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Davenport Resources Pty Ltd

(ACN: 153 414 852)

Hatches Creek (EL 22912 and EL 23463)

Annual Technical Report

25th July 2014 to 24th July 2015

Combined Report #: GR262

Target Commodities: Tungsten, Gold, Copper, Bismuth

25th July 2014 – 24th July 2015
Prepared by: Mick Wilson
Date: 9th September 2015
Distribution: Davenport Resources Pty Ltd, GWR Group Limited
NTDOR/NTGS, Darwin, NT

Summary Sheet

Titleholder	Davenport Resources Pty Ltd
Operator	Arunta Resources Limited
Tenements	EL 22912 & EL 23463
Combined Report #	GR262
Tenement Manager / Agent	Australian Mining and Exploration Title Services (AMETS)
Mine / Project Name	Hatches Creek
Report Title	Hatches Creek (EL 22912 and EL 23463) Annual Technical Report 25th July 2014 to 24th July 2015
Personal author (s)	Wilson, M
Corporate author	Arunta Resources Limited
Target Commodity or Commodities	Tungsten, Gold, Copper, Bismuth
Date of report	15th September 2015
Datum / Zone	GDA94/Zone 53
250,000 Map sheet	Frew River (SF 53-3)
100,000 Map sheet	Hatches (5956)
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Abstract

The Hatches Creek tungsten project consists of two tenements EL 22912 and EL 23463, located approximately 320km north east of Alice Springs. Work undertaken during the year ending 24 July 2015 consisted of resource estimation, metallurgical testwork and a sacred site survey.

A maiden Inferred Resource estimate of the surface mineralised stockpiles and dumps surrounding the historical mine workings was completed identifying an Inferred Resource of 225,000 tonnes grading 0.58% WO_{3}.

Investigative metallurgical test work was completed on two bulk samples collected from Pioneer and Treasure waste dumps and this has shown that it is possible to produce a 42`% WO_3 *concentrate with a recovery of 66%. The test work has also produced significant grades of Cu, Mo, and Au in the pre-float sulphide concentrate, showing potential for these to be recovered as saleable by-products. The concentrates from Pioneer also produced high Bi which is deleterious, however high Bi is only present at Pioneer and Green Diamond*

A sacred site survey was undertaken by Central Land Council (CLC) on site on the 15th June 2015 and a site meeting was held with the Traditional Owners on the 16th June

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1.0 Summary

During the period 25th July 2014 to 24th July 2015, activities undertaken included;

- A maiden Inferred Resource estimate of the surface mineralised stockpiles and dumps surrounding the historical mine workings was completed yielding an Inferred Resource of 225,000 tonnes grading 0.58% WO_3
- Investigative metallurgical test work was completed on two bulk samples collected from Pioneer and Treasure waste dumps and this has shown that it is possible to produce a 42`% WO_3 concentrate with a recovery of 66%, as summarised below. The test work has also produced significant grades of Cu, Mo, and Au in the pre-float sulphide concentrate, showing potential for these to be recovered as saleable by-products. The concentrates from Pioneer also produced high Bi which is deleterious, however high Bi is only present at Pioneer and Green Diamond

WO_3 Summary	**WO_3**	**Circuit yield WO_3**
	%	**%**
Super Concentrate	66.30	27
Concentrate	36.50	39
Concentrate - Total	**42.70**	**66**
Middlings	**4.30**	**9**
Tailings	**0.10**	**25**
calc head	**0.77**	**100**

- A sacred site survey was undertaken by Central Land Council (CLC) on site on the 15th June 2015 and a site meeting was held with the Traditional Owners on the 16th June

Activities planned for 2015 -2016 include the following;

- Flying of an unmanned aerial vehicle ("UAV") survey, to collect high quality aerial photography and 3D imagery
- Completion of a scoping study level economic assessment of the viability of the dumps project
- Initiation of systematic exploration to test hard rock potential

Introduction

The Hatches Creek project is located in the central portion of the Northern Territory. The project area is located approximately 325km north east of Alice Springs and 160km south east of Tennant Creek (Figure 1)

The tenements cover the historical Hatches Creek mining field, which was known as the Hatches Creek Wolfram Field, within which numerous underground mines exploited quartz veins containing wolframite and to lesser extent scheelite, bismuth and copper mostly to the water table or just below it. Mining of eluvial deposits containing wolframite, gold and copper also occurred.

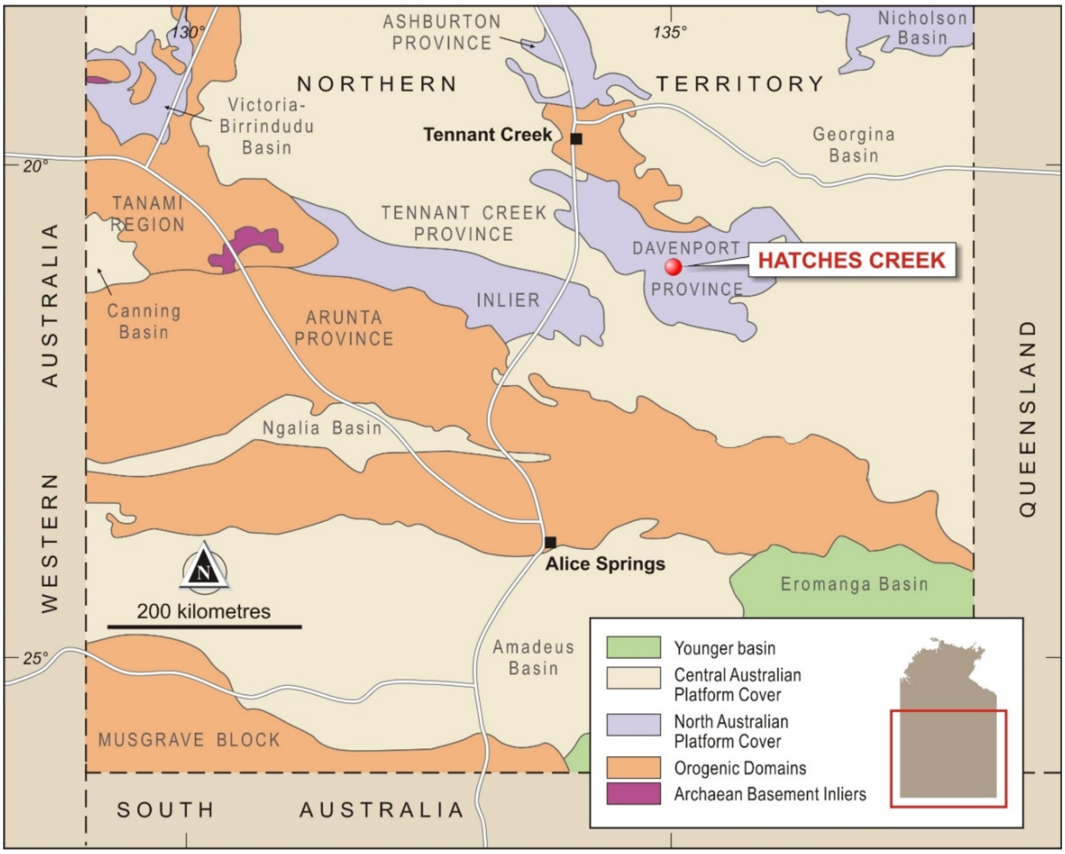

Figure 1; Hatches Creek Location Plan

The tenements are on Aboriginal Freehold Land which means access to the area is restricted to permit holders only by the CLC. Formal work programmes must be lodged with CLC to allow any ground disturbing activities to commence (i.e. clearing and drilling).

3.0 Tenure

The Hatches Creek project comprises two granted exploration Licences EL22912 and EL23463. The tenements are held by Davenport Resources Pty Ltd, which is a 100% owned subsidiary of Arunta Resources Limited. In January 2015, NT Tungsten Pty Ltd (100% owned subsidiary of GWR Group Limited) executed a binding Heads of Agreement with Davenport, where it can earn a 50% interest by spending $1,500,000 over a period of two years on project development and exploration. GWR are currently managers of the project.

Tenement details are summarised in Table 1 and Figure 2

Table 1
Tenement Summary

Tenement	Area	Status	Grant Date	Expenditure Commitment
EL22912	8 Blocks	Live	25/07/2007	$83,375
EL23463	2 Blocks	Live	25/07/2007	$71,875

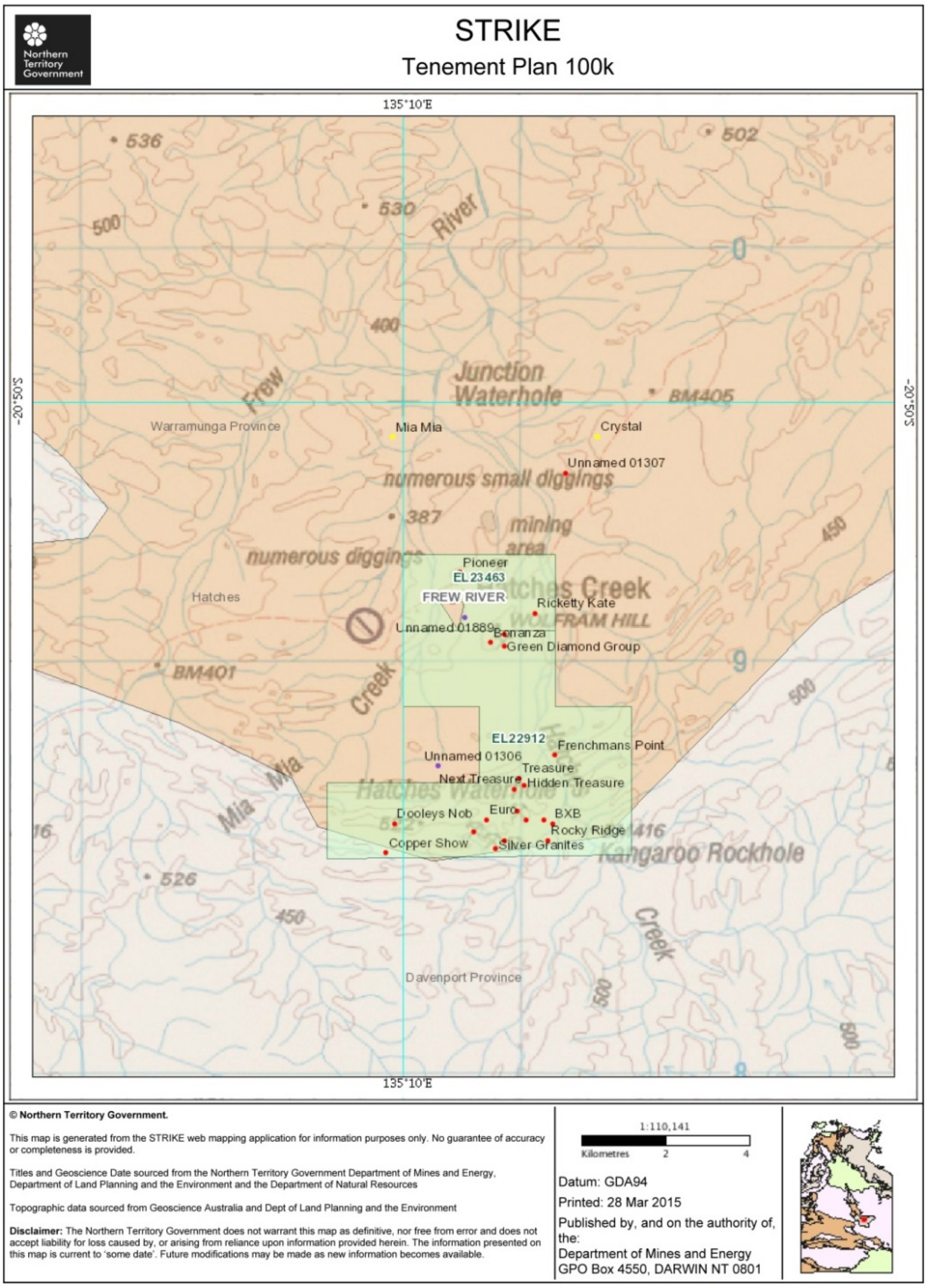

Figure 2

Figure 2; Hatches Creek Tenement Plan

4.0 Geology

The Hatches Creek tenements are underlain by a Palaeoproterozoic sequence of weakly metamorphosed clastic sedimentary and felsic volcanic rocks. The sequence is intruded by mafic igneous sills, with sandstone as the dominant sedimentary lithology. The sequence has been subjected to folding and faulting and has been cut by numerous narrow quartz reefs which follow lines of shearing. The quartz reefs are mineralised, the main minerals of economic interest are wolframite and scheeelite, although bismuth, gold and copper mineralisation is also present within them. The average tungsten grade of the mined reefs was between 1% and 5% WO3.

The mineralised reefs are present in groups. The average reef width is 30cm, with the maximum width being 1.5m. The maximum strike length of any one reef is around 170m however en echelon lines of reefs are up to 1.5km in length. The reefs strike in two main directions, just east of north, parallel to the main fault direction, and east-northeast. The north-striking reefs dip at moderate to steep angles either to the west or the east; those striking easterly dip at moderate to steep angles to the south. The majority of the reefs are within volcanic or intrusive rocks, rather than in the sandstone units. The more mafic host rocks (gabbro, diorite) appear to have been important host rocks for some of the significant mineralisation in the area.

5.0 Previous Exploration

In 1937 a brief ground reconnaissance of the area was made by officers of A.G.G.S.N.A., and an area, including the mining field, was photographed from 12,000 feet. This was followed by a detailed survey of the mines and the geology of the surrounding country in 1940. The area was photographed again, from 5,500 feet, in the same year (A.G.G.S.N.A., 1941).

During the Second World War the demand for wolfram increased; considerable interest was shown in the field and several brief reports were prepared (Knight, 1942; Raggatt, 1943; Sullivan, 1943).

C. J. Sullivan examined the Treasure Mine in 1951 (Sullivan, 1951) and contributed a section on Hatches Creek in The Geology of Australian Ore Deposits' (Sullivan, 1953).

E. B. Jensen (1955) dealt with the treatment of the complex wolfram-scheelite-bismuth ore from the Pioneer Mine.

A ground party from the Bureau of Mineral Resources completed a regional geological survey of the Davenport Range in October, 1956 (Smith, Stewart, & Smith, 1960). In the same year an airborne scintillometer survey of selected areas in the Davenport Range commenced also conducted by the Bureau of Mineral Resources, and was completed in 1957. Maps showing the results of this survey were released in late 1957.

In 1968 Northern Territory Minerals were granted an Authority to Prospect over a portion of the Hatches Creek Wolfram field. Several costeans were dozed over numerous prospects with varying degrees of success, and a diamond hole was completed at the treasure deposit. The tenement was relinquished in 1970.

Minscope were granted an exploration licence in the area in 1987 and focused on gold exploration. Several costeans were dozed over the different deposits with some anomalous intersections but nothing of economic significance was intersected.

Exploration in 2008 by Thor (Hatches Creek Pty Ltd) staff included a site meeting with the CLC and Traditional Owners. Reconnaissance mapping and sampling of old workings was also permitted whilst in the area. The visit confirmed that mineralisation in the Hatches Creek area occurs as narrow quartz veins (up to 1.5m thick and extending up to 200m in individual lenses) hosting exceptionally high grade tungsten as wolframite crystals several centimetres in length in some cases. Virtually all veins visited at old prospects and exploration pits had visible wolframite evident. Historical underground mining at most prospects has been very efficient and removal of vein material has been extensive in some cases to the water table or just below.

174 rock chip samples (Figure 2) were collected over old workings and from abandoned stockpiles associated with the historic mining centre. Assaying included Au, Ba, Fe, Mo, Sn, W, Cu Ni, Pb, Ag, and Bi.

A strong gold association was noted at the Pioneer mine area with maximum gold results of up to 7.24ppm Au and an average of 0.92ppm Au. At Hit and Miss group a number of samples also returned favourable Molybdenum assays up to 1.98% from the Chinaman's Shaft area.
Prospects visited during the 2008 reconnaissance included the following:

Pioneer Mine Area

Significant gold assays were received from the Pioneer workings at the north end of the Wolfram Field, with gold results up to 7.24g/t Au returned from mullock dumps at the Pioneer Mine. It is also interesting to note that anomalous gold assays were returned from battery sands at the mine up to 2.48g/t Au. Visible wolfram was noted in many of the quartz rock chip samples with a number of results >1% W, average of the 27 samples taken returned 1.16% W and 0.3% Bi.

Green Diamond Group

At Green Diamond tungsten mineralisation in the Hatches Creek area occurs as narrow quartz veins (up to 1.5m thick and extending up to 200m in individual lenses) hosting exceptionally high grade tungsten as wolframite crystals several centimetres in length. Virtually all veins visited at old prospects and exploration pits had visible wolframite evident. Thirty eight samples were taken returning an average grade of 1.42% W with a maximum of 7.22% W. Significant copper mineralisation was evident at numerous locations also associated with quartz veining occurring as malachite and/or azurite. High grade copper mineralisation was evident at the Green Diamond Prospect where quartz vein mullock from deeper parts of the workings exhibited massive sulphide veining containing bornite and chalcopyrite. High concentrations of malachite and azurite were also noted with four sample returning results of >1% Cu.

Hit and Miss Group

Results from quartz vein samples at Hit and Miss returned and average grade from 23 samples of 2.9% W, with a maximum of 13.3%. Other than the obvious wolframite mineralisation, significant molybdenite was noted in mullock at the "Hit & Miss" workings near Chainman's shaft with assays returning up to 1.98%. The workings are associated with one of the largest veins noted to date although numerous other parallel vein sets occur at the prospect. Not tabled here are the anomalous Ag assays of up to 46pmm and Sn to 0.2%.

Copper Show Group

The "Copper Show" workings were also located and are worthy of some follow up work as the main lode is easily traceable in later costeans that have been excavated. Along with the ubiquitous wolframite mineralisation the main lode exhibits significant malachite and azurite. The prospect area is relatively flat lying surrounded by hills.

Treasure Group

The Treasure Group workings cover an extensive area that occupies two valleys and the sides of the adjacent hills in the central part of the Hatches Creek Wolfram Field. The Treasure Group also boasted a general store a generator shed and large living quarters and office the ruins of which are still evident. Workings extend to the west previously known as "Next Treasure". The results returned consistently high tungsten assays from visible wolframite in quartz veining, the average tungsten grade returned 3.84% W. Only minor copper was noted in hand specimens in this area which is reflected in the assays.

Work by Thor in 2009 included locating all historical prospect maps based on GPS data collected from the reconnaissance visit in the previous year. All historical aerial photography was acquired to aid in locating old tracks and prospects. Recent colour photography was not an effective tool for locating prospects and in particular tracks that have not been disturbed for around 50 years. Significant potential exists for additional mineralisation under cover between the northern and southern areas of EL22912.

Work undertaken by Davenport Resources Pty Ltd from 25th July 2013 to 24th July 2014 focused on establishing an estimate the above ground resource contained within historic mine dumps and stockpiles. A total of 106 bulk and rock sample were taken from historically mined areas; Pioneer, Black and Green Diamond, Treasure Group, Hit and Miss and other minor historic workings. Results from the sampling work indicated that grade of the waste dumps and tailings averaged between **0.5% and 1% WO3**

A volume survey was also completed establishing that approximately 200,000 tonnes of rock material in the form of waste (mullock) dumps and battery tailings could be available for treatment. Preliminary metallurgical test work was undertaken and this indicated that economic recoveries could be achieved with the best result of coming from the Hit or Miss Area with test work on the composite sample C (generated from the mullock sampling) producing a concentrate **of** 47.5% WO3 recovering 74% of the contained WO3.

6.0 Year 7 Work Completed

During the year, a maiden Resource estimate of the mineralised surface stockpiles, metallurgical testwork and a sacred site survey was completed

6.1 Resource Estimate

Independent mineral resource consultant; Mr. Tony Ryall, was commissioned by Arunta to complete a JORC compliant resource estimate for the stockpiled surface mineralisation. This identified an Inferred Resource estimate of 225,000 tonnes @ 0.58% WO3 (0.2% WO3 cut off) as summarised in Table 2. The independent resource report is provided in Appendix 1

Table 2

Inferred Resource Estimate for Hatches Creek Historic Mine Dumps and Alluvial above a 0.2%WO^3 Cut Off

Prospect Location	Material Type	Volume (M3)	In Situ Density (g/cc)	Dump Compact Bulk Density	Resource Tonnes	Weighted Grade uncut%WO3:	Weighted Grade Cut 1.5%WO3
Pioneer	BS	14,260	2.80	2.60	37,076	0.60	0.60
Pioneer	MD	19,426	2.85	2.50	48,565	0.80	0.70
Treasure	MD	11,516	2.85	2.50	28,790	0.78	0.76
Hit or Miss	MD	9,351	2.80	2.45	22,909	0.76	0.68
Black Diamond	MD	6,262	2.80	2.45	15,342	0.58	0.57
Green Diamond	MD	1,252	2.80	2.45	3,067	0.86	0.86
Copper Show	MD	1,012	2.80	2.45	2,479	1.79	0.96
Masters Gully	MD	4,301	2.80	2.45	10,537	0.40	0.40
Hens and Chickens	MD	2,008	2.80	2.45	4,919	0.40	0.40
White Diamond	MD	1,564	2.80	2.45	3,831	0.40	0.40
Silver Granite	MD	3,664	2.80	2.45	8,977	0.40	0.40
Kangaroo Group	MD	4,112	2.80	2.45	10,074	0.40	0.40
Alluvial All Areas	AL	15,000	2.50	1.90	28,500	0.25	0.25
Total		**95,773**			**225,066**	**0.62**	**0.58**

*BS: Battery Sands Dumps
*MD: Historic “Mullock” Dumps
*AL: Alluvials

6.2 Metallurgical Testwork

In October 2014, Davenport Pty Ltd collected three bulk samples of approximately 500kg each from the Pioneer, Green Diamond and Treasure dumps respectively. In January 2015 the Pioneer and Treasure samples were submit to Nagrom laboratory in Perth for metallurgical test work to determine if a viable WO3 concentrate could be produced. Extensive testwork was undertaken between January 2015 and June 2015

Extract Mineral Processing (“EMP”) was commissioned to manage the testwork program and in consultation with Nagrom evaluate the results and decide upon the direction of the test work.

In July 2015 EMP prepared a comprehensive report on the test work results and this can be found in Appendix 2. A summary of the report’s findings are provided below;

Summary of EMP Report July 2015

An investigative metallurgical test work program on 2 bulk samples of Hatches Creek Pioneer and Treasure material has been completed at Nagrom. The aim of the program was determine the amenability of the samples to upgrade WO_3 using gravity, magnetic characterisation and flotation test work.

The test work has upgraded WO_3 from a calculated combined Pioneer/Treasure head grade of 0.77% WO_3 to 42.7% WO_3 concentrate with an overall circuit recovery of 66%. A super-concentrate grading 66.3% WO_3 and 27% circuit recovery resulted from flotation test work summarised in Table 3

Table 3
Summary of Metallurgical Testwork Results

WO_3 Summary	WO_3 %	Circuit yield WO_3 %
Super Concentrate	66.30	27
Concentrate	36.50	39
Concentrate - Total	**42.70**	**66**
Middlings	**4.30**	**9**
Tailings	**0.10**	**25**
calc head	**0.77**	**100**

The test work has also produced significant grades of Cu, Mo, and Au in the pre-float sulphide concentrate, having potential to be recovered as saleable by-products.

Bi levels in the concentrate (1.5% or 10500 ppm) are well above the industry standard for APT and FeW production. Upper limits on Bi on WO_3 concentrates for APT production is 0.5%. The general specification requirement is 0.15% max with higher levels incurring price penalties in the order of \$2/mtu for 0.15- 0.3% Bi, \$4/mtu for 0.3-0.5% Bi and \$6/mtu at 0.5%.

Due to money and sample mass constraints, the number of Bi -50% concentrate and middlings flotation tests were inadequate to fully optimise the flotation conditions. No flotation test work was conducted on the tailings despite it containing 25% of the overall circuit WO_3. This means that the overall result of the test work program is indicative only, with upside potential. The test work results have however provided enough information to have confidence that there is sufficient WO_3 grade and recovery to progress to a scoping level study at Hatches Creek

Depending upon the outcome of the scoping study and nature of the project moving forward, it is recommended that any metallurgical test work required should include, but not limited to;

1. Establishing a full set of physical characteristics including hardness, crushability and abrasion indices that will feed into a process design.

2. Increasing the recovery of WO_3 from the middlings and tailings by;
    - ➢ Targeting increased depression of Si in the middlings to improve the WO_3 concentrate grade.
    - ➢ Optimisation of WO_3 collector in the middlings to maximise WO_3 recovery.
    - ➢ Developing a test work plan aimed at increasing recovery of WO_3 from the tailings.
3. Optimisation of the roughing and cleaning stages for the -50% concentrate.
4. Cleaning of the final concentrate to a standard specification grade for a WO_3 concentrate (~65%).
5. Reduction of Bi in the intermediate and/or final concentrate to a saleable specification.
6. Recovery of potentially valuable by-products including Au, Cu and Mo.

6.3 Scared Site Survey

A sacred site clearance survey was undertaken by Central Land Council and Traditional Owners ("TO's") at the Hatches Creek project on the 15th June 2015. A formal report has yet to be received from CLC.

On the 16th of June a liaison meeting was held with the TO's on site at Hatches Creek. This meeting was attended by approximately 30 TO's, representatives of CLC including David Young and Mick Wilson and Allan Ashwin (GWR employee and Aboriginal Martu Elder from Wiluna) representing GWR.

7.0 Proposed Exploration Activities

The following activities are planned for the coming year;

7.1 UAV survey

An unmanned aerial vehicle ("UAV") aerial photography survey over 5 areas at Hatches Creek is planned . The purpose of the survey was to collect high quality 3D imagery of the major mining areas which will allow very accurate volumes of the historical mine dumps to be collected and as such increase confidence in the Inferred Resource. The high quality aerial photography / photogrammetry should also facilitate the creation of a GIS based database of the historical mines and workings as defined in Bulletin 6 (1961).

7.2 Additional metallurgical testwork

It is planned to collect additional bulk samples from the major historical mine dumps and further testwork be undertaken to;

- Determine size analysis at various screen sizes
- Determine loose and compact bulk density's
- Determine if the deposits are amenable to ore sorting technologies

7.3 Scoping study

Complete a scoping study level economic assessment to determine the viability of the dumps project.

Appendix 1
Independent Resource Report

HATCHES CREEK MINERAL FIELD TUNGSTEN RESOURCE ESTIMATES HISTORIC MINED DUMPS AND ALLUVIAL CHANNELS INDEPENDENT CONSULTANT REPORT

FOR ARUNTA RESOURCES LTD

RESOURCE INVENTORY FOR HATCHES CREEK MULLOCK DUMPS AND ALLUVIAL

Prospect Location	Material Type	Volume cu m	In Situ Specific Gravity g/cc	Dumps: Compact Bulk Density	Resource Tonnes	Weighted Grade %WO3: No Upper cut:	Weighted Grade Upper Cut 1.5%WO3	Category And Confidence Level
Pioneer	BS	14,260	2.80	2.60	37,076	0.60	0.60	Inf- 1
Pioneer	MD	19,426	2.85	2.50	48,565	0.80	0.70	Inf- 1
Treasure	MD	11,516	2.85	2.50	28,790	0.78	0.76	Inf- 1
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Kangaroo Group	MD	4,112	2.80	2.45	10,074	0.40	0.40	Inf-2
Alluvial All Areas	AL	15,000	2.50	1.90	28,500	0.25	0.25	Inf-3
Total		**95,773**			**225,066**	**0.62**	**0.58**	**Inferred**

*BS: Battery Sands Dumps
*MD: Historic "Mullock" Dumps
*AL: Alluvials

Key Notes and Assumptions:
1 Excellent documented geological and historic mining production records as background support for these resource estimates.
2 Lower cut grade 0.2%WO3 applied throughout.
3 Upper cut grade 1.5% WO3 based on 97th percentile log probability
4 In Situ SG based on Nagrom Laboratory gas pychnometry on 30 samples from various prospects and material types
5 Density reduction from SGs for compacted dumps (voids / moisture) assumed: approx. 10% for Battery Sands, 15% for Mullock Dumps (coarser) and 25% for Alluvials

6 All Bulk sample assays from two programs in Nov 2013 and June 2014 applied- 110 samples. Rock chip sample assays not applied to this resource estimate do confirm very high grade tenor of veined tungsten mineralization.
7 Each prospect is considered a separate resource
8 In all cases un sampled dumps have been assigned the weighted average grade of sampled dumps.
9 **Confidence Level 1:** Prospects with significant sample assay data (Pioneer, Treasure, Hit or Miss, Black Diamond, Green Diamond, Copper Show) are at higher level of confidence .
10 **Confidence Level 2:** Prospects not sampled mostly have been assigned a more conservative grade reflecting a lower recorded production.
11 **Confidence Level 3**: Alluvials, based on estimated average thickness of 1metre and 30% of alluvial area above cut- off grade. Inferred Resource is at a lower level of confidence and reduced grade, but based on sampling, metallurgical testing and field evidence/ historical – has significant potential .
12 Volumes of dumps measured from average dump area at base x height and GPS located. In total 166 dumps measured this way.
13 Metallurgical testing on composite dump samples shows high tungsten concentrate grades from (29% to 47% WO_3) with up to 78% recovery achieved via a simple gravity and magnetics circuit

SUMMARY:

The Hatches Creek Minerals Field located almost 500kms NE of Alice Springs was mined in the early part of the 20th Century for tungsten and other by products. A number of prospects were mined in tandem with periods of an elevated tungsten price. Because of the very high grade nature of this tungsten mineral field a lot of hand sorting meant that material of high grade has been retained in mullock and battery sands dumps as well as alluvial drainage systems. Records of the geology and tungsten production from at a dozen different prospects is very well documented.

The high grades reported in bulk dump samples , the basis of this resource estimation is evident from recent work by Arunta, with rock chip and bulk sampling programs conducted. This work has included dump surveys and metallurgical testwork on composite samples.

Approximately 100 bulk samples collected from 166 dumps in two separate programs form the basis of current resource estimates , although a number of the regional prospects and associated have also been considered on the basis of production records, similar geology, and demonstrated alluvial potential.

In summary an Inferred Resource of just over 200,000 tonnes at over 0.5% WO3 mainly from existing mullock dumps and alluvial material can be realistically estimated for the main prospects with

Database

The database used for this resource estimate of historic stockpiles of mined material has been provided by Arunta Resources from data collected since their acquisition of the project during 2013.

The database is based on two bulk sampling programs conducted over 65000 cu meters of stockpiles (166 stockpiles) covering 5 material mined from 5 main sets of workings: Pioneer, Treasure, Hit or Miss, Black Diamond and Green Diamond.

The sampling protocols have been described in an earlier ASX release by the company and reported within the JORC compliance statement accompanying that release by John Young of Arunta Resources.

It is considered that there is sufficient data that an Inferred Resource for this material can now be reported.

The estimate is further supported by metallurgical testwork, based on volume to tonnage assessments made with supportive density data.

A higher level of QA/QC would be required to bring this to Indicated.

The sampling has included

- Photography of the various materials and rock types
- Detailed location and descriptions of material type and geology for each sampled stockpile
- A good surface understanding of structural controls on the mineralization and plan maps illustrating sample locations in relation to surface geology and workings.
- Assessment of the volume of each stockpile based on application of a hexahedron shape assessing the basal area the height and the upper area using GPS for better control on larger stockpiles and by visual estimate in smaller.
- A total identified 166 stockpiles for a total estimated volume of approximately 65,000 cu meters of stockpile material.
- Sampling has been conducted on approximately 70% of this dump volume material. In total approximately 100 samples have been collected as typically 10 to 20kg samples, from several spear positions around stockpiles.
- Sampling of stockpiles has been carried out from spear sampling with a sufficient number and volume of stockpiles sampled to provide a representation of grade range.
- Importantly the focus has been to sample the larger stockpiles as therefore covering a larger representative material volume than sampling of smaller stockpiles.
- Gas pychnometer SG has been applied to 30 samples covering all 6 prospects and range of mullock dump. Battery stockpile and alluvial samples representing the three sample sources. All SGs lie in the range of 2.7 to 3.05 and mostly within the range of 2.75 to 2.9, consistent with the lithologies hosting the quartz wolfram veined structurally complex mineralization.
- These SGs are in situ and consistent with reported averages for dolerites, acid volcanic and sedimentary host rocks.
- These samples are from stockpiles and despite compacted over 60 years – the rocks have an inherent porosity and significant voids component.
- All assaying has been conducted by Nagrom Laboratories in Perth.
- Sampling and assaying has been reported to ASX previously through the JORC compliance process by John Young of Arunta Resources.
- In some cases stockpiles have more than one sample assigned. The nature of the veined mineralization inherently means there is high deposit variance associated with WO_3 grade estimation, particularly in mullock dump sampling.
- However there is enough appropriate size and methodology in the sampling to see consistency in character within each prospect- so local geological influence on grade is broadly evident.

- The underground geology mined can only be learned and understood from historic records and geological mapping from pre 1950s.
- In most operations sampled stockpiled ore from a known area of an orebody that was grade control drilled would be classified as Measured Resource. In this case the source of stockpiled material is not certain and with this and other limitations categorizing as Inferred is considered appropriate.
- A further fact is the known high grading that occurred in past mining so that the possibility of positive nugget effect may exist
- Rock chip sampling was also conducted and demonstrates significantly higher grade than reported from stockpile sampling. The rock chip assay results range from to and have not been applied in this estimate, as they are considered to bias results.
- Dumps that were not sampled are assumed to contain the average grade of that Prospect and Material type --- this seems a fair call
- Dumps that have been multiple sampled – the average grade is used

Summary of Dump Surveying and Sampling

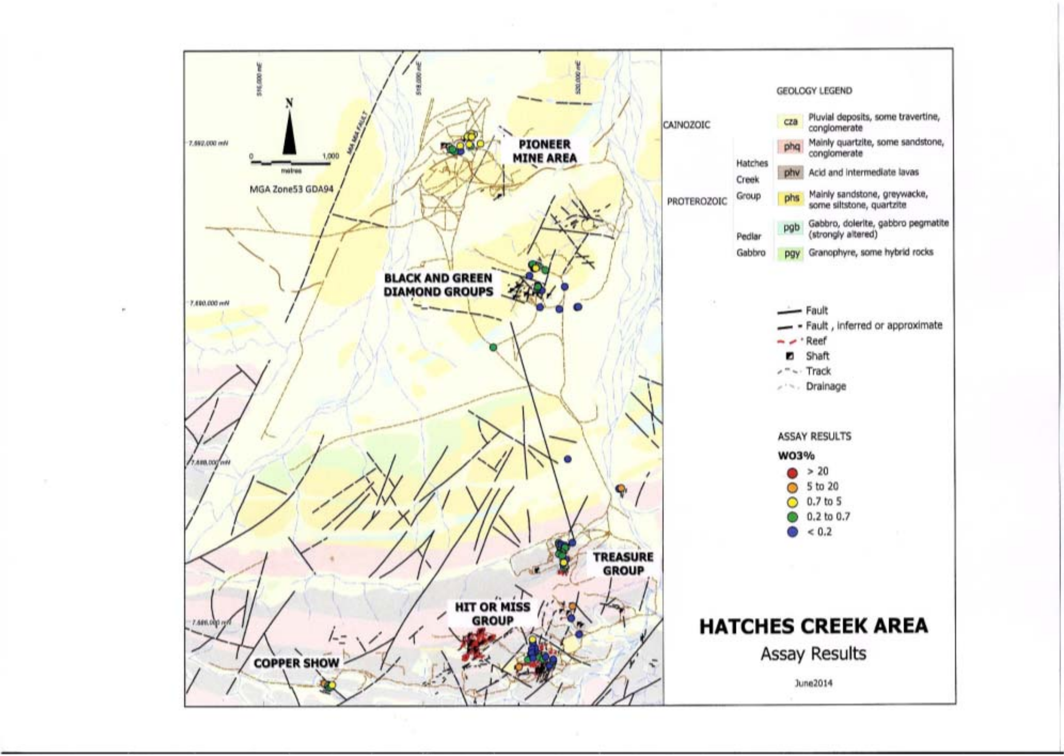
Workings	**Volume m^3**	**Sample Nos Applied**
Pioneer Battery Sands	14,260	PRO2-4,6,19-23, HCB015,016,021,022
Pioneer Mullock + Alluvial	20105	PRD1,5,8,13,15-18,24-34 PRD09,10
Green Diamond	1,252	GDD01-03, HCB031,032
Black Diamond	6,262	BDD01-10, HCB023/024
Treasure	12,130	TRD01-29, HCB033-040
Hit or Miss	11,148	HOM01-14,HCB007-014
Masters Gully*	4,301	
Hen&Chickens*	2,008	
White Diamond*	1,564	
Silver Granites + others*	3,664	
Copper Show	1,012	HCB001 -006
Kangaroo Group*	4,112	
Total Volume Survey at Hatches Creek	**81,819 m^3**	

*Samples not taken at these workings

Bulk Sampling Summary Statistics

	Material Type	**Total No of Dumps Recorded**	**Total No of Dumps Sampled**	**Dumps Sampled % Number**	**Total Surveyed Dump Vol m3**	**Total Volume of Dumps Sampled**	**Dumps Sampled %Volume**	**#.Sub grade dumps <0.2%WO3**
Pioneer	BS	12	6	**50**	14,260	10,408	**73**	NIL
Pioneer	MD	21	5	**24**	19,426	11,732	**60**	NIL
Treasure	MD	29	14	**48**	11,516	8,147	**71**	3
Hit or Miss	MD	67	14	**21**	11,148	6,167	**55**	9
Black Diamond	MD	33	8	**24**	6,262	4,429	**72**	NIL

Green Diamond	MD	3	3	**100**	1,252	1252	**100**	NIL
Copper Show	MD	5	5	**100**	1,012	1,012	**100**	NIL
Total		172	55	**30**	63,864	42,135	**66**	12



Sampling Methodology

As for the collection of samples in November 2013 during the course of the June survey a total of 51 samples were collected from various waste dumps and occasionally with two or three samples collected from individual waste dumps. The location of the sample was selected by the geologist with an eye to selecting a representative sample from the dump. Material collected for the sample was of gravel size and smaller with all cobble sized material being left out. GPS coordinates were recorded for each sample site. The sample was collected using a shovel into the stockpile and about 10kg of material was placed in a polyweave bag or two calico sample bags. Samples were occasionally collected from various parts of the pile.

- Sampling has been conducted on approximately 70% of this dump volume material from these prospects , typically 10 to 20kg samples, from several spear positions around stockpiles.
-
- Sampling of stockpiles has been carried out sufficient stockpiles and volume sampled to provide a good representation of grade range.

- Importantly the focus has been to sample the larger stockpiles as therefore covering a larger representative material volume than sampling of isolated smaller stockpiles.

Dump Survey Methodology

At the start of the survey it was decided to try two different methods to survey waste dumps. For the one method the top and basal area of the waste dump was measured using the area measuring facility present on the Garmin GPS. This application allows the user to measure the size of an area by walking around it with the GPS. It was decided to measure the top and basal areas of waste dumps with the GPS and then estimate the height of the waste dump. This data would then be used to calculate the volume of the dump treating it as a cone.

In the second method the waste dump was treated as a hexahedron and its area measured by pacing the top of the dump to determine its length and width. As the dumps had sloped sides it was necessary to include a correction factor to the length and width so as to allow for the calculation to be performed treating the dump as having vertical sides. For both methods the height of the dump was visually estimated. This was thought to provide sufficient accuracy as many of the dumps were situated on sloped surfaces and could vary in height from as much as 3 to 10m in some instances additionally what the surface beneath the waste dump was unknown.

In a number of instances small waste dumps were present adjacent to larger waste dumps. Most of these were less than 10 m^3 in these instances the volume of these small waste dumps were visually estimated. In some instances 10s of shallow workings were found concentrated in an area with numerous small waste piles. On these occasions the volumes of these small waste dumps were visually estimated.

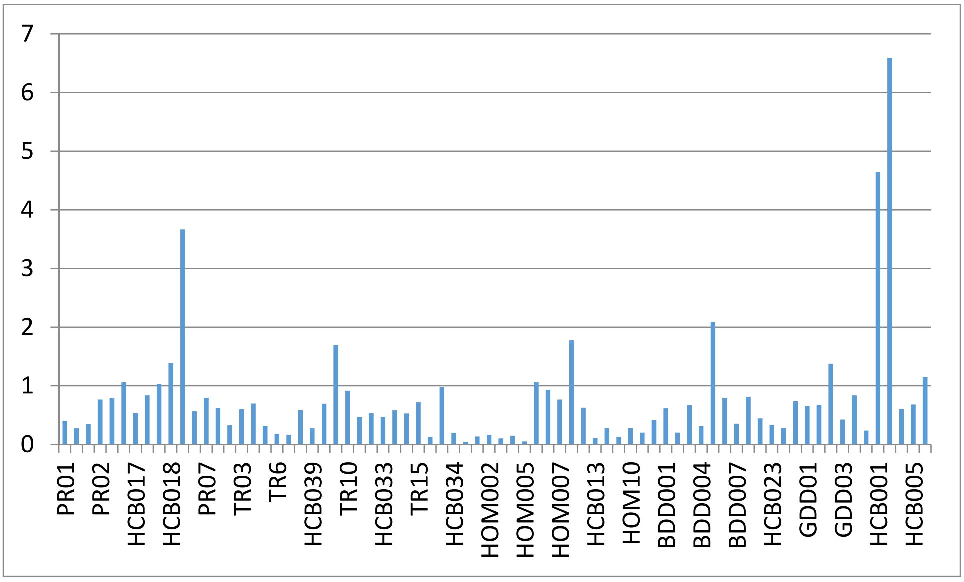
During the course of the survey it was discovered that measuring the area of the waste dump with the GPS was problematic. One problem was that in the case of small waste dumps the GPS could badly under estimate the area. ***It also became obvious that pacing the area of the dump to allow calculation as a hexahedron was much quicker than using the GPS to measure both the top and basal area of waste dumps. Furthermore calculating the volume of a waste dump could be quickly and easily accomplished in the field using the hexahedron method*** *whereas calculating the volume of the waste dump as a cone could not be done quickly and easily in the field.* ***Therefore it was decided, after measuring top and basal areas of seven waste dumps with the GPS, to discontinue this method to save time and allow for immediate calculation of the waste dump volume in the field. It was also thought that the hexahedron method allowed for sufficient accuracy for the purposes of this survey. In addition it is considered that the volumes estimated using the hexahedron method are conservative in nature and if anything underestimate the volume of the material surveyed within the Hatches Creek mining field.***

The Hexahedron volume calculation was done using the formula below:

$$v = l \times w \times h \quad \text{Where: } v = \text{volume in } m^3$$

l = length in m
w = width in m
h = height in m

HISTOGRAM PLOT OF WO3 PERCENT Assays: A top cut of 1.5% approx. 97% percentile



Metallurgical Evidence of Production Potential

The results highlight the potential to process the significant volumes of previously mined material available at surface, providing an immediate pathway to progress commercial development of the Hatches Creek Project.

The Company has received the results from 10 composite samples that were submitted to metallurgical process specialists Nagrom Pty Ltd for simple wet gravity test work. The 10kg samples were crushed and wet tabled with further concentration by magnetic separation and gravity separation.

The results were from six historical prospect areas – Pioneer, Black Diamond, Green Diamond, Treasure Group, Copper Show and Hit or Miss. Four of these areas have returned excellent grade and recovery results, with concentrate grades ranging from 20% to 47% WO_3 from Green Diamond, Treasure Group, and Hit or Miss with recoveries ranging from 65% to 75%.

The results highlight the potential to process the significant volumes of previously mined material available at surface, providing an immediate pathway to progress commercial development of the Hatches Creek Project.

Importantly some of these composites included alluvial material further justifying its existence in the resource estimate.

Composite	Sample No	Northing GDA	Easting GDA	Location	Sample Type	Weight (Kg)	%WO3	Estimated %WO3
A	HCB004	7685333	516953	COPPER SHOW	ALLUVIAL	17.79	0.24	**0.24**
						17.79		
B	HCB006	7685370	516993	COPPER SHOW	MULLOCK DUMP	23.20	1.15	**0.92**
	HCB005	7685343	516970			23.20	0.68	
						46.40		

C	HCB007	7685756	519481	HIT OR MISS	MULLOCK DUMP	13.56	1.06	**1.15**
	HCB009	7685736	519493			11.29	1.78	
	HCB010	7685718	519510			11.51	0.63	
						36.36		
D	HCB017	7692225	518763	PIONEER	MULLOCK DUMP	12.06	0.54	**0.78**
	HCB018	7692227	518735			14.38	1.39	
	HCB019	7692044	518647			14.01	0.09	
	HCB020	7692094	518512			14.25	1.03	
						54.71		
E	HCB015	7692253	518813	PIONEER	BATTERY SANDS	13.63	0.84	**0.57**
	HCB016	7692266	518782			15.18	0.79	
	HCB021	7692066	518522			13.87	0.28	
	HCB022	7692041	518570			13.93	0.35	
						56.61		
F	HCB023	7690574	519549	BLACK DIAMOND	MULLOCK DUMP	12.61	0.33	**0.59**
	HCB024	7690622	519526			14.57	0.81	
						27.18		
G	HCB025	7690482	519495	BLACK DIAMOND	ALLUVIAL	9.64	0.15	**0.32**
	HCB026	7690551	519694			11.97	0.45	
						21.61		
H	HCB031	7690343	519608	GREEN DIAMOND	MULLOCK DUMP	10.97	0.84	**1.11**
	HCB032	7690357	519598			10.98	1.38	
						21.95		
I	HCB033	7686892	519892	TREASURE	ALLUVIAL	9.01	0.47	**0.33**
	HCB034	7686925	519921			9.52	0.20	
						18.53		
J	HCB039	7687080	519930	TREASURE	MULLOCK DUMP	12.65	0.28	**0.96**
	HCB040	7687009	519885			11.90	1.69	
						24.55		

SUMMARY HISTORY

The following is an excerpt from BMR Bulletin No 6

THE GEOLOGY AND MINERAL RESOURCES OF THE HATCHES CREEK WOLFRAM FIELD,

NORTHERN TERRITORY
BY
G. R. RYAN- 1961

“The Hatches Creek Wolfram Field lies almost 500kms by road north-east of Alice Springs, Northern Territory, Australia.

Mining of wolfram began in 1913, but production was erratic and at times the field was closed completely owing to an unfavourable market for wolfram.

A big drop in the price of tungstic oxide caused all the mines on the field to close by late 1957.

Official records indicate that 2,840 tons of wolfram and scheelite concentrates valued at £1,294,110 were produced between 1913 and 30th June, 1958; but the recording of production before 1940 was unreliable and the actual production is thought to be about 3,000 tons. In addition to wolfram the field has produced 5.58 tons of bismuth valued at £4,400 and 68.759 tons of copper ore valued at £7,148.

Wolframite is the commonest source of tungsten, but scheelite is present in some mines. Minerals of bismuth, copper, molybdenum, lead, tin, and iron have been identified in association with the tungsten minerals. Minor amounts of gold are also present.

The mineral-bearing lodes lie in sedimentary rocks, in volcanic rocks, and in intrusive rocks; in places the lodes traverse several types of country rock.

The oldest rocks in the area are arenaceous sedimentary rocks of the Hatches Creek Group of (?) Upper Proterozoic age. These have been folded and faulted, and intruded by the Pedlar Gabbro. Granite, which is believed to be the source of the mineral deposits, intrudes feldspar porphyry four miles south of the field. Metasomatic and thermal alteration of the Hatches Creek Group adjacent to the Pedlar Gabbro is common.

The mineral-bearing fluids intruded zones of shearing formed during the earlier main phase of faulting, and accompanied further slight movement. The mineral deposits lie in groups of lodes; within each group one or two lodes are dominant. Each lode consists of a zone of shearing, along which one or more quartz reefs, lying side by side or en- echelon, have been emplaced.

The lodes carry between 1.0% and 3.0% of tungstic oxide. Most reefs are between 6 inches and 18 inches wide; a few are larger. The deposition of wolfram was effected by features on the reefs which caused an increase in the rate of loss of temperature and pressure in the mineral solutions.

Tungsten minerals have been won from a great number of lodes, which have been divided into sixteen groups.
The Pioneer Group, the Treasure Group, and the Hit or Miss Group have contributed about 40% of the total tungsten production from Hatches Creek, and the Pioneer Group has been the sole source of bismuth. Copper ore has been sold from the Copper Show Group and the Hit or Miss Group.

Descriptions of all the groups are included, but attention has been concentrated on the more important lodes. Activity on the mining field has always been closely related to the demand for tungsten. High prices stimulate widespread activity on a large number of poorer lodes, but only a few lodes can support operations at a moderate price. The *greater part of the ore near the surface has now been removed, and the time is approaching when the development of the tungsten deposits will require the investment of considerable capital, and the use of more efficient methods of mining and exploration.*

There are no measured or indicated reserves of ore; inferred ore reserves total 1,000 tons of 65% WO, concentrate. Protore is estimated at about 500 tons.

LOCATION AND ACCESS

The Hatches Creek Wolfram Field (the Field) lies 457 kilometers by road north east of Alice Springs (Figure 1). The Project is accessed from Alice Springs via the Stuart Highway for 323 kilometers; turn right 40 km north of Barrow Creek then proceed for 134 kilometers east along a well graded tourist/station road arriving at the Pioneer Mine workings at the northern end of the Field. This road is normally accessible , but may be closed if heavy rain floods the creeks. Hatches Creek may also be reached by a good earth road from Bonney Well on the north side of the Davenport Range, arriving via Kurundi and Kuruncelli stations.

The Hatches Creek Wolfram Field is situated near the north-eastern end of the Davenport Range in the Northern Territory. The Stuart Highway, a bituminized road linking the port of Darwin with Tennant Creek and Alice Springs, crosses the Range at the north-western end. Wolfram deposits are present at Wauchope, The Devil's Marbles, Mosquito Creek, Kurunelli, Epenarra, Elkedra, and Hatches Creek; only the Wauchope and Hatches Creek deposits have proved to be economically important. The Wauchope deposits have been investigated by Sullivan (1952). The Hatches Creek Wolfram Field is defined for the purpose of this report as a roughly triangular area bounded by Hatches Creek, Mia Mia Creek, and the Hit or Miss Gully (Pl. 2). The area extends roughly 12kms in a northerly direction and is about 5kms across at the base. Only two small tungsten bearing and some reefs reputed to carry gold lie outside this area.

The project area comprises 16 groups of old tungsten mines and is contained within Exploration Licences EL 22912 and EL 23462 (Arunta 100%).

MINING HISTORY

The Hatches Creek Tungsten field has been one of Australia’s most prolific tungsten producing areas and contains by far the highest grade ore in the western world averaging 2.5% W03 throughout the multiple vein systems.

The earliest recorded production was in 1915 during WW1 and the last recorded production was early 1958. The total recorded production in the intervening 43 years was 2,840 tonnes of 65% W03 concentrates (Table 1) although the actual production is estimated at approximately 3,000 tonnes due to unreliable production records prior to 1940.

The value today of the historical production is approximately US$65Million which is remarkable considering the Field was mined only to a maximum depth of 60 meters. All the tungsten bearing lodes are reported to be continuous at depth

JORC Code, 2012 Edition – Table 1 report

Section 1 Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.*	• Bulk Samples were taken as cut channels on Mullock dumps. Rock chips and grab samples were selected samples of visibly mineralized material., and weighed between 0.5kg and 1.7kg. All sample material is derived locally within 5m of sample location. • Bulk samples were between 5kg and 24kg in weight • All samples were individually labelled and documented
	• *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	• Mullock samples were taken perpendicular across general trend of the dump over distance of 1 to 3m.
	• *Aspects of the determination of mineralisation that are Material to the Public Report.* *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• The majority of samples were generated by the collection of 5 kg to 24kg of material on site and have been shipped to the Laboratory. Samples will be; • Crushed to P100 2mm Riffle Split each crushed sample: o ½ split Head Analysis o Reserve Remainder • All samples were be analysed for WO3, Sn, Fe2O3, MnO, SiO2, Al2O3, TiO2, CaO, MgO, As, P, S, Mo, Cu, Bi, Au, Ag and LOI1000.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• No Drilling was used to collect these samples

Criteria	JORC Code explanation	Commentary
Drill sample recovery	*Method of recording and assessing core and chip sample recoveries and results assessed.*	• No Drilling was used to collect these samples
	• *Measures taken to maximize sample recovery and ensure representative nature of the samples.*	• No Drilling was used to collect these samples
	• *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• No Drilling was used to collect these samples
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	• Geology was logged by geologist and located by using a hand held GPS • Descriptions exist for all samples in the database
	• *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.*	• Sample descriptions are has primarily been quantitative and contain some components of semi-quantitative analysis • Photographs of sample sites are available.
	• *The total length and percentage of the relevant intersections logged.*	• Estimated
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*	• No Drilling was used to collect these samples. • Whole rock or mullock samples were taken, these was no preparation of sample on site.
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	• Whole rock or mullock samples were taken
	• *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.*	• No duplicates were taken.
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• Bulk samples were a minimum of 10kg. These are appropriate for early stage assessment.
Quality of assay data and	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	• No Assays completed at this stage.

Criteria	JORC Code explanation	Commentary
laboratory tests		
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.*	• No geophysical tools were used to determine any element concentrations.
	• *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	• No Assays completed at this stage.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.*	• No field duplicates were submitted in this sample program • No Assays completed at this stage. • Sample information is recorded at the time in hard copy format
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*	• An electronic database containing collars, surveys, assays and geology will be complied into the company's database. • Data verification was undertaken by checking assays and collars against hard copy logs.
	• *Discuss any adjustment to assay data.*	• No adjustment has been required
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	• Sample locations have been surveyed by handheld GPS only.
	• *Specification of the grid system used.*	• The GPS locations were recorded MGA (GDA94, Zone 53) coordinates.
	• *Quality and adequacy of topographic control.*	• No topographic control
Data spacing and	• *Data spacing for reporting of Exploration Results.*	• Grab or bulk samples representivity cannot be assessed as they are localized samples.

Criteria	JORC Code explanation	Commentary
distribution		
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	• Sample space is not sufficient, material sampled is local in nature, and not continuous with regard to geology.
	• *Whether sample compositing has been applied.*	• No compositing at this stage.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	• The sample orientations are deemed appropriate.
	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• No orientation-based sampling bias has been identified.
Sample security	• *The measures taken to ensure sample security.*	• Chain of custody for samples were managed by Arunta personnel. Samples were delivered to Nagrom laboratory by freight company.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• No Audits or reviews have been completed

1. Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites*	• Exploration Licences 22912 and 23462 are 100% are held by Davenport Resources Limited a 100% owned subsidiary of Arunta Resources Limited.

Criteria	JORC Code explanation	Commentary
	• *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• All statutory approvals have been acquired to conduct exploration. • No known impediments.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• Thor Mining PLC, were the last company to explore the area in 2008.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Hatches Creek tenements are underlain by Palaeoproterozoic sequence of weakly metamorphosed clastic sedimentary and felsic volcanic rocks. The sequence is intruded by igneous sills. Sandstone is the dominant sedimentary lithology. The sequence has been subjected to folding and faulting and has been cut by numerous narrow quartz reefs which follow lines of shearing. The quartz reefs are mineralised, the main mineral of economic interest being wolframite, although bismuth, gold and copper mineralisation is also present within them. The average tungsten grade of the mined reefs was between 1% and 5% WO3. • The mineralised reefs are present in groups. The average reef width is 30cm, with the maximum width being 1.5m. The maximum strike length of any one reef is around 170m however en echelon lines of reefs are up to 1.5km in length. The reefs strike in two main directions, just east of north, parallel to the main fault direction, and east-northeast. The north-striking reefs dip at moderate to steep angles either to the west or the east; those striking easterly dip at moderate to steep angles to the south. The majority of the reefs are within volcanic or intrusive rocks, rather than in the sandstone units. The more mafic host rocks (gabbro, diorite) appear to have been important host rocks for some of the significant mineralisation in the area.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:*	• No Drilling conducted

Criteria	JORC Code explanation	Commentary
	o *easing and northing of the drill hole collar* o *elevation or RL (Reduced Level – elevation above sea level in meters) of the drill hole collar* o *dip and azimuth of the hole* o *down hole length and interception depth* o *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• No weighting techniques have been used all results have been reported • Where results have been discussed, a simple arithmetic average has been used.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• Results are from bulk samples or rock chips, no geometry or width are able to be reported.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• See Figures 1
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• All results have been reported
Other substantive	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical*	• Description of sample type and size has been reported, bulk samples were 5-24kg. Rock chips were 0.5 to 1.7kg

Criteria	JORC Code explanation	Commentary
exploration data	*survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• Further metallurgical testing of samples are required.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Explanation	Commentary
Database Integrity	• *Measures taken to ensure data has not been corrupted by, for example, transcription or keying errors, between its initial connection and its use for Mineral Resource estimation purposes* • *Data validation procedures used*	• *Data checked between lab results sheets and from excel spreadsheets provided from client and cross checked. Data signed off under previous Reporting of Exploration Results statement- attached.* • *Cross checking between lab sheets and Client excel sheets*
Site Visits	• *Comment on any site visit undertaken by the Competent Person and the outcome of those visits* • *If no site visits have been undertaken indicate why this is the case*	• *Site visit not conducted as assessment is based on recent sampled stockpiles of historic dumps evident on satellite photography, well documented and geologically described with volumes measured by geologist. A site visit was not considered necessary in this case as excellent historic records exist for resource estimate based on measured / mined stockpiles from historic production not in- ground mineralisation..*
Geological Interpretation	• *Confidence in or conversely the uncertainty of the geological interpretation of the mineral deposit* • *Nature of the data used and any of the assumptions made* • *The effect, if any, of any alternate interpretations om Mineral resource estimation* • *The use of geology in guiding and controlling Mineral resource estimation* • *The factors affecting continuity both of grade and geology*	• *A strong geological knowledge is documented from past BMR mining history and geological publication by Ryan 1961of each prospect within the mineral field. Detailed description of mineralisation, structural and alteration control on tungsten mineralisation is described in each case and a robust regional geological interpretation of this mineral field has resulted.* • *Data used included locations and volume measurements of dumps by area/ height measurement, supported by SGs*

		measurements on host rocks, and assay results from two separate bulk sample phases on 10 to 20kg samples. Assumptions made of 10% to 15% SG reduction on conversion to compacted dump density after over 50 years stockpile compaction. • *Being stockpiles the mineral resource estimation is factually based, not subject to significant interpretation .Host rocks for each prospect are recorded and visually confirmed from photos of each location and dump material type as well as excellent geological volume by Ryan in 1961 BMR Report.* • *The geology has been used in SG estimation for tonnage estimates from dump volumes and confirmatory of underground source. Alteration zoning and structural intersections control mineralisation more than lithology. Rock chip samples show veined nature and not used directly in estimates but show mineralisation grade potential before mine dilution considered. Bulk samples reflect mine ability* • *Lode structure orientations and extent have been mapped/ recorded / reflected in stockpile locations, geology and grade. Historic mining involved hand selection of very high grade. Alluvial drainage patterns define lower grade trends, some economic*
Dimensions	• *The extent and Variability of the Mineral resource expressed as length (along strike or otherwise) plan width and depth below surface to the upper and lower limits of the mineral resource*	• *Separate resource estimations were conducted on each project area. All resource estimations based on surface historic mined dumps and minor alluvial component. Volumes of dumps were estimated from area and height using GPS .Sampling of dumps was carried out by shovel spearing. Grade weighting was applied based on dump volumes and then tonnages from SG data . Unsampled dumps within each prospect were applied the average grade of sampled dumps. Alluvial estimates were based on sampling and drainage trends from Aerial photography*
Estimation and Modelling techniques	• *The nature and appropriateness of the estimation technique applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If computer assisted*	• *Grade weighting based on sample location and dump size/ tonnage for each prospect and material type/ host rock was a key assumption. Upper and lower cut off grades were applied described below. The upper cut at the 97^{th} percentile and lower cut based on current production economics. Each*

	estimation is used include a description of software and parameters used. • The availability of check estimates, previous estimates and/ or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • Assumptions made about recovery of by products • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation) • In the case of block model interpolation, the block size in relation to the average spacing and search employed. • Any assumptions behind modelling of selected mining units	resource reflects a separate prospect – considered a separate geological domain Where dumps were not sampled but contained within a known mineralised domain average grade of that prospect domain was applied. No computer assisted or interpolation applied. • Each sample is representative of approximately 2000 tonnes of ore grade stockpile material, which is similar to current RC grade control drilling definition on a 12.5m x 12.5m spacing.. • No check estimates except independent sampling programmes but production records and detailed lode mapping available to support grades and tonnages achieved • No assumptions made on recovery of by-products. Historic mining recovered tungsten . Sulphur values up to 0.5% noted from 41 HCB samples. Otherwise no deleterious elements evident. Any sulphur issue will be addressed in an NOI and if necessary contained in any future mining. • Block modelling interpolation not applicable in this case. • SMUs Not applicable in this case
Estimation and Modelling Techniques Continued	• Any assumptions about correlations between variables • Description of how the geological interpretation was used to control the resource estimates • Discussion of basis for using or not using grade cutting or capping • The process of validation, the checking process used, the comparison of model data to drill hole data and use of reconciliation data if available	• Mineral zonation and a varying association of mineralisation is evident between prospects. Previous mining has been successful for tungsten throughout all prospects and Nagrom testing also—detailed correlations not assessed as part of this estimate • All Dumps appear to be directly aligned on lode structural trends. Any dumps not sampled were assigned average grade of dumps from that prospect / domain. Alluvials were included in resource based on alluvial drainage patterns. • Used modest grade upper cut cutting to 1.5%WO3 (approximating the 97% intersection with sample on log probability distribution . A top cut was applied because of vein irregularity with likely nugget effect – evident in rock chip samples. Rock chip results not used in resource estimation because not mineable.. Rock chip results however show the grade character of the resource • Validation by comparing stockpile record against production records for each prospect. Validation by sampling being conducted in two separate programmes and sample weights considered. (except by large parcel throughput)

		• *Sampling results grade weighted against the size stockpile represented*
Moisture	• *Whether the tonnages are estimates on a dry basis or with natural moisture and the method of determination of the moisture content*	• *Moisture contents not determined but estimate within mineral lattice at less than 5%. Dumps been exposed for 60 years in a semi desert environment..*
Cut Off Parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied*	• *A lower cut-off grade of 0.2%WO3 was applied as economic cut and both an uncut and an upper cut of 1.5%WO3 applied as best case for mine ability at bulk sample scale. The lower cut-off grade reflects likely current economics of production . The upper cut approximates 97th percentile on log probability.*
Mining factors or assumptions	• *Assumptions made about possible mining methods, minimum mining dimensions and internal (or if applicable, external) mining dilution. In considering potential mining methods for eventual economic extraction, the assumptions made regarding mining methods may not always be rigorous. Where this is the case this should be reported with an explanation of the basis of the mining assumptions made*	• *Not applicable to this estimate as resource already mined and on surface. Future mining will be able to utilise past information.*
Metallurgical Factors or assumptions	• *The basis for assumptions about metallurgical amenability. In considering potential metallurgical methods assumptions made when reporting Mineral resources may not always be rigorous. In this case this should be reported with the basis of metallurgical assumptions made*	• *Extensive compositing and metallurgical testwork from Nagrom and historic mining records and treatment have been favourable to recovery. There appear to be no metallurgical issues of concern as gravity and magnetic separation on sample composites have been successful..*
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. In considering potential environmental impacts of the mining and processing option the determination of potential environmental impacts, especially for greenfields projects, may not be well advanced , the status of early consideration of these environmental impacts should be reported. Where these have not been considered this should be reported with an explanation of environmental assumptions made.*	• *Existing Mining areas Isolated with minor topographic gradient that could be exploited for containment of tailings into a dam if necessary.. A current Environmental historical liability exists which Arunta plans to address should it proceed to production.* • *May need to investigate sustainable water supply and low sulphur assays recorded should ensure any Acid mine drainage could be contained . Arunta will address this and assumption will be that if any environmental concern from sulphur then planned to be contained..*
Bulk density	• *Whether assumed or determined. If assumed the basis for this. If determined, the method used, whether wet or dry, frequency of the measurements, the nature, size and representativeness of the samples*	• *Detailed assessment carried out by gas pychnometer on 30 samples for various prospects and material types,* • *Assumption on voids and porosity content applied later. Battery sands fine grained a 10% sg reduction and Mullock*

	• *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vughs, porosity etc) moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density used in the evaluation process of the different materials*	*Dumps up to 15% reduction for density. The material on stockpile has been compacted for 50 years in a semi desert environment. Both low inherent moisture and voids content assumed.*
Classification	• *The basis for the Classification of the Mineral resources into varying confidence categories* • *Where appropriate account has been taken of all relevant factors (relative confidence in tonnage / grade estimations , reliability of input data , confidence in continuity of geology and metal values, quality, quantity and distribution of the data.* • *Whether the result appropriately reflects the Competent Persons view of the deposit.*	• *Mined ore stockpiles in modern mining would be classified Measured Resource. However in this case - because of historic mining and therefore uncertainty on exact underground location and grade control practices and since e not every stockpile was sampled in some prospects it is therefore classified as Inferred throughout.* • *Based on availability of relatively reliable recent stockpile tonnage measurement and sampling/ assay data , continuity of stockpiles along lodes from excellent production and geological records as well as satellite photography and the number and size of samples collected . Given the amount of high grading conducted in historic mining Areas of little confidence have been excluded subject to confirmatory sampling*
Audits or reviews	• *The results of any audits or reviews of mineral resource estimates*	• *None conducted previously on these stockpiles.*
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the estimate using an approach or procedure deemed appropriate to the Competent Person. For example application of statistical or geostatistical procedures to quantify the accuracy of the resource stated confidence limits or if not deemed appropriate a qualitative discussion of factors that could affect the relative accuracy and confidence of the estimate.* • *Statement should specify whether it relates to global or local estimates and if local, state the relative tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and procedures used.* • *The statements of relative accuracy and confidence of the estimate should be compared with production data where available*	• *All estimates have been conducted on prospects/domains as local estimates.* • *Different prospect areas have some difference in confidence based on degree of sampling, evidence of geological control. Prospects called Pioneer, Treasure, Hit or Miss, Green Diamond, Black Diamond and Copper Show have highest level of confidence Inferred level 1 . The other prospects are Inferred Level 2 with measured stockpiles but not sampled – assumed a lower grade than above. The alluvials despite only applying a 30% estimate of area ads being above cut off relatively lower level of confidence, Level 3 but based on production evidence this is likely conservative.* • *Assumption any unsampled stockpiles along mined lodes are assigned average resource grade of that prospect* • *Production data supports the resource estimate* • *Statement relates to local estimates*

COMPETENT PERSONS STATEMENT

The information in this report that relates to the mineral resource estimation is based on work completed by Mr. Anthony Ryall who is a member of the Australian Institute of Mining and Metallurgy. Mr. Ryall is an Independent Consultant with sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'.

<u>ANTHONY RYALL</u>

Appendix 2
Hatches Creek Metallurgical Testwork Report

Extract Mineral Processing

Metallurgical Test Work Report

Hatches Creek

To:	**Craig Ferrier**
Company:	**GWR Ltd**
From:	**Leigh Wardell-Johnson**
Date:	**23 July 2015**

Executive Summary

An investigative metallurgical test work program on 2 bulk samples of Hatches Creek Pioneer and Treasure material has been completed at Nagrom. The aim of the program was determine the amenability of the samples to upgrade WO_3 using gravity, magnetic characterisation and flotation test work.

The test work has upgraded WO_3 from a calculated combined Pioneer/Treasure head grade of 0.77% WO_3 to 42.7% WO_3 concentrate with an overall circuit recovery of 66%. A super-concentrate grading 66.3% WO_3 and 27% circuit recovery resulted from flotation test work.

The test work has also produced significant grades of Cu, Mo, and Au in the pre-float sulphide concentrate, having potential to be recovered as saleable by-products.

Bi levels in the concentrate (1.5% or 10500 ppm) are well above the industry standard for APT and FeW production. Upper limits on Bi on WO_3 concentrates for APT production is 0.5%. The general specification requirement is 0.15% max with higher levels incurring price penalties in the order of $2/mtu for 0.15- 0.3% Bi, $4/mtu for 0.3-0.5% Bi and $6/mtu at 0.5%.

Due to money and sample mass constraints, the number of Bi -50% concentrate and middlings flotation tests were inadequate to fully optimise the flotation conditions. No flotation test work was conducted on the tailings despite it containing 25% of the overall circuit WO_3. This means that the overall result of the test work program is indicative only, with upside potential. The test work results have however provided enough information to have confidence that there is sufficient WO_3 grade and recovery to progress to a scoping level study at Hatches Creek

Depending upon the outcome of the scoping study and nature of the project moving forward, it is recommended that any metallurgical test work required should include, but not limited to;

7. *Establishing a full set of physical characteristics including hardness, crushability and abrasion indices that will feed into a process design.*

8. *Increasing the recovery of WO_3 from the middlings and tailings by;*

- *Targeting increased depression of Si in the middlings to improve the WO_3 concentrate grade.*

- *Optimisation of WO_3 collector in the middlings to maximise WO_3 recovery.*

- *Developing a test work plan aimed at increasing recovery of WO_3 from the tailings.*

9. *Optimisation of the roughing and cleaning stages for the -50% concentrate.*

10. *Cleaning of the final concentrate to a standard specification grade for a WO_3 concentrate (~65%).*

11. *Reduction of Bi in the intermediate and/or final concentrate to a saleable specification.*

12. *Recovery of potentially valuable by-products including Au, Cu and Mo.*

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1.0 BACKGROUND

In January 2015 GWR and Arunta Resources Ltd entered into a Farm-in Joint Venture Agreement under which GWR could earn a 50% interest in the Hatches Creek Tungsten Project by spending $1.5m on development and exploration. GWR funds were to be initially directed to the completion of metallurgical test work, preparation of a scoping study and receipt of all relevant approvals to conduct mineral processing activities.

In November 2014 Arunta had commenced a program of metallurgical test work on bulk samples taken from stockpiles of mineralised mine waste, tailings and alluvial material. The material located at the 11 largest historic mines within the Hatches Creek Tungsten Field had formed the basis for a maiden inferred resource of 225,066 tonnes grading 0.58% WO_3 *(announced by Arunta in September 2014).*

Extract Mineral Processing (EMP) was commissioned to manage the test work program and in consultation with Nagrom evaluate the results and decide upon the direction of the test work program.

This report will detail the results of the metallurgical test work program and provide recommendations for any future test work or studies and implications on processing options.

2.0 THE SAMPLES

In October 2014 a bulk sampling program was completed by Arunta, which involved the collection of 3 x 500kg samples from previously stockpiled material at Pioneer, Treasure and Diamond using a mini excavator. The entire profile of the stockpile was dug to ensure representivity of the samples. The samples were placed into 1 tonne bulker bags and dispatched to Nagrom Metallurgical Laboratories in Kelmscott WA.

The aim of Nagrom's test work was to optimise the processing parameters to achieve the best possible recoveries of Wolframite ($FeMnWO_4$*) and Scheelite (*$CaWO_4$*) and to determine the specifications of the final concentrates.*

To reduce test work costs, it was decided to exclude the Green Diamond sample from the initial test work program as it represented a smaller proportion of the stockpile relative to Pioneer and Treasure.

Table 1 below shows the head grade of the major constituents that make up the samples.

Sample	**WO_3 %**	**Fe %**	**CaO %**	**SiO_2 %**	**S %**	**Mo %**	**Cu %**	**Bi ppm**	**Au ppm**
Pioneer	*0.74*	*11.94*	*3.54*	*63.49*	*0.45*	*0.010*	*0.15*	*1061*	*0.28*
Treasure	*0.62*	*4.79*	*0.51*	*82.82*	*0.025*	*0.009*	*0.02*	*111*	*0.16*

Table 1: Head Grade

3.0 TEST WORK PROGRAM

Figure 1 below is a summary the original test work program as per Q01752, the scope of work designed by Nagrom in consultation with Arunta personnel. A copy of Q01752 can be found in the attached appendices.

A detailed block flow diagram of the test program was not requested at the time.

As with all test work, the scope of the program was subject to change as results were received. This meant that additional tests were required, namely scavenging test work of spiral tailings and flotation of gravity concentrate and middlings. This test work is described in Q02034A and Q02141A respectively, found in the attached appendices.

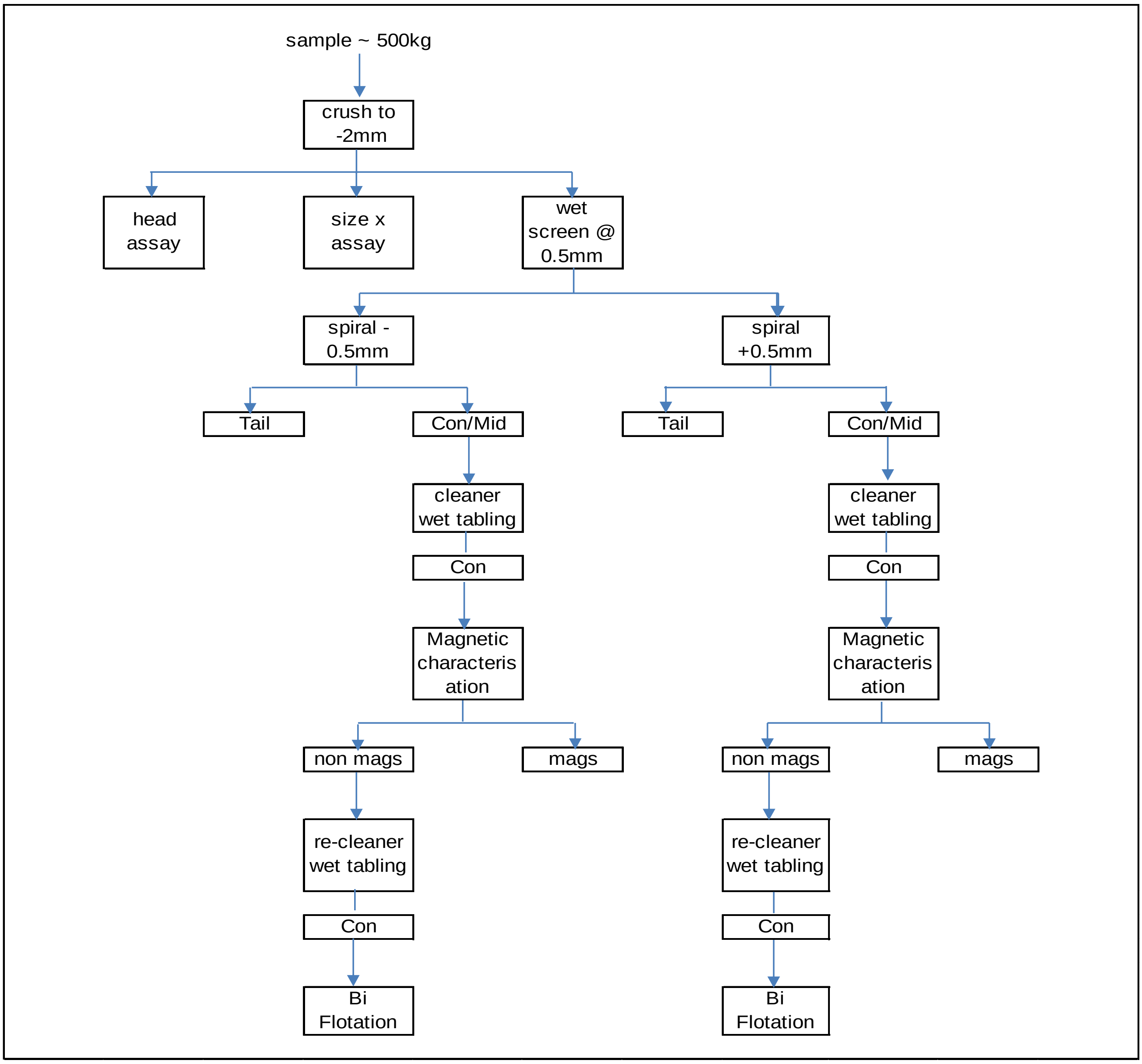


Figure 1: Test Work Program

3.1 TEST WORK RESULTS

The full suite of metallurgical test work results can be found in the attached Nagrom test work report in the appendices.

3.1.1 Wet Screening

The Pioneer and Treasure samples were individually crushed to p100 2mm and wet screened over a 0.5mm screen to produce -2.0 +0.5mm and -0.5mm subsamples prior to spiral test work. This was to ensure optimum conditions for coarse spiralling (-2.0 +0.5mm) and fine spiralling (-0.5mm) and subsequent recovery of WO3.

Table 2 and 3 shows the mass split and assays of the -2.0 +0.5mm and -0.5mm fractions for both Pioneer and Treasure material.

Pioneer

Size (mm)	Mass kg	Mass Split %	WO_3 %	Distn WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
+0.5	336	63	0.60	50	11.0	65.6	3.6	0.3	0.004	0.10	493	0.2
-0.5	198	37	0.99	50	13.1	61.6	3.4	0.5	0.017	0.21	1499	0.5
calc head	534	100	0.74	100	11.8	64.1	3.5	0.4	0.009	0.14	866	0.3

Treasure

Size (mm)	Mass kg	Mass Split %	WO_3 %	Distn WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
+0.5	309	67	0.57	59	3.8	86.4	0.2	0.01	0.007	0.02	64	0.01
-0.5	153	33	0.79	41	6.5	78.1	0.8	0.04	0.011	0.03	134	0.03
calc head	462	100	0.64	100	4.6	83.7	0.4	0.02	0.008	0.02	87	0.01

Tables 2 and 3: Mass spit and assays

In both the Pioneer and Treasure samples, there is a higher mass split of material to the +0.5mm fraction and a % WO_3 upgrade in the -0.5mm fraction.

Higher grades of Fe_2O_3, S, Mo, Cu, Bi and Au were also observed in the -0.5mm fraction.

Distribution of WO_3 between the coarse and fine fractions for the Pioneer sample was equal, whereas for the Treasure sample 59% of the WO_3 reported to the +0.5mm fraction.

3.1.2 Spiralling

Pioneer and Treasure coarse (-2mm +0.5mm) and fine (-0.5mm) material were fed through a laboratory spiral rig to recover concentrate, middlings and tailings streams. The coarse and fine spiral set up were chosen to ensure maximum recovery of WO_3.

Table 4 to 7 below shows the result of the coarse and fine spiral test work for Pioneer and Treasure.

Pioneer -2.0mm + 0.5mm coarse spiral

Size (mm)	Mass kg	Mass Split %	WO_3 %	Distn WO3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Conc	28	8	4.3	57	14.0	57.4	4.7	1.11	0.045	0.40	2238	0.87
Mids	87	26	0.5	22	11.7	64.8	3.9	0.34	0.005	0.11	423	0.28
Tail	215	65	0.2	21	10.5	66.8	3.3	0.20	0.003	0.07	420	0.20
calc head	330	100	0.7	100	11.1	65.5	3.5	0.31	0.007	0.11	575	0.28

Pioneer -0.5mm fine spiral

Size (mm)	Mass kg	Mass Split %	WO_3 %	Distn WO3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Conc	18	9	8.0	70	19.3	45.3	6.1	2.70	0.072	0.97	6018	1.93
Mids	18	9	0.3	2	14.0	60.7	4.6	0.40	0.015	0.17	839	0.58
Tail	154	81	0.4	28	12.4	63.8	2.9	0.29	0.011	0.15	1170	0.33
calc head	190	100	1.1	100	13.2	61.8	3.4	0.53	0.017	0.23	1598	0.50

Treasure -2.0mm + 0.5mm coarse spiral

Size (mm)	Mass kg	Mass Split %	WO_3 %	Distn WO3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Conc	15	5	3.3	29	6.2	79.9	0.4	0.03	0.027	0.04	123	0.01
Mids	44	14	1.4	35	4.4	84.8	0.2	0.01	0.013	0.02	67	<0.005
Tail	245	81	0.2	36	3.3	87.1	0.2	0.01	0.004	0.01	56	0.01
calc head	304	100	0.6	100	3.6	86.5	0.2	0.01	0.006	0.02	61	0.004

Treasure -0.5mm fine spiral

Size (mm)	Mass kg	Mass Split %	WO_3 %	Distn WO3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Conc	8	5	7.7	58	14.9	61.2	1.3	0.25	0.084	0.12	584	0.16
Mids	6	4	0.5	3	6.3	79.6	0.6	0.04	0.010	0.03	132	0.02
Tail	134	91	0.3	39	5.5	79.8	0.8	0.04	0.007	0.03	127	0.09
calc head	148	100	0.7	100	6.1	78.8	0.8	0.05	0.011	0.03	152	0.09

Tables 4 to 7: Coarse and fine spiral test work

In all spiral tests, there has been an upgrade in the %WO_3 to the concentrate with recoveries ranging from 29% for the Treasure coarse spiral to 70% for the Pioneer fine spiral, suggesting that that the Pioneer sample appears to have a *coarser liberation size. Fe_2O_3, Au, Mo and Cu all upgraded with S in the concentrate fraction suggesting that these constituents maybe associated with sulphides and potentially recovered via a simple sulphide flotation.*

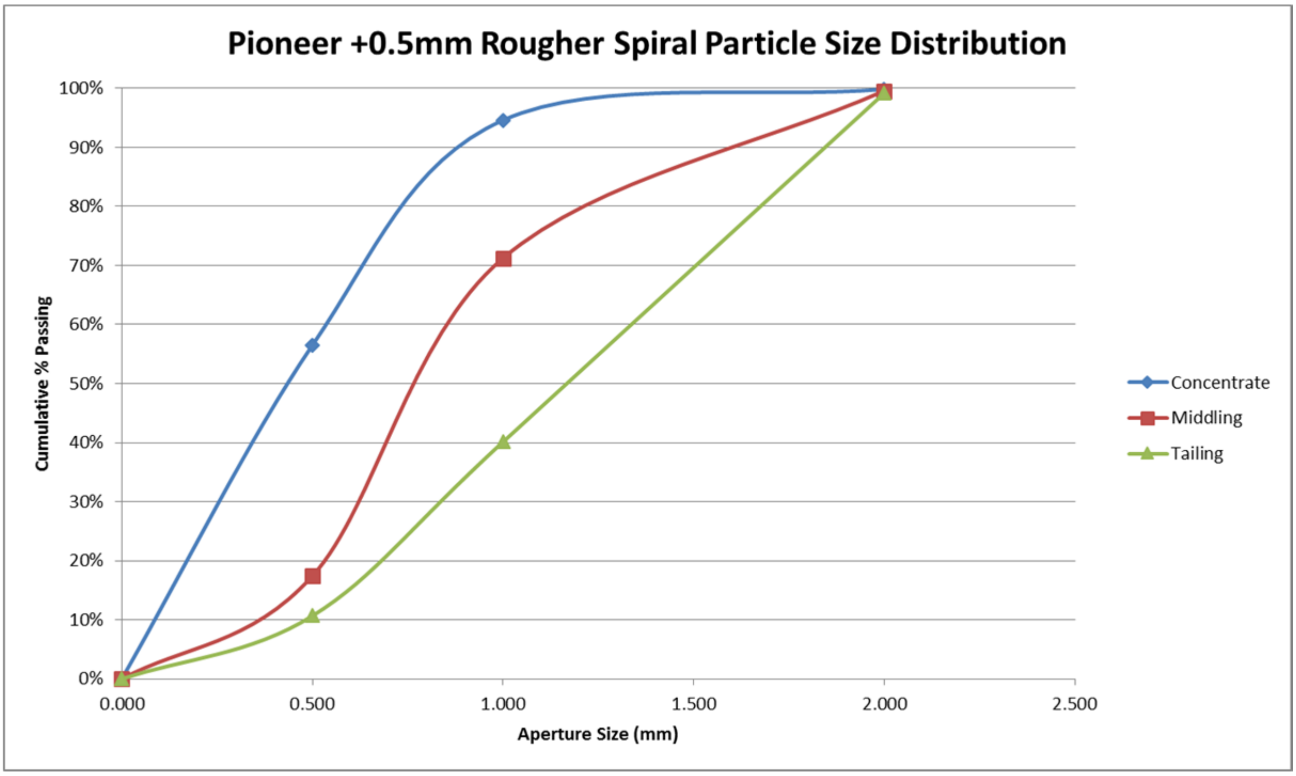
Elevated Bi, particularly in the Pioneer samples is a deleterious contaminant in a WO_3 concentrate and was subject to flotation test work to reduce its grade down to a marketable level (see section 3.1.4.1 – Bismuth Flotation).

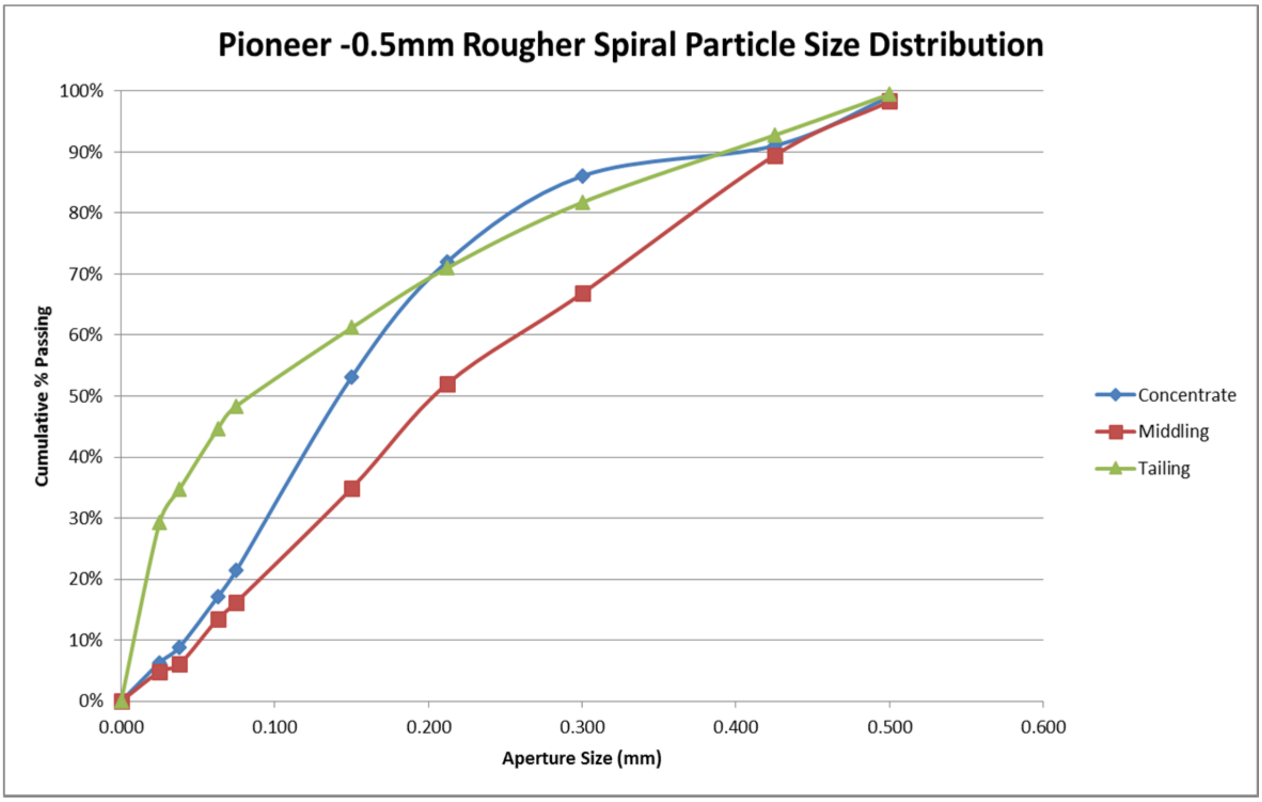
Concentrate and middlings streams from each spiral were recovered and combined according to their mass deportment and sent for cleaning on a Wilfrey wet table. The spiral tailings contained a higher than expected

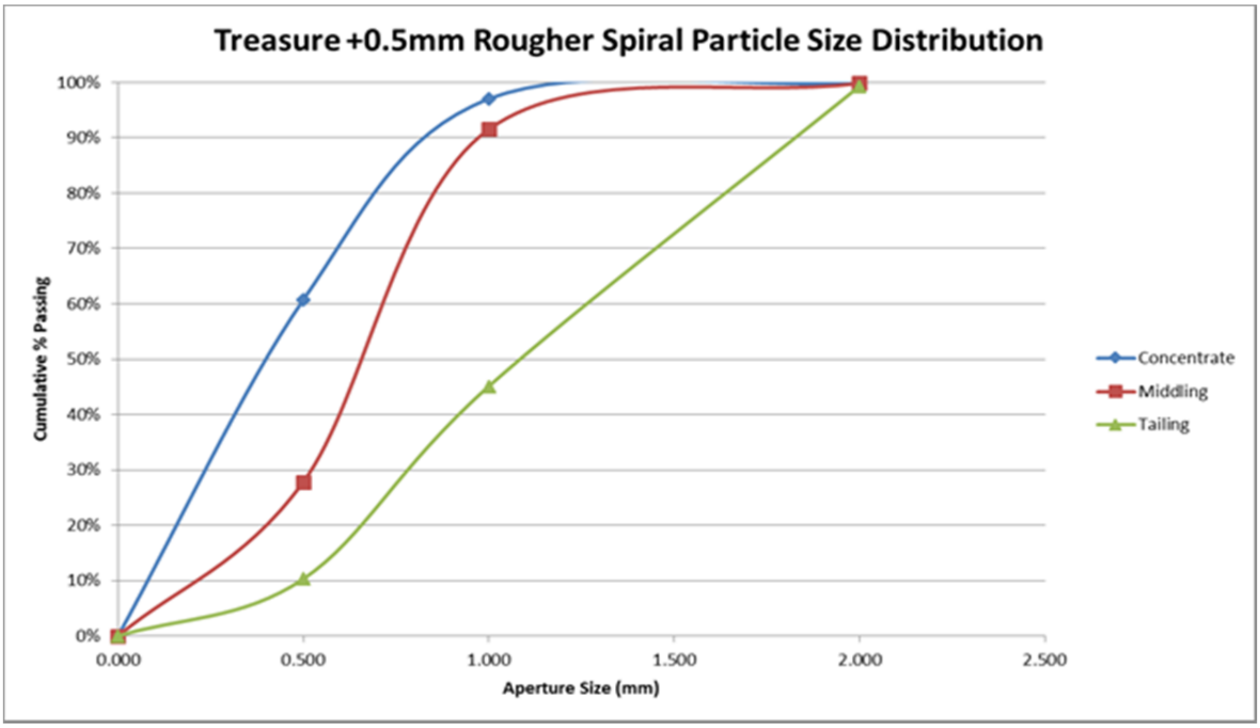
proportion of the total circuit WO_3, ranging from 21% in the Pioneer coarse spiral sample to 39% in the Treasure fine spiral sample. For this reason, it was decided to increase the scope of the original program to include test work to recover WO_3 from the tailings.

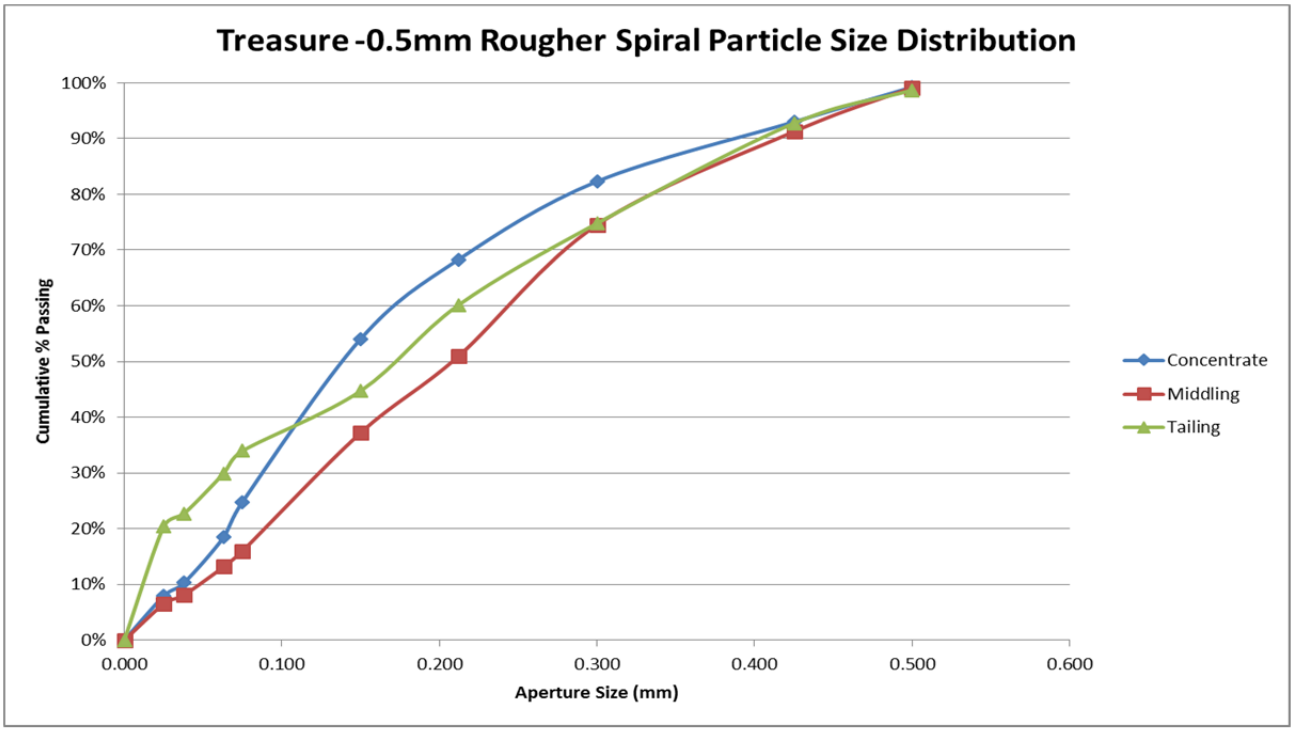
The concentrate, middlings and tailings streams were wet screened to produce particle size distributions (PSD's) to determine if there they were related in anyway.

Figures 2 to 5 show the PSD's for the concentrate, middlings and tailings for the Pioneer and Treasure coarse/fine spirals.


Pioneer +0.5mm Rougher Spiral Particle Size Distribution
Cumulative % Passing
100%
90%
80%
70%
60%
50%
40%
30%
20%
10%
0%
0.000
0.500
1.000
1.500
2.000
2.500
Aperture Size (mm)
Concentrate
Middling
Tailing


Pioneer -0.5mm Rougher Spiral Particle Size Distribution
Cumulative % Passing
Aperture Size (mm)
0%
10%
20%
30%
40%
50%
60%
70%
80%
90%
100%
0.000
0.100
0.200
0.300
0.400
0.500
0.600
Concentrate
Middling
Tailing


Treasure +0.5mm Rougher Spiral Particle Size Distribution
Cumulative % Passing
Aperture Size (mm)
0%
10%
20%
30%
40%
50%
60%
70%
80%
90%
100%
0.000
0.500
1.000
1.500
2.000
2.500
Concentrate
Middling
Tailing



Figures 2 to 5: PSD's for the concentrate, middlings and tailings

As the PSD's show, the concentrate portion in both the Pioneer and Treasure coarse spirals has the finest size distribution compared to their respective middlings and tailings. The fine spirals was a different story with the tailings producing a finer PSD below 200 microns for Pioneer and 100 micron for Treasure. This is more than likely associated with very fine clay particles in the samples being carried by the water medium.

3.1.3 Tabling – Spiral Concentrate/Middling Composite and Spiral Tailings

The concentrate and middlings streams from each run of the Pioneer and Treasure spiral test work were combined according to their mass deportment and cleaned via tabling. Super-concentrate (+50% WO_3), concentrate (-50% WO_3), middlings and tailings were recovered from each table run.

The tailings samples from Pioneer and Treasure spiral test work were subject to rougher tabling followed by cleaner tabling. As with the concentrate/middling composite, a super-concentrate, concentrate, middlings and tailings sample were recovered from each table run.

All coarse size fractions were dry tabled and fine size fractions wet tabled at 20% solids w/w.

Although the coarse and fine components of both the Pioneer and Treasure material were tabled separately, the coarse and fine results were combined leaving an overall tabling result for Pioneer and Treasure.

Tables 8 and 9 show the combined coarse and fine results of the Wilfrey tabling of the Pioneer and Treasure spiral concentrate/middling composite and spiral tailings.

Pioneer - Tabling	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Super-concentrate (+50%)	57.4	0.8	18.2	2.9	9.5	4.74	0.05	0.76	25210	0.00
Concentrate (-50%)	16.9	66.8	21.9	31.0	7.6	4.17	0.08	1.38	9471	3.40
Middlings	2.0	14.7	17.5	52.1	5.8	1.24	0.02	0.40	1355	0.29
Tailings	0.1	17.6	10.9	67.3	3.1	0.19	0.01	0.09	516	0.21
calc head	**0.8**	**100.0**	**11.6**	**65.3**	**3.4**	**0.37**	**0.01**	**0.15**	**838**	**0.31**

Treasure - Tabling	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Super-concentrate (+50%)	52.0	11.0	28.0	3.0	2.8	1.00	0.34	0.34	3705	1.80
Concentrate (-50%)	9.8	61.2	15.5	60.2	0.9	0.12	0.09	0.11	402	0.04
Middlings	2.6	2.9	17.9	60.1	1.6	0.14	0.05	0.11	458	0.23
Tailings	0.2	24.9	3.8	86.2	0.3	0.01	0.00	0.01	67	0.02
calc head	**0.6**	**100.0**	**4.3**	**85.0**	**0.4**	**0.02**	**0.01**	**0.02**	**86**	**0.02**

Tables 8 and 9: Coarse and fine results

The recovery of WO_3 to the combined super-concentrate and concentrate ranged from 68% for Pioneer and 62% for Treasure. Using a weighted average, the %WO_3 grade of the combined super-concentrate and concentrate for Pioneer was 17.1% and 11.2% for Treasure.

In general, Pioneer has higher grades of Cu, Bi and Au compared to Treasure in the feed and final concentrate. There may be greater opportunity in the Pioneer material to extract valuable by products in parallel to WO_3 extraction.

This opportunity should be explored in greater detail in subsequent test work phases.

The higher head grades of CaO in Pioneer suggests that this deposit contains a higher proportion of Scheelite relative to Treasure.

3.1.4 Magnetic Characterisation

The super-concentrate and concentrate from the fine and coarse tabling, including the spiral tailings, were recovered and separately processed through a series of magnetic separation tests of varying gauss. The reason for magnetic separation was twofold, firstly to clean the concentrates of any magnetic iron constituents and secondly to separate Scheelite from Wolframite as they cannot be sold as a combined concentrate. Scheelite is non-magnetic and Wolframite paramagnetic, meaning that Wolframite can be magnetically concentrated at high gauss using wet and/or dry high intensity magnetic separation depending upon its chemical structure and crystallography.

The full set of magnetic characterisation results is included in the full circuit summaries for Pioneer and Treasure, and can be found in the attached appendices

Tables 10 to 29 below shows the results of the magnetic characterisation tests on the gravity concentrates.

Coarse Table Superconcentrate

Pioneer - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	3.9	1.1	51.9	21.1	4.6	3.38	0.01	0.15	731	0.10
4000G Magnetics	40.5	18.8	23.6	17.2	4.7	1.69	0.01	0.26	1142	2.24
12980G Magnetics	8.7	35.7	19.5	41.6	6.0	1.50	0.01	0.74	933	0.62
12980G Non Magnetics	29.0	44.4	6.2	41.4	8.2	4.62	0.38	1.29	21088	2.39
calc head	**15.6**	**100.0**	**18.1**	**38.8**	**6.4**	**2.34**	**0.10**	**0.81**	**5761**	**1.13**

Coarse Table Concentrate

Pioneer - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	0.7	1.3	43.3	30.2	5.2	1.95	0.01	0.11	386	0.53
4000G Magnetics	5.9	12.5	25.5	38.4	5.7	1.54	0.01	0.17	587	0.15
12980G Magnetics	1.1	33.9	17.6	49.2	5.9	0.80	0.01	0.33	524	0.11
12980G Non Magnetics	5.1	52.4	3.5	80.9	2.3	1.22	0.11	0.23	5625	1.95
calc head	**2.2**	**100.0**	**15.9**	**55.0**	**5.0**	**0.97**	**0.03**	**0.29**	**1663**	**0.54**

Coarse Spiral Tailings - Superconcentrate

Pioneer - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	1.5	1.1	62.0	11.4	3.8	4.72	0.01	0.14	457	0.00
4000G Magnetics	1.4	1.6	62.0	11.8	3.8	4.68	0.02	0.13	478	0.00
14410G Magnetics	12.2	39.4	23.8	29.4	8.5	5.33	0.03	2.22	2418	0.85
14410G Non Magnetics	40.8	57.9	15.3	14.5	11.1	13.12	0.20	0.46	41827	0.00
calc head	**15.3**	**100.0**	**33.0**	**21.1**	**7.7**	**6.83**	**0.06**	**1.24**	**10404**	**0.42**

Coarse Spiral Tailings - Concentrate

Pioneer - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	0.3	1.3	50.7	20.6	5.1	2.83	0.01	0.12	344	0.00
4000G Magnetics	2.2	13.2	30.2	29.6	6.8	1.05	0.01	0.11	418	0.11
15340G Magnetics	1.1	32.8	20.5	42.9	8.8	2.21	0.01	0.90	696	1.04
15340G Non Magnetics	9.6	52.6	5.9	68.3	5.0	4.42	0.12	0.16	10806	4.28
calc head	**2.3**	**100.0**	**22.7**	**42.3**	**7.7**	**2.38**	**0.02**	**0.63**	**1883**	**1.22**

Tables 10 to 13: Pioneer Coarse Particle Gravity Magnetic Characterisation

Fine Table Superconcentrate

Pioneer - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	2.9	0.3	74.0	10.0	2.3	6.53	0.01	0.16	1232	0.37
4000G Magnetics	43.6	14.8	28.6	7.9	3.2	1.73	0.01	0.14	2114	2.95
17000G Magnetics	36.2	24.3	26.4	9.9	5.7	9.76	0.02	4.71	9528	4.08
17000G Non Magnetics	63.6	60.6	6.8	1.2	15.5	6.01	0.20	0.19	50600	8.63
calc head	**48.0**	**100.0**	**20.5**	**5.6**	**9.6**	**6.55**	**0.10**	**1.64**	**26644**	**5.76**

Fine Table Concentrate

Pioneer - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	2.0	2.1	58.9	14.8	4.1	6.07	0.02	0.22	3084	0.31
4000G Magnetics	4.5	15.3	29.1	30.9	6.5	1.12	0.00	0.18	1414	0.30
15950G Magnetics	3.5	22.1	23.3	34.7	8.2	7.23	0.02	3.89	4272	0.91
15950G Non Magnetics	32.0	60.5	9.6	31.2	11.0	7.49	0.50	0.39	31651	5.01
calc head	**7.9**	**100.0**	**25.7**	**31.6**	**7.8**	**5.53**	**0.09**	**2.08**	**7499**	**1.31**

Fine Spiral Tailings - Superconcentrate

Pioneer - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	43.9	7.1	35.7	7.3	2.1	1.31	0.03	0.09	2065	IS
4000G Magnetics	43.9	16.0	35.7	7.3	2.1	1.31	0.03	0.09	2065	IS
17100G Magnetics	45.3	21.1	24.9	6.7	5.4	7.60	0.08	2.76	12320	IS
17100G Non Magnetics	63.9	55.7	4.9	2.5	16.5	4.71	0.09	0.12	49256	IS
calc head	**53.6**	**100.0**	**18.6**	**4.9**	**9.6**	**4.47**	**0.07**	**0.77**	**26685**	**IS**

Fine Spiral Tailings - Concentrate

Pioneer - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	5.2	3.3	69.4	6.9	2.6	5.29	0.03	0.24	2866	IS
4000G Magnetics	11.3	8.9	35.1	16.4	4.8	1.92	0.01	0.19	1914	IS
16580G Magnetics	13.6	25.7	30.9	17.1	7.4	12.45	0.08	5.91	9293	IS
16580G Non Magnetics	52.0	62.0	13.3	2.5	14.1	10.98	0.28	0.29	46154	IS
calc head	**22.2**	**100.0**	**32.5**	**11.7**	**8.0**	**9.20**	**0.11**	**2.62**	**16838**	**IS**

Fine Spiral Tailings - Cleaner Table Superconcentrate

Pioneer - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	2.3	2.1	55.3	15.7	5.0	3.92	0.04	0.22	3339	IS
4000G Magnetics	2.1	5.9	35.6	25.5	6.2	1.32	0.01	0.14	1491	IS
16410G Magnetics	2.3	20.1	24.2	33.7	10.5	3.87	0.03	2.03	3067	IS
16410G Non Magnetics	38.9	71.9	13.0	13.1	14.4	9.73	0.34	0.35	34801	IS
calc head	**7.0**	**100.0**	**27.0**	**28.2**	**9.8**	**4.13**	**0.06**	**1.32**	**6876**	**IS**

Fine Spiral Tailings - Cleaner Table Concentrate

Pioneer - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	1.0	2.7	49.8	22.7	5.7	2.87	0.04	0.18	1698	IS
4000G Magnetics	0.2	8.3	24.9	41.0	8.4	0.24	0.00	0.05	519	IS
16850G Magnetics	0.4	11.8	17.2	44.9	10.6	1.53	0.01	0.80	1247	IS
16850G Non Magnetics	13.7	77.1	6.8	56.6	9.8	4.50	0.23	0.21	0	IS
calc head	**1.3**	**100.0**	**21.2**	**43.2**	**9.4**	**1.22**	**0.03**	**0.39**	**836**	**IS**

* IS - insufficient sample

Tables 14 to 19: Pioneer Fine Particle Gravity Magnetic Characterisation

Coarse Table Superconcentrate

Treasure - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	1.7	0.1	87.8	5.6	0.4	0.21	0.01	0.05	72	0.19
4000G Magnetics	60.8	19.5	30.0	2.9	0.3	0.00	0.09	0.04	167	0.09
12760G Magnetics	35.1	75.1	28.1	20.1	0.8	0.02	0.18	0.21	332	0.13
12760G Non Magnetics	3.5	5.3	4.5	78.1	1.2	0.20	0.10	0.06	512	0.06
calc head	**24.7**	**100.0**	**20.3**	**40.4**	**0.9**	**0.09**	**0.14**	**0.14**	**383**	**0.10**

Coarse Table Concentrate

Treasure - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	0.7	0.1	84.2	7.4	0.3	0.13	0.01	0.03	32	0.08
4000G Magnetics	10.6	0.9	49.5	25.3	0.6	0.19	0.20	0.13	674	0.52
12980G Magnetics	10.0	89.9	22.7	44.7	0.4	0.00	0.09	0.13	244	0.04
12980G Non Magnetics	0.5	9.1	3.1	87.3	0.5	0.04	0.03	0.02	94	0.01
calc head	**3.9**	**100.0**	**10.7**	**71.8**	**0.5**	**0.03**	**0.05**	**0.05**	**147**	**0.02**

Coarse Spiral Tailings - Superconcentrate

Treasure - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	1.2	0.1	85.8	4.1	0.4	0.39	0.00	0.04	45	IS
4000G Magnetics	55.4	19.4	35.7	2.3	0.3	0.03	0.13	0.07	222	IS
14770G Magnetics	46.8	76.2	32.7	9.0	0.9	0.07	0.22	0.30	477	IS
14770G Non Magnetics	13.8	4.3	1.9	60.7	6.8	1.13	0.48	0.72	7139	IS
calc head	**42.4**	**100.0**	**30.7**	**14.7**	**1.6**	**0.21**	**0.23**	**0.31**	**1301**	**IS**

Coarse Spiral Tailings - Concentrate

Treasure - Magnetic Characterisation	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
1000G Magnetics	0.8	0.2	86.2	4.5	0.3	0.13	0.01	0.04	64	0.00
4000G Magnetics	24.9	14.8	54.7	10.5	0.5	0.05	0.16	0.10	448	0.00
14310G Magnetics	12.2	79.6	27.8	38.9	0.7	0.04	0.12	0.22	347	0.02
14310G Non Magnetics	0.5	5.4	1.3	90.7	1.3	0.05	0.02	0.03	173	0.08
calc head	**5.6**	**100.0**	**14.2**	**67.6**	**1.0**	**0.05**	**0.06**	**0.11**	**244**	**0.05**

Tables 20 to 23: Treasure Coarse Particle Gravity Magnetic Characterisation

Fine Table Superconcentrate

Treasure - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	5.6	0.2	84.7	7.4	0.7	0.80	0.03	0.06	432	0.00
4000G Magnetics	56.4	27.4	34.9	1.7	0.4	0.07	0.13	0.06	346	0.28
16994G Magnetics	55.4	58.2	29.4	2.8	1.4	0.22	0.29	0.35	954	0.22
16994G Non Magnetics	44.4	14.2	2.7	3.5	11.2	5.19	0.92	0.39	17920	10.61
calc head	**52.7**	**100.0**	**27.5**	**2.7**	**2.8**	**1.03**	**0.35**	**0.27**	**3648**	**1.98**

Fine Table Concentrate

Treasure - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	5.9	1.2	76.4	6.7	1.2	0.85	0.10	0.14	947	0.17
4000G Magnetics	20.1	11.8	59.8	7.6	0.8	0.23	0.20	0.17	930	0.11
17000G Magnetics	21.7	77.2	41.4	16.4	1.3	0.32	0.22	0.46	918	0.52
17000G Non Magnetics	12.1	9.8	3.6	50.9	6.6	2.89	0.48	0.19	4002	0.51
calc head	**19.4**	**100.0**	**38.9**	**20.4**	**2.1**	**0.73**	**0.25**	**0.37**	**1405**	**0.45**

Fine Spiral Tailings - Superconcentrate

Treasure - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	57.0	6.6	33.5	2.4	0.7	0.15	0.11	0.07	544	IS
4000G Magnetics	57.0	30.0	33.5	2.4	0.7	0.15	0.11	0.07	544	IS
17670G Magnetics	58.8	48.4	27.4	2.4	1.6	0.26	0.24	0.28	1370	IS
17670G Non Magnetics	57.2	15.1	1.3	3.8	14.5	3.44	0.57	0.38	0	IS
calc head	**57.9**	**100.0**	**25.7**	**2.6**	**3.2**	**0.70**	**0.24**	**0.22**	**854**	**IS**

Fine Spiral Tailings - Concentrate

Treasure - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	8.7	5.1	73.4	7.1	1.3	0.82	0.10	0.23	1274	IS
4000G Magnetics	36.9	27.3	48.8	3.7	0.5	0.10	0.16	0.11	558	IS
17670G Magnetics	36.1	46.8	38.0	6.9	1.6	0.55	0.25	0.61	1320	IS
17670G Non Magnetics	43.7	20.8	2.9	7.1	12.8	5.45	0.74	0.48	0	IS
calc head	**32.3**	**100.0**	**41.8**	**6.2**	**3.0**	**1.25**	**0.28**	**0.40**	**926**	**IS**

Fine Spiral Tailings - Cleaner Table Superconcentrate

Treasure - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	1.7	1.3	86.7	3.9	0.5	0.34	0.05	0.06	144	IS
4000G Magnetics	14.6	22.3	58.1	9.9	0.9	0.10	0.17	0.17	783	IS
17650G Magnetics	12.3	60.8	41.8	21.3	1.6	0.27	0.15	0.41	867	IS
17650G Non Magnetics	11.4	15.5	3.0	51.0	9.6	2.81	0.30	0.22	4271	IS
calc head	**11.6**	**100.0**	**42.6**	**22.4**	**2.6**	**0.65**	**0.17**	**0.31**	**1326**	**IS**

Fine Spiral Tailings - Cleaner Table Concentrate

Treasure - Magnetic Characterisation	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
1000G Magnetics	0.8	0.9	83.0	6.2	0.7	0.24	0.07	0.07	113	IS
4000G Magnetics	2.6	7.5	53.7	21.8	1.9	0.12	0.11	0.13	766	IS
17200G Magnetics	2.7	78.4	24.3	48.1	1.1	0.08	0.05	0.16	385	IS
17200G Non Magnetics	0.8	13.1	2.9	84.1	2.9	0.21	0.02	0.03	273	IS
calc head	**2.0**	**100.0**	**20.2**	**57.7**	**1.7**	**0.13**	**0.04**	**0.11**	**363**	**IS**

* IS - insufficient sample

Tables 24 to 29: Treasure Fine Particle Gravity Magnetic Characterisation

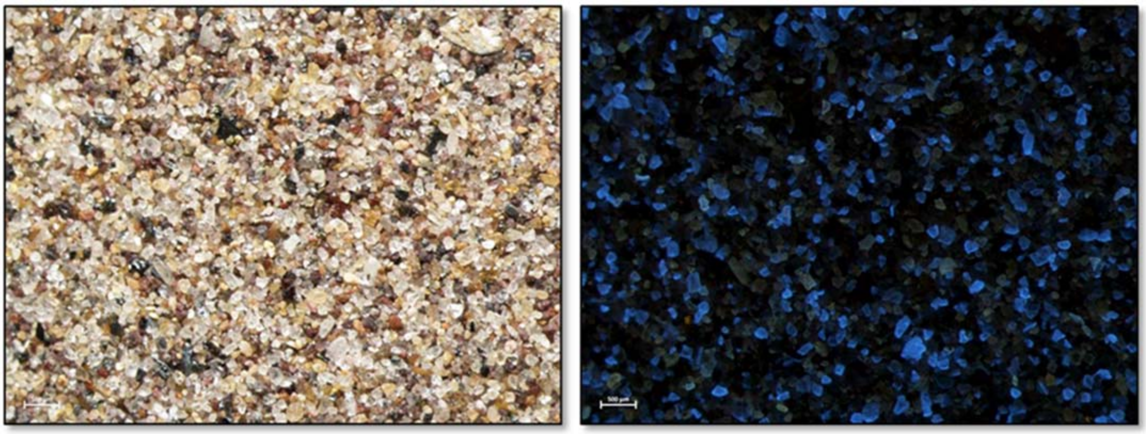


Figure 6: 44% WO3 in the Treasure fine spiral non-magnetics, one image under UV light

In all tests, minimal WO_3 reported to the magnetic concentrate at 1000 gauss as expected. WO_3 that reported to the 1000 gauss concentrate was more than likely entrained within the highly magnetic constituents as ultra-fines.

For the Pioneer material, the majority of the WO_3 reported to the non-magnetic fraction and the magnetic concentrate (>4000 gauss). For the coarse super-concentrate and concentrate, WO_3 has upgraded significantly in the 4000 gauss and non-magnetic concentrates. This suggests the combined presence of Wolframite in the magnetic concentrate and Scheelite in the non-magnetic concentrate. The magnetic characterisation for the fine material resulted in an upgrade of WO_3 in the non-magnetic concentrate only.

Bi, Mo and Au also upgraded to the non-mags fraction. Apart from 4000 gauss, S was reasonably well distributed between magnetic and non-magnetic concentrates.

For the Treasure material, the majority of WO_3 reported to the magnetic concentrate at >= 4000 gauss. This suggests the presence of weakly magnetic Wolframite at a greater proportion relative Scheelite. For the coarse super-concentrate and concentrate, WO_3 has upgraded significantly >=4000 gauss. The magnetic characterisation for the fine material resulted in a reasonably even upgrade of WO_3 profile across the magnetic and non-magnetic concentrates.

In general, Bi and S were upgraded to the non-magnetic concentrate and Cu was reasonably well distributed between the magnetic and non-magnetic concentrates.

Magnetic and non-magnetic portions recovered at the various gauss were classified as either +50%concentrate, -50% concentrate, middlings or tailings.

The results of the gravity test work were combined with the magnetic characterisation results to provide a combined picture of the overall results.

Tables 30 and 31 provide a summary of the various material classification post magnetic characterisation of gravity concentrates.

Pioneer	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Super-concentrate (+50%)	63.3	16.9	6.9	1.2	15.5	6.09	0.20	0.19	50464	8.22
Concentrate (-50%)	12.3	59.6	15.9	46.6	5.8	3.22	0.09	1.11	6486	1.61
Middlings	1.2	5.8	20.7	45.9	6.2	1.22	0.01	0.39	643	0.26
Tailings	0.2	17.8	11.1	67.1	3.2	0.20	0.01	0.09	519	0.21
calc head	**0.8**	**100.0**	**11.6**	**65.4**	**3.4**	**0.37**	**0.01**	**0.14**	**857**	**0.29**

Treasure	WO_3	Stage yield WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Super-concentrate (+50%)	57.1	15.2	30.8	2.5	1.0	0.15	0.20	0.19	581	0.17
Concentrate (-50%)	17.6	52.9	28.4	32.9	1.1	0.25	0.16	0.21	788	0.28
Middlings	1.1	4.4	8.5	76.5	0.7	0.06	0.02	0.05	183	0.05
Tailings	0.2	27.5	3.7	86.3	0.3	0.01	0.00	0.01	66	0.01
calc head	**0.6**	**100.0**	**4.3**	**85.0**	**0.4**	**0.02**	**0.01**	**0.02**	**84**	**0.02**

Tables 30 and 31: Various Material Classification

For Pioneer, 16.9% of the WO_3 has reported to the +50% concentrate at a grade of 63.3% and 59.6% has reported to the -50% concentrate at a grade of 12.3% after gravity and magnetic characterisation test work. In total 77% of the WO_3 has been recovered to the concentrate at a weighted grade of 15% WO_3.

For Treasure, 15.2% of the WO_3 has reported to the +50% concentrate at a grade of 57.1% and 52.9% of the WO_3 has reported to the -50% concentrate at *a grade of 17.6% after gravity and magnetic characterisation test work. In total 68% of the WO_3 has been recovered to the concentrate at a weighted grade of 21% WO_3.*

Both Fe and Bi species have upgraded into the concentrate, in the case of Fe some of this is likely associated with the presence of Wolframite. Bi in the Pioneer super concentrate is particularly high considering that a typical tungsten final product specification would contain <300ppm Bi.

Reviewing the assay data, there is significant other mineralisation occurring within the Pioneer material. Cu (0.19%), Au (8.2 ppm) and Bi (50,464 ppm) may be associated with the high S grade of 6%. In the case of CaO grade of 15.5%, this is more likely associated with the presence of Scheelite. Mo (0.2% in the super-concentrate) appears evenly distributed between Pioneer and Treasure.

3.1.5 Flotation

3.1.5.1 Bi Flotation

The original test work plan made provision for 4 x Bi flotation tests only. Bi is a contaminant in a WO_3 concentrate and attracts significant price penalties

According to Wogen (Metals trading company based in London), the upper limits on Bi on tungsten concentrates for APT production is 0.5%. The general specification requirement is 0.15% max with higher levels incurring price penalties in the order of $2/mtu for 0.15- 0.3% Bi, $4/mtu for 0.3-0.5% Bi and $6/mtu at 0.5%. Above 0.5% Bi the concentrate becomes unsaleable.

Two sub-samples of Pioneer coarse table super-concentrate "non-magnetics" material were split out, stage ground to a p80 of 0.106mm in preparation for Bi flotation test work. The starting set of flotation conditions were based on Nagrom's previous Bi flotation experience.

Table 32 and 33 below shows the starting set of flotation conditions and the result of the first Bi flotation test.

Bismuth Flotation Parameters	Test 1
Cell Size (L)	0.5
Impellor Speed (RPM)	1000
Cond. Pulp Density (%w/v)	20
Float . Pulp Density (%w/v)	20
pH	8.0
pH Modifier	NaOH/HCl
Eh (mV)	-
Hostaflot LIB (g)	100
NaHS (g 10%)	-
MIBC (drops)	-
Total Conditioning Time (min)	5
Total Flotation Time (min)	10
Average Temperature (°C)	27.5

Pioneer - Bi float test 1	WO_3	Stage yield Bi	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Concentrate	4.1	51.0	34.9	8.1	2.3	30.53	2.35	10.48	85532	IS
Tailings	33.6	49.0	2.5	46.8	9.3	0.92	0.13	0.26	13271	2.90
calc head	**29.4**	**100.0**	**7.1**	**41.3**	**8.3**	**5.10**	**0.45**	**1.70**	**23465**	**2.49**

Tables 32 and 33: Set of Flotation Conditions

The results were poorer than expected, with only 51% of the Bi recovered to the concentrate. 49% remains in the tailings (WO_3 concentrate) at a grade of 13271 ppm.

To try and improve the removal of Bi, a second flotation test was undertaken using a change in flotation parameters. 3.35g of 10% NaHS solution was added as an activator, 1 drop of MIBC was added to improve the stability of the froth and the total conditioning time was increased from 5 to 10 minutes.

Table 34 shows the result of the Bi flotation test under the revised set of conditions.

Pioneer - Bi float test 2	WO_3	Stage yield Bi	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Concentrate	6.7	40.0	36.6	6.2	3.2	30.64	1.33	9.79	74316	0.00
Tailings	33.4	60.0	3.0	45.4	9.2	1.62	0.25	0.52	16573	3.84
calc head	**30.0**	**100.0**	**7.2**	**40.4**	**8.4**	**5.31**	**0.38**	**1.69**	**23911**	**3.35**

Table 34: Bi Flotation Test

The results of float test 2 were worse than float test 1 with 60% of the contained WO3 reporting to the concentrate (tailings).

According to E Broughton Jensen's publication, "Tungsten at Hatches Creek – Central Australia Mining and Treatment (1955), Bi is present in 2 forms at Hatches Creek, Bismuthinite (Bi2S3) and Bismutite (Bi2CO3O2). It is most likely that the remaining Bi in the WO3 concentrate is in the form of Bismutite as it is anticipated that the sulphide containing Bismuthinite would have readily floated.

At this point in the program, no direction forward for improving the Bi flotation result was discussed and was put aside in favour of rougher and cleaner flotation on the WO3 concentrates.

Given the results of the gravity and magnetic characterisation test work, the test work scope was expanded to include cleaner flotation of the +50% and -50% concentrate and rougher and cleaner flotation of WO3 from the middlings.

For the concentrate and middlings flotation test work, it was decided between EMP and Nagrom to combine the Pioneer and Treasure samples to ensure enough sample mass and to cut down on the cost by halving the number of float tests. Details of the make - up of the composite can be found in the attached appendices.

The results of pre-flotation tests using cyclone overflow material (tailings) from another tungsten project were used to determine the beginning flotation conditions for the Hatches Creek material.

Pioneer/Treasure composites were stage ground to p100 0.15mm as an estimate to effect the required liberation of WO_3 prior to flotation. No previous mineralogical investigations had been done and the grind size was estimated using photographs and experience from previous test work programs. Consideration should be given to mineralogy test work following the *conclusion of the current test work to better understand the liberation characteristics of the ore.*

The Pioneer/Treasure concentrate composite was split into 2 categories, +50% and -50% WO3 as it was anticipated that each category would require a different flotation regime in order to maximise the grade and yield of WO_3.

3.1.5.2 +50% Flotation

3 flotation tests, A, B and C on the +50% concentrate have been completed in total. Test A was aimed at floating a straight WO_3 concentrate with no sulphide pre-float.

Table 35 and 36 show the starting flotation parameters and results for test A.

Flotation Parameters	+50% A
Cell Size (L)	1.25
Cond. Impellor Speed (RPM)	1000
Float. Impellor Speed (RPM)	750
Pulp Density (%w/v)	20
pH	9.0
pH Modifier	Na_2CO_3/HCl
Na_2SiO_3 (g/t)	2000
Flomin C9621 (g/t)	1550
Hostflot H-3403 (g/t)	-
FS2 (g/t)	-
Flotigam EDA (g/t)	-
MIBC (drops)	7
Total Conditioning Time (min)	32
Total Flotation Time (min)	13
Average Temperature (°C)	22.0

Float Test A	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Concentrate 1	5.3	0.3	24.4	1.3	1.2	30.51	12.97	17.62	94082	IS
Concentrate 2	27.3	0.3	18.3	4.8	5.4	16.90	5.07	8.81	115378	IS
Concentrate 3	35.9	0.8	15.5	2.5	8.4	15.55	2.61	5.41	102807	IS
Concentrate 4	55.9	1.4	9.0	3.5	13.3	6.92	0.52	1.58	59516	IS
Concentrate 5	70.8	32.8	4.3	0.5	17.6	3.53	0.05	0.13	26230	IS
Concentrate 6	66.0	23.0	8.3	0.8	16.1	5.97	0.06	0.11	27163	IS
Concentrate 7	60.6	9.7	12.5	1.3	14.3	8.14	0.07	0.11	31990	IS
Concentrate 8	57.1	1.7	14.6	2.9	12.8	8.86	0.10	0.16	31007	IS
Concentrate 9	50.7	0.4	15.6	8.4	9.3	4.37	0.11	0.20	35489	IS
Concentrate 10	50.7	0.2	15.6	8.4	9.3	4.37	0.11	0.20	35489	IS
Concentrate 11	50.7	0.3	15.6	8.4	9.3	4.37	0.11	0.20	35489	IS
Concentrate 12	54.3	0.2	16.2	4.8	11.0	7.49	0.09	0.15	26535	IS
Concentrate 13	54.3	0.3	16.2	4.8	11.0	7.49	0.09	0.15	26535	IS
Tailings	46.7	28.6	23.7	13.1	2.5	1.30	0.07	0.10	25047	IS
calc head	**57.6**	**100.0**	**13.8**	**5.4**	**10.8**	**4.59**	**0.39**	**0.60**	**29887**	**IS**

* IS - insufficient sample

Tables 35 to 36: Starting flotation parameters

2000 g/t of Na_2SiO_3 was added to assist in the depression of SiO_2 and 1550 g/t of Flomin C9621 was added as WO_3 collector. The test was started and maintained at a pH of 9, based on previous experience floating WO_3 bearing minerals.

As the results show, there is a preferential recovery of sulphides to the first 3 concentrate "pulls". Concentrate 1 assays 30.5% S down to 15.5% in the 3rd concentrate. Concentrate 1 has also produced significant Cu (17.6%) and Mo (13%) grades. They are significant enough to warrant grade and recovery optimisation in future test work programs. Bi, most likely in the form of Bismuthinite has concentrated with the sulphides in the first 4 concentrates. The remaining Bi is evenly distributed across the remaining concentrates and tailings. As previously stated, the grade of Bi needs to be in the order 300 ppm in the final concentrate to be in line with concentrate specifications.

Between concentrate pull 5 to 8, approx. 67% of the contained WO_3 is recovered at average grade of 67.2% WO_3. 28.6% remain in the flotation tailings.

It was decided to conduct another test (test B) with the aim of pre-floating a sulphide concentrate and improving the recovery of WO_3. Flomin C9621 was replaced with 300 g/t of Hostflot H-3403 to assist the sulphide flotation and 800 g/t of FS2 was added as a collector for WO_3. Frother was increased to improve the stability of the froth and flotation time increased from 13 mins to 21 mins. All other conditions remained unchanged.

Table 37 shows the results of flotation test B.

Float Test B	WO_3 %	Stage yield WO3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Pre-float 1	3.9	0.5	49.7	2.3	0.9	41.80	1.52	5.45	64767	IS
Pre-float 2	16.2	0.2	36.3	4.3	3.4	28.29	0.57	2.67	103507	IS
Pre-float 3	27.4	0.4	27.6	5.3	5.4	19.68	0.53	1.83	96397	IS
Concentrate 1	38.2	1.1	14.8	6.5	7.5	10.31	2.06	0.78	120249	IS
Concentrate 2	44.6	0.9	12.0	6.7	8.7	5.95	0.96	0.39	108254	IS
Concentrate 3	42.1	0.4	10.1	6.8	8.9	5.64	0.86	0.77	133125	IS
Concentrate 4	47.5	0.6	9.9	6.5	9.8	3.77	0.26	0.26	110120	IS
Concentrate 5	50.2	0.6	9.8	7.0	10.6	2.83	0.16	0.21	77750	IS
Concentrate 6	51.1	0.6	10.2	6.5	10.5	2.49	0.09	0.19	77205	IS
Concentrate 7	65.5	2.4	3.2	2.0	16.0	1.77	0.23	0.08	20297	IS
Concentrate 8	55.5	0.6	9.1	6.6	11.8	1.29	0.07	0.11	34486	IS
Concentrate 9	54.4	0.5	9.0	5.8	11.6	1.58	0.07	0.12	45361	IS
Concentrate 10	55.0	0.8	9.6	6.1	11.5	1.63	0.07	0.29	42534	IS
Tailings	64.8	90.5	9.9	4.4	12.4	0.48	0.08	0.05	20742	1.51
calc head	**58.4**	**100.0**	**13.1**	**4.4**	**11.3**	**4.21**	**0.24**	**0.51**	**30256**	**1.23**

* IS - insufficient sample

Table 37: Results of flotation test B

Float test B produced an unexpected but pleasing result. Instead of floating WO_3 into the concentrate, the test has successfully reverse floated Si leaving a 64.8% WO_3 in the tailings with a yield of 90.5%. S grades in the pre-float increased from test A to test B as expected, however Cu and Mo grades decreased in the pre-float for no apparent reason.

To confirm the success of the silica reverse flotation result in test B, a third float test, test C was conducted. FS2 collector and Na_2SiO_3 silica depressant were removed, 205 g/t of Flotigam EDA was as an alternative

WO_3 collector. Conditioning time was reduced to 20 minutes and flotation time increased to 25 minutes. All other conditions remained unchanged.

Table 38 shows the results of flotation test C.

Float Test C	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Pre-float 1	27.8	4.3	31.2	2.1	5.5	24.84	1.83	4.48	53294	IS
Pre-float 2	37.2	4.7	27.5	3.1	7.5	19.75	0.81	1.44	38105	IS
Pre-float 3	34.9	1.4	28.9	3.4	6.9	20.44	0.77	1.02	48710	IS
Concentrate 1	16.5	0.3	32.0	10.5	3.3	21.76	2.81	0.70	62935	IS
Concentrate 2	20.0	0.3	27.9	13.5	4.3	15.24	0.80	0.66	68709	IS
Concentrate 3	17.9	0.3	22.8	20.8	4.1	8.45	0.31	0.54	80852	IS
Concentrate 4	21.9	0.4	22.5	19.9	4.9	7.06	0.13	0.56	64017	IS
Concentrate 5	32.7	0.8	18.8	12.6	6.8	6.12	0.18	0.41	49878	IS
Concentrate 6	27.1	2.5	12.1	36.3	6.7	1.80	0.08	0.34	47443	IS
Tailings	65.5	85.2	10.1	3.2	12.3	0.44	0.06	0.06	19866	IS
calc head	**55.5**	**100.0**	**14.4**	**5.4**	**10.6**	**4.93**	**0.32**	**0.59**	**28257**	**IS**

* IS - insufficient sample

Table 38: Results of flotation test C

Due to insufficient mass pull in some of the concentrates, some concentrates were combined to enable an assay to be completed.

The grade and yield of WO_3 in the tailings (concentrate in this case) is similar to that in test B. Interestingly though, 10.3% of the contained WO_3 reported to the pre-float concentrate compared to 1.1% in test B. Sulphide collectors should not collect oxides, the most likely explanation is fine WO_3 particles being entrained in the froth. Consideration should be given to de-sliming prior to flotation in future metallurgical test work campaigns.

Fe, Cu and S grades in the pre-float concentrate were generally lower than that of test B.

In all 3 tests, Bi levels in the concentrate far exceed the concentrate target grade of <0.03% (300ppm). Future test work programs will need to address the amenability of the ore to reducing Bi at or below this target grade.

Given that test C had produced a similar result to test B and a marketable WO_3 concentrate grade of sufficient recovery had been produced, it was decided to dis-continue any more test work on the +50% material given limitations in the test work budget.

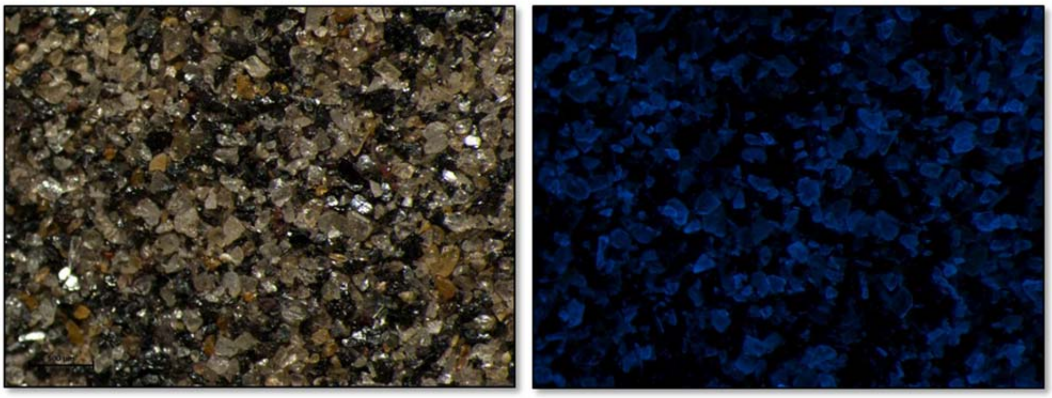


Figure 7: 65% WO3 concentrate from combined Pioneer/Treasure +50% flotation test C, one image under UV light

3.1.5.3 -50% Flotation

7 flotation tests, test D, E, F, G, H, I, J were completed on the -50% concentrate as part of this test program. Test D mimicked the conditions for test A in the +50% concentrate and was aimed at floating a straight WO_3 concentrate with no sulphide pre-float.

Table 39 shows the results of flotation test D

Float Test D	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Concentrate 1	34.8	14.7	22.1	2.6	8.8	18.10	0.78	11.64	5909	IS
Concentrate 2	32.0	3.4	23.2	6.6	8.1	16.58	0.61	9.21	10513	IS
Concentrate 3	36.8	5.2	22.9	9.3	9.1	12.97	0.33	2.56	6081	IS
Concentrate 4	37.5	6.6	24.8	7.6	9.9	12.66	0.22	1.42	7233	IS
Concentrate 5	28.3	4.7	23.9	12.2	11.5	7.77	0.12	0.62	9599	IS
Concentrate 6	22.5	2.7	23.7	23.9	6.3	3.47	0.10	0.27	7891	IS
Concentrate 7	21.9	2.5	24.4	26.7	3.9	2.64	0.10	0.23	7177	IS
Concentrate 8	14.5	0.5	23.6	33.5	2.9	1.11	0.09	0.23	4628	IS
Concentrate 9	14.5	0.4	23.6	33.5	2.9	1.11	0.09	0.23	4628	IS
Concentrate 10	9.7	0.1	24.1	36.5	2.7	0.91	0.08	0.26	4025	IS
Concentrate 11	9.7	0.2	24.1	36.5	2.7	0.91	0.08	0.26	4025	IS
Concentrate 12	17.8	1.1	24.1	29.7	2.8	1.42	0.08	0.19	4618	IS
Concentrate 13	17.8	0.4	24.1	29.7	2.8	1.42	0.08	0.19	4618	IS
Tailings	11.3	57.6	23.6	41.8	2.9	0.23	0.07	0.13	2194	IS
calc head	**15.4**	**100.0**	**23.6**	**35.5**	**3.9**	**2.59**	**0.14**	**1.14**	**3234**	**IS**

* IS - insufficient sample

Table 39: Results of flotation test D

As seen in the +50% results, there is a preferential recovery of sulphides to the first 4 to 5 concentrate "pulls". Concentrate 1 assays 18.1% S, 11.6% Cu and 0.78% Mo. There is a steady decrease in the grade of S, Cu and Mo through concentrate "pulls" 2-5.

Unlike the result in test A, approx. 30% of the WO_3 has reported to the sulphide concentrates (pulls 1-4). There is no obvious reason for the high recovery of WO_3 to the first 4 concentrates, the most likely cause being the

entrainment of fine particles of WO_3 minerals within the froth. 57.6% of the WO_3 reported to the tailings at grade of 11.3%.

As with test A, Bi, most likely in the form of Bismuthinite has concentrated with the sulphides with the remaining Bi evenly distributed across the remaining concentrates.

As with the +50% concentrate, it was decided to conduct another test (test E) with the aim of pre-floating a sulphide concentrate and improving the recovery of WO_3. Conditions for test B were replicated, with the exception that less drops of MIBC frother were added.

The results of test E showed that both the grade and recovery of WO_3 in the sulphide pre-float concentrates dropped dramatically. Recovery of WO_3 to the *remaining concentrates was very poor with 85% of the WO_3 reporting to the tailings.*

Nagrom and EMP consulted chemical manufacturer Clariant to interpret the results and to recommend a flotation regime to take forward into the next flotation test F.

Table 40 shows the flotation regime recommended by Clariant to use in test F.

Flotation Parameters	-50% F
Cell Size (L)	1.25
Cond. Impellor Speed (RPM)	1000
Float. Impellor Speed (RPM)	750
Pulp Density (%w/v)	20
pH	9.0 (Pre-Float Con 1-3) 9.0 (Con 1-4) 8.0 (Con 5-6) 7.2 (Con 7-8) 6.2 (Con 9-12)
pH Modifier	NaOH/HCl
Na_2SiO_3 (g/t)	4000
Flomin C9621 (g/t)	-
Hostflot H-3403 (g/t)	300
FS2 (g/t)	3500
Flotigam EDA (g/t)	-
MIBC (drops)	-
Total Conditioning Time (min)	28
Total Flotation Time (min)	24
Average Temperature (°C)	19.0

Table 40: Flotation regime test F

The major changes for test F included a gradual decrease in the pH from 9.0 to 6.2 to see the effect of pH on flotation performance (if any), an increase in Na_2SiO_3 for Si depression and an increase in FS2 from 800 g/t to 3500 g/t.

Table 41 below shows the result of flotation test F.

Float Test F	WO_3 %	Stage yield WO_3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Pre-float 1	1.6	0.5	46.7	4.8	0.8	36.44	0.84	17.10	10292	IS
Pre-float 2	4.6	0.3	37.2	20.5	2.0	20.38	0.36	6.23	15862	IS
Pre-float 3	6.1	0.3	32.9	21.6	2.3	14.63	0.33	5.00	16921	IS
Concentrate 1	11.4	0.8	23.1	28.7	3.5	5.35	0.51	1.45	9961	IS
Concentrate 2	10.4	0.5	23.6	30.3	3.3	4.04	0.19	0.88	11200	IS
Concentrate 3	11.3	0.4	21.6	31.2	3.6	2.64	0.13	0.52	9482	IS
Concentrate 4	11.3	0.4	21.6	31.2	3.6	2.64	0.13	0.52	9482	IS
Concentrate 5	14.4	0.5	20.9	24.7	4.5	4.95	0.14	0.55	23279	IS
Concentrate 6	14.4	0.4	20.9	24.7	4.5	4.95	0.14	0.55	23279	IS
Concentrate 7	41.0	1.1	12.3	13.2	11.2	2.87	0.30	0.33	25489	IS
Concentrate 8	36.8	1.1	13.6	15.3	10.2	2.48	0.26	0.32	21496	IS
Concentrate 9	60.0	12.7	6.9	3.3	17.7	0.44	0.32	0.07	2110	IS
Concentrate 10	59.7	9.9	10.2	4.3	14.4	0.31	0.27	0.09	2362	IS
Concentrate 11	45.2	15.5	23.3	10.4	4.9	0.70	0.16	0.21	4085	IS
Concentrate 12	39.7	10.5	27.8	10.8	4.2	0.46	0.15	0.22	4573	IS
Tailings	10.7	45.3	22.8	42.5	3.2	0.13	0.07	0.24	2491	IS
calc head	**16.8**	**100.0**	**23.6**	**33.5**	**4.1**	**2.66**	**0.15**	**1.26**	**3978**	**IS**

Table 41: Results of flotation test F

There is a significant increase in the WO_3 grade and yield from concentrate 7 through to concentrate 12, corresponding with a pH range 7.2 down to 6.2. This result is contrary to previous WO_3 flotation test work programs undertaken elsewhere, where it is generally accepted that WO_3 flotation is optimised at pH 9-10 range. It is difficult to determine the exact reason for the difference, a theory might be that the result of test F may be influenced by the poly-metallic nature of the sample.

Recovery of WO_3 was poor with 45.3% of the contained WO_3 reporting to the tailings.

Also of significance is the high grade Cu from pre-float 1, grading 17.1% Cu which could be considered a valuable by-product.

In order to increase the recovery of WO_3, a new set of flotation conditions were set. This included repeating the pre-float conditions from test F, dropping the pH to 6 during the conditioning stage with the collector prior to WO_3 flotation. All other parameters remained the same.

Table 42 below shows the results of test G.

Float Test G	WO_3 %	Stage yield WO3 %	Fe_2O_3 %	SiO_2 %	CaO %	S %	Mo %	Cu %	Bi ppm	Au ppm
Pre-float 1	1.2	0.5	46.8	6.0	0.7	35.65	0.78	16.41	14110	7.88
Pre-float 2	6.6	0.6	31.7	25.8	2.5	12.75	0.19	4.35	11402	IS
Pre-float 3	7.8	0.4	28.0	29.3	2.7	7.38	0.16	2.28	11615	IS
Concentrate 1	47.5	42.6	19.3	6.8	9.5	0.84	0.24	0.22	4553	1.19
Concentrate 2	38.8	10.3	27.5	11.7	5.5	0.41	0.19	0.26	4912	IS
Concentrate 3	38.6	12.1	30.7	10.4	3.3	0.43	0.16	0.28	4999	IS
Concentrate 4	31.1	8.9	33.7	13.7	3.4	0.33	0.16	0.29	4430	IS
Concentrate 5	22.9	6.3	35.6	17.8	3.5	0.34	0.13	0.31	3824	IS
Concentrate 6	18.8	4.1	36.3	20.7	3.7	0.30	0.14	0.28	3360	IS
Concentrate 7	16.2	2.8	36.2	21.8	3.9	0.31	0.12	0.26	3208	0.35
Concentrate 8	13.1	1.8	34.9	26.1	4.1	0.24	0.12	0.22	2364	IS
Concentrate 9	9.5	1.4	34.0	29.4	4.2	0.23	0.11	0.21	2060	0.19
Concentrate 10	8.3	0.9	32.1	33.0	4.4	0.18	0.09	0.18	1644	IS
Concentrate 11	6.5	0.8	31.1	34.9	4.4	0.17	0.09	0.17	1584	IS
Concentrate 12	5.5	0.6	29.2	38.7	4.5	0.15	0.09	0.16	1235	IS
Tailings	1.9	5.8	15.6	58.6	2.3	0.02	0.02	0.07	606	IS
calc head	**15.0**	**100.0**	**23.8**	**35.5**	**3.7**	**2.48**	**0.13**	**1.15**	**3086**	**0.62**

* IS - insufficient sample

Table 42: Results of test G

Reducing the pH to 6 straight after the pre-float has had a dramatic effect on the recovery of WO_3. The first 3 concentrates have recovered 65% of the contained WO_3. Overall recovery of WO_3, excluding that recovered in the pre-float, was 92.8%. This is a massive increase compared to float test F recovery of 55%.

Au has upgraded to 7.88 g/t to pre-float concentrate 1. Unfortunately, due to the sample size, it has been difficult to have enough mass to consistently assay for Au. It is recommended that future test work programs have adequate starting mass to ensure that all potential valuable by-products, including Au, can be measured.

All other assays followed a similar trend to test F.

The set of flotation conditions for test G provided the flotation regime for a combined rougher cleaner flotation test, test H. The aim was to replicate the result of test G to produce a rougher concentrate and re-float the concentrate through 2 cleaning stages to a marketable grade.

Table 43 show the starting conditions for the rougher cleaner test H.

Flotation Parameters	-50% H Rougher	-50% H Cleaner 1	-50% H Cleaner 2
Cell Size (L)	1.25	1.25	0.50
Cond. Impellor Speed (RPM)	1000	1000	1000
Float. Impellor Speed (RPM)	750	750	750
Pulp Density (%w/v)	20	9	16
pH	9.0 (Pre-Float Con 1-3) 6.0 (Con 1-8)	6.0	6.0
pH Modifier	NaOH/HCl	NaOH/HCl	NaOH/HCl
Na_2SiO_3 (g/t)	4000	2000	1000
Flomin C9621 (g/t)	-	-	-
Hostflot H-3403 (g/t)	300	-	-
FS2 (g/t)	2500	-	-
Flotigam EDA (g/t)	-	-	-
MIBC (drops)	4	2	3
Total Conditioning Time (min)	22	10	10
Total Flotation Time (min)	20	4	4
Average Temperature (°C)	19.0	19.0	19.5

Table 43: Rougher cleaner test H

The cleaning was conducted in 2 stages, with the concentrate from cleaning stage 1 used as feed for cleaning stage 2. 2000 g/t of Na_2SiO_3 was added to *cleaner stage 1 to depress Si, and a further 1000 g/t was added to cleaner stage 2. Apart from a few drops of MIBC and NaOH/HCl to maintain pH, no other chemicals were added to the cleaning stages.*

Table 44 shows the results of flotation test H.

Float Test H	WO_3	Stage yield WO3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Pre-float Concentrate	3.1	1.7	40.9	12.9	1.3	27.24	0.48	12.32	11696	0.00
Cleaner Concentrate	37.6	51.6	27.6	7.8	7.4	0.75	0.19	0.30	4726	0.00
Cleaner Tail	21.4	41.3	30.3	24.0	4.6	0.31	0.14	0.21	3163	0.39
Rougher Tail	1.6	5.4	15.1	59.4	2.2	0.04	0.03	0.07	575	0.05
calc head	**14.1**	**100.0**	**23.7**	**36.2**	**3.8**	**2.38**	**0.13**	**1.11**	**2948**	**0.13**

* IS - insufficient sample

Table 44: Results of flotation test H

Float test H has produced a 37.6% WO_3 concentrate at a recovery of 51.6%. Comparing the result from test G, the first 4 concentrates produced a weighted average concentrate grade of 42% WO_3 at a recovery of 74%. Clearly more work is required to optimise the cleaner flotation.

Due to sample mass and budgetary constraints, it was decided not to continue optimising the -50% concentrate flotation. It is recommended that any future test work include the optimisation of the roughing and cleaning stages for the -50% concentrate.

3.1.5.4 Middling Flotation

Due to budget constraints, only 1 flotation test on the Hatches Creek middlings was completed, test I. Priority was given to flotation tests on the +/-50% concentrates.

The flotation regime from float test G was replicated, dropping the pH to 6 during the conditioning stage with the collector prior to WO_3 flotation. All other parameters remained the same.

Table 45 below shows the result of rougher float test I on the Hatches Creek Pioneer and Treasure middlings composite.

Float Test I	WO_3	Stage yield WO3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au
	%	%	%	%	%	%	%	%	ppm	ppm
Pre-float 1	0.8	1.8	32.4	32.1	2.3	21.98	0.15	7.83	5999	IS
Pre-float 2	0.9	0.8	27.4	39.8	3.1	15.41	0.07	3.49	9215	IS
Pre-float 3	1.1	0.5	19.2	47.3	3.6	8.10	0.06	1.49	9608	IS
Concentrate 1	5.3	51.8	21.0	35.1	12.4	0.28	0.06	0.07	2289	IS
Concentrate 2	2.9	6.6	21.9	42.1	9.1	0.23	0.04	0.07	2112	IS
Concentrate 3	2.2	4.0	23.3	42.4	7.8	0.16	0.04	0.08	1636	IS
Concentrate 4	1.9	4.6	23.7	42.1	7.8	0.14	0.03	0.07	1434	IS
Concentrate 5	1.7	4.9	23.4	43.4	7.4	0.11	0.02	0.07	971	IS
Concentrate 6	1.5	4.0	22.8	44.6	7.4	0.10	0.02	0.07	794	IS
Concentrate 7	1.0	4.0	21.7	47.3	7.1	0.06	0.02	0.05	504	IS
Concentrate 8	0.7	3.4	19.0	52.3	6.6	0.05	0.01	0.03	349	IS
Concentrate 9	0.6	1.9	18.2	53.8	6.4	0.03	0.02	0.04	294	IS
Concentrate 10	0.4	1.4	15.9	58.2	5.9	0.01	0.01	0.03	207	IS
Concentrate 11	0.3	0.7	14.9	60.6	5.5	0.02	0.01	0.02	198	IS
Concentrate 12	0.3	0.8	13.5	63.4	5.1	0.00	0.01	0.02	165	IS
Tailings	0.1	8.7	5.1	83.0	1.4	0.00	0.00	0.00	64	IS
calc head	**0.8**	**100.0**	**10.6**	**69.9**	**3.7**	**0.55**	**0.01**	**0.18**	**583**	**0.00**

* IS - insufficient sample

Table 45: Rougher float test I

For a one off first test, test I produced a very promising result. Just on 52% of the contained WO_3 was recovered in concentrate 1 at a grade of 5.3%, up from a head grade of 0.8%. Both Fe and Si species are at high grades in the concentrates, with a small proportion of the Fe likely associated with wolframite. It is recommended that future tests would target increased depression of Si to improve the WO_3 concentrate grade as well as optimisation of WO_3 collector to maximise WO_3 recovery.

Like previous tests, Cu has upgraded to the pre-float to 7.8% and further supports the possibility of Cu becoming a valuable by-product.

It is recommended that future metallurgical test work programs include mineralogical investigations of flotation feed and products as this will greatly assist in determining the optimum flotation conditions.

4.0 CIRCUIT SUMMARY

At the conclusion of all gravity and flotation test work, the results of Pioneer and Treasure were combined using a weighted average of the respective stockpiles to produce an overall test work circuit summary. The circuit summary was broken down into unit processes to show the grade and recovery results at specific parts of the test work flow sheet.

Table 46 below shows the overall circuit summary for the combined Pioneer and Treasure Results.

Flowsheet Stage		WO_3	Fe_2O_3	SiO_2	CaO	S	Mo	Cu	Bi	Au	CIRCUIT Deportment						
		%	%	%	%	%	%	%	ppm	ppm	Mass Yield	WO_3	Fe_2O_3	SiO_2	S	Cu	Bi
Pioneer stockpile (t)	85641																
Treasure stockpile (t)	28790																
Head Grade	Feed	0.7	10.0	69.0	2.7	0.3	0.01	0.1	670	0.2	100%	100%	100%	100%	100%	100%	100%
Spiralling	Cons	5.5	14.3	57.9	4.1	1.3	0.05	0.5	2853	1.0	8%	57%	12%	7%	35%	32%	30%
	Mids	0.7	10.2	69.2	3.0	0.3	0.01	0.1	389	0.2	18%	15%	18%	18%	16%	15%	10%
	Tails	0.3	9.5	70.3	2.4	0.2	0.01	0.1	569	0.2	74%	28%	70%	76%	49%	54%	60%
Tabling	Sup-Cons	56.0	20.6	2.9	7.8	3.8	0.12	0.7	19799	0.5	0.04%	4%	0.2%	0.002%	2%	1%	2%
	Cons	15.1	20.3	38.4	5.9	3.2	0.08	1.1	7190	2.6	3%	64%	8%	2%	30%	27%	28%
	Mids	2.1	17.6	54.1	4.8	1.0	0.03	0.3	1130	0.3	4%	11%	7%	3%	15%	12%	7%
	Tails	0.2	9.1	72.0	2.4	0.1	0.01	0.1	403	0.2	92%	21%	86%	95%	53%	61%	64%
Mag Sep + Tabling	Sup-Cons	61.7	12.9	1.6	11.9	4.6	0.20	0.2	37913	6.2	0.2%	16%	0.4%	0.004%	3%	1%	10%
	Cons	13.6	19.0	43.1	4.6	2.5	0.11	0.9	5052	1.3	3%	57%	7%	2%	29%	27%	23%
	Mids	1.2	17.6	53.6	4.8	0.9	0.01	0.3	527	0.2	3%	6%	6%	3%	12%	11%	3%
	Tails	0.2	9.2	71.9	2.5	0.2	0.01	0.1	405	0.2	93%	21%	86%	95%	56%	61%	64%
Total Gravity/Mag Sep	Sup-Cons	59.2	15.5	3.7	9.6	4.0	0.31	0.5	25933	3.2	1%	38%	1%	0.03%	7%	3%	18%
	Cons	15.6	21.5	36.4	5.0	3.3	0.11	1.3	4914	1.0	2%	34%	5%	1%	21%	21%	13%
	Mids	2.0	22.6	46.0	5.5	1.6	0.03	0.6	976	0.4	2%	4%	4%	1%	11%	10%	3%
	Tails	0.2	9.3	71.7	2.5	0.2	0.01	0.1	405	0.2	96%	24%	90%	98%	61%	66%	66%
Flotation	Sup-Cons	66.3	7.7	1.1	16.1	5.3	0.07	0.2	28412	IS	0.3%	27%	0.3%	0.004%	5%	0.4%	10%
	Cons	36.5	28.6	10.3	6.2	0.7	0.19	0.3	5879	0.3	1%	39%	2%	0.1%	3%	2%	13%
	Mids	4.3	23.3	17.4	4.3	18.6	7.90	10.7	57791	0.0	1%	9%	3%	1%	21%	21%	8%
	Tails	0.1	5.8	81.9	1.4	0.1	0.01	0.1	129	0.1	97%	25%	94%	99%	71%	76%	68%
Whole Circuit Summary	**Cons**	**42.7**	**24.3**	**8.4**	**8.2**	**1.7**	**0.16**	**0.3**	**10557**	**IS**	**2%**	**66%**	**3%**	**0.1%**	**8%**	**3%**	**23%**
	Mids	**4.3**	**23.3**	**17.4**	**4.3**	**18.6**	**7.90**	**10.7**	**57791**	**0.0**	**1%**	**9%**	**3%**	**1%**	**21%**	**21%**	**8%**
	Tails	**0.1**	**5.8**	**81.9**	**1.4**	**0.1**	**0.01**	**0.1**	**129**	**0.1**	**97%**	**25%**	**94%**	**99%**	**71%**	**76%**	**68%**

Table 46: Overall Circuit Summary

The overall circuit summary shows that at the conclusion of this test work phase, a 43% WO_3 concentrate has been made with an overall WO_3 recovery *of 66%. 9% and 25% of the total circuit WO_3 remain in the middlings and tailings respectively. Significant grades of S (18.6%), Cu (10.7%) and Mo (7.9%) are evident in the middlings portion.*

The test work has also shown that a +65% WO_3 super-concentrate can be made via flotation with an overall circuit recovery of 27%.

Bi in the final concentrate is very high at 10577ppm. Typical WO_3 concentrates target Bi < 300ppm. Future test work programs will require a major focus on the reduction of Bi, either to final product specification or at least low enough to enable blending to be an option.

5.0 CONCEPTUAL PLANT DESIGN

Given the small nature of the Hatches Creek stockpile re-treatment project (~ 225,000t), there is a need to minimise the capital cost to ensure that it is in line with the size of the project. This will have a direct impact on the process design and grade/yield of WO_3 that can be produced.

Consideration should be given to a capital efficient process design that produces an "intermediate" grade product on site at Hatches Creek that can be sold as is or transported and cleaned in another facility.

A conceptual plant design at site may consist of a simple communition circuit and gravity concentration in the form of spirals and gravity tables, or consideration of optical sorting technology. However, it should be noted that no physical metallurgical test work has been done including hardness, crushability, BWi, RWi and abrasion indices. This information will impact the design of any communition circuit.

The intermediate product, if not sold as is, could be cleaned via magnetic separation and flotation. This stage could be completed at another facility, for example in a pilot plant operated in a metallurgical testing laboratory where existing process equipment, assaying facilities and qualified staff are available. The grade of Bi in the concentrate is very high and requires specific attention. At the current grade of 10557 ppm, even an intermediate product grade concentrate would be un-saleable.

At the completion of the current metallurgical test program, it is recommended that a scoping study be prepared that addresses the issues of process design, capital and operating cost estimates, product/revenue model for different *grade concentrates, location of processing and contracting model for processing.*

6.0 RECOMMENDATIONS

The test work program has demonstrated the amenability of the Pioneer and Treasure material to the recovery of WO_3 at a saleable WO_3 grade.

In summary, Table 47 shows the test work has produced the following WO_3 concentrate grade and recovery.

WO_3 Summary	WO_3 %	Circuit yield WO_3 %
Super Concentrate	66.30	27
Concentrate	36.50	39
Concentrate - Total	**42.70**	**66**
Middlings	**4.30**	**9**
Tailings	**0.10**	**25**
calc head	**0.77**	**100**

Table 47: WO_3 concentrate grade and recovery

Given that the concentrate, either intermediate or final grade, contains elevated Bi, there is considerable doubt whether a 225kT stockpile would underpin the capital investment required to develop a processing plant that is capable of replicating the unit processes of the test work program and reducing Bi to below industry standards. For this reason, it is questionable whether or not the investment required in attaining more bulk representative samples and another metallurgical test work campaign is worth it for a project of this size. It is also highly unlikely that the stockpile is representative of the greater Hatches Creek resource. Amongst other variables, there is a high likelihood that the sulphide mineralisation has oxidised over time and has adversely affected the flotation test work during this program.

Perhaps money is better spent in drilling the wider Hatches Creek project and developing an inferred resource definition, from which more representative fresh metallurgical test work samples can be collected and the costs required for the additional test work can be more justified. This may be an outcome of the planned scoping study at Hatches Creek.

Depending upon the outcome of the scoping study, should further metallurgical test work be required, the program needs to focus on;

1. *Ensuring that a sufficient size test work sample is collected so that test results are not compromised due to low sample mass. It is recommended that future metallurgical test work samples will originate from fresh RC and diamond core drilling.*

2. *Ensuring the sample(s) are taken according to an industry ISO standard. ISO 3082 – Sampling of Iron Ore Fines, would provide the sampling fundamentals to ensure representivity of the Hatches Creek material.*

3. *Recovery of a large enough concentrate mass for marketing purposes.*

4. *Establishing a full set of physical characteristics including hardness, crushability, RWi, BWi and abrasion indices that will feed into a process design.*

5. *Ore and flotation feed and product mineralogy to determine the texture of tungsten minerals, the degree of sulphide oxidation and liberation size.*

6. *Increasing the recovery of WO_3 from the middlings and tailings. At the conclusion of this phase of test work, 34% of the circuit WO_3 was still contained in the middlings/tailings. This can be achieved by;*

    - *Targeting increased depression of Si in the middlings to improve the WO_3 concentrate grade.*

    - *Optimisation of WO_3 collector in the middlings to maximise WO_3 recovery.*

    - *Developing a test work plan aimed at increasing recovery of WO_3 from the tailings.*

7. *Optimisation of the roughing and cleaning stages for the -50% concentrate.*

8. *Evaluation of ore sorting technology as a pre-concentration step.*

9. *Cleaning of the final concentrate to a standard specification grade for a WO_3 concentrate (~65%).*

10. *Separation of Wolframite and Scheelite using magnetic separation as they cannot be sold as a combined concentrate.*

11. *Reduction of Bi in the final concentrate to a saleable specification.*

12. *Recovery of potentially valuable by-products including Au, Cu and Mo.*

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Appendices

Nagrom Testwork