



CLEVELAND MINE, LUINA, TASMANIA

MINERAL RESOURCE REPORT

For

ROCKWELL MINERALS LIMITED

Job No. 5119_G
Doc No. 3555v2.doc
Date: April 2013
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Quality
ISO 9001
SAI GLOBAL

FINAL REPORT

Textual abbreviations used in this report

Aberfoyle	Any of the Aberfoyle and Cominco group of companies including, for example, Aberfoyle Ltd, Cleveland Tin N.L., Aberfoyle Exploration Pty Ltd and Cominco Exploration Pty Ltd.
EL	Exploration Licence.
HMS	Heavy Media Separation.
JORC	Joint Ore Reserve Committee (of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geosciences and Minerals Council of Australia).
Lynch Mining	Lynch Mining Pty Ltd.
Mining One	Mining One Pty Ltd
MRT	Mineral Resources Tasmania (a State Government Department).
Rockwell	Rockwell Minerals Limited.
Western Metals	Western Metals Limited.

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EXECUTIVE SUMMARY

Rockwell Minerals Limited is investigating the tin, copper and tungsten hard rock deposits and tin and copper tailings at the Cleveland Mine at Luina on the West Coast of Tasmania. The Cleveland Mine is an underground tin and copper mine which closed in 1986 after which the surface area of the mine site was rehabilitated.

The Cleveland Mine is located at Luina about 80km by sealed road from Burnie.

The Cleveland mine occurs within Exploration Licence 7/2005 (“EL7/2005”) which is held by Lynch Mining Pty Ltd (“Lynch Mining”), 50%, and Rockwell, 50%. Rockwell has entered into an Option to Purchase with Lynch Mining to acquire the remaining 50% of EL7/2005.

During the life of the Cleveland operations, Aberfoyle Limited and Cleveland Tin N.L. (a subsidiary of Aberfoyle Ltd) mined and treated 5.7M tonnes of ore for a production of ~24,000 tonnes of tin and ~10,000 tonnes of copper in concentrate

Adequate geological records have survived from the time of the Aberfoyle operations to allow for the estimation of Mineral Resources and reporting of the Mineral Resources in accordance with the JORC Code. The validity of the Aberfoyle data was considered to be reliable having been accumulated by Aberfoyle during the period when that company was the major operator in the tin and tungsten mining industry with four operating tin and tungsten mines in Australia. In addition, re-splitting and re-assaying of Aberfoyle drill core, stored at Mineral Resources Tasmania, confirmed the reliability of the Aberfoyle sampling and tin and copper assaying methods.

Aberfoyle drilled 2040 diamond drill holes into the deposits for a total drilled length of about 130,000m.

The mineralisation in the Cleveland mine is principally of two styles: tin and copper bearing semi-massive sulphide lenses replacing limestone and a tungsten bearing quartz stock-work.

The tin and copper bearing lenses are more or less vertically dipping, lenticular deposits with strike lengths of up to 550 metres, across strike thicknesses of up to 30 metres and down-dip extents of up to 800 metres.

The tungsten bearing stock-work is currently considered to dip vertically and has a known strike length of about 300 metres, an across strike width of up to 300 metres and a down-dip extent of about 900 metres.

Tin occurs as cassiterite and in very minor amounts as stannite, copper occurs principally as chalcopyrite and in very minor amounts as stannite, and tungsten occurs as wolframite.

Mineral Resources have been estimated for tin and copper in the tin and copper bearing lenses, tungsten in the tungsten bearing stock-work, and tin and copper in tailings.

For the hard rock deposits, a block model with a parent block size of 10m X 10m X 10m was made with sub-calling allowed to a minimum block size of 2.5m X 2.5m X 2.5m.

A bulk density of 3.1 tonnes/m³ was considered reasonable for the tin and copper bearing lenses and 2.85 tonnes/m³ for the tungsten bearing stock-work. The tonnage of tailings was estimated from the reported operating records of the Aberfoyle processing plant.

All tonnage estimates were made in dry tonnes.

Sn and Cu grades were interpolated into the blocks representing the lenses using an inverse distance squared method.

WO₃ grade was interpolated into the blocks representing the stock-work by ordinary kriging.

Sn and Cu grades for the tailings were estimated from the reported operating records of the Aberfoyle processing plant.

No mining or metallurgical assumptions were made during the estimation of the Mineral Resource for this report.

For tin and copper resources, a Sn equivalent grade was estimated from the Sn and Cu grades, assuming metal prices current at the time of writing, that is, US\$25000 per tonne for tin and US\$8500 per tonne for Cu.

Cleveland Tin and Copper Mineral Resource				
31 March 2012				
0.35% Sn cut-off				
Category	Tonnage	% Sn	% Cu	% Sn Equivalent
Indicated	4,239,000	0.70	0.28	0.80
Inferred	1,880,000	0.64	0.19	0.70
Total	6,119,000	0.68	0.25	0.77

Cleveland Tungsten Mineral Resource		
31 March 2012		
0.20% WO₃ cut-off		
Category	Tonnage	% WO₃
Inferred	3,980,000	0.30
Total	3,980,000	0.30

Cleveland Tin and Copper in Tailings Mineral Resource				
31 March 2012				
0% Sn cut-off				
Category	Tonnage	% Sn	% Cu	% Sn Equivalent
Inferred	3,850,000	0.30	0.13	0.34
Total	3,850,000	0.30	0.13	0.34

The Cleveland tin copper deposit is open at depth and along strike. In addition, there are at least two shallow targets for lenses which outcrop or come close to outcrop, notably Luck's and Henry's No 1 pod. There are areas within the current extents of the mine where extensions to the known deposit are likely, as well as the so far unestimated, remnant resources. The tungsten mineralisation in Foley's stock-work has only been estimated down to 850m RL. There is considerable scope to increase the tungsten resource below this level.

1 INTRODUCTION

Rockwell Minerals Limited (“Rockwell”) is an Australian public unlisted company based in Queensland. Rockwell is investigating the mineral deposits and tailings at the Cleveland Mine at Luina on the West Coast of Tasmania. The Cleveland Mine is an underground tin and copper mine which closed in 1986 after which the surface area of the mine site was rehabilitated.

The Cleveland Mine is located at Luina about 80km from Burnie (see Figure 1).

The Cleveland mine occurs within Exploration Licence 7/2005 (“EL7/2005”) which is held by Lynch Mining Pty Ltd (“Lynch Mining”), 50%, and Rockwell, 50%. Rockwell has entered into an Option to Purchase with Lynch Mining to acquire the remaining 50% of EL7/2005. EL7/2005 has an area of 18 square kilometres (see Figure 2) and includes the Cleveland Mine, the Cleveland tailings dams and the area formerly occupied by the surface infrastructure for the former Cleveland operations.

There are resources of tin, copper and tungsten at Cleveland in the form of semi-massive sulphide, quartz stock-works and greisen, and tailings.

Cleveland was operated by Cleveland Tin N.L., Aberfoyle Limited and other Aberfoyle group companies (collectively referred to in this report as “Aberfoyle”).

This report is a report of the tin and copper Mineral Resources in the Cleveland mine in accordance with the 2004 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves of the Joint Ore Reserve Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geosciences and Minerals Council of Australia (“JORC Code”).

Much of the information available for this report was compiled by Aberfoyle and associated parties over the period 1968 to 1986. The principal sources of information are listed in the References. There is a repository of data and information regarding the Cleveland Mine which was accumulated by Aberfoyle and now held at the offices of the Burnie Research Lab, 39 River Road, Burnie, Tasmania.



Figure 1: Location of the Cleveland Mine

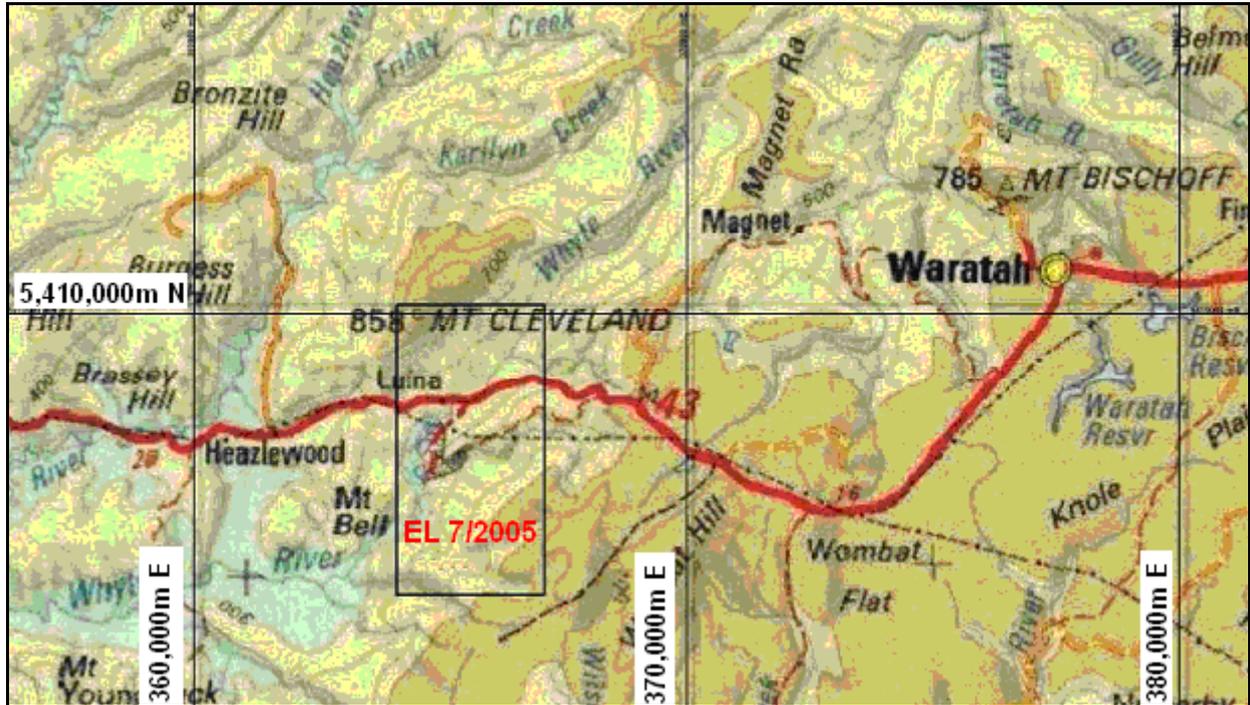


Figure 2: Location of Exploration Licence 7/2005 (Coordinates are based on Australian Geodetic Datum 1966)

2 PROPERTY LOCATION, ACCESS AND TENURE

2.1 Property Location and Access

The Cleveland mine is located at Luina about 80km from Burnie (see Figure 1). Access to the mine is by way of the sealed all weather road which runs from Burnie through Waratah and Luina to Savage River.

The topography around the mine is relatively steep and rugged with elevations ranging from about 300m to over 500m above sea level. The mine was developed beneath Crescent Hill which rises to an elevation of 520m while the former township of Luina, the former mine infrastructure and the tailings dams lie in the valleys of the Whyte River and Deep Creek (see Figure 3).

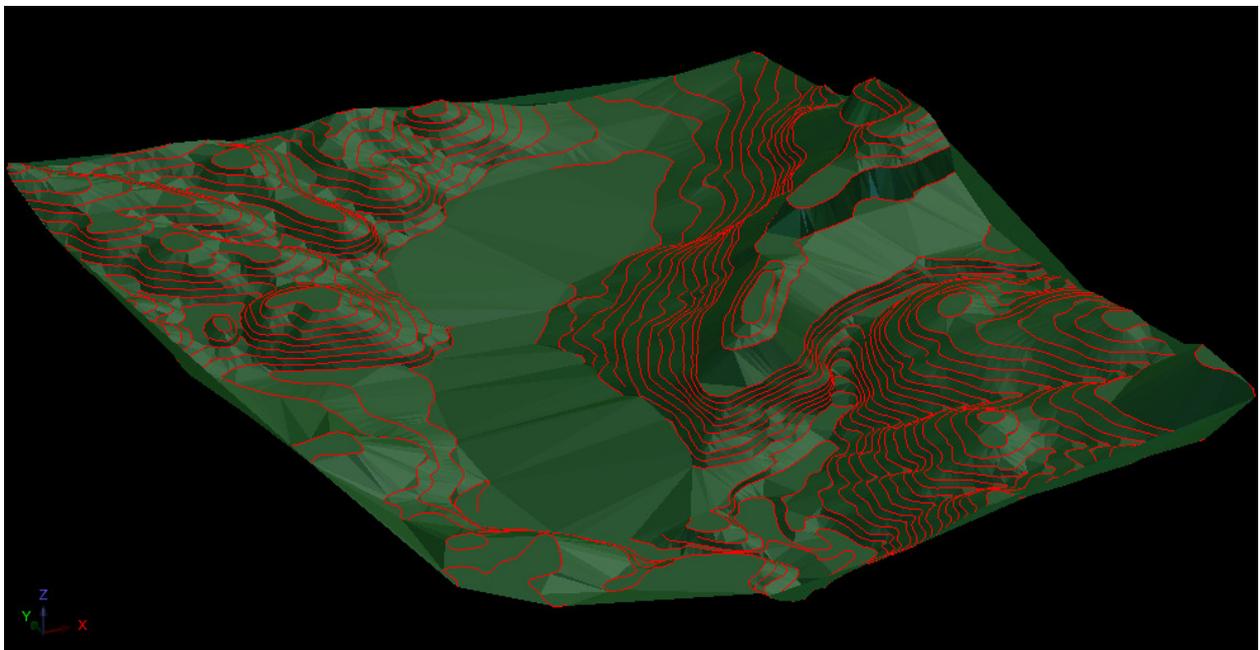


Figure 3: Topography at the Cleveland Mine

2.2 Property Tenure

Mining One has not reviewed the type or status of the exploration tenure at Cleveland.

Also, Mining One has not reviewed the type or status of land titles at Cleveland nor the types or status of any Federal, State or Local Government laws or regulations which may restrict or define the uses of the land in the areas covered by the Cleveland Exploration Licence.

3 HISTORY OF EXPLORATION AND MINING

3.1 Discovery and Early Mining

Outcrops of gossan were first discovered by prospectors at Cleveland in about 1898. It is likely that the gossan was originally hoped to be the outcrop of a silver lead deposit. Tin was first identified at Cleveland in 1900 by Harcourt Smith, Government Geologist. Mining by the Cleveland Tin Mining Co NL commenced in 1908 but the mine closed in 1917 after production of about 275 tonnes tin in the form of cassiterite concentrate. (Reid, 1923)

From 1935 to 1939, the Mt Bischoff Tin Mining Co undertook exploration by small scale underground mining but no tin production occurred during this period (Porte, 1937).

3.2 Exploration and Modern Mining

Exploration by the Tasmania Mines Department and the Australian Bureau of Mineral Resources in the 1950s identified the potential for relatively large cassiterite bearing sulphide ore bodies (Hughes, 1952, 1953a, 1953b, 1954 and Keunecke, O. and Tate, K.H., 1954). Consequently, in 1961, the Aberfoyle Tin Development Partnership acquired the leases at Cleveland and commenced systematic exploration based on geological mapping and using diamond drilling (Mason et al., 1963). Since then, over 2000 diamond drill holes have been drilled into the deposit and its known depth has been demonstrated to about 700 metres below the surface (Barth, 1986).

In the 1960s and early 1970s, the mines of the Aberfoyle group of companies were Australia's chief producers of tin and tungsten. The group operated Cleveland tin and copper mine, the Storeys Creek and Aberfoyle tin and tungsten mines in North-East Tasmania, and the Ardlethan tin mine in southern New South Wales.

Mining of ore at Cleveland commenced in 1968 using trackless methods for mining and ore haulage to the surface. The Cleveland Mine was among the first in the world to use trackless mining and the mine now extends to a depth of over 500 metres below surface (see Figure 4).

Production from the mine during the Aberfoyle operation from 1968 to 1986 was 5,645,000 tonnes at a grade of 0.68% Sn and 0.29% Cu (see **Error! Reference source not found.**). The Cleveland mine produced about 24,000 tonnes of tin in concentrate and about 10,000 tonnes of copper in concentrate.

Cleveland closed in 1986. The tin price fell significantly in 1985 following the collapse of price support from the International Tin Council (see Figure 5). In addition, Aberfoyle had turned its attention to lead, zinc and copper following the discovery of the Que River deposit in 1974 and the Hellyer deposit in 1983. It is, perhaps, no coincidence that Cleveland was closed in June 1986, the same month that the Hellyer orebody was exposed in the Hellyer Decline.

Table 1: Historical Summary of Exploration and Mining at the Cleveland Mine

1898	S.C. Coundon, Prospector	Pegged leases over gossan for possibility of silver and lead.
1900	Harcourt Smith Government Geologist Department of Mines, Tasmania	Identified cassiterite in gossan.
1908 - 1917	Cleveland Tin Mining Company N.L.	Mined oxidised ore for tin.
1923	A.M. Reid Government Geologist Department of Mines, Tasmania	Recognised fissure lodes and replacement lodes.
1935-1937	Mount Bischoff Tin Mining Company	Small scale underground exploration: Battery, Smithy, Lucks, Khaki, Hall's, Henry's recognised.
1937	Q.J. Henderson Government Geologist Department of Mines, Tasmania	Described the work undertaken by the Mount Bischoff Tin mining Company.
1945	S.W. Carey Government Geologist Department of Mines, Tasmania	Reported all deposits were of replacement style.
1952-1954	T.D. Hughes Government Geologist Department of Mines, Tasmania	Postulated that the ore would continue in depth. Recommended cutting of a grid and geophysical surveys.
1953-1954	O. Keunecke and K.H. Tate Bureau of Mineral Resources Commonwealth of Australia	Concluded self-potential and magnetic surveys anomalies suggested that sulphide mineralisation may extend beyond the old workings.
1961-1965	Aberfoyle Tin Development Partnership	Explored the area with diamond drilling and proved up sufficient resources for mining.
1968-1986	Cleveland Tin N.L. and Aberfoyle Limited	Mined tin and copper ore.

**Table 2: Cleveland Mine – Mine Production during Aberfoyle Operations
(Compiled from Aberfoyle Ltd and Cleveland Tin N.L. annual reports)**

Year	Ore Treated tonnes	Ore Treated	
		% Sn as cassiterite	% Cu
1968	?	?	?
1969	256,865	0.85	0.40
1970	281,875	0.79	0.37
1971	305,726	0.73	0.42
1972	357,498	0.78	0.40
1973	505,806	0.76	0.32
1974	314,210	0.75	0.25
1975	289,018	0.78	0.32
1976	363,036	0.73	0.27
1977	393,275	0.66	0.22
1978	388,579	0.53	0.18
1979	352,977	0.52	0.24
1980	367,866	0.47	0.21
1981	439,304	0.51	0.22
1982	350,300	0.64	0.28
1983	277,700	0.71	0.25
1984	180,300	0.71	0.26
1985	137,000	0.80	0.25
1986	83,700	0.92	0.41
Total	5,645,035	0.68	0.28

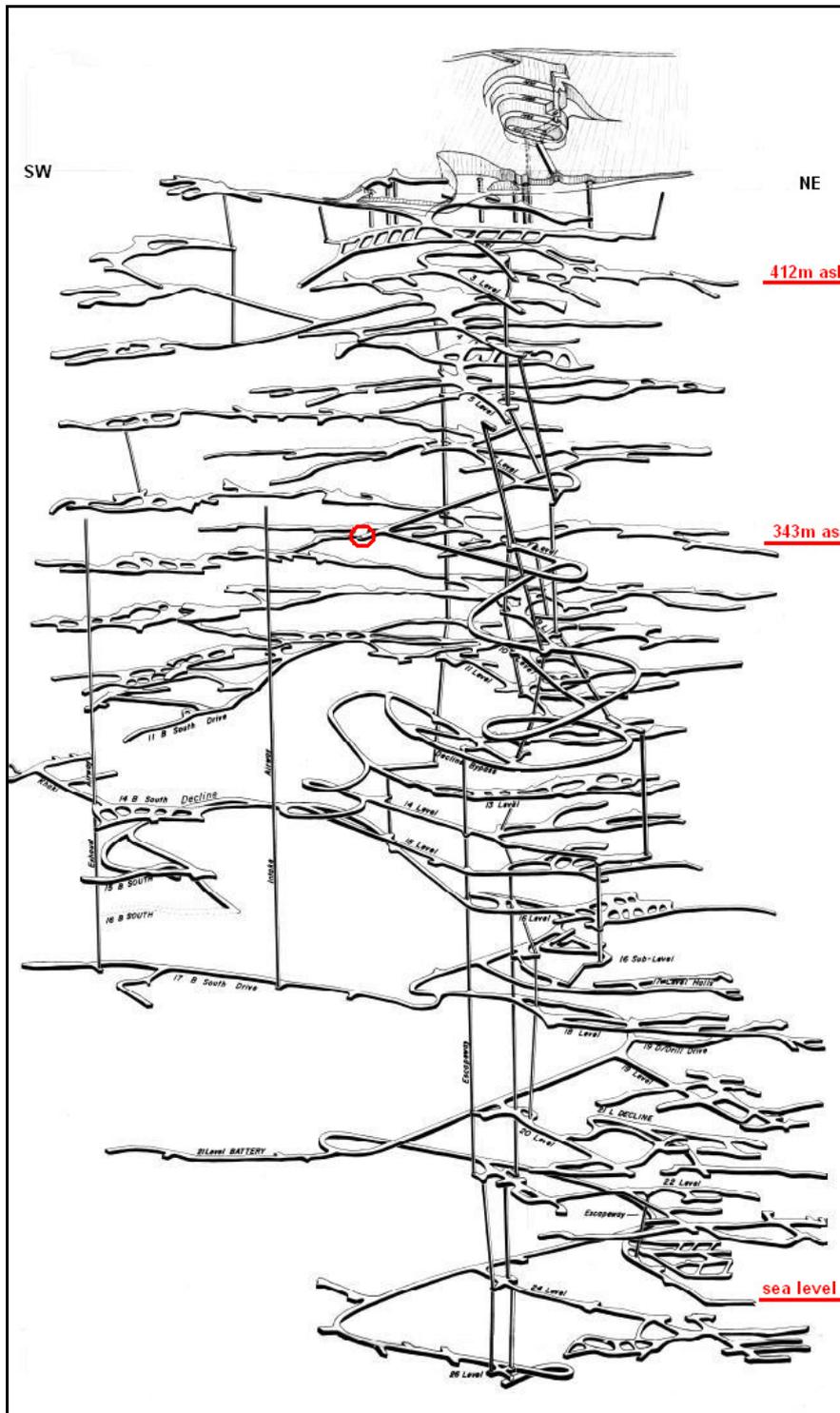


Figure 4: Cleveland Mine – Perspective View

The entrance to the Cleveland Decline is circled; the mine was over 500m deep: the heights above sea level (asl) are shown on the right.

(Taken from Aberfoyle records)

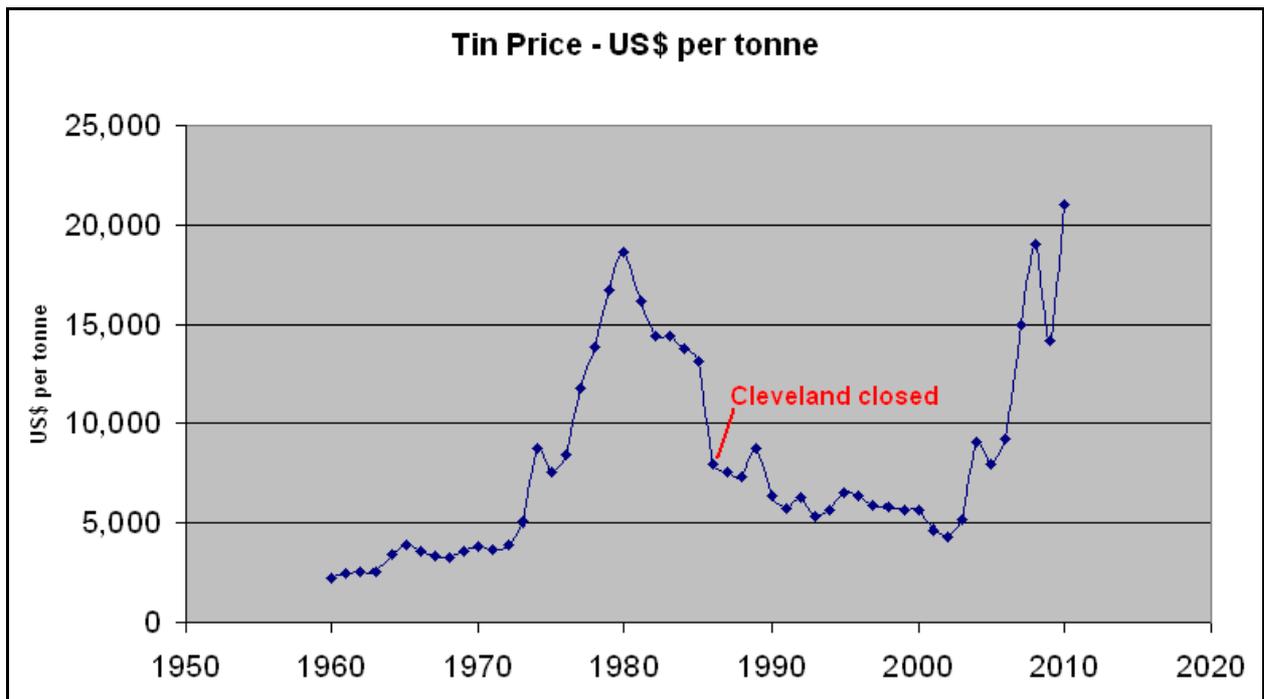


Figure 5: Tin Price 1960 to 2010 – US\$ per tonne
(Compiled from Minerals Year Books of the Bureau of mines of the United States Geological Survey)

4 GEOLOGY

4.1 Ore Minerals in the Cleveland Mine

Worldwide, tin is won from cassiterite, copper from chalcopyrite and tungsten from wolframite or scheelite (see Table 3). Tin and copper grades are usually expressed as % Sn and % Cu respectively; tungsten grades are usually expressed as % WO₃.

Tin, copper and tungsten occur at Cleveland: tin occurs principally as cassiterite and in very minor amounts as stannite; copper occurs principally as chalcopyrite and also in minor amounts as stannite; tungsten occurs as wolframite.

Table 3: Tin, Copper and Tungsten Ore Minerals

Ore Mineral	Formula	Metal Content	Occurrence at Cleveland
cassiterite	SnO ₂	78.6% Sn	Yes
stannite	Cu ₂ S.FeS.SnS ₂	27.5%Sn, 29.5%Cu	Yes
wolframite	(Fe,Mn)WO ₄	76.4% WO ₃	Yes
scheelite	CaWO ₄	80.5% WO ₃	No

Traditional processing of tin bearing ore recovers only cassiterite. Consequently, it is important to know how much of the tin in a deposit occurs as cassiterite. Stannite is soluble in nitric acid whereas cassiterite is not and soluble tin assays are assays of tin occurring in stannite and the determination of the grade of tin as cassiterite is calculated by determining the total tin and the soluble tin grade and then subtracting one from the other:

$$\text{grade of tin as cassiterite} = \text{total tin grade} - \text{soluble tin grade}$$

This was the practice by Aberfoyle at Cleveland.

4.2 Tin Ore Processing

Tin is relatively scarce in the Earth's crust averaging about 2 parts per million compared, for example, with copper which averages about 63 parts per million. Tin is differentiated in magma and is associated with granitic and rhyolitic rocks.

Cassiterite is currently the only tin bearing mineral of economic significance although small amounts of tin are recovered from stannite and other complex sulphide minerals.

Cassiterite is relatively heavy with a specific gravity of about 7gm per cubic centimeter.

Traditionally, cassiterite bearing ore from hard rock deposits, such as those at Cleveland, is reduced in size by crushing and the crushed material is passed through a Heavy media separation ("HMS") process where some less dense, tin poor material is floated off and discarded. The remaining ore then passes to grinding so that the cassiterite grains are mostly liberated from the other minerals present. Then, coarser cassiterite is concentrated by gravity methods using jigs, spirals and shaking tables. Sulphide minerals present are then removed by sulphide flotation and finer cassiterite is concentrated by cassiterite flotation into a cassiterite

concentrate usually assaying over 60% Sn. The processing recovery of tin is commonly in the range of 55% to 75%. The cassiterite concentrate is then smelted to recover the tin in a traditional tin smelter.

An alternative method of tin concentration, known as fuming, was first suggested in the 1970s.. This process mirrors that just described but a cassiterite concentrate assaying only about 10% Sn is produced. The processing recovery of tin by fuming is hoped to be as high as 85% to 90%. The cassiterite concentrate is then smelted to recover the tin in a special tin smelter which employs a smelting process known as fuming.

4.3 Tasmanian Tin Geology

In Tasmania, tin deposits are associated with granite of Devonian-Carboniferous age, including the deposits at Renison, Cleveland and Mt Bischoff on the West Coast, and Aberfoyle and Storeys Creek in the North-East (see Table 4). The deposits are hosted in older sedimentary and metamorphic rocks.

For example, granite outcrops at Pine Hill about 2 kilometres south of Renison and has been intercepted in drilling below the mine; granite outcrops within 5 kilometres of the Cleveland Mine; granite was exposed in the lower workings of both the Storeys Creek and Aberfoyle Mines.

The geological settings are similar to those of the tin deposits of Cornwall and Devon which are also associated with granite of Devonian-Carboniferous age and hosted in older sedimentary and metamorphic rocks.

In particular, the larger deposits in Tasmania are hosted in part by rocks of the Cambrian Crimson Creek Formation or its stratigraphic equivalent, notably the Renison, Cleveland and Mt Bischoff deposits. To date, Renison has produced over 200,000 tonnes of contained tin, Mt Bischoff about 60,000 tonnes and Cleveland about 36,000 tonnes.

Table 4: Tin, Tungsten and Copper Production from Tasmanian Tin Mines
(All figures, except those for Cleveland are from Green, 1990; figures for Cleveland are from Aberfoyle reports of operations at Cleveland.)

Mine	Ore- tonnes	% Sn	%WO3	% Cu
Renison ¹	11,300,000	1.18	-	-
Mt Bischoff	5,590,000	1.56	-	-
Cleveland	5,650,000	0.68	-	0.29
Aberfoyle	2,100,000	0.91	0.28	-
Storey's Creek	1,100,000	0.18	1.09	

1. Note that Renison is still in production.

4.4 West Coast Geology

In Tasmania, the principal metal mines are associated with Devonian-Carboniferous granite, or the Cambrian Mt Read Volcanic rocks, or the Precambrian metamorphic rocks of the Arthur Mobile Belt. Tin and tungsten deposits and some silver-lead-zinc deposits are associated with Devonian-Carboniferous granite; lead-zinc, copper and gold deposits are associated with the Mt Read Volcanics; large iron deposits are associated with the Arthur Mobile Belt. (See Figure 6).

Tasmania's three largest tin mines occur on the West Coast: Renison, 35 kilometres south of Cleveland, Mt Bischoff, 15 kilometres north-east of Cleveland, and Cleveland (refer to Table 4). Renison has been in production for over a century, Mt Bischoff was mined from 1872 to 1947 and for a brief period in 2009 and 2010, and Cleveland was mined from 1908 to 1917 and from 1968 to 1986.

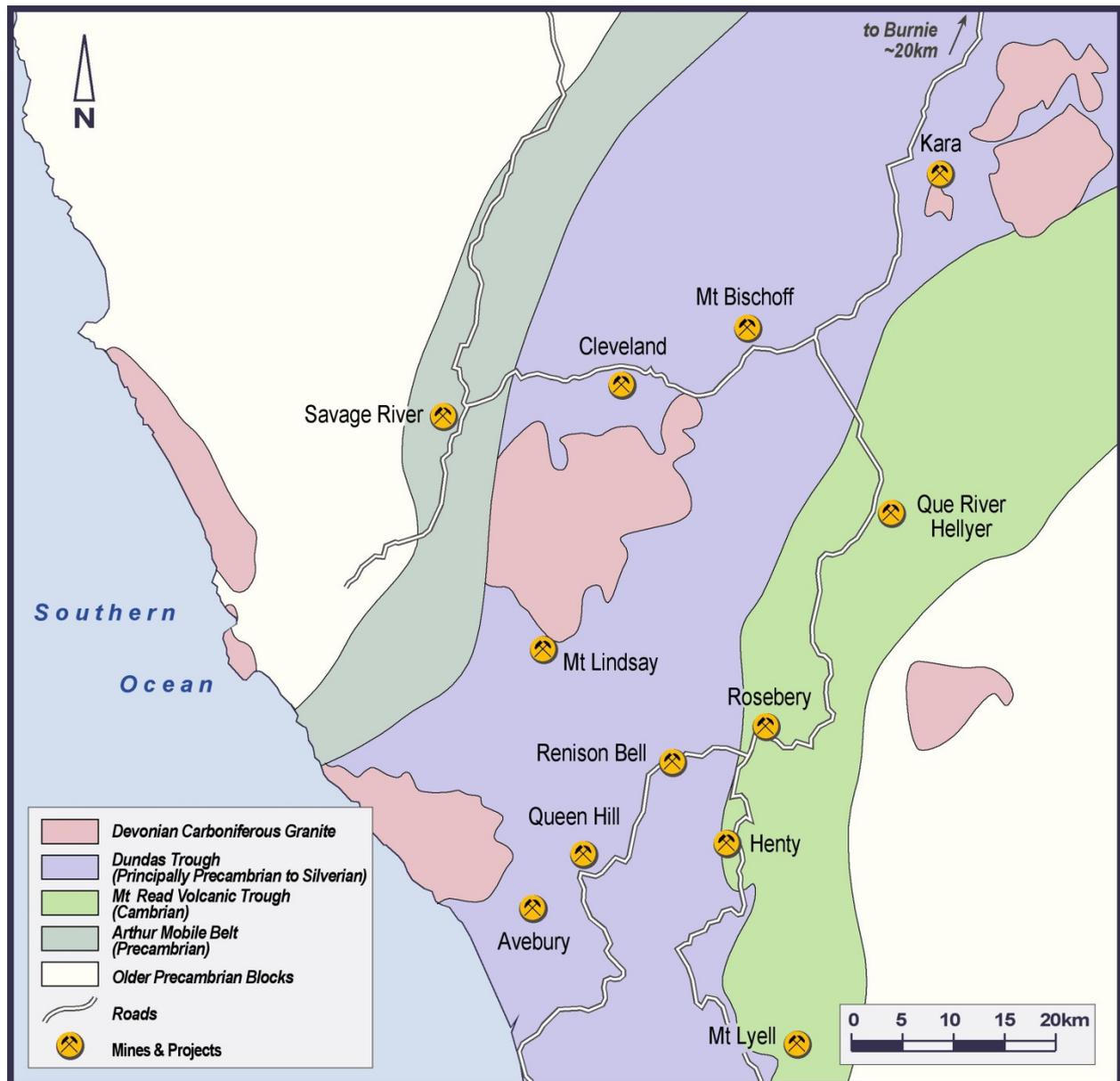


Figure 6: Geological Regimes and Principal Mines on the West Coast of Tasmania

4.5 Local Geology

The Cleveland and Mt Bischoff tin mines both occur to the north of the outcrop of the Meredith granite (see Figure 6 and Figure 7).

Two other mines occur in the area: the Magnet silver lead mine between Cleveland and Mt Bischoff and the Godkin Mine to the south of Cleveland (see Figure 7). The more significant of these was the Magnet Mine which operated from 1895 to 1940 and produced 630,000 tonnes of ore containing 38,000 tonnes of lead and 8,000,000 ounces of silver (Cottle, 1953).

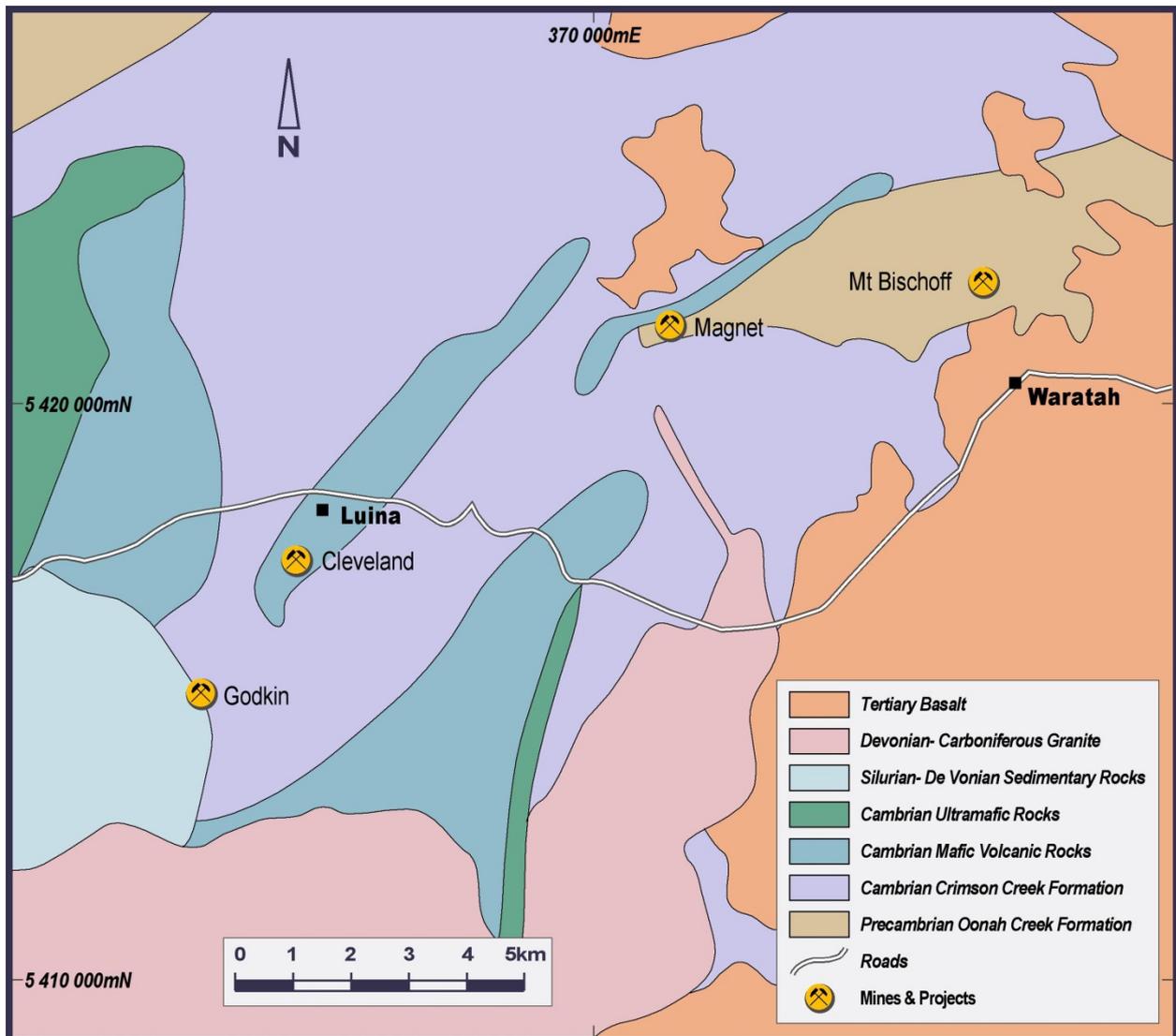


Figure 7: Local Geology around Cleveland

4.6 Mine Geology

At Cleveland, the known tin, copper and tungsten mineralisation occurs within a series of sedimentary rocks belonging to Hall's Formation (see Figure 8). Hall's Formation is underlain by the sandstone and overlain by volcanics. All these sedimentary rocks are of Cambrian age and, while they were originally deposited horizontally, they have been tilted and are now more or less vertical.

The sedimentary rocks were intruded by the Devonian-Carboniferous Meredith granite. A quartz porphyry dyke occurs in the bottom of the mine below 350m from the surface.

The mineralisation at Cleveland is of two styles:

- Tin and copper bearing semi-massive sulphide lenses consisting of pyrrhotite and pyrite with cassiterite and lesser chalcopyrite and stannite, and quartz, fluorite and carbonates. Sulphide minerals make up 20% to 30% of the mineralisation.
- Tungsten bearing quartz stock-work and minor greisen with wolframite.

The semi-massive sulphide has formed by the replacement of limestone and is geologically similar to the tin bearing semi-massive and massive sulphide mineralisation at Renison.

The known tin and copper bearing semi-massive sulphide mineralisation occurs entirely within Hall's Formation, as faulted, more or less vertically dipping, lenticular deposits with strike lengths of up to 550 metres, across strike thicknesses of up to 30 metres and down-dip extents of up to 800 metres (see Figure 9). The deposits are referred to as lodes, and from east to west, these are:

- Battery, Smithy and Luck's Lodes,
- Khaki Lode
- Hall's Lodes,
- Henry's Lode,

The tungsten bearing stock-work formed around and possibly from the quartz porphyry dyke. The dyke dips vertically and has a known strike length of 100m, an across strike thickness of up to 60m and a down-dip extent of 800 metres (Jackson et al., 2000).

The tungsten bearing quartz stock work formed by the intense quartz veining which forms a halo around the quartz porphyry dyke.

The tungsten bearing quartz stock-work is known as Foley's Zone (see Figure 10). Foley's Zone is currently considered to dip vertically and has a known strike length of about 300 metres, an across strike width of up to 300 metres and a down-dip extent of about 900 metres (Dronseika, 1983).

The quartz stock work and the semi-massive sulphide mineralisation overlap in part although the quartz stock-work contains very low tin grades, generally less than 0.2% Sn.

Modelling of the granite, based on geophysical gravity survey, indicates that the top of the granite is nearly 4 kilometres deep at Cleveland (see Figure 11) (Leaman and Richardson, 1989 and 2003). This is important because

- at Renison, the tin mineralisation continues to the granite,
- in Cornwall, tin mineralisation continues into the granite, and
- the granite is deeper than thought by explorers from the 1960s to the 1980s.

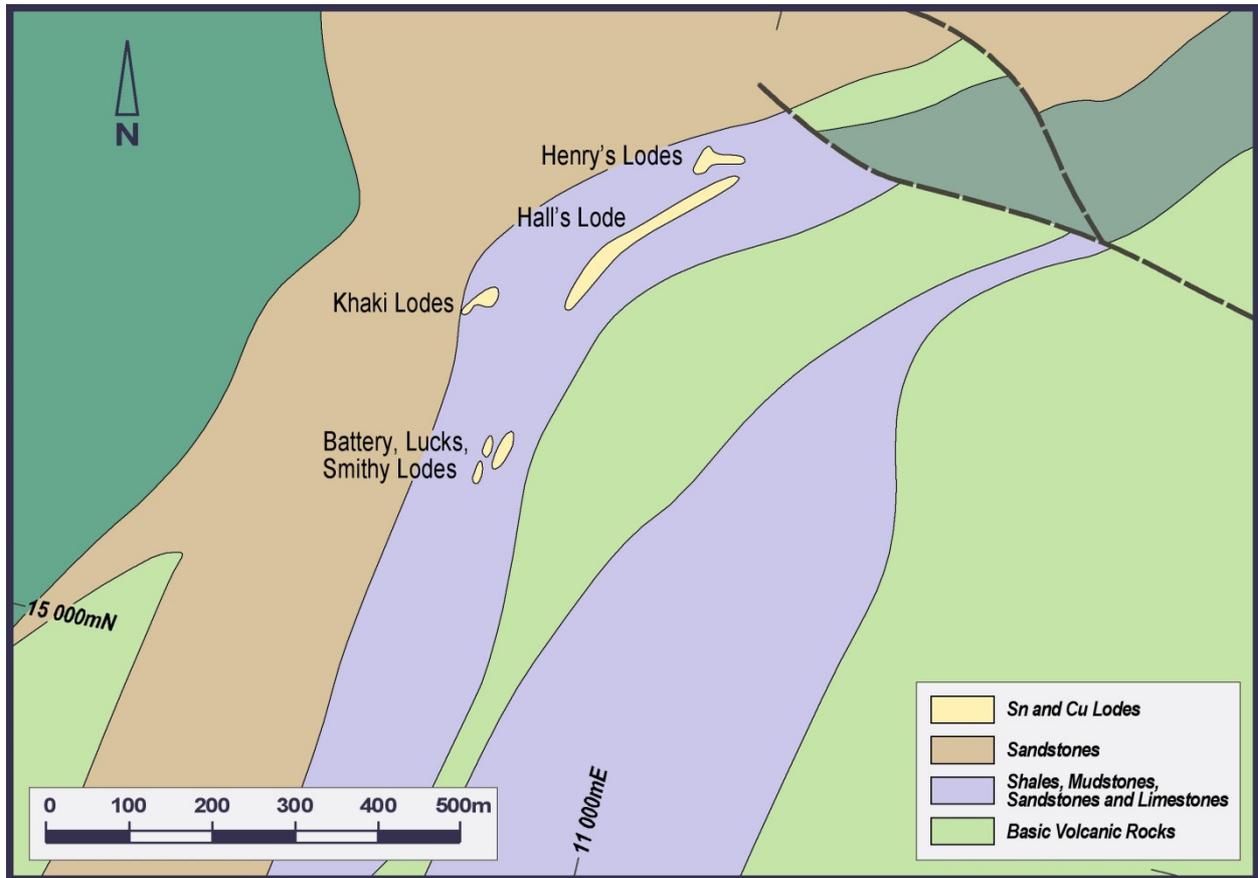


Figure 8: Geology near the Cleveland Mine

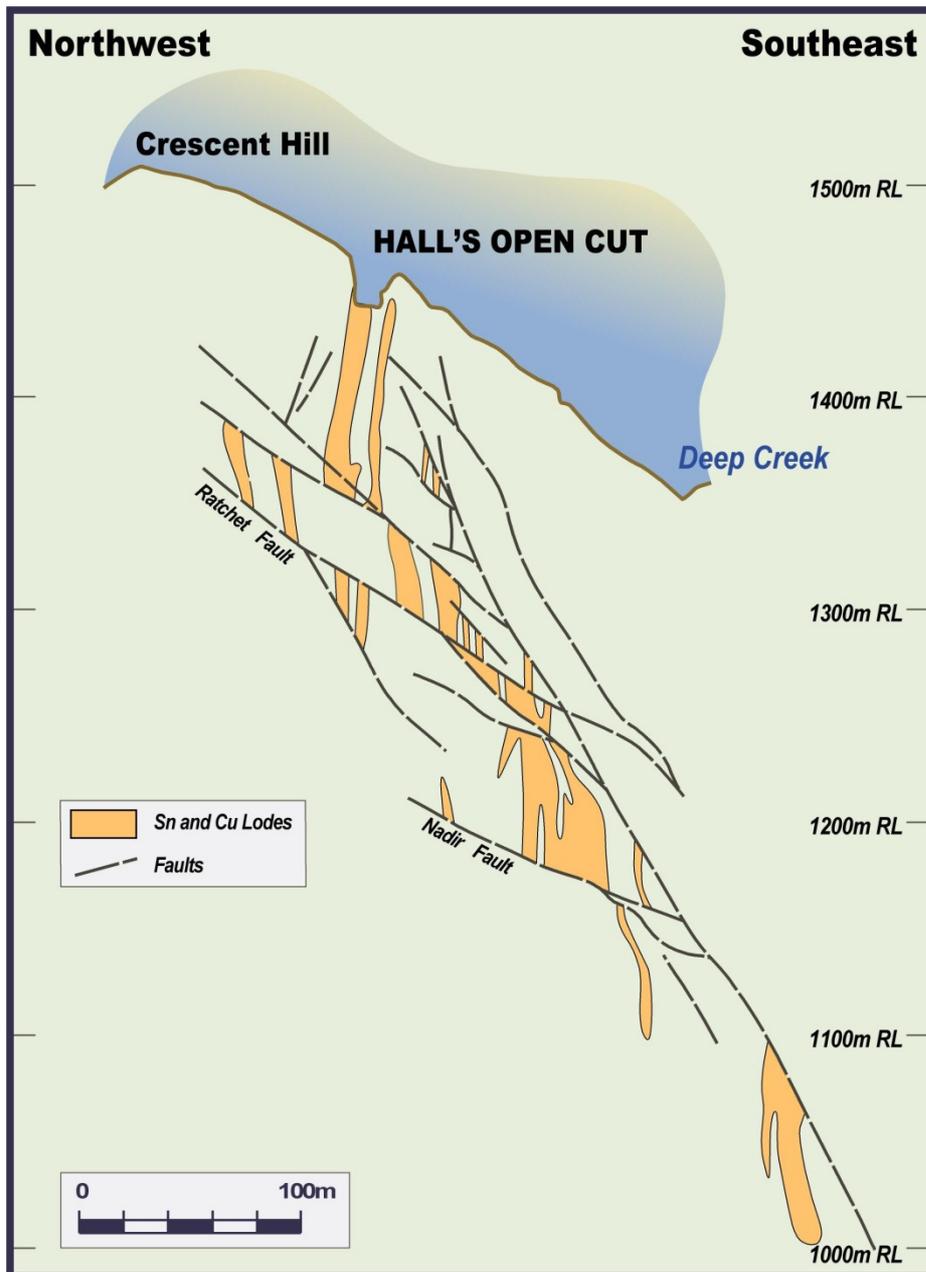


Figure 9: Tin Copper bearing Semi-massive Sulphide Mineralisation on Cross-section Number "N"
(after Collins et al., 1989)

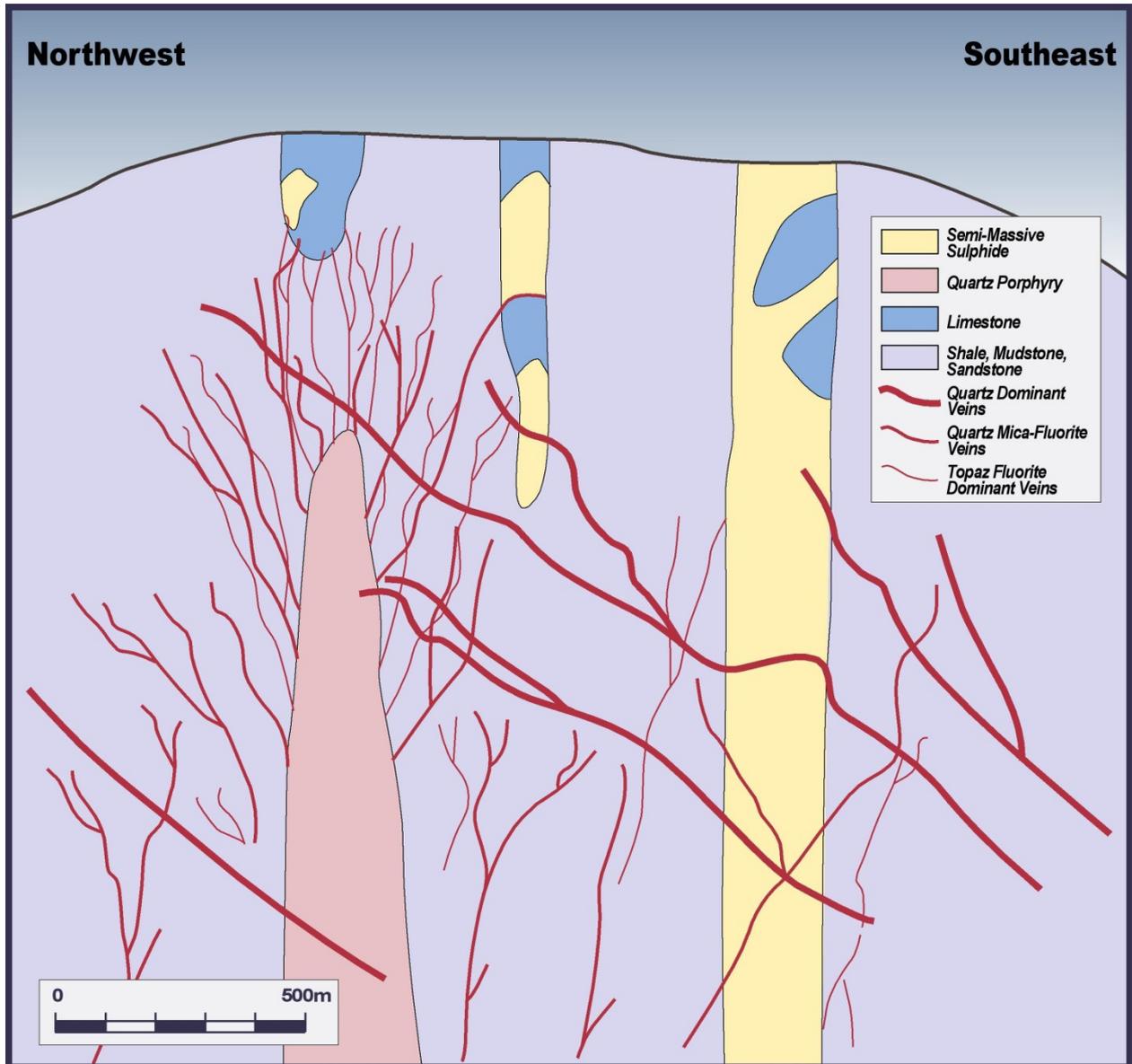


Figure 10: Schematic Representation of Foley's Zone – Cross Section
(After Jackson et al., 2000)

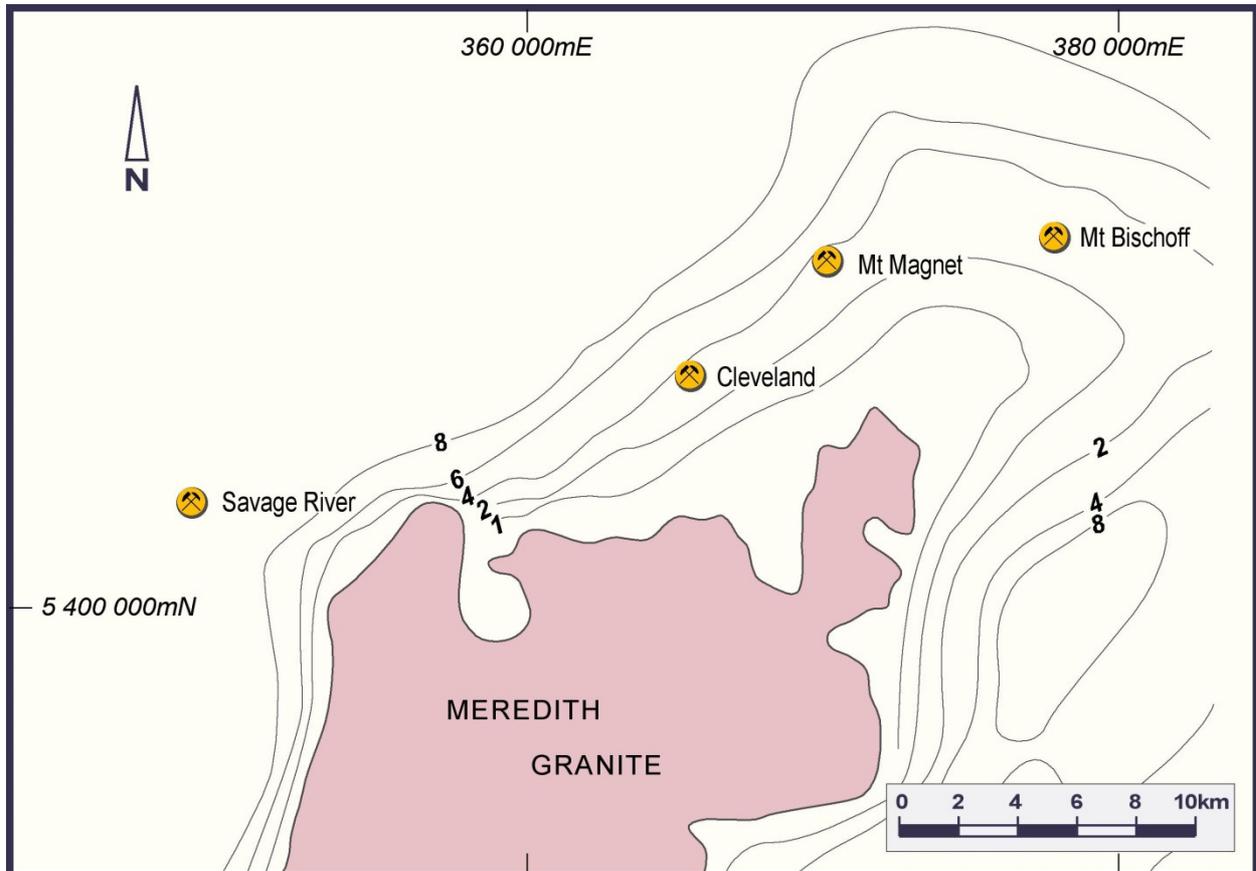


Figure 11: Depth to Granite in the Cleveland Area
(After Leaman and Richardson, 1989 and 2003)

4.7 Cleveland Ore

The principal aim of processing at Cleveland was the recovery of tin rather than copper. This was because the tin price was then, as now, significantly greater than the copper price and the tin grade of the ore was greater than the copper grade (see Table 5).

The cassiterite in the semi-massive sulphide mineralisation is fine grained with grains generally being in the range 0.02mm to 0.07mm across. The fine grain size dictates the extent to which the ore must be ground to release the cassiterite from the other minerals present so that it can then be recovered. As the grind size becomes finer, the cassiterite becomes more difficult to recover in a traditional processing plant which uses gravity and flotation methods to recover the cassiterite. During the Aberfoyle operations at Cleveland, tin and copper recovery both averaged about 60% (see Table 5).

**Table 5: Cleveland Mine – Mill Performance during Aberfoyle Operations
(Compiled from Aberfoyle Ltd and Cleveland Tin N.L. annual reports.)**

Year	Ore Treated tonnes	Feed Grade		Sn Recovered from Cassiterite		Cu Recovered	
		% Sn as cassiterite	% Cu	tonnes	recovery	tonnes	recovery
1969	256,865	0.85	0.40	1,136	52%	378	37%
1970	281,875	0.79	0.37	1,472	66%	717	69%
1971	305,726	0.73	0.42	1,557	69%	823	65%
1972	357,498	0.78	0.40	1,874	67%	969	69%
1973	505,806	0.76	0.32	2,668	69%	1216	76%
1974	314,210	0.75	0.25	1,513	65%	529	69%
1975	289,018	0.78	0.32	1,370	60%	565	61%
1976	363,036	0.73	0.27	1,519	60%	617	62%
1977	393,275	0.66	0.22	1,386	54%	420	49%
1978	388,579	0.53	0.18	1,236	60%	458	65%
1979	352,977	0.52	0.24	1,106	60%	422	49%
1980	367,866	0.47	0.21	1,080	63%	414	53%
1981	439,304	0.51	0.22	1,337	60%	518	55%
1982	350,300	0.64	0.28	1,457	65%	602	60%
1983	277,700	0.71	0.25	1,182	60%	405	59%
1984	180,300	0.71	0.26	668	53%	300	64%
1985	137,000	0.80	0.25	525	48%	198	59%
1986	83,700	0.92	0.41	434	56%	140	41%
Total	5,645,035	0.68	0.28	23,519	61%	9,691	60%

5 THE DATABASE

5.1 Existing Data

Most of the information used for this report was compiled by Aberfoyle and associated parties over the period 1968 to 1986. There is a large repository of data and information regarding the Cleveland Mine which was accumulated by Aberfoyle and which is now held at the offices of the Burnie Research Lab, 39 River Road, Burnie, Tasmania. A catalogue of this data was made by Rockwell and listed in a spreadsheet named “110723 BPRL Cleveland archive catalogue – Master.xlsx”

The Aberfoyle data includes drill logs, maps, reports and survey information. Drill logs exist for all holes drilled for geological purposes. A full set of 1:500 working geological cross-sections exists for the mine.

MRT holds reports regarding exploration activity at and around Cleveland from around 1900 to the present day. These are all available via the Tiger Database on the MRT website. Many of these reports refer directly to the Cleveland Mine. Amongst the reports held by MRT is a report of the remaining resources made at mine closure (Dronseika, 1986).

MRT also holds many paper mine plans and sections. Of particular note is a set of plans and cross-sections showing the mine workings and bearing the date 1982. Despite the date on the plans, the plans appear to show the mine workings at mine closure, although this still requires clarification. The mine plans show all development outlines and relevant survey station locations and numbers but not RLs. However, a file of survey station numbers with RLs was discovered in the Aberfoyle data held in Burnie; this data is considered critical and has been entered into digital form.

MRT also holds drill core for 87 diamond drill holes at its Core Store at Mornington, Tasmania. For this report, samples taken from this core were used to verify the Aberfoyle assay data (see 5.2 Reliability of the Aberfoyle Data).

5.2 Reliability of the Aberfoyle Data

The Aberfoyle data was compiled during Aberfoyle’s ownership of Cleveland from 1961 to 1986. During that time, Aberfoyle drilled 2040 diamond drill holes into the Cleveland deposit and mined about 5.65 million tonnes of ore over a period of 18 years, from 1968 to 1986. In the 1970s, Aberfoyle was one of the leading tin producers in Australia with four operating tin mines: Cleveland, the Storeys Creek and Aberfoyle mines in North-East Tasmania, and the Ardlethan mine in southern New South Wales. The author worked for Aberfoyle from 1970 to 1973 and was acquainted with many of the Aberfoyle staff who worked at Cleveland. He knows of no reason to doubt the technical competence of the geologists who worked at Cleveland, in fact, Aberfoyle personnel were recognised as efficient operators with good record keeping and reporting practices.

During the work for this report, only one significant shortcoming in the data was discovered: only 1,547 of the drill logs included registered coordinates of the collars. However, it appears that all holes were plotted on the 1:500 working sections and, for the work for this report, the coordinates of about 300 of the 493 holes which did not have registered coordinates were measured from the cross-sections.

Corollary validation of the data as it was used for the work for this report has confirmed the reliability of much of the data, for example, the collars of diamond drill holes drilled from the

surface plot on the surface digital terrain model. There are discrepancies between the digital database and the data presented on the Aberfoyle working sections and these are discussed below (see 5.3 Digital Database).

An assessment of the validity of the Aberfoyle assay data was made by quartering some of the core held in the MRT Core Store and submitting the samples for assay at the Burnie Research Lab, a division of AMMTEC Ltd and part of the ALS Group – a diversified international analytical laboratory group present in 40 countries through a network of over 140 laboratories. Staff at the Burnie Research Lab are recognised experts in sample preparation and assaying for Sn.

111 samples were prepared by quartering drill core from parts of drill holes which had previously been split, sampled and assayed by Aberfoyle. The samples covered a range of Sn assays from about 0% to about 4% Sn. Sn content was determined by fused bead XRF, soluble Sn content by acid digest and AAS, and Cu content by AAS.

The length weighted average of the re-splits Sn assays was 0.84% compared with the length weighted average of the Aberfoyle Sn assays of 0.78%. As expected with smaller samples, the variance of the re-splits Sn assays was higher than the variance of the Aberfoyle Sn assays: 0.59 %² compared with 0.44%² (the “support effect”).

A graph of the two sets of Sn assays shows reasonable correlation between the two data sets with no obvious systematic error (see Figure 12). Note that this is not a graph of re-assays of the same samples but a graph of samples taken, as far as possible and within the limits of the depth records in the core trays, adjacent to each other. Given these factors these results are a good confirmation of the reliability of the Aberfoyle Sn assay data.

The length weighted average of the re-splits Cu assays was 0.28% the same as the length weighted average of the Aberfoyle Cu assays. As expected with smaller samples, the variance of the re-splits Cu assays was higher than the variance of the Aberfoyle Cu assays: 0.21%² compared with 0.13%² (the “support effect”).

A graph of the two sets of Cu assays shows reasonable correlation between the two data sets with no obvious systematic error (see Figure 13). Note, again, that this is not a graph of re-assays of the same samples but a graph of samples taken, as far as possible and within the limits of the depth records in the core trays, adjacent to each other. Given these factors these results are a good confirmation of the reliability of the Aberfoyle Cu assay data.

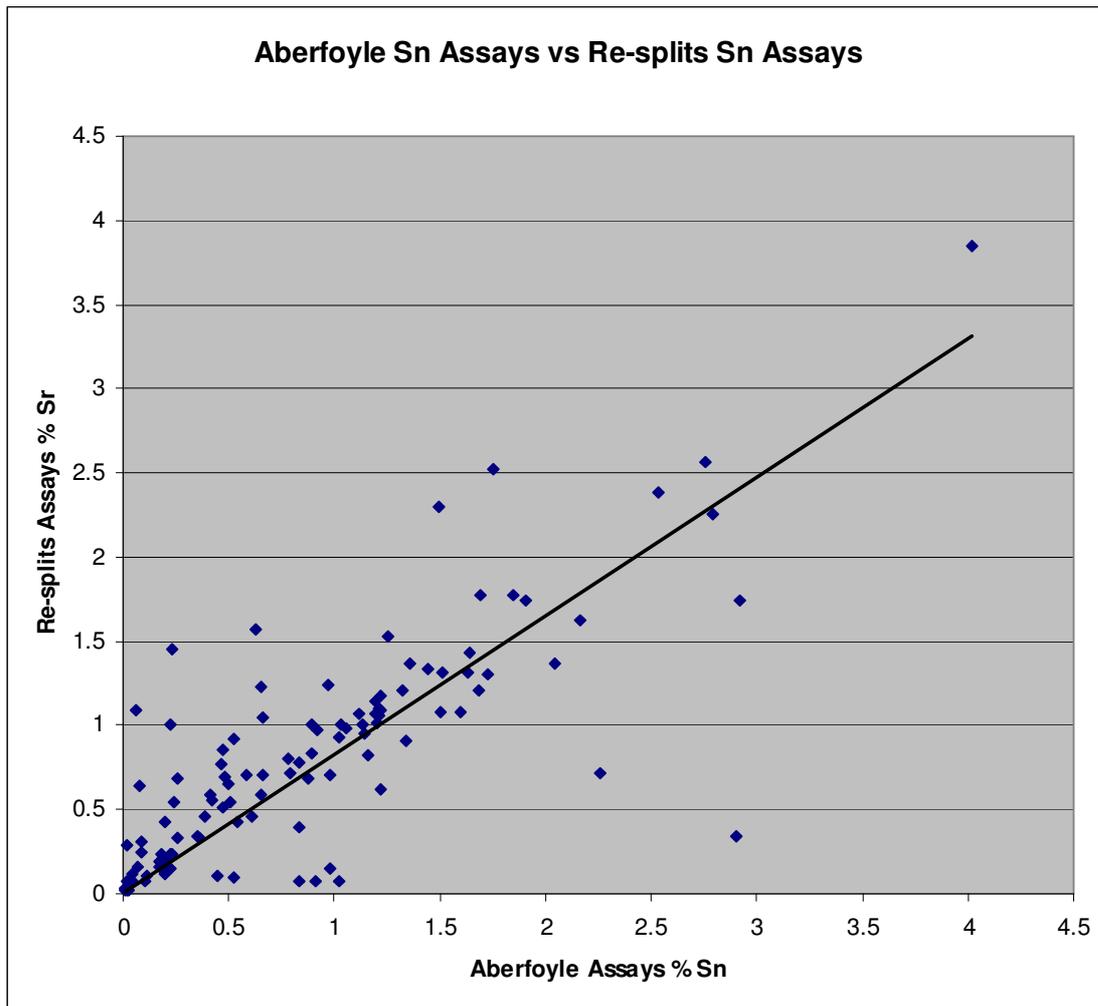


Figure 12: Aberfoyle Sn Assays vs Re-splits Sn Assays
(111114 Cleveland drill logs – check assay comparison_MM.xls)

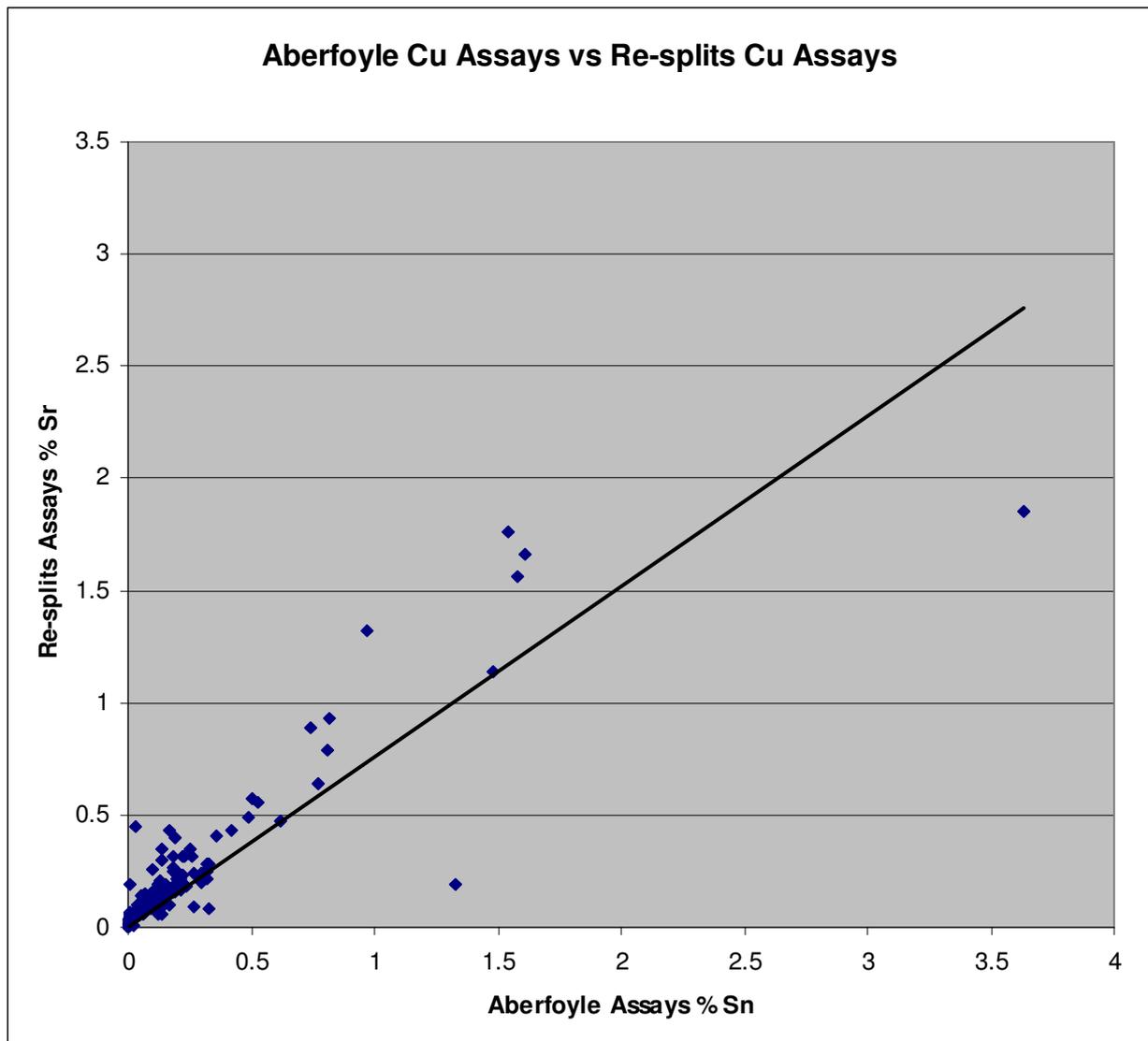


Figure 13: Aberfoyle Cu Assays vs Re-splits Cu Assays
(111114 Cleveland drill logs – check assay comparison_MM.xls)

5.3 Digital Database

Creation of a digital database of the Aberfoyle data is continuing and the Aberfoyle data digitised so far includes:

- diamond drill hole collar locations;
- diamond drill hole surveys;
- assays of diamond drill core;
- lode intercepts, that is from depths and to depths of lodes as recorded by Dronseika in his 1986 resource report (Dronseika, 1986);
- mined out perimeters as recorded by Dronseika in his 1986 resource report (Dronseika, 1986);
- surface contours over the mine area;

- the location of the Cleveland Decline; this is correct in plan view but levels are approximate only.

5.4 Reliability of the Data for the Tin and Copper Estimate

The nature of defects in the sampling techniques and data for this resource estimate and their likely impacts on the quality of the resource report are listed in Table 6. The Assessment and Resource Reporting Criteria listed in Table 7 have been categorised using the terms listed under Nature of Defects.

Table 6: Nature of Defects - Likely Impact on Resource Report Quality and Likely Impact on Subsequent Use of the Resource Report

Nature of Defects	Likely Impact on Resource Report Quality	Likely Impact on Subsequent Use of Resource Report
Acceptable: no defects	None	None
Defects tolerable	Low	No material impact
Defects not acceptable	Quality severely compromised	Severely compromised

Table 7: Cleveland Sn and Cu Resource Estimate – Sampling Techniques and Data

Cleveland Sn and Cu Resource Estimate	
Sampling Techniques and Data	
Sample recovery.	<p>Acceptable.</p> <p>A sampling of drill logs by the author did not reveal that core loss was a problem during diamond drilling. The reliability of core recovery was confirmed in discussions with a contemporary Aberfoyle geologist. Aberfoyle reported that core recovery at Cleveland was consistently good (Cox, 1967). This is in accordance with the reported ground conditions in the Cleveland mine which have been reported as competent to highly competent (Everett, 1977) and Buckland, 1980).</p>
Logging.	<p>Acceptable to tolerable.</p> <p>A sampling of drill logs by the author indicated that the logs contained adequate locational, sampling and assay data. Lithological logging was not always carried out but, given the nature of the mineralisation, even though not ideal, this lack is tolerable.</p>
Sub-sampling techniques and sample preparation.	<p>Acceptable.</p> <p>Drill core was split longitudinally and crushing and pulverising were subject to specific and definite protocols. Aberfoyle paid particular attention to sampling technique and sample preparation (Cox, 1967). The reliability of sub-sampling techniques and sample preparation has been confirmed by re-sampling and re-assaying of existing drill core by Rockwell (McKeown, 2011).</p>
Quality of assay data and laboratory tests.	<p>Acceptable.</p> <p>Assays were conducted at the Tasmanian Mines Department Laboratory at Launceston and at Cleveland and check samples, although not recorded in the drill logs, were in use (Cox, 1967). The reliability of the assays is also partly confirmed by reconciliations of resources to production (Dronseika, 1986). The reliability of Sn assays has been confirmed by re-sampling and re-assaying of existing drill core by Rockwell (McKeown, 2011).</p>
Verification of sampling and assaying.	<p>Tolerable.</p> <p>Some verification of assay data was carried out routinely. Check samples, although not recorded in the drill logs, were in use (Cox, 1967). The reliability of the assays is also partly confirmed by reconciliations of resources to production (Dronseika, 1986).</p>

Location of data points.	<p>Tolerable.</p> <p>Locations of drill hole collars, channel samples and mine workings were established by Surveyors. However, about 25% of holes did not have collar coordinates and these will have to be assumed from the available cross-sections. Also, it has become evident from the work for this report that the down-hole surveys need to be reviewed, especially for long holes.</p>
Data spacing and distribution.	<p>Acceptable.</p> <p>Data spacing was sufficient for estimation of grades by inverse distance e squared method and for classification according to the JORC Code. Generally, geological interpretation did not extend more than half a cross-section (approximately 10m) beyond diamond drill hole intercepts.</p>
Orientation of data in relation to geological structure	<p>Acceptable.</p> <p>The strike and dip of the host geological formation and of the sulphide cassiterite mineralisation was well known from the beginning of systematic evaluation by Aberfoyle in 1961 and the drill holes were generally angled accordingly.</p>
Audits or reviews.	<p>Acceptable.</p> <p>There are no known audits or reviews by personnel outside Aberfoyle. However, there was a culture of internal reviewing of the geological procedures including at least one review of sampling methods (Cox, 1967).</p>

5.5 Reliability of the Data for the Tungsten Estimate

The nature of defects in the sampling techniques and data for this resource estimate and their likely impacts on the quality of the resource report are listed in Table 8. The Assessment and Resource Reporting Criteria listed in Table 9 have been categorised using the terms listed under Nature of Defects.

Table 8: Nature of Defects - Likely Impact on Resource Report Quality and Likely Impact on Subsequent Use of the Resource Report

Nature of Defects	Likely Impact on Resource Report Quality	Likely Impact on Subsequent Use of Resource Report
Acceptable: no defects	None	None
Defects tolerable	Low	No material impact
Defects not acceptable	Quality severely compromised	Severely compromised

Table 9: Cleveland WO₃ Resource Estimate – Sampling Techniques and Data

Cleveland WO₃ Resource Estimate	
Sampling Techniques and Data	
Sample recovery.	<p>Acceptable.</p> <p>A sampling of drill logs by the author did not reveal that core loss was a problem during diamond drilling. The reliability of core recovery was confirmed in discussions with a contemporary Aberfoyle geologist. Aberfoyle reported that core recovery at Cleveland was consistently good (Cox, 1967). This is in accordance with the reported ground conditions in the Cleveland mine which have been reported as competent to highly competent (Everett, 1977) and Buckland, 1980) and, in the quartz porphyry host rock, as excellent (Dronseika, 1983).</p> <p>Core recovery in the WO₃ mineralisation was in excess of 95% (Dronseika, 1983).</p>
Logging.	<p>Acceptable.</p> <p>5500m of core was logged in detail noting country rock, wall-rock alteration, structures, mineralogy, vein thickness and vein to core angle (Dronseika, 1983).</p> <p>A sampling of drill logs by the author indicated that the logs contained adequate locational, geological, sampling and assay data.</p> <p>In addition, there are 64 petrological and mineralogical descriptions made under the microscope by AMDEL and Latrobe University (included in Dronseika, 1983).</p>
Sub-sampling techniques and sample preparation.	<p>Acceptable.</p> <p>Drill core was split longitudinally and crushing and pulverising were subject to specific and definite protocols. Aberfoyle paid particular attention to sampling technique and sample preparation (Cox, 1967).</p>
Quality of assay data and laboratory tests.	<p>Acceptable.</p> <p>Samples were routinely assayed in the laboratory at Cleveland. Thirty samples were re-split and re-assayed by AMDEL Laboratories. Some samples were re-assayed by AMDEL Laboratories. The correlation of assay results for WO₃ was acceptable (Hample and Waters, 1983).</p>
Verification of sampling and assaying.	<p>Acceptable.</p> <p>Samples were routinely assayed in the laboratory at Cleveland. Thirty samples were re-split and re-assayed by</p>

	AMDEL Laboratories. Some samples were re-assayed by AMDEL Laboratories. The correlation of assay results for WO ₃ was acceptable (Hample and Waters, 1983).
Location of data points.	Acceptable. Locations of drill hole collars, channel samples and mine workings were established by Surveyors.
Data spacing and Distribution.	Acceptable. Data spacing was sufficient for creation of useable WO ₃ variograms with relatively low nugget effect and ranges for spherical models of up to 150m (McArthur, 1983 in Dronseika, 1983).
Orientation of data in relation to geological structure	Acceptable. The strike and dip of the quartz porphyry intrusion and the quartz vein stock-work mineralisation was well known from the beginning of systematic evaluation by Aberfoyle in 1970 and the drill holes were oriented accordingly.
Audits or reviews.	Tolerable. There are no known audits or reviews by personnel outside Aberfoyle. However, there was a culture of internal reviewing of the geological procedures including at least one review of assaying methods (Hample and Waters, 1983).

5.6 Reliability of the Data for the Tin and Copper Tailings

The nature of defects in the sampling techniques and data for this resource estimate and their likely impacts on the quality of the resource report are listed in Table 10. The Assessment and Resource Reporting Criteria listed in Table 11 have been categorised using the terms listed under Nature of Defects.

Table 10: Nature of Defects - Likely Impact on Resource Report Quality and Likely Impact on Subsequent Use of the Resource Report

Nature of Defects	Likely Impact on Resource Report Quality	Likely Impact on Subsequent Use of Resource Report
Acceptable: no defects	None	None
Defects tolerable	Low	No material impact
Defects not acceptable	Quality severely compromised	Severely compromised

Table 11: Cleveland Tailings Resource Estimate – Sampling Techniques and Data

Cleveland Tailings Resource Estimate	
Sampling Techniques and Data	
Sample recovery.	Acceptable. The tailings grade is based on sampling in the Cleveland Mill and subsequent metallurgical mass balances from 1968 to 1986.
Logging.	Not applicable.
Sub-sampling techniques and sample preparation.	Acceptable. Sampling in the Cleveland Mill was subject to metallurgical mass balances from 1968 to 1986.
Quality of assay data and laboratory tests.	Acceptable. Samples were taken routinely in the Cleveland Mill and routinely assayed in the laboratory at Cleveland. Assaying in the Cleveland Mill was subject to metallurgical mass balances from 1968 to 1986.
Verification of sampling and assaying.	Acceptable. Samples were taken routinely in the Cleveland Mill and routinely assayed in the laboratory at Cleveland. Assaying in the Cleveland Mill was subject to metallurgical mass balances from 1968 to 1986. Drilling of the tailings dams in 2008 confirmed the grade of the tailings (Brewer, 2008).
Location of data points.	Not applicable.
Data spacing and Distribution.	Acceptable. Sampling in the Cleveland Mill was routine and subject to metallurgical mass balances from 1968 to 1986. A very large number of tailings samples were taken during that time, probably at least one per day from 1968 to 1986.
Orientation of data in relation to geological structure	Not applicable. However, drilling in 2008 suggested that there is layering in the dams.
Audits or reviews.	Acceptable. Aberfoyle made estimates of tonnage and grade of tailings made in 1981 (Foo, 1981) which were confirmed in 2008 (Moony, 2008).

6 GEOLOGICAL INTERPRETATION

Aberfoyle geologists described the tin and copper bearing deposits at Cleveland as lodes. For the interpretations for this report, before the interpretations were made, no assumptions were made regarding the origin of the lodes.

For this report, the interpretation of the lenses of mineralisation on each cross-section was based on:

- Aberfoyle's geology cross-sections: these exist as paper cross-sections showing drill holes and development geology coloured for lithology. These were drawn during the period of operation ending in 1986.
- Interpretations by Aberfoyle geologists: interpreted lode boundaries are shown in pencil on the Aberfoyle cross-sections.
- Dronseika's list of lode intercepts: Dronseika (1986) included a list of lode intercepts in drill holes which included depths from and to and lode name.
- Sn assays in drill holes: the Aberfoyle cross-sections did not show any assays so the Aberfoyle lode interpretations were based entirely on lithology. This may seem to be a geologically sound approach but, unfortunately, many holes drilled towards the end of the mine life do not have lithology plotted on the Aberfoyle cross-sections. In any event, Sn assays must be indicators of the locations of tin lenses. The use of Sn assays must be tempered by the fact that in hydrothermal deposits tin mineralisation will leak outside the main zones of mineralisation.
- The extent of tin assaying in drill holes: Aberfoyle geologist's selected likely looking intersections for assaying for Sn, so, the extent of Sn assaying is an indication of the logging geologist's opinion of the extent of the Sn bearing rock.
- Aberfoyle's cross-sections showing mine openings: these exist as paper cross-sections showing development and stope openings. The locations of development openings were determined by Aberfoyle's surveyors. Stopes boundaries are depicted by straight lines and are, most likely, based on planned stope outlines. Nearly all stopes were long hole open stopes, left unfilled and, before the invention of cavity monitoring systems ("CMS") there was no way for actual stope shapes to be surveyed. The extent of cross-cuts through lenses is some guide as to the operators' view of the extent of the lenses.
- The interpretation of the lenses on adjacent cross-sections.

A general view of the tin and copper bearing lenses is shown in Figure 14.

The lenses occur in three structural domains separated by two flatly dipping faults: Ratchet fault and Nadir fault. For this report, the upper domain has been designated domain 1, the middle as domain 2 and the lower as domain 3. Lens coding takes this into account, for example, Halls B lens below Ratchet fault but above Nadir fault has the code HLB2.

It is worth noting here that the fault referred to by Aberfoyle geologists as En fault, a fault structurally parallel to sub-parallel to the lodes is almost certainly not a fault but an unconformity.

Foley's stock-work was modelled within a 0.2% WO₃ threshold. Using this threshold value resulted in a bounding shape for the stock-work as shown in Figure 15.

A zone known as Foley's North, located to the north of Aberfoyle's interpretation of Foley's stock-work was identified by Aberfoyle (Dronseika, 1983). Foley's North was based on an intersection in C0969. The interpretation of the stock-work for this report has incorporated all the relevant intersections, including the intersection in C0969, into a single body.

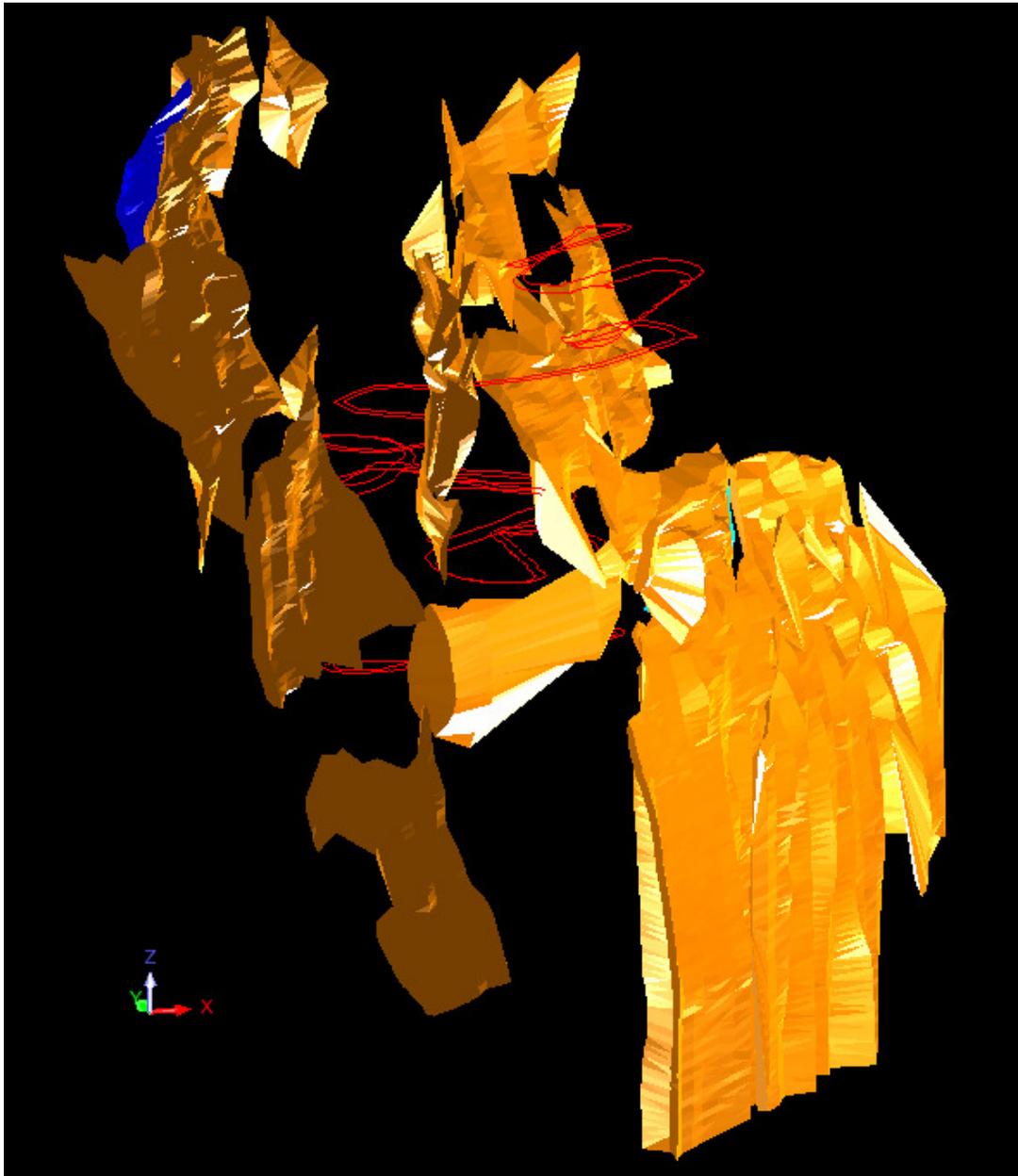


Figure 14: Oblique View of the Tin Copper Lenses Estimated for this Report

This is a view looking north-east. The vertical distance from the top of the upper-most lens to the base of the lower-most lens is about 700m. The Cleveland decline is shown in red.

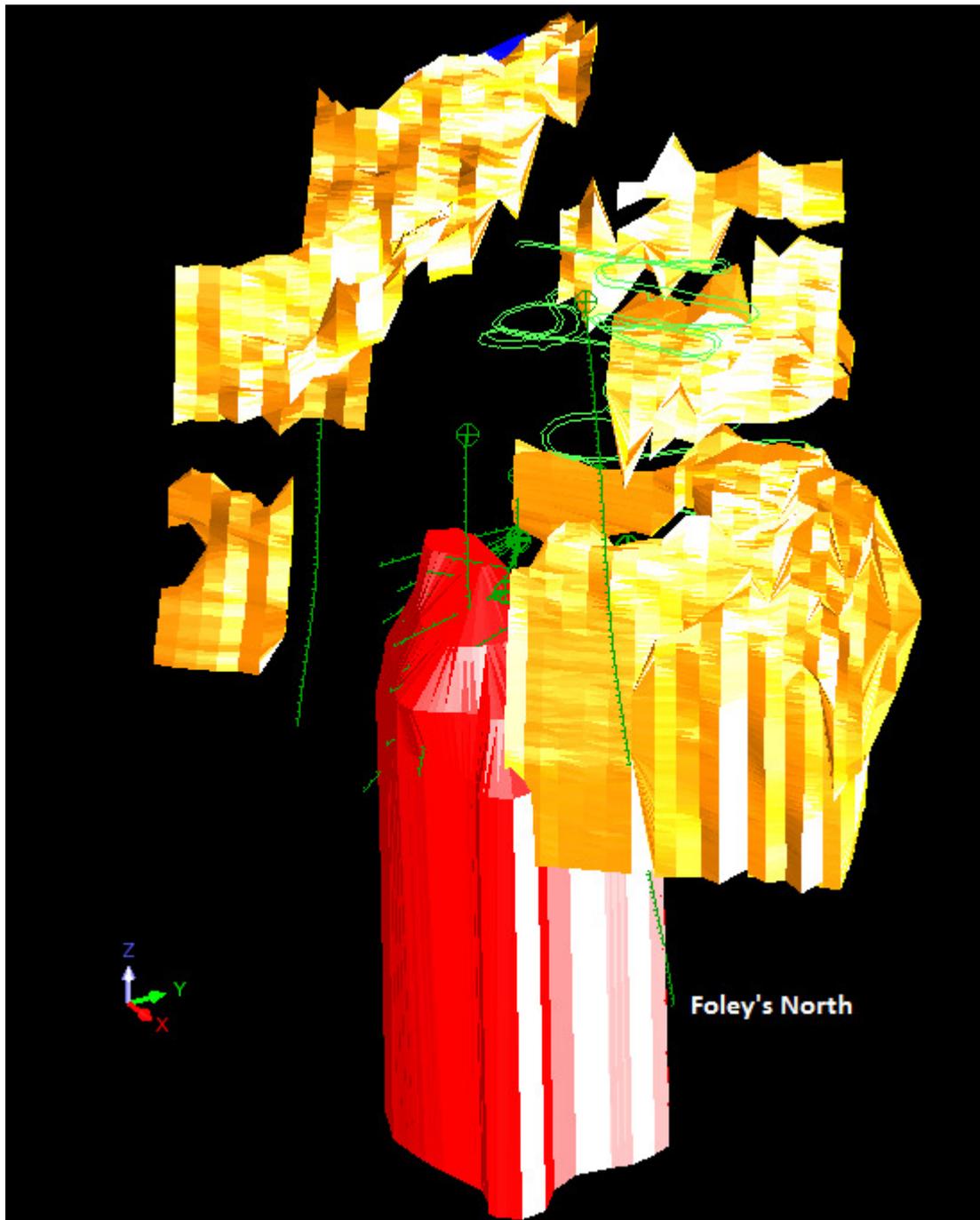


Figure 15: Oblique View of the Tungsten Stock-work Estimated for this Report

This is a view looking north-west. The vertical distance from the top of the upper-most lens to the base of Foley's stock-work is about 1100m. The Cleveland decline and Foley's drill holes are shown in green, and the boundary of Foley's stock-work in red. The location of the area known as Foley's North was just to the left of the annotation.

7 TIN AND COPPER RESOURCE ESTIMATE

7.1 Compositing Length for Tin and Copper Resource Estimation

All assays for the estimate of the Mineral Resources came from sampling of diamond drill core.

A histogram of the sample lengths of the raw assays of all data shows that most assays were taken over lengths of less than 1.0m with the mode of the histogram occurring at 0.8m to 1.0m (see Figure 16).

A compositing length of 1.0m was used for this resource estimate.

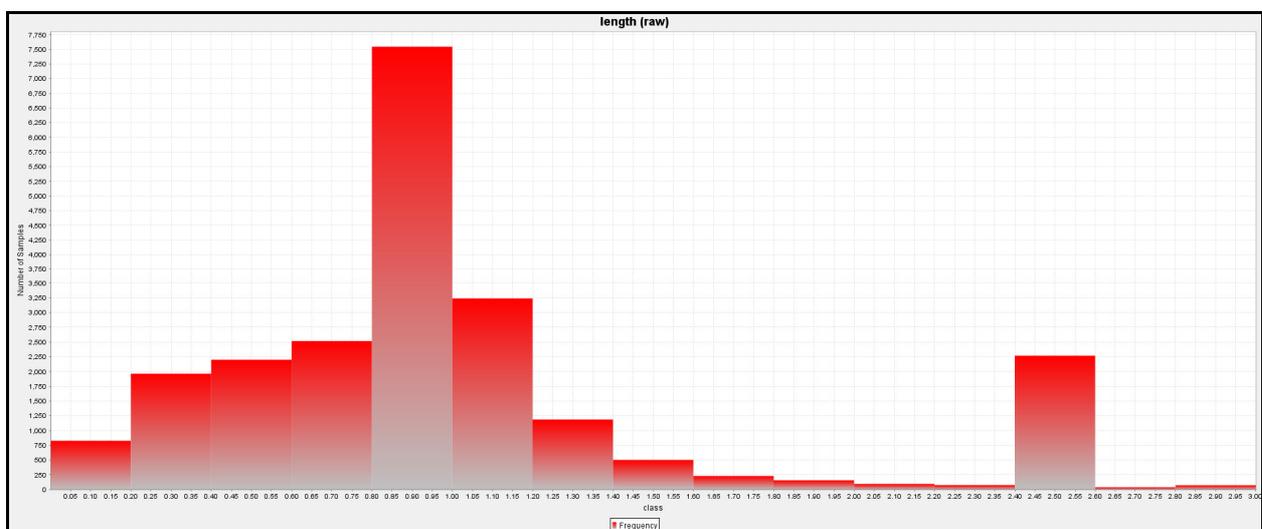


Figure 16: Histogram of all Sample Lengths in the Database

Note that the mode is at 0.8 to 1.0m and bin size is 0.2m. The peak at 2.5m represents the samples from intersections into Foley's stock-work. (samples_all1.str)

7.2 Basic Statistics for Tin and Copper Mineralisation

Histograms, log histograms and log-probability plots for % Sn, % Cu and % Soluble Sn of the composited samples were made (Figures 17 to 25).

As expected, the distributions appear log-normal and the log-probability plots are close to linear suggesting that there is only one population of assays present in the data. In addition, there are no rogue outliers, that is, high grade assays that do not fit the distributions and which consequently indicate the need for cutting of high grades.

There are no significant statistical correlations between % Sn, % Cu and % Soluble Sn (see Table 12).

The mean Sn and Cu grades of the lenses estimated for this report are listed in Table 13.

Table 12: Correlation Coefficients for % Sn, % Cu and % Soluble Sn, all Lode Samples
(comps_all_1.str)

	% Sn	% Cu	% Soluble Sn
% Sn	1	0.30	0.39
% Cu	0.30	1	0.26
% Soluble Sn	0.39	0.26	1

Table 13: Mean Grades of Composited Samples by Lens

Lens	Code	Mean Grades		
		% Sn	% Cu	%Sn soluble
Hall's	HLC1	0.71	0.36	0.05
	HLD1	0.87	0.57	0.10
	HLB2	0.74	0.26	0.06
	HLC2	0.71	0.36	0.11
	HLA3	0.72	0.37	0.04
	HLB3	0.57	0.25	0.03
	HLC3	0.56	0.27	0.02
	HLD3	0.63	0.40	0.02
Henry's	HN31	0.89	0.43	0.05
	HN32	0.79	0.41	0.05
	HN33	0.76	0.30	0.02
Khaki	KK	0.76	0.21	0.06
Luck's	LL	0.73	0.27	0.09
Battery	BT	0.95	0.33	0.05
	BTE	0.69	0.23	0.08
	BTW	0.50	0.13	0.05
B South	BS	0.62	0.26	0.04

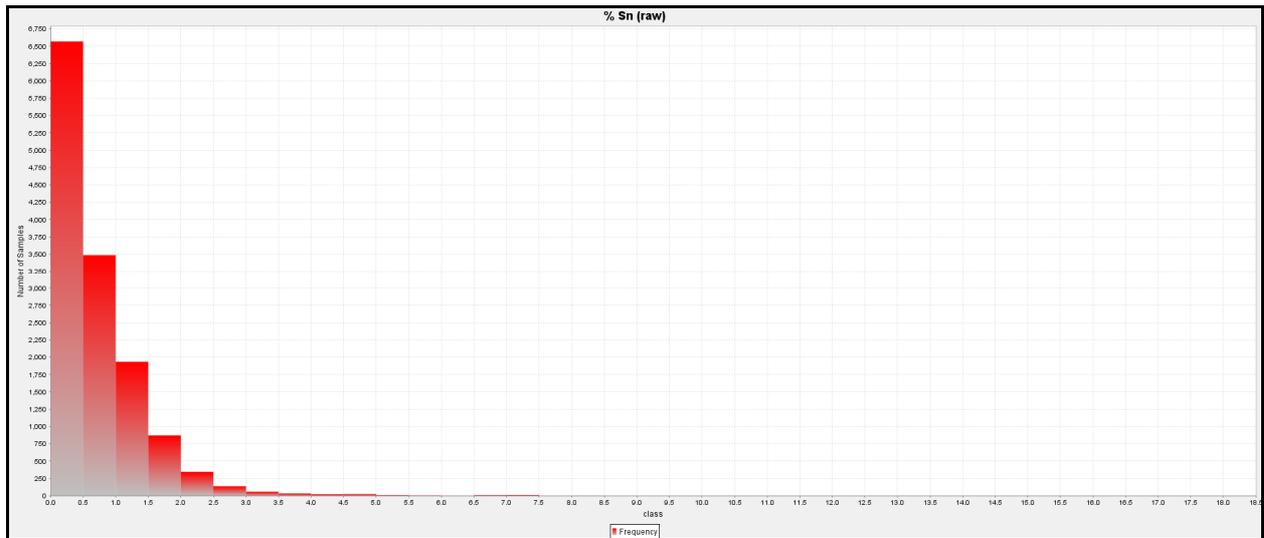


Figure 17: Histogram % Sn all Composited Lode Samples (comps_all_1.str)

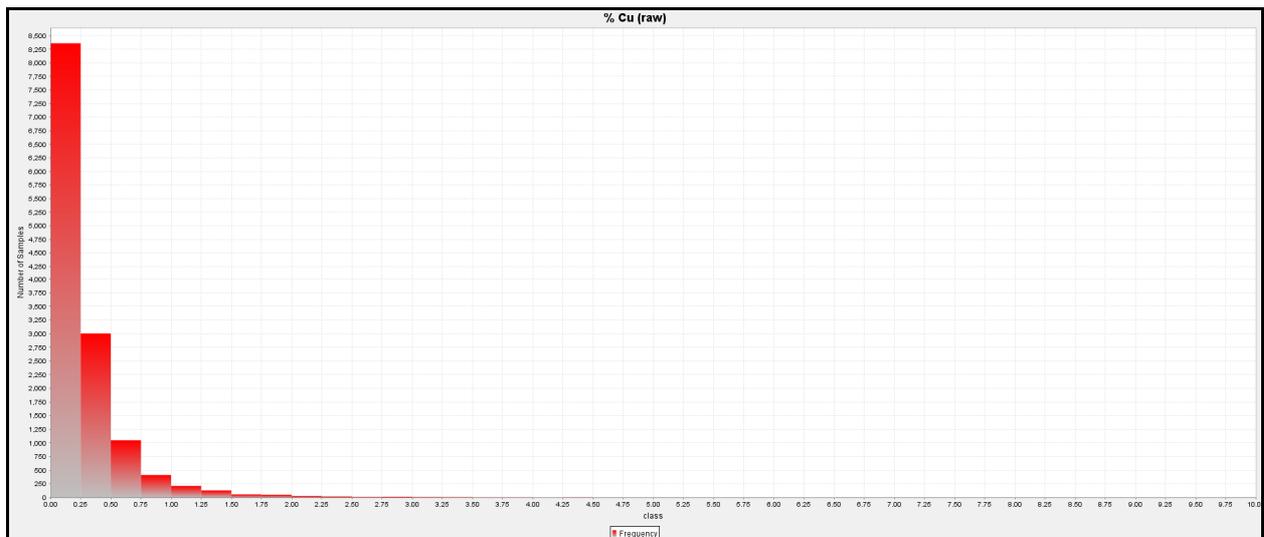


Figure 18: Histogram % Cu all Composited Lode Samples (comps_all_1.str)

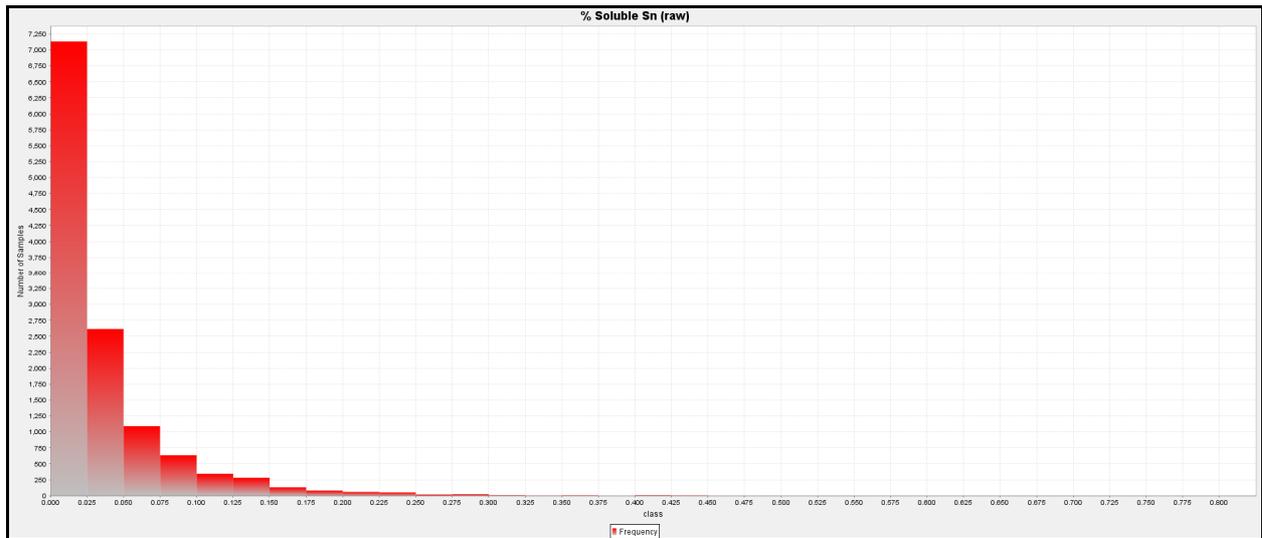


Figure 19: Histogram % Soluble Sn all Compositied Lode Samples (comps_all_1.str)

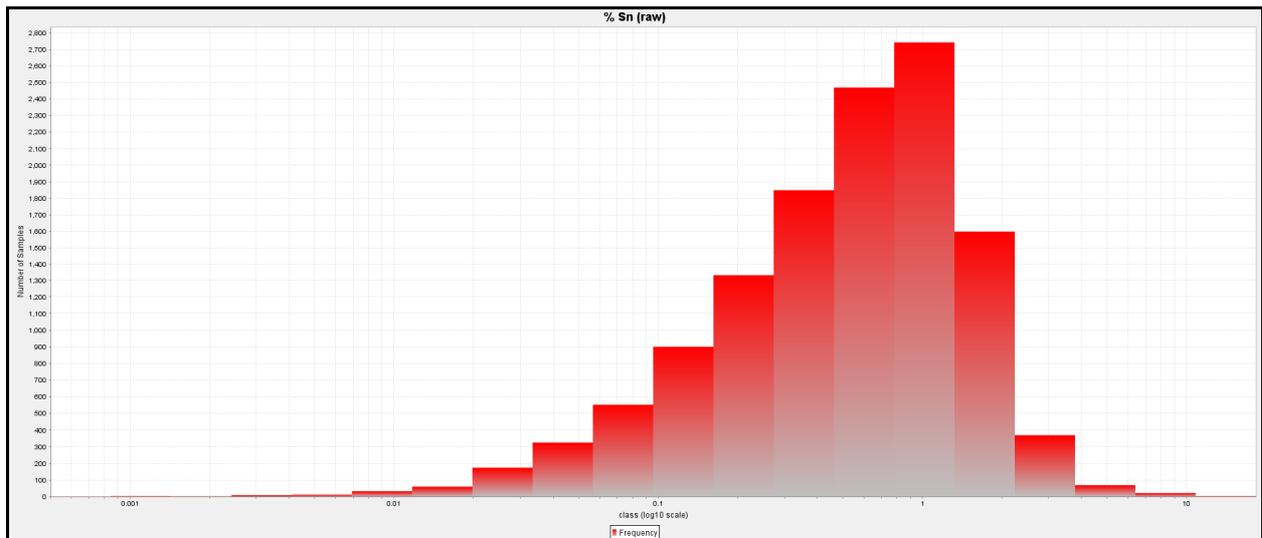


Figure 20: Log-histogram % Sn all Compositied Lode Samples (comps_all_1.str)

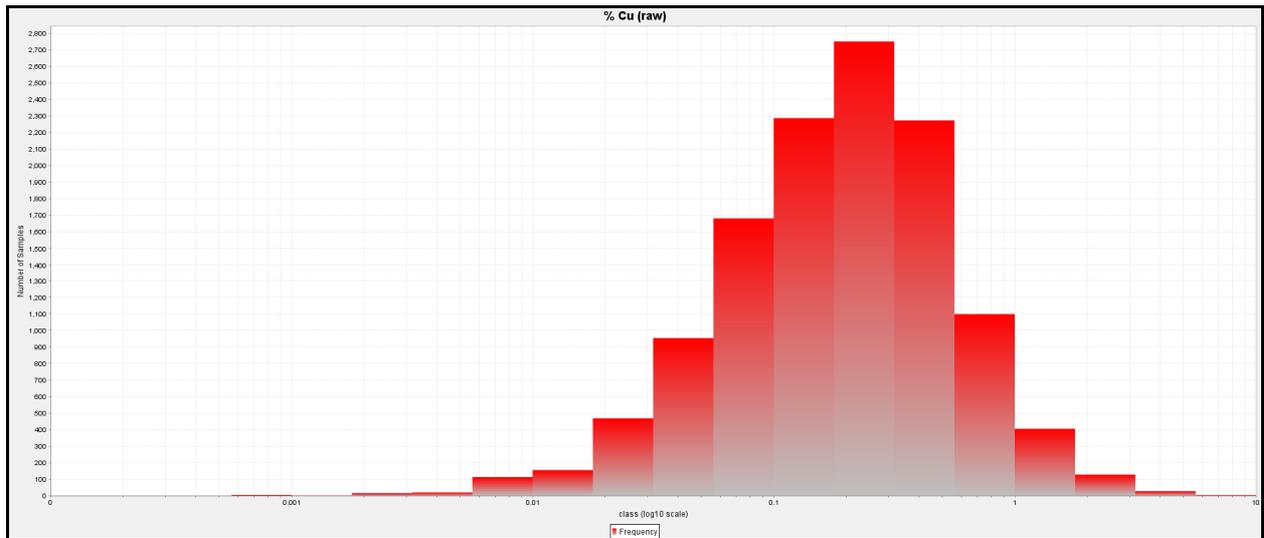


Figure 21: Log-histogram % Cu all Composited Lode Samples (comps_all_1.str)

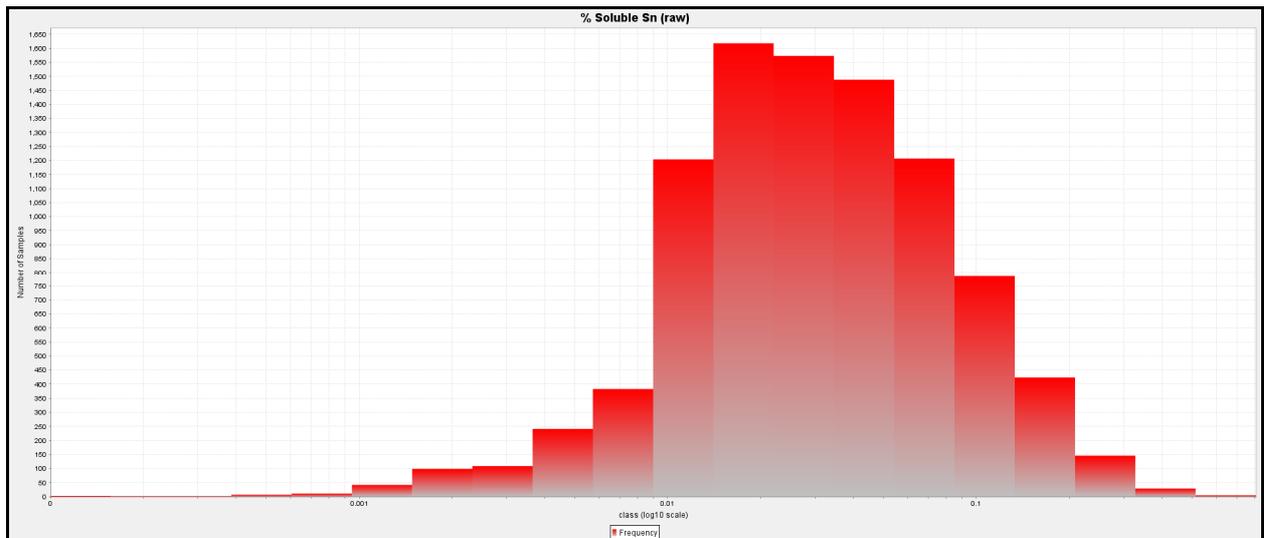


Figure 22: Log-histogram % Soluble Sn all Composited Lode Samples (comps_all_1.str)

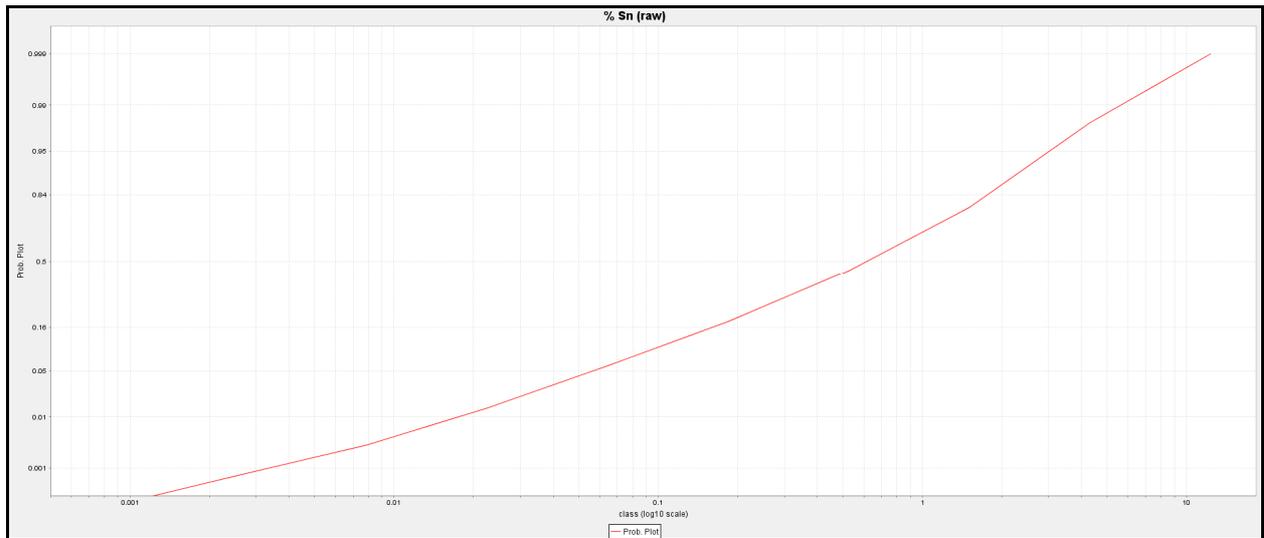


Figure 23: Log-probability Plot % Sn all Composited Lode Samples (comps_all_1.str)

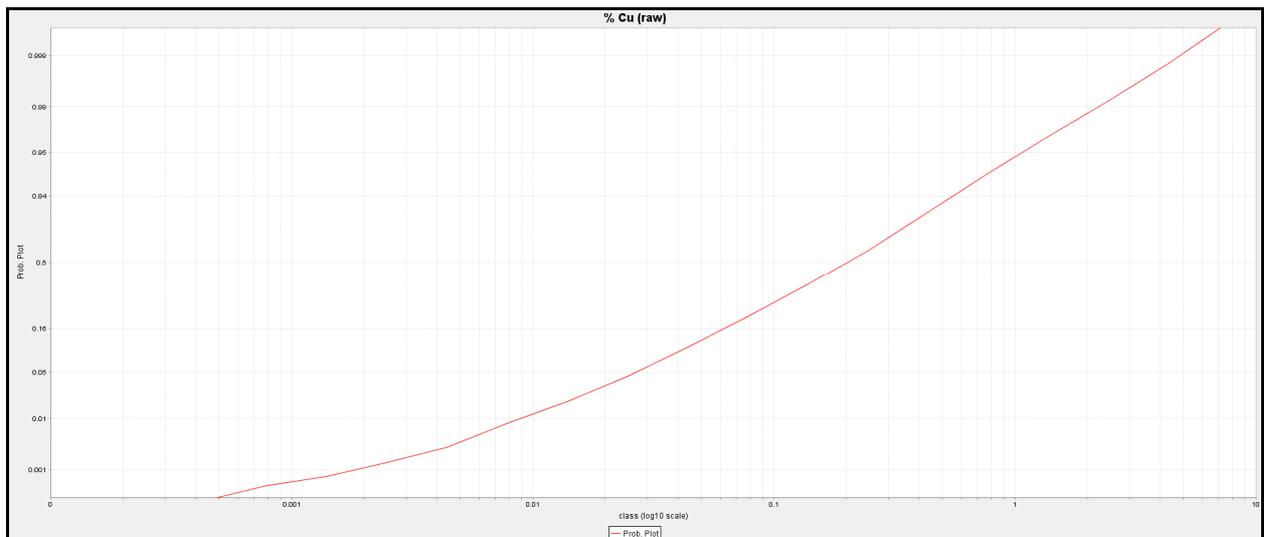


Figure 24: Log-probability Plot % Cu all Composited Lode Samples (comps_all_1.str)

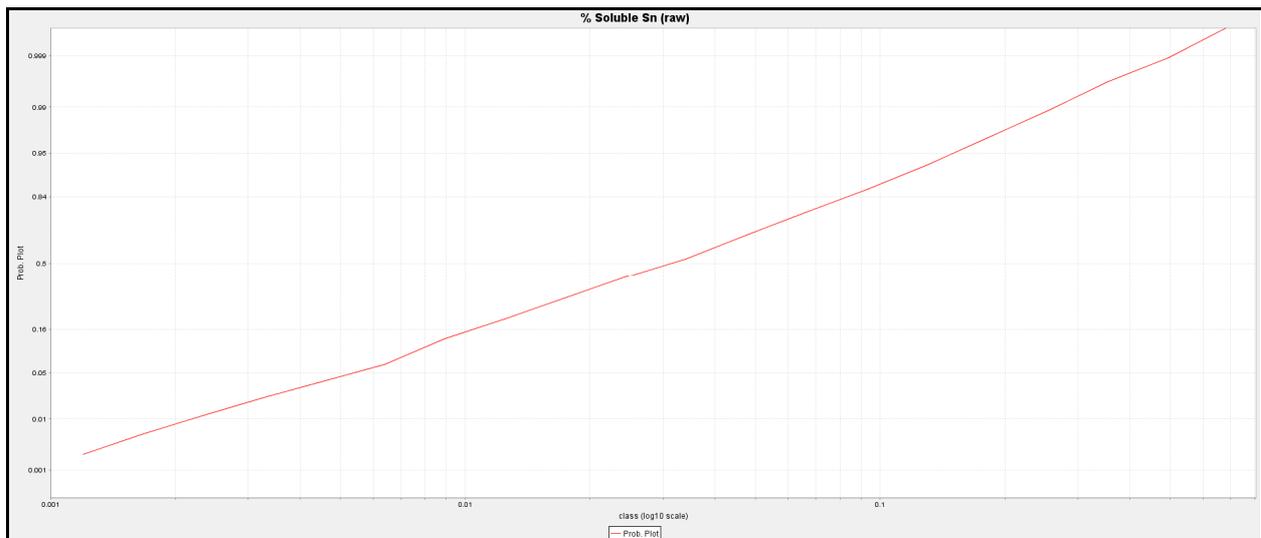


Figure 25: Log-probability Plot % Soluble Sn all Composited Lode Samples (comps_all_1.str)

7.3 Variography for % Sn

Variography was attempted using Sn assays of all the composited lode samples

A down-hole, pair-wise relative experimental variogram for % Sn was prepared. Relative variograms can give a clearer experimental variogram in situations where there is a tendency for a larger range of grades with increasing mean grade of the samples. This tendency is referred to as the proportional effect.

The down-hole pair-wise relative experimental variogram for % Sn was reasonably well structured (see Figure 26) and a spherical model was fitted to the experimental variogram:

$$\gamma(h) = 0.45 + 0.12\text{Sph}3.5(h) + 0.18\text{Sph}6.5(h)$$

The nugget effect represented about 60% of the total variance which is high but reasonably typical for hydrothermal tin deposits.

Directional pair-wise relative experimental variograms were prepared. The directions of continuity for these variograms corresponded to the general strike and dip of the mineralised lenses. For example, see Figure 27 which is the along-strike pair-wise relative experimental variogram fitted with the model:

$$\gamma(h) = 0.45 + 0.12\text{Sph}1.0(h) + 0.13\text{Sph}15(h)$$

The down-dip variogram was similar to the along-strike variogram but the across-dip variogram had a much shorter range, less than 5m. Because of the short ranges, use of the modelled variogram parameters for kriging inputs would tend to reduce grade interpolations to simple averages within the search radii being used. This would render the choice of search ellipsoid parameters as a critical factor in the grade interpolations. To avoid this effect, an inverse distance squared method ("ID2") of interpolation was used.

ID2 is widely used when variography has been unsuccessful and geostatistically based methods such as OK cannot be implemented. ID2 provides good global estimates if the samples used to make the estimate are not clustered and is commonly used in the estimation of grades for hydrothermal tin deposits.

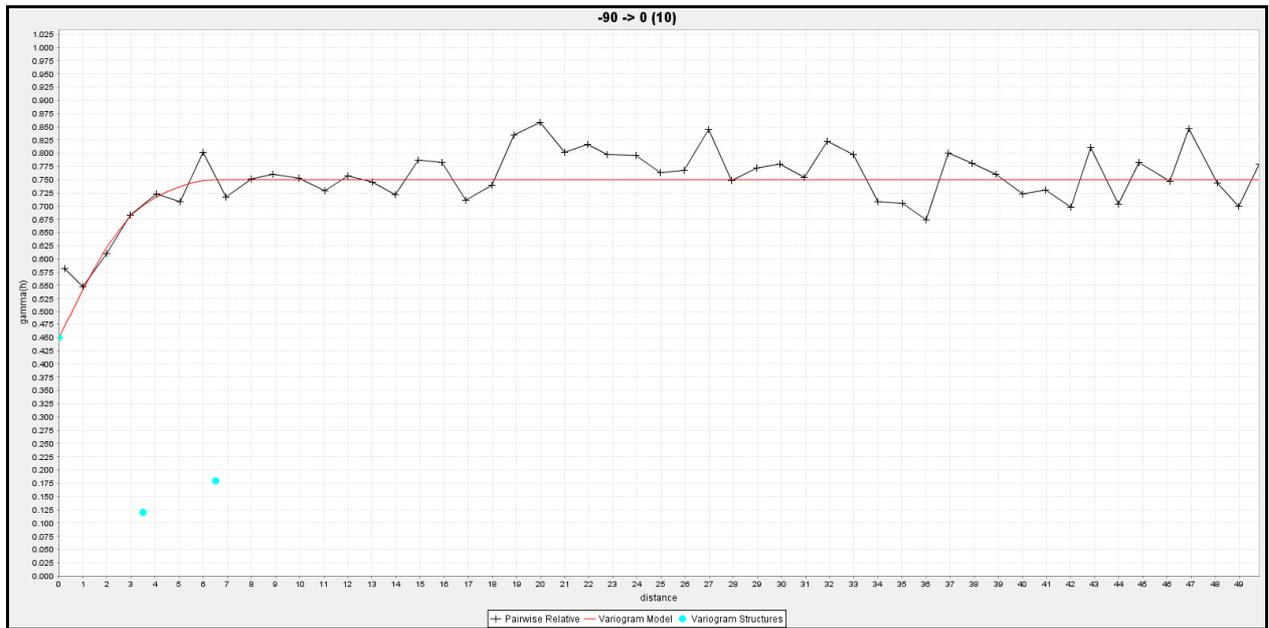


Figure 26: Down-hole Pair-wise Relative Variogram for % Sn all Composited Lode Samples (comps_all_1.str)



Figure 27: Directional 00°/045° Pair-wise Relative Variogram for % Sn all Composited Lode Samples (comps_all_1.str)

7.4 Bulk Density for Tin and Copper Resource Estimation

The Aberfoyle database contained 960 samples from within mineralised lenses for which specific gravity had been determined. The mean of the specific gravities was 3.1 g/cm³.

There was no significant correlation between specific gravity and % Sn, % Cu or % Soluble Sn (see Table 14).

The principal gangue sulphide mineral present at Cleveland is pyrrhotite. A bulk density of 3.1 tonnes/m³ for pyrrhotite bearing limestone implies that the rock contains about 20% pyrrhotite which is in line with descriptions of the deposit. A bulk density of 3.1 tonnes/m³ was used for this resource estimate.

The last estimate by Aberfoyle (Dronseika, 1986) used bulk densities of 3.05 and 3.08 tonnes/m³.

Table 14: Correlation Coefficients between Assays and Specific Gravity in Composited Samples with SG Measurements
(comps_all_1.str)

	% Sn	% Cu	% Soluble Sn
SG	0.22	0.18	0.11

7.5 Global Tin and Copper Resource Estimates

7.5.1 Wireframes and Block Model

Wireframes were created for the lenses listed in Table 17.

For Hall's lenses, above Ratchet fault, only lenses C and D were modelled. Lenses A and B above Ratchet fault were not modelled because they have been extensively mined in the past and only marginal remnants and internal pillars remain.

The wireframe and block model volumes of the lenses modelled are listed in Table 17. There is excellent agreement between the volumes of the wireframes and the block model.

Note that these are the undepleted volumes, that is, these volumes include mineralisation which has been mined in the past. Provision was made to exclude this mined material from this resource estimate (see 7.5.3 Depleted Global Tin and Copper Resource Estimate).

A block model, named mocleveland1302.mdl, was created with the extents listed in Table 15. The block model was rotated in plan to be parallel to the general strike of the lenses.

The attributes listed in Table 16 were included for each block in the block model.

**Table 15: Extents of Block Model
(mocleveland1202.mdl)**

Direction	Minimum	Maximum	User Block size	Minimum Block Size
Y	14,400m N	15,600m N	10	2.5
X	10,600m E	12,100m E	10	2.5
Z	500m RL	1600m RL	10	2.5
Surpac Rotation Parameters				
Bearing (around Z)			-47.838 ^o	
Dip (around Y)			0 ^o	
Plunge (around X)			0 ^o	

Table 16 Attributes Included in the Block Model for Tin and Copper Estimation

Attribute	Description
lode_code	The lode code for each lens as listed in Table 14.
sn_id2	Sn grade interpolated by inverse distance squared method.
cu_id2	Cu grade interpolated by inverse distance squared method.
solsn_id2	Soluble Sn grade interpolated by inverse distance squared method.
numsam	Number of samples used to interpolate Sn grade into the block.
numsamcu	Number of samples used to interpolate Cu grade into the block.
numsamsolsn	Number of samples used to interpolate soluble Sn grade into the block.
mined_out	mined_out = 1 for mined out blocks.

Table 17: Wireframe and Block Model Volumes

Lens	Code	Domain	Volume m³	Block model volume m³
Hall's	HLC1	1	15,902	15,922
	HLD1	1	79,254	78,703
	HLA2	2	6,534	309,406
	HLB2	2	303,522	
	HLC2	2	45,997	46,328
	HLA3	3	131,068	130,469
	HLB3	3	1,840,848	1,841,234
	HLC3	3	874,922	874,391
	HLD3	3	39,832	40,125
Henry's	HN31	3	184,553	184,344
	HN32	3	39,677	40,078
	HN33	3	46,227	46,109
Khaki	KK		619,507	606,734
Luck's	LL		17,184	17,125
Battery	BT		126,179	125,703
	BTE		19,935	19,625
	BTW		25,533	25,141
B South	BS		273,820	273,875
Total			4,690,494	4,675,312

7.5.2 Global Resource Estimate

Sn, Cu and soluble Sn grades were interpolated into the block model using an ID2 method (see 7.3 Variography for % Sn).

The orientations of the search ellipsoids were aligned parallel to the strike and dip of each lens and are listed in Table 18.

The dimensions of the search ellipsoids and other parameters are listed in Table 19. The dimensions were chosen to allow for interpolation of grades into all blocks representing lenses.

Grades were interpolated into each lens using only composited samples from within the lens.

The undepleted global resource estimate is listed in Table 20 and a cross-section through the block model is shown in Figure 28.

Table 18: Search Ellipsoid Orientations for Sn and Cu Interpolations

Lens	Code	Orientation
Hall's	HLC1	strike 045 ^o , dip vertical
	HLD1	strike 045 ^o , dip -65 ^o to 135 ^o
	HLA2	strike 045 ^o , dip -65 ^o to 135 ^o
	HLB2	
	HLC2	strike 045 ^o , dip -65 ^o to 135 ^o
	HLA3	strike 045 ^o , dip vertical
	HLB3	strike 045 ^o , dip vertical
	HLC3	strike 045 ^o , dip vertical
	HLD3	strike 045 ^o , dip vertical
Henry's	HN31	strike 045 ^o , dip vertical
	HN32	strike 045 ^o , dip vertical
	HN33	strike 045 ^o , dip vertical
Khaki	KK	strike 030 ^o , dip -65 ^o to 120 ^o
Luck's	LL	strike 020 ^o , dip -90 ^o to 120 ^o
Battery	BT	strike 055 ^o , dip -65 ^o to 145 ^o
	BTE	strike 030 ^o , dip -90 ^o to 120 ^o
	BTW	strike 020 ^o , dip -90 ^o to 110 ^o
B South	BS	strike 030 ^o , dip -90 ^o to 120 ^o

Table 19: Search Parameters

Ellipsoid axis	Direction	Axis Radius
Major	down-dip	200m
Semi-major	along-strike	40m
Minor	across-strike	10m
Other Parameters		
Discretisation	2 * 2 * 2	
No of samples	3 to 15	

**Table 20: Undepleted Tin and Copper Global Resource Estimates
(mocleveland1302.mdl, numsam>0)**

Lens	Code	Global Resource			
		Tonnes	% Sn	% Cu	%Sn soluble
Hall's	HLC1	49,358	0.74	0.39	0.04
	HLD1	243,980	0.91	0.59	0.03
	HLB2	959,159	0.66	0.23	0.05
	HLC2	143,617	0.86	0.55	0.10
	HLA3	404,453	0.75	0.28	0.02
	HLB3	5,306,328	0.34	0.16	0.02
	HLC3	2,710,611	0.47	0.17	0.02
	HLD3	124,388	0.64	0.40	0.01
Henry's	HN31	408,038	0.86	0.14	0.02
	HN32	124,242	0.74	0.45	0.05
	HN33	142,939	0.72	0.34	0.01
Khaki	KK	1,880,877	0.76	0.21	0.04
Luck's	LL	53,088	0.87	0.30	0.06
Battery	BT	389,680	0.83	0.33	0.04
	BTE	60,838	0.70	0.22	0.07
	BTW	77,936	0.50	0.14	0.02
B South	BS	849,013	0.58	0.24	0.03

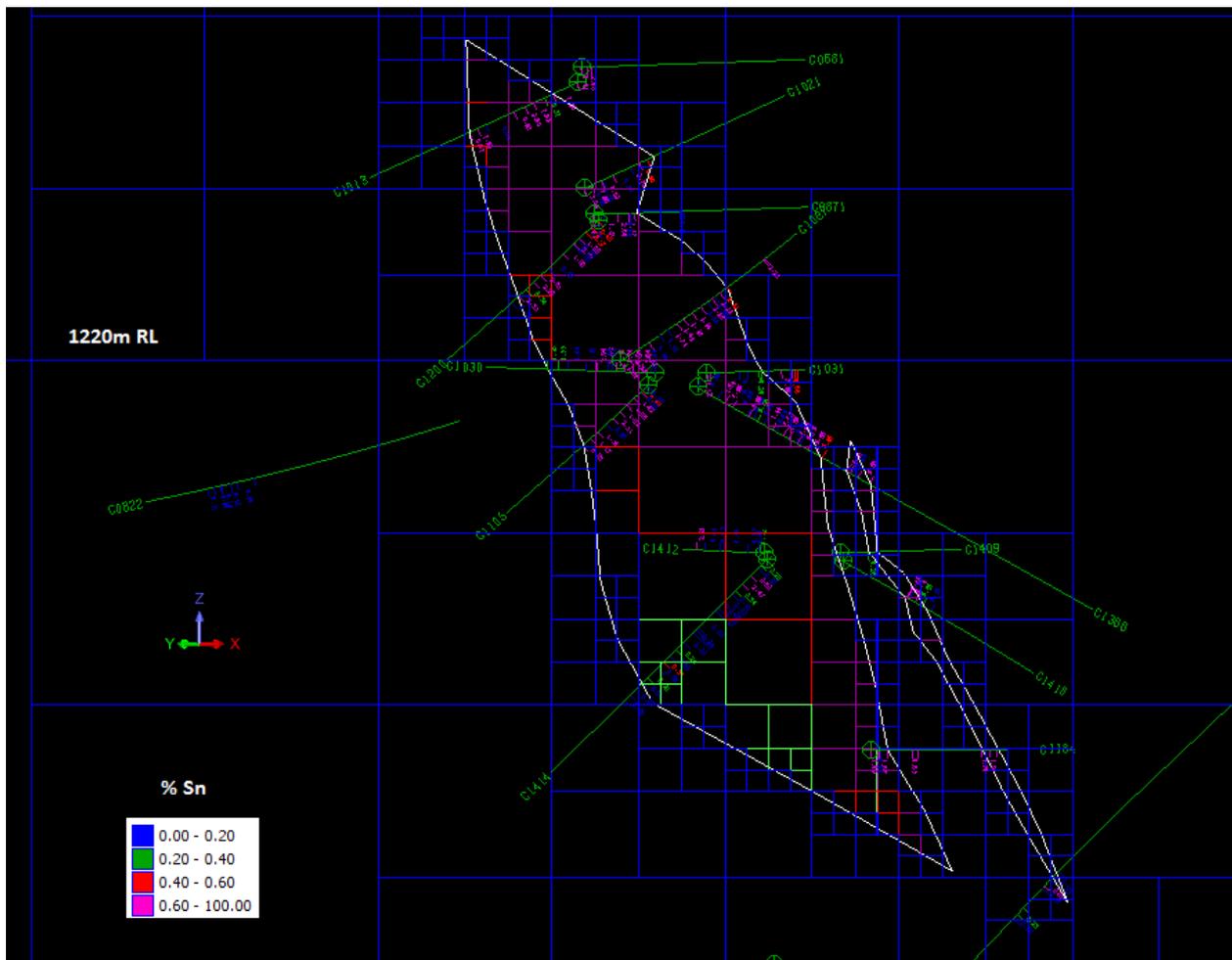


Figure 28: Cross-section through the Tin and Copper Block Model - Section Q
Hall's B lens is the large lens shown with a white outline, Hall's C lens is the smaller lens.
(mocleveland1302.mdl, lode_hlb2_1.str, lode_hlc2_1.str)

7.5.3 Depleted Global Resource Estimate

Two dimensional outlines of the mined-out parts of lenses were included in Aberfoyle's closure resource report as longitudinal projections (Dronseika, 1986). These outlines were digitised (for example, see Figure 29) and used as cookie cutters to flag the blocks in the block model which had been mined out (see Table 21). No differentiation was made on the basis that some mine workings may not have taken out the full width of the mineralisation and, consequently, the full width of the mineralisation was depleted.

The depleted global resource is listed by lens in Table 22.

Table 21: Mining Depletions Applied

Lens	Code for this report	Depletion reference in Dronseika, 1986.
Hall's	HLC1	HCAR
	HLD1	HDAR
	HLA2	none
	HLB2	HLA1, HLA2, HLA3, HLBRN
	HLC2	HLCRN
	HLA3	HLAW, HLAE
	HLB3	HLBUT
	HLC3	HLCET, HLCUT
	HLD3	none
Henry's	HN31	HNUT
	HN32	HNWUT
	HN33	HNEUT
Khaki	KK	KHKW, KHKE
Luck's	LL	none
Battery	BT	none
	BTE	none
	BTW	none
B South	BS	BSTH

Table 22: Depleted Global Resource Estimates
(mocleveland1302.mdl, numsam>0, mined_out<1)

Lens	Code	Tonnes	% Sn
Hall's	HLC1	49,358	0.74
	HLD1	100,072	0.84
	HLB2	562,408	0.68
	HLC2	41,995	0.79
	HLA3	331,216	0.73
	HLB3	4,979,036	0.32
	HLC3	2,656,216	0.46
	HLD3	124,388	0.64
Henry's	HN31	362,022	0.85
	HN32	29,353	0.64
	HN33	51,247	0.70
Khaki	KK	521,478	0.60
Luck's	LL	53,088	0.87
Battery	BT	389,680	0.83
	BTE	60,838	0.70
	BTW	77,936	0.50
B South	BS	849,013	0.58
Total		11,239,344	0.47

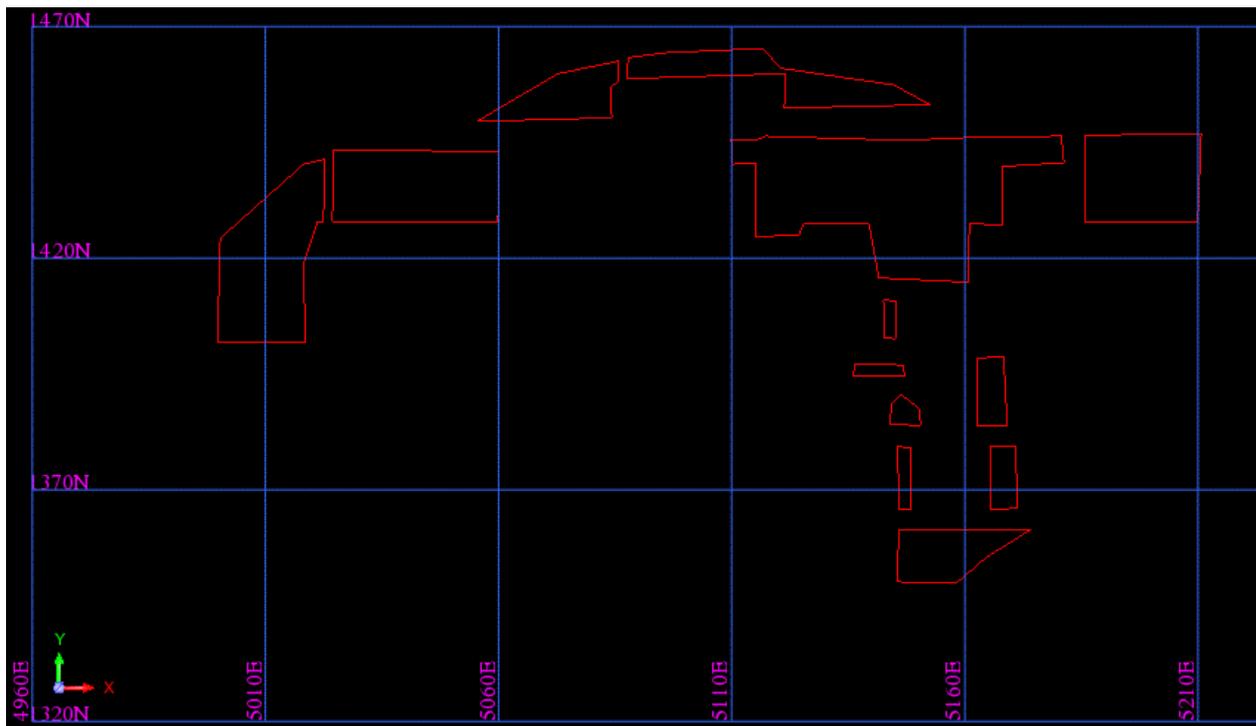


Figure 29: Depleted Areas of Part of Hall's A Lens

The depleted areas are outlined in red. The coordinates are coordinates relative to Hall's reference plane.
(lode_13_mined1.str)

7.6 Tin and Copper Mineral Resource Estimate

The grade-tonnage information for the depleted global resource is listed in Table 23. Note that the reliability of the estimates becomes lower with increasing cut-off grade, particularly where cut-off grade exceeds the average global grade of the deposit, that is, above about 0.5% Sn.

The Mineral Resource has been stated at a cut-off grade of 0.35% Sn.

A cut-off grade of 0.35%, at a tin price of US\$25,000 per tonne implies that material with a contained metal value of about US\$90 could be treated at a profit, which seems reasonable, even at relatively modest metallurgical recoveries. This was also the cut-off grade used by Aberfoyle for its final resource estimate (Dronseika, 1986).

A Sn equivalent grade was estimated from the Sn and Cu grades, assuming metal prices current at the time of writing, that is, US\$25000 per tonne for tin and US\$8500 per tonne for Cu, a ratio of % Cu to % Sn of 0.34. This was considered reasonable given that, as in the past, both metals would probably be recovered during processing.

Classification of Mineral Resources was based on the known geological continuity of the lenses. For parts of the deposit, where drilling intensity was adequate to reasonably reliably define the lens shapes and extents, were classified as Indicated Mineral Resources and the balance as Inferred Mineral Resources (refer to Table 24). This classification reflects the author's confidence in the location, quantity, grade, geological characteristics and continuity of the Mineral Resources.

The Mineral Resource is listed by class in Table 25 and by lens in Table 26.

**Table 23: Global Resource, Grade-tonnage Information, Depleted for Mining
(mocleveland1202.mdl, 18_grade_tonnage_information.tcl, numsam>0 and numsamcu>0, mined_out<1)**

Cut-off % Sn	Tonnage	% Sn	Contained tin tonnes	% Cu	Contained copper tonnes
0.00	11,239,344	0.473	53,162	0.197	22,142
0.05	11,204,708	0.474	53,110	0.197	22,073
0.10	10,831,836	0.488	52,859	0.200	21,664
0.15	10,192,752	0.511	52,085	0.206	20,997
0.20	9,292,347	0.543	50,457	0.217	20,164
0.25	7,994,948	0.594	47,490	0.229	18,308
0.30	7,056,617	0.636	44,880	0.239	16,865
0.35	6,118,577	0.684	41,851	0.253	15,480
0.40	5,459,245	0.722	39,416	0.264	14,412
0.50	4,273,350	0.797	34,059	0.283	12,094
0.60	3,149,019	0.886	27,900	0.307	9,667
0.70	2,256,364	0.979	22,090	0.325	7,333
0.80	1,564,677	1.081	16,914	0.335	5,242
0.90	986,430	1.213	11,965	0.355	3,502
1.00	740,561	1.303	9,650	0.361	2,673

Table 24: Classification of Resources

Lens	Code	Domain	Classification
Hall's	HLC1	1	Indicated
	HLD1	1	Indicated
	HLA2	2	Indicated
	HLB2	2	Indicated
	HLC2	2	Indicated
	HLA3	3	Indicated
	HLB3	3	Indicated above 1000m RL Inferred below 950m RL
	HLC3	3	Indicated above 1020m RL Inferred below 1020m RL
	HLD3	3	Inferred
Henry's	HN31	3	Indicated above 1190m RL, north of L Section Inferred elsewhere
	HN32	3	Indicated
	HN33	3	Indicated
Khaki	KK		Indicated
Luck's	LL		Indicated
Battery	BT		Indicated
	BTE		Inferred
	BTW		Indicated
B South	BS		Indicated

Table 25: Tin and Copper Mineral Resources
 (mocleveland1302.mdl, numsam>0 and numsamcu>0, mined_out<1)

Cleveland Tin and Copper Mineral Resource				
31 March 2012				
0.35% Sn cut-off				
Category	Tonnage	% Sn	% Cu	% Sn Equivalent
Indicated	4,239,000	0.70	0.28	0.80
Inferred	1,880,000	0.64	0.19	0.70
Total	6,119,000	0.68	0.25	0.77

Table 26: Tin and Copper Mineral Resources by Lens
(mocleveland1302.mdl, numsam>0 and numsamcu>0, mined_out<1)

Cleveland Tin and Copper Indicated Mineral Resource 31 March 2012					
0.35% Sn cut-off					
Lode	Structural Domain	Tonnage	% Sn	% Cu	% Sn Equivalent
B South	-	631,000	0.71	0.27	0.80
Battery	-	387,000	0.84	0.33	0.95
Battery West	-	61,000	0.56	0.15	0.61
Hall's A	3	310,000	0.76	0.32	0.87
Hall's B	2	434,000	0.81	0.29	0.91
	3	1,122,000	0.61	0.23	0.69
Hall's C	1	44,000	0.79	0.41	0.93
	2	41,000	0.80	0.42	0.94
	3	572,000	0.62	0.27	0.71
Hall's D	1	97,000	0.86	0.59	1.06
Henry's 32	3	20,000	0.80	0.30	0.90
Henry's 33	3	49,000	0.72	0.52	0.90
KK	-	418,000	0.69	0.21	0.76
LL	-	53,000	0.87	0.30	0.97
Sub-total	-	4,239,000	0.70	0.28	0.80
Cleveland Tin and Copper Inferred Mineral Resource 31 March 2012					
0.35% Sn cut-off					
Lode	Structural Domain	Tonnage	% Sn	% Cu	% Sn Equivalent
Battery East	-	55,000	0.74	0.24	0.82
Hall's B	3	369,000	0.44	0.16	0.49
Hall's C	3	1,037,000	0.61	0.17	0.67
Hall's D	3	97,000	0.73	0.48	0.89
Henry's 31	3	322,000	0.93	0.37	1.06
Sub-total	-	1,880,000	0.64	0.19	0.70

7.7 Reliability of the Tin and Copper Resource Estimate

The nature of defects in the estimation and reporting for this resource estimate and their likely impacts on the quality of the resource report are listed in Table 6. The Assessment and Resource Reporting Criteria listed in Table 27 have been categorised using the terms listed under Nature of Defects.

Table 27: Cleveland Sn and Cu Resource Estimate – Assessment and Reporting Criteria

Estimation and Reporting of Mineral Resources	
Database integrity.	<p>Tolerable.</p> <p>The specific measures taken by Aberfoyle to ensure database integrity are not known but the creation of a digital database is allowing for on-going review of the integrity of the data.</p>
Geological interpretation.	<p>Acceptable.</p> <p>The geological interpretation was devised by the author of this report (see 6 GEOLOGICAL INTERPRETATION).</p> <p>Halls Formation, the geological formation which contains the lenses of mineralisation, dips nearly vertically and is known over a strike length of 700m, an across strike width of about 200m, and a down-dip extent of over 800m (Ransom and Hunt, 1975 and Dronseika, 1986). Halls Formation and the lenses of mineralisation are interpreted as being offset by flat lying faults (Ransom and Hunt, 1975 and Everett, 1977).</p>
Dimensions.	<p>Acceptable.</p> <p>Halls Formation, the geological formation which contains the lenses of mineralisation, dips nearly vertically and is known over a strike length of 700m, an across strike width of about 200m, and a down-dip extent of over 800m (Ransom and Hunt, 1975 and Dronseika, 1986).</p> <p>The spatial dimensions for each lens were quoted in the resource report (Dronseika, 1986).</p>
Estimation and modelling techniques.	<p>Acceptable.</p> <p>Mineralisation was modelled as three dimensional blocks from 10m X 10m X 10m to 2.5m X 2.5m X 2.5m in size. Grade estimates of total Sn, Cu and acid soluble Sn were made by inverse distance squared and depletion was made for mining.</p> <p>No assumptions were made about the recovery of by-</p>

	<p>products.</p> <p>No estimates of S grade were made.</p>
Moisture.	<p>Acceptable.</p> <p>All assays were reported on a dry basis.</p>
Cut-off parameters.	<p>Acceptable.</p> <p>The resources were reported using a cut-off grade of 0.35% total Sn which was reasonable given the current tin price (see 7.6 Tin and Copper Mineral Resource Estimate).</p>
Mining factors or assumptions.	<p>Acceptable.</p> <p>Resources were estimated, not reserves, and no mining factors were applied.</p>
Metallurgical factors or assumptions.	<p>Acceptable.</p> <p>Resources were estimated, not reserves, and no mining factors were applied.</p>
Bulk density.	<p>Acceptable.</p> <p>A bulk density of 3.1 tonnes/m³ was used This was very similar to those which were used in earlier estimates by Aberfoyle. (See 7.4 Bulk Density)</p>
Classification.	<p>Acceptable.</p> <p>The resources were classified as Indicated and Inferred based on current understanding of geological and grade continuity (see 7.6 Tin and Copper Mineral Resource Estimate).</p>
Audits or reviews.	<p>Acceptable.</p> <p>The process used to create this resource estimate was reviewed by Rod Williams, geologist with Norvale Pty Ltd, and the report has been peer reviewed by Stuart Hutchins, Senior Geologist with Mining One.</p>
Discussion of relative accuracy/confidence	<p>Tolerable.</p> <p>See Classification above.</p>

7.8 Historical Tin and Copper Resource Estimate

The historical estimate listed in Table 28 has never been reported in compliance with the JORC Code. The resource estimate was made by Aberfoyle geologists at mine closure in 1986 (Dronseika, 1986). It is included here for reference only.

The Aberfoyle estimate differs from the estimate for this report because:

- the estimates for the two reports are based on somewhat different geological interpretations and modelling methods;
- the Aberfoyle estimate included estimates of remnants above Ratchet fault but these were not estimated for this report; and
- the Aberfoyle estimates included an allocation of 75,000 tonnes for broken stocks which have not been included in this report.

**Table 28: Cleveland Mine – Tin and Copper Estimate 1986
(Dronseika, 1986)**

Cleveland Mine – Tin and Copper Estimate 1986			
Measured and Indicated Category			
0.35% Sn (total) Cut-off Grade			
Lens	tonnes	% Sn (total)	% Cu
Hall's	3,400,000	0.71	0.33
Henry's	450,000	0.81	0.47
Khaki and Lucks	680,000	0.70	0.22
B South and Battery	620,000	0.57	0.22
Total	5,200,000	0.70	0.31
Inferred Category			
0.35% Sn (total) Cut-off Grade			
Lens	tonnes	% Sn (total)	% Cu
Hall's	1,100,000	0.69	0.2
Khaki and Lucks	40,000	0.73	0.15
B South and Battery	190,000	0.86	0.35
Total	1,330,000	0.72	0.22

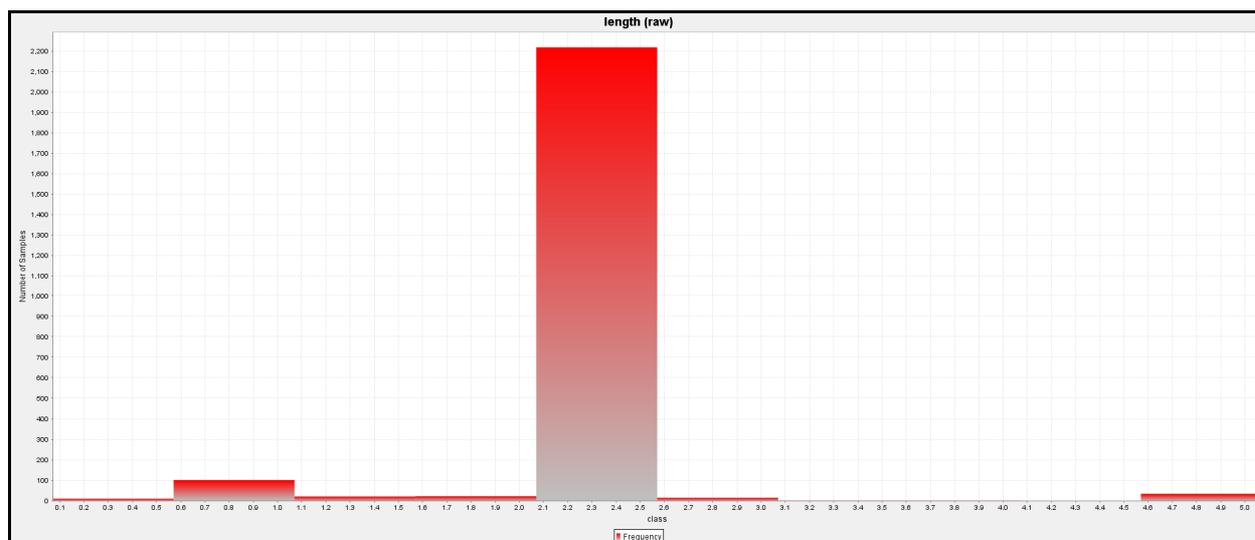
8 TUNGSTEN RESOURCE ESTIMATE

8.1 Compositing Length for Tungsten Resource Estimation

All assays for the estimate of the Mineral Resources came from sampling of diamond drill core.

A histogram of the sample lengths of the raw assays of the data for Foley's stock-work shows that most assays were taken over lengths of 2.5m (see Figure 30).

A compositing length of 2.5m was used for this resource estimate.



**Figure 30: Histogram of all WO₃ Sample Lengths
(samples_foleys_every_1.str)**

8.2 Basic Statistics for WO₃

A histogram, log histogram and log-probability plot for % WO₃ for composited samples within the stock-work as defined within a 0.2% WO₃ threshold (see 6 GEOLOGICAL INTERPRETATION) were made (Figures 31 to 33).

The distribution appears log-normal and the log-probability plot is nearly linear suggesting that there is only one population of assays present in the data. In addition, there are no rogue outliers, that is, high grade assays that do not fit the distributions and which consequently indicate the need for cutting of high grades.

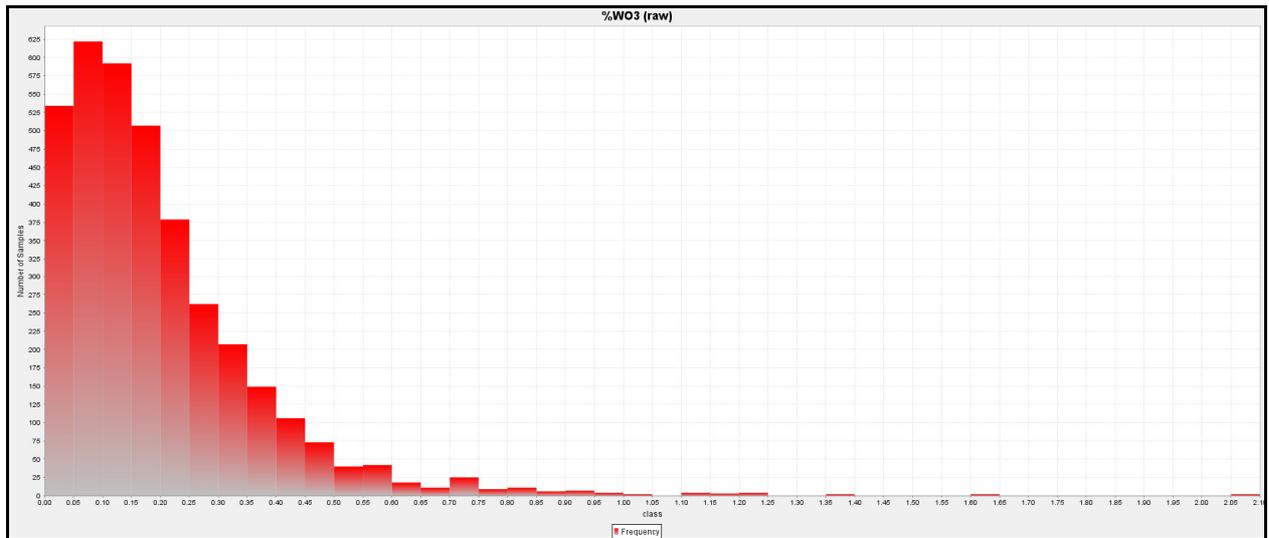


Figure 31: Histogram of % WO₃ for all Composited Samples from within Foleys Stock-work defined within a 0.2% WO₃ Threshold (comps_foleys_1.str)

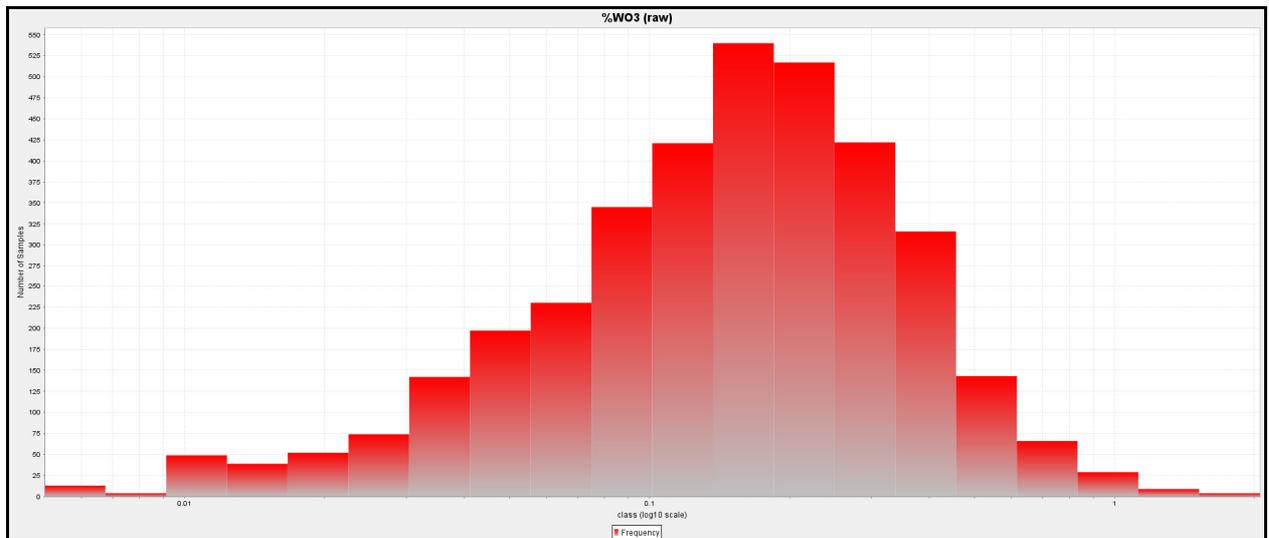


Figure 32: Log-histogram of % WO₃ for all Composited Samples from within Foleys Stock-work defined within a 0.2% WO₃ Threshold (comps_foleys_1.str)

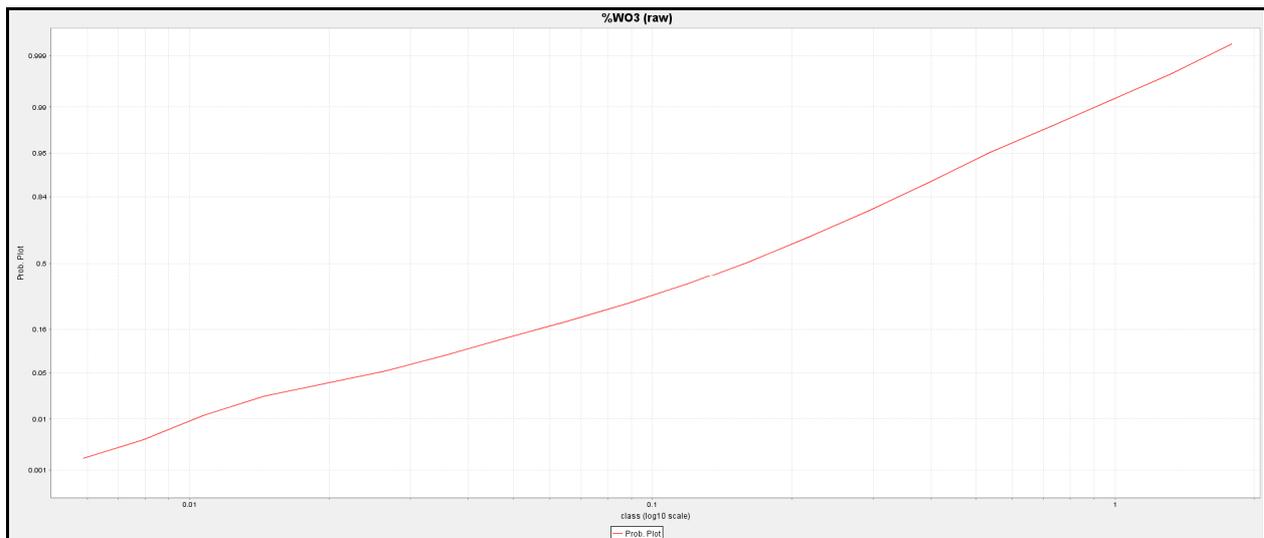


Figure 33: Log-probability Plot of % WO₃ for all Composited Samples from within Foleys Stock-work defined within a 0.2% WO₃ Threshold (comps_foleys_1.str)

8.3 Variography for % WO₃

Directional variography was successful for % WO₃ for the composited samples within Foley’s stock-work as defined within a 0.2% WO₃ threshold (see 6 GEOLOGICAL INTERPRETATION).

Directional pair-wise relative experimental variograms (see Figures 34 to 36) were fitted with the following spherical models:

$$+40^{\circ}/090^{\circ} \quad \gamma(h) = 0.10 + 0.18\text{Sph}7.5(h) + 0.16\text{Sph}75(h)$$

$$00^{\circ}/000^{\circ} \quad \gamma(h) = 0.10 + 0.18\text{Sph}5(h) + 0.16\text{Sph}50(h)$$

$$-50^{\circ}/090^{\circ} \quad \gamma(h) = 0.10 + 0.18\text{Sph}5(h) + 0.16\text{Sph}50(h)$$

The nugget effect represented about 22% of the total variance.

The principal direction of grade continuity was +40° towards 090° which may correspond to the average dip direction of the WO₃ bearing veins.

The success of the variography meant that WO₃ grades in the stock-work could be interpolated by ordinary kriging and this was undertaken (see 8.5.2 Tungsten Resource Estimate).



Figure 34: Directional 40⁰/090⁰ Variogram for % WO₃ for all Composited Samples from within Foleys Stock-work defined within a 0.2% WO₃ Threshold (comps_foleys_1.str)

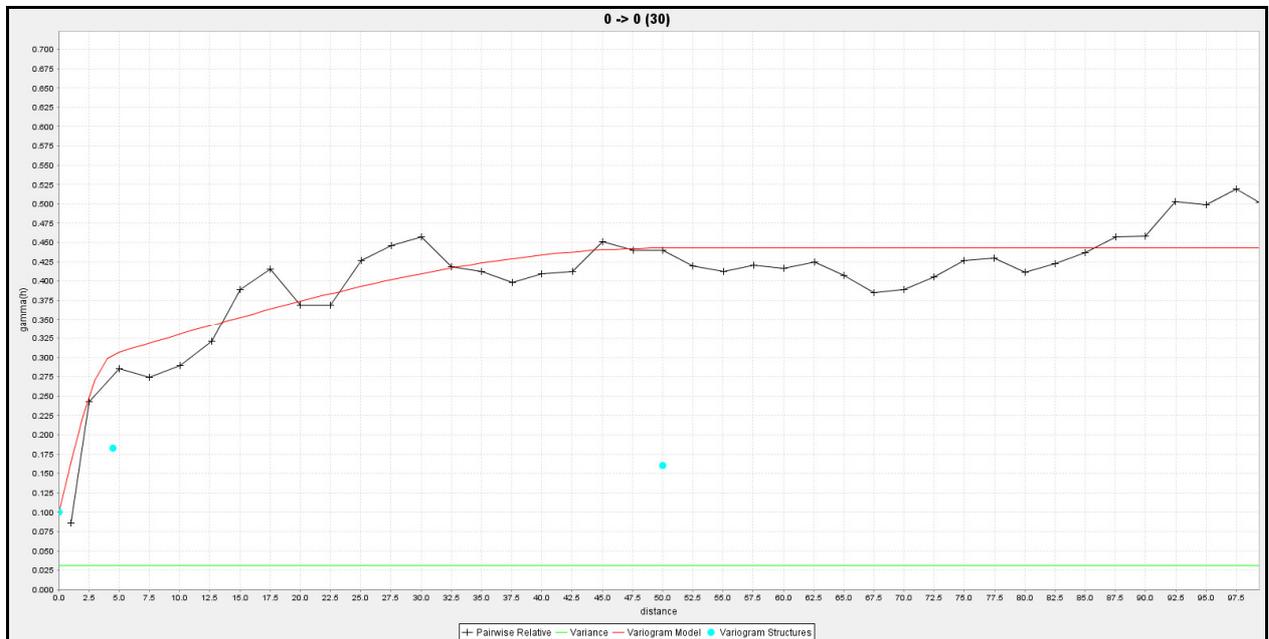


Figure 35: 0/0 Directional 00⁰/000⁰ Variogram for % WO₃ for all Composited Samples from within Foleys Stock-work defined within a 0.2% WO₃ Threshold (comps_foleys_1.str)

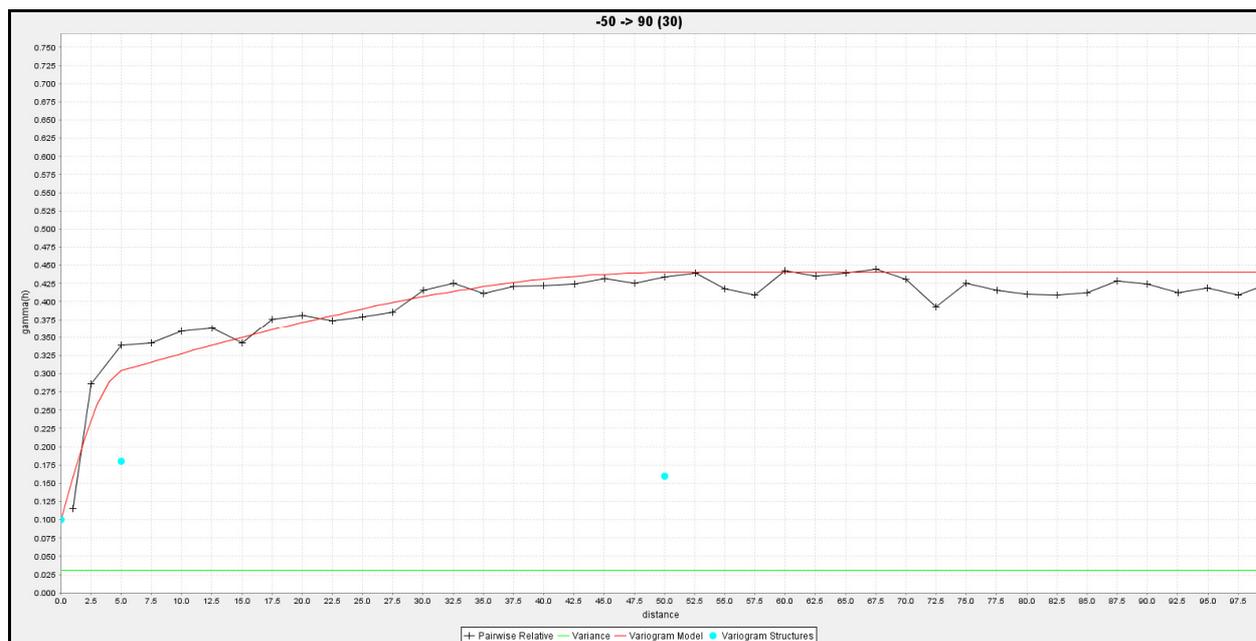


Figure 36 Directional -50⁰/090⁰ Variogram for % WO₃ for all Composited Samples from within Foleys Stock-work defined within a 0.2% WO₃ Threshold (comps_foleys_1.str)

8.4 Bulk Density for Tungsten Resource Estimation

The Aberfoyle database contained 1235 samples from the composited samples within Foley’s stock-work as defined within a 0.2% WO₃ threshold. The mean of the specific gravities was 2.86 g/cm³.

A bulk density of 2.85 tonnes/m³ was used for this resource estimate.

The last estimate by Aberfoyle (Dronseika, 1983) used a bulk density of 2.87 tonnes/m³.

8.5 Tungsten Resource Estimate

A wireframe was created for the stock-work as defined within a 0.2% WO₃ threshold (see Figure 15).

The same block model as used for the tin and copper resource estimates was used for the tungsten estimate in Foley’s stock-work (see 7.5.1 Wireframes and Block Model).

The attributes listed in Table 29 were included for each block in the block model.

Table 29: Attributes Included in the Block Model for Tungsten Estimation

Attribute	Description
lode_code	The lode code for each lens as listed in Table 14.
wo3_ok	WO ₃ grade interpolated by ordinary kriging.
numsamwo3	Number of samples used to interpolate WO ₃ grade into the block.

WO₃ grades were interpolated into the block model using ordinary kriging (see 8.3 Variography for % WO₃).

The orientation of the search ellipsoid was aligned along the directions of the variograms (see 8.3 Variography for % WO₃ and Table 30).

The dimensions of the search ellipsoid and other parameters are listed in Table 31. The dimensions were chosen to allow for interpolation of grades into all blocks representing the stock-work above 850m RL and the search radii were in proportion to the ranges of the variogram models.

Foley's stock-work does not have a distinct geological boundary: it does not appear to occupy a particular geological structure or a particular lithology. The boundary derived for the stock-work (see 6 GEOLOGICAL INTERPRETATION and Figure 15) was based simply on the available WO₃ grades from samples from holes which intersected the stock-work. This is a fuzzy boundary, sometimes referred to as a soft boundary.

Estimation of grades within the soft boundary using only samples from within the boundary would have over-estimated the grade of the stock-work. Consequently, the WO₃ grade was interpolated into the blocks within Foley's stock-work, as defined within a 0.2% WO₃ threshold, using all WO₃ assays from samples from holes which have intersected the stock-work or that occur in the neighbourhood of the stock-work.

The deeper holes into Foley's stock-work did not intersect the boundary of the stock-work as defined by the 0.2% WO₃ threshold. The lowest intersections of the boundary are at about 850m RL and this level was taken as a lower limit for the resource estimation for this report.

The volume of the Foley's Stock-work, as defined by the 0.2% WO₃ threshold, above 850m RL was 3,454,998m³.

The global resource estimate was 9,845,058 tonnes at 0.19% WO₃ (see Table 32). A cross-section through the block model is shown in Figure 37.

The Mineral Resource has been stated at a cut-off grade of 0.20% WO₃.

A cut-off grade of 0.20% and a WO₃ price of US\$35,000 per tonne implies that material with a contained metal value of about US\$70 could be treated at a profit. This appears reasonable even at relatively modest metallurgical recoveries. This was also the cut-off grade used by Aberfoyle for its final resource estimate (Dronseika, 1983).

The tungsten Mineral Resource has been classified as Inferred. The geological and grade continuity are currently not well enough understood to allow for a higher classification.

At a cut-off grade of 0.20% WO₃, the Inferred Mineral Resource was 3.9M tonnes at 0.30% WO₃ (see Table 33).

Table 30: Search Ellipsoid Orientation for WO₃ Interpolation

Lens	Code	Orientations
Foleys	FOL	strike 000 ^o , dip -50 ^o /090 ^o

Table 31: Search Parameters

Ellipsoid axis	Direction	Axis Radius
Major	40 ^o /090 ^o	100m
Semi-major	00 ^o /000 ^o	66m
Minor	-50 ^o /090 ^o	66m
Other Parameters		
Discretisation	2 * 2 * 2	
No of samples	1 to 10	

**Table 32: Global Tungsten Resource, Grade-tonnage Information
(mocleveland1202.mdl, numsamok>0)**

Cut-off % Sn	Tonnage	% WO ₃	Contained WO ₃ tonnes
0.00	9,845,058	0.190	18,706
0.05	9,595,727	0.194	18,616
0.10	7,564,078	0.225	17,019
0.15	5,597,756	0.260	14,554
0.20	3,981,227	0.295	11,745
0.25	2,387,766	0.341	8,142
0.30	1,454,970	0.383	5,573
0.35	866,934	0.423	3,667
0.40	360,792	0.494	1,782
0.45	216,867	0.544	1,180
0.50	136,622	0.585	799

Table 33: Tungsten Mineral Resources
(mocleveland1302.mdl, numsamok>0)

Cleveland Tungsten Mineral Resource 31 March 2012		
0.20% WO ₃ cut-off		
Category	Tonnage	% WO ₃
Inferred	3,980,000	0.30
Total	3,980,000	0.30

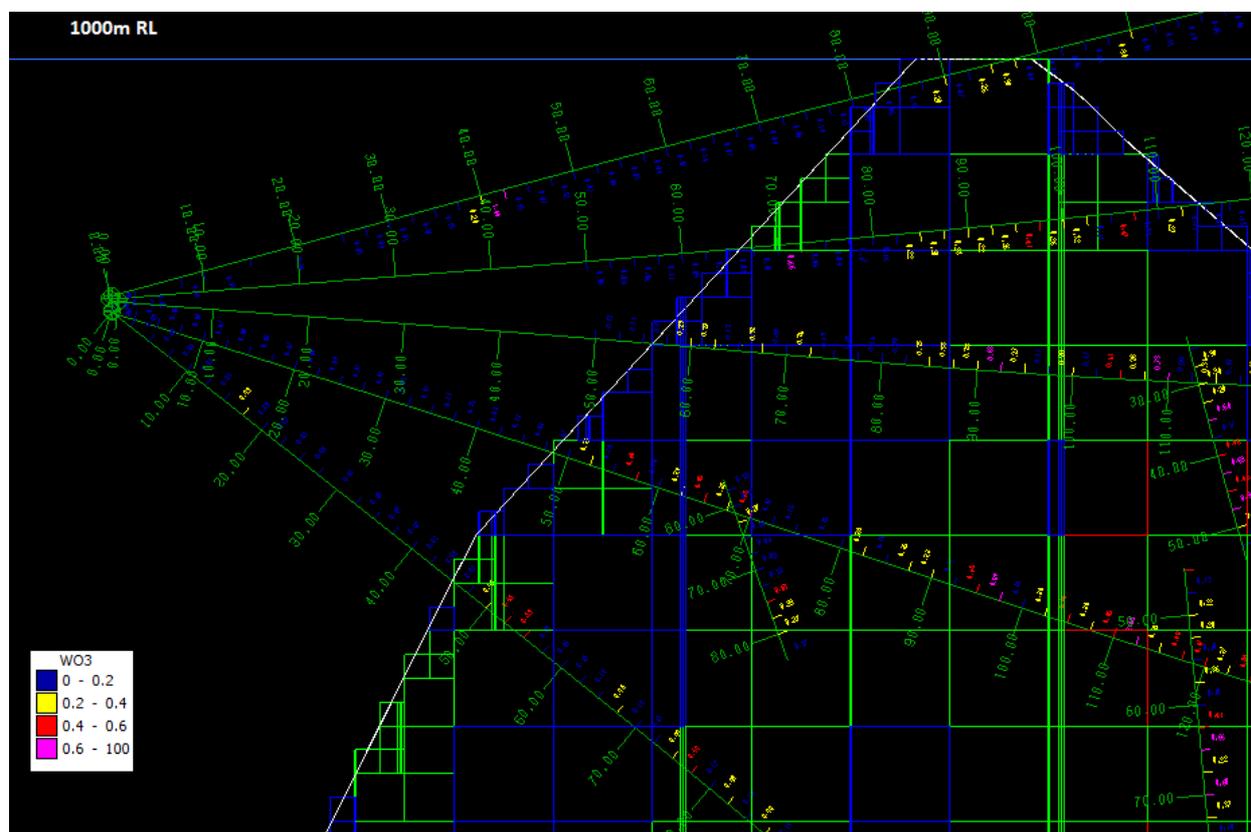


Figure 37: Oblique Cross-section through the Tungsten Block Model
The boundary of Foley's stock-work as defined within the 0.20% WO₃ threshold is shown as a white line. The largest blocks are 10m X 10m. (mocleveland1302.mdl, foleys_strings1.dtm)

8.6 Reliability of the Tungsten Resource Estimate

The nature of defects in the estimation and reporting for this resource estimate and their likely impacts on the quality of the resource report are listed in Table 8. The Assessment and

Resource Reporting Criteria listed in Table 34 have been categorised using the terms listed under Nature of Defects.

Table 34: Tungsten Resource Estimate - Assessment and Reporting Criteria

Estimation and Reporting of Mineral Resources	
Database integrity.	<p>Tolerable.</p> <p>The specific measures to ensure database integrity are not known but the descriptions of the work carried out by Dronseika (Dronseika, 1983) and McArthur (McArthur, 1983) imply a systematic and thorough approach.</p>
Geological interpretation.	<p>Acceptable.</p> <p>The geological interpretation was devised by the author of this report (see 6 GEOLOGICAL INTERPRETATION).</p> <p>The mineralisation was interpreted as a stock-work of quartz veins surrounding an intrusion of quartz porphyry in the form of a steeply dipping stock. The quartz porphyry stock is known from near sea level to 600m below sea level. The wolframite bearing quartz vein stock-work is known from 300m above sea level to 600m below sea level. The maximum extents of the quartz vein stock-work are 900m vertically, 300m along strike and 300m across strike.</p>
Dimensions.	<p>Acceptable.</p> <p>The wolframite bearing quartz vein stock-work is known from 300m above sea level to 600m below sea level. The maximum extents of the quartz vein stock-work extends are 900m vertically, 300m along strike and 300m across strike. (Dronseika, 1983)</p>
Estimation and modelling techniques.	<p>Acceptable.</p> <p>Mineralisation was modelled as three dimensional blocks from 10m X 10m X 10m to 2.5m X 2.5m X 2.5m in size.</p> <p>Grade estimates of WO₃ were made by ordinary kriging.</p> <p>No assumptions were made about the recovery of by-products.</p> <p>No estimates of S grade were made.</p>
Moisture.	<p>Acceptable.</p> <p>All assays were reported on a dry basis.</p>
Cut-off parameters.	<p>Acceptable.</p> <p>The resources were reported using a cut-off grade of 0.2%</p>

	<p>WO₃ which was reasonable given the current tungsten price (see 8.5 Tungsten Resource Estimate).</p>
<p>Mining factors or assumptions.</p>	<p>Acceptable.</p> <p>Resources were estimated, not reserves, and no mining factors were used.</p>
<p>Metallurgical factors or assumptions.</p>	<p>Acceptable for Inferred Category</p> <p>Metallurgical test work to estimate the flotation response for wolframite was undertaken by Aberfoyle Central Metallurgical Services; the note which refers to the test work reports concentrate grades which were achieved but not mill recoveries (Thomas, 1983).</p> <p>However, in the author's experience, given the size of the wolframite grains reported in drill logs and in thin and polished sections and the generally magnetic nature of wolframite, mill recoveries of the order of 60% to 80% could be expected.</p> <p>Metallurgical test work was carried out to determine whether photometric ore sorting would be practically useful to beneficiate the wolframite bearing quartz stock-work mineralisation. Results were not encouraging (Hamill, 1981).</p>
<p>Bulk density.</p>	<p>Acceptable.</p> <p>Bulk density of 2.85 tonnes per cubic metre was used based on 1235 specific gravity measurements on core samples (see 8.4 Bulk Density for Tungsten Resource Estimate).</p>
<p>Classification.</p>	<p>Acceptable.</p> <p>The resource was classified as Inferred based on current understanding of geological and grade continuity (see 8.5 Tungsten Resource Estimate).</p>
<p>Audits or reviews.</p>	<p>Acceptable.</p> <p>This report has been peer reviewed by Stuart Hutchins, Senior Geologist with Mining One.</p>
<p>Discussion of relative accuracy/confidence</p>	<p>Acceptable.</p> <p>The estimate reflects the current understanding of the geological and grade continuity (see 8.5 Tungsten Resource Estimate). The accuracy of the estimates is adequate for classification as an Inferred Mineral Resource.</p>

9 TIN AND COPPER TAILINGS RESOURCE ESTIMATE

9.1 Tin and Copper Tailings Resource Estimate

The operating statistics for the Cleveland mill are tabulated in Table 35. This information was accumulated by Rockwell from Cleveland and Aberfoyle Annual Reports and data reported by Foo (1981).

The Cleveland mill discharged two waste streams: a coarse waste stream from heavy media separation (“HMS floats”) and a tailings stream of finely ground material. The HMS plant scalped coarse reject material from the ore stream prior to the grinding and subsequent treatment of the ore.

The tailings are stored on site in two tailings dams, the surfaces of which have been re-vegetated. None of the tailings has been removed from the site since they were placed.

The quantity and grade of tailings discharged from the Cleveland mill was estimated (see Tables 36 and 37) using the following steps (see Table 37).

Step 1.

The tonnage of Sn and Cu concentrate was estimated from the Sn and Cu in concentrate, assuming grades of 50% Sn and 20% Cu for concentrate. For 1969, these tonnages were:

$$\text{tonnage of Sn concentrate} = 1,136 / 50\% = 2,272 \text{ tonnes}$$

$$\text{tonnage of Cu concentrate} = 377.5 / 20\% = 1,887 \text{ tonnes}$$

Step 2.

The tonnage of tailings was estimated as the tonnage of mill feed less the tonnage of Sn concentrate, Cu concentrate and HMS floats, for example, in 1969 this was:

$$256,865 - 2,272 - 1,887 - 34,934 = 217,772 \text{ tonnes.}$$

Step 3.

The Sn as cassiterite grade of mill feed and HMS floats, Sn as cassiterite recovery of the mill circuit, and production of Sn in cassiterite concentrate were recorded in the mill operating figures.

The grade of Sn in tailings was estimated from these figures by subtracting the tonnage of Sn in HMS floats plus the tonnage of Sn in concentrates from the tonnage of Sn in mill feed, and dividing by the tonnage of tailings (from Step 2), for example in 1969 this was:

$$((256,865 * 0.85\%) - (34,934 * 0.21\%) - (1136)) / 217,772 = 0.44\% \text{ Sn}$$

Step 4.

The Cu grade of HMS floats was not recorded in the available data. For this estimate, the Cu grade of HMS floats was estimated to be 25% of the Cu grade of the feed. This was in line with the average Sn grade of the floats (0.17% Sn) versus the average Sn grade of the feed (0.68%

Sn) over the life of the mine (see the bottom line of Table 36). For example in 1969, the Cu grade of the floats was estimated to be:

$$25\% \text{ of } 0.40 = 0.10\% \text{ Cu}$$

Step 5.

The Cu grade of mill feed, Cu in copper recovery of the mill circuit, and production of Cu in copper concentrate were recorded in the mill operating figures. The Cu grade of HMS floats was estimated as described in Step 4. The grade of Cu in tailings was estimated from these figures by subtracting the tonnage of Cu in HMS floats plus the tonnage of Cu in concentrates from the tonnage of Cu in mill feed, and dividing by the tonnage of tailings (from Step 2), for example in 1969 this was:

$$((256,865 * 0.40\%) - (34,934 * 0.10\%) - (377.5)) / 217,772 = 0.28\% \text{ Cu}$$

Step 6.

The annual totals of tailings and contained Sn and Cu tonnes were summed over the life of the mine and the Sn and Cu grades were calculated.

The quantity and grades of the tailings have been estimated from the operating statistics of a competently run operating mill and are reasonably reliable. However, the spatial distribution of the tailings, both for tonnage and grades is not known and consequently the resource of tailings has been classified for this report as Inferred (see Table 35).

A zero cut-off grade has been applied to the estimate. It has been assumed that if the tailings are to be treated, all the tailings will be treated.

No assumptions have been made about mining factors or metallurgical recoveries.

A Sn equivalent grade was estimated from the Sn and Cu grades, assuming metal prices current at the time of writing, that is, US\$25000 per tonne for tin and US\$8500 per tonne for Cu a ratio of % Cu to % Sn of 0.34. This was considered reasonable given that, as in the past, both metals would probably be recovered during processing.

Table 35: Tin and Copper in Tailings Mineral Resources

Cleveland Tin and Copper in Tailings Mineral Resource				
31 March 2012				
0% Sn cut-off				
Category	Tonnage	% Sn	% Cu	% Sn Equivalent
Inferred	3,850,000	0.30	0.13	0.34
Total	3,850,000	0.30	0.13	0.34

Table 36: Cleveland Mill Operating Statistics 1968 to 1986 (mine life)
(Cleveland Historical Production 3 Apr 2013.xlsx)

Year	Ore Treated	HMS Floats				Head Grade		Contained Metal		Sn in Con	Sn Recovery from cassiterite	Cu in Con	Cu Recovery
		% Wt	Tonnes	Sn tonnes	% Sn	% Sn as cassiterite	% Cu	Sn in Feed	Cu in feed				
1969	256,865	14%	34,934	75	0.21	0.85	0.40	2,193	1,026	1136.0	51.8%	377.5	36.8%
1970	281,875	23%	64,831	92	0.14	0.79	0.37	2,234	1,040	1472.2	65.9%	717.4	69.0%
1971	305,726	27%	83,769	119	0.14	0.73	0.42	2,247	1,275	1557.2	69.3%	822.6	64.5%
1972	357,498	27%	97,954	207	0.21	0.78	0.40	2,789	1,414	1873.9	67.2%	968.5	68.5%
1973	505,806	25%	124,934	175	0.14	0.76	0.32	3,844	1,619	2667.9	69.1%	1216	76.0%
1974	314,210	23%	72,268	103	0.14	0.75	0.25	2,357	786	1512.5	64.6%	529.4	68.8%
1975	289,018	20%	56,648	108	0.19	0.78	0.32	2,254	925	1369.6	60.3%	565.1	60.7%
1976	363,036	30%	107,822	303	0.28	0.73	0.27	2,650	980	1519.1	60.0%	617.0	62.3%
1977	393,275	37%	143,939	292	0.20	0.66	0.22	2,596	865	1386	53.5%	420	48.7%
1978	388,579	38%	147,271	206	0.14	0.53	0.18	2,059	699	1236	59.6%	458	65.2%
1979	352,977	41%	144,721	202	0.14	0.52	0.24	1,849	847	1106	59.8%	422	49.2%
1980	367,866	43%	158,182	222	0.14	0.47	0.21	1,729	773	1080	62.7%	414	53.3%
1981	439,304	31%	137,359	234	0.17	0.51	0.22	2,240	966	1337	60.1%	518	54.7%
1982	350,300	31%	109,530	187	0.17	0.64	0.28	2,231	998	1457	65.3%	602	60.3%
1983	277,700	31%	86,830	148	0.17	0.71	0.25	1,973	692	1182	59.9%	405	58.5%
1984	180,300	31%	56,375	96	0.17	0.71	0.26	1,272	471	668	52.5%	300	63.7%
1985	137,000	31%	42,836	73	0.17	0.80	0.25	1,094	337	525	48.0%	198	58.8%
1986	83,700	31%	26,171	45	0.17	0.92	0.41	770	343	434	56.4%	140	40.8%
Total	5,645,035	30%	1,696,375	2,886	0.17	0.68	0.28	38,382	16,056	23,519	61.3%	9,691	60.4%

**Table 37: Cleveland Mill Estimation of Tailings Quantity and Grade
(Cleveland Historical Production 3 Apr 2013.xlsx)**

Year	Feed			HMS Float			Tailings		
	tonnes	%Sn as cassiterite	% Cu	tonnes	%Sn as cassiterite	% Cu	tonnes	%Sn as cassiterite	% Cu
1969	256,865	0.85	0.40	34,934	0.21	0.10	217,772	0.44	0.28
1970	281,875	0.79	0.37	64,831	0.14	0.09	210,512	0.31	0.12
1971	305,726	0.73	0.42	83,769	0.14	0.10	214,730	0.26	0.16
1972	357,498	0.78	0.40	97,954	0.21	0.10	250,953	0.27	0.13
1973	505,806	0.76	0.32	124,934	0.14	0.08	369,456	0.26	0.08
1974	314,210	0.75	0.25	72,268	0.14	0.06	236,270	0.31	0.09
1975	289,018	0.78	0.32	56,648	0.19	0.08	226,806	0.33	0.14
1976	363,036	0.73	0.27	107,822	0.28	0.07	249,091	0.32	0.11
1977	393,275	0.66	0.22	143,939	0.20	0.06	244,464	0.37	0.15
1978	388,579	0.53	0.18	147,271	0.14	0.05	236,546	0.26	0.07
1979	352,977	0.52	0.24	144,721	0.14	0.06	203,934	0.26	0.16
1980	367,866	0.47	0.21	158,182	0.14	0.05	205,454	0.20	0.13
1981	439,304	0.51	0.22	137,359	0.17	0.06	296,681	0.22	0.12
1982	350,300	0.64	0.28	109,530	0.17	0.07	234,846	0.24	0.13
1983	277,700	0.71	0.25	86,830	0.17	0.06	186,481	0.34	0.12
1984	180,300	0.71	0.26	56,375	0.17	0.07	121,089	0.41	0.11
1985	137,000	0.80	0.25	42,836	0.17	0.06	92,124	0.53	0.12
1986	83,700	0.92	0.41	26,171	0.17	0.10	55,961	0.51	0.31
Totals	5,645,035	0.68	0.28	1,696,375	0.17	0.07	3,853,169	0.30	0.13

9.2 Reliability of the Tin and Copper Tailings Resource Estimate

The nature of defects in the estimation and reporting for this resource estimate and their likely impacts on the quality of the resource report are listed in Table 10. The Assessment and Resource Reporting Criteria listed in Table 38 have been categorised using the terms listed under Nature of Defects.

Table 38: Tin and Copper Tailings Resource Estimate - Assessment and Reporting Criteria

Estimation and Reporting of Mineral Resources	
Database integrity.	Tolerable. The specific measures to ensure the integrity of the Aberfoyle metallurgical data are not known but, given that the data was collected at a large operating mill, it is reasonable to assume that the data is sound.
Geological interpretation.	Not applicable.
Dimensions.	Acceptable. TD1 is 300m long, 100m wide and a maximum depth of about 20m. TD2 is 400m long, 0m to 200m wide and has a maximum depth of about 35m.
Estimation and modelling techniques.	Tolerable. There is no model of the tailings deposits. The tonnages and grades were estimated directly from records of tailings reported as discharged from the Cleveland Mill.
Moisture.	Acceptable. All assays were reported on a dry basis.
Cut-off parameters.	Acceptable. The resource is quoted at 0.0% Sn (total) cut-off grade. Given that all the tailings will probably need re-treatment without selectivity, this is reasonable.
Mining factors or assumptions.	Not applicable. All the tailings will probably need re-treatment without selectivity, this is reasonable.
Metallurgical factors or assumptions.	Acceptable for Inferred Category. Foo (1981) considered that mill recoveries from treatment of run of mine ore of 65% for Sn could be maintained under best operating conditions at the time. This is

	<p>considerably better than the mill recoveries during the routine operation of the mill up until that time. This implies that some, at least, of the tin in the tailings dams should be recoverable.</p> <p>Stribley et al. (1984) reported that mill recoveries from pilot scale treatment of tailings of between 33% and 45% for Sn were attainable using conventional gravity and flotation processing and 48-69% Sn recovery using pre-concentration by flotation and matte fuming.</p>
Bulk density.	<p>Not applicable.</p> <p>Tailings discharge from the Cleveland Mill was measured in tonnes.</p>
Classification.	<p>Acceptable for Inferred Category.</p> <p>The resources were not classified in the historical estimates. However, given they would be classified as Inferred Mineral Resources today.</p>
Audits or reviews.	<p>Tolerable.</p> <p>The method of estimation of the tailings resource was reviewed by Mike Adams of Rockwell.</p>
Discussion of relative accuracy/confidence	<p>Acceptable.</p> <p>The quantity and grades of the tailings have been estimated from the operating statistics of a competently run operating mill and are reasonably reliable. However, the spatial distribution of the tailings, both for tonnage and grades is not known and consequently the resource of tailings has been classified for this report as Inferred.</p>

10 EXPLORATION POTENTIAL

The Cleveland tin copper deposit is open at depth and along strike. In addition, there are areas within the current extents of the mine where extensions to the known deposit are likely.

In the short term, there are targets which can be tested by relatively short holes drilled from within the mine either from existing development or from new development headings. New development will be required to gain access to drilling platforms which will allow for drilling of internal extensions of known lodes, internal exploration for new lodes and infill down-dip drilling of known lodes. Underground drilling cannot be undertaken until the mine is sufficiently dewatered and rehabilitated.

In the short term, there are two ways to add to the existing resource inventory:

- estimation of the, so far unestimated, remnant resources, and
- small possible additions to near surface or outcropping lenses.

Estimation of the remnant resources depends on acquiring reliable three dimensional data of what has been mined out to date. At present, we only have access to three dimensional data (In Dronseika, 1986) of what has been mined, so, only the remnants that lie along strike and up and down dip of a lens can be estimated. The remnants which occur on the hangingwall and footwall of a lens cannot be estimated.

Acquiring a reasonable three dimensional view of the mine, from a comprehensive digitised set of plans and sections, would enable a reasonable estimate of the remnants to be made adding to the known resource. In the short term, this is feasible using the plans and cross-sections made at mine closure that would then enable an estimate of the remnant resources to be made.

There are at least two shallow targets for lenses which outcrop or come close to outcrop, notably Luck's and Henry's No 1 pod. It is possible that short holes drilled into these targets could add some modest tonnages to the current resources.

The tungsten mineralisation in Foley's stock-work has only been estimated down to 850m RL (see 8.5 Tungsten Resource Estimate). There is considerable scope to increase the tungsten resource below this level (for example, refer to Figure 15). This work will have to wait until the mine is sufficiently dewatered and rehabilitated and will likely require some extra mine development to provide appropriate drilling platforms.

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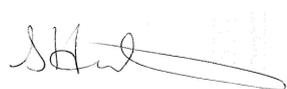
DOCUMENT INFORMATION

Status	Final
Version	2
Print Date	11 th April 2013
Author(s)	Mick McKeown
Reviewed By	Stuart Hutchin
Pathname	P:\5119_G Bright Phase Resources Cleveland Mine\WPO\3555v2.doc
File Name	3555v2.doc
Job No	5119_G
Distribution	PDF emailed to client

DOCUMENT CHANGE CONTROL

Version	Description of changes/amendments	Author (s)	Date
2	Final Report	M. V. McKeown	11-04-13

DOCUMENT REVIEW AND SIGN OFF

Version	Reviewer	Position	Signature	Date
2	Stuart Hutchin	Geology Manager		11-04-13