

**Independent review of Underground Mining at the Dolphin Mine on
King Island : Revision 1**

Prepared for: King Island Scheelite Project

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Executive Summary

The King Island Scheelite – Dolphin Project has been developed in recent years as an Open Cut project and this is reflected in the Geological Digital Resource model that has been prepared.

In order to assess an underground mining option for the Dolphin Mine a Digital Resource model which discriminates between previously recognised individual ore horizons is required from which a revised Mining Inventory for the underground project option may be derived.

The MRO (mining reserve optimiser) utilised by AMC is an effective tool for assisting with mining inventory estimation, some additional control and removal of post pillars, skin pillars, isolated ore units and ore sterilised by ground conditions will be required at the next level of study.

The old main decline contained numerous steel arch sets which were extensively timber packed above the sets and back failure has previously developed in this zone causing eventual set failure. Rehabilitation of the decline by removal or repair of the sets is not viable but the option of placing a 100mm sprayed (fibre reinforced concrete) arch inside the sets may present a viable means of re-entry. Rehabilitation using shotcrete is therefore proposed as the preferred means of mine access whilst the option of development of a new decline from -150m RL may be considered a fall back option to mitigate any risk associated with the rehabilitation option.

As there is some uncertainty at this stage regarding the actual underground Mining Inventory it is not possible to comment with any certainty upon the viability of a potential underground operation except to comment that sufficient Mining Inventory to deliver 1.5-2.0Mt of ore at 1.3% WO₃ would, in my opinion, be required to present a project that was sufficiently robust to withstand both the risk associated with remnant mining and the sensitivity of the project to factors such as price, mill recovery and head grade.

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1. Scope of work

The intent of this report was to briefly review previous reports by AMC and Coffey and provide an opinion regarding the re-opening of the Dolphin Mine as an underground ore source.

2. Introduction

This report presents a brief technical review of the mining and geotechnical aspects of the proposed underground operation at the Dolphin Mine on King Island, Tasmania. In addition to a technical review this report is intended to highlight any risk associated with the above options and the potential impact on cost and project value.

The King Island Dolphin Mine ceased operation in 1990 due to a failure to market and sell the concentrate produced from the operation and the high costs associated with the “Artificial Scheelite Plant” – over one year of concentrate production was held in storage at the Power Plant warehouse shed at the time of closure.

Prior to closure the underground operation was producing ore from the Lower Wedge, Mid Wedge, Lower Pit and Southern Swan orebodies solely by post pillar cut and fill methods.

Shortly before closure the lower main decline collapsed at -250m RL and a by-pass had commenced to regain access to the lower workings.

The mine workings have now been flooded with fresh water for up to 20 years.

AMC have produced two reports, of a scoping level of accuracy, to examine the option of a stand alone underground mining operation and have shown that under certain physical and financial circumstances it might be possible to achieve a viable underground operation.

As underground manager at the time of closure, and having worked closely with the operation between 1980 & 1982 and 1984 & 1991, the author of this report will attempt to add to AMC’s findings in the light of additional personal operational experience of the mine.

3. Geology

The Dolphin mine orebody is a skarn orebody – in simple terms - created when a granite igneous intrusion altered the neighbouring host rocks by contact metamorphism. Typically skarns appear as haloes in the country rock reflecting the outline of the intrusion. In the case of the Dolphin orebody subsequent faulting has disrupted and dislocated the skarn deposit. The three main elements of the orebody the pit, wedge and central orebodies represent faulted sections of the original skarn lying between a series of well defined fault zones. The fault zones at Dolphin Mine are associated with poor to extremely poor ground conditions

The economic component of the Dolphin Mine orebodies may be said to be contained within the C Lens horizon. C Lens has historically been subdivided into distinct units or domains as follows: -

- Lower C banded hornfels group associated with low to medium scheelite grades.
- Marble Marker – an unmineralised geological marker between Upper and Lower C.
- Upper C massive garnet hornfels associated with medium to high scheelite grades.
- PGH pyroxene garnet hornfels is a distinct lighter coloured very hard unit containing generally low background levels of scheelite but is occasionally mineralised with high scheelite grades in a nuggetty manner

The Upper C lens formed the principal mining target with the Lower C unit also being mined where the grade was economic. The PGH was not frequently mined because of grade and metallurgical factors.

The B lens unit also may be found throughout the Dolphin Mine. In the past the B lens was not mined as a specific target because of inconsistent grade and metallurgical properties.

The geological units and the major faults at the mine with the exception of the Grassy River Fault which was remote from underground operations may be noted in the following Fig 1.



Figure 1: Geological Features and Faults at -100m Level

4. Ore resources and Mining Inventories

Specific JORC compliant underground mining ore resources and ore reserves have not been proposed at this stage for the Dolphin Mine and none are presented or implied in this report.

As a reference the 1990 reserve statements by King Island Scheelite are used in this report, as are mining inventories estimated by AMC and by Polberro Consulting (PC). All such data should be considered in this report as a *Mining Inventory* that is “An unclassified diluted tonnage and grade estimate that can be expected to be recovered from a stope, region of or whole of a mine that may be utilised for initial conceptual study subject to the understanding that it does not represent a reserve”.

AMC and PC mining inventories have been estimated utilising the digital resource model *ModFin.dm* produced by AMC principally as a means of estimating open cut inventories.

The model *ModFin.dm* treats the whole of the C Lens including the pgh as a single unit which is appropriate for an open cut examination but, in my opinion, needs to be controlled within distinct domains for an underground option where mining will need to be more discriminatory to improve the opportunity of developing an economic project.

Previously the PGH unit was not often mined because the mineralogical properties were such that the scheelite grade was sporadic with a low background grade and the unit was metallurgically difficult to treat. The Upper C garnet hornfels unit contained homogeneous evenly distributed higher scheelite grade material whilst the Lower C banded hornfels was of a lower grade with a more varied grade distribution.

It is my opinion that any digital resource model prepared for an underground study of the Dolphin orebody should constrain the PGH, Upper C and Lower C units within distinct domains which have no influence on adjacent domains.

The following Mining Inventory estimates were derived utilising a 0.7% WO₃ cog: -

Table 1 - Mining Inventories for Post Pillar Mining Underground at Dolphin Mine

	Source	Date	Tonnes(M)	WO3%	Mtu's(M)
1	North Ltd*	June 1990	1.03	1.13	1.16
2	AMC 109034	July 2009	1.96	1.1	2.15
3	AMC 109085	Nov 2009	2.15	1.05	2.26
4	PC*	Feb 2010	2.46	0.9	2.06
4a	PC*	Mar 2010	1.04	1.1	1.16

*- Dilution & Recovery factors applied

In the above table estimate 1 was based on an existing polygonal sectional resource whilst estimates 2, 3 & 4 were derived from AMC digital resource model ModFin.dm.

Differences in estimates 2 & 3 against estimate 4 are explained as follows: -

- Estimates 2 & 3 incorporate post pillar and skin pillar resource in/or adjacent to filled stopes which can not be recovered by an underground mining operation (see Figure 2). Estimates 2 & 3 also includes satellite ore pods and small ore cells that are unlikely to be economic to extract. (See Appendix 3 – Isometric Plot of AMC109034 mining inventory)
- Estimate 4 was produced by delineating areas that have a realistic prospect of recovery based upon the author's operating experience at the Dolphin Mine and is therefore significantly different to the global scoping approach used in Estimates 2 and 3. Some high grade sections of the Lower Wedge orebody – in particular – were excluded because of extremely poor ground conditions.
- Estimate 4 includes lower grade material that it may reasonably be expected will need to be mined to recover higher grade cells in the resource model.
- Estimate 4a was generated artificially by removing cells within the defined mining shapes below a 0.7% cut off for comparison purposes only.

Estimates 4 and 4a were developed purely as a check on Estimates 2 & 3 using the same digital resource model and do not represent a proposed mining inventory. A proposed mining inventory may be derived from a revised block model and taking into account the following considerations.

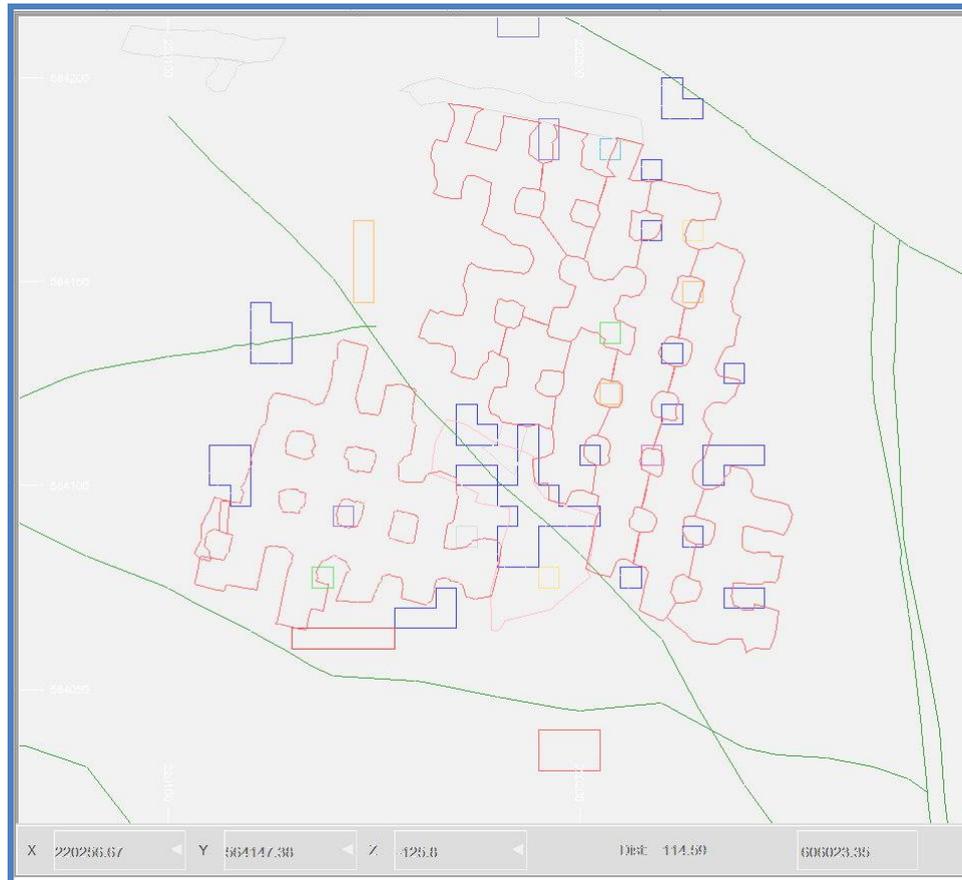


Figure 2: Estimate (MRO) incorporating post pillar and skin pillar resources in filled stopes

In addition to providing a more discriminating digital resource model an in depth examination of the mining methods and applied recovery factors will need to be performed. The factors considered in making this statement are as follows: -

- A review of the remaining ore and the archived plans of the old workings indicate that most of the remaining stopes have a limited number of horizontal post pillar lifts prior to approaching open development or filled stopes above them.
- Transverse bench stoping of 10m to 14m vertical height should be viable in most areas of the mine. Bench stopes would be transverse and recover the central 8-9m of each 14m panel allowing waste or sand filling to be used. Recovery for this type of bench stoping would be 64%.
- Stopes approaching a filled undercut should be mined to within 10m of the undercut and then transverse uphole stopes recovered to within 2m of the old floor. Recovery for this type of benching would be 51%.

The mining method will therefore be a mixture of post pillar (R=82%), transverse bench stopes (R=64%) and uphole retreat transverse stopes (R=51%) filled with waste rock or low cost hydraulic classified mine tailings (or other).

Dilution has been factored at 10% in the PC and old North mining inventory estimates at zero grade. In fact many of the stope boundaries lie either within the high grade lens or bordering medium grade material thus the dilution is either HG or MG ore and only on a limited occasion is it actually waste. This should be addressed at the next reserve calculation to avoid underestimating the grade of the reserves.

Reconciliation

The following excerpts from the 1990 Ore Reserve report give some insight to ore grades at the time of closure.

3. RECONCILIATION

3a. Upper C Lens

As predicted in the 1989 Reserve Statement the Southern Swan Geological Resource was underestimated with an Extraction Indicated Resource grade of 1.696% compared to a Resource estimate of 1.36% W03.

Similarly Lower Wedge Geological Resource grades were underestimated with an Extraction Indicated Resource grade of 1.939% in the Lower Wedge 220ml stope compared to a Geological Resource of 1.52% W03, whilst the Extraction Indicated Resource in the Lower Wedge 260ml was 2.17% W03 compared to a Geological Resource of 1.49% W03.

In other areas the reconciliations were more realistic with the exception of the Lower Pit which appears to have been overestimated in the Geological Resource Statement of July 1990, however, the occurrence of considerable low grade dilution continues to make Lower Pit Reserve grades difficult to predict.

Second Part follows.....

Figure 1

LOCATION	ACTUAL EXTRACTED GRADE	PREDICTED RESERVE GRADE	HIGH/ LOW
Upper Pit	1.206	1.13	- 6.3%
Lower Pit	0.897	1.34	+49.4%
Mid Wedge	1.503	1.37	-11.1%
Lower Wedge 220	1.763	1.50	-14.9%
Lower Wedge 260	1.974	1.43	-27.6%
Lower Central	0.997	1.04	+ 4.3%
Southern Swan	1.542	1.29	-16.3%

3b. Lower C Lens

To date the predictions for the Lower C Lens have proved an unreliable estimate for the planning of extracted grades as follows:-

Figure 2

LOCATION	ACTUAL EXTRACTED GRADE	PREDICTED RESERVE GRADE	HIGH/ LOW
Lower Pit	0.717	1.19	+66%
Mid Wedge	0.846	1.10	+30%
Lower Wedge 220	0.925	1.16	+25%
Lower Wedge 260	1.201	1.08	-10.1%
Pit Dag	0.9	1.10	+15.4%

With the exception of the Lower Wedge 260ml all grades appear to be overestimated by between 15 and 66%. The major reason being that a 1% cut-off is unrealistic as a tool for defining an extractable resource for the Lower C Lens due to the wide variations in grade within a small area which occur within the Lower C Lens, this in turn makes it impossible to give a realistic appraisal of potential dilution levels. The 0.7% W03 Cut-Off Resource is a far more practical tool, for defining the Lower C Lens, as this cut-off envelopes a much higher proportion of the total resource which would have to be extracted in a practical mining operation.

5. Geotechnical Report

No historical evidence was available in a format regarding domains, rock mass quality and other geotechnical properties of the rock, rock mass and structure at the Dolphin Mine other than colloquial evidence. In general terms the following may be stated - based on operational experience - for the relevant areas. The following units are ordered from HW to FW: -

Pyroxene Garnet Hornfels – generally formed the hangingwall of all orebodies except where not present due to faulting. Good competent rock type generally hard to very hard.

Garnet Hornfels – main Upper C ore horizon generally very competent massive rock type except in Lower Wedge and to a lesser extent the Mid Wedge where an increase in structural discontinuities can lead to formation of discrete blocks 1.5-2.0m in size. This unit is exceptionally good in the Central, Pit and Swan orebodies due to the reduced level of discontinuities in these units.

Banded Hornfels – the Lower C ore horizon is generally weaker and less competent than the Upper C horizon with foliation like discontinuities in addition to random discontinuities. This unit was particularly of low rock mass quality and presented poor ground conditions in the Lower and Mid Wedge orebodies.

Banded Footwall Beds – More strongly foliated and thus of lower quality than the banded hornfels typically presenting poor to very poor ground conditions.

Lower (meta) Volcanics – Generally poor to extremely poor conditions may be encountered within this unit consisting of variably weathered volcanic material ranging from a foliated volcanic rock mass with chloritic coatings to completely broken chlorite and graphite coated rubble.

Biotite Pyroxene Hornfels and Quartzite units form the basement above the granite intrusion. The biotite pyroxene hornfels tended to be shale like in appearance and performance whilst only limited exposure of the quartzite (unknown) and granite (pink-fresh- strong-unjointed rock mass) units were made.

A significant component of the old main decline and access development is sited in the weakest of the footwall units (bfb, bh and lower volcanics).

Coffey have produced a Geotechnical Investigation Report for the Open Cut Dolphin Project MINENHIL00237AB. This report contains the best available quantified descriptions of rock types, rock mass data and fault structures.

The geotechnical domains defined in the report are suitable for open cut activity but they are too broad to use for an accurate underground assessment.

The Coffey report⁵ defines the major Dolphin Mine faults but gives little indication of actual conditions. In practice the Northern Boundary Fault, Wedge Fault and Decline Fault were the most difficult of the faults to deal with typically containing zones of greasy (chlorite, graphite) coated rubble and reduced stability within the vicinity of the fault zone.

The Dolphin Mine was relatively dry with a total water pumped averaging 23 litres per second of which 14 litres per second was natural inflow and the remainder water from drilling and filling operations. This contradicts the understanding that may have been given in the Coffey report⁵ that the mine was wet (p45) and the faults were by inference interconnected.

In-situ stress measurements at the mine indicated a generally low level of in-situ stress as follows: -

Table 2: In-situ stress field at -75m RL

Level	Maximum Mpa	Intermediate Mpa	Minimum Mpa
-75m RL	4.5 (N-S)	3.1(E-W)	2.0(Vert)

Results from the many tests (>100) conducted at the mine were highly variable, the tests included numerous hollow inclusion and USBM biaxial cell tests and were conducted to derive the summary stress field. A copy of some of the test results is shown in Appendix 2. Actual strength test results for intact rock were generally high. As a result of the high strength, the low in-situ stress levels and the highly variable rate of discontinuities it is discontinuities that play the dominating controlling influence on ground conditions at the mine.

Table 3: UCS Strength (Miller 1980)

Rock Type	UCS Mpa
Pyroxene Garnet Hornfels	103.6
Pyroxene Garnet Hornfels	195.5
Garnet Hornfels	122.6
Garnet Hornfels	96.6
Marble Marker	161.9

6. Dolphin Mine Access

6.1. Historical Development

The operating mine was accessed via a main decline with ventilation/fill reticulation development levels at the 75m Level, old 100m Decline and 150m Level. All mine ventilation was provided by twin Richardson 1615CY ventilation fans drawing 120m³/sec through two 1.8m diameter ventilation raises between the surface and the 75m Level. The ventilation return circuit linked 150 Level (via the 100 level) to the 75 level. All operating stopes were linked to the 150m level by 1.8m diameter raises with flow rates controlled by brattices on the 150 level.

Much of the main decline between -150m RL and -260m RL was supported by steel arch sets.

The decline sets have failed previously (and been repaired) at the 150m Level, at the former entrance to the 150m Level, and below the turn off to the Lower Pit workings at -190m RL. Both repair operations were lengthy, potentially hazardous and costly.



Figure 3 Main Decline Set Repairs at 150 Lev and -190m RL

A major decline failure occurred at the -250m RL T14 shortly before the mine closure. This failure was massive and unrepairable necessitating the development of a decline bypass – the original decline was located too close to the orebody in the footwall (Lower C lens) and its replacement would have been required to effectively recover all of the Lower Wedge orebody.

Shortly after the T14 decline failure damage was observed in the steel arch sets at the Main Decline/Q11 development junction at -235m RL. A combination of gradual twisting of the steel sets and damage to the timberwork was noted – it should be assumed that some set failure has occurred at this site.



Figure 4: Decline Failure at T14 -250m RL. Damage developing at -235m RL Q11

A main decline by-pass was planned (Southern Decline) and started from both ends several years before the mine closed to replace the section that eventually failed at -255m RL. The lower section of this by pass suffered a major sidewall failure during development and was abandoned.

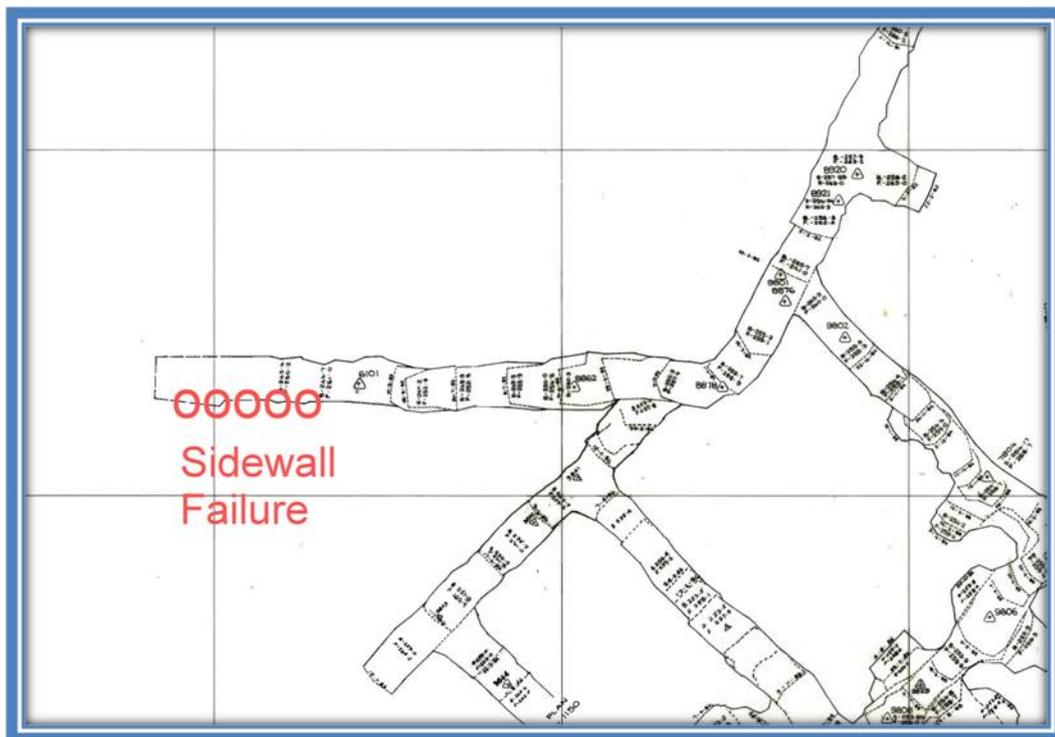


Figure 5: Sidewall failure (on occluded structure) in South Decline

The probable cause of the failure was an unexpected intersection with the Swan Fault - some metres north of its anticipated position which was intersected just before the decline was due to enter granite.

6.2. AMC Scoping Studies

AMC considered a number of scoping options for mine re-entry including a completely new decline and rehabilitation of the steel setted sections of the decline.

A new decline is a consideration that is a realistic alternative given the amount of damage and failure that may have occurred in the setted sections of the main decline. The principal difficulties with the option are as follows: -

- The decline would need to be remote from the ore horizon in order to pass through undisturbed ground because of the difficulty of avoiding existing development.
- Development access from a new decline to the ore horizons would be extensive and would need to thread between existing access development and the old decline in order to access the remaining ore.
- Development of a new decline makes dewatering of the existing workings more difficult and increases the potential hazard because the decline development would be remote from the flooded workings.

The option of dismantling and rehabilitating the steel sets or backs above them would, in my opinion, be too hazardous and costly and should not be considered. The presence of unravelling type ground above the steel sets and the poor quality of timber packing of the voids between the sets and the old backs that was carried out has probably resulted in migration of the backs above the sets to between 2 and 3 times the original decline height in places.

6.3. PC

For the purposes of this brief review and to provide realistic options for re-entry PC consider the following to be realistic alternatives.

1. A full rehabilitation option. This option involves the following: -
 - Rehabilitate all unsetted declines with 75mm shotcrete.
 - Cover the interior of all decline sets with a 100mm shotcrete shell
 - Mine a decline bypass from -220m RL to -260m RL.
2. Develop a shaft from -75m RL to the -275m RL of 200m depth and (approx) 4.5m diameter with the shaft development linked to existing development where practicable.
3. Develop a decline from -135m RL to -250m RL replacing all existing setted declines and some orebody access development. The unsetted decline from Surface to -150m RL is utilised.

For the purpose of this review all options have been reviewed to a scoping level only and are ranked as follows: -

1. Rehabilitate all existing development and develop a Bypass decline to access below the -240m RL..
2. Develop a new decline from -135m RL to -250m RL – rehabilitate and utilise the existing unsetted decline from surface to -150m RL.
3. Develop a shaft from -75m RL to -275m RL

The major capital associated with each option is shown in Table 5. Costs shown include re-access development and rehabilitation required for each case plus an allowance to establish an escape route. A typical program of development and dewatering works for the rehabilitation option is shown in Table 4.

The rehabilitation option has the lowest capital cost but carries the highest risk. This risk of the rehabilitation option is mitigated by the second option of a new decline which can be conducted as a fall back option if the existing decline below -150m RL is unrecoverable.

The shaft option presents a high cost without any further reduction in risk and is excluded.

Table 4 Staged Rehabilitation and dewatering Program

Rehabilitation/Development	ID	Units	Total Drive	Settled Drive	Standard Support	Stage	Purpose
Dewater to -80m RL	D1					0	Dewater development
Portal	D1	m	10	7	3	1	Access
Portal to I15	D1	m	100	0	100	1	Access
I15 to N12	D2	m	150	15	135	1	Access
75 LEV to Surface Vent Rise (2)	D3	m	240	0	240	1	Ventilation & Dewatering
I15 to G16	D1	m	200	0	200	1	Access
Refurbish E13 Manway	E1	m	17	0	17	1	Escapeway
Recover rest of 75m LEVEL	D3	m	290	30	260	1	Ventilation & Dewatering
Dewater stopes to -80m RL						1	Dewater stopes
Dewater to - 100m RL	D1					1	Dewater development
Old Decline N12 to R13	D2	m	120	0	120	2	Ventilation & Dewatering
Decline G16 to L17	D1	m	90	17	73	2	Access
Old Decline R13 to M17	D2	m	165	20	145	2	Ventilation & Dewatering
Re-establish Vent 100 to 75	V1	m				2	Ventilation & Dewatering
Break into K17 Rise	A1	m	10	0	10	2	Escapeway refurbishment and ventilation
Dewater stopes to -100m RL						2	Dewater stopes
Dewater to - 120m RL	D1					2	Dewater development
L17 to J19	D1	m	175	15	160	3	Access
Access L17, J16 & H16	D4	m	50	0	50	3	Old accesses to assist with stope dewatering
Dewater stopes to -120m RL						3	Dewater check on stopes
Dewater to - 140m RL	D1	m				3	Dewater development
J19 to N17	D1	m	150	50	100	4	Access
Break into K17 Rise from J19	A1	m	35	0	35	2	Escapeway refurbishment and ventilation
Dewater stopes to -140m RL						4	Dewater stopes
Dewater to - 160m RL	D1					4	Dewater development
N17 to J14	D1	m	150	135	15	5	Access
150 LEV	D5	m	800	20	780	5	Ventilation , production, dewatering
Escapeway and Pump Raises	D6	m	220	40	180	5	Escapeway, dewatering ventilation
Dewater stopes to -160m RL						5	Dewater stopes
Dewater to - 180m RL	D1	m				5	Dewater development
J14 to N13 Junction	D1	m	200	200	0	6	Access
I11 to N11	D7	m	220	0	220	6	Production Pit, Central, Pit Dag
Dewater stopes to -180m RL						6	Dewater stopes
Dewater to - 200m RL	D1	m				6	Dewater development
N13 to Q13S	D1	m	90	60	30	7	Access
N13 to M8	D8	m	200	0	200	7	Dewatering
193 Q13 to Q16	D4	m	80	80	0	7	Dewatering
Q16 to R15	D4	m	140	0	140	7	Dewatering & FW access
Dewater stopes to -200m RL						7	Dewater stopes
Dewater to - 220m RL	D1					7	Dewater development
Q13S to S12 T/Off	D1	m	140	50	90	8	Access
L.Wedge 200 Level	D4	m	210	30	180	8	Stope Access/Stoping Level/Dewatering M.Wedge
200R13	E2	m	90	0	90	8	Escapeway and pump station
Refurbish escapeway to 150L	E2	m	60	0	60	8	Maintain escape route to surface
Refurbish escapeway 215-200	E2	m	20	0	20	8	Maintain escape route to surface
M8 to N13	D8	m	100	0	100	8	Natural Sump for Pit Orebody/Dewatering stopes
M8 to Pit Dag/Dodo	D4	m	50	0	50	8	Old open stope/stoping/dewatering
220 L Wedge Access	D4	m	140	20	120	8	Stope Access/Dewatering
Dewater stopes to -220m RL						8	Dewater stopes
Dewater to - 240m RL	D1					8	Dewater development
S12 T/Off - 240 LEV	D1	m	140	126	14	9	Access & Production
L.Wedge 220 LEV Lift 2	D4	m	110	0	110	9	Stope Access & Production
Q11 Development	D4	m	90	20	70	9	Dewatering old stopes (Pit Central)
Develop S12 - Q10 By-pass	D9	m	300	0	300	9	New development
Dewater stopes to -240m RL						9	Dewater stopes
Dewater to - 270m RL	D1					9	Dewater development
240 LEV - T11	D1	m	30	10	20	10	Access & establish sump
L.Wedge 240 LEV	D4	m	200		200	10	Recover stoping horizon
Refurbish escapeway 250-215	E2	m	35	0	35	10	Maintain escape route to surface
Refurbish escapeway 250-265	E2	m	20	0	20	10	Maintain escape route to surface
Finish bypass	D9	m				10	
Dewater to 280	D1					10	Dewater development
Dewater stopes to 260						10	Dewater stopes
Southern Decline	D1	m	110	0	110	11	Access & dewatering
P9 Southern orebody access	D4	m	55	0	55	11	Stoping
Swan Stope	D4	m	100	0	100	11	Stoping
Mid Central Q9	D4	m	15	0	15	11	Dewatering Stope
Decline up to R14 Escapeway	D1	m	60	0	60	11	Escapeway completed
Decline Down to S11 Sump	D1	m	45	0	45	11	Dewatering Stopes
Dewater to -284m RL	D1	m				11	Access & dewatering
Dewater Stopes to -270						11	Dewatering Stopes
L.Wedge 260 LEV	D4	m	120	20	100	12	Production

The rehabilitation requires that a total of more than 5.5km of development including declines, access, levels, raises, escape ways and stope development needs to be recovered of which approximately 1.0km contains steel arch set support. As a conservative estimate the rehabilitation option will cost of the order of \$10.7m as indicated in Table 5. (No contingency shown as these costs are for comparison of options only).

Table 5 Access Options Cost Estimate

Option 1: Rehabilitate decline	Total	Rate	Cost
Development	300	4,000	1,200,000
Rehab Setted areas	965	2,250	2,171,250
Rehab Development	4,680	1,500	7,020,000
Refurbish escapeway	152	1,500	228,000
Air leg Development	45	2,500	112,500
			10,731,750
Option 2: Replace lower decline	Total	Rate	Cost
New Decline 150m to -260m RL	1,100	4,000	4,400,000
Development re-access costs	350	4,000	1,400,000
Rehabilitate Development	2,700	1,500	4,050,000
Rehab Stopes & Access	1,190	1,500	1,785,000
Re-establish escapeway	1	500,000	500,000
			12,135,000
Option 3: Shaft Access	Total	Rate	Cost
Shaft Sink and Line	200	20000	4,000,000
Off Shaft Development	222.5	4000	890,000
Equipping and Installation	1	9000000	9,000,000
Workshop	1	920000	920,000
Rehab Stopes & Access	1,200	1500	1,800,000
Development	720	4000	2,880,000
Escapeway Costs	1	500,000	500,000
			19,990,000

7. Mining Methodology

7.1. Historical

Post Pillar Cut and Fill

At Dolphin Mine the principal mining method in use was post pillar cut and fill. The post pillar mining method utilised 8m wide x 4-5m high drives and cross cuts with 6m square pillars on a chequerboard pattern i.e. a 14m centre grid. The pillars were fixed in position and exposed by each successive lift. The horizontal cut at the time of closure was 4m for each successive “lift”.

The fill used was coarse classified mine tailings which could be walked upon during pouring and worked upon within forty eight hours. In addition to natural drainage through the fill and decanting from the top of the fill drainage towers were maintained and built up with each stope lift. Drainage towers were picked up by survey and re-accessed at the end of each mining cycle.

Stope backs were supported by twin strand cable support on a 3.5m square pattern with cables installed up to 20m in length providing support for successive lifts when the cables were reduced to 4-5m in length the next 20m set was installed to ensure coverage was maintained.

Undercut or sill development was a full face development initial undercut lift – all subsequent “lifts” were taken as horizontal slices mined using a development jumbo with a cut consisting of 3-4 rows of holes using the gap between the fill and the previous lift backs as the free face rather than using a development cut.

In addition to cable support for the control of large slabs and wedges primary support consisting of cement grouted rebar and when appropriate rolled steel weldmesh was used.

An aspect of the post pillar method was the utilisation of 14m x 14m Regional Pillars to carry Regional Hangingwall load on the understanding that post pillars only carried the load of the immediate backs and prevented back failure. There was never any evidence that Regional Pillars conducted significant load and almost certainly they performed in a similar manner to the post pillars exercising control over the immediate backs in the vicinity only.

Sill Benching

In a number of instances undercut sills were developed ahead of the cut and fill production face of active post pillar stopes by the former operators of the mine in order to support production schedules that may have demanded higher grade material than was otherwise available. Once the cut and fill production face advanced to within 6m of the (open) upper sill then the remaining material was

benched using 64mm production upholes. This method was successfully utilised in the Pit and Swan stopes where the ground conditions consisted of massive, competent garnet hornfels and pyroxene garnet hornfels. A sill failure however was recorded, at firing time, in the poorer quality ground conditions in the Mid Wedge stope in which a 40m length of 6m x 8m bench dropped to the lower level.

Open Stoping

Open stoping was trialled in the Lower Wedge in the early 1980's (1982-4). The first trial stope W62 was mined successfully but left open for a long period (>6months) before its ultimate collapse. The stope collapse significantly destabilised ground remaining near to the stope in W60 and W64 panels with clear development of open structures and movement of blocks toward the failed opening. The failed stope was eventually filled and the upper failure chimney tracked to the -150m level. The W62 stope was 14m wide and was mined from 260-200 level in the W62 panel, the stope would have been successful if mined and filled within a time frame of 1-2 months as would normally be anticipated.

A second open stope was later mined in the Pit Dag/Dodo lens – this stope was quickly mined and filled with no associated problems and minimal dilution.

No other mining methods were utilised at the Dolphin Mine.

7.2. AMC

AMC have determined that post pillar mining represents the most financially viable approach in the underground scoping studies^{1 2}. The AMC selection of an underground stoping methodology is, in my opinion, appropriate.

7.3. PC

Examination of ore in the AMC digital resource block model indicates that the majority of the remaining ore is contained within the Wedge orebody in the Mid Wedge and Lower Wedge.

Detailed examination of these orebodies indicates that the ore is broken into sub-stopes within the Mid and Lower Wedge as follows: -

Wedge Remnant 280 stope - The original Wedge Remnant 280 stope was abandoned on lift 02 because of severe ground control issues. Any remnant ore can not be accessed until the Lower Wedge 260 stope is completed.

Lower Wedge 260 stope – The Lower Wedge 260 stope consists of the current active post pillar stope lift (2) above -255m level, the 240m Lower Wedge level development and 240 floor ramp. This stope will need to be mined as a post pillar until the resulting backs come within 10m (minimum) of any open or filled stoping

level above. If the upper development is open then the remaining ore between the levels may be mined as simple down hole vertical benches 9m wide (with 5m rib pillars) running transverse to strike along the original stope crosscuts such that each bench can be mined and filled individually. If the development above is filled as will be the case when the Lower Wedge 220m Level is approached then a retreat uphole bench may be mined, and preferably back filled, under a 2-3m back crown pillar. The uphole benching directly beneath Lower Wedge 220 level should be conducted only when the Lower Wedge 220 stope is completed.

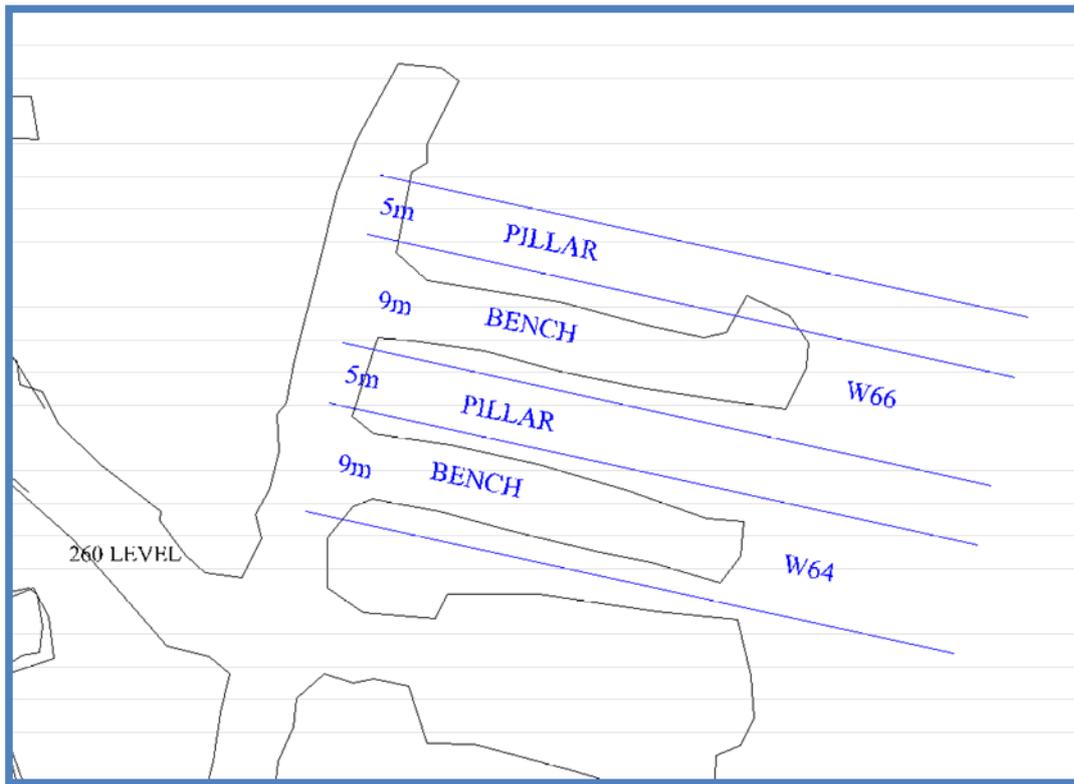


Figure 6: Benching Option when approaching levels above

Lower Wedge 220 stope – The Lower Wedge 220 stope consists of ore between the Lower Wedge 220 post pillar current lift, the Lower Wedge 200 Development and ore remaining above the Lower Wedge 200 level. There is a rib pillar above the 200 level (on 41 pillar panel) separating the stope from the main part of the Mid Wedge stope but there were encroachments beyond the rib pillar boundary where high grade ore was mined and a concrete crown pillar left in the vicinity of W60 to W64. The result is that this stope will also be a mix of post-pillar and transverse benching.

Mid Wedge Stope – The Mid Wedge stope is a quite complex mix of sills, post pillar faces, rib pillars and footwall & hangingwall remnants. If the project proceeds to a full feasibility the exact process (mix of post pillar and bench stoping) should be defined so that an accurate mine inventory and risk analysis can be performed – most particularly for the Mid Wedge.

The material from the Mill is currently relatively coarse with between 15-20% of the material only being in the -325 mesh size range. The mill tailings are currently pumped to the tailings classification plant for a single pass through a cyclone with the underflow reducing the -325 mesh component to the 5-8% range. The overflow is returned to the slimes dam. The grade of the slimes portion is higher than the coarser classified fill material (0.3 to 0.5% W03 compared to 0.17 to 0.20% W03) and thus the slimes dam could represent a resource at some future time, whilst the fill will not. The underflow from the cyclone is dropped onto a stockpile which is allowed to drain. When fill is required underground it is collected with a Cat 988 Front End Loader and tipped into a hopper which feeds onto a belt which runs the material through water sprays, which re-pulp the fill to around 50% solids. The mixture is then fed underground, using gravity only, through 110mm poly pipe (Class 12 for vertical runs, Class 6 for horizontal runs) to the eventual destination where the fill is run into the stope and allowed to drain.

Flygt pumps are used to collect excess run off and decanted water in the stope, in addition drainage is facilitated using drainage towers which are built up through the stope and act as a 'continuous drainage medium. The towers are built out of 150mm square timbers 1.5 metre long built into a crib tower. The gaps between the timbers are covered with permeable plastic cloth held in place by cyclone mesh. At the bottom of the tower will be a 165mm diameter drain hole to allow passage of drainage water.

Provided properly classified material is used, machinery can operate on the surface within 48 hours of filling.”

The type of fill arrangement that will be suitable for a new operation will depend upon the fill size range of the proposed mill tailings – if practicable a cheap to build and run coarse classified tailings option should not be overlooked.

8. Mining Project Schedules

8.1. AMC

AMC have presented development and production schedules as annualised production and development metres to support the scoping studies at scoping study order of accuracy. That is appropriate for the level of accuracy of the study.

8.2. PC

The review by PC also utilises simple scoping study annualised production rates rather than a specific production schedule.

Detailed development, production and fill schedules should be produced for the full feasibility stage as there are likely to be variations in grade and availability of ore/fill that will need to be reflected in the scheduling. The impact of delays to stages of the re-access program should also be modelled.

9. Financial Evaluation

9.1. Operating

Operating costs utilised in the AMC scoping studies are generic scoping study costs and as such are appropriate.

9.2. Capital Costs

Capital costs utilised in the AMC scoping studies were based upon the various re-entry options and appeared to be appropriate.

Both operating and capital costs are appropriate for a generic mine at scoping level however there may be significant variations from the norm in the case of Dolphin mine re-entry, rehabilitation, development and production and these variations need to be closely examined at the feasibility study stage.

9.3. Modelling

Both AMC and KIS have modelled the financial forecast of a scoping level option.

From AMC forecast data in the scoping studies a set of models was reconstructed by PC for the underground stand alone option.

At the same time a financial model prepared by KIS was examined.

At this stage the total capital costs, total mining inventory, annual feed tonnage and grade and mill recovery are not defined to a stage where the modelling outcome is reliable but the models all clearly indicate a high degree of sensitivity in the following areas: -

- Head Grade
- Mill Recovery
- Price
- Total Mining Inventory

It was noted when manipulating the above models that a Mining Inventory of +1.5Mt at 1.3% WO₃ produced a very robust project whilst any fall in grade below 1.0% produced a marginal project.

It was also noted mill recoveries were modelled at between 85% and 90% whereas historically mill recovery ranged from 72% to a target of 82%. From financial modelling it is clear that the mill recovery will also be a major factor governing the success of any project.

10. Risk

Risk factors that should be dealt with at or before the full feasibility stage which lie within the scope of this report may be defined as follows: -

Mining Inventory – There is a risk of overvaluing/undervaluing the project. The current mining inventories, in my opinion, are limited in their application because the digital block model *ModFin.dm* does not define the separate ore domains (pgh, gh, bh). In addition the mining inventory should be based upon areas with a realistic chance of exploitation and not include post pillars, skin pillars, isolated cells & pods or mineralisation sterilised by poor ground conditions.

Mine Re-access – The risk associated with mine re-access is the introduction of cost, delay and operator hazard by the selection of an inappropriate course of re-entry. The lowest risk from a development perspective would be a shaft from the surface however the cost and power supply appear to be prohibitive. A shaft option has a higher potential risk in the dewatering area, as does the new decline option, because the development of both is remote from the existing old workings. The principal risk with the rehabilitation option is that the existing development may be in a poorer condition than anticipated - this risk can be quantified as \$1.4m which is the additional cost of the alternative decline option (-135m to -250m RL).

Dewatering/Inundation – There is a potential risk of inundation of workings if the dewatering of development and stoping areas is not conducted in a controlled manner. The rehabilitation option presents the most controlled dewatering options as the lowering of the water levels can be controlled as the rehabilitation process advances. The shaft and new decline options would present a higher risk of perched water remaining until a flow path was presented, potentially at a high head on say a development breakthrough to undrained old workings. Extensive use of probe drilling and monitoring of the water levels in the old development would be required.

Stability of old stopes – The stoping areas in the mining inventory are all existing workings which in many cases contain both an active face position and advance sill development. Any significant deterioration that has taken place during the period of flooding has the potential to reduce mining inventory.

Product Demand and Price – The project is very sensitive to price variation which is outside of the scope of this review.

Mill recovery - The project is very sensitive to variation in mill recovery which is outside of the scope of this review.

11. Recommendations

The following are recommended based upon the findings of this review: -

- Develop an underground digital resource model with separate domains for PGH, Upper C (Garnet hornfels) and Lower C (Banded garnet hornfels) to permit better definition of high grade resources.
- Recalculate the Mining Inventory/Reserve from the revised block model ensuring that all post pillars, skin pillars, remote ore pods, small ore pods and sterilised ore are excluded from the assessment. Any ore in development pillars should be clearly indicated in the inventory as pillar ore.
- Assume that the rehabilitation of existing decline development is the preferred re-access option.
- Following the above conduct Financial Analysis to determine if the project is viable or determine the target mining inventory to generate a viable project.

12. Statement of capability

This review has been undertaken by Alan Fudge, of Polberro Consulting. Alan Fudge is a qualified mining and geotechnical engineer with over 30 years experience of mining projects principally in Australia, UK and Zambia. He has extensive experience in the areas of reserve estimation, mining method selection, providing of mining and geotechnical technical support (for underground and surface mining operations), scheduling, mine design and project feasibility. He has been involved in all stages of mine development from feasibility to closure and has worked at both open-pit and underground mines.

Alan Fudge is a Member of the AUSIMM and has prepared reserve statements as a competent person as defined by the 2004 Edition of the JORC code. He has worked on and produced reserve statements for tin, copper, nickel, zinc, lead, tungsten, magnetite, gold and uranium.

Limitations and consent

This report is provided to the King Island Scheelite Pty Ltd in the context of an independent review of the mining and geotechnical aspects of the underground mining studies conducted on behalf of KIS for the Dolphin Mine and should not be used or relied upon for any other purpose.

The report has been prepared using information available to the author at the time of writing. The opinions stated herein are given in good faith and with the belief that the basic assumptions are factual and correct and the interpretations reasonable.

This report is not intended for use as a public document nor, in whole or in part, in a public document without written consent to the form and context in which it appears.

Statement of independence

Alan Fudge has no material interest or entitlement in the securities or assets of King Island Scheelite Pty Ltd or any associated companies.

13. References

The following documents and data were referenced and form the principal basis for inferences drawn and recommendations made in this report. Only documents that could be studied and commented on within the time frame of this review are included and areas not covered by these reports and references have not been considered in this review.

1. AMC 109058 Dolphin and Bold Head Mine Scoping Study - November 2009
2. AMC 109034 Underground Scoping Study – July 2009
3. Archived 1:500 Development and Stopping Plan Set
4. Geological Resource Model *Modfin.dm*
5. MINENHIL00237AB Coffey Geotechnical Investigation - July 2009
6. In situ and long term stress field monitoring at King Island Scheelite Dolphin Mine – Unpublished internal report - A.Fudge 1982

14. Appendices

Appendix One

North Ltd Mining Inventory 1990

Source: - Unpublished Reserve estimate tabulation for ore above 0.7% COG prepared in 1990

Dolphin Mine Geological Resource and Mining Reserve above 0.7% cut off

STOPPING	AREA	+1% WO3 Resource		0.7-1% Resource		Pillar Resource		Mineable Resource		Mining Reserve	
		Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade
Wedge	Upper C	693,400	1.57	6,000	0.92	284,500	1.56	408,900	1.57	311,139	1.40
Wedge	Lower C	348,900	1.20	256,300	0.84	81,200	1.21	267,700	1.20	337,960	.91
Central	Upper C	81,800	1.34			42,600	1.32	39,200	1.36	32,785	1.20
Central	Lower C	46,900	1.43	11,600	0.82	21,100	1.43	25,800	1.43	27,617	1.12
Pit	Upper C	244,500	1.32	18,100	0.81	95,100	1.31	149,400	1.33	141,962	1.12
Pit	Lower C	46,400	1.31	1,200	0.88	15,700	1.31	30,700	1.32	26,688	1.14
Pit Dag	Lower C	20,000	1.31			8,100	1.31	11,900	1.31	10,469	1.14
Dodo	Upper C	16,700	1.24					16,700	1.24	12,428	1.09
Auk	Upper C	21,700	2.21					21,700	2.21	14,534	1.94
Pheasant	Upper C			44,900	0.85					30,073	.75
Coast	Upper C	1,400	1.37	26,100	0.75			1,400	1.37	18,419	.69
Auk	Lower C	4,000	1.14	14,000	0.89			4,000	1.14	12,162	.82
Swan	Upper C	50,000	1.36	8,400	0.84			50,000	1.36	49,275	1.17
Total		1,575,700	1.42	386,600	0.83	548,300	1.43	1,027,400	1.41	1,025,510	1.13

Table 6: - Mining Inventory Estimate 4

Item	From	To	Tonnes	WO3%	mtu	ID	R	D	Tonnes	WO3%	mtu
B Lens 1	-55	-70	30,798	0.65	19,926	PP	0.82	0.1			
Upper Central	-60	-75	79,011	0.66	52,463	PP	0.75	0.1			
UC Dec Pillar	-60	-75	72,956	0.71	51,726						
Upper Pit 1	-60	-80	89,978	0.88	78,911	PP	0.75	0.1	74,232	0.80	59,183
B Lens 2	-75	-90	42,231	0.67	28,126	PP	0.82	0.1			
B Lens 3	-70	-90	52,821	0.76	40,144	pp	0.82	0.1			
Upper Wedge	-65	-95	67,549	0.95	64,104	PP	0.7	0.1	52,013	0.86	44,873
Upper Central 2	-80	-110	55,408	0.74	40,725	PP	0.6	0.1			
Mid Wedge 1	-85	-120	73,781	0.68	50,392	PP	0.6	0.1			
Central RP	-75	-125	52,955	0.91	48,401	LH	0.6	0.2	38,128	0.76	29,041
Mid Wedge 2	-110	-125	26,288	1.09	28,549	PP	0.6	0.1	17,350	0.99	17,129
Mid Central 1	-120	-135	15,751	1.10	17,263						
B Lens 5	-125	-155	59,261	0.62	36,860	PP	0.82	0.1			
Mid Central 2	-105	-165	234,901	0.86	202,485	PP	0.82	0.1	211,881	0.78	166,037
Mid Wedge 3	-135	-165	650,761	0.83	540,782	PP	0.8	0.1	572,670	0.76	432,626
Mid Pit 3	-145	-165	168,373	0.90	151,367	PP	0.82	0.1	151,872	0.82	124,121
Mid Pit 2	-145	-170	147,348	1.22	179,765	PP	0.82	0.1	132,908	1.11	147,407
Dodo/Pit Dag	-155	-185	68,102	0.89	60,338						
B Lens 4	-170	-195	23,476	0.65	15,330						
Lower Wedge 4	-180	-195	62,363	0.93	57,811	PP	0.82	0.1	56,251	0.84	47,405
Lower Wedge 9	-180	-195	117,564	1.04	121,914	PP	0.82	0.1	106,043	0.94	99,969
Lower Wedge 3	-202	-212	226,195	1.07	242,029	PP	0.8	0.1	199,052	0.97	193,623
Lower Wedge 7	-220	-235	274,830	1.03	282,800	PP	0.8	0.1	241,850	0.94	226,240
Lower Wedge 2	-220	-235	206,222	0.89	182,919	PP	0.8	0.1	181,475	0.81	146,335
South 1	-230	-250	59,119	0.77	45,581	PP	0.82	0.1	53,325	0.70	37,376
Lower Wedge 6	-235	-255	296,732	0.94	277,741	PP	0.8	0.1	261,124	0.85	222,193
Lower Wedge 1	-265	-285	72,271	1.26	91,134	PP	0.75	0.1	59,624	1.15	68,350
South 2	-245	-290	97,070	0.57	55,427	PP	0.82	0.1			
			3,424,115	0.90	3,065,012				2,409,798	0.86	2,061,909

Estimate 4 – Inclusive of sub 0.7% WO3 cells

Table 7:- Mining Inventory Estimate 4a

Item	From	To	Tonnes	WO3%	mtu	M	R	D	Tonnes	WO3%	mtu
B Lens 1	-55	-70	11,170	1.30	14,543	PP	0.82	0.1	10,075	1.18	11,926
Upper Central	-60	-75	29,923	1.07	32,048	PP	0.75	0.1	24,686	0.97	24,036
UC Dec Pillar	-60	-75	19,704	1.08	21,300						
Upper Pit 1	-60	-80	46,490	1.26	58,624	PP	0.75	0.1	38,354	1.15	43,968
B Lens 2	-75	-90	14,274	1.11	15,773	PP	0.82	0.1	12,875	1.00	12,934
B Lens 3	-70	-90	21,650	1.22	26,456	pp	0.82	0.1	19,528	1.11	21,694
Upper Wedge	-65	-95	50,850	1.20	60,766	PP	0.7	0.1	39,155	1.09	42,536
Upper Central 2	-80	-110	23,732	1.09	25,868	PP	0.6	0.1	15,663	0.99	15,521
Mid Wedge 1	-85	-120	35,778	0.89	31,878	PP	0.6	0.1			
Central RP	-75	-125	32,414	1.17	37,957	LH	0.6	0.2	23,338	0.98	22,774
Mid Wedge 2	-110	-125	19,816	1.33	26,355	PP	0.6	0.1	13,079	1.21	15,813
Mid Central 1	-120	-135	9,451	1.58	14,885						
B Lens 5	-125	-155	17,093	1.06	18,170	PP	0.82	0.1			
Mid Central 2	-105	-165	171,827	0.99	170,796	PP	0.82	0.1			
Mid Wedge 3	-135	-165	470,772	0.95	447,704	PP	0.8	0.1			
Mid Pit 3	-145	-165	106,862	1.12	119,151	PP	0.82	0.1	96,390	1.01	97,704
Mid Pit 2	-145	-170	82,661	1.81	149,947	PP	0.82	0.1	74,560	1.65	122,957
Dodo/Pit Dag	-155	-185	48,273	1.04	50,300						
B Lens 4	-170	-195	15,317	0.86	13,203						
Lower Wedge 4	-180	-195	52,616	1.02	53,668	PP	0.82	0.1			
Lower Wedge 9	-180	-195	97,824	1.16	113,182	PP	0.82	0.1	88,237	1.05	92,810
Lower Wedge 3	-203	-213	178,558	1.25	222,840	PP	0.8	0.1	157,131	1.13	178,272
Lower Wedge 7	-220	-235	213,118	1.19	252,971	PP	0.8	0.1	187,544	1.08	202,377
Lower Wedge 2	-220	-235	152,253	1.05	160,322	PP	0.8	0.1			
South 1	-230	-250	38,026	0.91	34,604	PP	0.82	0.1			
Lower Wedge 6	-235	-255	217,122	1.09	237,314	PP	0.8	0.1	191,067	0.99	189,851
Lower Wedge 1	-265	-285	67,731	1.34	90,963	PP	0.75	0.1	55,878	1.22	68,222
South 2	-245	-290	19,014	0.82	15,610	PP	0.82	0.1			
			2,264,319	1.11	2,517,201				1,047,561	1.11	1,163,394

Estimate 4a - Exclusive of sub 0.7% WO3 cells

Appendix Two

Hollow Inclusion In-situ stress test results - Fudge 1982

Summary of Stress Field Calculations for H.I. Cell Tests			
Principal Stresses	Bearings	Elevations	Test No.
14.2 5.8 5.4	193.8 104.1 280.9	0.9 18.2 71.8	2/3 (-116m)
-13.8 -14.9 -30.6	186.5 91.8 181.2	71.4 1.7 18.5	4/4 (-95m)
2.2 0.2 -1.9	283.9 0.5 194.1	1.3 84.0 5.5	4/5 (-95m)
-1.0 -1.4 -2.0	274.6 44.5 356.0	20.8 59.4 21.4	5/3 (-116m)
6.1 1.2 -3.1	316.6 49.5 221.5	3.8 36.8 53.0	7/1 (-95m)
2.7 0.5 -3.7	348.9 76.7 295.5	9.7 12.6 74.0	8/2 (-95m)
0.5 -0.2 -2.6	273.6 86.2 357.6	64.0 25.8 2.9	8/3 (-95m)
1.3 1.0 -2.8	189.2 62.9 331.3	76.2 8.3 11.0	11/2 (-95m)
0.3 -1.1 -3.1	226.6 10.9 283.9	71.1 15.5 10.5	11/4 (-95m)
-0.9 -1.1 -1.2	200.6 163.7 280.5	29.2 55.1 17.5	11/7 (-95m)
-6.0 -7.0 -11.2	303.0 135.2 214.2	18.9 70.7 3.8	12/3 (-112m)
-0.3 -0.4 -4.6	285.3 103.1 194.0	49.2 40.8 1.1	18/5 (-75m)
-0.4 -0.6 -1.0	260.4 51.9 322.5	80.9 8.0 4.3	22/5 (-100m)
2.9 0.9 -0.5	291.4 9.1 247.8	28.5 21.3 53.2	22/7 (-100m)
0.0 -2.2 -7.0	262.1 69.7 340.0	80.7 9.1 2.0	27/1 (-100m)

(Compression negative)

Convention used: Compression -ve.

Appendix Three

Isometric Plot of MRO output of AMC109034 Mining Inventory Wireframes showing the inclusion of a large scatter of isolated cells, satellite pods and post pillars within total inventory.

