake valves	integrated in main valve block

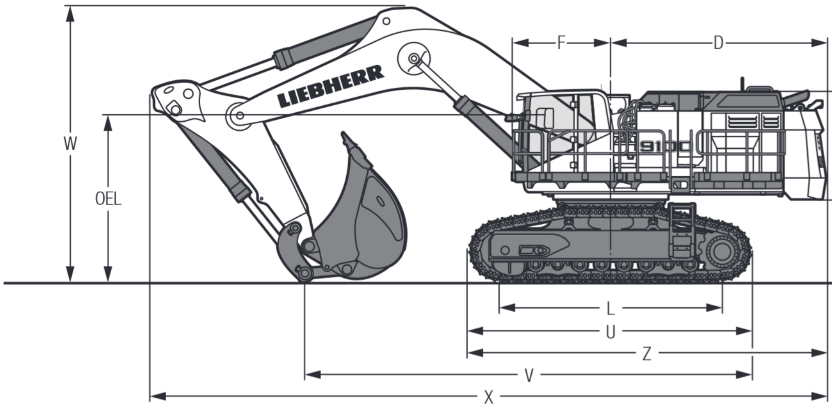
Central Lubrication System

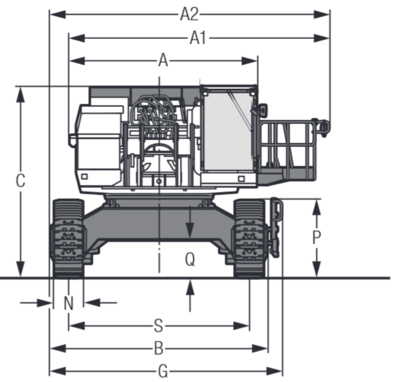
Type	single line lubrification system, for the entire attachment / swing ring bearing and teeth
Grease pumps	1 hydraulic pump for attachment / swing ring bearing lubrification, 1 electric pump for swing teeth lubrification
Capacity	27 l / 7.1 gal bulk container for attachment / swing ring bearing, separated 8 l / 2.1 gal container for swing ring teeth
Refill	via quick connections and grease filters for both containers

Attachment

Design	box-type, combination of resistant steel plates and cast steel components
Hydraulic cylinders	Liebherr design
Hydraulic connections	pipes and hoses equipped with SAE flange connections
Pivots	sealed, low maintenance
Pivots bucket-to-stick Pivots bucket-to-link	O-ring sealed and completely enclosed

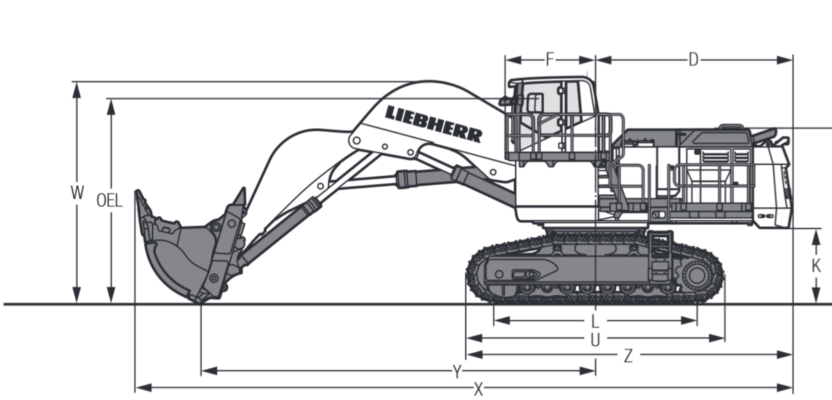
Dimensions

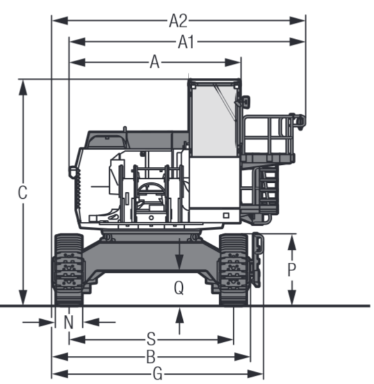




		mm / ft in
A		4,059 / 13'3"
A1		5,443 / 17'9"
A2		5,856 / 19'2"
B		4,685 / 15'4"
C		4,143 / 13'6"
D		4,630 / 15'2"
F		2,107 / 6'9"
G		4,995 / 16'4"
H		4,114 / 13'5"
K		1,803 / 5'9"
L		4,810 / 15'8"
N	500 / 1'6" 600 / 2'	750 / 2'5"
P		1,663 / 5'5"
Q		812 / 2'7"
S		3,900 / 12'8"
U		6,107 / 20'
Z		7,683 / 25'2"
OEL	Operator's eye level	3,533 / 11'6"

	Stick length m / ft in	Mono boom 7.60 m / 24'9" mm / ft in	Mono boom 9.20 m / 30'2" mm / ft in
V	3.20 / 10'5"	9,660 / 31'7"	11,445 / 37'6"
	4.50 / 14'8"	– / –	9,930 / 32'6"
	5.60 / 18'4"	– / –	9,890 / 32'5"
W	3.20 / 10'5"	6,035 / 19'8"	6,210 / 20'4"
	4.50 / 14'8"	– / –	6,800 / 22'3"
	5.60 / 18'4"	– / –	7,550 / 24'8"
X	3.20 / 10'5"	14,560 / 47'8"	16,080 / 52'8"
	4.50 / 14'8"	– / –	15,385 / 50'5"
	5.60 / 18'4"	– / –	14,825 / 48'6"

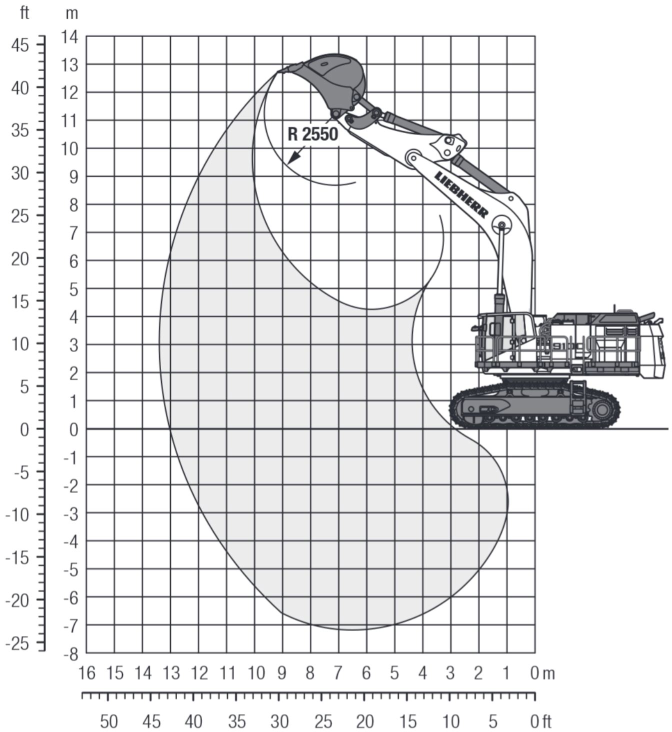




		mm / ft in
A		4,059 / 13'3"
A1		5,443 / 17'9"
A2		5,856 / 19'2"
B		4,685 / 15'4"
C		5,340 / 17'5"
D		4,630 / 15'2"
F		2,107 / 6'9"
G		4,995 / 16'4"
H		4,114 / 13'5"
K		1,803 / 5'9"
L		4,810 / 15'8"

		mm / ft in
N	500 / 1'6" 600 / 2'	750 / 2'5"
P		1,663 / 5'5"
Q		812 / 2'7"
S		3,900 / 12'8"
U		6,107 / 20'
V1		12,350 / 40'5"
W1		6,035 / 19'8"
X1		15,530 / 51'
Z		7,683 / 25'2"
OEL	Operator's eye level	4,733 / 15'5"

Backhoe Attachment (Standard)
with Boom 7.60 m / 24'9"



Digging Envelope

Stick length	**m**	3.20
	ft in	10'5"
Max. digging depth	**m**	7.15
	ft in	23'5"
Max. reach at ground level	**m**	13.00
	ft in	42'7"
Max. dumping height	**m**	8.65
	ft in	28'4"
Max. teeth height	**m**	12.70
	ft in	41'7"

Forces

Max. digging force (ISO 6015)	**kN**	415
	lbf	93,296
Max. breakout force (ISO 6015)	**kN**	560
	lbf	125,893

Operating Weight and Ground Pressure

The operating weight includes the basic machine with boom 7.60 m / 24'9", stick 3.20 m / 10'5" and bucket 7.50 m³ / 9.8 yd³.

Undercarriage		**HD**	
Pad width	**mm / ft in**	600 / 2'	750 / 2'5"
Weight	**kg / lb**	112,717 / 248,498	113,733 / 250,738
Ground pressure*	**kg/cm² / psi**	1.79 / 25.46	1.44 / 20.48

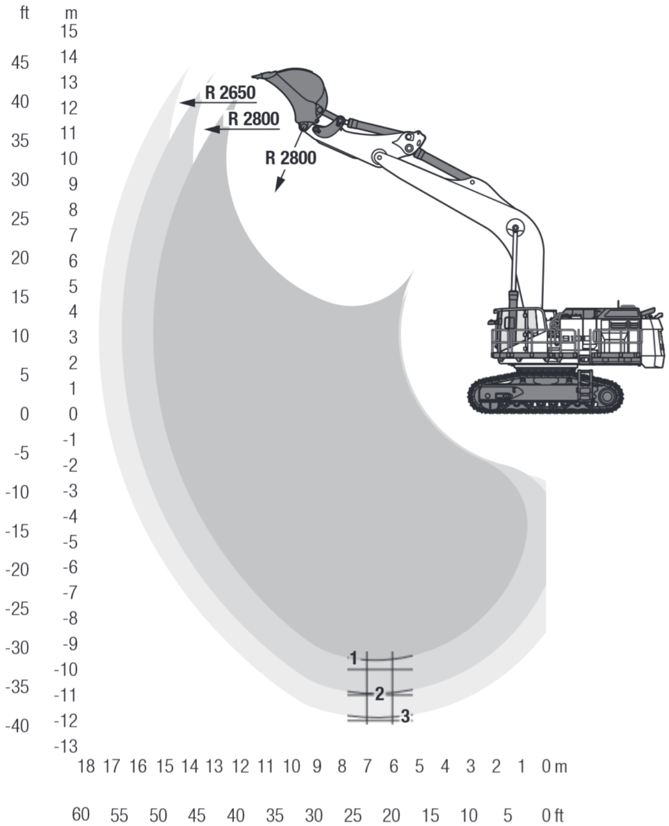
* according to ISO 16754

Backhoe Buckets

For materials class according to VOB, Section C, DIN 18300		< 5	< 5	< 5	5 – 6	5 – 6	5 – 6	5 – 6	7 – 8	7 – 8	7 – 8
Typical operation according to VOB Section C, DIN 18300		GP	GP	GP	HD	HD	HD	HD	XHD	XHD	XHD
Capacity ISO 7451	**m³**	9.00	8.40	7.70	8.00	7.00	7.50	7.00	7.50	6.50	6.00
	yd³	11.8	11.0	10.1	10.5	9.2	9.8	9.2	9.8	8.5	7.8
Suitable for material up to a specific weight of	**t/m³**	1.5	1.65	1.8	1.65	1.8	1.8	1.95	1.65	2.0	2.2
	lb/yd³	2,528	2,781	3,034	2,781	3,034	3,034	3,287	2,781	3,371	3,708
Weight	**kg**	7,200	7,000	6,900	7,700	7,200	7,450	7,200	8,520	7,710	7,420
	lb	15,873	15,432	15,212	16,976	15,873	16,424	15,873	18,783	16,998	16,358

GP: General purpose bucket with Liebherr Z90 teeth
HD: Heavy-duty bucket with Liebherr Z100 teeth
XHD: Heavy-duty rock bucket with Liebherr Z100 teeth

Backhoe Attachment
with Boom 9.20 m / 30'2"



Digging Envelope

		1	2	3
Stick length	**m**	3.20	4.50	5.60
	ft in	10'5"	14'8"	18'4"
Max. digging depth	**m**	9.64	10.94	11.90
	ft in	31'6"	35'9"	39'
Max. reach at ground level	**m**	15.02	16.20	17.20
	ft in	49'3"	53'1"	56'4"
Max. dumping height	**m**	8.40	8.90	9.40
	ft in	27'6"	29'2"	30'8"
Max. teeth height	**m**	13.16	13.60	13.90
	ft in	43'2"	44'6"	45'6"

Forces

		1	2	3
Max. digging force (ISO 6015)	**kN**	410	330	285
	lbf	92,172	74,187	64,071
Max. breakout force (ISO 6015)	**kN**	530	530	530
	lbf	119,149	119,149	119,149

Operating Weight and Ground Pressure

The operating weight includes the basic machine with boom 9.20 m / 30'2", stick 4.50 m / 14'8" and bucket 4.50 m³ / 5.9 yd³.

Undercarriage		**HD**	
Pad width	**mm / ft in**	600 / 2'	750 / 2'5"
Weight	**kg / lb**	112,464 / 247,941	113,480 / 250,181
Ground pressure*	**kg/cm² / psi**	1.78 / 25.32	1.44 / 20.48

* according to ISO 16754

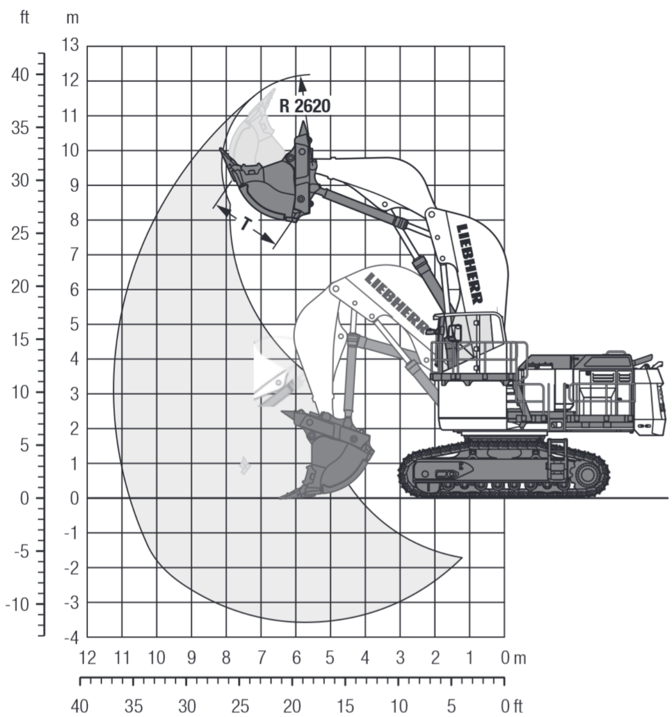
Backhoe Buckets

For materials class according to VOB, Section C, DIN 18300		< 5	5 – 6	5 – 6	5 – 6	5 – 6	5 – 6
Typical operation according to VOB Section C, DIN 18300		GP	HD	HD	HD	HD	HD
Capacity ISO 7451	**m³**	6.50	5.80	5.50	4.50	3.80	3.20
	yd³	8.5	7.6	7.2	5.9	5.0	4.2
Suitable for material up to a specific weight of							
with stick 3.20 m	**t/m³**	1.2	1.5	1.8	2.0	2.2	–
with stick 10'5"	**lb/yd³**	2,023	2,528	3,034	3,371	3,708	–
with stick 4.50 m	**t/m³**	–	1.2	1.4	1.8	2.0	2.2
with stick 14'8"	**lb/yd³**	–	2,023	2,361	3,034	3,371	3,708
with stick 5.60 m	**t/m³**	–	–	1.2	1.5	1.8	2.0
with stick 18'4"	**lb/yd³**	–	–	2,023	2,528	3,034	3,371
Weight	**kg**	6,800	7,100	6,300	5,300	4,600	4,000
	lb	14,991	15,653	13,889	11,685	10,141	8,819

GP: General purpose bucket with Liebherr Z90 teeth
HD: Heavy-duty bucket with Liebherr Z100 teeth

Face Shovel Attachment

with Boom 5.30 m/17'4"



Digging Envelope

Stick length	**m**	3.70
	ft in	12'1"
Max. reach at ground level	**m**	10.70
	ft in	35'1"
Max. dumping height	**m**	8.00
	ft in	26'2"
Max. crowd length	**m**	3.70
	ft in	12'1"
Bucket opening width T	**mm**	2,000
	ft in	6'6"

Forces

Max. crowd force at ground level (ISO 6015)	**kN**	545
	lbf	122,521
Max. crowd force (ISO 6015)	**kN**	704
	lbf	158,266
Max. breakout force (ISO 6015)	**kN**	585
	lbf	131,513

Operating Weight and Ground Pressure

The operating weight includes the basic machine with shovel attachment and bucket 7.30 m³/ 9.6 yd³.

Undercarriage		**HD**	
Pad width	**mm/ft in**	600/2'	750/2'5"
Weight	**kg/lb**	116,391/256,598	117,407/258,838
Ground pressure*	**kg/cm²/psi**	1.84/26.17	1.49/21.19

* according to ISO 16754

Face Shovel Buckets

For materials class according to VOB, Section C, DIN 18300		< 5	< 5	5 – 6	5 – 6	5 – 6	5 – 6	7 – 8	7 – 8	7 – 8
Typical operation according to VOB Section C, DIN 18300		GP	GP	HD	HD	HD	HD	XHD	XHD	XHD
Capacity ISO 7546	**m³**	9.00	7.80	7.80	7.30	6.70	5.90	7.30	6.70	5.80
	yd³	11.8	10.2	10.2	9.6	8.8	7.7	9.6	8.8	7.6
Suitable for material up to a specific weight of	**t/m³**	1.3	1.7	1.6	1.8	2.0	2.4	1.5	1.8	2.2
	lb/yd³	2,191	2,865	2,697	3,034	3,371	4,045	2,528	3,034	3,708
Weight	**kg**	12,700	11,500	12,200	11,600	11,200	10,600	13,400	12,600	11,800
	lb	27,999	25,353	26,896	25,574	24,692	23,369	29,542	27,778	26,015

GP: General purpose bucket with Liebherr Z90 teeth
HD: Heavy-duty bucket with Liebherr Z100 teeth
XHD: Heavy-duty rock bucket with Liebherr Z100 teeth

Optional Equipment

Undercarriage

Narrow track pad width
Large track pad width
Removable side frames
HD travel gear for muddy applications
Rock protection for idler wheel
Protection for undercarriage center frame
Full length chain guide

Uppercarriage

Increased fuel tank capacity (24h operation)
Grid protection for front headlights
Semi-automatic swing brake with joystick control
Wiggins couplings for ground level access service
Wiggins fast fueling system with Multiflo Hydrau-Flo
Steel grease lines on swing ring
Hydraulically operated 45° access stair
Swing ring scrapers
External grease refill station (hydraulic-powered)
Right-hand bumper
External starting device
Rock protection for swing gear and grease lines

Hydraulic System

Oil cooler inlet screen

Operator’s Cab

4-point seat belt
Cab elevation (500 mm/1'6" / 1,200 mm/3'9" / 1,600 mm/5'3")
Cab pressurization/cab pressurization with HEPA filter
FOPS top guard with additional sun protection
Operator comfort package
Front protective grid
Pre-heating system for cab
Roof glazing
External louvers

Attachment

Piston rod guard for bucket cylinder (BH)
Piston rod guard for hoist cylinder (FS)
Piston rod guard for stick cylinder (FS)
Quick change coupling

Specific Solutions

Arctic package (–20 °C / – 4 °F, – 35 °C / – 31 °F, – 50 °C / – 58 °F)
Sound attenuation package
Hydraulic arrangement for special application (hammer/shear/grapple/coupler)

Safety

Additional LED lighting with timer (for main access)
Automatic fire suppression system
Additionnal emergency stop (ground level)

General

Maritime transport packaging

Proposition 65

WARNING: This product can expose you to chemicals, including exhaust emissions, includiong lead and lead compounds, which are known to the State of California to cause cancer, birth defects or other reproductive harm.
For more information see: www.P65warnings.ca.gov/diesel

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Appendix E Reserve Report

MT CARBINE PROJECT

2021 ORE RESERVES ESTIMATE

DECEMBER 2021

Document Issue and Approvals

Document Information

Project:	**Mt Carbine 2021 Ore Reserves**
Document Number:	
Title:	**Mt Carbine Project 2021 Ore Reserves Estimate**
Client:	**EQ Resources Limited**
Date:	**8th December 2021**

Contributors

	Name	Position	Signature
Prepared by:	Tony O'Connell	Principal Mining Consultant	A. J. O'Connell
Reviewed by:			
Approved by:	**Tony O'Connell**	**Principal Mining Consultant**	A. J. O'Connell

Distribution

Company	Attention	Hard Copy	Electronic Copy
EQ Resources Limited	**Peter Jukes**	**No**	**Yes**

PURPOSE OF REPORT

Measured Group Pty Ltd have prepared a report on the Ore Reserves of the Mt Carbine Project for the Directors of EQ Resources. The Ore Reserves are estimated as at December 31st 2021.

The purpose of the report is to provide for the company, an objective assessment and estimate of the Ore Reserves contained within the Mt Carbine Project, that have been prepared in accordance with the requirements of the 2012 edition of the Australian Code for Reporting of Mineral Resources and Ore Reserves.

COMPETENT PERSON STATEMENT

This Reserves Estimate for the Mt Carbine Project has been prepared by a team of consultants under the guidance of Mr Tony O'Connell.

The Mt Carbine Project consists of:

- The Mt Carbine low grade stockpile (LGS); and
- The Mt Carbine open pit.

The estimates of Open Cut Ore Reserves for the Mt Carbine Project as at 31st December 2021 presented in this report have been prepared in accordance with the requirements of the 2012 edition of the Australasian Code for Reporting of Mineral Resources and Ore Reserves (**2012 JORC Code**).

Mr O'Connell is a qualified Mining Engineer, (BE (Mining), University of Queensland), has over 22 years of experience in the global mining industry and is a member of the Australasian Institute of Mining and Metallurgy (**AusIMM**). Mr O'Connell has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and the activity being undertaken to qualify as a Competent Person as defined in the JORC Code. Mr O'Connell consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

Neither Mr O'Connell, Measured Group Pty Ltd or Optimal Mining Solutions Pty Ltd has any material interest or entitlement, direct or indirect, in the securities of EQ Resources Limited or any associated companies. Fees for the preparation of this report are on a time and materials basis only.

Mr O'Connell consents to the release of the report, in the form and context in which it appears.

A. J. O'Connell

Tony O'Connell

Bachelor of Engineering (Mining), University of Queensland

Member AusIMM - 230490

EXECUTIVE SUMMARY

This document forms the supporting documentation for the ore reserve estimate, prepared according to *The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, December 2012*, as at 31st December 2021 for the Mt Carbine Project. The Mt Carbine Project consists of:

- The Mt Carbine low grade stockpile (LGS); and
- The Mt Carbine open pit.

Measured Group Pty Ltd (Measured Group) has been engaged by EQ Resources Pty Ltd (EQR) to prepare a Statement of the Ore Reserves for its fully owned Mt Carbine Tungsten Project (Mt. Carbine).

Mt Carbine is an operating tungsten mine and rock quarry located at the northern end of the Atherton Tableland approximately 130 km by sealed highway from the closest major centre of Cairns. EQR acquired the mine and associated quarry in June 2019 and has been operating the mine and quarry concurrently, with the mine currently processing tailings and low grade ore stockpiles located on the site that are remnant from previous operations on the site. The mine is well supported by existing services and infrastructure.

The current plan for Mt Carbine is to recommence the old open pit, which was shut in the late 1980's, whilst continuing to process the LGS. The site is currently permitted to process up to 100,000t per annum of ore, however an amendment to the current environmental authority has been submitted to allow up to 1,000,000t per annum of ore to be processed. The open pit is forecast to be developed at approximately 5mtpa with ore delivery to the plant fluctuating between 250ktpa and 600ktpa. Rehandling of the LGS will top up the total feed into the processing plant to approximately 1mtpa.

The processing plant generates a 50% WO3 concentrate which will be sold on the open market. Mt Carbine currently has off-take agreements for the W03 concentrate which it currently produces. The concentrate will be sold into a market with ongoing strong demand.

A Mineral Resource Statement compliant with the 2012 JORC Code has been prepared by Mr. Chris Grove, a full time employee of Measured Group. The

Resources are split into two sections, one for the LGS and one for the open pit as summarised in the table below.

Mt Carbine Mineral Resources				
September 2021				
Resource	Classification	Tonnes (Mt)	Grade (% WO_3)	WO_3 (mtu)
Low Grade Stockpile Resources				
	Indicated	12.00	0.075	900.000
In-Situ Hard Rock Resources				
	Indicated	2.40	0.74	1,776,000
	Inferred	6.81	0.59	4,017,900
	Sub-total	**9.21**	**0.63**	**5,793,900**
	Total	**21.21**		**6,693,900**

The Ore Reserve was estimated as of 31st December 2021 by a team of mining experts from DAS Mining Solutions, Optimal Mining Solutions and Measured Group. The Competent Person for the Ore Reserve Estimate is Mr Tony O'Connell of Optimal Mining Solutions who contracts to Measured Group.

Open cut Ore Reserves have been estimated by applying modifying factors to the Mineral Resources. The modifying factors include practical pit limits which were based on the current economic limits, determined using indicative operating costs, metallurgical parameters, geotechnical constraints and projected revenue. Other modifying factors included mining losses, mining recovery and dilution factors. An economic evaluation of the mine plan and schedule was completed as part of the estimation process, with the project generating a positive net present value. It should be noted that no revenue was accounted for from the current quarrying services as part of the economic evaluation.

All the Reserves are classified into their respective category based on the level of detail completed in the mine plan and the level of confidence in the Resource estimate. In the categorisation of Reserves, all Indicated Resources have been classified as Probable Reserve. There are no Proven Ore Reserves. No Inferred Resources have been included in the Ore Reserve estimate.

A 0.2% W03 cut-off has been applied in the open pit resource model, however once loss and dilution is applied the minimum open pit ore grade mined is 0.12% WO3. The average ROM grade of the open pit Ore Reserve is 0.713%. The LGS has been classified as a large homogenous orebody which contains an average of 0.075% WO3.

The Ore Reserves for the low grade stockpile and open pit are summarised in the tables below.

LOW GRADE STOCKPILE ORE RESERVE ESTIMATE AS AT 31ST DECEMBER 2021		
Reserve Category	**ROM Tonnes (mt)**	**WO3 %**
LGS - Proved	-	-
LGS - Probable	10.126	0.075%
LGS - Total	**10.126**	**0.075%**

OPEN PIT ORE RESERVE ESTIMATE AS AT 31ST DECEMBER 2021		
Reserve Category	**ROM Tonnes (mt)**	**WO3 %**
Open Pit - Proved	-	-
Open Pit - Probable	1.263	0.713%
Open Pit - Total	**1.263**	**0.713%**

The Resources are reported inclusive of the Ore Reserves. The Ore Reserves have been estimated using the same geological model as the Mineral Resource Statement.

The open pit Ore Reserves are accompanied by 14.0mt of waste which provides an overall ROM strip ratio of 11:1 t/t.

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1. INTRODUCTION

1.1 Process

The process adopted for completing the 2021 Mt Carbine Project Ore Reserve Estimate is described below:

- Geological models have been prepared by Measured Group, with Ore Resources updated and declared in September 2021 for both the LGS and open-pit.
- Economic pit limits and subsequent detailed pit designs for the open pit were completed by DAS Mining Solutions.
- The design stage outputs were 3-dimensional solids in the Vulcan mine planning package. The mine designs included pit wall batters, berm offsets, access ramps and subdivisions into mining benches for truck and shovel waste.
- The insitu ore solids were interrogated against the latest geological model, including the modelled qualities for all ore solids.
- The details for each solid were imported into Comet for processing, analysis and scheduling.
- Minimum mining thicknesses, ore losses and dilution factors were applied to the ore solids to calculate ROM values.
- Concentrate with 50% WO3, for both the LGS and open pit, were calculated for all ore solids based on the metallurgical factors compiled from historical performance at Mt Carbine plus forecast factors from new processing infrastructure.
- Unit cost values were applied to all mining and processing activities to calculate the total cost for each tonne of concentrate.
- Forecast sale prices were applied to the concentrate produced to calculate the overall revenue generated.
- Annual cash flows were calculated based on the calculated operating costs, forecast capital costs and revenue generated. The annual cash flows were discounted at 8% to determine the overall net present value of the project.
- The Ore Reserves have been categorised as Probable based on the Ore Resource confidence, the level of detail in the mine planning and considering all the modifying factors to quantify the risks surrounding the project.
- Checks of all quantities and qualities quoted in this report have been undertaken and all work peer reviewed internally by Optimal Mining Solutions and Measured Group.

1.2 Location

Mt Carbine is an operating tungsten mine and rock quarry located 130 km north-west of the city of Cairns in Far North Queensland, Australia.

The mine is at the northern end of the Atherton Tableland approximately two hours (130 km) by sealed highway from the port and major centre of Cairns and 45 minutes from Port Douglas. There is a small historic hotel and caravan park adjacent to the mine site and a small town. The mine location is shown in Figure 1.1.

Figure 1.1: Mt Carbine Location

.

1.3 History

Wolframite was first discovered on the slopes of Carbine Hill in the 1890s and a number of small-scale mines operated in the vicinity prior to 1920. During this period, the town of Mount Carbine had a population of 400. At the end of World War One, the price of tungsten collapsed, and the township was virtually deserted until the 1970s when Queensland Wolfram Pty Ltd commenced mining by means of an open pit mine. In 1973 Roche Brothers Mining commenced developments of a mine on the project site which became a major world tungsten producer. Mining continued until 1986 when declining prices again forced closure. The mine was placed on a care and maintenance basis until 1991 when the plant and equipment was sold. Since 1987 the mine site has been operated as a quarry by Mt Carbine Quarries Pty Ltd. Increases in tungsten prices prompted Carbine Tungsten Limited (previously known as Icon Resources Limited) to assess mining and reprocessing of tailings at the site.

The contemporary mining operations started in 2012 and included a tailings re-processing pilot plant immediately north of the tailings storage facility. The contemporary mining activities were placed into care and maintenance in May 2016. The quarry repurposes waste rock from stockpiles and tailings on-site and has not significantly altered the site. In January 2020, the project to recover tungsten units from historical low grade material stockpile and tailings materials, that comprised the quarry inventory, commenced.

1.4 Tenure

1.4.1 LAND OWNERSHIP AND MINERAL RIGHTS

The Mt Carbine mining area is located within two Mining Leases, ML4867 and ML4919 totalling approximately 366 hectares. In June 2019, EQ Resources Pty Ltd acquired Mt Carbine Quarries Pty Ltd and has 100% ownership of the two leases and surrounding exploration tenements.

Mt Carbine currently operates as a mine and quarry. The current approvals allow for the extraction, crushing and screening of up to 1Mtpa of material for use as quarry product and the processing through the existing processing plant of up to 100,000tpa of ore.

A map of the EQ Resources mining leases at Mt Carbine are shown in Figure 1.2.

Figure 1.2: Mt Carbine Tenements

1.5 Infrastructure

Mt Carbine is an operational site well serviced by existing on site infrastructure to support the mining and quarrying operations.

1.5.1 Site Offices and Facilities

Mt Carbine is currently fitted with site offices and facilities that are being utilised by operations personnel. The onsite facilities include:

- Office buildings
- Laboratory
- Ablutions facilities
- Car park
- Crib areas
- Phone and internet

1.5.2 Site Access

The access to the mine and quarry is by a well maintained dirt road directly off the Mulligan Highway. Signage, gates and the necessary parking facilities are in place for the acceptance of staff and site visitors. The site access road is shown in Figure 1.3.

The access to the processing plant is from an access road on the southern side of the Mulligan Highway. The processing plant also has the requisite existing infrastructure to manage the access of vehicles and personnel to the site.

Figure 1.3: Mt Carbine Access Road

1.5.3 Site Roads

The site has well maintained light and heavy vehicle roads within the mining and quarrying operational areas. Site has developed a traffic management plan which make use of the existing site roads in a loop configuration minimising light vehicle and heavy vehicle interactions.

The standard site haul road is shown in Figure 1.4.

Figure 1.4: Mt Carbine Site Haul Road

1.5.4 Site Upgrade

The Ore Reserves estimate is underpinned by a mine plan which proposes to increase production through the processing plant up to 500ktpa. Ore will be fed into the processing plants from two sources: the current LGS and the open pit.

This increase in ore processing will require additional processing plant infrastructure as well as facilities to accommodate the recommencement of mining operations in the open pit, which ceased in the late 1980's.

The strategy for the site infrastructure upgrade is to utilise as much as possible of the existing site infrastructure and only construct new infrastructure as required to support the increase in mining quantities. Overall changes to the footprint of the mining and associated crushing, screening, sorting and processing operations is forecast to be minimal, with only minor site infrastructure modifications required to support the upgraded facilities.

The plan for the sitewide infrastructure upgrade is shown below in Figure 1.5.

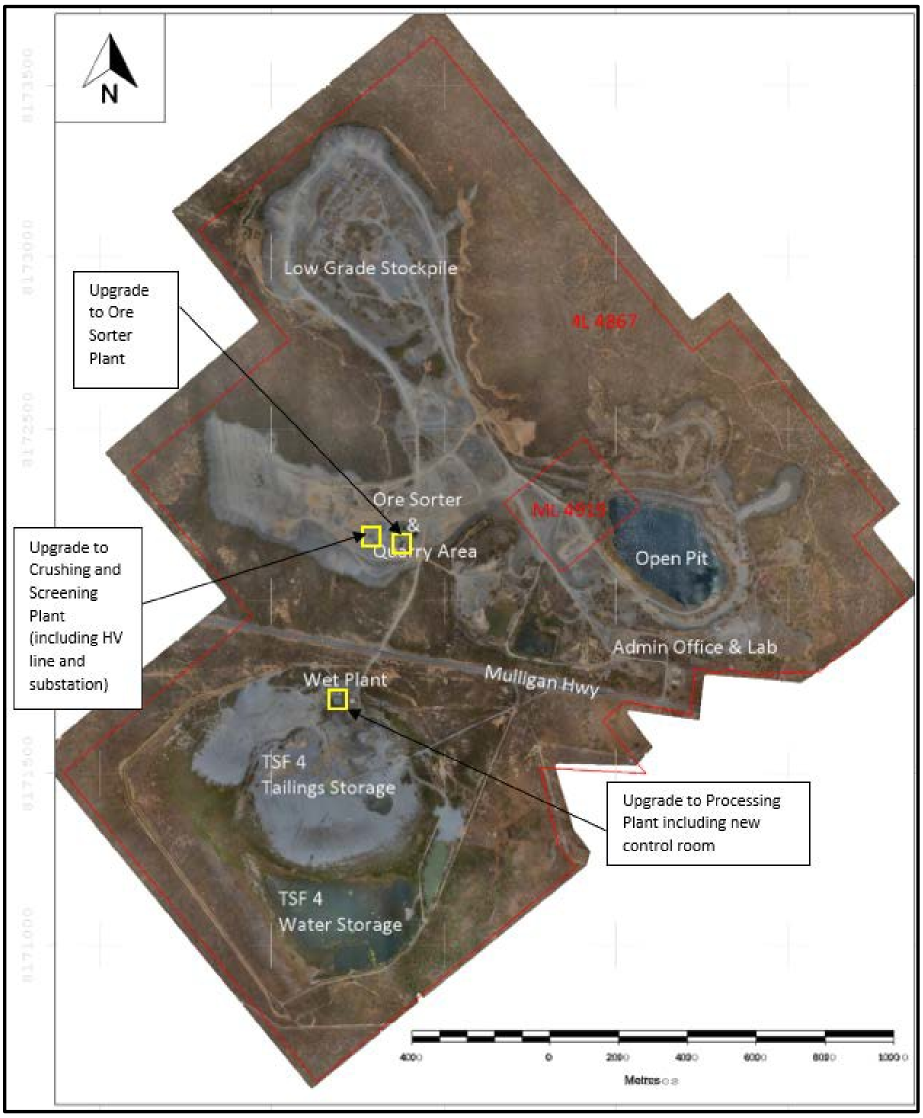

Figure 1.5: Mt Carbine Upgrade Plan

The details of the existing site infrastructure supporting the operations are detailed below in section 3.5.3.

1.6 Approvals

1.6.1 Current Environmental Status

The surrounding land use is rural-urban dominated by the nearby Mount Carbine township, low-intensity cattle grazing, mining and exploration, and conservation (the Brooklyn Nature Refuge1). The background land tenure (Lot 13 on SP254833) is Brooklyn Nature Refuge, which is held by the Australian Wildlife Conservancy as a rolling term lease – pastoral (Title Reference 17664140); a special condition of this lease is to allow quarry material to be removed.

Environmental values (EVs) associated with the project include air, acoustic, water, wetlands, groundwater, and land.

Air

The air quality EV for the Mount Carbine area is described as having an airshed that is typical of a rural area impacted by agricultural activities, mining and exploration activities, and transport infrastructure activities on sealed and unsealed roads.

The closest nearby sensitive receptors are the Mt Carbine Hotel and Mt Carbine Roadhouse, approximately 700m east of the project site. Other nearby sensitive receptors located farther eastward are five residences off Mulligan Highway and the Mount Carbine township area.

Acoustic

The noise levels in the Mount Carbine area are typical for a small rural area, with traffic being the main source of noise during day and night. Light vehicles, cattle trucks, and semi-trailers use the Mulligan Highway, which passes through the project area, on a daily basis. The quarry and mining operations have produced noise from on-site crushing, truck and loader movements, light vehicle traffic, and mine processing plant for many years. Sensitive locations for noise and vibration are the Mt Carbine Hotel and Mt Carbine Roadhouse, approximately 700m east of the project site, five residences off the Mulligan Highway and the Mount Carbine township area.

Water (surface water, groundwater and wetlands)

The project area is within the Manganese Creek and Holmes Creek catchments, which drain to the Mitchell River approximately 5 km downstream. Manganese Creek and Holmes Creek are intermittent watercourses that are dry for most of the year.

EVs are not nominated under the Environmental Protection (Water and Wetland Biodiversity) Policy 2019 for the part of the Mitchell River basin relevant to the project receiving environment.

There are no wetlands of national or international significance mapped in the project site or the receiving environment (DES 2021a). There are no High Ecological Value Waters (watercourses), High Ecological Value Waters (wetlands) or Wetlands of High Ecological Significance mapped in the project site or the receiving environment (DES 2021b).

1.6.2 Current Approvals

The land relevant to the project site is used for quarry operations and mining activities as per the respective licenses (EA EPPR00438313, dated 16 March 2021 for the quarry and EA EPML00956913, dated 1 December 2020 for the mine).

The Mineral and Energy Resources (Financial Provisioning) Act 2018 came into force on 1 April 2019. This act includes provisions that replaced the financial assurance arrangements for resource activities under the Environmental Protection Act 1994 (EP Act) with the requirement to provide either surety or a percentage contribution to the Financial Provisioning Scheme (based on the

Estimated Rehabilitation Cost (ERC) for the project). DES has decided the ERC amount for EA EPML00956913 to be approximately $1.05m for the period 25 June 2021 to 24 June 2022.

Notifiable activities are defined in Schedule 3 of the EP Act. No notifiable activities are planned to occur as part of the quarry activities under EA EPPR00438313. Lot 13 on Plan SP254833 is included on the Environmental Management Register (EMR) as the site has been subject to the following notifications associated with the mining activity undertaken pursuant to EA EPML00956913: Mine Waste, Mineral Processing, Petroleum Product or Oil Storage.

Environmentally relevant activities (ERAs) are defined in Environmental Protection Regulation 2019 (EP Reg). The ERAs are licensed under EA EPPR00438313 for the quarry activity and under EA EPML00956913 for the mine activity.

With regards to the requirement for an End of Waste (EOW) code for the Mt Carbine development, the regulator (i.e. DES) has determined that that an EOW approval or code is not required.

Reforms to the regulatory system regarding rehabilitation that came into effect on 1 November 2019 require that mining projects have a Progressive Rehabilitation and Closure Plan (PRCP) and PRCP schedule. All projects in Queensland will need to comply with the new regulatory system before 1 November 2022; at this time the EA holder has not been invited by DES to initiate the PRCP process.

1.6.3 Mining Leases

The Mt Carbine project is located on two leases: ML4867 and ML4919. The details of both mining leases are shown in Table 1.1.

Table 1.1 – Mt Carbine Leases

Item	**ML 4867**	**ML 4919**
Permit number	ML 4867	ML 4919
Permit type	Mining Lease	Mining Lease
Permit status	Granted	Granted
Permit sub-status	None	None
Lodge date	23 December 1971	30 November 1972
Approve date	25 July 1974	22 August 1974
Expiry date	31 July 2022	31 August 2023
Authorised holder name	MT. CARBINE QUARRIES PTY. LTD.	MT. CARBINE QUARRIES PTY. LTD.
Mineral	CU, FE, MO, SN, W, MT, Q, SI	CU, PB, SN, W, ZN
Permit sub-type	Mineral	Mineral
Native Title category	Granted before 1 January 1994	Granted before 1 January 1994
Area (ha)	358.5	7.891
Permit name	MT CARBINE NO 1	NEW DCL
Permit number other	4867	4919
Permit type abbreviation	ML	ML
Previous permit number	ML2523MARE	ML2888MARE
Permit ID	108011	108023

Both mining leases expire within the planned duration of the mine plan. ML4867 expires on 31st July 2022 whilst ML4919 expires on 31st August 2023. EQR has commenced the process for acquiring a new mining lease over the Mt. Carbine area, which will be underpinned by a bankable feasibility study.

The competent person believes that it can be expected that the company will gain the required mining leases and environmental authorities to allow for extraction and processing of the open pit and low grade stockpile as planned. The mine is critical to the livelihood of the township and has a positive effect on the surrounding areas. Tungsten is seen as a key future element and part of the state government's plan to develop the northern regions of the state.

1.7 Topography and Land Use

The topography around the Mt Carbine project is dominated by the surrounding high hilly and mountainous areas with very steep slopes. The mine and processing plant areas are mostly situated on a relatively flat alluvial plain.

The land has been disturbed by historical mining activities which have been active since the late 1800's. Land use within the project area is licensed for mining and quarrying. The immediate surrounding land use is conservation (Brooklyn Nature Refuge), with low intensity beef cattle grazing, mining and exploration, and the township of Mount Carbine.

The Mt Carbine agricultural land evaluation classification is Class 4 on the lower/flatter areas and Class 5 on the steeper/higher areas.

2. GEOLOGY

2.1 Regional Geology

Mt Carbine is located within the Siluro-Devonian Hodgkinson sedimentary province. The thick sedimentary sequence was subjected to complex folding and regional metamorphism before, and during, extensive granitic intrusions in the Carboniferous and Permian.

Within the permit, the north-northwest trending Hodgkinson Formation turbidite and siltstone sequence is intruded by the Mareeba Granite dated at 277 My, and the Mt Alto Granite dated at 271±5 My. Contact metamorphic aureoles marked by the formation of cordierite Hornfels that surround the granite intrusive, and numerous acid intermediate dykes intrude the metasediments. In the western portion of the tenement, a prominent metabasaltic- chert ridge is a significant stratigraphic component of the Hodgkinson Formation.

Fluids from the large granite batholith (>400 km2) were the source of hydrothermal fluids for mineral deposition around the margins of the intrusive. The Mt Carbine deposit is a direct result of these fluids travelling out from the granite into surrounding structured ground.

There appears to be a preference for the higher grade tungsten mineralisation to be located on failed fold hinges, associated with the isoclinal folding of the Hodkinson Formation. These locations have the highest structural deformation and have allowed these fluids to penetrate into structures and deposit quartz and minerals. The granites associated with Mt Carbine are 'S' Type Granites, which can mobilise tin, tungsten, molybdenum and rare earth elements in fluids and deposit these as the main economic minerals.

2.2 Local Geology

The Mt Carbine tungsten deposit is similar to sheeted vein-type tungsten deposits in South China and these are divided into endo-contact (granite hosted) and exo-contact (wall-rock hosted) types. Mt Carbine is interpreted to be an exo-contact type.

The vertical structural zoning model for vein-type exo-contact tungsten deposits observed in China directly applies to the Mt Carbine vein system. The model is being incorporated in an evolving exploration model for the Mt Carbine and Mt Holmes vein systems, with Mt Holmes considered to be situated closer to the underlying mineralising granite than Mt Carbine (Figure 2.1).

The simplified conceptual geological model of the Mt Carbine area is based on that of Mt Holmes:

- Deposition of the Siluro-Devonian Hodgkinson Formation sequence.
- Several stages of complex folding and faulting of the Hodgkinson Formation.
- The intrusion of minor andesite and dolerite dykes.
- The intrusion of mineralising granite plutons with associated hornfels in the country rock.
- Emplacement of major sheeted quartz-wolframite-tin veining and hydrothermal alteration of wall-rock.
- The intrusion of post mineralisation dykes.

In the open-cut pit, the following rock types are observed in the order of abundance:

- **Metasediments** – a range of hornfelsed mudstones and interbedded rudites. The major rock unit in the pit can look similar to a slate with prominent cleavage. Various alterations from pervasive silicification are present, represented as a hornfelsed cordierite chloritic rock. Typically breakage planes are along cleavage and schistosity planes.
- **Metavolcanics** – located on the eastern end of the pit and the south side of the Southwest Fault (SWF) this unit is pale green with greenschist facies alteration. It forms about 20% of waste material and is less likely to contain

mineralisation as it is a peripheral unit. It contains locally hard siliceous chert bands that form some of the larger rocks on the waste dump

- **Quartz Veins** – this rock type makes up to 10% by volume of the waste material and is found in all sizes but typically less than 20 cm. It presents as powder and shards throughout much of the dump material, which is interpreted to be a product of shattering during blasting. As previously discussed, quartz veins can be barren or can contain tungsten mineralisation.
- **Dyke Material** – two types of dykes are observed:
  1. Pale uniform fine-grained felsic dyke that is exposed as a 10-15m wide dyke at the western end of the pit; and
  2. Dark green/grey basic dyke that is present on most benches as a 0.5-1m dyke cross- cutting the open pit.

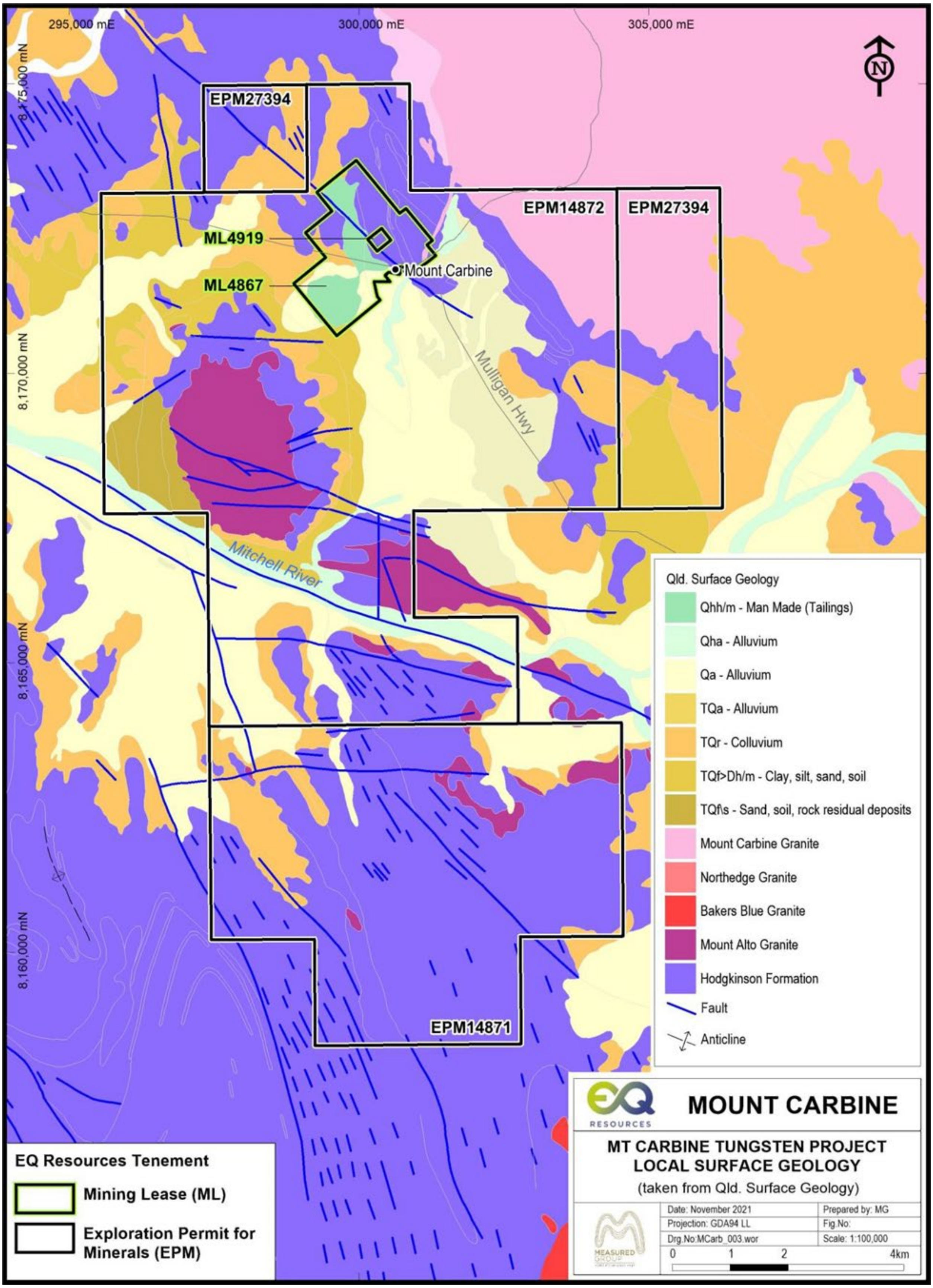

Figure 2.1: Mt Carbine Local Geology

2.3 Structure

Mt Carbine sits at a spur on a major arc parallel fault called the South Wall Fault (SWF), along with the Mossman Orogeny trend, which can be traced through the Hodkinson formation for over 100 km in strike length. The inflection point is likely due to a change in compressional regime due to oblique pressures present at an intersection of a major fault junction, the South West Fault (SWF). The SWF is a thrust fault formed at the time of compression and development of regional isoclinal folding of the basement rock, remaining active through to post tungsten mineralisation movement.

This terminology on the local scale with this thrust also called the 'South Wall Fault' (SWF) at Mt Carbine was kept. The SWF truncates the tungsten orebody at an angle of 70 degrees to the grid north. It forms a boundary fault on the southwest side of the mineralisation. Evidence suggests it is a reverse thrust fault, and by studying stratigraphic marker beds (chert- metabasalt unit) it is postulated that the throw is of the order of 200-300 m. The truncated parts of the Mt Carbine Tungsten mineralisation should still be open at depth in the footwall region of this fault. Figure 2.3 shows the location of the SWF in the open pit.

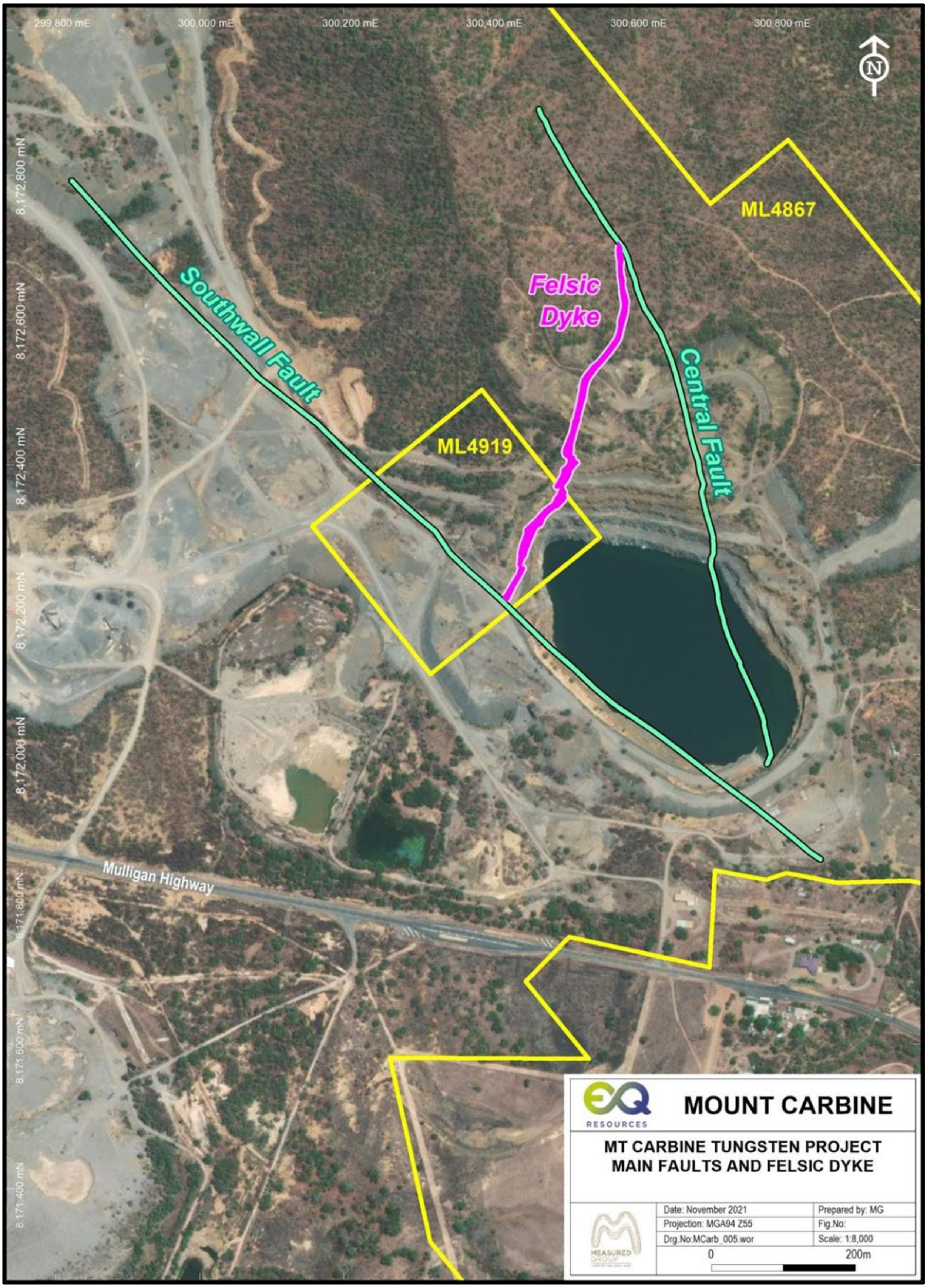

Figure 2.2: Mt Carbine Open Pit Structure

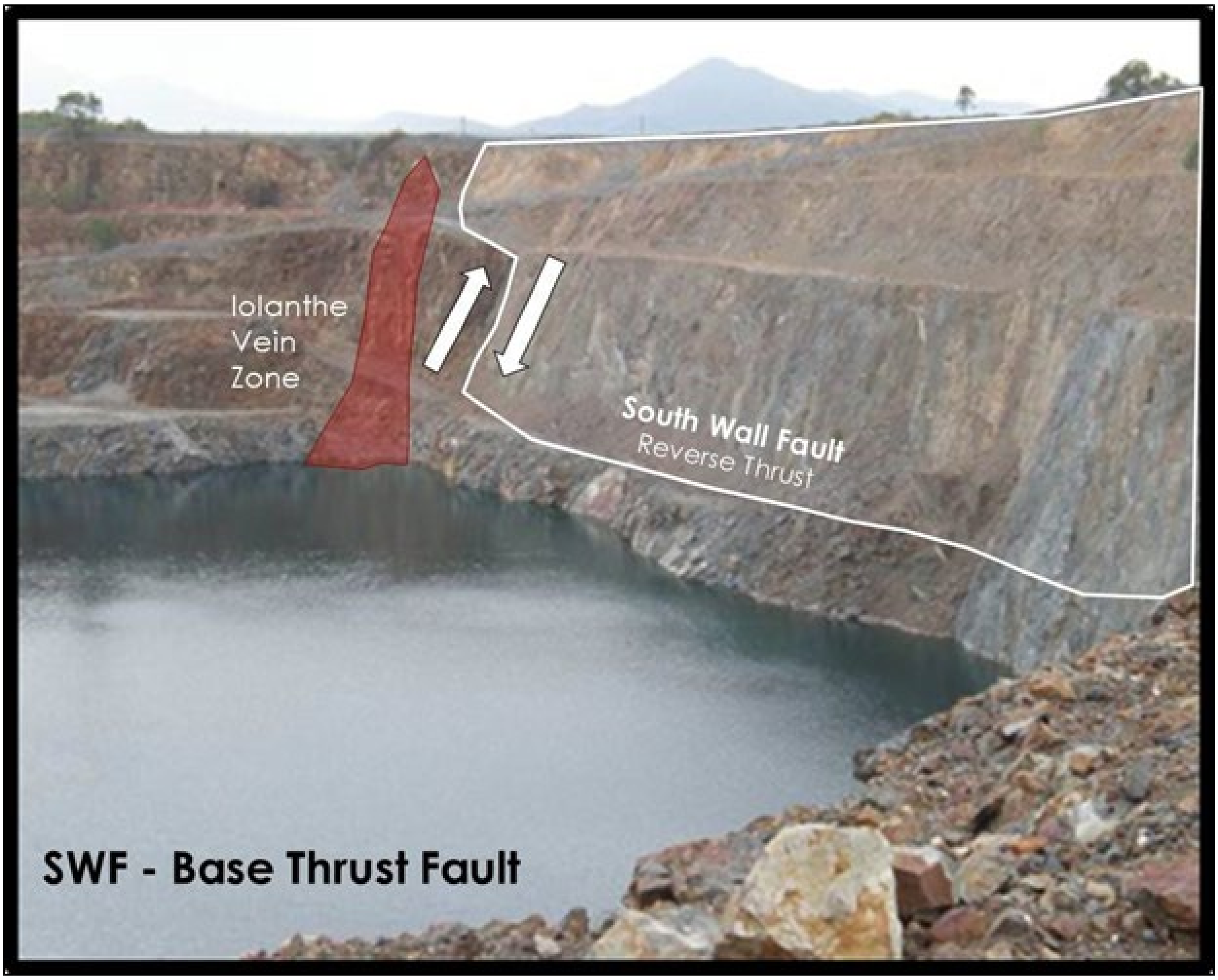

Figure 2.3: South Wall Thrust Fault

Other minor faults are typically orientated on a north-south strike direction and exhibit localised movement. The Central / Iron Duke and Christmas Faults both show strike-slip movement and in the case of the central fault, there is strike-slip movement across a dyke of 120m in a left lateral direction. Whereas minimal throw is noted on the Christmas fault.

Within the confines of the pit, the rocks are hornfelsed but several deformation lineations can still be seen i.e. S0 bedding, S1 minor folding and S2 isoclinal folding planes. The mineralised veins postdate this basement deformation, and there is little or no movement on the pit scale.

Veins can be traced over vertical distances of 300-400m and strike distances for over 1,200m with very few offsets. Occasionally in the pit, a regular low angle fault occurs that locally shifts the veins up to 3-4 metres. This low angle fracture regime has a tendency to form blocks which will require geotechnical considerations for underground mining.

2.4 Low Grade Stockpile

During mining operations undertaken by Queensland Wolfram Limited, 22 Mt was mined from the pit, of which 12 Mt of low grade material was sent directly to the Low Grade Stockpile (LGS). 10 Mt was optically sorted to extract white quartz from the ore, which resulted in 6 Mt of reject material (now since disposed) and 4 Mt of higher-grade ore that was processed.

A nearly complete record of mine production, including the amounts of mined rock consigned to the LGS has been compiled by EQR using published and unpublished archives, including using reports for State Royalty returns. Head grades were not recorded, rather they were calculated from the recovered grade using a nominal 70% recovery. The calculated head grade for the mine using this method was 0.14% WO3. Several authors have subsequently postulated a higher feed grade based on a lower recovery at the processing plant with the head grade being as high as 0.16% WO3.

During mining, grade control in the pit was difficult since the mining process focused on quartz vein content, with the percentage of quartz used to decide whether material was ore or waste. Since the completion of mining, geological interpretations have suggested that an early major barren quartz vein intrusion event occurred. This resulted in the processing of increased amounts of barren quartz, and the wasting of mineralised material to the LGS. The lack of an effective grade control system was instrumental in allowing higher-grade material to be dumped to the LGS.

The LGS consists of material ranging from fines to large boulders. It is largely heterogeneous and consists of layers of similarly sized material, which reflects the position of the mine at the time of emplacement. Cross sections through the LGS confirm the cyclic nature of the emplacement of material, with layers of similar sized material observed. Significant work has been completed to understand the size distribution of the LGS

2.5 Recent Exploration Programs

EQR completed 16 angled diamond drill holes (EQ001 – EQ016) in 2021, for a total of 4,074 m. The drill holes targeted high-grade ore shoots below the current pit, to improve confidence in the lithology, structural interpretations, and mineralisation limits and to improve the resolution of geological models.

Figure 2.4 shows the drill hole location map relative to the historical pit. The red coloured locations are showing the 2021 drilling program whilst the green locations are historical drill holes.

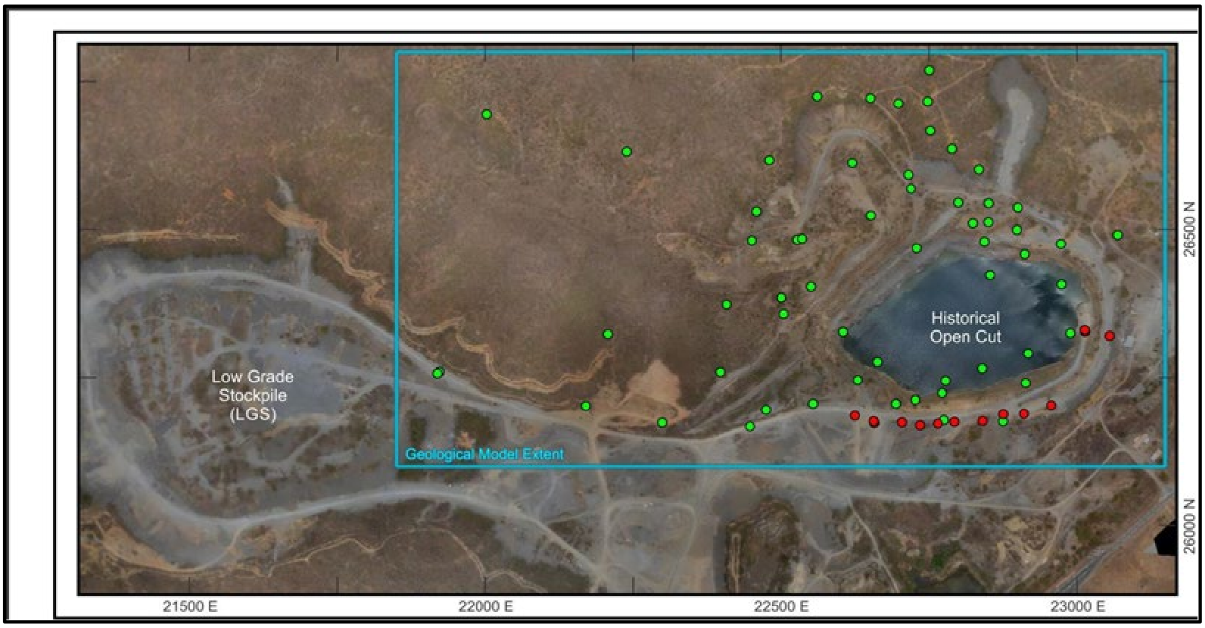

Figure 2.4: Mt Carbine Exploration Holes

2.6 Data Supporting Ore Resource and Reserve Estimates

2.6.1 Open Pit

All zones of potential mineralisation were logged and sampled by cutting the selected core interval in half using a diamond saw along the centre core orientation line mark. Before cutting and sampling, the core was logged for zones of visual mineralisation, with wolframite and scheelite recorded by their visual contained percentage.

Scheelite glows under ultraviolet light, and although difficult to distinguish under ordinary light from quartz-carbonate, it is visual under the shortwave 254 nm UV light. A common technique to estimate grade in core is to trace out individual crystals to determine the overall percentage shown on the face of the core. The mineralisation was often observed as very coarse tungsten mineral crystals of up to 10 cm in size.

All quartz veins intersected in drill core were assayed as separate samples. Where the veins were more than 1m in downhole length, the sample was broken into two or more samples - each with a maximum of 1m interval. The minimum vein assayed was 5 cm in width, because the mineralisation often occurs in narrow widths of 5 cm to 500 cm and it is important to assay each narrow mineralised zone. On either side of the mineralised zone, samples were taken of the host rock at intervals of 1m to ascertain whether the mineralisation had disseminated into the host rock.

X-ray fluorescence assay techniques were used to determine the tungsten grade (ME-XRF15b). Using this technique, a fusion disk is created for the representative sample of the core sample, it is created by grinding the sample to achieve a homogenous sample (<200 microns). The sample is then melted in an arc furnace to produce a clear fused disc, which is then x-rayed with the fluorescence recording spectral peaks.

The instrument used to determine the tungsten grade is a Bruker multi-shot XRF machine with an X-ray scan of 1 minute applied to each disk to ascertain the light and heavy elements. The XRF machine is calibrated by the laboratory to maintain reliable and repeatable results.

Approximately 10% of each batch that is sent to the laboratory includes check samples, which are submitted alternatively as being either a blank, a tungsten standard or a repeat sample with a known grade. This process was successfully used to resolve an issue with samples 100216 and 100217, which are samples vein and host rock (respectively). The results for these samples did not match the visual grade determination or the weights of the samples and it was established that the grade of 0.72% was in the vein, not the host rock. It was concluded that samples were mistakenly switched at the laboratory, and this was rectified prior to loading into the assay database.

ALS was used for assaying samples. ALS is a NATA accredited laboratory that conducts internal and external round-robin analysis to maintain its certification and to ensure their equipment is correctly calibrated and reliable. Final samples were bagged and prepared for transport to Brisbane via road or rail. Reserves from the assayed samples have been archived for future re-sampling. Chain of custody between EQR and ALS requires both parties to record and check sample and/or batch numbers on dispatch/receipt of sample shipments and check for any signs of tampering or damage.

2.6.2 Low Grade Stockpile

To determine the grade distribution of the LGS, a comprehensive sampling programme was developed to achieve representative sampling of the stockpile material. The sampling that was undertaken to achieve this is summarised below, while Figure 2.5 shows the location of the samples:

- **Sites Selection** – The dump was divided into quadrants with a major and minor sample location being marked. In two of the quadrants, two sample sites were selected to see repeatability.
- **Sample Size** - 6 trench samples (each trench taken at approximately 10m wide x 5m deep x 40m length) was deemed to be representative of that part of the dump each comprising a 3,500t sample.
- **Method** – The sample was collected using 25t trucks and a 30t excavator being careful to load all the material from the sample trench and the run over the weighbridge to determine weight before being added to a large stockpile. A total of 22,000t was collected from the 6 separate locations. This was then cone and quartered down to a subset sample of 2,000t which was fully crushed to a nominal 40 mm and sampled.

The bulk sample average was determined to be 0.075% WO3.

Further sampling of the LGS for environmental permitting purposes involved taking 80 grab samples from the surface of the stockpile. Each sample was approximately 20 kg of minus 100 mm material. The average grade of these samples was 0.088% WO3.

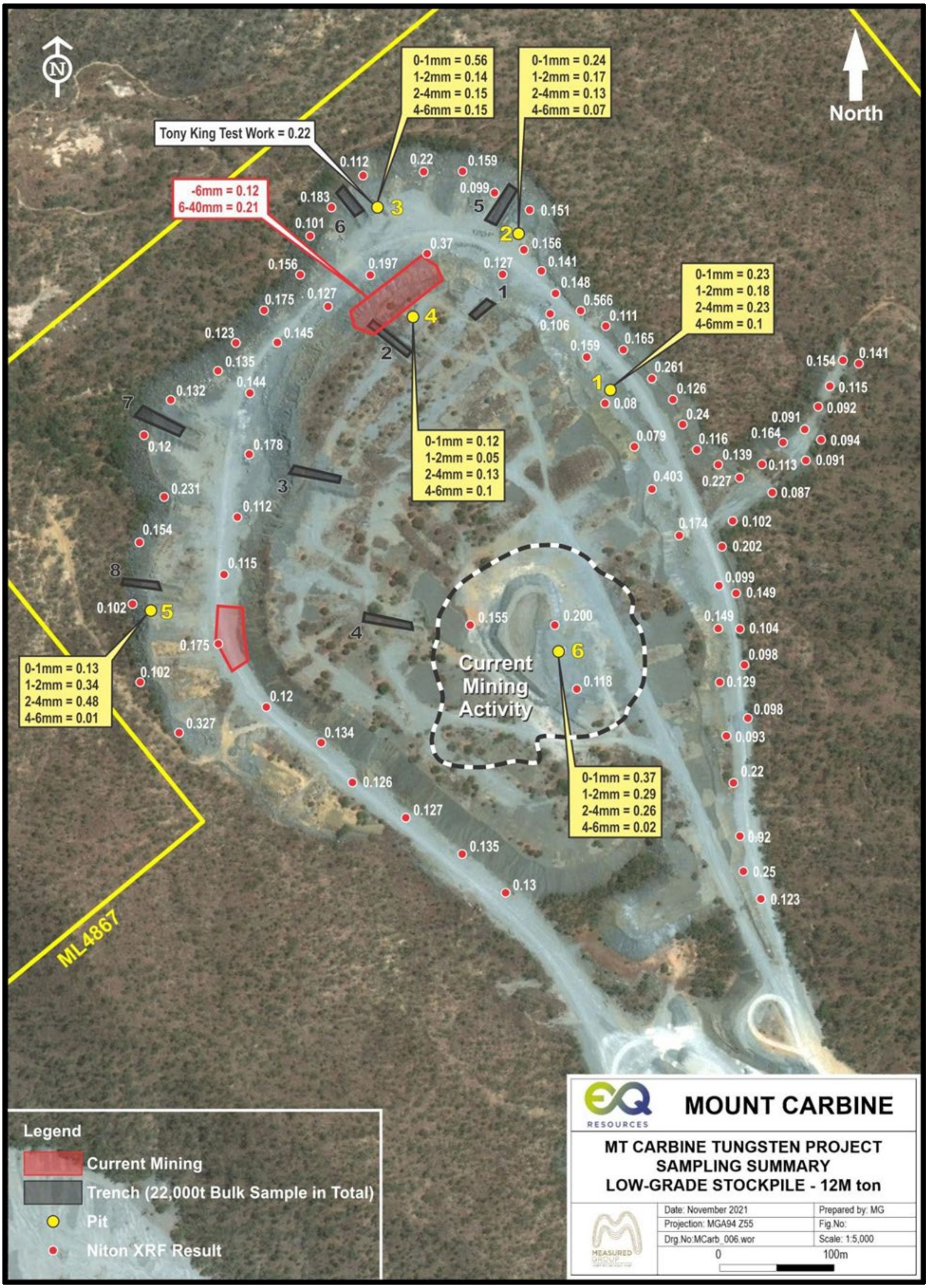

Figure 2.5: Mt Carbine LGS Sampling Locations

3. Mine Planning

3.1 Mine Setting

The key features of Mt Carbine which directly influences the mine design, equipment allocation and the schedule are:

- Topography is relatively flat within the proposed open pit area with minor undulation across the deposit. The topography rises to the north-west with a 120m high hill located in the central region of the tenement. The crest of the hill is located approximately 500m to the north of the current ore sorter.
- A total of 160 individual tungsten veins have been modelled across the deposit. The planned open pit mines 71 of the modelled veins.
- The veins are near vertical and vary in width from approximately 30 centimetres to over 10 metres.
- The mine is developed in two phases, with the first finishing off the previously started phase from the 1980's and the second phase will develops the open pit approximately 80m deeper than phase one.
- Manganese Creek and Holmes Creek drain the site and flow to Mitchell River approximately 5 km away. Both these creeks are intermittent watercourses that are dry for most of the year.
- No major powerlines cross the planned open pit mining area.
- There are no endangered ecosystems within the planned mining area.

An isometric view of the current open-pit, LGS, ore sorter and offices is shown in Figure 3.1 which an aerial photograph of the current open pit provided in Figure 3.2.

Figure 3.1: Mt Carbine Mine Setting

Figure 3.2: Current Mt Carbine Mine Open Pit

3.2 Pit Optimisation

3.2.1 Economic Pit Limits

The economic pit limits were calculated using the Vulcan Pit Optimisation module. The module requires several key sets of input data, including:

- Resource model,
- Loss, dilution and recovery factors,
- Metallurgical factors,
- Unit cost rates,
- Revenue assumptions,
- Geotechnical parameters.

3.2.2 Pit Design Parameters

The design of the majority of the open pit is based on the current wall design parameters, which have performed very well since mining ceased in the 1980's, as well as design requirements for drill and blast plus ramp access.

The dig design parameters for the majority of the open pit are summarised in Table 3.1.

Table 3.1 – Pit Design Parameters

Item	Units	Value
Final Wall Batter Angle	degs	70
Final Wall Bench Height	m	20
Final Wall Bench Width	m	8.5
Access Ramp Width	m	15
Access Ramp Maximum Grade	%	10

The mine plan assumes that articulated dump trucks will be used for all waste and ore haulage. These trucks have a width over mirrors of 3.6m, which requires a minimum ramp width of 12.6m when the standard 3.5 x truck width factor is applied.

3.2.3 South Fault Wall

The primary area of geotechnical risk in the open pit exists along the southern wall near the south wall fault. The geotechnical advise for this wall is to construct a slightly lower angled wall and rock bolt each bench as per the cross sectional profile shown in Figure 3.3.

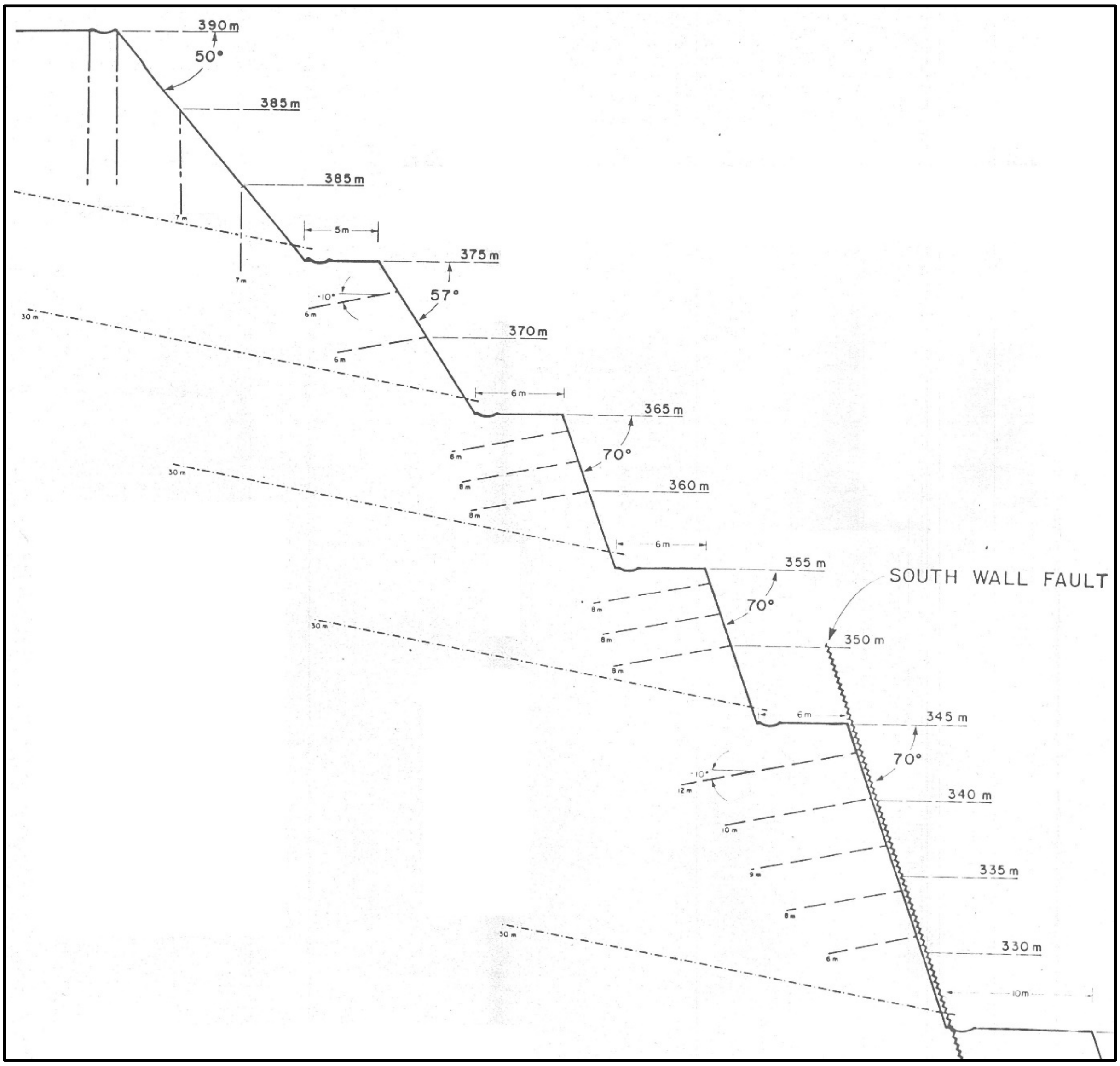

Figure 3.3 –South Wall Geotechnical Design

The geotechnical analysis of this wall indicates that the current pit design may require rock bolting at close intervals to minimise the probability of this wall causing geotechnical disruptions.

Capital and operating cost allowances have been made in the financial assessment to monitor and treat this wall as it is exposed in the final wall. The two upper benches in the south wall, which are located in weathered material, have been excavated at 50 degrees and 57 degrees as specified by the geotechnical assessment.

3.2.4 Cost Assumptions

Unit costs have been supplied by EQ Resources for current operational practices and Ausenco for processing costs of the upgraded facilities. Drill and blast plus open pit mining costs were estimated based on the required fleet. The average unit costs used are shown in Table 3.2.

Table 3.2 – Average Unit Costs for Economic Pit Limit Calculation

Cost Item	Units	Unit Cost
Drill & Blast	$/t	$1.00
Waste Mining	$/t	$3.50
Ore Mining	$/t	$3.50
Ore Processing	$/ore t	$14.50
Rehabilitation	$/total t	$0.20
State Royalty	% of revenue	2.70%

3.2.5 Revenue Assumptions

Revenue assumptions are based on the forecast concentrate prices for WO3 APT price of US$31,500 per tonne with a AUD:USD exchange rate of 0.73 applied. Historical realized price adjustment factors were then applied as well as discounts for producing a concentrate with 50% WO3. The assumed metallurgical recovery is 75%. The final assumed realised price for each tonne of 50% WO3 concentrate is AU$15,100 for the pit optimisation work.

Despite currently generating income from quarry material, no revenue has been generated from this procedure as part of the economic pit shell calculation evaluation of the Reserves.

The old open pit currently contains water which is being pumped out. The base of the pit beneath the water has been compiled from old survey maps plus depth sounding information taken from the water level. The starting topography for the open pit, with the current water area shaded dark blue, is shown in Figure 3.4.

Figure 3.4 –Open Pit Starting Topography

The total economic pit shell, which the Ore Reserves Estimate is based on, is shown in Figure 3.5.

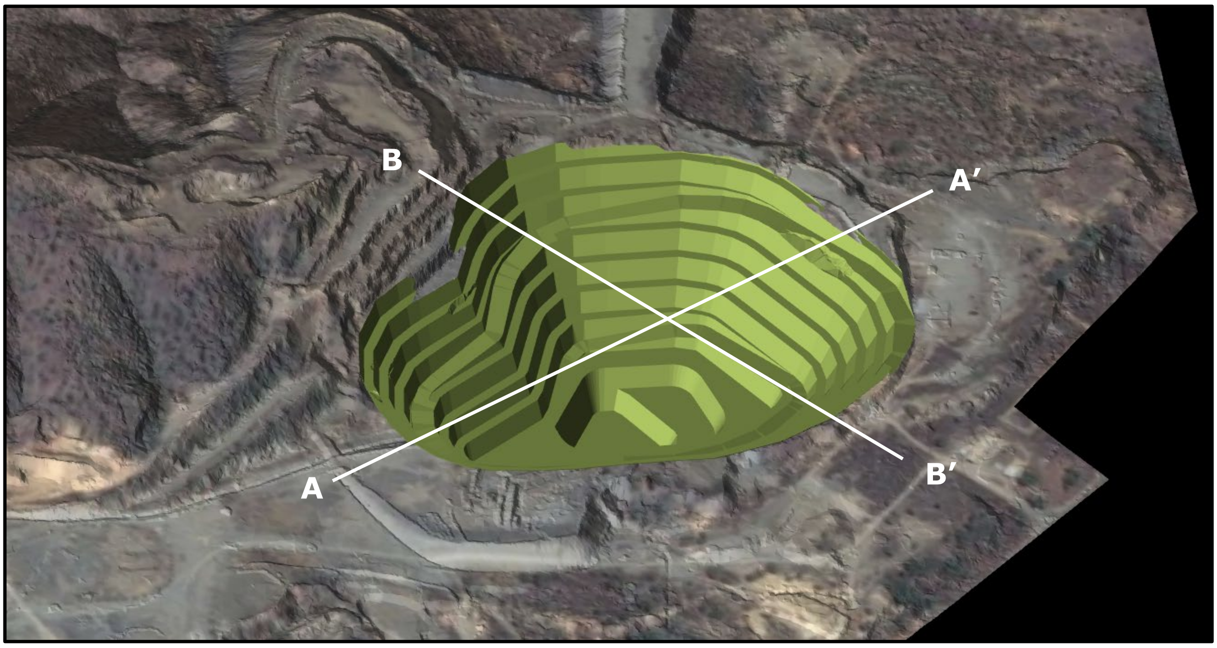

Figure 3.5 – Economic Pit Shell for Mt Carbine

The economic pit shell shown in Figure 3.5 was broken into 3.5m high benches and interrogated against the resource model before being scheduled.

Cross sections A-A′ and B-B′ as shown in Figure 3.6 and Figure 3.7 are shown below with 20m benches (white lines) and modelled tungsten veins (red) indicated.

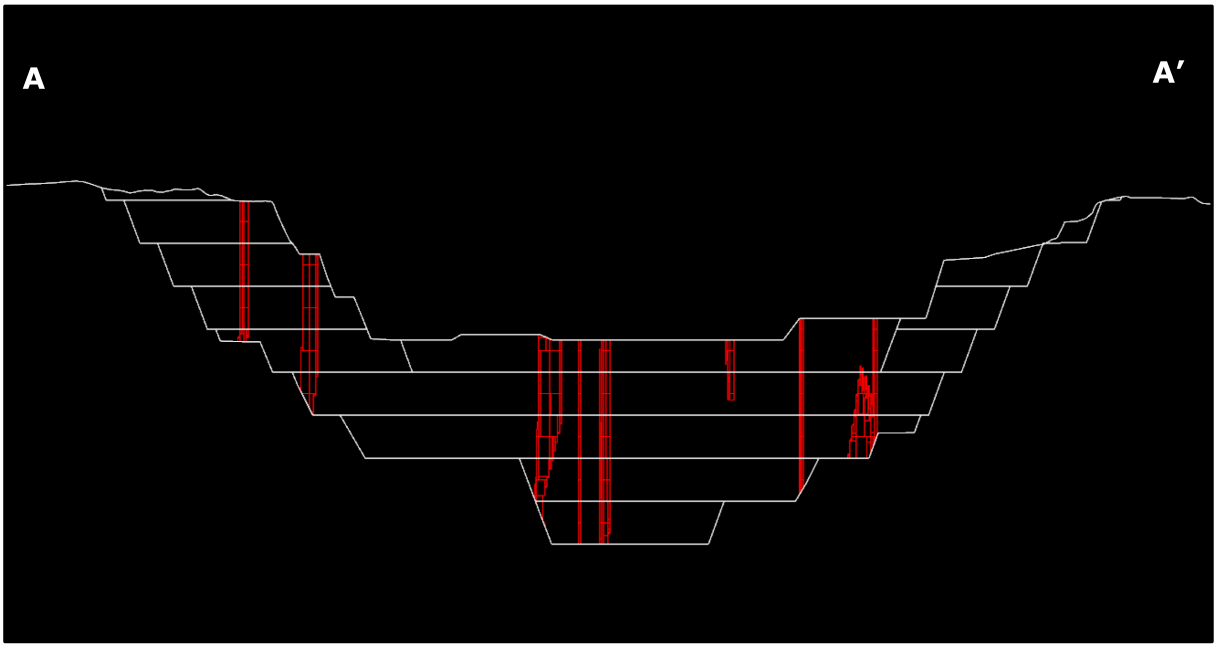

Figure 3.6 – Cross Section A-A′

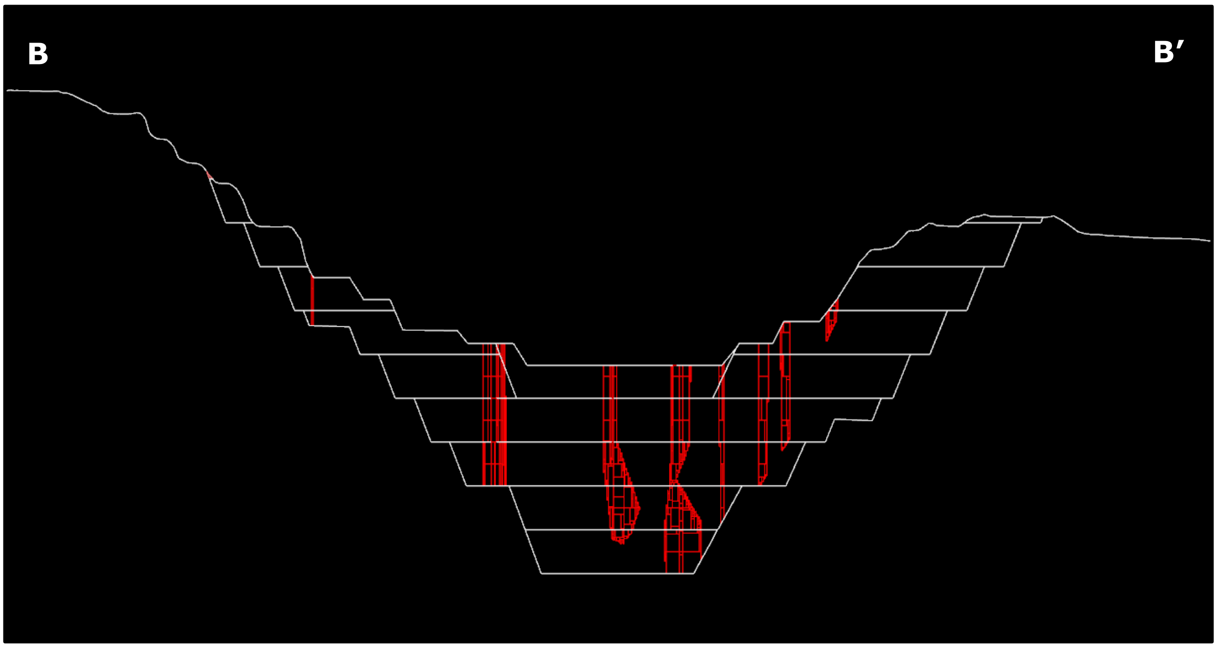

Figure 3.7 – Cross Section B-B′

The starting topography for the LGS was calculated from forecast production information up to December 31st 2021. It is estimated that 10.126mt of ore will be remaining in the LGS at this date.

The location of the LGS ore blocks is shown in Figure 3.8.

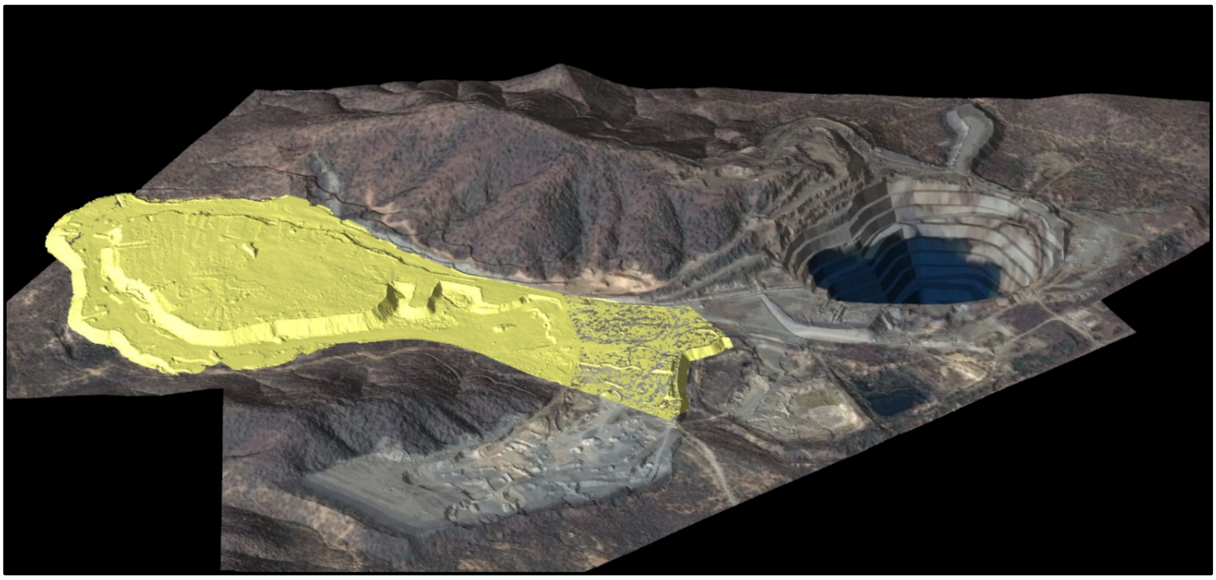

Figure 3.8 – Low Grade Stockpile Ore Blocks

3.3 Mining Method

The low grade stockpile is currently processed by a 50t excavator and accompanying 45t articulated dump trucks (ADTs). The Ore Reserves estimate assumes that this process will continue throughout the proposed life of the schedule.

The open pit will be developed using two primary fleets:

- 1 x 120t class excavator loading 45t ADTs,
- 1 x 50t class excavator loading 45t ADTs.

The mining fleets will be supported by ancillary equipment including 2 x dozers, a grader and a water cart. Topsoil stripping is not required as the planned pit is within the footprint of the previous open pit disturbance area.

For scheduling and costing purposes, it has been assumed that all material will be drilled and blasted, however some upper waste material may be free dug. Due to the vertical nature of the modelled tungsten veins, drill and blast of the waste material will be undertaken to the edge of the vein and then removed. The ore will then be drilled and blasted in a small shot to reduce losses and minimise dilution.

3.4 Mining Layout

3.4.1 Pit Limits

An isometric view of the open pit with modelled tungsten veins (red solids) is displayed in Figure 3.9.

Figure 3.9 – Open Pit and Tungsten Veins

3.5 Mining Assumptions and Modifying Factors

Modifying factors were applied to the insitu quantities and qualities to convert from an insitu basis to a ROM basis. Metallurgical modifying factors were then applied to calculate product information.

3.5.1 Loss and Dilution

Mining dilution has already been included in the resource model, with an average of 85% dilution by volume added to the insitu tungsten veins. This dilution has been applied veins whose assayed length is less than 2m, reflecting that the veins will need to be mined at a minimum width of 2m to recover the majority of the Resource with the planned mining equipment. The grade of the modelled veins has accordingly be downgraded to reflect the addition of this dilution.

An additional 16% dilution has been added on top of the mining dilution built into the resource model to account for additional dilution around the edges of the veins which are thicker than 2m.

Between the two dilution assumptions, approximately 101% dilution has been applied to the insitu tungsten veins.

Global losses equal to 1% of the modelled tungsten veins has been applied. This low loss factor complements the large dilution value which aims to recover as much of the insitu tungsten as possible.

3.5.2 Metallurgical Factors

The mined ore passes through a network of processing facilities, with the process summarised in Figure 3.10.

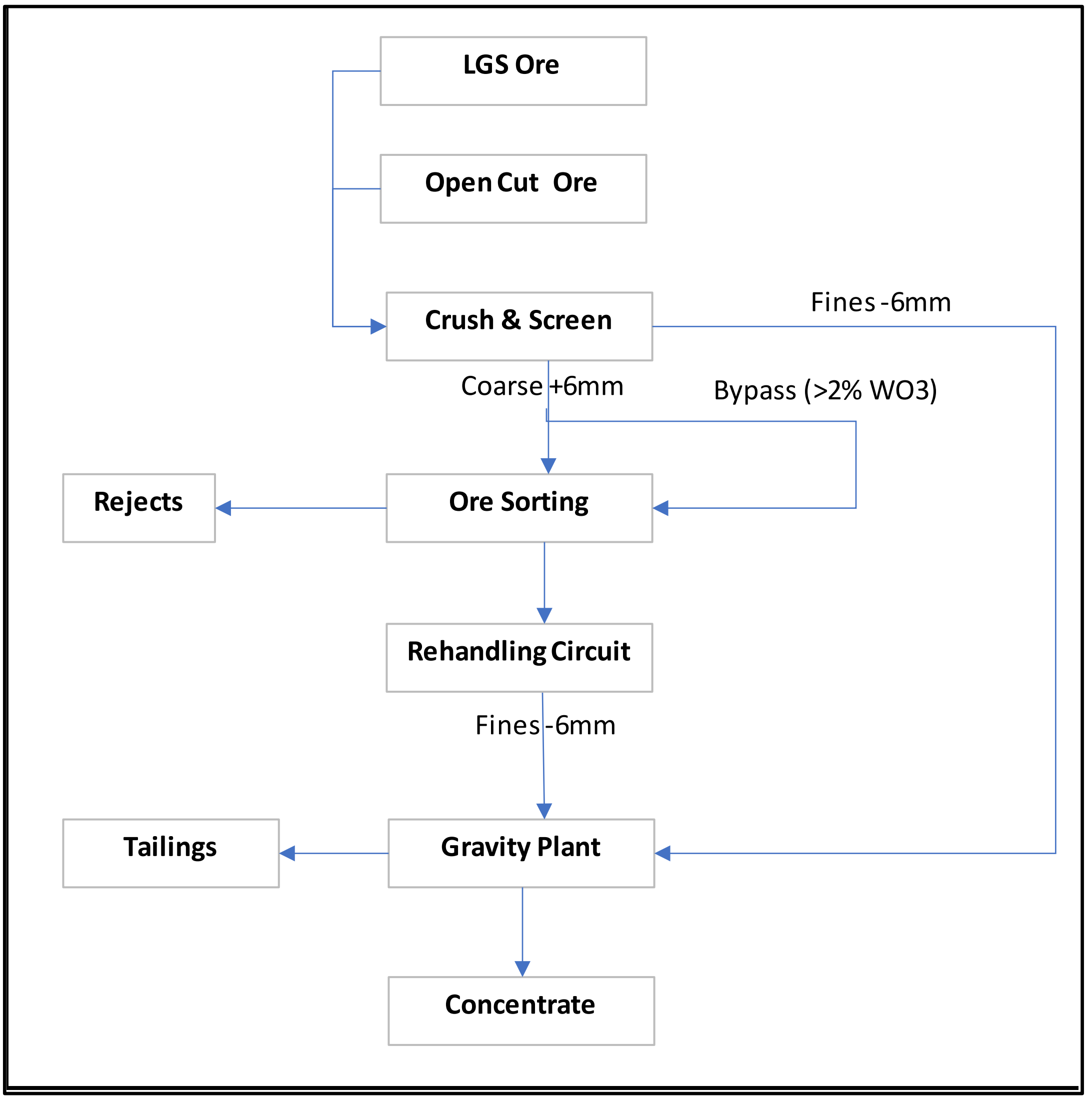

Figure 3.10 – Mt Carbine Ore Processing Flowchart

The above flow chart includes a combination of both existing infrastructure and proposed new facilities.

Table 3.3 summarises the average metallurgical factors achieved at each section of the processing flowchart.

Table 3.3 – Average Metallurgical Factors

Factor	Value
Ore Sorter Tungsten Recovery	90.3%
Wet Plant Fines Tungsten Recovery	79.3%
Wet Plant Coarse Tungsten Recovery	90%
Wet Plant Total Tungsten Recovery	82.7%
Total Tungsten Recovery	80.3%

3.5.3 Existing Infrastructure

The existing crushing and screening flowsheet consists of two stages crushing and dry screening circuits to produce two products:

1. –6mm wet plant feed
2. +6mm, -40mm ore sorter feed

Run of mine (ROM) ore (-700mm) is delivered to the fixed jaw crusher which has a closed side setting of -75mm. The jaw crusher discharge belt transfers primary crushed ore onto a 900mm wide screen feed conveyor. The screening plant consists of a mobile fitted with two decks to split the feed into two streams:

1. Oversize (+40mm) to the cone crusher circuit
2. Undersize (-6mm) to the -6mm stockpile

The secondary cone crusher discharge is fed onto a belt conveyor and recirculates back to the sizing screen for separation into product sizes.

The existing ore sorter consists of a single hopper feed point, dry screen to dress ore before the ore sorter and a single ore sorter. The ore sorter circuit produces two products:

1. Rejects
2. +6mm to -40mm ore sorter oversize that is crushed by a cone crusher to -6mm for feed into the processing plant.

Ore is fed into the existing processing plant and onto a wet screen which separates the -6mm material and the +6mm material. The +6mm material is sent back to the ore sorter for processing.

The -6mm particles are pumped to a pulse jig where the high density, tungsten bearing particles are concentrated and pumped to a secondary wet screen with 0.8mm panels on the screen. The +0.8mm particles are fed to a rolls crusher and then pumped back to the front of the screen while the -0.8mm sized material is dewatered and sent to six shaking tables.

The shaking tables produce a rougher concentrate which is pumped to a final cleaner table. The tailings from the rougher tables are pumped back to the screen, to be jigged once more to minimise losses and increase recovery. The cleaner table produces a final concentrate which is bagged immediately. The tailings from the cleaner table are pumped back to the secondary screen, to undergo sizing and crushing once more to ensure minimal losses.

A significant amount of data is available on the metallurgical performance of the existing processing infrastructure.

3.5.4 New Infrastructure

The upgraded ore sorter circuit flowsheet has been prepared by Mincore, a minerals processing and engineering consultancy.

The ore sorter will be upgraded to accommodate the proposed increase in annual ore tonnage. The treatment rate will be 80tph to achieve an annualised throughput of 525,600 tonnes.

Additional processing infrastructure, which will allow the site to mine up to 1mtpa of ore, has been designed and costed (for both capital and operating costs) by

Ausenco, a multinational engineering consultancy firm, in 2021 to feasibility level of detail.

The proposed additional processing infrastructure will process ore at a rate of 60tph. Historical performance data plus results of metallurgical test work Minerals Institute for the ore sorter has been referenced when calculating the performance of the ore sorting facilities.

Historical performance data plus results of laboratory metallurgical testing completed by Ausenco as part of the plant expansion project has been referenced when analysing the performance of the processing plant.

3.6 Equipment Waste Allocation

Waste will be mined by both the 120t and 50t excavators, however the majority of the waste will be excavated by the large excavator as the small excavator will be the only dig unit which mines ore.

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3.7 Detailed Mine Schedules

A detailed schedule has been compiled in Spry based off the economic pit shell as described in Section 3.2.

The mine schedule comprises of five main processes:

- Drilling
- Blasting
- Waste mining
- Ore mining
- Low grade stockpile mining

Two stripping fleets were included in the schedule, a 120t excavator and a 50t excavator. The 120t excavator mines only waste whilst the 50t excavator is able to mine ore and waste. All waste and ore was assumed to require drilling and blasting.

The productivities for all drilling and blasting plus mining equipment is summarised in Table 3.4

Table 3.4 – Equipment Productivities

Equipment	Process	Rate
Drill	Waste Drilling	40m/hr
	Ore Drilling	40m/hr
Blast Crew	Waste Loading	0.5t/hr
	Ore Loading	0.5t/hr
50t Excavator	Waste Mining	176 t/hr
	Ore Mining	150 t/hr
	Ore Mining – LGS	176 t/hr
120t Excavator	Waste Mining	437 t/hr
	Ore Mining – LGS	386 t/hr

Practical dependencies and constraints were applied to ensure the correct progression of the equipment and the mine in general. Benches were developed away from the ramp with two active faces allowed on each bench.

The strategy of the schedule was to develop the open pit in a logical manner and top up the annual ore quantity to 1mt via feed from the low grade stockpile. The ramp up in production is based on the estimated dates for receiving the required approvals which are summarised below:

- EA approval to process 500,000tpa - Estimated on 1/4/2022
- EA approval to process 1mtpa - Estimate on 1/1/2023

Dump and haulage modelling was completed as part of the detailed schedule, with a swell factor of 25% applied to all prime material. All waste and ore is assumed to be loaded onto 45t articulated dump trucks.

Mine physicals and mine scheduling results for the detailed equipment schedule is provided in Table 3.5 with a summary of major physicals and qualities for the schedule provided in Figure 3.11 to Figure 3.13.

Table 3.5 – Annual Quantities and Qualities

Item	Units	2022	2023	2024	2025	2026-2033	2034	TOTAL
Tonnes								
Waste Tonnes	t	0	5,616,886	5,999,519	2,393,770	0	0	**14,010,174**
Ore Tonnes	t	0	171,304	620,077	472,647	0	0	**1,264,028**
LG Ore Tonnes	t	385,010	828,696	379,923	527,353	1,000,000	5,247	**10,126,228**
Total Tonnes	**t**	**385,010**	**6,616,886**	**6,999,519**	**3,393,770**	**1,000,000**	**5,247**	**25,400,431**
Drill Metres								
Waste Drill	m	0	306,403	328,547	186,334	0	0	**821,284**
Ore Drill	m	0	5,723	20,483	16,235	0	0	**42,441**
Total Metres	**m**	**0**	**312,126**	**349,029**	**202,570**	**0**	**0**	**863,725**
Blast Tonnes								
Waste Blast	t	0	4,398	4,760	2,612	0	0	**11,770**
Ore Blast	t	0	89	319	241	0	0	**648**
Total Tonnes	**t**	**0**	**4,487**	**5,079**	**2,853**	**0**	**0**	**12,419**
Processing								
Crushed & Screened Tonnes	t	385,010	1,000,000	1,000,000	1,000,000	1,000,000	5,247	**11,390,257**
Crushed & Screened WO3 Grade	%	0.075%	0.185%	0.471%	0.376%	0.075%	0.075%	**0.146%**
Fines Tonnes	t	138,604	360,000	360,000	360,000	360,000	1,889	**4,100,492**
Fines Grade	%	0.113%	0.277%	0.706%	0.565%	0.113%	0.113%	**0.219%**
Ore Sorter Feed Tonnes	t	246,406	640,000	640,000	640,000	640,000	3,358	**7,289,764**
Ore Sorter Feed Grade	%	0.054%	0.133%	0.338%	0.271%	0.054%	0.054%	**0.105%**
Ore Sorter Product Tonnes	t	39,425	102,400	102,400	102,400	102,400	537	**1,166,362**
Ore Sorter Product Grade	%	0.304%	0.749%	1.909%	1.527%	0.304%	0.304%	**0.592%**
Ore Sorter Rejects	t	206,981	537,600	537,600	537,600	537,600	2,821	**6,123,402**
Gravity Plant Feed Tonnes	t	178,029	462,400	462,400	462,400	462,400	2,426	**5,266,855**
Gravity Plant Feed Grade	%	0.155%	0.381%	0.972%	0.778%	0.155%	0.155%	**0.301%**
Concentrate Tonnes	t	464	2,966	7,560	6,046	1,205	6	**26,679**
Concentrate WO3 Tonnes	t	232	1,483	3,780	3,023	602	3	**13,340**
Tailings Tonnes	t	177,565	459,434	454,840	456,354	461,195	2,420	**5,240,176**

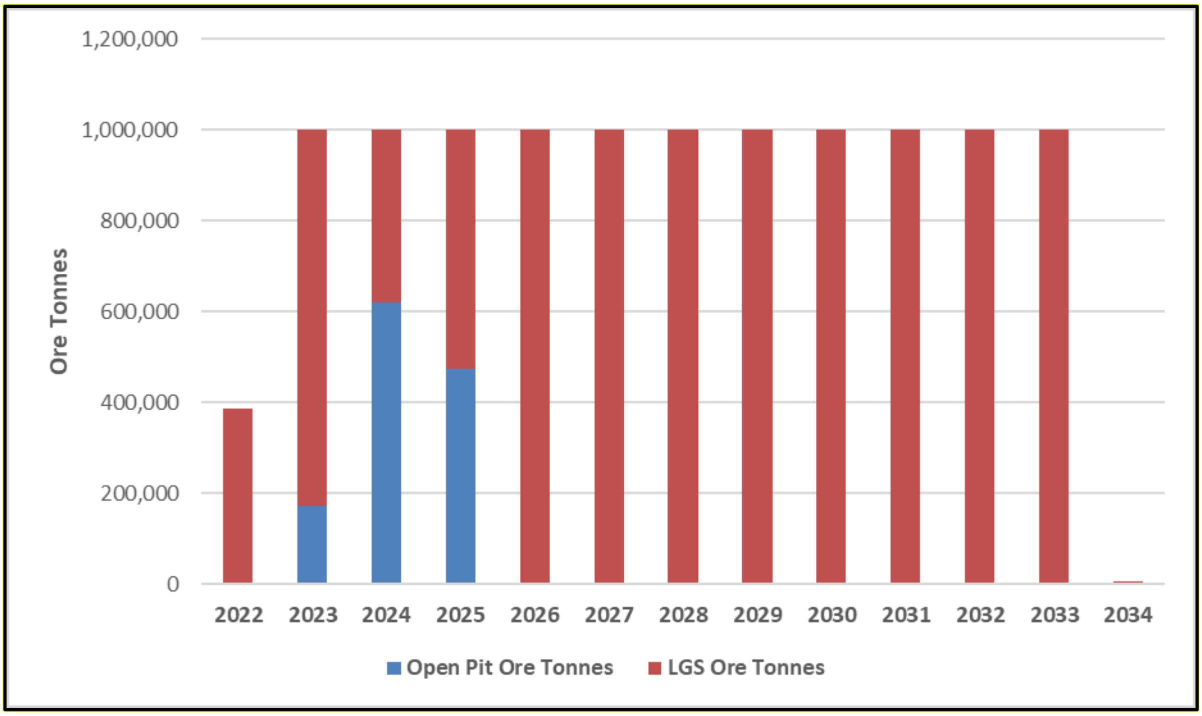

Figure 3.11 – Annual Ore Tonnes

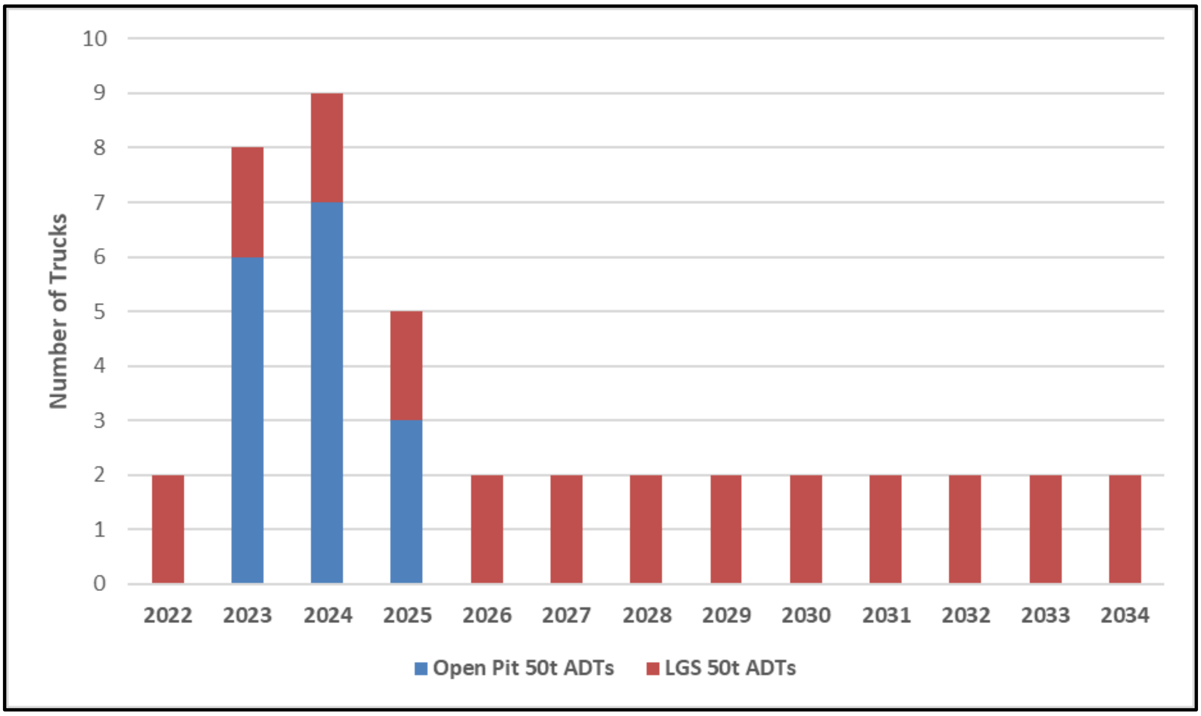

Figure 3.12 – Annual Truck Numbers

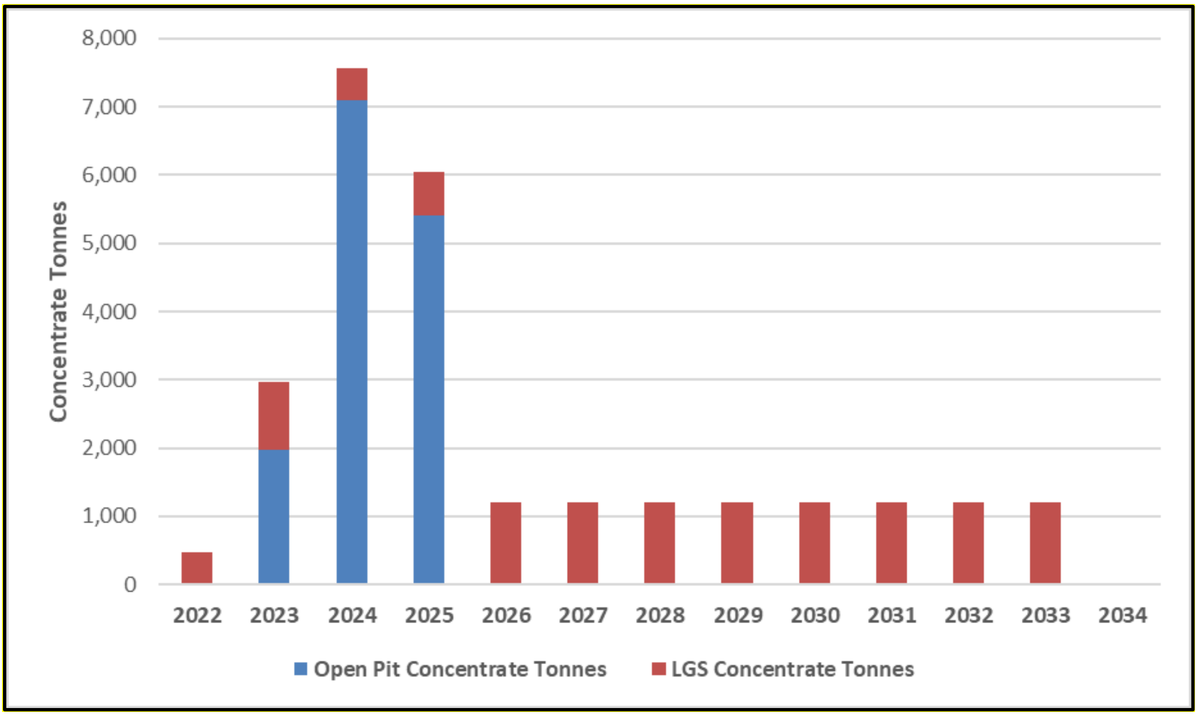

Figure 3.13 – Concentrate Production

Stage plans showing the progression of the mine at the end of each year of the operation of the open pit are provided in Figure 3.14 to Figure 3.18.

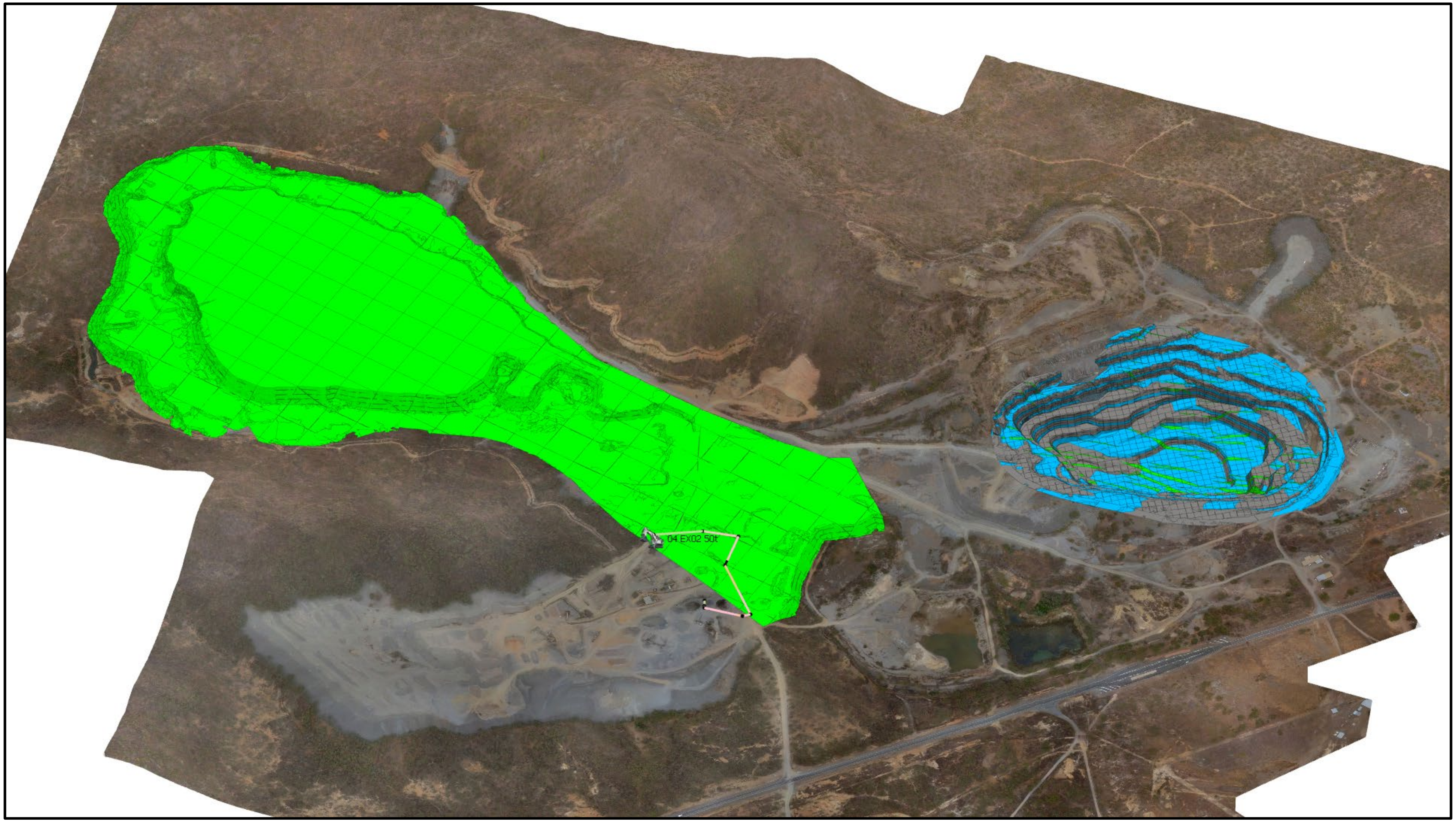

Figure 3.14 – Starting Topography

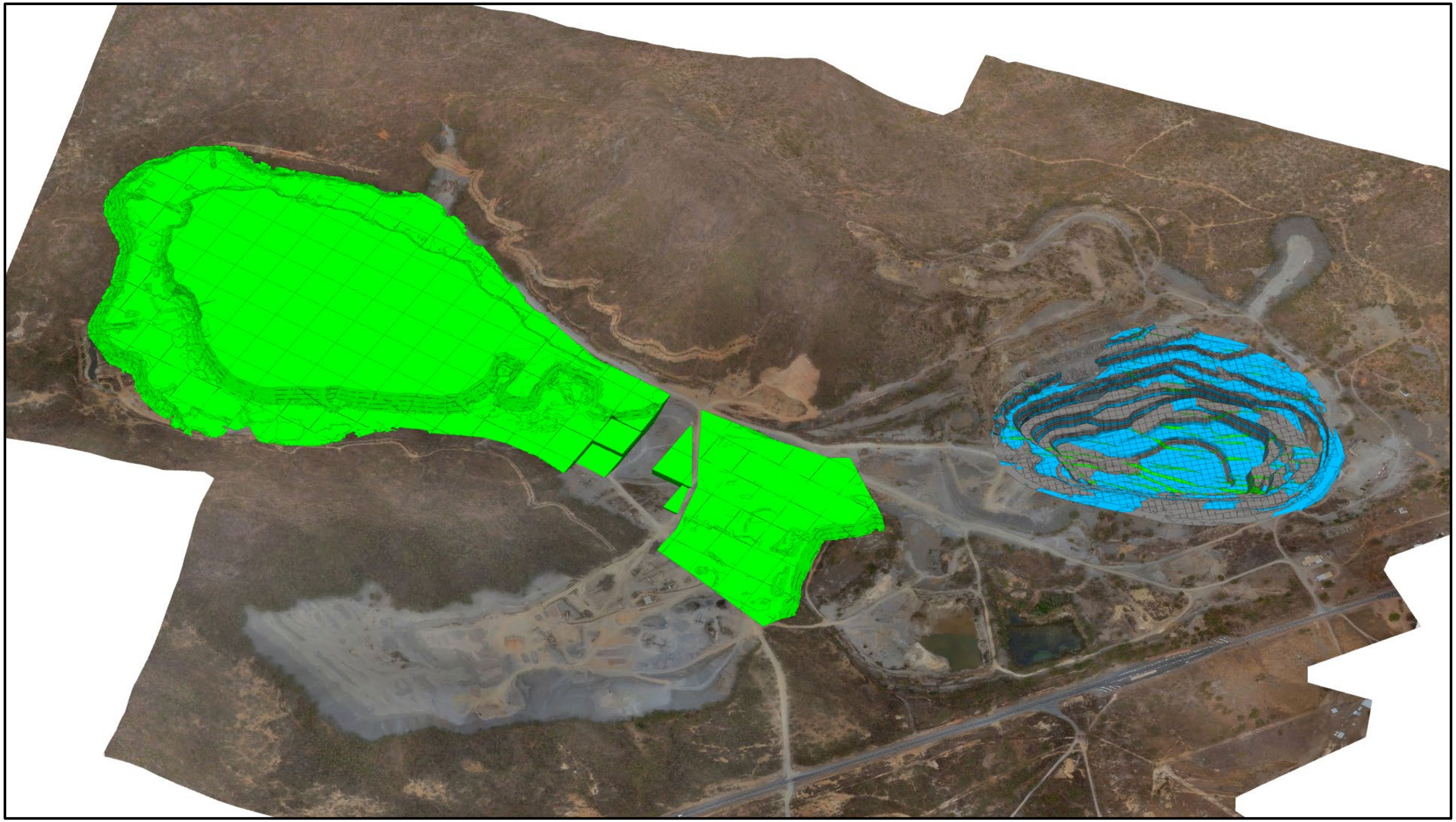

Figure 3.15 – Mine Status at end of 2022

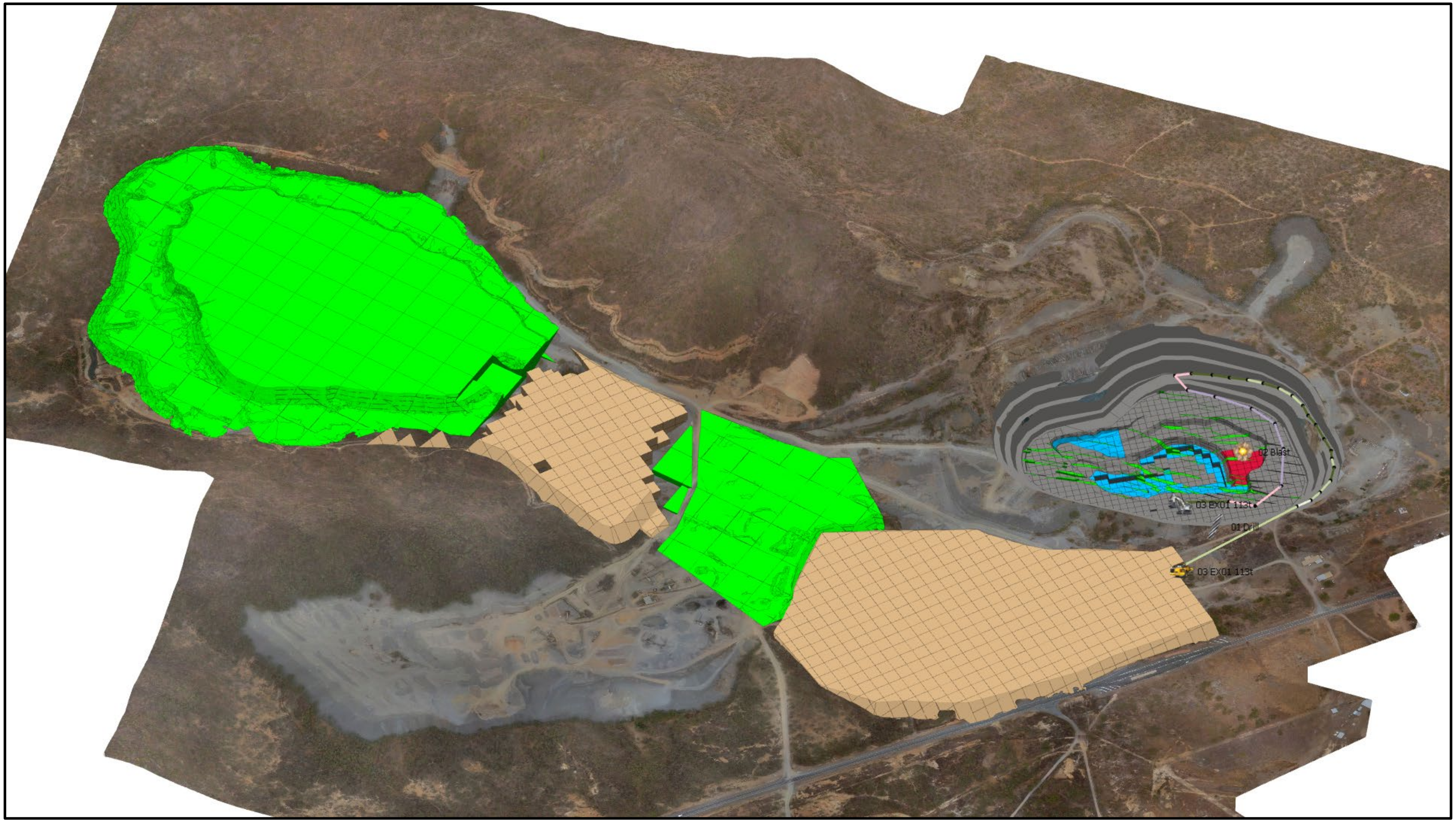

Figure 3.16 – Mine Status at end of 2023

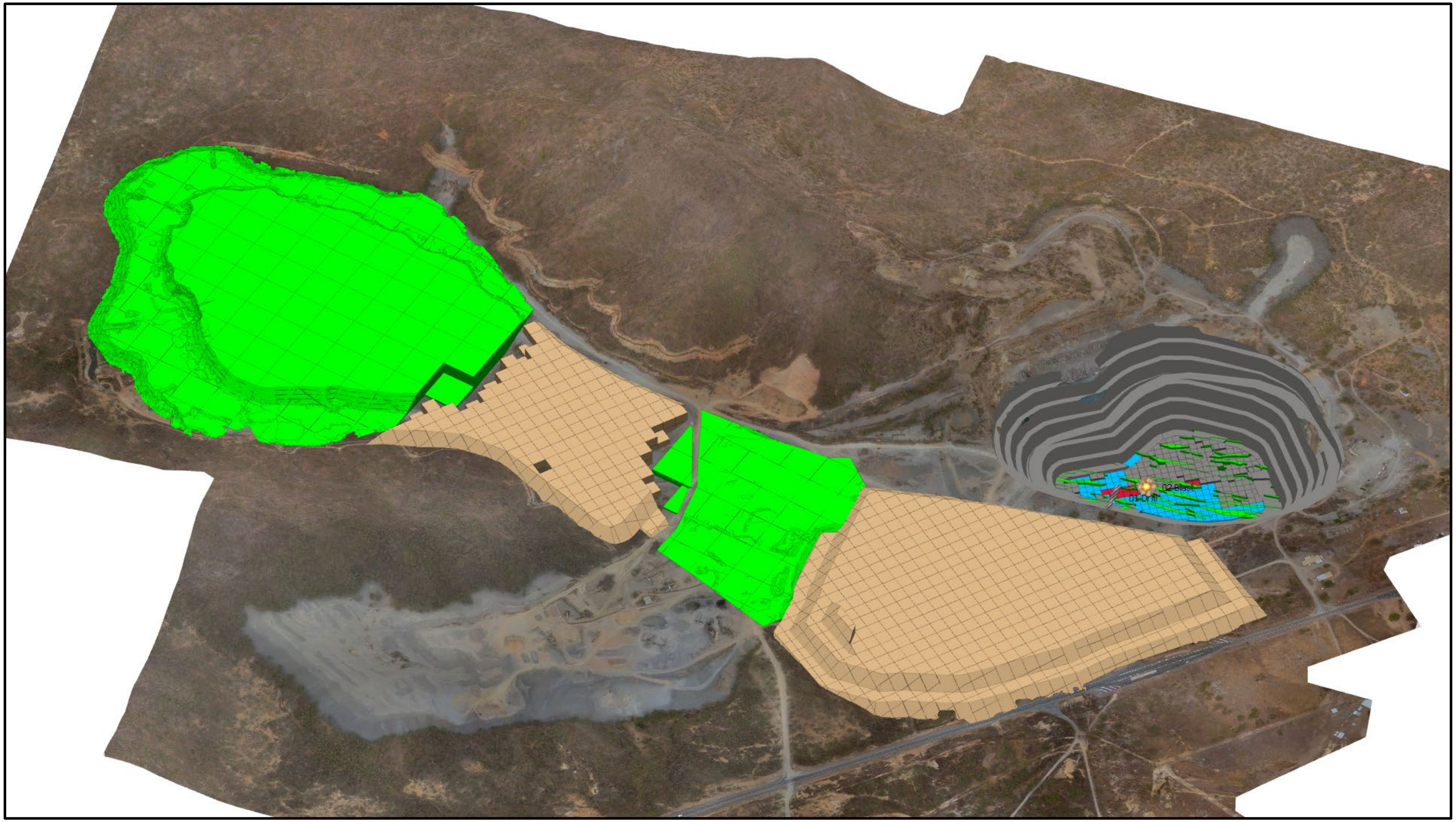

Figure 3.17 – Mine Status at end of 2024

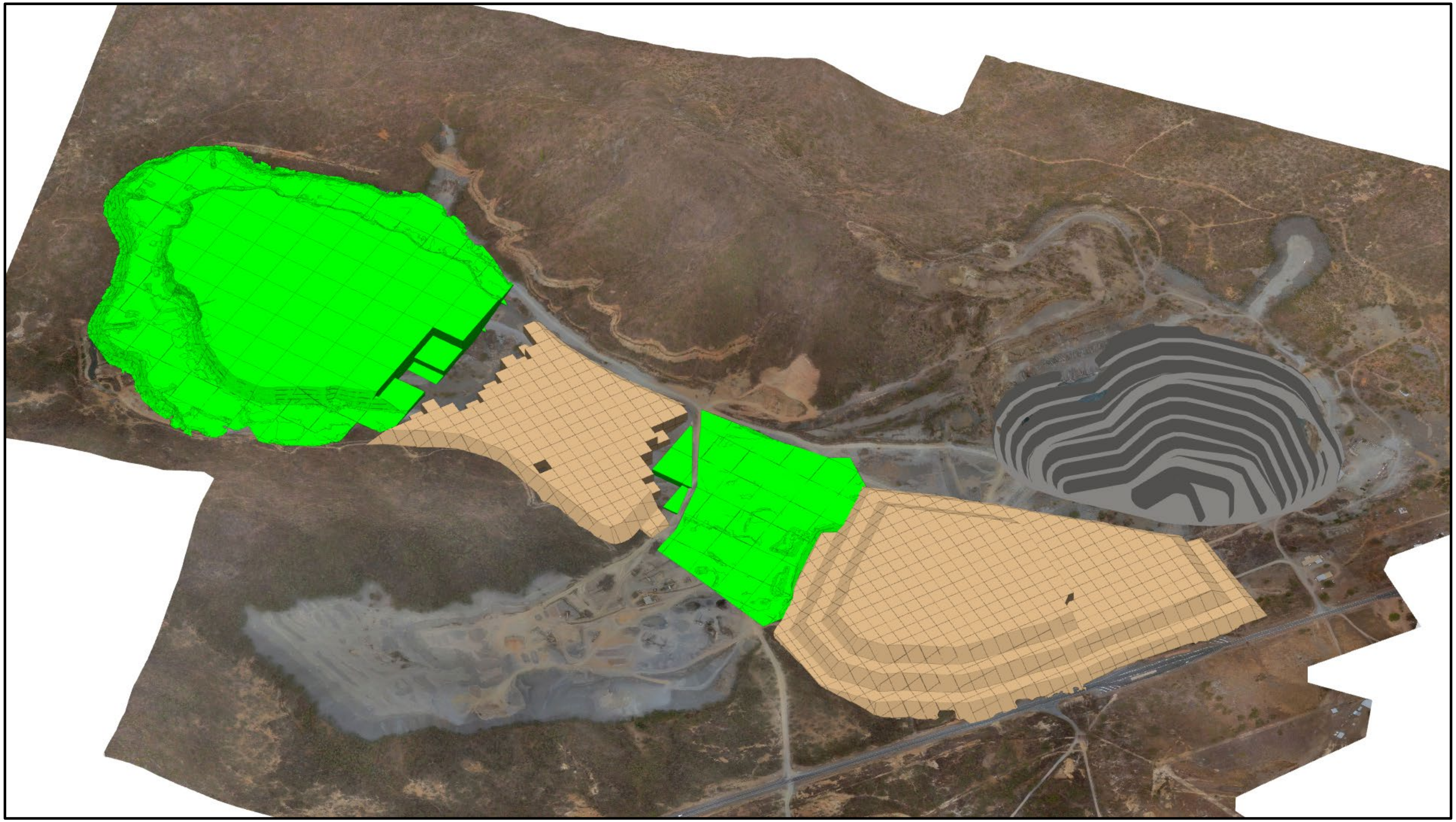

Figure 3.18 – Mine Status at end of 2025

4. Financial Modelling

4.1 Financial Assumptions

A financial model was compiled based on the physicals generated from the detailed schedule. The financial modelling was based on a contractor operation undertaking all mining and processing activities with EQR providing the management and administrative roles. The details of the unit costs applied in the financial model are sensitive and have not been disclosed in this report.

Allocations for initial capital estimates were generated from a combination of sources including:

- feasibility level information for processing infrastructure,
- feasibility level information for the upgrade of power and water infrastructure,
- Market capital costs for mining equipment,
- Estimate decommissioning and mine closure costs,
- Contractor mobilisation and demobilisation.

4.2 Equipment Numbers

The number of trucks required were calculated from the haulage modelling which determined the annual truck hours required for ore, waste and low grade stockpile mining. Ancillary equipment hours, for items such as graders and dozers, were then calculated using industry standard factors which are summarised below in Table 4.1.

Table 4.1 – Ancillary Equipment Number Calculation Factors

Cost Item	Units	Unit Cost
Dozer - Pit	hrs/exc. hr	1.50
Dozer - Dump	hrs/exc. hr	0.50
Grader	hrs/truck hr	0.12
Water Truck	hrs/truck hr	0.15
Service Truck	$/fleet hr	0.05

Fleet sizes were calcualted by dividing the total hours for each piece of equipment by the annual operating hours achieved. These values vary by equipment with priority equipment achieving 5,600 hours annually whilst low utilisation equipment achieve 3,500 hours.

The maximum size of the mining fleet, when the open pit is in fully production, is summarised in Table 4.2.

Table 4.2 – Maximum Mining Fleet Numbers

Cost Item	Maximum Fleet Size
120t Excavators	1
50t Excavators	1
45t Articulated Dump Trucks	9
Drills	2
Dozers	2
Graders	1
Water Trucks	1
Service Trucks	1

4.3 Mining and Processing Unit Costs

4.3.1 Cost Assumptions

Unit costs have been supplied by EQ Resources for current operational practices and Ausenco for processing costs of the upgraded facilities. Mining costs, including all drill and blast, were split into equipment, labour and fuel costs with benchmark fuel burn rates applied for all mobile equipment.

Processing costs were based on historical performance at Mt Carbine as well as feasibility level assessments completed by Ausenco and Mincore on the proposed upgraded processing facilities.

Application of all operating costs, both fixed and variable, results in an average production cost of approximately $10,250 per concentrate tonne across the life of the mine. The production cost of each tonne of 50% WO3 concentrate produced from the low grade stockpile is $13,100, whilst concentrate generated from the open-pit costs approximately $7,860 per tonne.

4.3.2 Revenue Assumptions

Revenue assumptions are based on the forecast concentrate prices for WO3 APT price of US$31,500 per tonne with a AUD:USD exchange rate of 0.73 applied. Historical realized price adjustment factors were then applied as well as discounts for producing a concentrate with 50% WO3. The final assumed realised price for each tonne of 50% WO3 concentrate is AU$15,100.

Current off-take agreements consider the following potential deleterious elements:

- Sulphur
- Tin
- Molybdenum
- Lead

- Arsenic
- Water

None of the above elements have been modelled in either the low grade stockpile or open-pit geological models. However, forecast sale prices, which align with current off-take agreements, apply a substantial penalty to the benchmark tungsten concentrate price reflecting the potential presence of deleterious elements which Mt Carbine concentrate may contain. Historically, Mt Carbine has produced concentrate with relatively high levels of arsenic, however the processing plant proposed by Ausenco contains an arsenic removal module which will be used when levels of this element become too high.

Historically, Mt Carbine concentrate has been sold to customers in several locations including Europe, the United States, Vietnam and China reflecting the acceptance of the product in the open market.

Despite currently generating income from quarry material, no revenue has been generated from this procedure as part of the economic pit shell calculation evaluation of the Reserves.

4.4 Financial Results

The net present value of the operation was calculated using a discount rate of 8%. The project generated a good net present value figure with all levels of the planned open pit and section of the low-grade stockpile economical to mine and process.

The competent person is satisfied that the proposed mining activities within the mine plan which underpins the Ore Reserve estimate are economical to mine and process.

5. Reserve Estimate

5.1 Reserve-Resource Clarification

All Ore Reserves have been classified as Probable Reserves which are subsets of areas of Indicated Resources category.

5.2 Physical Limits

The physical limits of the open pit and low grade stockpile Ore Reserves are shown in Figure 5.1 and Figure 5.2 respectively.

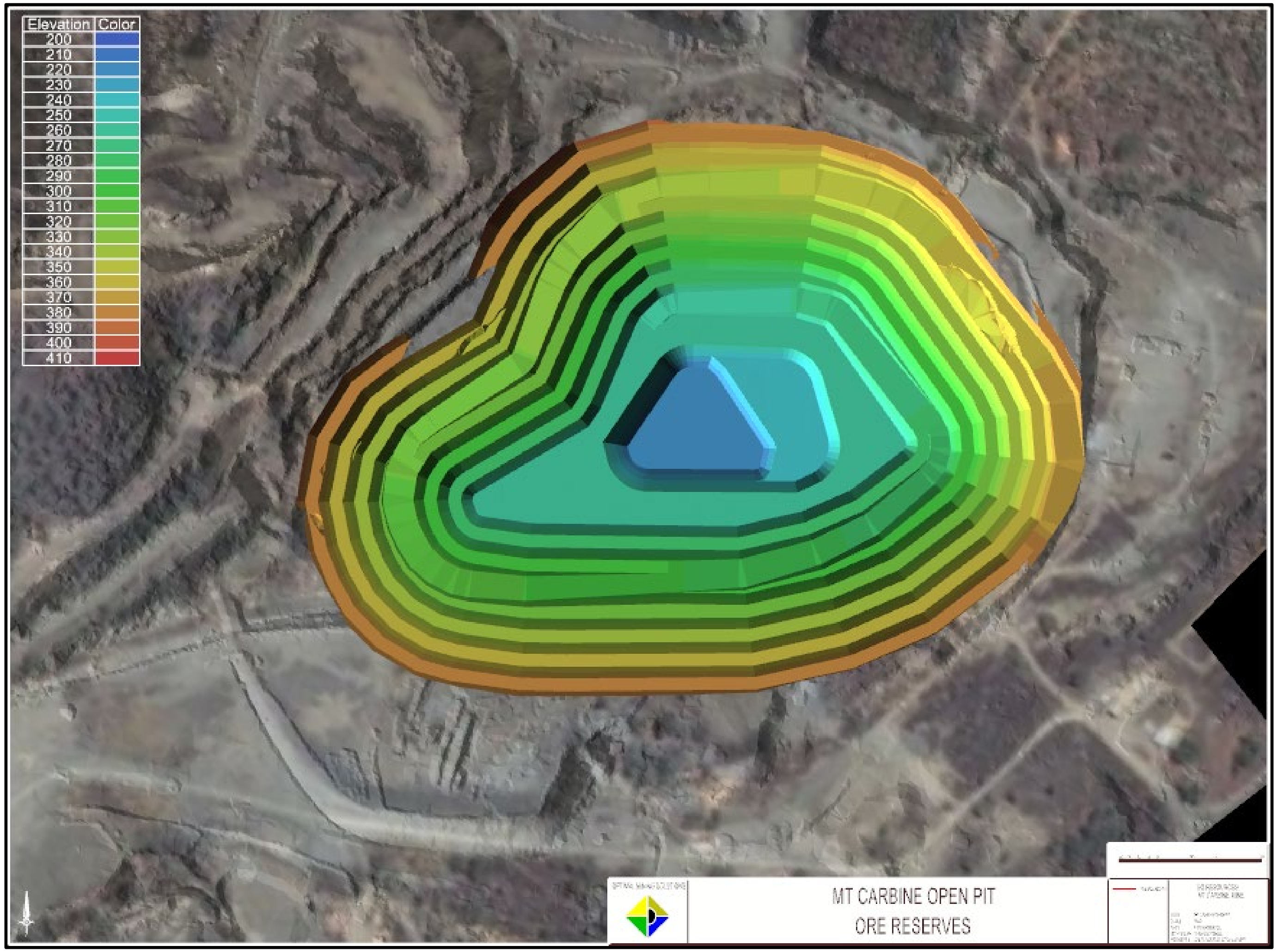

Figure 5.1 – Open Pit Ore Reserve Areas

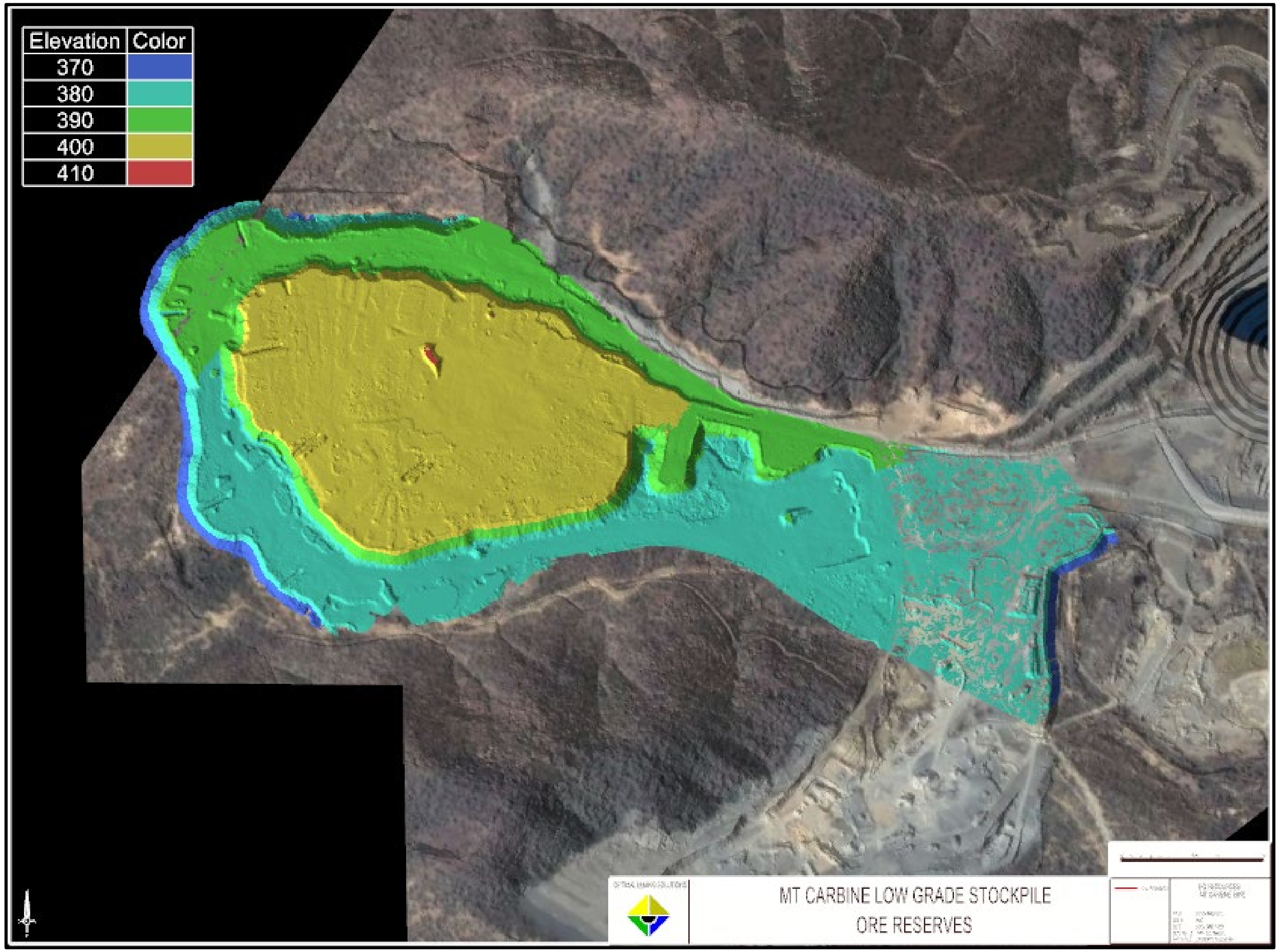

Figure 5.2 – Low Grade Stockpile Ore Reserve Areas

5.3 Ore Reserves

The estimated Ore Reserves for the open pit and low grade stockpile are presented in Table 5.1 and Table 5.2 respectively.

Table 5.1 – Mt Carbine Project – Low Grade Stockpile Reserves Estimate

Reserve Category	ROM Tonnes (mt)	WO3 %
LGS - Proved	0	-
LGS - Probable	10.126	0.075%
LGS - Total	**10.126**	**0.075%**

Table 5.2 – Mt Carbine Project - Open Cut Reserves Estimate

Reserve Category	ROM Tonnes (mt)	WO3 %
Open Pit - Proved	0	-
Open Pit - Probable	1.263	0.713%
Open Pit - Total	**1.263**	**0.713%**

** Tonnages in the above table are expressed on a ROM basis, incorporating the effects of mining losses and dilution.*

5.4 Accuracy of Estimate

Small differences may be present in the totals due to tonnage information being rounded.

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Section 4 Estimation and Reporting of Ore Reserves

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• *Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.* • *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	• The Ore Reserves have been based on two separate block models, one for the low grade stockpile and the other for the open pit operation. • The geological model used to develop the final low grade stockpile resource model was generated by Measured Group Pty Ltd in August 2021 and is titled 'Mt_Carbine_LGS_20210820.bmf'. • The geological model used for the open pit operation was developed by Measured Group in September 2021 and is titled "Mt_Carbine_20210918.bmf". • The Mineral Resources are inclusive of the Ore Reserves.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• The competent persons are yet to visit site due to previous travel restrictions imposed by the Queensland government. However, the competent person has been provided with a large number of digital photos and videos of the current operation including the old open pit, the low grade stockpile, processing plant, tailings storage and dumping areas. The photos also provided information on the condition of the current infrastructure on site including the processing plants. • The provision of this information was deemed to be adequate for the competent persons to be comfortable that the current site infrastructure plus proposed new infrastructure is suitable for ongoing mining and processing at the site.
Study status	• *The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.* • *The Code requires that a study to at least Pre-Feasibility*	• The Ore Reserves are based on mining information that has been completed to an acceptable level of detail at the time of estimation, with a detailed mine design cut into

Criteria	JORC Code explanation	Commentary
	Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	two phases and subdivided into 3.5m high benches. A bankable feasibility study has commenced, which will continue to improve the level of detail in the mining section of the project. • The designed pit solids were intersected with the latest geological model and then adjusted for loss and dilution. • A bench-by-bench schedule was compiled with the insitu, ROM and product information for each dig solid analysed in a financial model. • Upgrades to the processing equipment have been completed to a feasibility level of detail by Ausenco. • Key performance parameters such as unit operating costs, metallurgical parameters, etc. have been based on historical performance at site where practical.
Cut-off parameters	• *The basis of the cut-off grade(s) or quality parameters applied.*	• A cut-off grade of 0.075% WO3 has been applied for calculation of the Reserve within the low grade stockpile. • A cut-off grade of 0.2% WO3 has been applied in the open pit geological model, however after loss and dilution calculations are completed, the final feed grade to the processing plant is as low as 0.121%. • A cut-off grade analysis has indicated that these two parameters are conservative and generate decent cash flows.
Mining factors or assumptions	• *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).* • *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.*	• Mining of the low grade stockpile is currently being undertaken by a 50t excavator and fleet of 45t articulated dump trucks. • Front end loaders with, ~6m^3 buckets, will be used around the crushers, screens, ore sorters and for general clean up. • Mining of the open cut operation will be completed in two phases: ○ Phase 3 - completion of the current pit design

Criteria	JORC Code explanation	Commentary
	• *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.* • *The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).* • *The mining dilution factors used.* • *The mining recovery factors used.* • *Any minimum mining widths used.* • *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.* • *The infrastructure requirements of the selected mining methods.*	down to RL300 ○ Phase 4 – completion of a new phase which widens most of the current pit, with the exception of the north-west corner, down to RL200. • The open pit will be mined with 1 x 120t class excavator, with a $7m^3$ bucket and a fleet of 45t articulated dump trucks. A secondary fleet with a 50t excavator, fitted with a ~$3m^3$ bucket, will also be used to mine ore as cleanly as possible. The two fleets will move up to 5.4mt of waste and ore annually. • The open pit operations will be supported by ancillary equipment including a grader, water cart and dozers. • A $3m^3$ bucket on the secondary excavator will allow the excavator to selectively mine the relatively thin orebodies and keep dilution quantities to a minimum. • All waste and ore will be drilled and blasted at a powder factor of approximately 0.8 kg/bcm of material. • Due to the vertical and relatively this nature of the orebodies, grade control will be paramount. It is proposed to complete grade control via a combination of mapping, face sampling and grade control drilling, utilizing mostly angled holes. • Open-pit ramps have been designed at 10% maximum gradient at a width of 15m which is greater than 3.5 times the largest vehicle regularly operating on the ramp, the 45t ADTs. • Geotechnical parameters for the majority of the open pit are based on the existing pit's design which has performed well and remained relatively unchanged since mining stopped in the 1980s. The key geotechnical parameters for the open pit wall are:

Criteria	JORC Code explanation	Commentary
		○ Batter height - 20m ○ Batter angle - 70 degrees ○ Berm width - 8.5m • The only area of geotechnical risk exists on the southern wall near the south wall fault. A geotechnical analysis of this wall indicates that the current pit design will require rock bolting at close intervals to minimise the probability of this wall causing geotechnical disruptions. Capital and operating cost allowances have been made in the financial assessment to monitor and treat this wall as it is exposed in the final wall. The two upper benches in the south wall, which are located in weathered material, have been excavated at 50 degrees and 57 degrees as specified by the geotechnical assessment. • Due to the presence of an ore sorter and valuable narrow veins, very high dilution values have been applied on the assumption that the entire vein will be mined and sent to the processing plants. • Dilution has been added in both the geological model and the mining model. Between the two allocations, approximately 101% dilution has been added. • Conversely, due to the very high extraction quantities a low loss value (1%) has been applied to all modelled ore.
Metallurgical factors or assumptions	• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* • *Whether the metallurgical process is well-tested technology or novel in nature.* • *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.*	• The low grade stockpile at Mt Carbine is processed through a combination of crushers, screens, an ore sorter and wet plant circuits to generate a concentrate containing approximately 50% WO_3. **Existing Infrastructure** • The existing crushing and screening flowsheet consists of

Criteria	JORC Code explanation	Commentary
	• *Any assumptions or allowances made for deleterious elements.* • *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.* • *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	two stages crushing and dry screening circuits to produce two products: 1. –6mm wet plant feed 2. +6mm, -40mm ore sorter feed • Run of mine (ROM) ore (-700mm) is reclaimed from the low grade waste stockpile and is delivered to the fixed jaw crusher. The jaw crusher has a closed side setting of -75mm. The jaw crusher discharge belt transfers primary crushed ore onto a 900mm wide screen feed conveyor. The screening plant consists of a mobile fitted with two decks to split the feed into two streams: 1. Oversize (+40mm) to the cone crusher circuit 2. Undersize (-6mm) to the -6mm stockpile • The secondary cone crusher discharge is fed onto a belt conveyor and recirculates back to the sizing screen for separation into product sizes. • The existing ore sorter consists of a single hopper feed point, dry screen to dress ore before the ore sorter and a single ore sorter. The ore sorter circuit produces two products: 1. Rejects 2. +6mm, -40mm ore sorter oversize that is crushed by a cone crusher to -6mm for feed into the processing plant. • Ore is fed into the existing processing plant and onto a wet screen which separates the -6mm material and the +6mm material. The +6mm material is sent back to the ore sorter for processing. • The -6mm particles are pumped to a pulse jig where the high density, tungsten bearing particles are concentrated and pumped to a secondary wet screen with 0.8mm panels on the screen. The +0.8mm particles are fed to a rolls crusher and then pumped back to the front of the screen while the -0.8mm sized material is dewatered and sent to six

Criteria	JORC Code explanation	Commentary
		shaking tables. • The shaking tables produce a rougher concentrate which is pumped to a final cleaner table. The tailings from the rougher tables are pumped back to the screen, to be jigged once more to minimise losses and increase recovery. The cleaner table produces a final concentrate which is bagged immediately. The tailings from the cleaner table are pumped back to the secondary screen, to undergo sizing and crushing once more to ensure minimal losses. • A significant amount of data is available on the metallurgical performance of the existing processing infrastructure. **New Infrastructure** • The existing ore sorter will be upgraded to accommodate the proposed increase in annual ore tonnage. The treatment rate will be 80 tph to achieve an annualised throughput of 525,600 tonnes. • The upgraded ore sorter circuit flowsheet has been prepared by Mincore, a minerals processing and engineering consultancy. • Additional processing infrastructure, which will allow the site to mine up to 1mtpa of ore, has been designed and costed (both capital and operating) by Ausenco in 2021, a multinational engineering consultancy firm, to BFS-level of detail. • The proposed additional processing infrastructure will process ore at a rate of 60tph. • Historical performance data plus results of metallurgical test work completed by the Sustainable Minerals Institute in 2021 has been referenced when analysing the performance of the ore sorter. • Historical performance data plus results of laboratory

Criteria	JORC Code explanation	Commentary
		metallurgical testing completed by Ausenco as part of the plant expansion project has been referenced when analysing the performance of the processing plant. • Current off-take agreements consider the following potential deleterious elements: ○ Sulphur ○ Tin ○ Molybdenum ○ Lead ○ Arsenic ○ Water • None of the above elements have been modelled in either the low grade stockpile or open-pit geological models. However, forecast sale prices, which align with current off-take agreements, apply a substantial penalty to the benchmark tungsten concentrate price reflecting the presence of deleterious elements which Mt Carbine concentrate may contain. Historically, Mt Carbine has relatively high levels of arsenic and the processing plant proposed by Ausenco contains an arsenic removal module which will be used when levels of this element become too high. • Historically, Mt Carbine concentrate has been sold to customers in several locations including Europe, the United States, Vietnam, and China reflecting the acceptance of the product in the open market.
Environmen-tal	• *The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable,*	• The site currently has all required environmental approvals to mine, crush and screen material from the pit. The mine and quarry activities occur on previously disturbed lands. • The surrounding land use is rural-urban (Mount Carbine

Criteria	JORC Code explanation	Commentary
	the status of approvals for process residue storage and waste dumps should be reported.	township), low-intensity cattle grazing, mining and exploration, and conservation (the Brooklyn Nature Refuge). • The background land tenure (Lot 13 on SP254833) is Brooklyn Nature Refuge, which is held by the Australian Wildlife Conservancy as a rolling term lease – pastoral (Title Reference 17664140); a special condition of this lease is to allow quarry material to be removed. • There are no wetlands of national or international significance mapped in the project site or the receiving environment. • There are no High Ecological Value Waters (watercourses), High Ecological Value Waters (wetlands) or Wetlands of High Ecological Significance mapped in the project site or the receiving environment. • Waste rock has historically shown minimal to no acid producing potential. Waste rock characterization has not been completed at Mt Carbine, therefore selective placement of this material has not been included as part of the scheduling and haulage modelling work.
Infrastructure	• *The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.*	• Mt Carbine is an operational site and is supported by well-established infrastructure for the current mine and quarrying operations. Current facilities include offices, laboratory, ablutions as well as crushing, screening and processing facilities. • Mt Carbine's current processing facilities can process ore at approximately 60tph, however this will be increased to accommodate the planned 1mt of ore mined annually. Capital costs for the required crushing, screening and processing infrastructure have been estimated to a BFS level of detail and included in the overall economic evaluation of the site. The competent persons are satisfied that enough detail has been included in the capital cost estimate for the

Criteria	JORC Code explanation	Commentary
		new processing facilities. • Access to site has already been established via the Mulligan Highway which runs through the operation. • Power to the site is currently supplied via two supplies segregated by the Mulligan highway into east and west. The eastern side is supplied by a 315 kVA overhead transformer whilst the western side is supplied by a 1000 kVA pad mounted transformer. Power is distributed across the site by 22kV above-ground power lines. • Raw water for processing and operational activities is currently sourced from the open-pit. An alternate raw water storage will be confirmed in upcoming studies. A capital allowance for the establishment of a new raw water storage facility has been applied in the financial model. • Potable water is trucked to Mt Carbine and stored onsite in storage tanks for use at the site facilities.

Costs	• *The derivation of, or assumptions made, regarding projected capital costs in the study.*
	• *The methodology used to estimate operating costs.*
	• *Allowances made for the content of deleterious elements.*
	• *The source of exchange rates used in the study.*
	• *Derivation of transportation charges.*
	• *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.*
	• *The allowances made for royalties payable, both Government and private.*
		• Capital costs have been estimated at a feasibility level of detail for all required infrastructure for a 1mtpa ore operation. Capital costs allocations include:
		    ○ Crushing and screening upgrades,
		    ○ Processing facilities upgrades,
		    ○ Raw water facility construction,
		    ○ Contractor facilities,
		    ○ Contractor mobilization and demobilization,
		    ○ Future studies,
		    ○ Ongoing exploration.
		• Operating costs have been estimated based on a contractor-based operation with 1 x 120t class excavator, 1 x 50t excavator, a fleet of 45t ADTs and supporting ancillary equipment. All waste will be drilled and blasted by a down-the-hole service drill and blast contractor.
		• Processing costs have been estimated based on current

Criteria	JORC Code explanation	Commentary
		operational costs for existing equipment and processes, such as tailings disposal, plus BFS-level estimates for new processing infrastructure. • A state government royalty equal to 2.7% of generated revenue has been included in the cost structure.
Revenue factors	• *The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* • *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	• The Reserves are based on a WO3 APT price of US$31,500 per tonne with a AUD:USD exchange rate of 0.73 applied. • Historical realized price adjustment factors were then applied as well as discounts for producing a concentrate with 50% WO3. • Despite currently generating income from quarry material, no revenue has been generated from this procedure as part of the economic evaluation of the Reserves.
Market assessment	• *The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* • *A customer and competitor analysis along with the identification of likely market windows for the product.* • *Price and volume forecasts and the basis for these forecasts.* • *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	• Tungsten carbide, which has hardness close to diamond, is the most popular form of tungsten. It is denser than steel and titanium, twice as hard as any steel grade, and has extremely high wear resistance. The product is widely used in construction, mining, and metal working applications and is forecast to continue to perform strongly on the global market. • Mt Carbine currently produces concentrate which is sold to multiple locations around the world. • In 2020, approximately 84,000 tonnes of tungsten was produced globally with 69,000 tonnes sourced from China. Mt Carbine is forecast to produce up to 3,700 tonnes of tungsten annually which will have minimal effect on the global market.
Economic	• *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation,*	• All costs and revenues which have been used in the financial model are in nominal terms and have been discounted by 8% to generate the overall net present value of the project.

Criteria	JORC Code explanation	Commentary
	discount rate, etc. • *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	• As Mt Carbine is an operating mine/quarry with significant existing infrastructure, capital expenditure is minimal and therefore the project is not sensitive to NPV discount rate. • The competent persons are confident that Mt Carbine will generate positive cash flows once the initial capital outlays are undertaken early in the schedule. The subsequent years generate enough free cash to adequately pay for the capital costs incurred in 2022.
Social	• *The status of agreements with key stakeholders and matters leading to social licence to operate.*	• The project has good community engagement and has been discussed verbally with the local stakeholders, particularly the Mt Carbine Caravan Park, which stands to be the most impacted, and the response has been positive. • EQ Resources in accordance with its requirements pays Native Title Administration Fees to the Nguddaboolgan Native Title Aboriginal Corporation (NNTAC) and maintains regular dialogue and communication with any relevant information pertaining to its activities. • The underlying pastoral leases on which Mt Carbine is located are held by Australian Wildlife Conservancy on a parcel of land known as Brooklyn Wildlife Sanctuary. A positive relationship exists between EQR and Australian Wildlife Conservancy. There are no anticipated issues with the landholder in relation to the project. • The project does not involve any new significant infrastructure, and changes to the current mining methods or other activities that could otherwise have a negative impact on the local community and stakeholders.
Other	• *To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* • *Any identified material naturally occurring risks.*	• The operation is contained within two mining leases: ML4867 & ML4919. • The land relevant to the project site is used for quarry operations and mining activities as per the respective

Criteria	JORC Code explanation	Commentary
	• *The status of material legal agreements and marketing arrangements.* • *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	licenses - EA EPPR00438313 for the quarry and EA EPML00956913 for the mine. • All environmental, surface access and operating licenses have been acquired to allow for between 100,000 and 1,000,000 tonnes to be mined, crushed and screened per annum. • Processing through the existing proposed plant is approved up to 100,000 tonnes per annum. • An amendment to the current EA will be lodged which will allow for up to 500,000 tonnes per annum to be processed through the processing plants. A pre-lodgement meeting has been completed, however no specific feedback has been received from DES. • The required water and solids circuit will remain within the existing disturbance areas accounted for in the ERC for the project, therefore it is unlikely that additional amendments to the EA for mining activities will be required. • Given that the EA amendment relates to a change in annual throughput with no material changes to the mining method or operational methodology, the competent persons believe that there are reasonable grounds for the required EA amendment to be approved.
Classification	• *The basis for the classification of the Ore Reserves into varying confidence categories.* • *Whether the result appropriately reflects the Competent Person's view of the deposit.* • *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	• All Reserves have been classified as Probable as the Resources have been fully categorized as Indicated. There are no Measured Resources.
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	• The Reserve assumptions, calculations and financial modelling has been internally reviewed by a team of experts. • No external audits of the estimate have been completed.

Criteria	**JORC Code explanation**	**Commentary**
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• The estimate of the Reserves at Mt Carbine has been derived from local assumptions based on historical and current performance indices at the site. • The cost of operating the open pit has been calculated from historical contractor rates at the site, adjusted for larger equipment and longer work hours.

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