March 14, 2012

GREY RIVER PROJECT

Preliminary Economic Assessment

Submitted to:
Playfair Mining Limited
Suite 520 - 470 Granville Street
Vancouver, BC V6C 1V5

Attention: Mr. Neil Briggs

REPORT

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1.0 SUMMARY

Playfair Mining Ltd (Playfair) commissioned Golder Associates Limited (Golder) to produce an updated National Instrument 43-101 compliant Preliminary Economic Assessment (PEA) for the Number 10 and Number 6 veins on the Grey River Tungsten property.

The property consists of 154 claims owned by Playfair that have been grouped into one mineral license (Number 015686M). The mineral license covers ground adjacent to the Local Service District of Grey River (community of Grey River) on the south coast of Newfoundland. Granitic rocks underlie the northern part of the claim group while amphibolites, quartz-mica schists, pelites and gneisses occupy the southern part. Younger pegmatites cut all rock types and can be locally abundant. Quartz veins hosting the tungsten mineralization commonly occupy a post-tectonic north to northeast trending fault orientation. Wolframite is the dominant tungsten-bearing mineral within the Number 10 Vein although scheelite (a calcium tungstate) is present in minor amounts.

The Number 10 Vein was discovered by a local prospector in the early-1950s. Later work done by the American Smelting and Refining Company (ASARCO) consisted of diamond drilling, trenching, sampling and the development of an underground adit. This work halted in 1970 when tungsten prices dropped. Playfair bought the property in 2004 from South Coast Ventures and drilled 15 HQ size holes on the Number 6 and 10 Veins in 2006 and added an additional 10 holes in 2008 targeting the lower portion of the Number 10 Vein. Geological mapping and sampling of other veins on the property accompanied the drilling.

A bulk sample was taken from one of the ASARCO trenches and submitted to SGS Lakefield Research Europe for metallurgical tests. Insufficient work has been done to this stage to develop a specific flowsheet for the deposit. Further metallurgical testwork is required to demonstrate that an acceptable grade concentrate at an acceptable metallurgical recovery can be achieved. There is a potential upside to the metallurgical results that have been completed to date, especially in terms of maximizing the mass pull to a 65% WO_3 concentrate. However, this must be demonstrated in the next phase of testwork.

During the data validation process and as part of the exploratory data analysis (EDA), a marked difference in the grade distribution between the historical data and the newer drill results was observed. Playfair assays generally returned lower WO_3 values. The difference is currently assumed to be related to the analytical procedure but needs to be evaluated by Playfair.

All indicated blocks in the resource were downgraded due to the lack of original assay certificates for the underground sampling, the difference in grade between the historical data and the newer Playfair data and the difficulty in capturing a representative sample with the small historical EX size drill core.

At the 0.2% Wo_3 cut-off and excluding all mineralization grading less than 0.2% WO_3 over a 1.0 minimum mining width, the updated undiluted mineral inventory indicated 1.2 million tonnes of Inferred mineralization grading 0.730% WO_3 containing 18.8 million pounds of tungsten trioxide or 853,000 metric tonne units (MTU). The bulk of this tonnage is in the Number 10 Vein which contained an Inferred mineral inventory of 1.06 million tonnes grading at 0.760% WO3 for a total of 804,800 metric tonne units of tungsten trioxide.

The complete undiluted mineral inventory was exported to Golder's Engineering team for further economic assessment.

The assessment of environmental and socio-economic considerations is preliminary at this stage and will require further study and development as project details and additional regional and site details become available.

Grey River is proposed as a low-tonnage, high grade operation, with a relatively free-milling ore, shown to be amenable to gravity separation methods, producing a potential concentrate of 60% with a tungsten recovery of 75%. A typical gravity/flotation-based plant processing material such as at Grey River would generally obtain between 85% to 92% recovery to a 65% to 70% concentrate. For this study a recovery of 85% to a 65% concentrate grade was assumed.

The underground mining resource is estimated to be 1,268,306 tonnes at a grade of 0.524% WO_3 using a minimum mining width of 2 m and 0.35% WO_3% cut-off grade and a blasthole open stoping mining method with pastefill. With about 2,400 tonnes per vertical meter of underground mining resource a 400 tpd operation is proposed at this level of engineering study. Using the base case economic parameters the pre-tax cash flow is estimated to be positive at $15.5 million over a mine life of about 9 years (price of $355 per MTU or $16/lb). The net present value for the base case using a 5% discount rate is $2.9 million. The project would generate a NPV (5%) of $47 million at an IRR of 27% at the current metal price of around $440 per MTU ($21 per lb) (at December 2011). The current size of the underground mining resource at Grey River limits the potential production rate and annual cash flow. To achieve higher production rates the deposit strike length needs to be longer but that would also necessitate additional decline, level and ventilation development thus increasing pre-production capital costs.

All of the resources used to develop the underground mining resource in this study are in the Inferred category. The Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary economic assessment will be realized. In order to develop mineral reserves for the Grey River Project as part of a pre-feasibility study the majority of Inferred mineral resources, and all of the underground mining resource as defined here, will have to be upgraded to the Indicated category.

No technical fatal flaws have been identified at this preliminary stage of study for the Grey River property.

2.0 INTRODUCTION

The Grey River Tungsten property is located on the south coast of Newfoundland adjacent to the community of Grey River. The property consists of one mineral licence (Number 12723M) held by Playfair Mining Ltd. of Vancouver, British Columbia, Canada. All of the claims within the licence (142 in all) are in good standing with excess credits sufficient for renewal until September 25, 2013. The tungsten deposit of interest is known as the Number 10 Vein and it is exposed in the eastern part of map sheet NTS 11P/11 (Ramea).

Tungsten mineralization was discovered on the property in 1954. Between 1954 and 1970 the ASARCO explored the Number 10 Vein using surface trenching, sampling and assaying techniques followed by surface diamond drilling and the establishment of 1703.5 metres (m) of underground workings. ASARCO also sampled 25 underground raises.

An in-house historical resource of 473,000 tonnes grading 0.97% WO_3 was estimated by ASARCO for the mineralization above the adit level of the Number 10 Vein. This historical estimate pre-dates the requirements of NI 43-101 and therefore it is not compliant with NI 43-101 and it should not be relied upon.

A diamond drilling program on the Number 10 Vein was carried out by Playfair in the summer of 2006. Sixteen HQ holes (37 millimetres (mm) core diameter) were completed for a total of 2922 m. Twelve of these holes were designed to replicate the results of the historic ASARCO drilling while the remaining four tested the exploration potential of the deposit.

A resource estimate was produced by Wardrop on June 2007 (Wardrop 2007) incorporating the result of the 2006 drill program. The NI43-101 compliant resources reported 852,000 tonnes of Inferred resources grading 0.858% WO_3 at a 0.2% WO_3 cut-off grade. The resource estimate was authored by Christopher Moreton, P.Geo. and the resource reported tonnages and grade at vein width with no dilution added.

The Wardrop 2007 resources were subsequently used in a PEA study titled, Preliminary Economic Assessment of the Grey River Property, authored by Christopher Moreton P.Geo., Wardrop, David Sprott, P.Eng. Golder Associates Limited (Golder) and Andrew Bamber P.Eng., Minesense Technologies Limited, and dated January 15, 2008 (Golder 2008). Golder reported an underground mining resource of 901,911 tonnes at 0.66% WO_3 determined from the Wardrop 2007 estimate. Golder used a longhole open stoping mining method with a 0.4% WO_3 cut-off and a minimum mining width of 2 meters.

In the summer 2008, Playfair added an additional 10 holes amounting to 3854 meter of NQ core drilling. All holes targeting the extension of the Number 10 Vein below the adit level.

In 2011, Playfair commissioned Golder to produce an updated Preliminary Economic Assessment (Scoping Study) on the Grey River property. The data from the 2008 program, as well as that from the historical programs, is used in the current report to estimate a NI 43-101 compliant underground mining resource for Grey River.

2.1 Terms of Reference

Golder Associates Ltd. (Golder) and Desautels Geoscience Ltd. (DGL) were retained by Playfair Mining Ltd. to complete an updated resource estimate and to update the Preliminary Economic Assessment (Scoping Study) of the Grey River property that is compliant with National Instrument 43-101 (NI 43-101).

The persons taking responsibility for specific sections of this report, and the extent of their responsibility for the purposes of NI 43-101 are shown in Table 2-1.

Table 2-1: Qualified Person Responsibilities for Various Sections of Report

Responsible Person	Independent QP	Company	Primary Area of Responsibility	Sections of Responsibility
Pierre Desautels, P.Geo.	Yes	Desautels Geoscience Ltd.	Site visit, resource estimate, geological sections	Inputs to Sections 1.0 to3.0, Sections 4.0 to 12.0, 14, 22 and Sections 24.1 and 25.1
David Sprott, P.Eng.	Yes	Golder Associates Limited	Study compilation, mine design, mining costs, mine and site capital costs, underground mining resource, economic analysis	Inputs to Sections 1.0 to 3.0 and 16.5 Sections 15, 16, 17, 18, 20 (except 20.1.2 and 20.2.3), 21, 24.2 and 25.2.
Andrew Bamber, P.Eng.	Yes	Minesense Technologies Limited	Metallurgy, processing, plants and associated costs	Inputs to Section 1.0 and 16.5, Section 13.0, 20.1.2, 20.2.3, 24.3 and 25.3.

3.0 RELIANCE ON OTHER EXPERTS

3.1 Environmental and Legal

Neither Golder nor DGL have verified the legal status or legal title of any of the claims and has not verified the legality of any underlying agreements for the subject property.

Michael Moore, P.Geo. of Playfair and manager of the exploration program provided input to Sections 4.0 and 11.0. The Environmental and Socio-economic considerations presented in Sections 19.0, 24.4 and 25.4 rely on work done by Bruce Bennett of Stantec. Mr. Bennett visited the Grey River project site in 2007.

3.2 Marketing

Marketing information in this report (Section 18.0) relies on information by Roskill Information Services Ltd. titled "Tungsten: Market Outlook to 2016 (10th edition)" (August 2011) (www.roskill.com). A specific marketing study was not done for this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The property is located adjacent to the fishing community of Grey River on the south coast of Newfoundland (Figure 4.1). The town of Grey River is situated at approximately latitude 47°34'N and longitude 57°6'W.

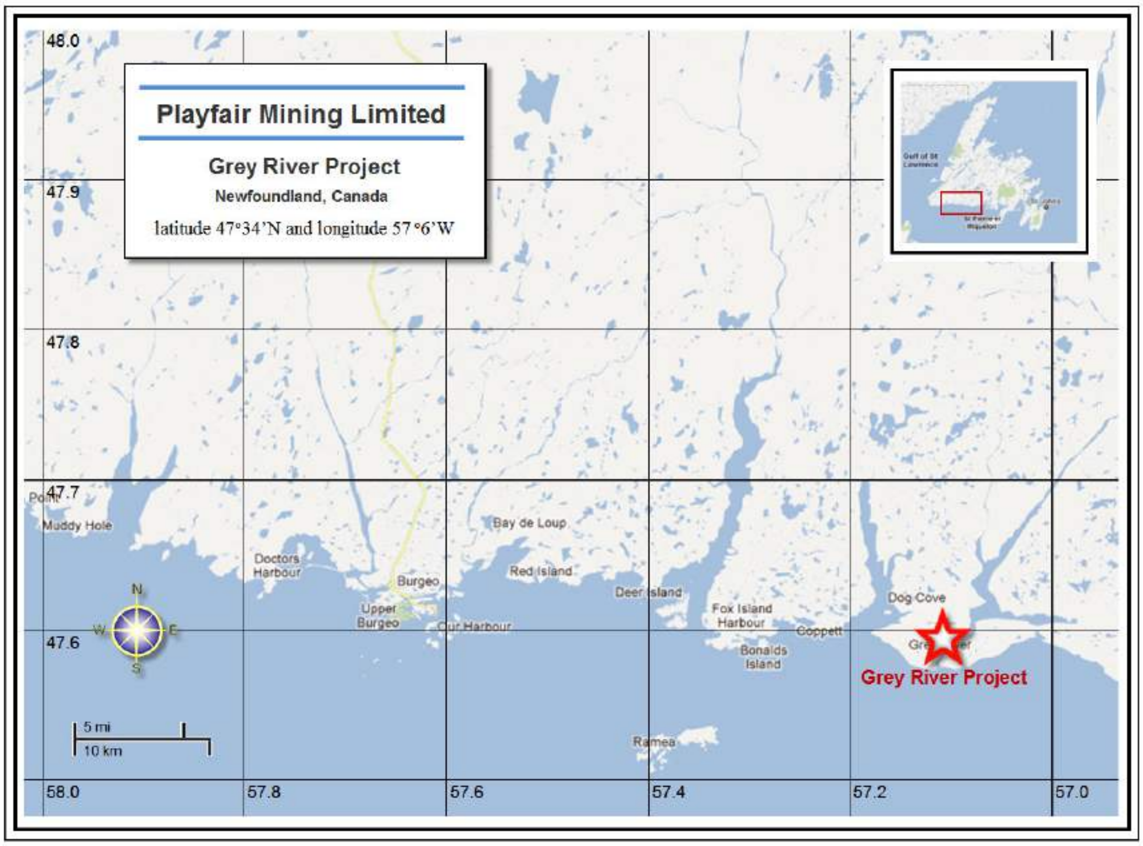

Figure 4.1: Location map for the Grey River Tungsten Property

The Grey River Tungsten property consists of 154 contiguous mining claims grouped into one mineral license (015686M) held by Playfair through a purchase agreement with South Coast Ventures. Since December 2008, this license replaces license 013436M and license 012723M. A review of the Newfoundland and Labrador government website by DGL shows that the mineral license is in good standing with the next report of work due November 24, 2011 and a renewal date of September 25, 2015. The mineral license overlaps the boundary of NTS map sheets 11P/10 and 11P/11 (Figure 4.2).

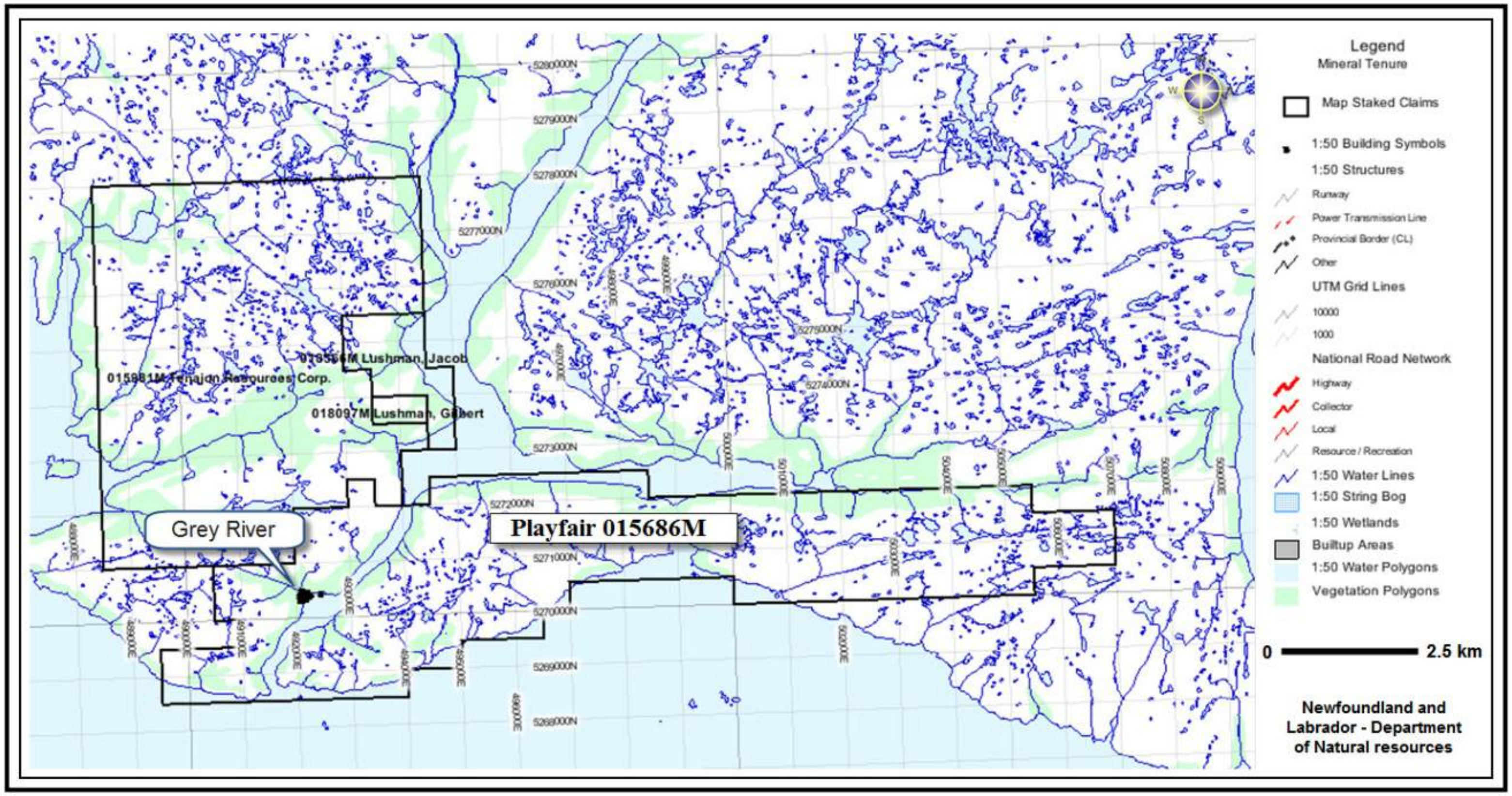

Figure 4.2: Playfair Mining Ltd. Licence 015686M

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility and Infrastructure

Grey River has a daily coastal boat service and a bi-weekly car ferry service from Burgeo, Newfoundland. Burgeo is a port town located approximately 40 kilometres (km) to the west of the property and is connected to the Trans Canada Highway by a paved road. The town of Stephenville is located approximately 130 km northwest of Burgeo. Both Stephenville and Deer Lake, located 170 km north of Burgeo, have airport facilities while Pasadena (20 km south of Deer Lake) is the base for two helicopter companies.

The mineral claims can be reached on foot from Grey River although a helicopter is the preferred mode of transport due to the rugged terrain in vicinity of the claims. The portal to the adit of the Number 10 Vein (developed by ASARCO) is accessible on foot using a short gravel trail from Grey River. The elevation of the portal is approximately 13 m (42 feet) above sea level.

ASARCO engineering drawings indicate that some infrastructure was designed in anticipation of mining the Number 10 Vein. None of this infrastructure was visible during the site visit except for a possible waste rock pad outside the portal. Local dock facilities exist at Grey River although it is speculated that these will need to be expanded when mining commences.

Grey River has a diesel generator that supplies electricity, internet service through a satellite link as well as a wharf owned by the Government of Newfoundland and Labrador.

5.2 Physiography, Elevation and Climate

The central part of the Grey River Tungsten property has an average elevation of 245 m (800 feet) above sea level (ASL). Topographic relief within the immediate vicinity of the Number 10 Vein varies from 200 to 275 m (650 to 900 feet) ASL. Sheer cliffs drop off directly to the sea level along the south and east sides of the property creating numerous hanging river valleys.

Scrub brush is intermixed with up to 60% outcrop in the higher elevations of the property while larger trees tend to be restricted to the valleys; the steep cliffs are virtually 100% outcrop. Overburden is less than one to five meters deep and it consists of various types of glacial tills.

The local climate for Grey River is temperate maritime (typical of the south coast of the island of Newfoundland). In general, the summers are mild although there are often days of thick fog that tend to moderate the temperature (highs of only 16°C are typical). The winters are cold but not as severe as mainland Canada with temperatures typically around the freezing mark (annual minimum temperature of -5.9°C). In contrast to the moderate temperatures the annual precipitation averages 1310 mm and this tends to fall between the months of July and November.

6.0 HISTORY

This section is taken directly from two reports supplied by Playfair (Dearin and Harris, 2006; Dadson, 2007).

The first mineral exploration in the Grey River area was carried out by the Buchans Mining Company Ltd. in 1955. Subsequent exploration by ASARCO (between 1957 and 1970) included: surface geological mapping, trenching and diamond drilling on five veins. In addition, an exploration adit was driven by ASARCO along the Number10 Vein which permitted the development of 20 raises and the collection of a 275 ton bulk sample for metallurgical tests by both ASARCO and CANMET. During these programs the only element of interest was tungsten.

ASARCO planned to produce tungsten from the adit in 1970 but this was postponed due to a drop in world tungsten prices. After 1970, the property changed hands several times but no further work was done. The claims expired in June 2000 and were map staked by South Coast Ventures after the Newfoundland government released the ground for staking.

Summarized below (Table 6-1) is a brief history of geological and exploration work carried out since 1955.

Table 6-1: Work History

Pre-1955	■ Tungsten mineralization was apparently discovered by a Mr. Rose of Grey River some years previously and was submitted to the Buchans Mining Company Ltd. in 1954-55 for analysis.
1955	■ July to October, a six man party carried out reconnaissance mapping and prospecting immediately north of Grey River and located numerous quartz-tungsten veins cutting granite gneiss. Trenching and sampling along the two more significant veins (Vein 10 and Vein 6) was carried out.
1956	■ June to October, a 16 man crew carried out mapping, prospecting, plane-table surveying, trenching and detailed trench sampling on Veins 10 and 6. This work formed the basis for future programs.
1957	■ A 25 man crew carried out a program of detailed mapping, trenching and sampling and defined the extent and grades of Veins 10 and 6. Twelve EX core holes (5913 feet) were drilled along Vein 10. Eleven of these holes in Vein 10 intersected 'ore grade' WO_3 values. A few parallel veins carry WO_3 values.
1958 – 1964	■ No work done on tungsten veins (Holes GR-17 to GR-19 were molybdenum exploration holes drilled away from the known veins).
1965	■ Seven EX core holes (GR-20 to GR-26 totalling 3078 feet) were drilled at the northern extremity of Vein 10 and 6 supposedly to intersect the veins near sea level in preparation for the proposed adit development.
1966	■ Two EX core holes (GR-27 and GR-28 totalling 1381 feet) were drilled on the northern section of Vein 10. Vein 10 extended an additional 1,200 feet to the north where it pinches out about 5,300 feet from the adit portal. WO_3 values appear to die out at the northern limit of the vein. ■ March 30: government authorization is approved for the driving of an adit. Development expenditure of $450,000 is approved for this work. ■ May: temporary camp set up. Topographic and triangulation survey network setup. ■ October: Bunkhouse and mess hall built in Grey River. A 38-foot long timber portal and 54 feet of adit is advanced by December 15.

1967	■ January 30 to December. 20: the adit is advanced by 1,292 feet. Seven underground core test holes (TH-1 to TH-7 totalling 582 feet) are drilled. ■ Generator building, repair shop, dry, dumping trestle, magazine and cap house completed. Cribbed wharf is started at the adit site. A mine lease application of 6.61 miles is applied for and boundaries are surveyed. ■ The Continental Ore Corp assessed the silica unit in the Gulch Cove area where silica values range from 96.98 to 99.21%.
1968	■ January 4 to December 16: the adit is advanced by 1,973 feet. ■ The adit wharf, compressor house and 240 feet of timbered dumping trestle were built. Six underground test holes (TH-8 to TH-13) are drilled horizontally. Nine adit core holes (GR-29 to GR-37) are drilled horizontally in the adit.
1969	■ Jan 4 to August 20: the adit is advanced 1,952 feet for a total adit length of 5,271 feet. The adit stopped as the tungsten-rich vein died out into a parallel fault zone. ■ Four underground EX core holes (GR-38 to GR-41 totalling 1,132 feet) were drilled from the crosscuts downward to test the extent of Vein 10 below the adit (results unknown). ■ Prior to May 15, eight raises averaging 27 feet high were driven for bulk sampling purposes along 820 feet of the southern part of Vein 10 (Section Lines 7950N to 8700N). Results ranged from 0.82% to 1.30% WO3% with an average value of 1.07% WO3. ■ Seven underground EX core holes (GR-42 to GR-48 totalling 680 feet) drilled. All collared in the adit face around Section line 8960N and they were drilled to locate the vein in advance of the adit. One section of Vein 10 was sampled twice by back-channel samples and once by face chip sampling. Results were comparable. Vein 10 was also re-sampled on surface in places by 12' by 12' channels (locations and results are unknown). ■ Seven surface EX drill holes (GR-49 to GR-55 totalling 1,643 feet) tested for tungsten in a series of parallel structures west of Veins 10 and 6.
1970	■ January to August: no exploration or development work is carried out. ■ September 5 to October 6: 25 six foot long raises were cut at 50 foot intervals along the vein. All broken rock, totalling 274.5 tons was carefully collected and shipped to the Mines Branch metallurgical Laboratory in Ottawa for detailed pilot plant studies.
1971	■ Pilot plant test work is completed at the Ottawa lab.
1976	■ Newfoundland Department of Mines and Energy assessed 12 million short tons of the silica unit by drilling. An average grade of 95.5% SiO_2 and 1.9% Al_2O_3 was quoted.
1979	■ September 10-13: channel sampling along the walls of four crosscuts ranging from 89 to 99 feet long. Some 81 channels each 5 feet long checked for low level tungsten values adjacent and away from Vein 10. Most samples are low but crosscut 8000N had one five foot assay of 1.4% WO_3 while crosscut 8400N had an assay of 0.40%. No follow-up work has been done. ■ A low grade resource of 25 million tons grading 0.1 to 0.2% WO_3 was postulated from this work and mapping in the southern end of the adit. **This historic resource pre-dates NI 43-101 and should not be relied upon**.
1985 – 1986	■ BP-Selco exploring for gold, locate values >1 g/t Au, with high Bi and Sb in the "quartz vein-silica body" on the eastern claims.

1995 – 1996	■ Several Grey River prospectors located base metal rich quartz veins with anomalous precious metals, moderate to high base metals but low tungsten values. This first independent-type exploration indicated the existence of a separate phase of veining with significant Au and Ag values.
1996 – 1997	■ Copper Hill Resources and Pearl Resources Ltd. of St. John's, Newfoundland option the prospector's claims and sample a number of newly discovered quartz veins. A number of grab samples on the current claims return high Au, Ag +/- Cu, Pb, Zn plus anomalous Bi. Copper Hill carries out an airborne EM and magnetic survey over a large area including the current claims area.
2003 – 2004	■ The claims expire due to a lack of funding. South Coast Ventures immediately stakes the current claims covering the high-grade Au-Ag rock samples. South Coast Ventures completes the first digital compilation of the 1960's Asarco work, the BP work and the 1996-97 rock sampling results.
2004	■ The property was sold to Playfair Mining Ltd. in 2004. During 2003-05 Fortis GeoServices Inc. compiled the 1986 to 2002 assessment work listed above and added it to the earlier digital compilation of work on the tungsten veins.
2006	■ Playfair Mining Ltd. completes 16 drill holes on the Number 10 and 6 veins to confirm grades and fill-in previously widely spaced drilling.
2007	■ Wardrop Engineering completes NI 43-101 inferred resource calculation on Number 10 Vein (852,000 tonnes of 0.858% WO3 @ 0.2% cut off) for Playfair Mining Ltd.
2008	■ Playfair Mining Ltd. completes 10 drill holes on the Number 10 and 6 veins to delineate additional tonnage and confirm Number 6 Vein. Golder & Assoc. completed a preliminary economic assessment on Number 10 Vein

6.1 Historical Resource Estimate

ASARCO estimated in 1970 a proven and probable "reserve" in one vein (the Number 10 Vein) using data from surface trenching and drilling as well as underground drifting, raising and bulk sampling. These figures are for the volume of rock between the adit level (40 feet ASL) and surface. Historical literature indicated a mineable, diluted "reserves" of 473,000 short? Tons grading 0.97% WO_3.

Golder and Playfair do not treat the "reserve" discussed in this section as current mineral resources or reserves. These estimates are historical in nature and are non-compliant with NI 43-101. They are discussed here purely for a record. These estimates are no longer relevant as they are being replaced by the NI 43-101 Mineral Resource estimate presented in this report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The project area is underlain by the Silurian-Devonian Burgeo Intrusive Suite and an east – west trending belt of Precambrian metamorphic rocks referred to as the Grey River Enclave. The contact between the intrusion in the north and the metamorphic rocks in the south is marked by a mylonitic shear zone. The Grey River Enclave typically consists of amphibolites, quartz-mica schists, pelites and gneisses. The schists and gneisses are believed to be derived from quartzites, sandstones, felsic tuffs and gabbro (relicts of these rock types are locally observed). Any bedding, along with the metamorphic foliation/banding, generally strikes E-W and dips steeply to the north. Minor post-tectonic ultramafic or mafic plugs and dikes intrude the metasedimentary rocks.

The Devonian Francois Granite intrudes the Enclave to the east of the property. Pegmatites cut all rock types and can be locally abundant. Three prominent fault sets have been documented: an E-W set is the most visible and it brings metasedimentary rocks into contact (which is typically mylonitic) with the granitic rocks. Quartz veins hosting the tungsten mineralization commonly occupy a younger north to northeast trending fault set. Figure 7.1 is a recent geological compilation showing the mineralized veins occurring directly north of Grey River within the boundaries of the old ASARCO surface grid (the claim outline on this map is out of date).

7.1 General Geology and Structure

The following description is modified from a report written by Dearin and Harris (2006):

"The area is divided into two main zones, metamorphosed sediments in the south and granites in the north. The sediments, which have been subjected to both regional and local metamorphism, strike east-west and dip steeply to the north. They represent a transition zone grading from high quartz members at the top to the more argillaceous members at the base. The upper members consist of quartzites, grits, greywackes, hornfels, slates and narrow limestone bands. The lower zone makes up the bulk of the formation and is composed of quartz-mica schists and hornblende gneisses. Cutting these sediments are several small ultrabasic plugs, narrow basic dykes and a great number of aplitic, pegmatitic and granitic dykes. Along the south margin of the sediments the granitic dykes and pegmatites constitute over 50% of the exposed outcrops. The granite bordering the sediments to the north is a coarse-grained pink variety with a low mafic content. The contact zone is highly contaminated with partially digested sedimentary remnants."

The metamorphic package consists of a unit of felsic tuff, (quartz–sericite schist) to the north and interlayered pelitic sediments and quartzites to the south. Amphibolite schist and meta-gabbro are evident locally, especially to the southwest. A 10 m to 400 m wide siliceous unit trends through the property from about 2.5 km west of Gulch Cove to the east end. This unit has previously been mapped as quartzite and/or quartz vein. Granular quartzite is evident locally but the unit is mainly fine-grained banded quartz with some white mica and >1% magnetite. Shearing is common and sheared pelite and mafic dyke occurs between silica 'lenses'. The unit likely represents a sheared quartzite but some hydrothermal silicification and/or quartz veining may be present.

The most prominent structural feature of the Grey River area is faulting. It occurs in the metamorphic and igneous rocks and is characterised by both normal and reverse senses of movement. The faults in the metamorphic rocks can be grouped into two main sets: an east-west set parallel to the schistosity and a south-east set cross-cutting the schistosity. A third set occurs only in the granites. Arising from this set of faults is a prominent fissure system of tensional origin striking north to northeast. These tension fissures act as the structural control for the tungsten veins. In general there is an absence of major movement along these fissures.

7.2 Mineralization – Tungsten Veins

The Grey River tungsten veins are typical fluorite-rich, wolframite-quartz greisen vein deposits. Wolframite is the dominant tungsten-bearing mineral although a number of small scheelite occurrences are known.

The quartz-wolframite veins cross-cut the metamorphic rocks but are also exposed within the granitic rocks to the north. Over 300 veins and lenses have been mapped on surface though only two or three have been aggressively evaluated. One of these, the Number 10 Vein, varies in width from 0.9 m to over 4.3 m, with average widths around 1.2 m (based on underground mapping). The Number 10 Vein has a strike length of at least 1600 m with the known mineralized shoot having a length of around 775 m. The vein is connected to the exposed mineralized veins on the surface (giving a minimum 240 m down-dip length) and it appears to increase in width with depth.

Higgins & Swanson (1956) give a more detailed summary on the mineralization based on their mapping and detailed observations of the mineralized veins exposed in trenches:

"Tungsten bearing veins of economic interest occur in the area shaded in red as shown on plan No. 2150. In this area several hundred veins have been found of which 300 were mapped and 300 others examined. The bulk of these veins are small lenses 40 to 50 feet in length and from one to two inches in width. Nine veins, two feet or more in width were stripped and sampled and of these only numbers 6 and 10 appear to be economically significant."

"The narrow quartz veins tend to hold a uniform thickness throughout their length while wide veins are characterized by quite irregular widths. The vein walls are sharp with a band of phlogopite mica separating the veins from the country rock."

"Fluorite is the most abundant non-metallic mineral (other than quartz) in the veins and may, in some cases run as high as one percent. Other non-metallic gangue minerals noted are apatite, beryl, scapolite, orthoclase, albite, muscovite and vesuvianite. Pyrite is the most abundant sulphide and, in the major veins, may account for over one percent. Chalcopyrite occurs sporadically in the wider veins but overall they will average less than 0.1% copper. Other sulphides noted were stibnite, molybdenite, arsenopyrite, sphalerite, galena and bismuthinite."

"Wolframite (WO_3) is the only important mineral in the veins of the Grey River area. The variety is manganese-rich with the ratio of MnO to FeO, in one sampled analyzed, being 15 to 9 (Note: this would be a huebnerite type from the wolframite mineral series $(Fe,Mn)WO_4$ ranging from $FeWO_4$ (ferberite) to $MnWO_4$ (huebnerite). The wolframite crystals are coarse grained and occur as irregular masses, well-defined monoclinic crystals or in radiating groups of bladed crystals. Scheelite is present but only in small quantities. It often replaces wolframite along the crystal surfaces and cleavage planes. Secondary minerals are fairly common on the exposed surfaces of the veins; limonite from the alteration of pyrite, tungstite secondary after scheelite, powellite after molybdenite and manganese hydroxides."

"Early in the field season a zonal arrangement of the mineralization was apparent; particularly the wolframite-molybdenite distribution. After several hundred veins had been examined the distribution of the wolframite, scheelite, molybdenite, chalcopyrite and galena were plotted and zonal curves calculated" (note that this data has never been updated and the various mineral distributions [tungsten, molybdenum, chalcopyrite in addition to relatively newly discovered gold mineralization] are now known to occur at significant distances from this 1956-era plot). “Pyrite, which is the most abundant metallic mineral, occurs everywhere and therefore has not been included in the zoning. It can be seen from the sketch that clear-cut zoning based on the temperature of formation of different minerals is not well defined as individual distribution curves cross each other. However, it appears that the high temperature mineralization decreases away from the centre of the mineralized area taken to be just west of vein number 10. The zonal arrangement also suggests that the mineralization is not directly related to the northern granite but to a source directly below the mineralized area.”

7.3 Number 10 Vein

“This is by far the most important vein found in the area. It occurs in a three thousand foot long fissure and has been exposed by intermittent trenches for approximately 2,000 feet. One hundred and sixteen channel samples were taken from the vein on the exposed sections between coordinates N593 & N1920”.

7.4 Number 6 Vein

“This vein lies two thousand feet northeast of vein number 10. Two sections of the vein were stripped; a 50 foot section and a 125 foot section separated by a gap of forty feet.”

7.5 Other Gold and Silver Rich Veins

During 1956 ASARCO located a quartz vein with high gold values (although no tungsten) in the Dog Cove Brook-Beaver Brook vicinity approximately 3.5 km north of Grey River (Bahyrycz, 1956). This showing, referred to as the Galena Vein Number 1, occurs in a shear zone cutting granitic rocks. Channel sampling of the vein returned values of up to 2.90 ounces per tonne gold (oz/t Au), 4.2 ounces per tonne silver (oz/t Ag), and averages of less than 0.5% copper (Cu), 15% lead (Pb) and 3% zinc (Zn) over a vein width of 2’ 2”. Later re-sampling of this vein by ASARCO-Abitibi Price returned gold values of 0.74 oz/t Au.

A graduate thesis by Gray (1958) noted occurrences of galena mineralization (with significant amounts of silver, gold and bismuth) in quartz veins cutting granitic rocks immediately east of Long Pond. No further exploration work was ever reported in this area. During 1995-97 and 2001 local prospectors located a number of high-grade sulphide-rich quartz veins, with assays exceeding 30 grams per tonne gold (g /t Au), cutting intrusive rocks immediately north of Long Pond.

Between 1995 and 1997 Grey River prospectors located sulphide-rich (10 to 15%) quartz veins west and south of Grey River. Precious metal values exceeded 9 to 21 g /t Au and 200 to 332 grams per tonne silver (g/t Ag) with high bismuth (greater than 440 parts per million) and anomalous to high base metal values (Jacobs, 1997). The following is modified from Jacobs (1997) who summarized the *rock sampling* results on and adjacent to the property as follows:

*“*Assays for gold showed slightly anomalous to highly anomalous results, including ten samples in the range of 17 parts per billion (ppb) to 251 ppb Au, two samples between 541 ppb (GR-2) and 755 ppb Au (GR-29) and four samples with values of 1,530 ppb (GR-9), 9,008 ppb (GR-26), 13,280 ppb (GR-24) and 21,355 ppb Au (GR-27). All anomalous gold values showed a general correlation with either of the base metals (Cu, Pb, Zn) and/or Ag; the best values, however, corresponded with the higher Pb and Ag values.*”*

Jacobs cautions that conclusions drawn on sample results, regarding maximum values and element correlations, are premature, as complete assay determinations have not yet been made for many samples. As well, assay correlations, in this sense, have only limited value due to the fact that most samples are taken from veins where mineralization is often inconsistent and of a generally localized nature.

The Grey River Tungsten Property contains multiple tungsten-rich quartz veins within undeformed, linear fractures. These fractures cross-cut the local metasedimentary and metavolcanic rocks and they appear to be spatially (but not necessarily genetically) associated with the northern granitic suite. All of the tungsten-carrying veins are oriented north-northeast. The better known mineralization is restricted to two veins called the Number 6 and Number 10 Veins.

Wolframite and scheelite are the dominant tungsten-bearing minerals in the veins although scheelite is better developed in the northern sections of the property where limey units are more common. Typically, wolframite crystals occur as coarse-grained, steel grey to black coloured clusters and disseminations within white-coloured quartz veining. Pyrite, pyrrhotite, chalcopyrite, bismuthinite, molybdenite, galena and fluorite may also be present. Sericitic alteration of wallrock is common on the hanging wall side of the Number 10 Vein and country rock inclusions have also been documented.

To date, the genetic model for the tungsten veins at Grey River is poorly understood. Although the deposits are in discrete veins, and appear to be spatially associated with the northern granitoids, there lacks conclusive evidence that the veins are linked to the exposed granite.

8.0 DEPOSIT TYPE

The Grey River Tungsten Property contains multiple tungsten-rich quartz veins within undeformed, linear fractures. These fractures cross-cut the local metasedimentary and metavolcanic rocks and they appear to be spatially (but not necessarily genetically) associated with the northern granitic suite. All of the tungsten-carrying veins are oriented north-northeast. The better known mineralization is restricted to two veins called the Number 6 and Number 10 Veins. Figure 8.1 shows the Grey River regional geology compilation.

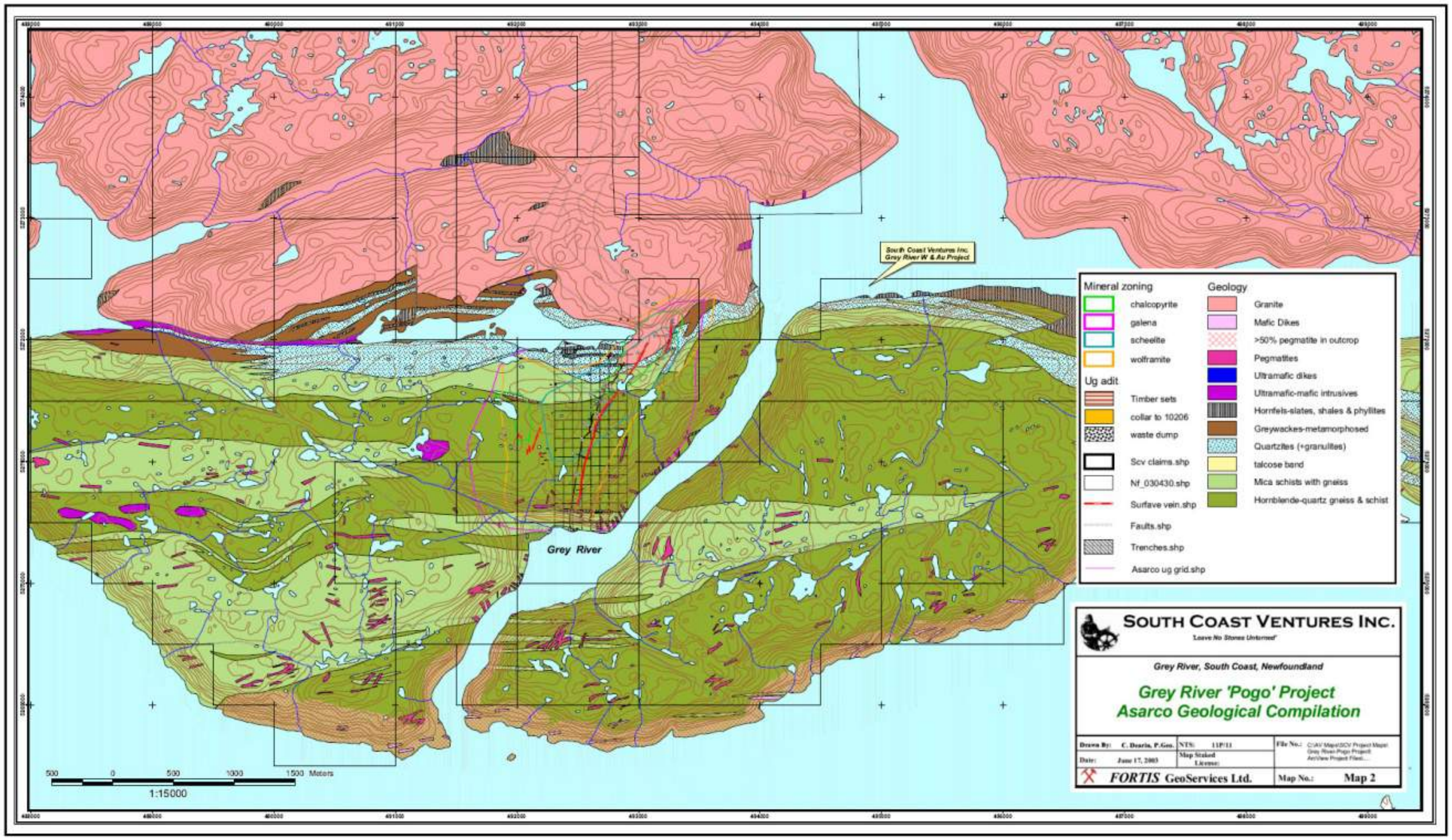

Figure 8.1: Grey River Regional Geology Deposit Types

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To date, the genetic model for the tungsten veins at Grey River is poorly understood. Although the deposits are in discrete veins, and appear to be spatially associated with the northern granitoids, there lacks conclusive evidence that the veins are linked to the exposed granite.

9.0 EXPLORATION

9.1 Summer 2006

In 2006, Playfair completed 16 HQ size holes (including one wedge hole). Of those, twelve holes targeted the Number 10 Vein and four holes targeted the Number 6 Vein. The programs main goal was to confirm the grade seen in the historical drill holes. In addition, Playfair re-sampled the ASARCO trenches on the Number 10 Vein (119 samples taken).

9.1.1 Adit

A brief inspection (by Playfair personnel) of the adit on the Number 10 Vein was carried out during the 2006 drill program and re-inspected during the 2011 site visit. Figure 9.1 shows a selection of photographs of the adit taken during the most recent site visit by the author.

A generator powering an underground fan is still in place. Playfair replaced the galvanized metal ventilation ducking with modern yellow polyethylene vent tubing that is now in need of minor repair. The adit is in surprisingly good condition with only minor falls of loose material over most of its length with exception to the fault area where a significant amount of loose material as accumulated on the drift floor. During the site visit, it was deemed unsafe to proceed through this area without rehabilitation. ASARCO did not routinely rock bolt the back of the drift but only supported selected areas as needed. The adit dimension is approximately 2.4 m by 2.4 m in size with a narrow gauge rail that appears in good condition.

The adit can be rehabilitated with some minor scaling, rock bolting and clean up. About one dozen rail cars, a cache of sample drums and galvanized ducting remains in the workings.

Adit entrance showing ventilation fan

Access drift showing old rail cars

Narrow section of the No 10 vein showing ASARCO sample location

Bulk sample raise in No 10 Vein

Figure 9.1: Underground adit photographs

9.2 Summer 2008

During the summer of 2008, Playfair completed an additional 10 NQ size holes targeting the Number 10 Vein below the adit level.

Playfair also commissioned Yates and Woods Limited (Yates and Woods) of Cornerbrook, Newfoundland to complete an accurate GPS survey of the Grey River Property. Yates and Woods conducted the survey using a Topcon GB500 Base station and a Topcon Hyper Plus Rover. The initial static survey was carried out on July 13, 2008, placing the Topcon GB 500 on point No. 2000 (on site). Yates and Woods surveyor traveled to Ramea and placed the Topcon Hyper plus on Monument No. 89G6164 and recorded at five second intervals for 37 min 15 sec. The survey team then traveled to Monument No. 89G6162 and recorded at five second intervals for 39 min 55 sec. The information collected was post processed later in the Topcon Tools Version 7.1.

All other points were surveyed using Topcon BG500 on point No. 2000 and standard RTK surveying practices with Hyper Plus Rover on July 14 and July 19, 2008.

The historical drill holes were originally surveyed in a local mine grid coordinate system. The new detailed survey allowed the correct placement of the historical holes in the NAD 27, Zone 21 coordinate system.

This work also allowed the geo-referencing and digitizing of a historical topographic map surveyed by Buchans Mining Co. Ltd. and dated September 14, 1957 (Dwg No. 2251 scale 1"-200') by Alicia Korpach, GIS specialist. With the digital map, 29 additional historical holes could now be added to the database, including 12 underground holes that were drilled by ASARCO in 1968 and 1969.

10.0 DRILLING

10.1 Pre-1970

The drill campaign in the Grey River area was carried out by the Buchans Mining Company Ltd. in 1955, which was followed by ASARCO between 1957 and 1970. With the exception of 13 test holes drilled from underground (TH1 to TH13), a total of 55 holes were recovered from historical logs. Holes during that period were drilled using coring equipment with no retrievable inner core barrel. Mostly EX (22.2 mm core) in size were used for short holes or a combination of AX (28.5 mm core) followed by EX for the longer holes.

Core recovery data is available on the logs. Considering the size of the holes and the equipment used at the time drilling was conducted, the core recovery appears to be moderate to good (> 75%) for the holes that were reviewed and is consistent with drilling in competent rocks.

Although not indicated on all logs, the down survey instrument used on longer holes appears to be a Tropari (from Pajari instruments Ltd.), which measure the azimuth and dip of the holes with a magnetic compass.

As shown in Table 10-1, thirteen historical holes were located in the field by Yates and Wood during the summer 2008 program. The remaining UTM coordinates for the holes were derived from the geo-referenced 1957 historical map or from a geo reference map of the ASARCO underground adit.

Table 10-1: Pre 1970 Diamond Drill Hole Summary for Grey River Pre 1970

Hole Number	UTM Coordinate		Elevation (m.)	Azimuth/Dip	Length (m.)	Survey
	East	North				
DDH-?	492695.0	5271813.0	300.0	0 / -90	100.0	Map
DDH3?-45d	492693.6	5271815.1	244.4	0 / -90	100.0	Y and W
GR-1	492792.1	5271592.6	236.0	120 / -60	143.6	Map
GR-2	492605.8	5271292.3	270.7	120 / -57	136.3	Map
GR-3	492699.8	5271383.3	258.0	120 / -57	30.2	Map
GR-4	492763.5	5271482.8	263.2	120 / -56	18.3	Map
GR-5	492645.2	5271269.2	265.4	120 / -57	23.2	Map
GR-6	492733.8	5271500.1	262.5	120 / -57	107.0	Map
GR-7	492687.9	5271362.8	258.3	120 / -70	23.2	Map
GR-8	492668.1	5271400.8	262.0	120 / -58	96.9	Map
GR-9	492575.5	5271309.4	276.0	120 / -62.5	343.6	Map
GR-10	492628.3	5271559.5	262.5	120 / -60	354.2	Map
GR-11	492222.3	5271048.3	248.0	55 / -64	19.5	Map
GR-12	492006.8	5271099.5	276.0	90 / -45	26.8	Map
GR-13	492411.5	5271026.9	300.0	90 / -45	12.8	Map
GR-14	492493.0	5271069.6	260.0	110 / -45	190.5	Map
GR-15	493214.1	5271858.7	231.0	292 / -37	23.8	Map
GR-16	492748.3	5271619.0	237.7	120 / -79	335.6	Y and W
GR-17	491962.0	5270710.5	300.0	45 / -30	30.5	Map
GR-18	491923.4	5271201.6	300.0	45 / -40	30.5	Map
GR-19	491930.9	5271258.9	300.0	300 / -30	27.4	Map
GR-20	493250.7	5272034.5	242.6	83 / -60	53.9	Y and W
GR-21	493249.9	5272034.4	242.6	0 / -90	89.3	Y and W
GR-22	493152.9	5272038.3	264.3	90 / -65	183.5	Y and W
GR-23	493152.6	5272038.3	264.3	88 / -83	253.3	Y and W
GR-24	493089.7	5271830.1	265.9	93 / -60	136.9	Y and W
GR-25	493009.8	5271896.2	251.5	120 / -60	74.7	Y and W
GR-26	493009.2	5271896.5	251.4	0 / -90	146.6	Y and W
GR-27	492834.6	5271746.2	257.4	121 / -46	182.9	Y and W
GR-28	492834.2	5271746.5	257.6	121 / -70	238.1	Y and W
GR-29	492494.8	5270899.5	14.0	90 / 0	51.5	Adit
GR-30	492491.0	5270899.2	14.0	40 / 0	62.5	Adit

Hole Number	UTM Coordinate		Elevation (m.)	Azimuth/Dip	Length (m.)	Survey
	East	North				
GR-31	492494.1	5270900.3	14.0	270 / 0	24.7	Adit
GR-32	492534.2	5271179.7	14.0	265.5 / 0	43.9	Adit
GR-33	492536.9	5271181.0	14.0	14 / 0	32.3	Adit
GR-34	492537.5	5271181.0	14.0	31 / 0	28.7	Adit
GR-35	492537.5	5271179.7	14.0	87 / 0	38.7	Adit
GR-36	492571.0	5271213.8	14.0	90 / 0	25.0	Adit
GR-37	492593.9	5271235.0	14.0	49 / 0	26.2	Adit
GR-38	492590.6	5271341.2	13.0	105.37 / -77	142.0	Adit
GR-39	492641.5	5271463.9	12.0	90 / -65	79.2	Adit
GR-40	492641.5	5271463.8	12.0	90 / -83	77.7	Adit
GR-41	492696.9	5271597.7	12.0	90 / -64.5	46.0	Adit
GR-42	492735.2	5271616.2	14.0	106 / 0	14.3	Adit
GR-43	492725.7	5271597.1	14.0	116 / 0	27.1	Adit
GR-44	492732.3	5271626.7	14.0	317 / 0	12.5	Adit
GR-45	492732.3	5271626.7	14.0	37 / 0	21.9	Adit
GR-46	492738.4	5271634.0	14.0	101 / 0	36.0	Adit
GR-49	492439.2	5271908.7	244.1	111 / -45	93.9	Map
GR-50	492137.4	5271878.3	244.1	115 / -45	75.9	Map
GR-51	492612.9	5271829.5	243.8	115 / -45	55.2	Map
GR-52	492612.9	5271829.5	243.8	115 / -65	61.0	Map
GR-53	492696.7	5271783.8	243.8	115 / -45	63.1	Map
GR-54	492481.4	5271529.9	291.9	115 / -45	61.0	Y and W
GR-55	492480.9	5271530.1	291.9	115 / -65	75.6	Y and W
Total Meter in Database					**4,809**	

10.2 Playfair 2006

Playfair carried out a diamond drilling program on the Grey River Tungsten property in the summer of 2006. Twelve holes, including one wedged hole, tested the Number 10 Vein while four other holes tested the Number 6 Vein to the north. Core was drilled to HQ (63.5 mm core) by Petro Drilling Co. Recovery was not indicated on the Playfair logs but appears to be good to excellent (>85%) in the holes that were examined during the site visit. Table 10-2 lists the 2006 collar location, length and drill direction. Down hole deviation surveys consisted of tropary tests typically taken at the end of the hole.

Table 10-2: 2006 Diamond Drill Hole Summary for Grey River

Hole Number	UTM Coordinate		Elevation (m.)	Azimuth/Dip	Length (m.)	Survey
	East	North				
GR-06-100	492589.2	5271366.9	271.3	120 / -50	156.2	Y and W
GR-06-101	492588.7	5271367.2	271.4	120 / -70	242.0	Y and W
GR-06-102	492643.1	5271476.9	265.6	120 / -50	125.0	Y and W
GR-06-103	492642.7	5271477.1	265.4	120 / -70	179.0	Y and W
GR-06-104	492704.0	5271565.9	249.4	120 / -52	152.0	Y and W
GR-06-105	492703.4	5271566.3	249.6	120 / -75	194.0	Y and W
GR-06-106	492795.9	5271665.5	235.2	120 / -60	179.0	Y and W
GR-06-106W	492795.9	5271665.5	235.2	120 / -60	119.0	Y and W
GR-06-107	492795.2	5271666.0	235.3	120 / -85	224.0	Y and W
GR-06-108	493157.3	5271980.5	270.2	90 / -50	153.0	Y and W
GR-06-109	493156.6	5271980.5	270.1	90 / -70	233.0	Y and W
GR-06-110	493182.1	5272079.2	254.3	90 / -51	164.0	Y and W
GR-06-111	493181.4	5272078.9	254.5	90 / -69	221.0	Y and W
GR-06-112	492533.8	5271254.8	278.5	120 / -50	196.0	Y and W
GR-06-113	492533.3	5271255.0	278.4	120 / -70	236.0	Y and W
GR-06-114	492627.3	5271415.3	263.8	120 / -64	149.0	Y and W
Total Meters in Database					**2,922 meters**	

10.2.1 Number 10 Vein

Drill holes GR-06-100 to 107 and GR-06-112 to GR-06-114 tested the tungsten mineralization in Vein 10. These holes were planned to intersect the vein at approximately 100 m and 200 m below surface in a position approximately half way between the sections drill-tested by ASARCO. The Number 10 Vein structure was intersected in all holes. From the drill logs the vein widths along the core varied from 0.5 m to 4.8 m and WO_3 assays varied from a low of 0.0003% (3 ppm) over 0.5 m to a high of 1.70% over 1.5 m.

10.2.2 Number 6 Vein

Holes GR-06-109 to 111 tested the down dip portion of the surface mineralization exposed in the Number 6 Vein area. Previous EX sized drilling on the Number 6 Vein returned lower grade results than the trenches. This may be due to low core recoveries (grinding of the core is common with standard drilling). Although alteration and veining was intersected in all four holes the results were disappointing with a high value of 0.40% WO_3 over 0.4 m.

10.3 Playfair 2008

Playfair continued diamond drilling on the Grey River Tungsten property in the summer of 2008. A total of 10 NQ (27 mm Core) holes were drilled on the property targeting the number 10 vein below the adit elevation north of coordinate 5,271,500N. Drilling was conducted by Cabo Drilling and the drill hole summary is presented in Table 10-3.

Table 10-3: 2008 Diamond Drill Hole Summary for Grey River

Hole Number	UTM Coordinate		Elevation (m.)	Azimuth/Dip	Length (m.)	Survey
	East	North				
GR08-115	492763.4	5271797.8	243.2	121.5 / -75	359.0	Y and W
GR08-116	492763.1	5271798.0	243.2	121.5 / -84	422.0	Y and W
GR08-117	492763.5	5271796.8	243.2	164 / -69	371.8	Y and W
GR08-118	492763.5	5271796.9	243.0	171 / -74	488.0	Y and W
GR08-119	492584.0	5271638.6	267.2	121.5 / -64	396.0	Y and W
GR08-120	492583.7	5271638.9	267.3	121.5 / -75	521.0	Y and W
GR08-121	492584.0	5271639.2	267.4	87 / -71	413.0	Y and W
GR08-122	492861.4	5271924.0	240.3	171 / -76	344.0	Y and W
GR08-123	492861.4	5271924.0	240.4	102 / -80	353.0	Y and W
GR08-124	492843.0	5271931.0	241.0	121.5 / -70	287.0	Garmin GPS
Total Meter in Database					**3,955 meters**	

10.3.1 Number 10 Vein

Hole GR08-115 intersected the Number 10 Vein from 296.4 m to 299.0 m. The intersection graded 0.13% W from 297.0 m to 299.0 meter. In the hanging wall of Vein Number 10, a low angle quartz vein was intersected with pyrite, chalcopyrite, fluorite and minor wolframite. The 2.4 meter intersection graded 0.3% W between 277.2 m and 279.6 m.

Drillhole GR08-116 intersected Number 10 Vein from 394.8 m to 399.9 m exhibiting approximately 1% Wolframite with minor scheelite, pyrite and chalcopyrite. The best result occurred in a 1.1 meter section starting at 397.9 grading 0.235% W.

In hole GR08-117 the Number 10 Vein graded 0.613% W in a 1-meter intersection starting at 302.8 m down the hole.

Hole GR08-118 possibly intersected Number 10 Vein between 407.1 m to 407.9 m. The zone is a highly altered quartz rich interval with moderate sericite. Hole GR08-119 intersected a quartz vein between 350.1 m to 353.8 m. There was no visible wolframite in the intersection.

GR08-121 intersected Number 10 Vein between 395.8 m to 398.4 m. The intersection displayed strong sericitic alteration with common fluorite, 3-4% Pyrite, molybdenite and chalcopyrite. Minor tungsten values were encountered, all below 0.02% W.

Number 10 Vein was intersected in hole GR08-122 between 319.85 m to 325.1 m. The 5.3 m intersection graded 0.287% W which includes a 0.6 m intersection grading 0.826% W from 319.8 to 320.4 and a 1.5 m intersection grading 0.551% W starting at 323.6 m. A structure exists in the hanging wall of Number 10 Vein between 294.4 m to 297.1 m. The 2.4 m intersection graded 0.895% W with copper values in excess of 0.5% copper.

Hole GR08-124 intersected Number 10 Vein between 250.5 m to 254.2 m. The 2.5 meter intersection graded 0.382% W

Hole GR08-120, GR08-123 failed to intersect a definitive Number 10 Vein although hole GR08-123 intersected a number of low angle quartz veins with significant tungsten values between 0.15 W to 0.8% W.

10.3.2 Upper Zone

Most holes drilled during the 2008 campaign intersected a zone of mineralization near the collar of the holes. The zone, typically in schist and gabbro contain 5% pyrite or less with minor fluorite, scheelite and wolframite and anomalous copper, lead, zinc and silver values. Figure 10.1 shows the drill hole plan map for the Number 6 and Number 10 veins.

Most notable intersection are in hole GR08-115 which returned 0.16% W from 35.5 m to 46 m. including a 3 meter section grading 0.51% W between 37.0 m to 40.0 m. A high angle section in hole GR08-116 graded 0.36% W from 43.9 to 46.4m. Anomalous copper, lead zinc and silver values were intersected between 35.0 m. to 38.9 m. in hole GR08-117. The intersection included a 0.5 meter section grading 0.468% W from 37.4 m. to 37.9 m.

In hole GR08-118, a high angle quartz vein with wolframite was intersected between 22.5 m to 23.7 m grading 1.180% W. This intersection was followed by a zone of anomalous silver, copper, lead and zinc between 23.7 m to 35.8 m containing significant tungsten. Most veins carrying wolframite are low angle fractures between 15 to 20 degrees to core angle. Best values occurs between 28.3 m to 29.2 m grading 0.217% W and between 33.8 m and 35.8 m grading 0.136% W.

Hole GR08-119 intersected a zone from 32.5 m to 47.0 m and again from 57.7 m to 66.0 m with anomalous silver, copper, lead and zinc but no significant tungsten results. Similar results was obtained in hole GR08-120 from 11.2 m to 34.7 m.

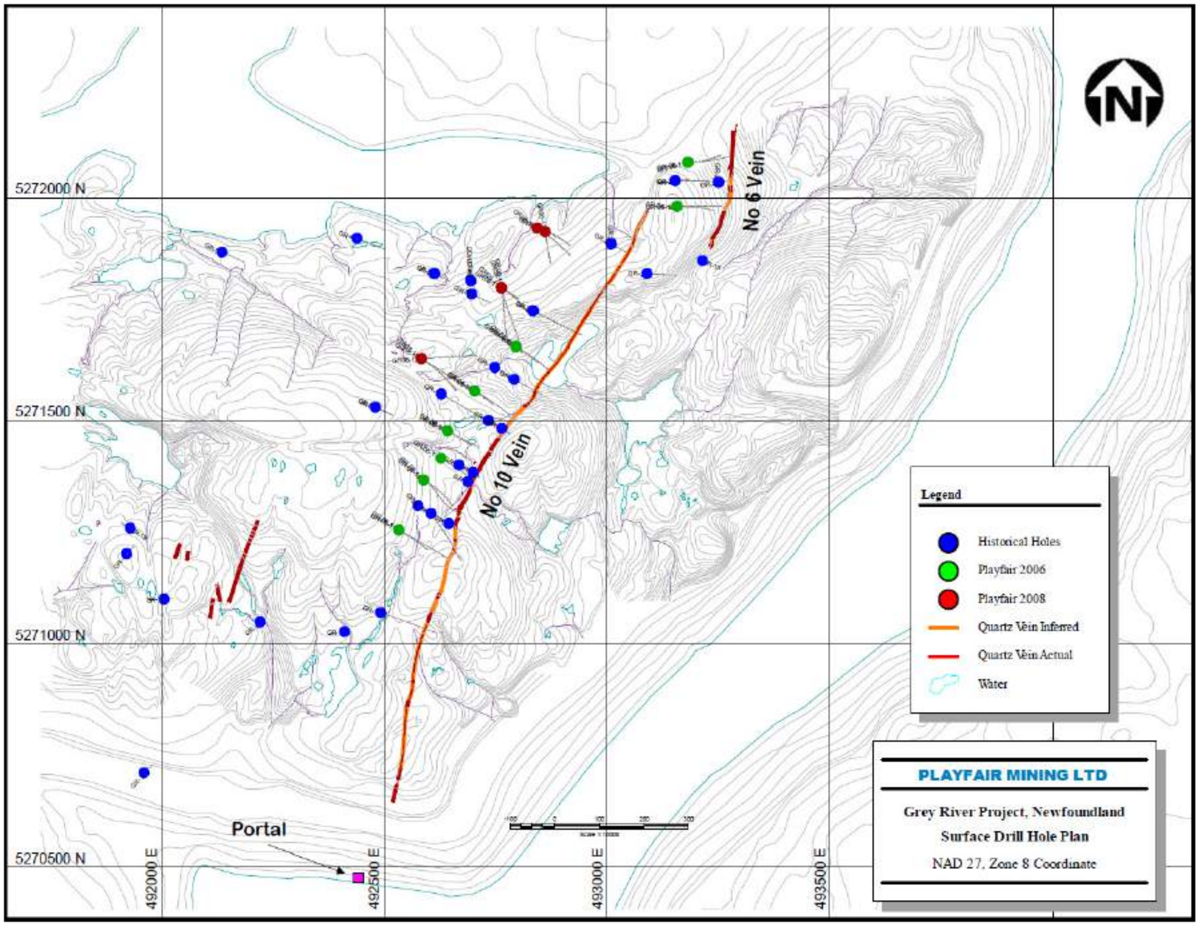


Figure 10.1: Drill hole plan map for the Number 6 and Number 10 veins

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Collection

Data from various sample types have been collected over the years at the Grey River Tungsten property:

a) ASARCO drill core samples from the pre-1970s exploration program.

b) ASARCO surface trench samples (channel samples).

c) ASARCO surface grab samples.

d) ASARCO underground face samples.

e) ASARCO underground back samples (two campaigns).

f) ASARCO raise/bulk samples.

g) Playfair drill core samples from the 2006 and 2008 drilling program.

Pre-1970 Drilling

ASARCO drilled 55 core holes on the Grey River Tungsten property, of those, 25 were drilled to intersect the Number 10 or Number 6 Vein. Most holes were drilled with an EX sized core or combination of AX followed by EX on longer holes. This drilling technique does not use wire-line equipment and is more prone to poor recovery. An examination of the preserved drill core in Buchans shows that the complete core from the mineralized sections were taken as samples. Consequently, there are no representative samples to check for any of the mineralized zones tested by the pre-1970s holes.

Trench Samples and Grab Samples

ASARCO excavated a total of 26 trenches over the Number 6 and 10 Veins. Seventeen of these trenches tested the mineralization in the Number 10 Vein. The method of sample collection and/or aggregation is unknown. Playfair collected a series of grab samples for portions of the Number 10 Vein as part of the 2006 exploration effort.

Underground Face and Back Samples

Face samples dating back to March 1969 were collected every 2 to 2.5 meter intervals, while the first campaign of back samples dated February 1970 were collected at 1 to 1.5 meter interval. ASARCO recorded the WO3%, Cu% assays along with the vein width. No details regarding the methodology used in the collection of these samples were available, however a document dated Nov. 16, 1979 describes a cross-cut sampling program and the procedure used for these samples is assumed to be similar to what was used by ASARCO. The Sampling was reportedly done with hammer and moils. The equipment was changed to a chisel-pointed bit during the cross cut sampling program due to contamination with the tungsten carbide circular point bit originally used. Samples were chipped onto a plastic sheet laying on a canvas then bagged within protective canvas bags. The exact equipment used for the back and face samples by ASARCO is unknown however during the site visit, it was apparent that the back samples were collected by hand tools and power saws were not used to cut "channels". Several campaigns of sampling exists in the database. The original data is from face samples and is dated March 1969. Another plan, dated February 5, 1970, shows face and back sample results. Two undated hand drawn historical plans showed a check back sampling program. This program was labelled as "Check channel South Plainfield". Only one assay value is indicated on the plans for the South Plainfield data and it is not known if the assay is in W% or WO_3%. Assuming the value is in WO_3%, results from this check sampling program indicated an average grade of 0.415% which is 0.349% lower than the ASARCO original back sample which graded 0.764 WO_3%. Figure 11.1 illustrates the ASARCO sampling on the right side of the drift. The data is inked with the text corresponding to the WO_3 grade and Cu grade over the vein width recorded in feet. On the left side are the hand drawn South Plainfield check sample results.

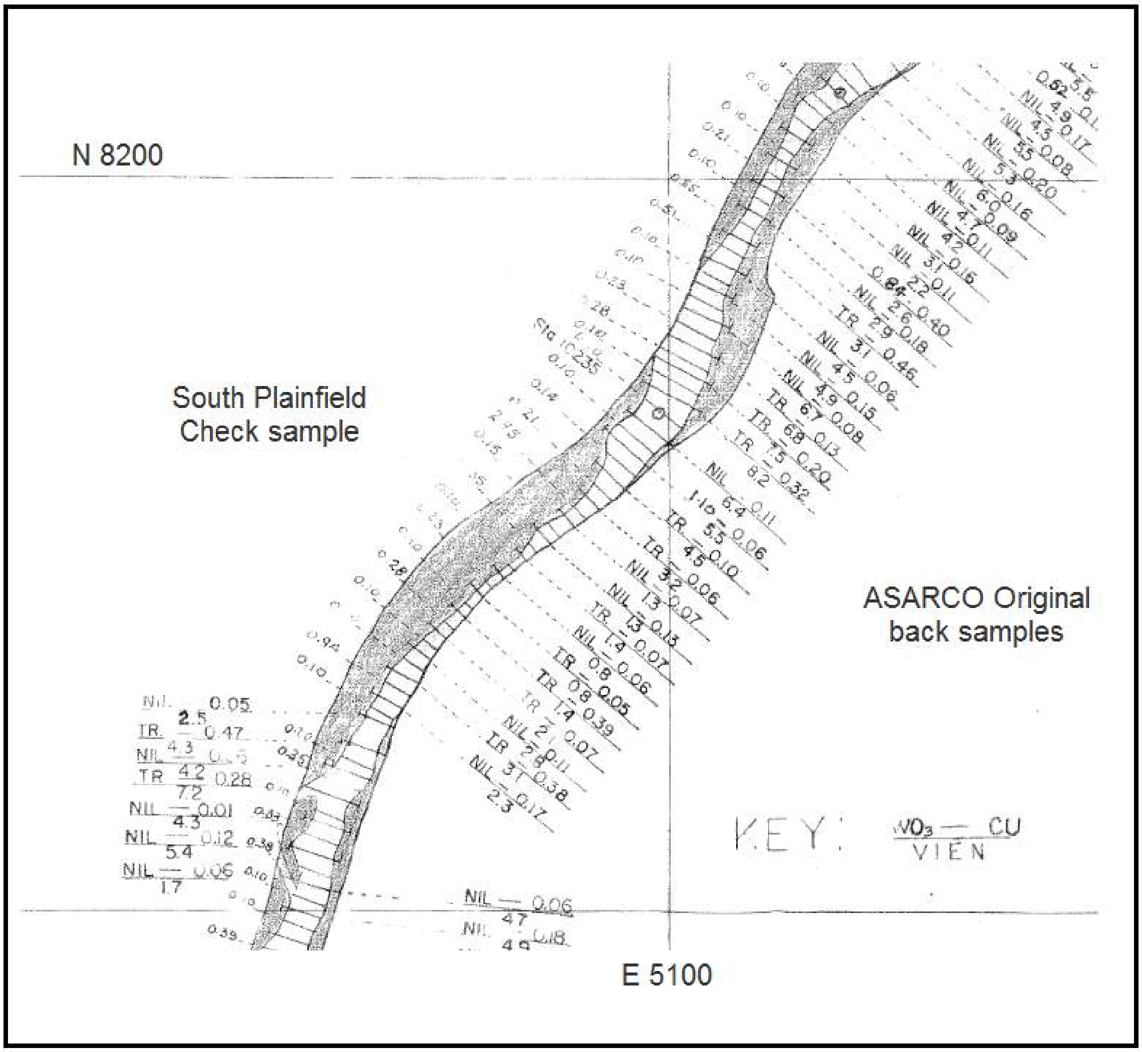
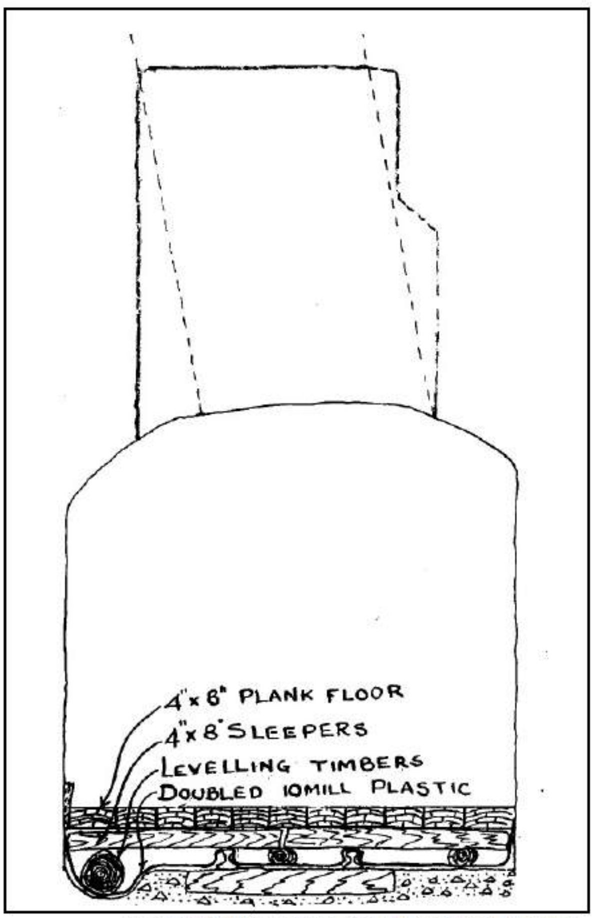


Figure 11.1: Example of a portion of the ASARCO underground back sampling program

Raise/bulk samples: Raise samples were also collected from underground (37 in total); there is no data for 25 of these samples. A document exists regarding the sampling collection of the bulk sampling program conducted by ASARCO. The document is a memorandum to Mr. E. M. Martin General Manager and is dated August 7, 1970. The procedure used for the collection of the bulk sampling program is summarized from this document as follows:

Each raise is marked and mapped in detail. The minimum width of the raise is 1.2 m x 1.5 m x 1.8 m to the width of the vein where necessary. A floor was built consisting of a 10 mil plastic sheet followed by a 10.2 cm x 20.3 cm x 2.4 m plank floor resting on 10.1 cm x 10.1 cm sleepers. Drilling was conducted along the dip of the vein then blasted. Each round was mucked into drums with the fines collected by sweeping the plank floor. Any fine material collected by the 10 mil plastic was placed in the drums. Plank and plastics were to be washed before been re-use for the next raise. Figure 11.2 shows the layout of the floor used in the collection of the samples.



From August 1970 - Asarco, Test Mill Sample memorandum

Figure 11.2: Flooring arrangement for bulk sample collection

Playfair 2006 and 2008 Drill Samples

Playfair completed a drilling program in 2006 on the Number 10 Vein that tested the vein above the adit level and replicated some of the ASARCO drill holes. All core samples were collected under the supervision of Mr. James Harris, P.Geo. of Playfair. HQ diameter core (63.5 mm) drilled by Petro Drilling Co. was descriptively logged on site, aligned, marked for sampling and then split in half, longitudinally, using a diamond saw blade.

The 2008 drill program was conducted under the supervision of Mr. James Harris P. Geo of Playfair and targeted the Number 10 Vein mineralization below the adit level. The NQ diameter core (47.6 mm) drilled by Cabot Drilling Co. was descriptively logged on site, aligned, marked for sampling and then split in half, longitudinally, using a diamond saw blade.

The remaining half of the drill core for the 2006 and 2008 drill program is stored in core boxes, cross stacked on wooden pallets in proximity to the drill pad for verification and future reference.

Playfair 2006 Bulk Sample

During the 2006 drilling program a bulk sample of approximately 4,550 kilogram (kg) was collected from the trench on the Number 10 Vein. The sample is stored on site in large tote bags for future metallurgical test work.

11.2 Sample Preparation and Analysis

Pre-2006 Drilling

The available documentation indicates that all of the samples for the pre-1970's drilling program, the trenching program and the underground sampling program, were shipped to, and analysed by, the ASARCO laboratory in Buchans, central Newfoundland. Some check samples were also assayed at an ASARCO laboratory in New Jersey (USA). A description of the method of analysis used in Buchans is given in Appendix A. The available documentation does not mention the use of blanks or Certified Reference Materials although there are a few comments on duplicate analyses. Pulps and/or sample rejects are not available for examination.

According to information supplied by Playfair, ASARCO analysed the tungsten samples using a colorimetric thiocyanate method. This procedure, explained in Appendix A, consists of fusing a sample with a sodium peroxide-sodium carbonate mixture, water leached and diluted to volume. An aliquot of the clear solution is acidified with sulfuric and hydrochloric acid. The tungsten ion is reduced with stannous chloride, potassium thiocyanate is added, and the color measured spectrophotometrically. The methodology dates back to 1943 and its stated that the main advantage of this method is the non-interference of Vanadium and the higher tolerance limit of Molybdenum (S.C Srivastava and al.). As of 1997, the spectrophotometric methods based on the thiocyanate were still popular. The method was reported to be sensitive and accurate as long as all tungsten is put in the solution.

2006 Drilling

One half of the sampled drill core was bagged, sealed and delivered to Eastern Analytical Ltd. in Springdale, Newfoundland where it was dried, crushed and pulped. Samples were crushed to -10 mesh and split using a riffle splitter to approximately 300 grams. A sample split was pulverized using a ring-mill to approximately 98% passing minus 150 mesh. The resulting pulp was then shipped to Acme Analytical Laboratories Ltd. of Vancouver, British Columbia, an ISO 9001:2000 accredited laboratory, where a 0.5 gm split was subjected to a phosphoric acid leach followed by tungsten analysis of the leachate by ICP-ES (Induced Couple Plasma Emission Spectroscopy). Any values higher than 100 ppm were assayed for tungsten. All coarse rejects are currently stored at Eastern Analytical Ltd. facilities and sample pulps are currently stored at the Acme Analytical Laboratories Ltd. (Acme) facilities in Vancouver.

Blanks, certified reference materials or field duplicates were not inserted into the sample stream therefore there is no independent way to monitor any quality control issues for the 2006 drilling program. Nevertheless, new pulps of the drill hole samples were created from the Acme coarse rejects and re-analyzed by SGS laboratories. A review of the data from the two laboratories shows that 7% (19 out of 285) of the samples have significantly different values. Wardrop attributed the difference to the nugget effect however; it may also be related to the analytical method used by Acme and SGS laboratories. Only the Acme dataset was used for the current resource estimate.

2008 Drilling

Core samples were collected under the supervision of Mr. James Harris, P.Geo. NQ diameter core was descriptively logged on site, aligned, marked for sampling and then split in half, longitudinally, using a diamond saw blade. One-half of the core is preserved on site in core boxes for verification and future reference.

The samples comprising the other half of the core were bagged, sealed and delivered to Eastern Analytical Ltd. in Springdale Newfoundland where they were dried, crushed and pulped. Samples were crushed to approximately -10 mesh and split using a riffle splitter to approximately 300 grams. The sample split was pulverized using a ring mill to approximately 98% minus 150 mesh.

The resulting pulp was then shipped to Eco Tech (Alex Stewart Geochemical) of Kamloops BC, an ISO 9001:2000 accredited laboratory, where a 0.2 gm split was subjected to a fusion digestion, then analyzed for tungsten by ICP-MS (Inductively coupled plasma mass spectrometry). All coarse rejects are currently stored at Playfair Newfoundland facilities and sample pulps have been discarded.

Blanks were not inserted into the sample stream so cross contamination cannot be independently monitored between sample batches. No field duplicates were inserted in the sampling chain however, new to the 2008 sampling program, Playfair introduce the use of standard reference material. The CDN-W-1 standard was purchased from CDN Laboratory Ltd. of Delta, BC. The recommended values and the "Between Lab" two-standard deviation are indicated on the certificate as 1.04 +/- 0.10% Tungsten and 0.458 +/- 0.042% Copper. According to CDN Laboratory, the material was sourced from the Cantung mine and consists primarily of pyrite, chalcopyrite and tungsten as scheelite.

The laboratory results for the standard shows excellent performance against the standard for both tungsten and copper with no failure as shown in Figure 11.3 and 11.4.

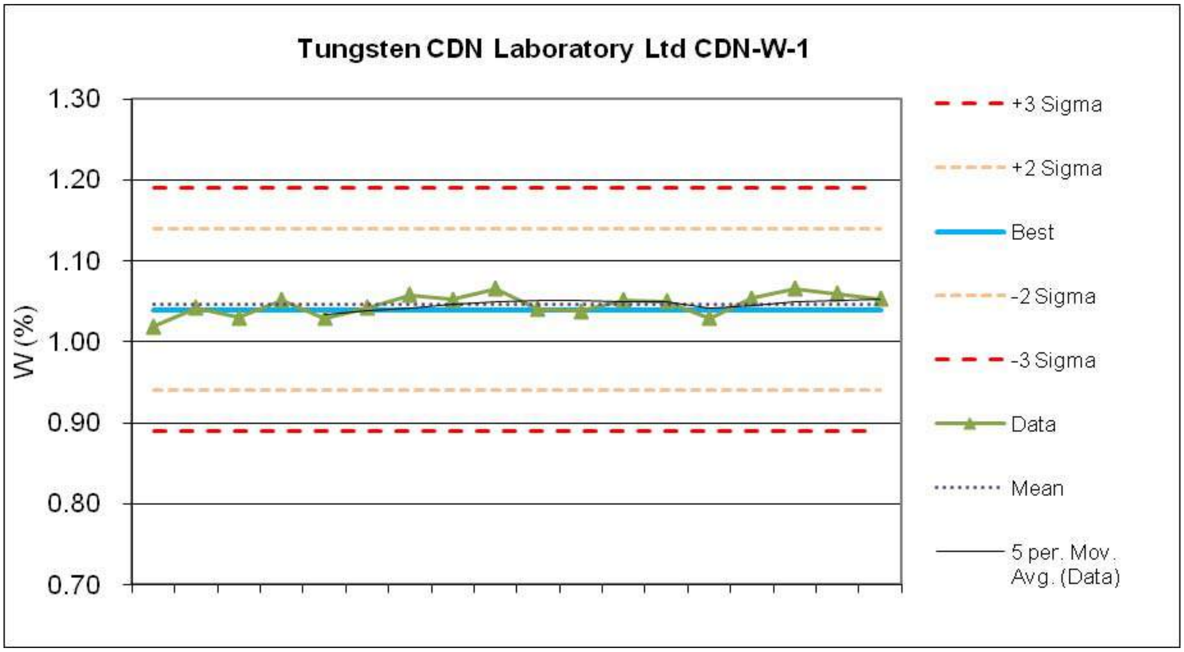


Figure 11.4: Tungsten Standard CDN-W-1

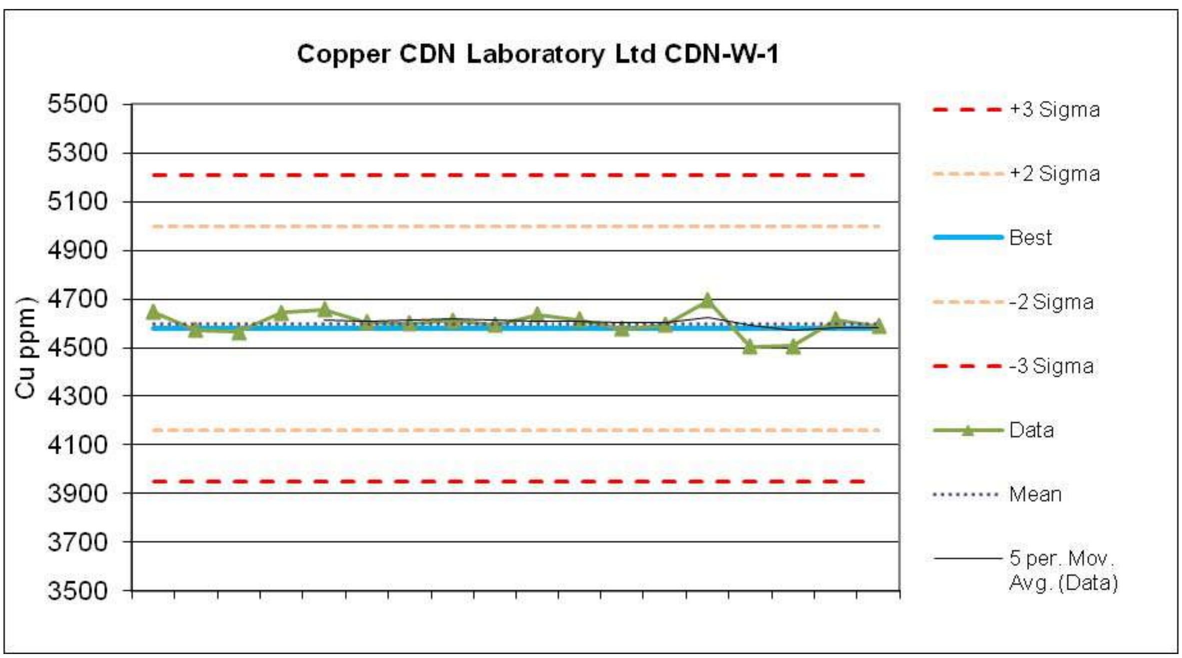


Figure 11.3: Copper standard CDN-W-1

11.3 Comments on the Data Quality and Assay

ASARCO invested significant amounts of money into the Grey River Tungsten property based on the quality of the tungsten data supplied by their laboratories. A great deal of preliminary work has been performed by ASARCO on the property including diamond drilling, the development of an adit, the extraction of raise samples and the development of some of the infrastructure around the portal. The historical data that was collected under ASARCO's supervision appear to follow the best industry practice at the time it was collected.

The probability plots for the various sample type (Section 14) indicates that the Playfair 2006 and 2008 drill results average are significantly lower grade than the historical samples. Check samples collected during the site visit also returned higher grade than Playfair samples. Since the determination of tungsten in low grade ores and geological samples is known to be difficult and challenging to assay. DGL suspect that wet analysis method used for the 2006 and 2008 drill core samples may leave some of the tungsten un-assayed.

For tungsten assay, ALS Chemex recommends using pressed pellet XRF for drill core while ActLabs recommend a neutron activation technique. The difference in these two methods is summarized below:

XRF is using 15g of sample press into a pellet then using X-ray to make the electrons jump to another orbital. Once the electrons go back to the original state it will emit an energy. By specific energy emitted as florescent the XRF detector is able to determine the elements and concentration.

Neutron Activation uses a high energy radioactive source to create isotope of the elements analyzed, followed by using the isotope radioactive decay to determine the elements and concentration. Usually there is a cool down period involved after activation and different isotopes will have different decay rates therefore the turnaround is usually longer.

Both methods are non destructive and don't involve the dissolution of the tungsten and are recommended by Srivastava and al (1996).

DGL recommend Playfair to re-run a portion of the Number 10 Vein pulp reject with either a XRF or INAA method. If the results of the check analysis indicate a different grade than the original samples, then the remaining vein material would need re-assaying. For the historical samples, a program of underground check assays can be initiated in the next exploration program along with a few twin holes. This program would also eliminate the lack of QA/QC on the historical samples.

DGL would also recommend Playfair to implement the submission of blanks and field duplicate in the sample string.

Based on the vein continuity, the number of samples collected and the fact that the Number 10 vein can be observed underground and on the surface, DGL believes that the supplied data are of sufficient quantity and quality to delineate an inferred resource. The lack of confidence in the analytical technique prevents the mineralization to be up-class to the indicated category.

12.0 DATA VERIFICATION

DGL has examined the records from the historical exploration and development work carried out on the Grey River Tungsten property. These records, which were made available by Playfair, consists of printed and digital data pertaining to exploration work carried out between 1954 and 2008.

12.1 Site Visit Summary by Wardrop

Wardrop visited the Newfoundland government core storage facility in Buchans, Newfoundland to examine the historical ASARCO drill core. Holes GR-1, GR-2, GR-8 and GR-10 were reviewed and it was also confirmed that the remaining holes were present in the storage facility. A variety of mineralized sections were checked and two issues are apparent:

1) The entire drill core within the mineralized zones were used for the ASARCO sample. This was the common practice for EX core due to its small diameter.

2) Re-drilling and grinding of the core is relatively common. This is a function of the standard drilling (non-wireline) technique used at the time. It is easily identified by footage tags that do not have the appropriate amount of drill core between them.

Wardrop also visited the Grey River Tungsten property to establish the coordinates of the drill collars for the 2006 program. In addition, the coordinates of the adit and one trench were determined and an attempt was made to gather the coordinates of the ASARCO drill pads. Due to snow coverage, it was not possible to examine the Number 10 Vein on surface.

12.2 Site Visit Summary by DGL

Mr. Pierre Desautels (P.Geo.) visited the Playfair' Grey River deposit, accompanied by Mr. Wallace Lushman, a local prospector residing in the community of Grey River, on October 05, 2011. Drilling was not active during the site visit since Playfair's exploration activity on the property stopped in 2008 following the completion of the drill program.

The 2011 site visit entailed reviews of the following:

- Overview of the geological setting of the Number 10 Vein.
- Surveying (topography and drill collar).
- Inspection of the higher grade core section from the 2006 and 2008 drill program.
- Core recovery.
- Inspect the ASARCO adit.
- Independent collection of character samples.

During the 2011 visit, DGL collected 3 character samples consisting of three half-core samples replicating the Playfair samples in hole GR08-115, 117 and 124, one core sample of approximately 10 cm long collected at 227.6 m in hole GR08-124 and one channel sample across the Number 10 Vein. DGL retained full custody of the sample from the Grey River deposit to Barrie, Ontario, where the samples were shipped to Activation Laboratories Ltd. located at 1428 Sandhill Drive, Ancaster, Ontario. The main intent of analyzing these samples was to confirm the presence of tungsten, copper, bismuth and molybdenum in the deposit by an independent laboratory not previously used by Playfair. Less relevant to the deposit, zinc and silver were also assayed.

At Activation Laboratories the samples were crushed (< 5 kg), split, and a 100 g cut was pulverized with mild steel (Activation Laboratories Code RX2).

Copper, silver, zinc, and molybdenum were analyzed by four acid digestion followed by ICP-OES (Activation Laboratories Code 8). Bismuth was analyzed by peroxide fusion followed by ICP-MS.

Tungsten was analyze by Wolfram Bergbau - WO_3 Induced Neutron Activation Analysis (INAA) which, according to Activation Laboratory, gives better results than wet chemistry such as fusion digestion, followed by ICP-MS because in most cases the method only requires the preparation of representative samples, i.e., pulverization or homogenization and this reduces the danger of contamination to a minimum and accelerates the whole analytical process. Solubilization of the sample is no longer necessary, which is difficult for tungsten since wolframite and ferberite tend to be resistant to the acid attack. From the assay results shown in Table 12-1, DGL character samples all returned higher tungsten concentration likely due to the difference in the analytical method used and the fact that the samples are 1/2 core splits and not pulp duplicates.

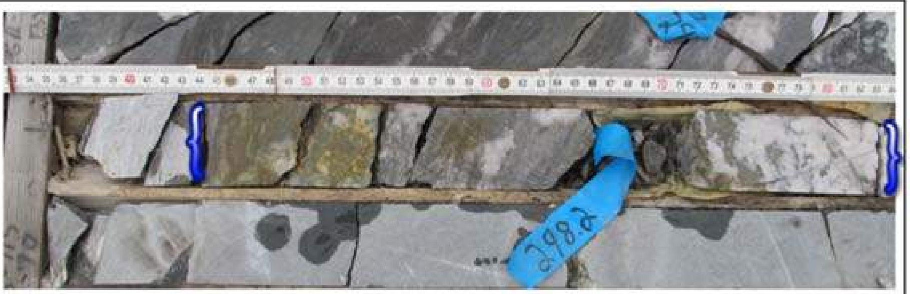
Table 12-1: Character Sample Results

Sample Type	Sample Number	WO_3 (%)	Bi (ppm)	Cu (%)	Mo (%)	Ag (ppm)	Zn (%)
GR08-124 (1/2 core)	DGL-001	0.840	508	0.285	< 0.003	13.0	0.03
	Playfair-83935	0.595	390	0.312	--	13.7	--
GR08-124 at 227.6	DGL-002	0.009	4	0.104	0.004	<3.0	0.02
	Playfair-83923	0.005	5	0.050	--	0.7	--
GR08-117 (1/2 core)	DGL-003	2.500	23	0.008	< 0.003	4.0	0.00
	Playfair-84344	0.773	75	0.017	--	0.9	--
GR08-115 (1/2 core)	DGL-004	0.175	665	0.054	0.015	10.0	0.02
	Playfair-84290	0.115	405	0.110	--	6.8	--
Across No10 Vein	DGL-005	5.180	416	0.006	<0.003	5.0	0.00
	No Playfair Equiv.	--	--	--	--	--	--

Despite the difference seen, the character samples were within an acceptable range with Playfair's original results and confirm the presence of tungsten in the original assay. However, the difference in the analytical method should be investigated further.

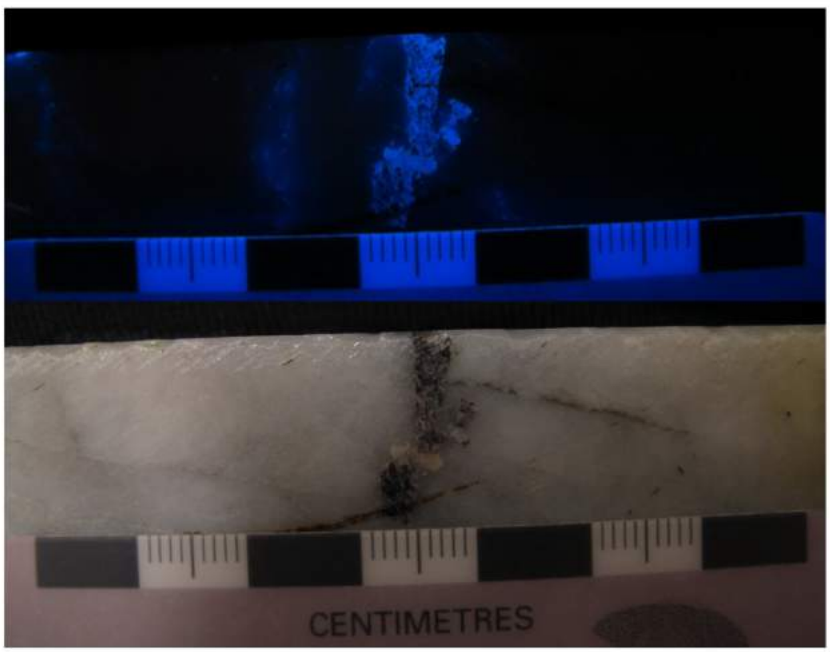
Geologists responsible for logging the core have no difficulty identifying the Number 10 Vein (shown on Figure 12.1) since the brecciated quartz is very distinct. During the site visit, sulphide was found to be present in minor amounts consisting mainly of pyrite with trace chalcopyrite. Wolframite was observed to occur in isolated clusters and sometimes large crystals rendering the mineralization difficult to sample with small diameter core. The hanging wall of Number 10 Vein was inspected by DGL in hole GR08-124. The zone consisted of quartz veinlets carrying fluorite and scheelite. The veinlets displayed a low core angle indicating that they may be dipping sub-parallel to the drill direction or strike in a E-W direction.

Scheelite was observed to be present with fluorite, apparently replacing wolframite along the crystal surfaces and cleavage planes. Figure 12.2 shows a sample collected between 302.8 m and 303.8 m in hole GR08-117 under UV light.



Example of Number 10 Vein in hole GR-08-115 between 298.0 m and 298.3 m (approximate grade of 0.150 WO3%)

Figure 12.1: Number 10 vein in core



Hole 08-117 from 302.8 m to 303.8 m Showing No 10 Vein mineralization
Top Image: Short Wave UV Light
Bottom Image: Day Light

Photo by DGL

Figure 12.2: Scheelite flourite and Wolframite in quartz

Playfair only sampled the mineralized sections of the core, which is considered appropriate for this deposit. Figure 12.3 shows a few selected images from the most recent site visit.

The underground inspection is described in Section 9.1.1 of this report and will not be repeated here.

Local Service District of Grey River

GR-08-115 casing

Coarse grained Wolframite GR-08-117 @ 303.8m

Number 10 Vein surface outcrop

Channel sample cut with power saw

Playfair bulk sample

Figure 12.3: Site visit photographs

12.3 Drill Hole Collar Validation

Collar coordinates were validated with the aid of a hand-held Garmin GPS map model 60CSx. A series of collars were randomly selected and the GPS position was recorded. The difference between these values and those recorded in the Gems database was calculated in an X-Y 2D plane with the following formula:

$$X - Y\ difference = \sqrt{(\Delta\mathrm{East})^2 + (\Delta\mathrm{North})^2}$$

Wardrop GPS coordinates were also added to the dataset since the position of the holes were adjusted after the 2008 survey conducted by Yates and Wood. There is good agreement between the historical drill collar coordinates and the coordinates determined during the site visit. As shown on Table 12-2, results indicated an average difference of 17.5 m in the X-Y plane and -2.8 m in the Z plane for the 10 hole collars where the instrument was located near the casing. The calculated differences in the X-Y plane are well within the accuracy of the hand held GPS unit used, which is typically influenced by the number of satellites seen by the instrument on the day of the survey.

Table 12-2: Collar Coordinate Validation

Gemcom Database Entry				GPS Point Recorded during Site Visit				Differences between GEMS and GPS	
Point-ID	**East**	**North**	**Elev.**	**Origin**	**East**	**North**	**Elev**	**X-Y Plane (m.)**	**Z Plane (m.)**
GR-06-102, 103	492643	5271477	265	Wardrop	492625	5271485	272	19.6	-6.5
GR-06-114	492627	5271415	264	Wardrop	492612	5271424	269	17.6	-5.2
GR-06-104, 105	492704	5271566	249	Wardrop	492684	5271576	266	22.1	-16.5
GR-16	492748	5271619	238	Wardrop	492731	5271625	240	18.3	-2.3
GR-06-106,107	492796	5271666	235	Wardrop	492781	5271670	238	15.2	-2.8
GR-27,28	492834	5271746	258	Wardrop	492819	5271755	260	17.7	-2.5
GR-1	492792	5271593	236	Wardrop	492775	5271596	240	17.4	-4.0
Portal	492441	5270469	14	Wardrop	492422	5270471	9	18.8	4.7
GR08-122	492861	5271924	240	DGL	492857	5271923	234	4.5	6.3
GR08-115 to 118	492763	5271797	243	DGL	492744	5271802	246	19.9	-2.8
GR06-114	492627	5271415	264	DGL	492610	5271423	271	19.0	-7.2
Portal	492441	5270469	14	DGL	492426	5270482	8	19.4	5.7
				Average Difference				17.5	-2.8

12.4 Digital Data Verification by Wardrop

Wardrop validated four of the holes in the database by comparing the original drill log data with the summary sheets supplied by the client. This sample population represents 9% of the total holes in the database (4 out of 45 holes).

Data verification validated the collar co-ordinates, length of holes, down-the-hole survey measurements (including azimuth and dip), as well as footage intervals of the assay samples and the lithological units. Tungsten values from non-drill hole samples were checked against the data on printed maps (for the trenches and grab samples) as well as plots of the underground sampling diagrams (for the back, face, raise and bulk samples). Minor errors are present in the lithology data set; two of the checked intervals differed by 0.1 metre while the other two intervals had a difference of 0.2 metre.

The coordinates for the four holes could not be confirmed because no field grid sketch is available for cross-reference. This is not critical given that the drill holes in the database are in UTM space (NAD 27 Zone 21) rather than grid coordinate space. As indicated above, the UTM coordinates for selected holes were confirmed during the site visit which suggests that all of the collar coordinates in the database are correct. Details of the verification are given in Table 12-3.

Table 12-3: Data Verification

Database Portion	Total Records	Error Records	Records with Errors	Records Validated
Collar	8	0	0%	Coordinates (easting, northing, elevation and depth).
Survey	16	0	0%	Survey depths, survey dips and survey azimuths.
Geology	132	8	6%	Names of units and downhole depths.
Assay	104	0	0%	Tungsten values and distances down hole.
Underground	993	1	0.1%	Tungsten and copper values. Width of samples.
Total	**1253**	**9**	**0.72%**	

12.5 Digital Data Verification by DGL

12.5.1 Historical Drill Holes

For this resource estimate and for reasons of completeness, the Playfair digital database was improved with the addition of forty one, pre-1970 drill holes on the property. Twenty three holes consisted of surface exploration drilling away from the Number 10 or Number 6 Veins. The remaining additional eighteen holes were underground definition drilling targeted at the Number 10 Vein ahead of the drift face with a few holes, drilled from the underground cross-cut, and targeted the Number 10 Vein below the adit level. Of these, nine definition holes were not sampled by ASARCO despite intersecting the Number 10 Vein. Only three holes collared from the underground cross-cut, GR-39, GR-40 and GR-41 had assay results that influenced the grade of the model below the adit level. Two holes GR-47 and GR-48 with available drill logs did not have collar coordinates and could not be located on any of the plans reviewed. Holes TH-1 to TH-13 were known to have been drilled but no logs existed for these holes. They are assumed to be definition holes drilled with extension steel attached to a Jackleg drill.

12.5.2 Historical Channel Samples

During the data validation process, one additional historical channel sample plan was located. The plan showed ASARCO channel samples and the South Plainfield check samples. The data from that plan was added to the database as part of this resource estimate. For the South Plainfield data, the sample width was assumed to be the same as the closest ASARCO sample. A total of 65 additional samples were added to the database. DGL also added the sample width and copper assays which were previously not entered in the database by Wardrop.

12.5.3 Trench and Rock Samples

Out of the 17 trench samples in the database only 11 had a sample width. The grade from the trenches originated from a plan view that was geo-referenced and digitized in the GEMS database. The WO_3% value on the plan appears to be an average of a number of channel samples and no assay certificates were available for review.

12.5.4 Assay Validation

The validation against the official, signed electronic version of the certificates in PDF format consisted of comparing the values on the certificate against the GEMS database entry. Certificates were obtained from Playfair for the 2006 and 2008 drill campaign. For the historical holes, some ASARCO assay certificate were located but not for all holes. For the holes that no certificate existed, the database value was compared to the value entered in the logs.

A total of 373 assay results were compiled from the certificates into an Excel spreadsheet (39% of the database) and matched against the sample number in the GEMS database. Hole GR-06-110 originally showed an import error originated by the author the validation rate was escalated to include all assays intersecting the number 10 and number 6 Vein for the 2006 and 2008 drill program. During the validation, a number of errors were encountered related to using the W% value in the WO_3% column, which, if left uncorrected, lowers the grade by 1.2611. Other issues were minor in nature and are related to rounding. The 2008 drill results were free of any errors.

All assays showing a grade variation in excess of 0.005 WO_3% when compared to the laboratory certificate were corrected. The assay validated and corrected by DGL, amounted to a total of 373 assays covering 37% of all assays for the 2006 and 2008 drill campaign and 69% of all historical drill core assays.

Notwithstanding the possible issue related to the analytical technique used, the Qualified Person regards the database to be free of major errors insofar as the assays in the database match the laboratory certificates or the valued entered on drill logs and plans.

13.0 METALLURGICAL TESTING AND MINERAL PROCESSING

13.1 Metallurgical Testing

Two metallurgical reports were completed by SGS Mineral Services to derive a preliminary flow sheet and cost estimate: 'Metallurgical Scoping Test work Report on a Sample of Wolframite Ore' dated May 31, 2006 and 'Phase 2 Metallurgical Scoping Test work Report on a Sample of Wolframite Ore', dated November 6, 2006.

Further metallurgical test work is required to demonstrate that an acceptable grade concentrate at an acceptable metallurgical recovery can be achieved. Individual gravity separation tests indicate that the quality of the sample preparation was insufficient to produce consistent metallurgical results. The test work indicates that testing was undertaken on a 'bulk sample', however, the indicated grade was relatively high compared to the current estimated feed grade. The potential mine feed is highly diluted compared to the sample tested by SGS and so metallurgical results from those reports are potentially optimistic compared to results achieved from a fully diluted sample.

The back-calculated feed grades vary from 2.32% to 4.01% WO_3, whereas the current underground mining resource has an estimated grade of 0.52%. Concentrate specification needs to be addressed in future test work. Furthermore, WO_3 cons destined for APT plants have a target size distribution specification. Further metallurgical work is strongly recommended in order to; establish a firm specification on the head grade for the feed; maximize recovery to an acceptable WO_3 concentrate; and, move beyond the scoping stage and simulate the unit processes of the proposed flow sheet on a representative sample of the feed material. This will help establish firm criteria for the metallurgical performance to produce an acceptable concentrate.

13.2 Mineral Processing

Grey River is proposed as a low-tonnage, high grade operation, with a relatively free-milling ore, shown to be amenable to gravity separation methods, producing a potential concentrate of 60% with a tungsten recovery of 75%. The concentrate is destined for Ammonium Para Tungstate (APT) plants either on the Continental US or alternately customers in Europe. However, the concentrate produced thus far does not yet meet typical feed grade or particle size specifications for conversion to Ammonium Paratungstate and it may be advantageous to produce a concentrate meeting these specifications directly in order to maximize value. It is generally accepted that concentrates grading >65% WO_3 attract a price premium on the market, and a further price premium is achieved in meeting the feed size specifications to the APT plant. Therefore, it is recommended that further test work be undertaken to meet these criteria. A generally applicable tungsten processing flow sheet has thus been selected for costing purposes, and flexibility to meet this option has been built into the flows sheet through the inclusion of a rod milling circuit as an alternative to an impact crushing option, in advance of the spiral plant.

13.2.1 Processing Methods

A generally applicable tungsten processing flow sheet has been selected for plant design and costing. Coarse ore concentration may be achieved by dense media separation or a heavy mineral jig as tested; however, a three-stage x-ray based sorting option is shown in place of the conventional jig/cyclone option based on the high expected levels of dilution in the ore and the poor indicated performance of the jig in this application. This decision is supported by recently published success at other operations in sorting similar feeds. Fine ore concentration is by a 3-stage heavy mineral spiral circuit. Pyrite rejection can be by gravity methods as indicated by the test work, or by reverse flotation of the gravity concentrate. Grey River is proposed as a low-tonnage, relatively high grade operation with a relatively free-milling feed material that is amenable to gravity separation methods. It may also be advantageous to produce a further refined concentrate that meets the APT plant feed specifications directly in order to maximize economic return. Flexibility to meet this option has been built into the flow sheet through the inclusion of a rod milling circuit, as an alternative to the impact crushing option ahead of the spiral plant.

In the dense medium, as well as the jig test work, it is felt that the upgrading demonstrated was insufficient to meet the concentrate criteria at a reasonable recovery. Therefore, the option to introduce the coarse concentrate into the grinding circuit has been provided for in the costing. Metallurgical results indicate that pyrite rejection may be achievable by gravity methods only, hence the inclusion of a third stage concentrating table or scavenger spiral. Provision has been made, however, for the rejection of pyrite from the concentrate by flotation should this be required.

13.2.2 Recoverability

The average metallurgical recovery stated in the SGS test work report was 79.3% to a 60% WO_3 concentrate. It is felt that these results, despite the lower feed grade, are possible pessimistic and that a typical gravity/flotation-based plant processing material such as at Grey River would generally obtain between 85-92% recovery to a 65-70% concentrate. The flow sheet presented here allows for a high mass pull to concentrate, while making provision for cleaning of the concentrate to ~65% by gravity in order to realize this potentially higher recovery and maintaining concentrate grade. A projected recovery of 85% WO_3 is thus used in this study but is provisional and should be confirmed by test work in the next phase of work. The proposed flow sheet is based on experience as well as the results from the metallurgical reports. The design criteria used in this study are presented in Table 13-1.

Table 13-1: Metallurgical Design Criteria

Criterion	Value
Material Type	Primary Tungsten
Mineralogy	Wolframite + Hubnerite + Scheelite with Pyrite
Grade	2.32% WO_3
Head Grade	0.53% WO_3
Mining Rate	400 tpd
Feed Topsize	200 mm
Mass Pull	2.83% (1.13 diluted)
Concentrate Grade	~65% WO_3

Details of the flow sheet can be found in Appendix B.

14.0 MINERAL RESOURCE ESTIMATE

14.1.1 Data

A mineral resource estimate has been completed by DGL for the Number 10 and Number 6 Vein on the Grey River Tungsten property in southern Newfoundland. Gemcom software GEMS 6.3 was used for the resource estimate. The metal of interest at the Grey River project is tungsten with possible copper, zinc and lead credits. Since approximately 85% of the raw data was also assayed for copper, the model was interpolated for that element; however since the metallurgical test work did not evaluate the possibility of extracting copper, the mineral resource will only report for tungsten.

Playfair provided a digital drill hole database in a Gemcom project that originated from the June 2007, NI43-101 report from Wardrop. Digital scan copies of the ASARCO adit, sampling plan and various historical reports were also provided by Playfair. The Playfair 2008 drill data, along with changes to the hole coordinates resulting from the Yates and Wood survey were added to the database along with other minor changes described in Section 12 of this report.

A total of fifty-five historical holes now exist in the database of which twenty-five holes were used for the resource estimation. The remaining thirty holes that were not used in the resource estimate consisted of twenty-one surface exploration holes not targeting the Number 10 or Number 6 Vein, eight underground holes that were discarded because of lack of sampling by ASARCO even though some of the holes intersected the Number 10 Vein and one underground hole that had 2 assays that did not line up with the expected position of the Number 10 Vein in relation to its position in the adit. Playfair 2006 and 2008 drill campaigns added an additional twenty-six holes, all of them used in the resource estimation. Table 14-1 summarizes the drill hole data used for the estimate.

Table 14-1: Drilling Data Records used for the Resource Estimate

Drill Core Holes	Origin	Number used in Resource	Total Meter	Assays
Used in Resources	Historical	25	3388.43	25
	Playfair 2006	16	2922.2	267
	Playfair 2008	10	3954.8	616
Not Used	Historical	30	1420.45	2
	Playfair 2006	0	n/a	n/a
	Playfair 2008	0	n/a	n/a
Total in Database		81	11685.88	910

Other data types were also used for the resource estimate. There were two campaigns of back sampling and one series of face samples completed by ASARCO and all three sets have been used in the resource model. The south Plainfield check samples were not use in the resource due to the missing sample length on plan and the lack of indication as to the unit (W% or WO_3%) used to express the assay and the analytical technique used.

ASARCO collected the underground face samples at a nominal average spacing of 2.7 m while the two series of back samples were collected at a nominal average spacing of 1 m. In order to minimize any bias in the search and interpolation procedures, these samples were de-clustered using a polygonal method.

Trench and grab samples were only used if the original data had a sample length associated with the assays.

A total of 12 grade originating from the ASARCO bulk sample program was also used for the area north of coordinate 5,271,650 N. Table 14-2 summarizes the data from other the source used in the resource estimate.

Table 14-2: Non-drilling Data Records used for the Resource Estimate

	ASARCO Face and Back Samples	South Plainfield Check Samples	ASARCO Bulk Samples	Rock Samples	Trench Samples
Total in Database	424	284	12	22	17
Used in Resource (De-clustered Data)	40	0	12	2	11

14.1.2 South Plainfield Check Sample Assessment

The 284 check back samples from the South Plainfield average 0.415 (%W or %WO3) compared to an average of 0.764% WO_3 for the 308 ASARCO back samples and 1.204% WO_3 for the 116 ASARCO face samples. If the South Plainfield check samples were in %W the grade difference would still be large once converted to %WO_3. The possible explanation for the difference in grade could be related to the sample length as South Plainfield may have sampled the entire drift as oppose to ASARCO, which sampled only the vein. The difference could also be due to the analytical technique, which is not specified in the South Plainfield data. Unfortunately, there is no recent or historical drill holes in the immediate vicinity of the drift which would allow for comparison of the grade. Therefore, DGL elected not to use the South Plainfield assays in the resource estimate but recommended Playfair to conduct a limited campaign of underground check sampling to confirm the grade of the ASARCO samples.

14.1.3 Bulk Sample Assessment

For the bulk sample grade assessment, data was filtered within a corridor of +50 / -50 meter from the drift elevation north of co-ordinates 5,271,650 N. The length weighted average grade for the bulk samples A and B were compared to the length weighted grade of the intersection of the Number 10 Vein in drill hole GR-06-107. Results indicated a bulk sample grade of 0.297% WO_3 which compared well with the drill hole grade of 0.339% WO_3. This was not the case for bulk Sample E and F when compared with hole GR-08-115 which showed a 0.271% WO_3 difference between the bulk sample grade of 0.610% WO_3 and the drill hole grade of 0.339% WO_3. Despite the difference, DGL believes that the bulk sample grade is likely more representative of the vein grade at the location the sample was taken, simply due to the sheer size of the sample and the fact that the bulk samples would not be affected by the difficulty in sampling coarse wolframite in drill core.

14.2 Geological Interpretation

The 3D wireframes developed to control the grade interpolation of the resource model were based primarily upon lithologies and partially on tungsten grades.

The geological wireframe was constructed using all drill hole intercepts within the quartz breccia vein typically logged as Vein 10, Vein 6 or quartz vein. During the construction of the wireframe, continuous zones of mineralization within Number 10 and Number 6 Vein favouring areas where the tungsten grade exceeded 0.05 WO_3. In rare cases, exception was made to include lower grade intercepts to allow zonal continuity. The wireframe model was also reconciled with the vein width in the ASARCO adit and the mapping of the vein on the surface plan. The wireframe construction was carried out in multiple steps as follows:

- The surface expression of the Number 6 and Number 10 as surveyed by ASARCO was digitized on a plan view, and the contacts were elevated to the topographical surface.
- Polylines describing the upper and lower contacts of the zones were digitized on the sections using the lithology as the primary guiding principle.
- The geological maps for the ASARCO exploration adit were geo-referenced in GEMS, and the geological interpretation was digitized from these plans.
- The digitized level plan polylines were connected to the sectional interpretation of the zone. Adjustment and re-interpretation were made to the sections if needed.
- The resulting plan and section polylines were both used to construct the wireframes.
- The model was validated in 3D against the drill holes, and adjusted once more if necessary.

Accessory wireframe on the hanging wall of vein 10 was constructed if a minimum 3 drill intercepts on at least 2 sections with a minimum of 2 assay exceeding 0.2% WO_3 were encountered. Of all the assays present on the hanging wall, only one zone (Vein 10a) could be constructed following these guidelines.

In all cases, the vein model was drawn at the vein width as indicated by the drill or map data with no minimum mining width added to the model.

Total un-diluted wireframe volume for the three wireframe constructed is shown in Table 14-3 below.

Table 14-3: Total Wireframe Volume

Vein Name	Volume in Cubic Meter
Vein 10	507,668
Vein 6	40,660
Vein 10A	37,001

The topography surface was constructed using the CanVec topographical data obtained from the Ministry of Natural resources, sheet 11P11 and merged with topographic contours originating from a 1"-200' scale map dated September 14, 1957 that was geo-referenced and digitized.

Figure 14.1 illustrates the model for the Grey River deposit.

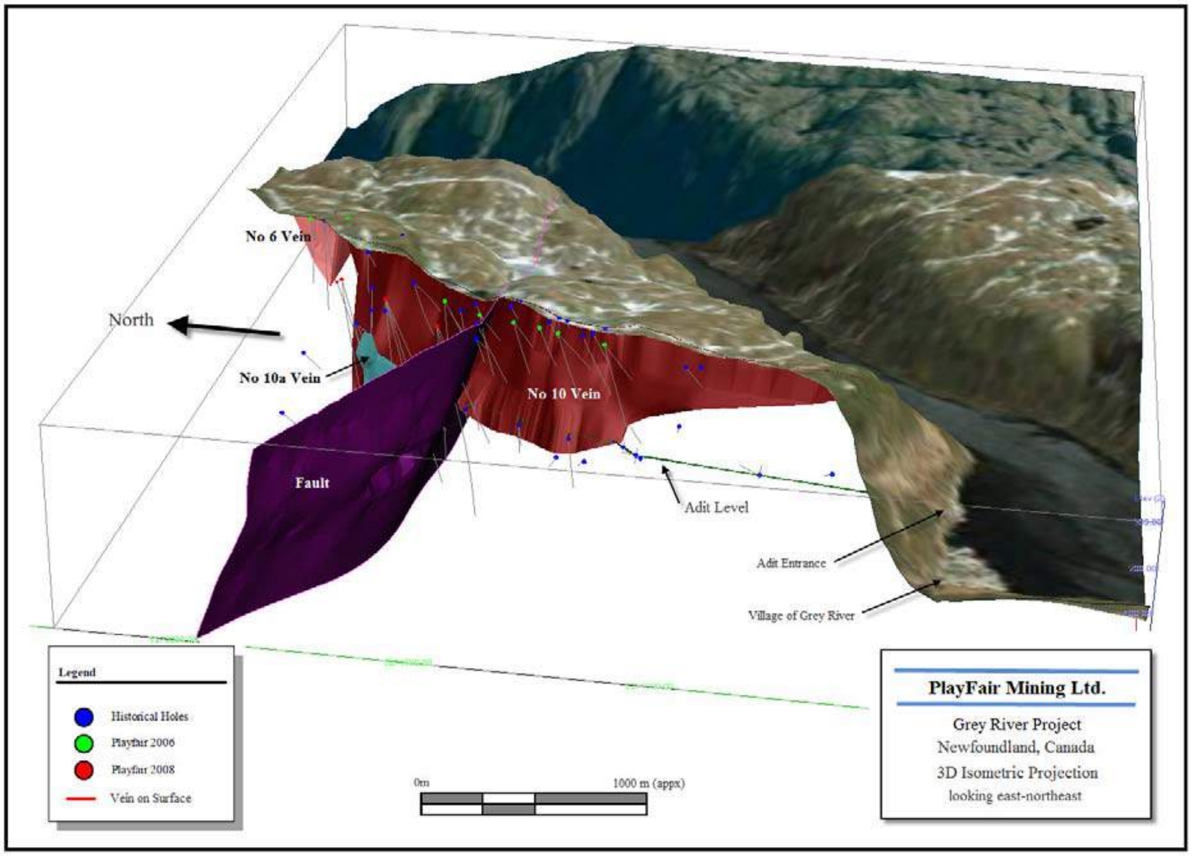


Figure 14.1: Position of the 3-D wireframe volume with contours

14.3 Exploratory Data Analysis

Exploratory data analysis is the application of various statistical tools to characterize the statistical behaviour or grade distributions of the data set. In this case, the objective is to understand the population distribution of the grade elements in the various units using such tools as histograms, descriptive statistics, and probability plots.

14.3.1 Assays

The raw assay statistics were evaluated by grouping all assays intersecting the Number 10, Number 10A and Number 6 Veins. Statistical analysis compared the trench, back and faces samples with the limited drill core assays, to verify whether a different sample type could be mixed with drill data and used in the resource interpolation.

Figure 14.2 shows probability plots for the various raw data types.

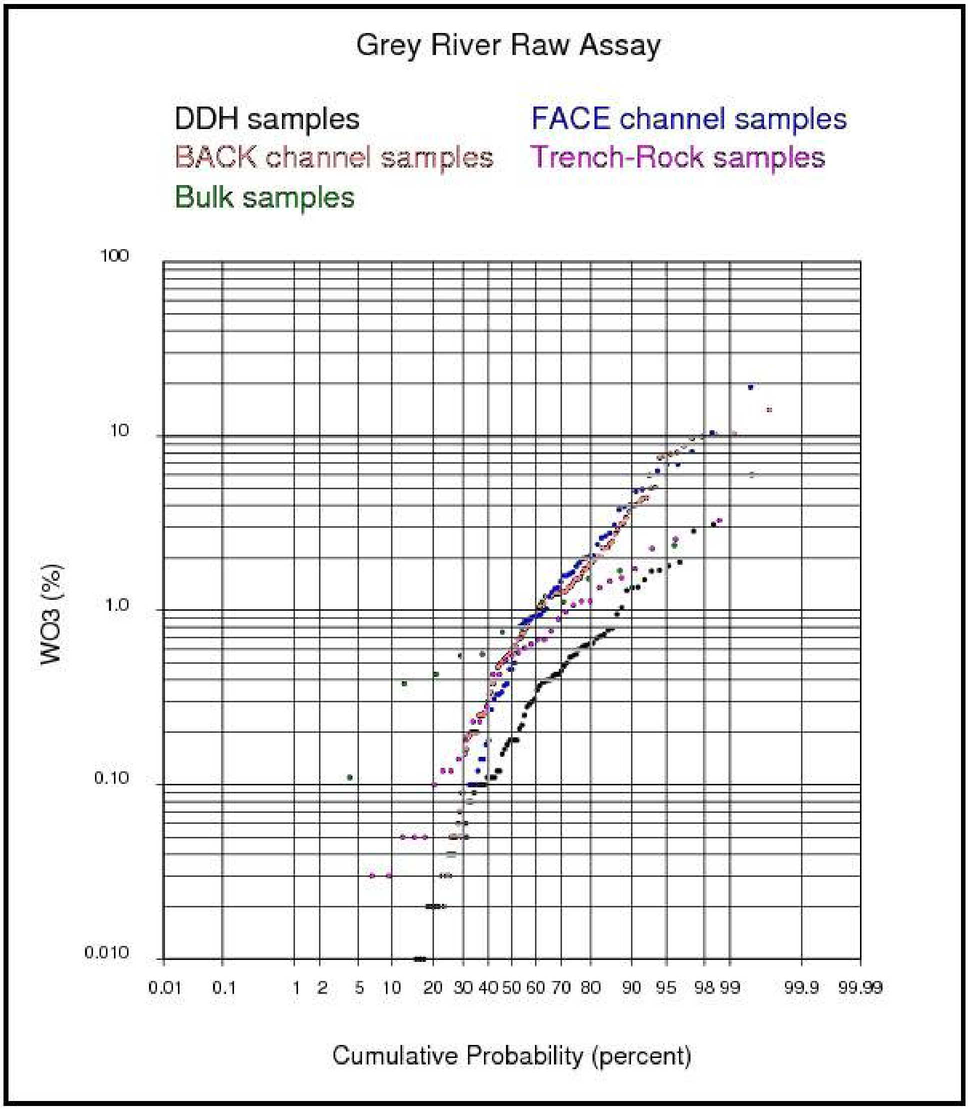


Figure 14.2: Population distribution of various sample types

The population distribution for trench, back, face and bulk samples all show higher grade than the drill hole distribution. The face and back samples distribution is virtually the same therefore the face and back samples were treated as one population in the remaining statistics presented in this report. For this study, the explanation related to the higher grade seen in the face, back, trench and bulk sample population is unknown but suspected to be an analytical procedure issue. It could also be attributed to the sample size because as stated earlier in this report, the coarse wolframite mineralization is difficult to assay with small diameter core drilling. The grade discrepancy could easily be resolved with limited re-sampling program by Playfair.

The frequency distribution of all samples shows a near log normal distribution with 98% of the WO_3% values below 10%. Table 14-4 provides descriptive statistics for the underground back and face samples along with the drill hole, trench and bulk samples for tungsten trioxide.

Table 14-4: Raw Assay Statistics (WO_3)

	All Sample	All DDH	Bulk	Rock	Trench	Face + Back	All Type No10	All Type No 6	All Type No10a
Valid Cases	543	99	12	2	11	424	520	12	11
Mean	0.813	0.456	0.954	0.650	0.875	0.884	0.838	0.151	0.321
Variance	3.496	0.652	0.418	0.029	0.549	4.274	3.628	0.040	0.295
Std. Dev.	1.870	0.808	0.647	0.170	0.741	2.067	1.905	0.201	0.543
Var. Coeff.	2.300	1.771	0.678	0.261	0.846	2.339	2.272	1.333	1.691
Minimum	0.000	0.000	0.120	0.530	0.000	0.000	0.000	0.003	0.005
Maximum	19.070	6.000	2.370	0.770	2.270	19.070	19.070	0.555	1.892
1st perc.	0.000	0.000	----	----	----	0.000	0.000	----	----
5th perc.	0.000	0.003	----	----	----	0.000	0.000	----	----
10th perc.	0.000	0.006	0.201	----	0.010	0.000	0.000	0.004	0.006
25th perc.	0.000	0.029	0.460	----	0.050	0.000	0.000	0.006	0.013
Median.	0.050	0.173	0.775	0.650	0.680	0.010	0.050	0.055	0.130
75th perc.	0.840	0.555	1.430	----	1.550	0.918	0.900	0.280	0.372
90th perc.	2.020	1.350	2.166	----	2.162	2.340	2.048	0.540	1.605
95th perc.	4.022	1.800	----	----	----	4.720	4.039	----	----
99th perc.	10.090	6.000	----	----	----	10.307	10.169	----	----

14.4 Capping

A combination of decile analysis and a review of probability plots were used to determine the potential risk of grade distortion from higher-grade assays. A decile is any of the nine values that divide the sorted data into ten equal parts so that each part represents one tenth of the sample or population. In a mining project, high-grade outliers can contribute excessively to the total metal content of the deposit.

Typically, in a decile analysis, capping is warranted if:

- The last decile has more than 40% metal.
- The last decile contains more than 2.3 times the metal quantity contained in the one before last.
- The last centile contains more than 10% metal.
- The last centile contains more than 1.75 times the metal quantity contained in the one before last.

The decile analysis results shown in Figure 14.3 indicated that grade capping was warranted.

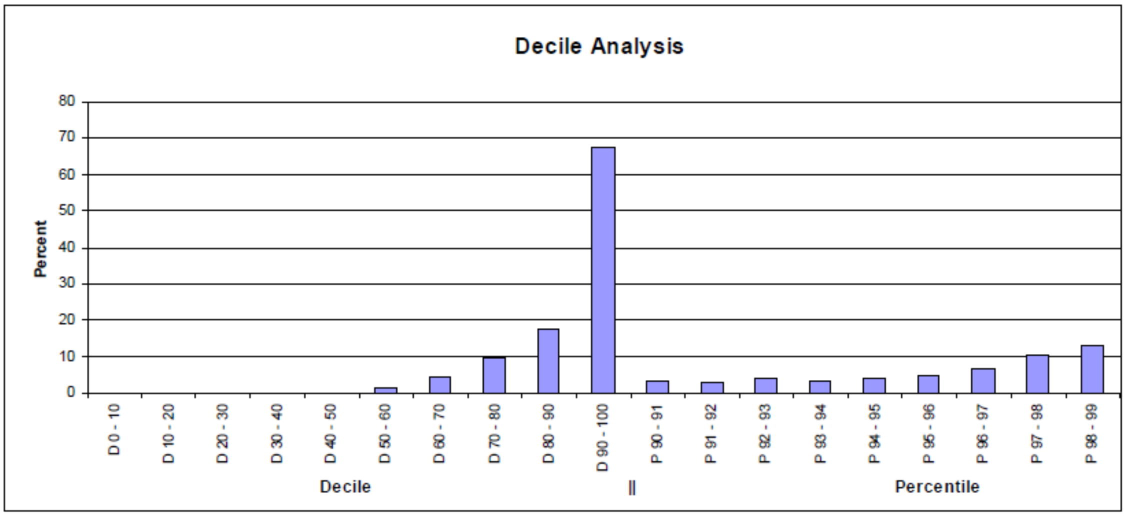


Figure 14.3: Decile analysis result

After conducting a careful examination of the data set, DGL elected to use an 8.5% WO_3 cap value for all assays prior to compositing. A total of 8 samples were capped.

14.5 Composite

14.5.1 Face and Back Sample De-clustering

The spatial locations of the face and back samples are not randomly or regularly spaced in relation to the entire model. If data is preferentially sampled when it is spatially auto correlated, the resulting histogram from the sample may not reflect the histogram of the entire population. For this reason, it is best to de-cluster the data. The face and back samples were de-clustered using a "pseudo" polygonal technique.

Polygons were drawn to be between 5 to 10 meters in length along the drift with their boundaries targeted at separating the high and low grade areas. No strict cut-off rule was used, the high/low grade areas were visually estimated based on a color-coding of the sample values. All drift and back samples within the polygons were length weighted average to produce one data point, located at the center of the polygon, for the block model input file. This methodology is different than Wardrop where every fifth sample were used in the de-clustering algorithm. There are two benefits to this methodology:

- All samples are considered in the average.
- The width of the sample is use as a weight maintaining the metal balance.

A total of 40 polygons were digitized averaging 10.6 samples per polygons with a 3.1 standard deviation. Samples were tagged with the polygon name, extracted into an XLS spreadsheet and the calculation of the average grade was performed outside Gemcom using a pivot table.

14.5.2 Drill Hole Composites

The average width of the underground sampling is 1.12 m with a 0.55 standard deviation. On surface, the trench and rock samples average 1.24 m with a 0.76 standard deviation. The drillhole sampling interval along the vein averaged 1.03 m in core length with a 0.69 standard deviation and a upper third quartile value of 1.30 m. Due to the narrow vein width and the number of single point composites generated by the de-clustering algorithm applied to the face and back samples, DGL elected to use a single vein composite for all drill holes intersecting the wireframe.

Assays were length-weighted averaged across the entire intersection with the vein and any grade capping was applied to the raw assay data prior to compositing. Gaps in sampling, if present, were composited at zero grade.

Once the composite file was populated with the drillhole composites, the surface and underground samples were added to the file. Table 14-5 presents the composite statistics.

Table 14-5: Composite Statistics (%WO_3)

	All Sample	All DDH	Bulk	Rock	Trench	Face + Back	All Type No10	All Type No 6	All Type No10a
Valid Cases	110	45	12	2	11	40	101	6	3
Mean	0.773	0.662	0.954	0.650	0.875	0.821	0.818	0.180	0.441
Variance	0.089	0.160	0.187	0.120	0.223	0.148	0.096	0.079	0.285
Std. Dev.	0.879	1.149	0.418	0.029	0.549	0.872	0.925	0.038	0.243
Var. Coeff.	0.938	1.072	0.647	0.170	0.741	0.934	0.962	0.194	0.493
Minimum	1.213	1.620	0.678	0.261	0.846	1.137	1.176	1.081	1.118
Maximum	0.000	0.000	0.120	0.530	0.000	0.000	0.000	0.006	0.084
1st perc.	6.000	6.000	2.370	0.770	2.270	3.490	6.000	0.504	1.004
5th perc.	0.000	----	----	----	----	----	0.000	----	----
10th perc.	0.000	0.000	----	----	----	0.001	0.000	----	----
25th perc.	0.005	0.003	0.201	----	0.010	0.004	0.005	----	----
Median.	0.137	0.095	0.460	----	0.050	0.127	0.144	0.009	0.084
75th perc.	0.465	0.263	0.775	0.650	0.680	0.429	0.550	0.124	0.236
90th perc.	1.123	0.836	1.430	----	1.550	1.425	1.150	0.358	1.004
95th perc.	1.875	1.620	2.166	----	2.162	2.319	1.967	----	----
99th perc.	2.604	3.024	----	----	----	3.404	2.803	----	----

14.6 Bulk Density

The available documentation for the ASARCO resource estimate suggests that a bulk density of 3.10 grams per cubic centimetre (g/cc) was used. Wardrop calculated a predicted density of 2.71 g/cc based on the WO_3% values returned for the assays. The calculation assumed a quartz/wolframite mixed with no other mineral such as fluorite, pyrite and scheelite. Based on the predicted value and experience with other similar deposit a value of 2.8 g/cc was used by Wardrop.

In 2008, Playfair submitted 18 samples went to Eco Tech for density measurements and supplemented the Eco Tech data with 40 additional field measurements.

The Eco Tech used approximately 50 grams of dry reject weighed into a dry phosphoric acid flask and the weight was recorded. Reverse osmosis water was added to cover the sample and it is swirled to ensure complete wetting of the sample and absence of all air. Once the sample has degassed, water was added to the 200ml line and then reweighed.

The field measurements used a conventional Ohaus Scout Pro balance. The balance was elevated and levelled with a thin wire suspended from the hook beneath the balance. The sample was then weighed in air suspended beneath the balance and weighed again suspended in a container of water. Playfair reported some difficulty weighting the sample outside due to the sensitivity of the balance.

The average of 2.73 g/cc for the Number 10 Vein returned by the Eco Tech laboratory is lower than the field measurement of 2.89 g/cc by Playfair. The difference is likely due to using reject as oppose to using full core samples.

For the purpose of this resource estimate, an average bulk density of 2.81 g/cc that has been derived by averaging all 21 Number 10 Vein samples (EcoTech and Playfair). Two outliers were omitted. Table 14-6 shows the data used in the bulk density determination.

Table 14-6: Bulk Density Samples in Number 10 Vein

Hole #	Meterage (m)	Material Type	Playfair (g/cm3)	Eco Tech (g/cm3)	Vn 10 (less Outliers) (g/cm3)
GR-102	at 108.2m	vein 10, approx 5% sulphides & wolframite	2.88		2.88
GR-103	at 155.1m	vein 10, 5% sulphides and wolframite	2.90		2.90
GR-105	at 173.7m	Vein 10, approx 1% wolframite	2.77		2.77
GR-105	at 172.5m	sericitic alteration & vein 10, 1% wolframite	2.91		2.91
GR-106w	at 111.0m	quartz vein 10, <1% wolframite, 1% sulphide	2.66		2.66
GR-117	302.9-303.3	Quartz Vein (minor pyrite and wolframite)	2.64		2.64
GR-117	303.3-303.7	Quartz Vein (moderate wolframite)	2.78	2.72	2.78
GR-118	407.0-407.5	Quartz Vein (minor pyrite and molybdenite)	2.70		2.70
GR-119	350.7-350.95	Quartz Vein (heavy sulfide mineralization)	3.36		3.36
GR-119	352.5-352.75	Quartz Vein (moderate sulfide mineralization)	2.92	2.83	2.92
GR-122	295.3-295.5	Quartz Vein, 10-15% sulphides & WO_3	3.22		3.22
GR-122	at 295.0	Quartz Vein, 30% sulphides & WO_3	4.13		
GR-124	252.0-253.0	Quartz Vein #10; with py, W, cpy & po		2.62	2.62
GR-124	253.0-254.0	Quartz Vein 10 (?)		2.60	
GR-115	297.0-297.6	Quartz Vein 10		2.71	2.71
GR-115	298.2-299.0	Quartz Vein 10		2.61	2.61
GR-116	397.9-399.0	Quartz Vein 10		2.74	2.74
GR-117	302.8-303.8	Quartz Vein 10		2.61	2.61
GR-119	350.1-351.6	Quartz Vein 10		2.82	2.82
GR-119	351.6-353.1	Quartz Vein 10		2.68	2.68
GR-121	395.8-396.7	Quartz Vein 10		2.72	2.72
GR-122	294.7-296.0	Quartz Vein 10		2.91	2.91
GR-122	319.8-320.4	Quartz Vein 10		2.77	2.77
					2.81

Cavey and Gunning (2006) used a bulk density of 2.8 g/cc for a resource estimate at the Panasqueira mine in Portugal. At this deposit the wolframite is developed in sheets of flat-lying quartz veins. Mineralogically, this mine is similar to the Number 10 Vein so the choice of 2.8 g/cc is deemed appropriate.

14.7 Spatial Analysis

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength of the spatial similarity of a variable with separation distance and direction. The correlogram measures the correlation between data values as a function of their separation distance and direction. If we compare samples that are close together, it is common to observe that their values are quite similar, and the correlation coefficient for closely spaced samples is near 1.0. As the separation between samples increases, there is likely to be less similarity in the values, and the correlogram tends to decrease toward 0.0. The distance at which the correlogram reaches zero is called the "range of correlation," or simply the "range." The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample; it is the distance over which sample values show some persistence or correlation. The shape of the correlogram describes the pattern of spatial continuity. A very rapid decrease near the origin indicates short scale variability. A more gradual decrease moving away from the origin suggests longer-scale continuity.

Variography was conducted for the Number 10 Vein using Sage 2001 software. Directional sample correlograms were calculated for tungsten in this single statistical domain along horizontal azimuths of 0, 30, 60, 90, 120, 150, 180, 210, 240, 270, 300, and 330 degrees. For each azimuth, a series of sample correlograms were also calculated at 15° dip increments. Lastly, a correlogram was calculated in the vertical direction. Using the complete suite of correlograms, an algorithm determined the best-fit model. This model is described by the nugget (C0) which was derived using down hole variograms; one or two nested structure variance contribution (C1, C2), ranges for the variance contributions and the model type (spherical or exponential). After fitting the variance parameters, the algorithm then fits an ellipsoid to all ranges from the directional models for each structure. The lengths and orientations of the axes of the ellipsoids give the final models of anisotropy.

In general terms, the variogram models were difficult to generate, due to the low point count. The "best" variogram was obtained at azimuth 030 degree, dip 0 degree, which coincided fairly well with the overall orientation of the Number 10 Vein indicating a range close to 50 meter. The remaining variograms were very poor.

Due to the unreliable variogram analysis, the search parameters were defined with respect to the orebody geometry.

The model was interpolated with a 3-pass scenario, the first pass was sized to reach at least the next drill section spacing, along the main axis of the mineralization. A second and third multiplier was used to set the subsequent search dimension for Pass 2 and Pass 3, leaving the ratio between the X Y and Z axis consistent with the vein geometry.

Due to the undulating nature of the deposit, three sub-domains were delineated for the Number 10 Vein. The sub-domains allowed for the rotation of the search ellipsoid, in order to optimize the sample search with the orientation of the vein, without resorting to any unfolding methodology. This was not necessary for Number 10a and Number 6 Vein.

Table 14-7 lists the final values used in the resource model for the range of the major, semi-major, and minor axis.

Table 14-7: Sample Search Parameters

Vein	Sub-Domain	GEM ZXZ	Range (Major, Semi-minor, Minor)		
			Pass1 (m)	Pass2 (m)	Pass3 (m)
No10 Vein	Sub Domain 1	-71,74,0	50, 50, 15	75, 75, 18	180, 180, 22
	Sub Domain 2	-87,75,0	50, 50, 15	75, 75, 18	180, 180, 22
	Sub Domain 2	90,-66,0	50, 50, 15	75, 75, 18	180, 180, 22
No 10A Vein	N/A	90,-66,0	50, 50, 15	75, 75, 18	180, 180, 22
No 6 Vein	N/A	-75,67,0	50, 50, 15	75, 75, 18	180, 180, 22

14.8 Resource Block Model

The block model was constructed using Gemcom's GEMS version 6.3™ software. A non-conventional model was considered for this deposit. The model would have a matrix of 1 single block along the model row of 100 m wide using the percentage ore to resolve the tonnage. However, due to the presence of the Number 10a Vein in the hanging wall of the Number 10 Vein, and in consultation with the underground engineering team, a conventional model using a matrix size 2 x 10 x 10 m was selected for mining selectivity considerations and the density of the dataset. The model replicated the matrix used in the Wardrop study and allows for easier comparison.

The block model was defined on the project coordinate system (UTM - NAD 27 ZONE 21) with a 30 rotation - counter clockwise. Table 14-8 lists the upper southeast corner of the model, and is defined on the block edge.

Table 14-8: Block Matrix Definition

Vein and Fill Model	Parameters
Easting	492,060
Northing	5,270,850
Top Elevation	300
Rotation Angle (counter clockwise)	-30
Block Size (X, Y, Z)	2 x 10 x 10
Number of Blocks in the X Direction	260
Number of Blocks in the Y Direction	175
Number of Blocks in the Z direction	50

The rock type model was coded by combining the geology model code with the sub-domain code, controlling the search ellipsoid orientation. The 1000 series code represents the Number 10 Vein, series 1100 the Number 10a Vein and the 2000 series represents the Number 6 Vein. The sub-domains were simply assigned a code of 1 to 3. A block-model manipulation-script calculated the final rock type code by adding the sub-domain code to the main geology code.

14.9 Interpolation Plan

The only element modeled are tungsten (as WO_3%) and copper (as Cu%) using nearest neighbour and inverse distance squared interpolation routines.

The interpolation was carried out in a multi-pass approach, with an increasing search dimension coupled with decreasing sample restriction, interpolating only the blocks that were not interpolated in the earlier pass.

Pass 1 uses an ellipsoid search with 6 samples minimum, and 15 maximum.

Pass 2 uses an ellipsoid search with 4 samples minimum, and a 15 maximum.

Pass 3 uses an ellipsoid search with 2 samples minimum, and 15 maximum.

A maximum number of samples per hole did not apply in this model due to the single composite per holes per vein.

Copper was interpolated similarly but used a minimum of 0.0001% Cu essentially discarding the data location where copper was not assayed.

Number 10, 10a and 6 Veins were treated as hard boundaries to each other's meaning that only the samples within the vein were used in the interpolation of that vein.

All sub-domain boundaries within the Number 10 Vein were treated as soft boundaries, allowing samples from one sub-domain to be used in the interpolation of the adjacent sub-domain. This is the correct methodology, since the sub-domains were only used to control the orientation of the sample search ellipsoids, and do not correspond to any known lithological contact or fault.

No blocks were interpolated outside the wireframe and a series of special models were created to facilitate the resource classification and validation of the model.

14.10 Mineral Resource Classification

Several factors are considered in the definition of a resource classification:

- CIM requirements and guidelines;
- Experience with similar deposits;
- Spatial continuity;
- Confidence limit analysis; and
- Geology.

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to the author that may affect the estimate of mineral resources. Mineral reserves can only be estimated on the basis of an economic evaluation that is used in a preliminary feasibility study of a mineral project, thus no reserves have been estimated. As per NI 43-101, mineral resources that are not mineral reserves do not have demonstrated economic viability.

The sampling density allowed Indicated and Inferred category. A block was considered indicated if the search ellipse (pass 1) must have found at least six, and no more than 15, composites. In addition, the distance to the nearest composite has to be less than 50 m. All other blocks were considered inferred.

A special Code 4 exists in the model. This non-reportable NI43-101 category is solely used for drill targeting and to provide guidance to the exploration effort. Material in this category originated from the downgrading of inferred material in the fringe of the deposit outside the core of the Number 10 Vein.

Following a discussion with Playfair and the statistical analysis of the raw assays, all blocks with an Indicated category were downgraded to Inferred for the following reasons:

- Face, back, trench and bulk samples are typically not as good of quality as drill core samples, due to bias that can be introduced in the sampling process. In this particular case, due to recovery in the small diameter historical drill core, one can argue that the face, back and especially the bulk samples may give a more representative grade.
- No original assay certificates for the face, back, bulk and trench sample program were found to validate the grade of the samples.
- There is a known issue regarding the low-grade assays from the South Plainfield check sample program that remains to be investigated.
- The lower grade in the drill core samples when compared to the historical surface and underground samples. This is possibly related to the assay methodology that needs to be investigated further.
- The difficulty of capturing a representative sample in the historical small diameter EX and AQ core.

Additional diamond drilling and an underground check sample program is required to improve the confidence level of all categories in the model.

14.11 Mineral Inventory Tabulation

Effective December 29, 2011 Desautels Geoscience Ltd (DGL) has estimated the mineral inventory for the Grey River Property utilizing data from 52 diamond drill holes. The Mineral inventory estimate takes into account all drilling information from the historical drill campaign conducted by ASARCO in the 60's and the more recent drilling by Playfair Mining in 2006 and 2008. The drill data was supplemented with historical underground channel and face samples, surface trench samples along with results from the ASARCO bulk sampling program.

For the purpose of this report, an Inferred global mineral inventory is presented in Table 14-9 within the entire Grey River deposit which includes material from the Number 10, 10a and 6 Veins combined. Tungsten prices are usually quoted per metric ton unit ("MTU" = 10 kilograms or one hundredth of a metric tonne) of contained tungsten trioxide (WO_3). One MTU therefore contains 10 kilograms of WO_3 or 7.93 kilograms of tungsten metal. In the following tables, rounding of tonnes as required by reporting guidelines may result in apparent differences between tonnes, grade and contained metal.

Excluding mineralization grading less than 0.2% WO3 over a 1.0 minimum mining width, the updated undiluted mineral inventory indicated 1.2 million tonnes of Inferred mineralization grading 0.730% WO_3 containing 18.8 million pounds of tungsten trioxide or 853,000 metric tonne units (MTU). The bulk of this tonnage is in the Number 10 Vein which contained an Inferred mineral inventory of 1.06 million tonnes grading at 0.760% WO_3 for a total of 804,800 metric tonne units of tungsten trioxide as shown in table 14-9.

Table 14-9: Mineral inventory at 0.2% WO_3 Cut-off.

	WO_3% Grade cut-off	Tonnage (T)	WO_3 %	WO_3 lbs.	WO_3 MTU's
Vein 10	>= 0.2	1,060,000	0.760	17,743,000	804,800
Vein 10a	>= 0.2	87,000	0.478	916,000	41,600
Vein 6	>= 0.2	22,000	0.320	155,000	7,000
Total		**1,169,000**	**0.730**	**18,815,000**	**853,400**

There is a possible copper credit that is not included in the mineral inventory since the metallurgical test work done so far did not consider the extraction of copper and, there is a lack of copper assays in some of the historical holes. DGL estimates the copper values could range between 0.13% and 0.14% as determined by selecting the estimated copper grade in the model at the 0.2% and 0.5% WO_3 cut-off and comparing this grade to the average grade of the composites.

Table 14-10, 14-11, 14-12 and 14-13 show the global mineral inventory by veins with all mineralization grading less than 0.2% WO_3 over a 1.0 minimum mining width removed from the inventory.

The entire mineral inventory was subsequently exported to Golder's Engineering team for further economic assessment.

Table 14-10: Global Inferred Mineral Inventories

Vein	WO_3% Grade Cut-off	Tonnage (Tonnes)	WO_3 grade %	Contained WO_3 (pounds)	WO3 MTU's (Metric tonne units) **
No10 Vein No6 Vein No10a Vein	>= 4.0	7,000	4.656	691,000	7,000
	>= 3.0	12,000	4.150	1,101,000	31,000
	>= 2.0	28,000	3.153	1,972,000	50,000
	>= 1.0	192,000	1.532	6,470,000	89,000
	>= 0.9	269,000	1.365	8,084,000	293,000
	>= 0.8	361,000	1.233	9,803,000	367,000
	>= 0.7	458,000	1.130	11,407,000	445,000
	>= 0.6	627,000	0.999	13,808,000	517,000
	>= 0.5	796,000	0.905	15,894,000	626,000
	>= 0.4	927,000	0.841	17,189,000	721,000
	>= 0.3	1,058,000	0.781	18,210,000	780,000
	>= 0.2	1,169,000	0.730	18,815,000	826,000

Table 14-11: Global Inferred Mineral Inventory (Vein No10)

Vein	%WO_3 Grade Cut-off	Tonnage (Tonnes)	WO_3 Grade %	Contained WO_3 (pounds)	WO_3 MTU's (Metric Tonne Units) **
No 10 Vein	>= 4.0	7,000	4.656	691,000	31,000
	>= 3.0	12,000	4.150	1,101,000	50,000
	>= 2.0	28,000	3.153	1,972,000	89,000
	>= 1.0	192,000	1.532	6,470,000	293,000
	>= 0.9	260,000	1.379	7,902,000	358,000
	>= 0.8	347,000	1.246	9,523,000	432,000
	>= 0.7	439,000	1.141	11,051,000	501,000
	>= 0.6	603,000	1.007	13,378,000	607,000
	>= 0.5	765,000	0.912	15,375,000	697,000
	>= 0.4	879,000	0.852	16,509,000	749,000
	>= 0.3	987,000	0.797	17,351,000	787,000
	>= 0.2	1,060,000	0.760	17,743,000	805,000

Table 14-12: Global Inferred Mineral Inventory (Vein No 10a)

Vein	$\%WO_3$ Grade Cut-off	Tonnage (Tonnes)	WO_3 Grade %	Contained WO_3 (pounds)	WO_3 MTU's (Metric Tonne Units) **
No 10a Vein	>= 1.0	0	1.001	0	0
	>= 0.9	9,000	0.952	182,000	8,000
	>= 0.8	14,000	0.913	280,000	13,000
	>= 0.7	19,000	0.869	356,000	16,000
	>= 0.6	24,000	0.818	430,000	20,000
	>= 0.5	31,000	0.751	519,000	24,000
	>= 0.4	43,000	0.666	634,000	29,000
	>= 0.3	60,000	0.577	760,000	34,000
	>= 0.2	87,000	0.478	916,000	42,000

Table 14-13: Global Inferred Mineral Inventory (Vein No6)

Vein	$\%WO_3$ Grade Cut-off	Tonnage (Tonnes)	WO_3 Grade %	Contained WO_3 (Pounds)	WO_3 MTU's (Metric Tonne Units) **
No 6 Vein	>= 0.5	0	0.000	0	0
	>= 0.4	5,000	0.449	46,000	2,097
	>= 0.3	11,000	0.396	99,000	4,470
	>= 0.2	22,000	0.320	155,000	7,027

Figures 14.4 and 14.5 show representative views (plan and section) through the $\%WO_3$ grade model.

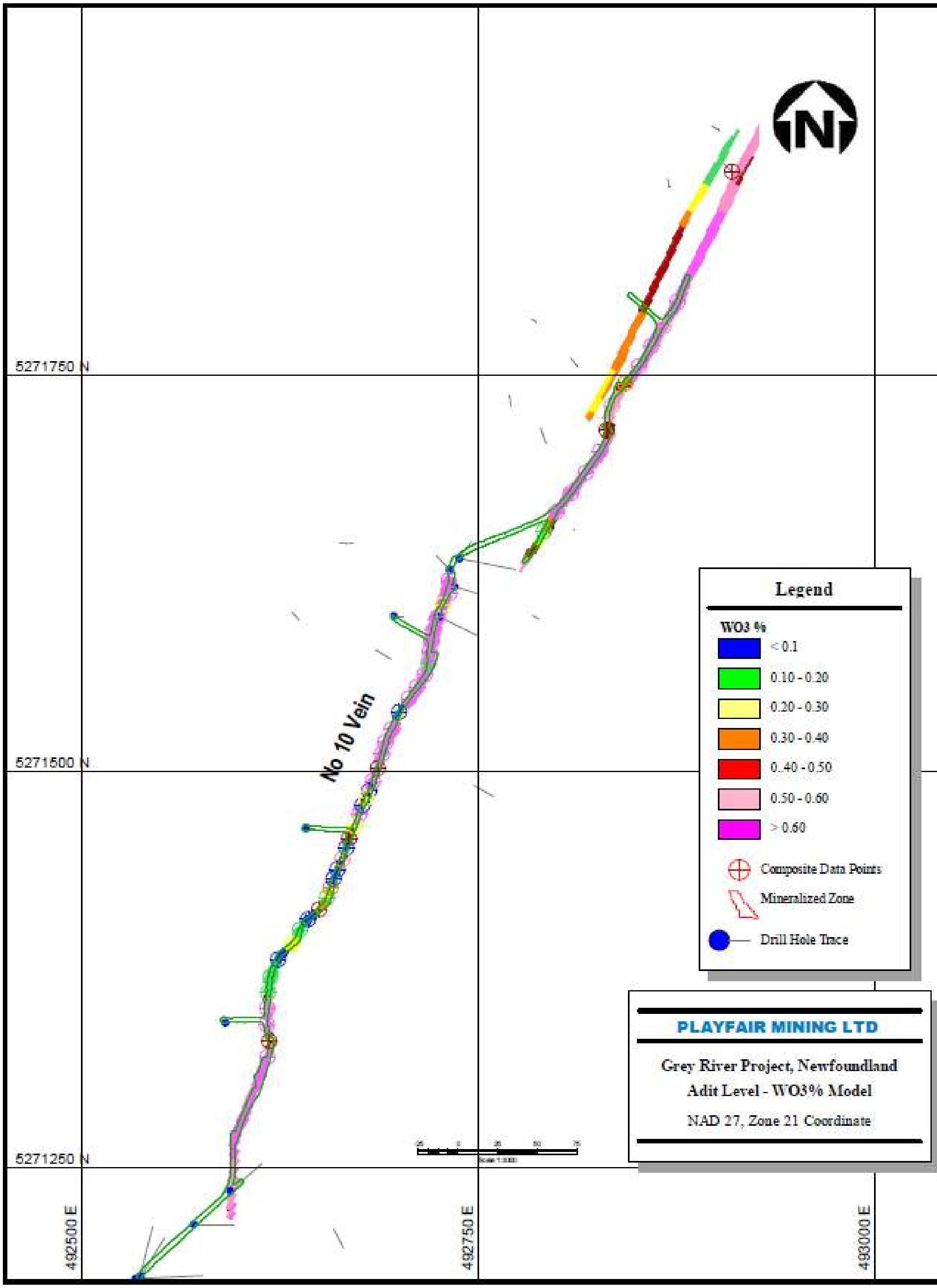


Figure 14.4: Representative plan view through the %WO_3 grade model (capped) at the drift elevation

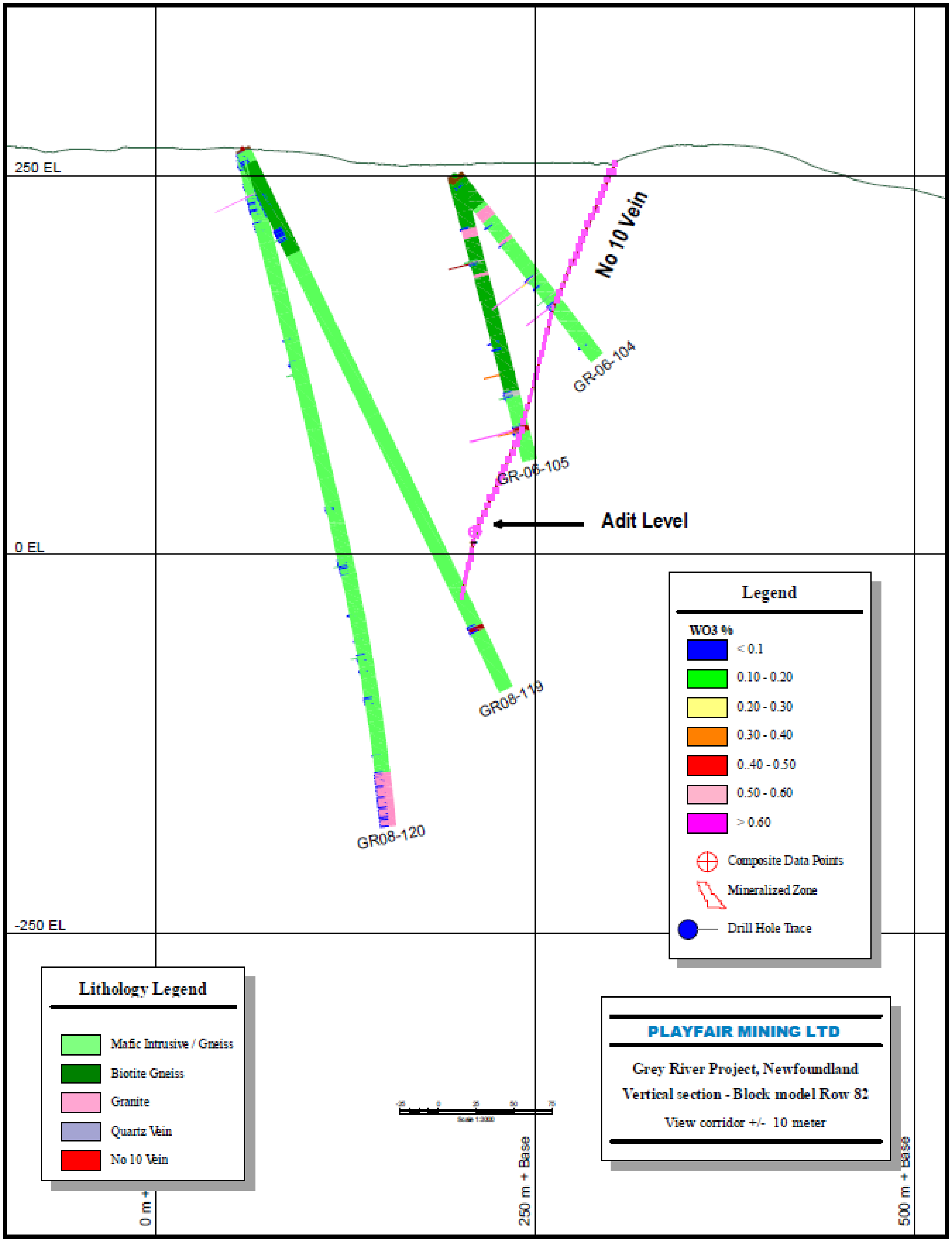


Figure 14.5: Representative section through the resource %WO_3 grade model (capped)

14.12 Mineral Inventory Validation

The grade models were validated by five methods:

1) Visual comparison of colour-coded block model grades with composite grades on section plots.
2) Comparison of the global mean block grades for inverse distance, nearest neighbour models, and composite and raw assay grades.
3) Comparison using grade profiles at 75 m spacing in the Y direction and 50 m spacing in the Z direction, looking for local bias in the estimate.
4) Naive cross validation test with composite grade versus block model grade.
5) Model compared with previous estimate.

14.12.1 Visual Comparison

The visual comparison of block model grades with composite grades shows a reasonable correlation between the values for the majority of the blocks. No significant discrepancies were apparent from the reviewed sections and plans. The orientation of the estimated grades on the sections follows well with the projection angles defined by the search ellipsoid.

14.12.2 Global Comparisons

The grade statistics for the raw assays, composites, nearest neighbour and inverse distance models were tabulated in Table 14-14 and Figure 14.6 shows the differences. Statistics for the composite mean grade when compared to the raw assay grade show a very small reduction in value attributed to the fact that the composites were not diluted to a minimum mining width. It is also a result of the fact that the entire vein width is sampled and the compositing process does not add any zero value from un-sampled interval. On a global basis, regardless of the methodology employed for the interpolation, the un-diluted composite grade average of 0.818% WO_3 is 20% higher than the interpolated grade of 0.667% WO_3. More importantly, the grade of the nearest neighbour and inverse distance at 0.00% WO_3 cutoff are within 3.9% of each other, showing that no global bias was introduced from the interpolation method used.

Table 14-14: Global Grade Comparisons at 0.00 Cut-off

Method	%WO_3 Capped
Raw Assays	0.838
Composite (un-diluted)	0.818
Nearest Neighbor	0.667
Inverse Distance	0.693

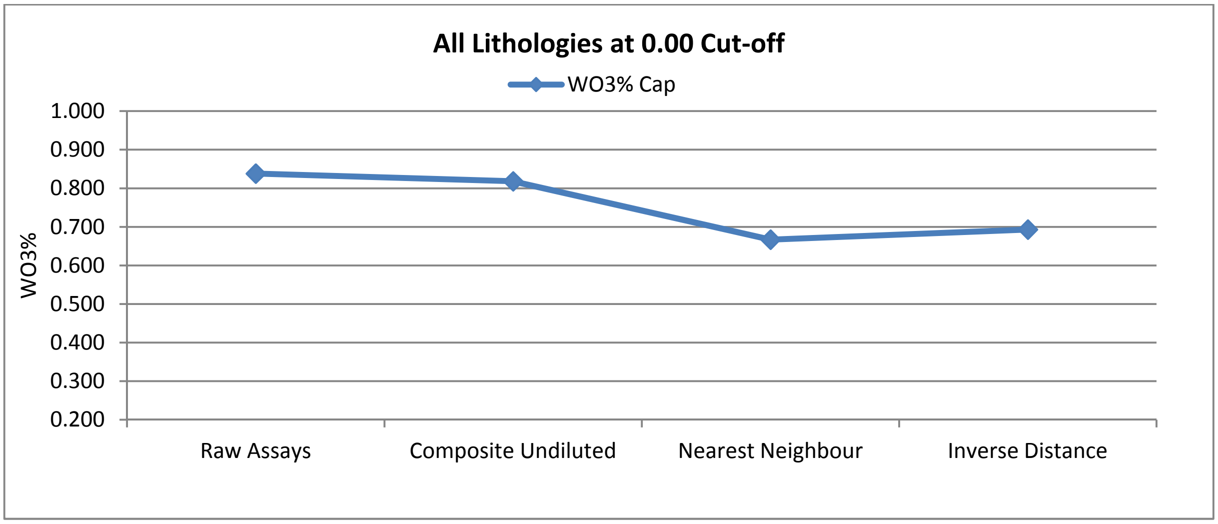


Figure 14.6: Global grade comparison at 0.00% WO_3 cut-off

14.12.3 Local Comparisons – Grade Profile

The comparison of the grade profiles (swath plots) of the raw assay, diluted composites and estimated grade for the main No. 10 Vein allows for a visual verification of an over or under-estimation of the model at the global and local scales. A qualitative assessment of the smoothing and variability of the estimates can also be observed from the plots. The output consists of three swath plots generated at 50 m intervals in the X direction, 75 m in the Y direction, and 50 m vertically for WO_3 capped model.

The Inverse distance estimate should be smoother than the nearest neighbour estimate; thus, the nearest neighbour estimate should fluctuate around the Inverse Distance estimate on the plots, or display a slightly higher grade. The undiluted composite line is generally located between the assay and the interpolated grade if there are a significant number of composites. A model with good composite distribution should show very few crossovers between the composite and the interpolated grade line on the plots. In the fringes of the deposits, as composite data points become sparse, crossovers are often unavoidable. The swath size also controls this effect to a certain extent; if the swaths are too small then fewer composites will be encountered, which usually results in a very erratic line pattern on the plots.

Due to the elongated nature of the No. 10 Vein, the best orientations for the swath plots are in the Y and Z axis, since the X axis is oriented parallel to the strike of the deposit, which is not ideal. The Wardrop un-diluted block model grade from the 2006 NI43-101 report was also plotted for comparison.

In general, the swath plots show agreement between the two interpolation methodologies used, with no major local bias. The assay line on the plots should normally be position above the composite line. In the charts presented in this report, this is not always the case. The assay line crosses over the composite line. The areas in questions were investigated and issue originated from the uneven raw assay sample length which has not been length weighted for the plots. On the Y Chart, the Wardrop capped %$W0_3$ model grade follows a similar pattern to the 2011 undiluted vein grade line confirming the use of an un-diluted composite in the Wardrop model. The areas South of 5,270,000 on the Y chart and between -30 and 30 elevation on the Z Chart are influenced greatly by the addition of the underground channel samples in 2011. Unfortunately the data on the chart does not show a clear trend as to the effect of the additional samples on the resource. Grade profiles are presented in Figures 14.6 and 14.7.

14.12.4 Naïve Cross-Validation Test

This methodology can be described as a statistical approach to the visual comparison method. It compares the average grade of the composites within a block with the estimated grade, which provides an assessment of the estimation process similar to the measured data. Pairing of these grades on a scattered plot gives a statistical evaluation of the estimates (see Figures 14.7 and 14.8). This methodology differs from "Jack-Knifing", which replaces a composite with a pseudo block at the same location. Jack-Knifing evaluates, and compares the estimated grade of the pseudo block against that of the composite grade.

It is anticipated that the estimated block grades should be somewhat similar to the composite grades within the block, without being of exactly the same value. The procedure typically returns a better correlation with inverse distance interpolation since the weights applied to the composite points are directly controlled by the distance of the composite to the block center.

A high correlation coefficient will indicate satisfactory results in the interpolation process, while a low correlation coefficient will be indicative of larger differences in the estimates and would suggest a further review of the process. Results from the pairing of the composited and estimated grades within blocks classified as inferred and pierced by a drill hole are presented in Figure 14.9. The R2 value shows a low correlation of 0.944 with no outliers removed. The composite versus Inverse Distance regression equation residuals were evaluated, and showed a distribution that is very slightly skewed positively.

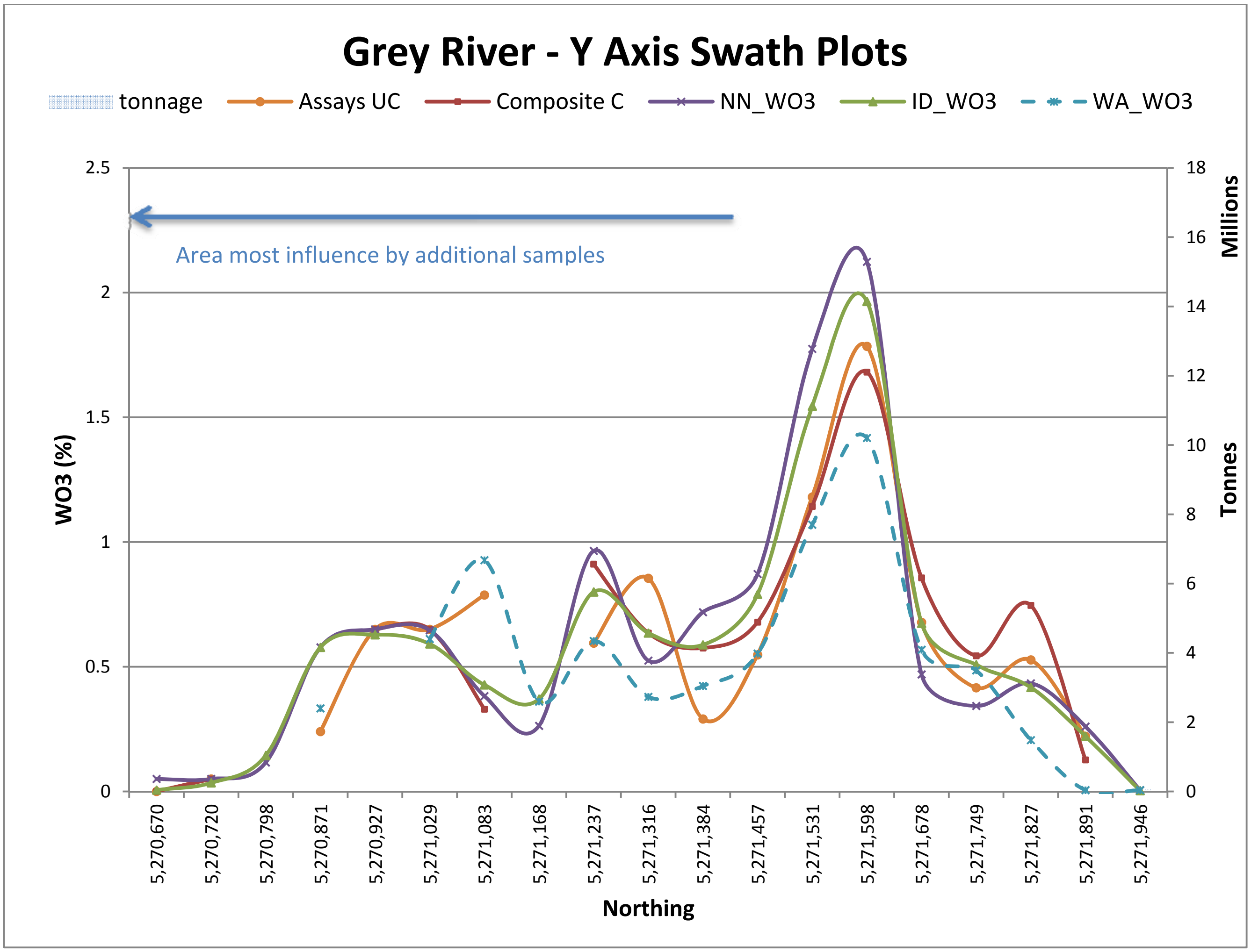


Figure 14.7: Y-axis swath plots for Vein 10 - Northing

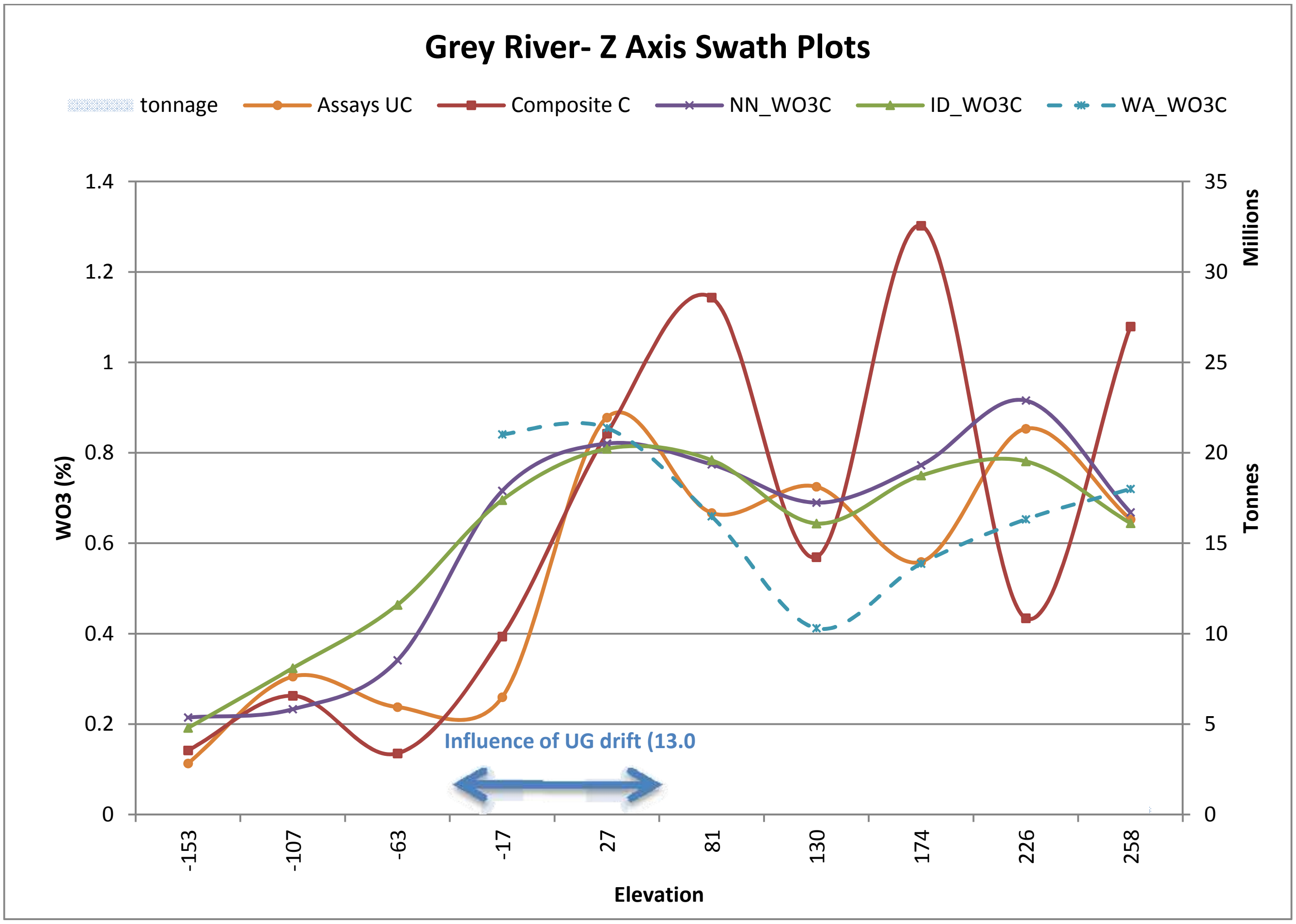


Figure 14.8: Z-axis swath plots for Vein 10 - Elevation

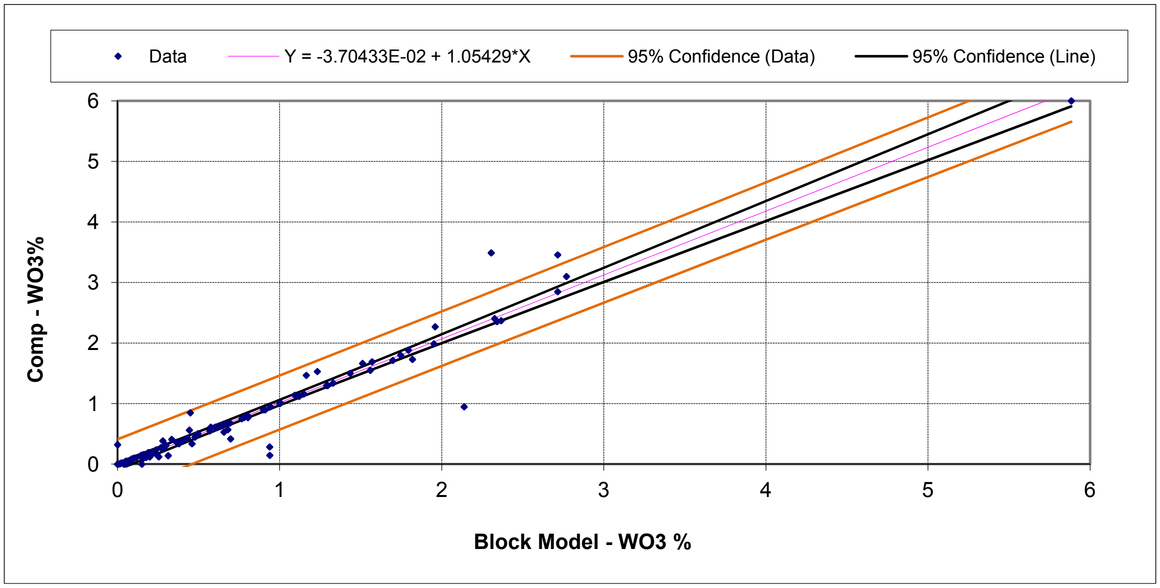


Figure 14.9: Naive cross validation regression results

14.12.5 Model Compare with Previous Estimate

The current resource was compared with the Golder Associate, grade estimate from PEA study dated January 15, 2008. The resource used in the PEA study replicated the resource published by Wardrop in June 2007, authored by Chris Moreton.

As shown in Table 14-15 at the greater than 0.2% WO_3 cut-off, the current global mineral inventory for vein 10, 10a and 6 returned 25% more pounds of metal that the Wardrop Resource. The grade is lower, 0.705% WO_3 in the current model versus 0.860% WO_3 in the Wardrop study. The changes are attributed to the inclusion of one additional sampling plan in 2011, which negatively affected the grade in the area of the ASARCO underground drift combined with the sample de-clustering methodology used.

Table 14-15: All Veins Compared to the Wardrop June 2007 Resource

	Desautels Geoscience 2011			WARDROP NI43-101			Metal Diff. (DGL-Ward.)	% Metal Change (DGL-Ward.)/Ward.
WO_3% Cut-off / Vein 10, 10a and 6	Tonnage (tonnes)	WO_3 %	WO_3 (lbs)	Tonnage (tonnes)	WO_3 %	WO_3 (lbs)		
>= 5.0	1,347	5.511	163,609	6,000	5.350	707,684	-544,075	-77%
>= 3.0	12,035	4.150	1,101,045	23,000	4.410	2,236,149	-1,135,104	-51%
>= 1.0	193,972	1.527	6,530,204	216,000	1.750	8,333,475	-1,803,271	-22%
>= 0.6	657,183	0.989	14,324,161	470,000	1.240	12,848,543	1,475,618	11%
>= 0.2	1,294,096	0.705	20,119,478	852,000	0.860	16,153,714	3,965,764	25%

Since Wardrop only modelled the No. 10 Vein, Table 14-16 compares the change in resources for that vein alone. At the 0.2% cut-off reported by Wardrop, the model returned 18% more metal, the tonnage increased by 39% or 331,000 tonnes. The grade was reduced by 0.130% WO_3 from 0.860% WO_3 in the Wardrop study to 0.730% WO_3 in the current model. The change in grade is directly attributed to the inclusion of a new sampling plan in 2011 which affected the grade negatively in the area of the ASARCO underground drift south of 5,271,000. The additional tonnage originated from the increase in size of the model below the Adit level originating from the new 2008 drill results.

Table 14-16: Vein 10 Compared to the Wardrop June 2007 Resource

	Desautels Geoscience 2011			WARDROP NI43-101			Metal diff. (DGL-Ward.)	% Metal Change (DGL-Ward.)/Ward.
$\%WO_3$ Cut-off Vein 10 Only	**Tonnage T**	**$\%WO_3$**	**WO_3 Lbs**	**Tonnage T**	**$\%WO_3$**	**WO_3 Lbs**		
>= 5.0	1,347	5.511	163,609	6,000	5.350	707,684	-544,075	-77%
>= 3.0	12,035	4.150	1,101,045	23,000	4.410	2,236,149	-1,135,104	-51%
>= 1.0	193,967	1.527	6,530,092	216,000	1.750	8,333,475	-1,803,383	-22%
>= 0.6	633,328	0.995	13,894,071	470,000	1.240	12,848,543	1,045,528	8%
>= 0.2	1,183,059	0.730	19,036,720	852,000	0.860	16,153,714	2,883,006	18%

15.0 UNDERGROUND MINING RESOURCES

15.1 Cut-off Grade

The cut-off grade for the Grey River Project was determined from the key economic parameters including production operating costs, metal price and milling recovery. The underground mining cost of $80 per tonne was estimated using experience with other similar projects and the MineCost model for North American underground mines as described in Section 20.1.1.

The milling cost used in the cut-off calculation is $11.50 per tonne milled and the derivation of this value is described in Section 20.1.2. General and administration costs and concentrate shipment charges are estimated to be $15 and $1 per tonne, respectively. These costs are also described in Sections 20.1.3 and 20.1.4, respectively.

A metal price of $355 per Metric Tonne Unit (MTU), or $16 per pound, is used in this study for the base case economic model. This value is shown in Figure 15.1 in relation to the three-year historic Tungsten APT prices as derived from the Bloomberg website for the period December 2009 to December 2011 (bloomberg.com). The prices at +30% ($465 per MTU) and -30% ($240 per MTU) are also shown on the Figure.

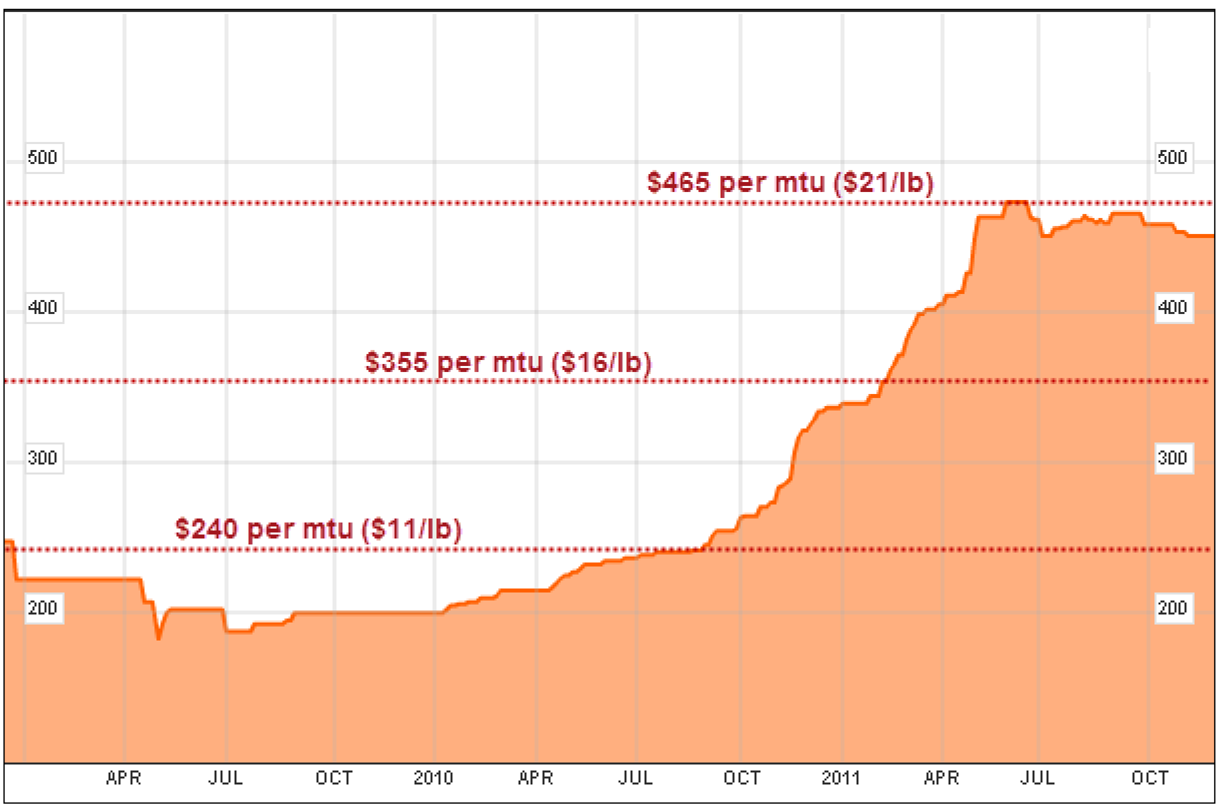


Figure 15.1: Three-year historic prices for Tungsten APT (USD/MTU). (bloomberg.com)

Table 15-1 summarizes the key economic parameters used to calculate a mining cut-off grade of 0.36% for the Grey River Project. A rounded cut-off grade of 0.35% was then used to determine the underground mining resources.

Table 15-1: Economic Parameters Used to Determine the Grey River Cut-off Grade

Parameter	Value	Units
Mining Cost	80	$US per t ore
Milling Cost	11.50	$US per t milled
G&A+Shipping	16	$US per t milled
Metal Price	16	USD per lb
Recovery	85	%
Selling Cost	0	$US per lb
Revenue	13.60	$US per lb
	300	per tonne
Cost	108	per tonne
Exchange Rate	1.00	CAD:USD
Cut-off Grade	**0.36**	%

15.2 Mining Recovery and Dilution

More detailed work is required to finalize expected mining recoveries and mining dilution. For this analysis a mining recovery of 95% was estimated. It is expected that mining dilution, as a percentage, would vary by vein thickness, however, an average overall value was used at this stage. An unplanned dilution factor of 15% at zero grade was applied to the conceptual mineable resource for production stopes and sill development. This dilution is expected to result from overbreak and stope wall failures. This dilution is in addition to the planned dilution that was incorporated in the conceptual mineable resource where the vein width was less than 2 m.

A minimum mining width of 2 m is proposed for the vein deposit. To calculate the planned dilution using this minimum width a mining stope model was created based on the geological resource model provided to Golder and described in Section 14.8. The methodology for creating this stope model is as follows:

- Horizontal slices were cut through the vein wireframe model at 20 m vertical intervals to reflect the sub-level interval or stope heights (a 20 m mineable crown pillar was included);
- Only blocks above a cut-off grade of 0.35% WO_3 were displayed and used to guide the stope definition;
- The stope hanging wall contact was then offset from the vein wireframe model by 0.5 m to reflect the fact that the miners must “see” the contact in the drift shoulder in order to follow the vein strike direction;
- If the vein wireframe width is less than 2 m then the stope footwall is offset 2 m from the hangingwall contact;
- If the vein wireframe width is greater than 2 m then the stope footwall is offset to the limit of the wireframe footwall; and

- The stope outlines on each 20 m vertical slice were then connected to form a single stope model solid.

The above methodology is depicted as a drawing in Figure 15.2 for vein widths less than, and greater than, the minimum mining width of 2 m.

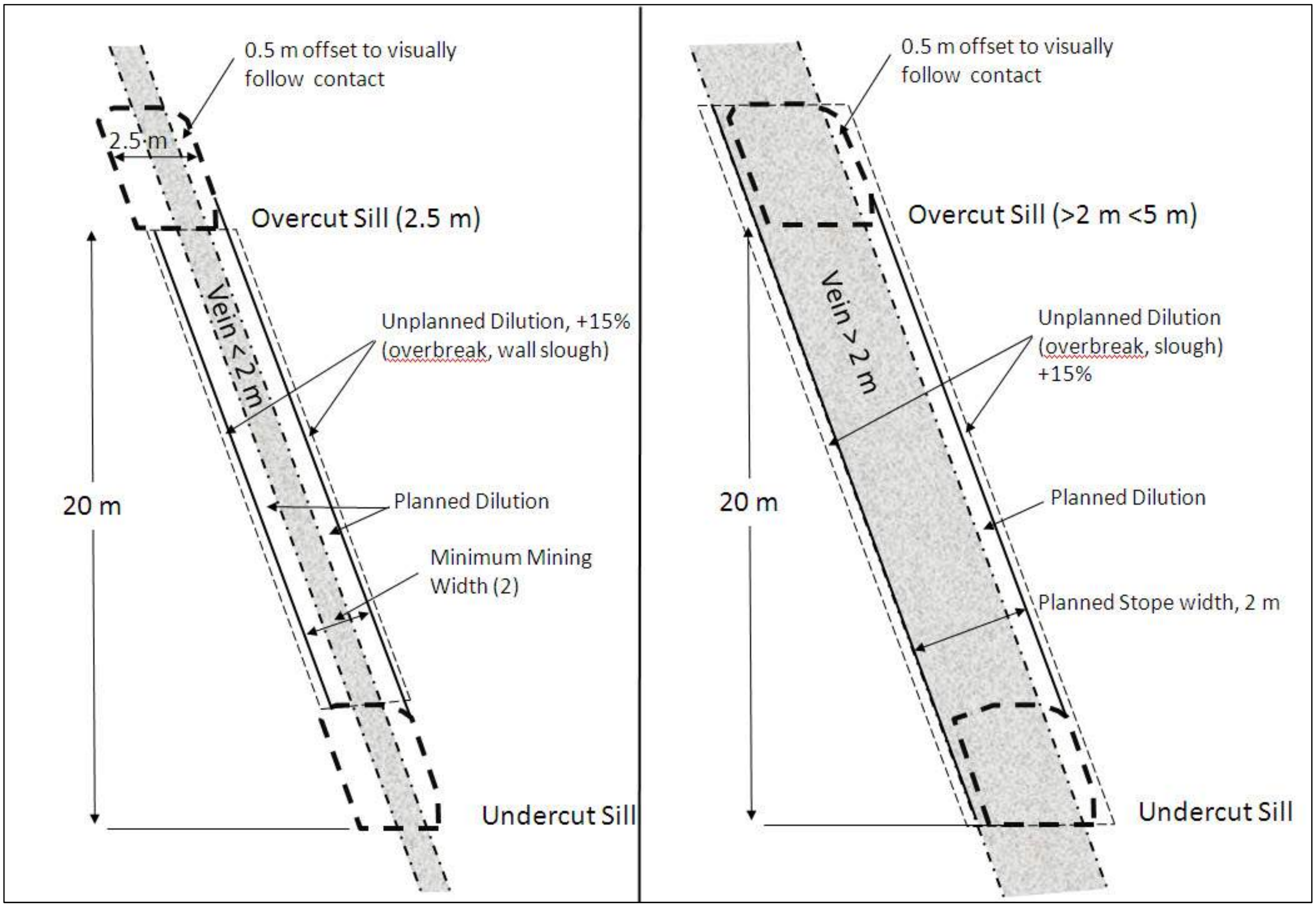


Figure 15.2: Methodology used to develop mineable LHOS stope shapes and dilution (planned and unplanned)

15.3 Underground Mining Resource

The underground mining resource was determined using the mineral resources as presented in Section 14. The DGL GEMS™ mineral resource block model was imported into Minesight™ software and a comparison of the resources is tabulated in Table 15-2 at a 0.30% cut-off.

Table 15-2: Comparison of GEMS™ Mineral Resource Model to Minesight™ Model at a 0.30% Cut-off

Model	Tonnes	Grade, %WO_3	Metal, Tonnes
DGL GEMS™ Model	1,157,700	0.759	8,785
Golder Minesight™ Model	1,158,689	0.759	8,791
Difference	*+0.09%*	*-0.02%*	*+0.06%*

The difference between the two models is less than ±0.1% and is considered acceptable.

The Minesight™ mineral resource block model is also shown in Figure 15.3.

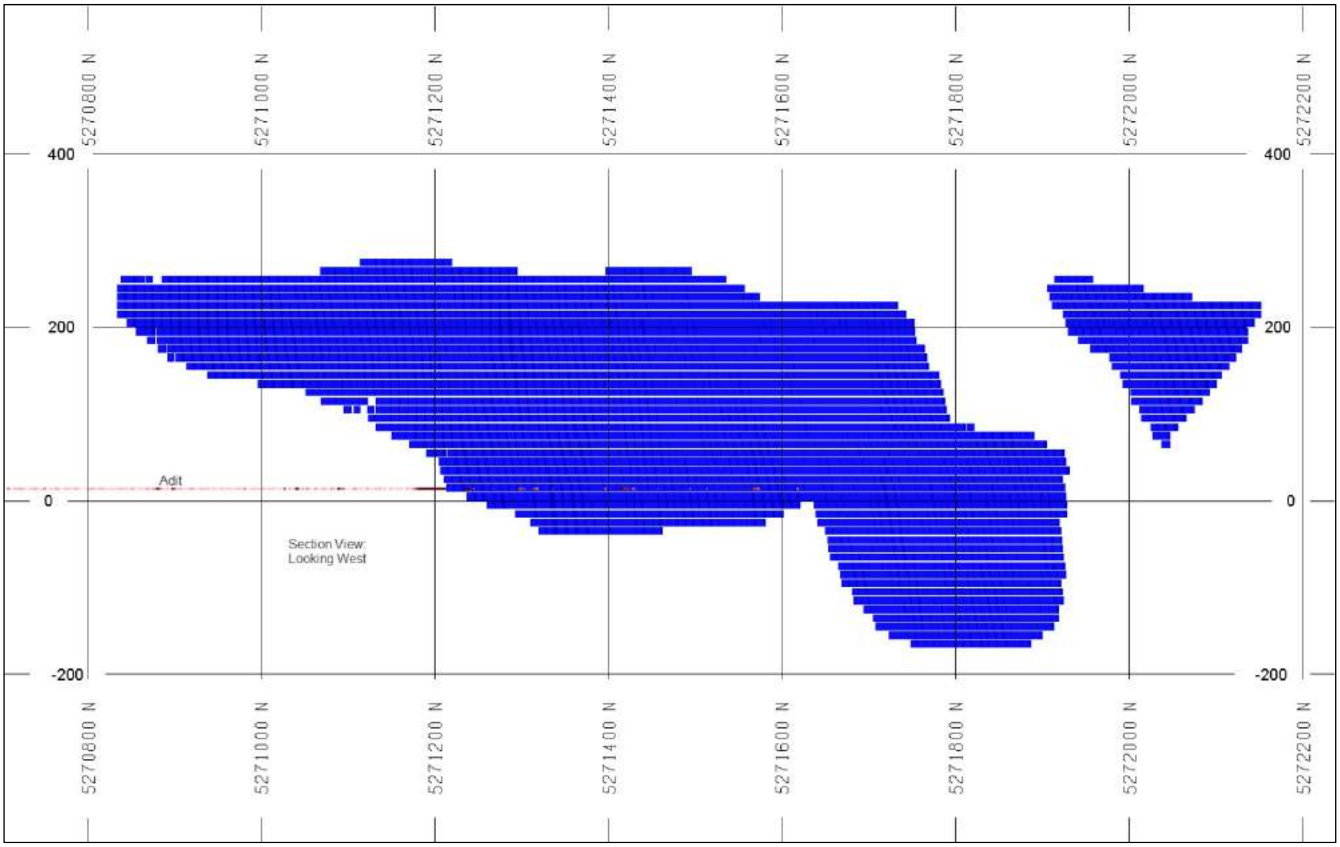


Figure 15.3: The Minesight™ mineral resource model used to generate the underground mining resources (no cut-off).

To define the underground mining resource the overall stope shape envelope created using the process described in Section 15.2 was divided into mining zones so that a preliminary production schedule could be produced. Figure 15.4 illustrates these zones as they are applied to the stope envelope (0.35% cut-off) and relative to the mineral resource model.

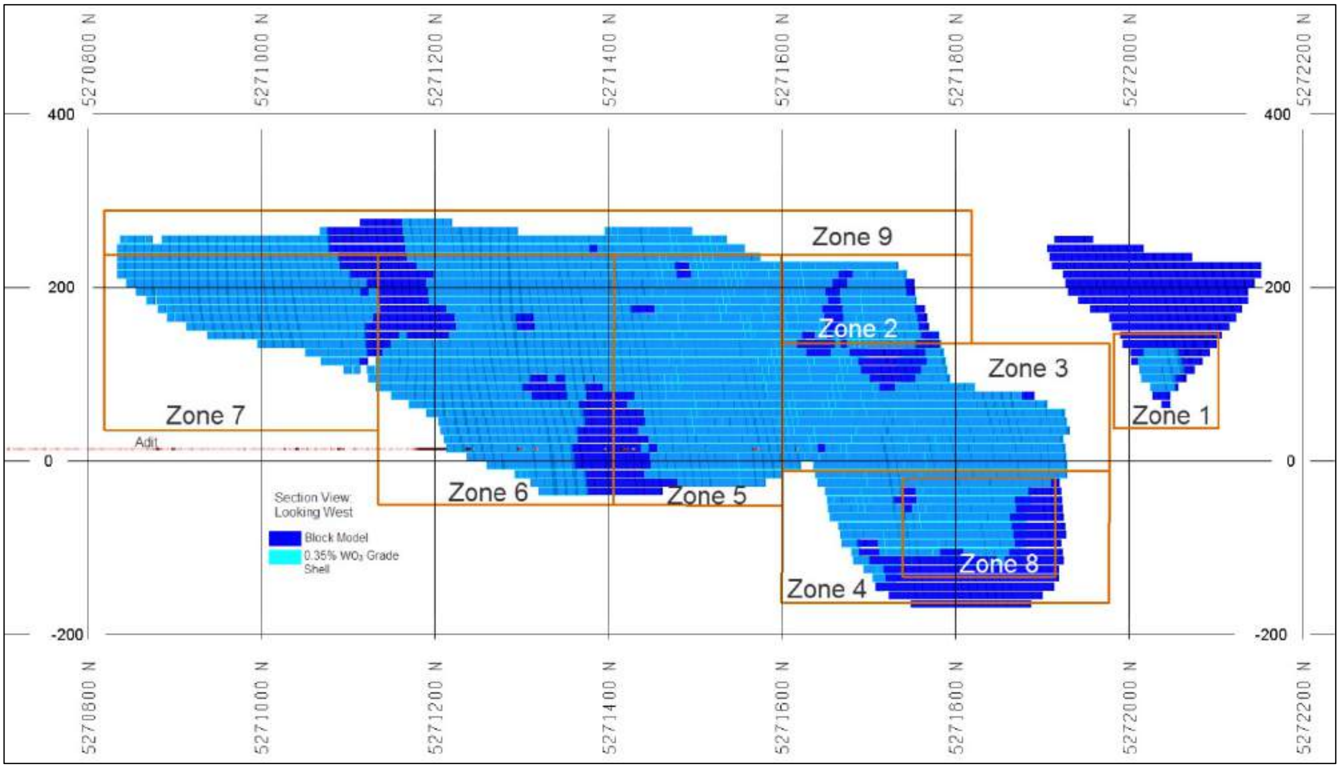


Figure 15.4: Potential mining zones above a 0.35% cut-off (light blue zones)

The mining zones were then further assessed for proximity to mining access and Zones 1 and 7 were both excluded due to their distance from the main ramp. Zone 7 would require either a separate portal and ramp or excessively long LHD haul distances of over 350 m. Zone 1 is too small to justify developing waste access to recover it. Finally, of the crown pillar (Zone 9), the portion above Zone 7 was also removed. The final underground mining resource is shown in Figure 15.5.

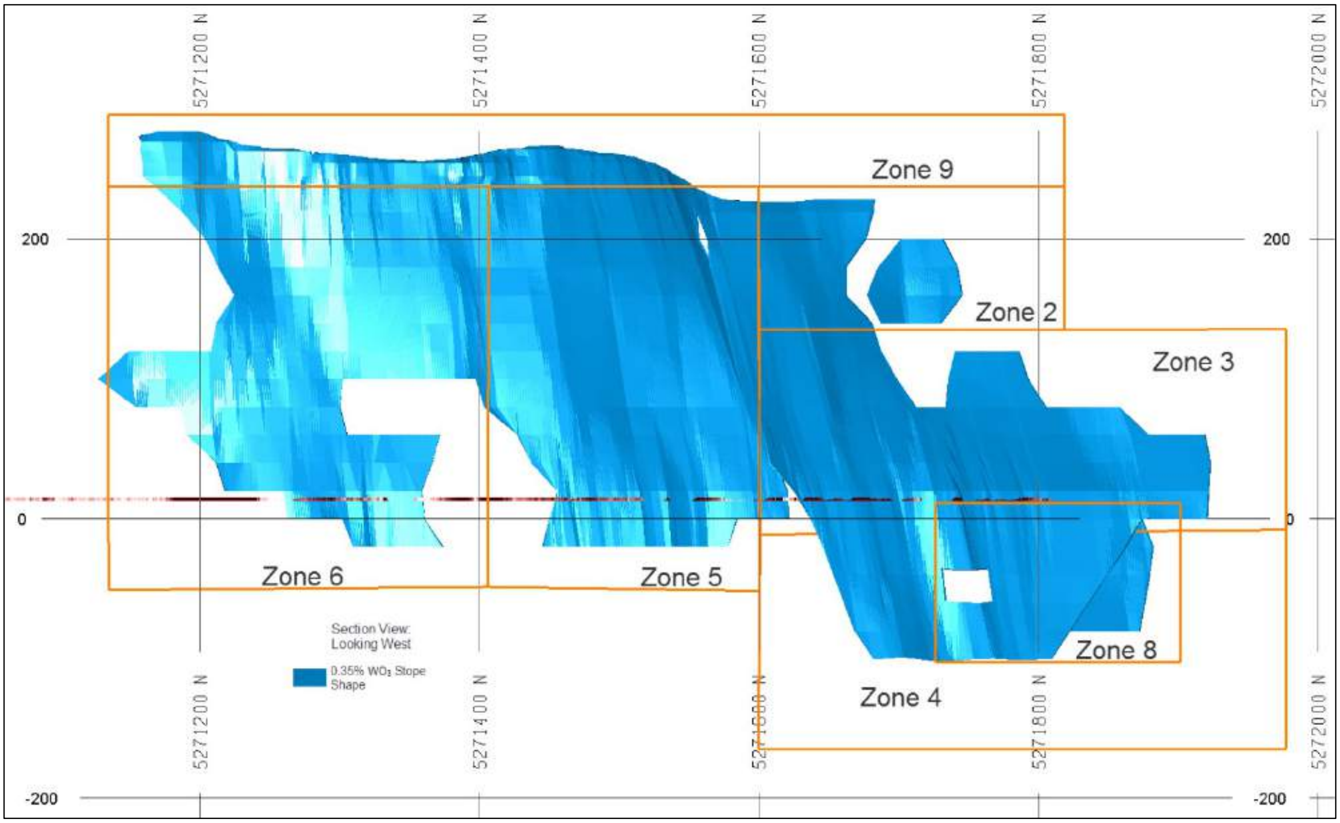


Figure 15.5: Underground mining resource zones used for mine design and economic analysis (Looking west and Zone 8 is offset to the west in front of Zone 4)

The final underground mining resource is also tabulated by zone in Table 15-3. These numbers were derived from the stope model and include all material and metal within the defined stope zones. So, even material below the 0.35% cut-off but within a defined stope shape would be included since, just as with waste material, there is no practical way to separate this material in the mining process. All material within the stope envelope is sent to the plant.

Table 15-3: The Total Underground Mining Resource by Zone (0.35% cut-off)

Zone	UG Mining Resources (tonnes)	Grade ($\%WO_3$)	Metal (tonnes WO_3)
Zone 2	87,126	0.581	506
Zone 3	191,975	0.480	922
Zone 4	163,564	0.366	599
Zone 5	436,988	0.586	2,561
Zone 6	292,107	0.467	1,364
Zone 8	62,075	0.416	258
Zone 9	101,224	0.781	791
Total Inventory	1,335,059	0.524	7,001
Total (95% Recovered)	**1,268,306**	**0.524**	**6,651 (14,662,945 lbs)**

The total mining resource used for underground mine planning and design and for the cashflow analysis is 1,268,306 tonnes at 0.524%. This number includes planned dilution (65%) and un-planned dilution (15%) and is 95% recovered from the mine.

This mining resource includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary economic assessment will be realized.

16.0 MINING

16.1 Geotechnical

Limited geotechnical data is available for the Grey River deposit at this stage. A review of ground conditions was completed by AMEC in December 2007 and concluded that the section of adit that was surveyed could be classified as having a "fair" rock mass rating under the Q system (Q = 7.5). In general, the rock mass examined is considered to be sound and was estimated to have an intact uni-axial compressive strength value of approximately 210 MPa. Also, numerous half-barrels were observed throughout and there was rare evidence of rockfall debris within the surveyed section (AMEC 2007). Sericitic alteration of wall rock is common on the hanging wall side of the Number 10 Vein which will require further geotechnical investigation in relation to maximum stope spans and potential for waste dilution. A more detailed geotechnical investigation and design exercise must be done for the next level of study.

Since the deposit outcrops at surface, a crown pillar 20 meters thick is assumed for this preliminary design. This pillar dimension is a conservative estimate based on the thickness of the mineralized zones at this elevation and empirical design methods for crown pillar design. This 20 m width is up to 3 times the width of the widest stope at the top of the deposit. It would also account for any weathered rock conditions and minimize any water inflow from the ground surface. The crown pillar remains part of the overall mineable resource since it could likely be mined to surface at the end of mine life. A more detailed geotechnical investigation and design exercise for the mining of this crown must be done in future studies.

16.2 Mining Method

The Grey River deposit is generally narrow-vein and steeply dipping with vein dip ranging from 70 to 80 degrees as shown in Figure 16.1.

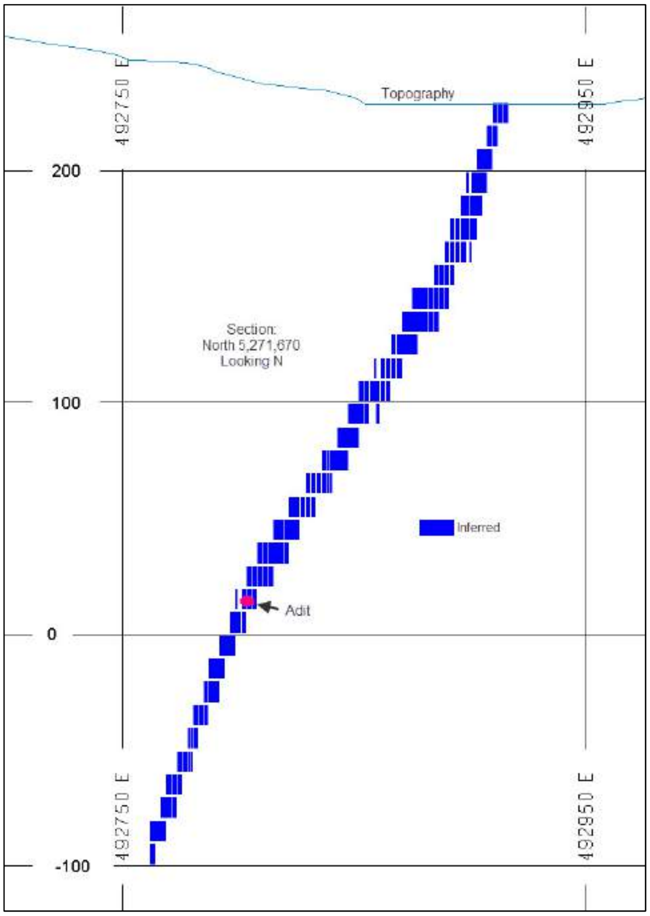


Figure 16.1: Cross section of the Grey River deposit illustrating the typical dip of mineralization

A longitudinal blasthole open-stoping method using delayed backfill was selected as the preferred mining method. Blasthole stope development would consist of sublevels on 20 metre vertical intervals (floor-to-floor). Stope development would include sill drives along the mineralized zone using 2.5 m by 3 m drifts. Slot raises would be developed in each stope panel to provide a free face for production blasting. As stated earlier in Section 15.2, a minimum planned mining width of 2 meters is used for stope dimensions. At this stage, it is envisaged that open stopes could be mined to three sub-levels high (60 meters) and around 15 meters along strike. Stope widths would vary depending on actual vein widths and range from 2 m to 4.5 m. Stope dimensions will have to be studied in greater detail as improved geotechnical data becomes available. A general depiction of the proposed mining method is shown in Figure 16.2.

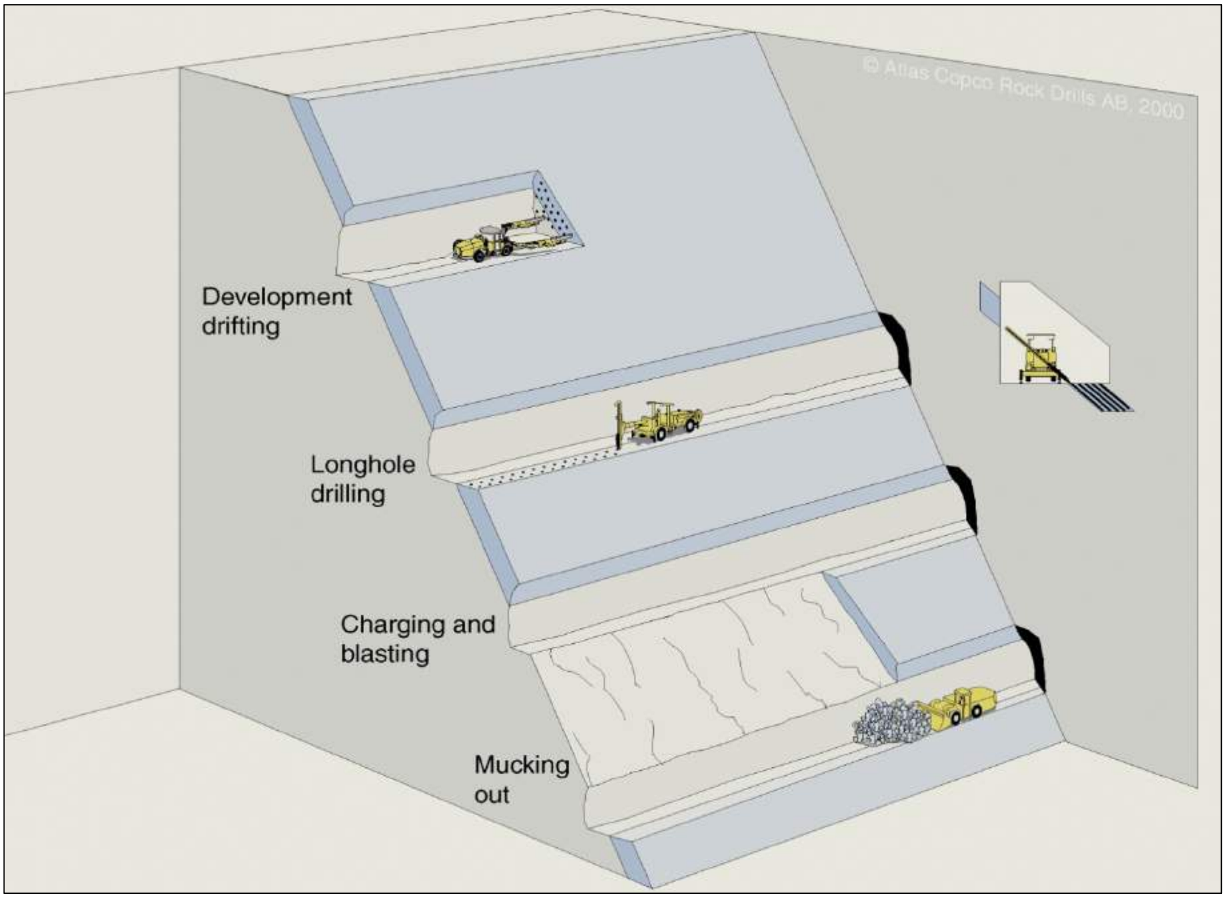


Figure 16.2: The proposed longhole open stoping mining method (Atlas Copco)

Access will be available at every third sub-level from the main decline, or at 60 m vertical intervals. Small longhole drills capable of drilling 64 mm diameter holes will be used to drill 18 m long holes along the vein (at an average dip of 75 degrees). Holes would be charged with packaged emulsion explosive and 3 to 6 blast rings will be blasted in sequence along the stope strike length, retreating from the outer regions of the deposit towards the center access location. Where possible the ramp access levels are located around the midpoint of each mining block to permit mining on both sides of the access. This is required to have up to 4 working fronts required to obtain the proposed production rate.

Blasted material will be removed at every third sub-level, or every 60 meters vertically, using small and narrow (1.5 m^3) diesel-powered load-haul-dump (LHD) machines. These loaders will transport the material along the sub-levels and out to a re-muck bay at the level access near the ramp. At this point, a larger loader (3 m^3) will re-handle this material and load it into 20-tonne haul trucks. These trucks will transport the material to surface within the main decline and then to the plant site located near the portal. Waste material from mine development will be handled similarly but will be trucked to a surface waste storage dump, also near the portal.

16.3 Mining Rate

The Grey River deposit has a narrow vein configuration that will limit the mining rate.

The proposed mining rate for the Grey River mine is 400 tpd or 146,000 tonnes per year. Figure 16.3 shows an empirical relationship between the tpd mined and the estimated tonnes per vertical meter of the underground mining resources. Another "rule-of-thumb" relationship is also plotted on Figure 16.3 and is based on a mining "rule-of-thumb" indicating that a maximum of 15 cm of a deposit can be mined vertically per day. The mining resources at Grey River have approximately 2,400 tonnes per vertical meter which translates to about 350 tpd from Figure 16.3. The data point for the proposed mine production of 400 tpd is plotted on the figure.

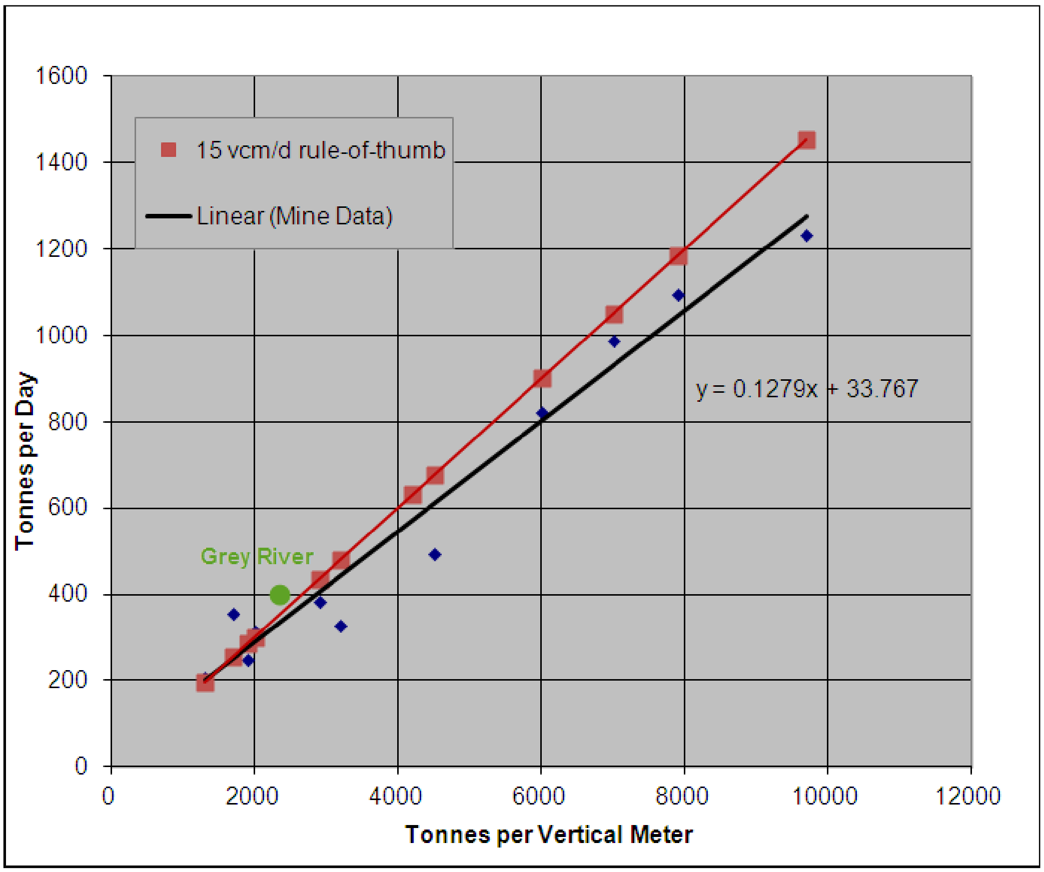


Figure 16.3: Relationship between tpd mined and the deposit vertical tonnes per meter (McCarthy, 1993)

Based on this data, the proposed production rate of 400 tpd is reasonable at this preliminary level of study. To obtain the proposed production rate it is estimated that a minimum of 4 workplaces will be required. These workplaces include all mining activities; drilling, blasting, mucking, support and backfilling.

16.4 Mine Design

16.4.1 Mine Layout

There is an existing underground adit at Grey River. It extends from Oceanside into the hillside within the host rock and then traverses into and along the mineralized vein. This adit was developed by ASARCO between 1966 and 1969 as an exploration drift and has been used to obtain bulk samples from the deposit. The total length of the drive is 1.9 kilometres and it has dimensions 2.5 meters by 2 meters. The mine plan uses the waste portion of this drift to provide an exhaust ventilation airway, as well as a secondary egress from the mine. Since the adit entrance is at the bottom of the cliff at Oceanside it would not be used for material haulage from the mine. Instead, a new decline with dimensions 4 m by 5 m would be developed from the surface of the deposit and down to the lowest mineralized extents to provide access for men, equipment and materials to each sub-level. This decline would be developed at a grade of 15% (1:6.6) and would be suitable for truck haulage using 20 tonne trucks (at dimensions 4.5 m by 4.5 m). Figure 16.4 shows an isometric view of the conceptual mine development.

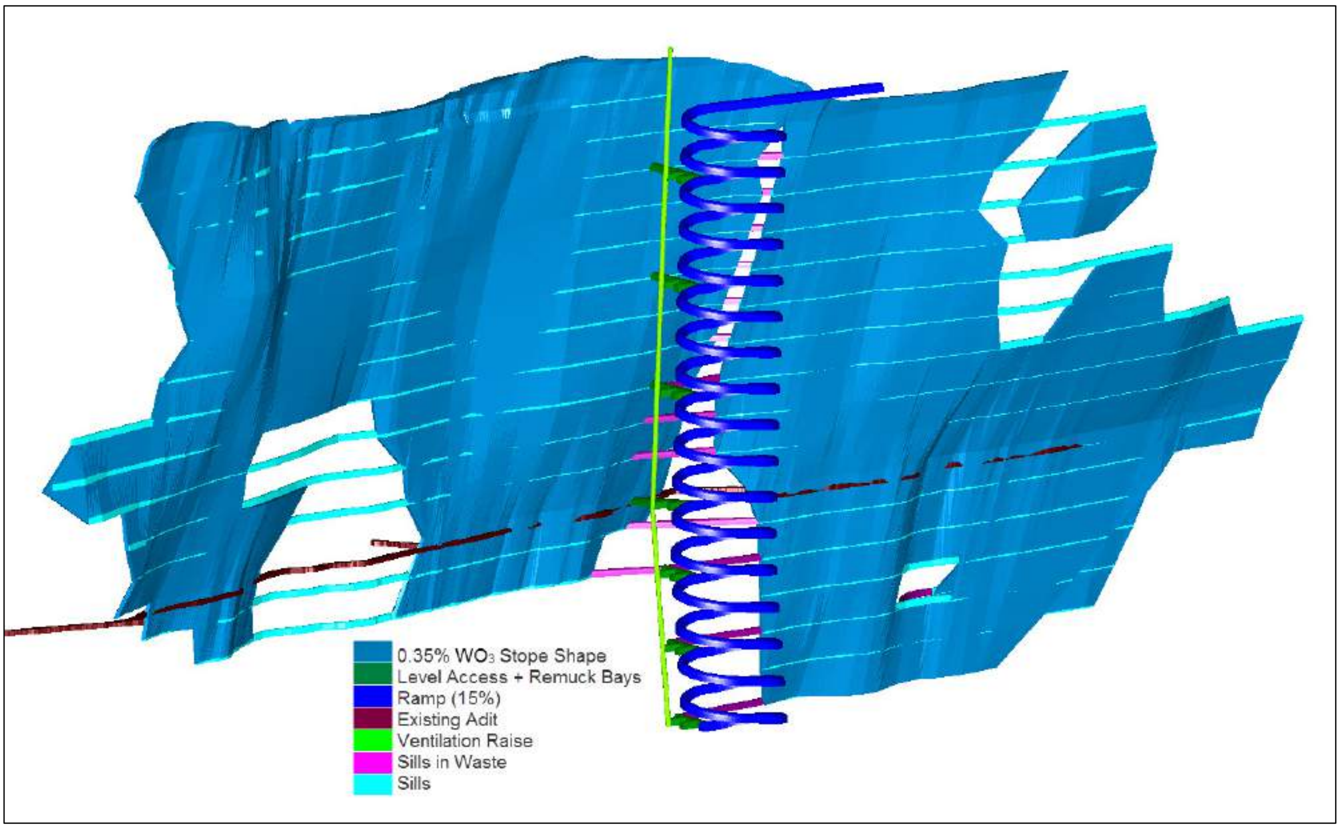


Figure 16.4: Conceptual mine development layout (looking northwest with no scale)

Mining the conceptual resource could commence once the main decline reached about the 200 m elevation (60 m below surface). At this point a bottom-up mining sequence could commence with trucks hauling mineralized material to surface and then to the plant site. At this elevation, good quality pastefill would be placed to permit mining from below. Mining on each sub-level will proceed from the outer extents inwards to the ramp access. Due to the relatively shallow depth of the deposit, ground stress induced problems are not expected in the final stopes (regional pillars) to be mined. However, this will require additional assessment at the next level of study. The next mining front could commence from 60 m below the first one and so on until the mining reached the bottom of the deposit. The ramp will be developed just in advance of mining.

The existing adit was considered for material haulage, however, the logistics of transporting material from the adit opening, and up the cliffside to the plant site were deemed unsuitable for several reasons including cost, exposure and visibility. For example, a cliffside hoisting system could be employed to move run-of-mine material up the hillside to the plant site. If the adit were used for primary haulage then the main decline could be developed smaller. This method would utilize raises within the deposit to transport mineralized material from the mining horizons and down to the adit elevation. Alternatively, a small underground hoisting system could also be used to move material to surface from the adit elevation. Trade-off studies on several possible material handling options could be completed in the next study.

Development quantities have been estimated based on the mine plan presented above to access each mineralized zone considered economically mineable. These quantities are summarized in Table 16-1. Sill development within the veins will amount to 220,621 tonnes or about 17% of the total underground mining resource. The majority of development waste will be hauled to surface and placed in a waste dump located near the portal. Further assessment is required to assess a suitable location for this dump and the potential for acid generation (Acid Rock Drainage, ARD) from this material on surface. Due to the planned mining and paste backfill methods there will be limited opportunity to dispose of waste rock in the mined out stopes.

Table 16-1: Estimated Total Life-of-mine Development Quantities

	Dimensions (m)	Total Length (m)	Total Tonnes
Main Ramp	4.5 x 4.5	2,555	129,547
Level Access	4 x 4	279	9,479
Ventilation Raise	2.5	383	5,598
Ventilation Raise Access	4 x 4	71	2,479
Re-muck Bays	4.5 x 4.5	71	2,479
Contingency	4 x 4	504	22,579
Total		**3,358**	**172,161**
Sills (in vein) (average)	2.5 x 3.5	10,080	220,621
Sills (in waste) (average)	2.5 x 3.5	680	15,290

Figure 16.5 shows a typical main level plan.

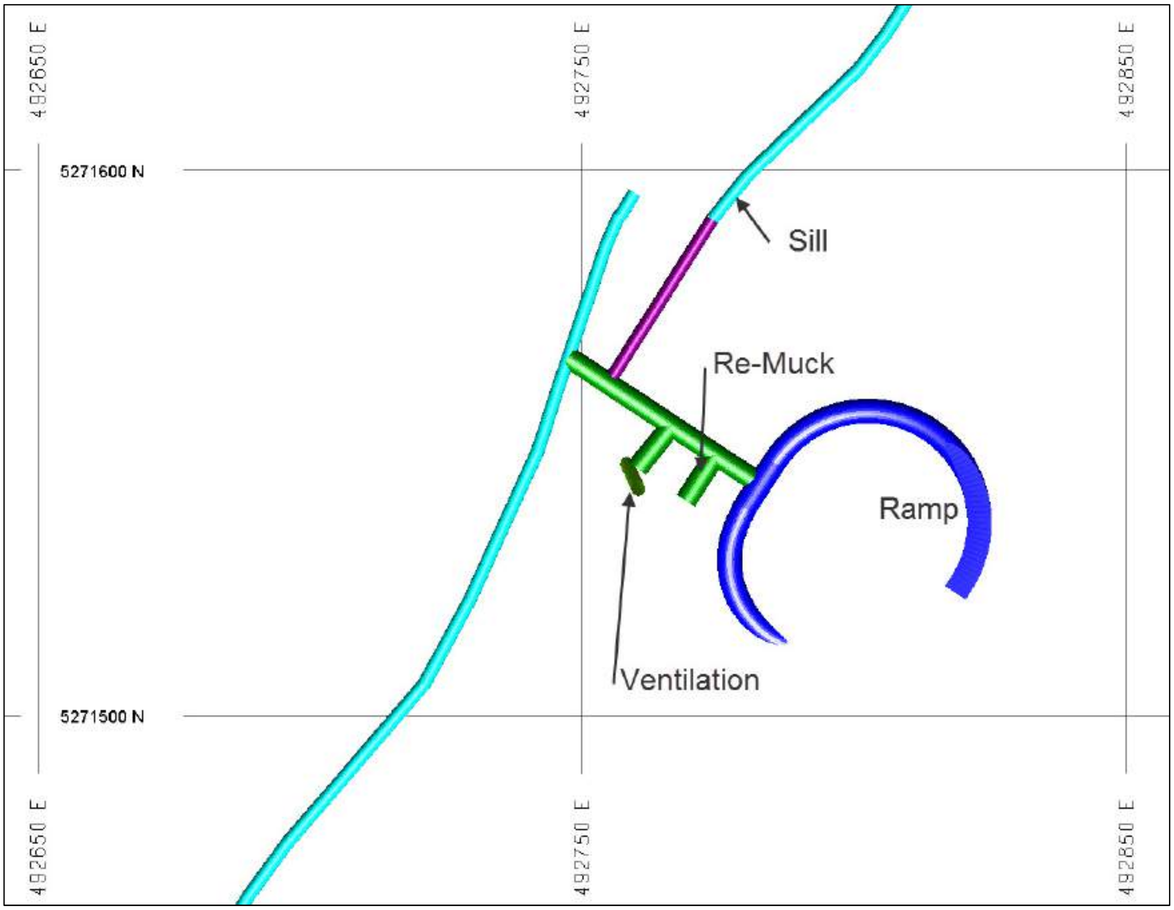


Figure 16.5: Typical main level plan

16.4.2 Ventilation

Mine ventilation is designed as a "pull-system" where a surface fan at the top of a 383 m long ventilation raise (~2.5 m diameter) "pulls" fresh air through the ramp and sub-level drives. The ventilation raise is situated in proximity to the main decline to provide ventilation during the development phase. The existing adit will be used as an exhaust opening and exhaust air would be divided between the main vent raise and the bottom adit. As the main decline commences development a raisebore machine or Alimak crew will develop the ventilation raise from the existing adit level to surface. A ladder way system will be installed inside the raise to facilitate personnel movement. This raise will then be accessed (developed to) as the main decline and level access development advances downward. This connection to the existing adit tunnel also provides a secondary egress for the mine before the main ramp also reaches the adit tunnel

Ventilation along the sub-levels will be done using smaller auxiliary fans and ventilation ducting that will flush ventilate to the ends of these drives. About 30 m^3/s of fresh air is required on each producing sub-level, sufficient for one 3 m^3 LHD and a truck. Total mine ventilation requirements would be expected to be in the range of 100 m^3/s (215,000 CFM) based on the diesel-powered mobile equipment needs (factored for utilization). The total un-factored estimate is 120 m^3/s. The estimate for air requirements is tabulated in Table 16-2.

Table 16-2: Estimated Total Mine Air Ventilation Requirements

Area	Equipment	Size	Quantity	Engine Size (kw)	Utilization	Ventilation Demand (m3/s)
Development	Jumbo	Single boom	2	110	33%	4.6
	LHD	1.5 m^3	1	72	65%	3.0
	Truck	20 t	1	375	100%	23.6
Production	LHD	1.5 m^3	1	72	65%	3.0
	LHD	3 m^3	1	150	65%	6.1
	Truck	20 t	1	375	100%	23.6
Service Vehicles	Scissor Lifts		2	110	50%	6.9
	Grader		1	110	50%	3.5
	Jeeps		3	110	65%	13.5
Contingency					15%	13
Total (m3/s)						**100**
Total (cfm)						**215,000**

** Ventilation factor used = .063 m^3/s/kwhr (100 cfm/bhp)*

16.4.3 Backfill

With limited geotechnical data or experience it is assumed that backfill will be placed in mined open stopes. The fill would act to stabilize stope walls, permit 95% recovery without leaving pillars, and provide regional stability to the mine. It would also keep stope mucking areas to a manageable size with less risk to, and more efficient use of, remote underground loaders. A preferred backfill material would be a cemented pastefill produced from the mill tailings. Further investigation and testwork will be required to confirm that the tailings are suitable for producing a pastefill. A paste backfill plant would be constructed on surface above the deposit and a pipeline would be used to deliver the paste to the mined stoping blocks as required. It is expected that 50% of the mill tailings could be sent back underground. This has the added benefit of reducing the size of the surface tailings deposition site.

16.4.4 Mobile Equipment

The mobile equipment requirements for the underground mine are listed in Table 16-3. This equipment is specified for narrow vein mining and is sized to fit within the drift dimensions presented in Section 16.4.1. The drilling equipment and small LHD's will need to be hoisted through raises onto captive sub-levels.

Table 16-3: Underground Mobile Equipment Requirements

Area	Equipment	Size	Quantity
Development	Jumbo/Long Tom	Single boom	2
	LHD	1.5 m^3	1
	Truck	20 t	1
Production	LHD	1.5 m^3	1
	LHD	3 m^3	1
	Truck	20 t	1
Service Vehicles	Scissor Lifts		2
	Grader		1
	Jeeps		3

16.4.5 Stationary Equipment

Stationary equipment and installations in the underground mine workings includes items such as fans, pumps, electrical distribution and pastefill line. A list of items is provided in Table 16-4.

Table 16-4: Underground Mine Stationary Equipment Requirements

Equipment	Size	Quantity
Electrical Distribution System	n/a	1
Pumps	11 – 45 kW	4
Main Fan	200 kW	1
Auxiliary fans	55 kW	4
Portable Refuge Chamber	16 person	2
Pastefill Line	400 m	2

16.5 Labour

Total site labour is estimated to be approximately 106 persons with 84 in the mine, 10 in administration and 12 in the plant. 32 persons are staff and 74 are hourly. The estimate is based on 24 hour/7 days per week schedule using a four-shift-on and four-shift-off crew rotation (two shifts per day). Some staff labour would be one shift per day only (engineering, office, etc).

16.6 Mine Production and Development Schedules

Detailed mine production and development schedules have not been generated for this preliminary study. The mine production schedule assumes 400 tpd for 365 days per year for 146,000 tonnes per annum. The plan is to commence mining in the upper levels in Zones 2, 6 and then 3. Then in Year 5, Zone 5 and 8 begin and Zone 4 commences in Year 6. The crown pillar, Zone 9, starts in Year 5 and continues until the end of mine life, Year 10. The grades associated to each zone were used to develop the preliminary schedule that generated the cashflow.

Mine development is completed as mining progresses downwards. That is, the main decline is extended below the current mining horizon in order to establish sills in the next stopes down. Main level access, re-mucks and ventilation access drifts are completed as the ramp descends. The main ventilation raise would be developed from the adit drift to surface when the main decline development starts. Then the raise would be developed into at each main access as the ramp descends. This will establish flow-through ventilation as each mining level is developed.

16.7 Mine Services

The underground mine will require compressed air and mine dewatering infrastructure in addition to the items listed below. The design of these items was not completed for this study.

Secondary Egress

Each mining zone requires two exit points; one main access/egress and the other in case of emergency. The mining zones are designed with ramp access and a ventilation raise connecting each main level. In addition, there will be small raises within the stopes and between sub-levels for personnel and equipment access. Secondary egress of the underground will be possible through the FAR down to the existing adit tunnel and then via the main ramp once it advances down to connect to the adit tunnel.

Refuge Stations

Portable refuge stations equipped with communications, emergency supplies and battery power will be positioned at a suitable location central to the mining activity. It can be re-located as mining progresses through the mine life. Two 16-person chambers will suffice for the underground workforce during one shift.

Shop

All of the major equipment repairs and routine maintenance will be handled by a shop on surface. The shop will share duties between the underground and surface equipment. A small repair bay will also be developed at a central underground location for small repairs and breakdowns.

Underground Communication

Communications underground will be through a leaky feeder radio system that will be installed in the main and internal ramps. At suitable locations (electrical substation, refuge stations) hard wired telephones will be also be installed.

17.0 PROJECT INFRASTRUCTURE

17.1 Site Layout Description

An isometric drawing of the proposed site layout is shown in Figure 17.1 and a plan view of the site layout is provided in Appendix C.

Figure 17.1: Isometric view of the Grey River project site infrastructure and underground mine workings (no scale)

The operation will be primarily travel-in, travel-out (with the exception of any local Grey River labour) with a camp and cafeteria on-site for all personnel. An access road is proposed from the community of Grey River to the top of the plateau where the site is located. This road will be about 2 km long at grades less than 10%. Other site access roads include access to the tailings facility, camp, plant site and mine portal. The total length of roads required is about 7 km.

Portable water, process water and water for other uses is required and will be drawn from nearby rivers and lakes. There is a Protected Water Supply nearby that is designated and regulated by the Provincial government. This approximate area is highlighted on the site plan figure. Overall site water management and hydrology studies will be needed at the next level of study.

All maintenance of mobile equipment will be done in a surface shop situated near the plant facility. A general warehouse will stock all materials required by the whole operation. An office building is required for managerial, administrative and technical personnel. First aid, training and security rooms are attached to the main building. A change house will be constructed for personnel lockers and a mine dry.

17.2 Electrical Power System and Site Distribution

The overall site power requirement is estimated to be about 2 MW with 1 MW for the underground mine, of which about half is needed for mine ventilation fans. The remaining 1 MW would supply the plant, camp and offices. For this study it is assumed that site power is generated on-site using diesel-powered generators. There is potential to tap into the power supply for the community of Grey River or to develop local hydro-electric generation. These options will require further study.

17.3 Other Site Facilities

A compressed air plant will be needed for the plant, mine and maintenance shops. Most underground mobile equipment will be diesel-powered, electro-hydraulic-type with on-board compressors. Underground operations and facilities will require a nominal supply of compressed air.

17.4 Waste Dumps

A 250,000 tonne waste dump (assuming a 30% swell factor) is situated close to the portal and will be used to permanently store underground development waste. There have been no acid mine drainage studies completed to date and an assessment of this waste material will be required.

17.5 Stockpiles

A small incremental stockpile will be located near the plant site. This is material that is excavated to access economic underground stope material, is below the economic mining cut-off, but is above the milling cut-off. It could be milled as low grade material at the end of mine life.

17.6 Tailings Management Facility

A tailings management area (TMA) will be required for the tails that are not sent underground as paste backfill. A possible location for these tails is identified on the site plan, however, no studies were done by Golder or others to confirm a suitable TMA. A conceptual tailings pipeline is also shown to transport the tails from the mill site to the TMA. It is estimated that there will be approximately 500,000 cubic meters of tails over the life of mine (assuming that 50% of the tails are sent underground as backfill).

18.0 MARKET STUDIES AND CONTRACTS

More than 20 tungsten-bearing minerals are known, but the principle minerals to produce ammonium paratungstate (APT) powder and tungsten metal powder are wolframite (the Grey River material), ferberite, and scheelite. The Grey River tungsten-bearing mine material will be upgraded to a concentrate on-site which will be shipped to an APT plant. Then it will be reduced to tungsten metal powder and further processed into tungsten carbide powder or ferrotungsten. Tungsten is produced from APT by reduction using hydrogen, followed by a second step using aluminum, potassium and silicon. The metal is then washed with hydrochloric acid and cast into ingots. End uses of tungsten include metalworking, mining, construction machinery and equipment, electrical and electronic machinery and equipment, transportation, lamps and lighting and chemicals.

The following section is from the website roskill.com and is a summary of their report titled "Tungsten: Market Outlook to 2016 (10th edition)" (August 2011):

"The tungsten market in the 1990s was characterised by oversupply from China and low prices, which meant that most western producers ceased production as prices were well below costs of production. However, in 2000 the Chinese government began the process of controlling its tungsten industry through the imposition of production and export quotas, and the removal of export rebates on tungsten products.

Over the years of excess supply, stockpiles of tungsten were built up by producers and also governmental organisations. These stockpiles overhung the tungsten market and tended to act as a brake on price rises. Most of the material contained in these stockpiles has now been sold and trends in tungsten prices have correlated more closely to the underlying supply/demand fundamentals since 2005/2006. Tungsten prices have risen strongly throughout 2010 and most of 2011, as most of the economies outside China recovered from the credit crisis-induced recession and demand for tungsten increased in parallel. At the end of September 2011, prices for Chinese APT had reached US$450-460 per metric tonne unit (MTU), compared to US$330 per MTU at the beginning of the year and US$200 per MTU at the beginning of 2010. APT prices are now well above the levels that were last seen in June 2005, when APT peaked at US$300 per MTU.

The outlook for the tungsten market is relatively positive as demand is expected to increase at almost 6%py to 2016, driven on by strong growth in China. Supply of tungsten will struggle to match demand growth at least until 2013, when some of the potential tungsten-producing projects are expected to begin production. However, any delays in commissioning of these projects would quickly see a growing deficit in the market with a resultant upward pressure on prices.

Very few of the significant new tungsten projects are expected to deliver any substantial tonnages of tungsten in 2012, so the market will be relying on existing producers to cope with any growth in demand. As a result, Roskill predicts a further tightening in supplies of tungsten and, therefore, further price rises, with an average APT price of US$475 per MTU. It is possible that the market will test the US$500 per MTU level at some point in 2012. Prices are then expected to ease between 2013 and 2015 as the bulk of planned new tungsten production capacity is expected to enter the market. Demand for tungsten is expected to continue to grow to 2016 and beyond, putting further pressure on the supply side and requiring more new capacity."

Playfair Mining Ltd. currently does not have any contracts to sell metal from the Grey River Project as it is still early in the development stages of the project. It is assumed in this study that tungsten concentrate, grading 65% WO_3, will be produced and shipped to an APT plant in either Europe or the American mid-west.

19.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section of the report was completed by Bruce Bennett of Stantec Limited who visited the Grey River project site in 2007.

The specific requirements for environmental assessment will ultimately be determined by the scope of the project, known, or assumed, environmental sensitivities, and public and stakeholder response to the project information in the registration document.

The following sections are provided to indicate what could be involved at different stages of the project.

19.1 Identification of Potential Issues and Baseline Studies

Following is a list of potential issues, in no particular order that could be raised in the environmental assessment process for the proposed project:

- Historic resources;
- Fish and fish habitat (freshwater);
- Fish and fish habitat (marine);
- Water resources (quality and quantity);
- Rare plants;
- Migratory birds (waterfowl and songbirds);
- Raptors;
- Vegetation/habitat type classification;
- Species at risk;
- Big game (moose, caribou, and black bear);
- Furbearers (otter, mink, Arctic hare, and others);
- Socio-economic environment; and
- Air quality.

Of these, we anticipate that baseline data will need to be collected for:

High Probability

- Fish and fish habitat (freshwater and marine);
- Water quality and quantity, and pond bathymetry (for proposed tailings/ sedimentation ponds if they are natural waterbodies);
- Historic resources (marine archaeology);
- Migratory birds (waterfowl and songbirds); and
- Rare plants.

Lower Probability

- Big game (caribou, moose and black bear); and
- Raptors.

The required level of effort will vary by issue and the project description will influence the profile of any issue.

19.2 Primary Issues

Two primary issues apply under most circumstances. The scope is determined by the size and extent of the proposed project.

19.2.1 Water Quality and Quantity

Detailed information is required on stream flow and the hydrological regime of water bodies that will be affected by the proposed project. This would apply to any requirements for water withdrawal or water releases. The provincially protected water supply for the community of Grey River will require careful consideration and cannot be impacted by the project.

Water quality sampling should be conducted on a regular or seasonal basis. Water samples should be analyzed for metals, mercury, nitrate, total organic carbon, sulphate, ammonia, total suspended solids and pH. The list could be reduced or expanded following a review of the initial sample results. It is recommended that baseline water quality conditions be documented by monthly or at least quarterly water sampling, pending discussions with government agencies. Sampling may be conducted by Playfair or by others if independent data collection is preferred. Either way, standard procedures for collection and analysis of samples will need to be followed.

19.2.2 Fish and Fish Habitat, and Pond Bathymetry

Current information on fish and fish habitat will be required for the environmental assessment and for the determination of harmful alteration, disruption or destruction (HADD) of productive habitat. This will apply to the marine area (if there is to be wharf construction or extension) and the ponds and streams at the top of the hill in areas to be developed for mining or processing.

A fish and fish habitat study for all affected water bodies would involve a review of available fisheries information followed by field studies as required. Potential field studies include determination of quality and quantity of stream habitat, (including water flow (depth and velocity) and substrate), and pond habitat. Sampling and surveys in July, August and September would likely represent natural seasonal hydrological variations at the project site.

Fish sampling in freshwater would consist of spot electrofishing selectively conducted at different locations in streams to determine the presence of fish and an indication of size class and areas of recruitment if fish are present. Index electrofishing (where the time fished is recorded and fishing effort is confined to a localized area) could be used to establish a comparative indication of fish density (i.e., standing stock). Short-term (tended) or overnight gillnet sets are conducted in ponds for the same purpose. Fish sampling would be conducted in July or August. Electrofishing is not permitted before 15 June, because fingerlings are still in the substrate and unable to swim about. To protect the breeding stock, electrofishing is not permitted during fall spawning. Areas that can be gillnetted are also restricted during the fall spawning period (after mid-September).

Surveys in the marine environment may be limited to visual surveys of the substrate (visual in shallow water and ROV or scuba in deeper water). This depends on the proposed activities in the marine environment and the presumed sensitivity of the area that will be affected. To determine fish (including shellfish) presence, sampling can be conducted or local informant knowledge can be obtained to characterize fish presence on a seasonal basis.

Marine aquaculture and marine fisheries, if present in the area will have to be considered from the perspective of potential disturbance by shipping or other activities and of potential for contaminant release (TSS, chemicals, fuel).

If Fisheries and Oceans Canada (DFO) determine that there will be a HADD (e.g., as a result of a tailings basin or marine wharf), compensation is usually determined on an area basis or sometimes on a productive capacity basis. Corresponding information will be required for the final determination of HADD and compensation. Habitat will be quantified using DFO guidelines and procedures that include bathymetry, substrate, flow and aquatic vegetation characterization. The HADD will be quantified on the basis of what is there and how much the Project will change things, based on the final project description.

19.2.3 Historic Resources (Stage 1)

While it may be unlikely that on-land archaeology surveys will be required, any proposal to infill marine areas for wharf construction could require an assessment of the potential for historic resources. There may be a requirement for surveys.

19.2.4 Rare Plants

A survey for rare plants within areas that would be affected by the project footprint might be required. Unique or rare vegetation communities and other areas with high potential to support rare plants would be targeted within the areas that would be physically disturbed. This work would need to be completed in mid-July to mid-August. The South Coast vegetative communities have been poorly studied, so it is difficult to judge the potential for issues at this stage.

19.3 Secondary Issues

These issues are no less important than those listed above; however, a smaller scale project (in size and extent) may not trigger a need to investigate original research or surveys on some of these issues. Some issues would likely be addressed through desk-top exercises and meetings with stakeholders and local informants.

19.3.1 Vegetation/Habitat Type Classification

To understand the existing environment and to assess the potential environmental effects of the proposed project on the terrestrial environment, terrestrial habitat classification might be required. The classification would be used to assess the relative availability of various habitat types in the region and within the area of the proposed project footprint. The classification would also be used to determine the scope of rare plant surveys and migratory bird surveys. Any existing vegetation inventory could be augmented by aerial photography and ground-truthing, as required.

19.3.2 Migratory Birds and Raptors

The proposed project may disturb habitat that provides nesting and forage for migratory birds. It is likely that a ground-based survey for songbirds (passerine birds), and an aerial survey for nesting raptors, particularly osprey and bald eagle, and waterfowl will be required. Surveys for migratory birds would need to be completed from mid-May to mid-June and from mid-June to July for raptors.

19.3.3 Species at Risk

There are species listed under the *Species at Risk Act* (*SARA*) and the provincial *Endangered Species Act* that may use habitat within the project area. Under Section 33 of *SARA*, damage or destruction of the residences of a species that is listed as endangered or threatened is prohibited. However, the minister having authority can enter into an agreement or issue a permit to engage in an activity affecting a listed wildlife species. Similar restrictions are identified under provincial legislation and would apply on all Crown lands. The implication of these *Acts* with respect to development of the proposed project would depend on the results of baseline data collection for listed species and the assessment of project effects. The need for dedicated baseline data collection for SARA species is not anticipated; however, this should be confirmed through the proposed consultation with the Inland Fish and Wildlife Division and Canadian Wildlife Service.

19.3.4 Big Game

There could be a requirement to assess potential effects on big game in the area, particularly for activities on the plateau. Focus could be on moose and caribou, but may also include black bear.

Moose

Moose would be present in suitable habitat throughout the area. The most sensitive issue with moose would be disturbance to wintering yards (areas where moose congregate during the deep snow periods). These are generally within forested areas and offer shelter and regenerating balsam fir for forage. It is assumed that a large amount of forested area will not be removed as a result of the project. However, if a large forested area is proposed to be removed, a winter aerial survey for moose yards may be required. Such a survey would be conducted in late February.

Woodland Caribou

The area of the proposed project is east of the identified range of caribou from the Buchans Plateau. Boreal woodland caribou are listed as Threatened under Schedule 1 of the *SARA*. However, under the provincial *Endangered Species Act*, only Labrador woodland caribou populations are listed as Threatened. Regardless, caribou populations on the Island are managed for hunting and are considered sensitive to disturbance during critical periods of their life cycle. Therefore, a requirement to consider caribou in an environmental assessment is likely.

The South Coast Barrens provides wintering habitat for caribou from the Middle Ridge and Sandy Lake/Grey River areas. Consultation with the Inland Fish and Wildlife Division would be required to determine the latest distribution information for caribou. The likelihood that a field program will be required may depend on the size of your proposed project; how much area, how many people and how does that compare with current activity levels such as snowmobiling, hunting, etc.

19.3.5 Socio-Economic Environment

An overview of the socio-economic environment in the area would obviously focus on Grey River because of its isolation. Aspects of the socio-economic environment to be considered would include labour and economy, employment equity, and land use, among other potential issues. Most required information should be available from existing sources including Statistics Canada, DFO, regional development groups, and provincial government databases.

20.0 CAPITAL AND OPERATING COSTS

20.1 Operating Cost Estimates

The operating cost estimates for mining, processing and general site administration are considered accurate to +/-40% in this preliminary study. Detailed first-principle engineering cost estimation has not been done and no actual quotations for equipment or consumables were used. The estimates are based on models, experience from similar operations and preliminary calculations.

20.1.1 Mining Costs

The underground mine operating cost was estimated from the MineCost model for Canadian underground mines. (MineCost 2009). The proposed longhole open stoping mining rate is 400 tpd, or 146,000 tonnes per annum, which is at the lowest extents of these models. The proposed method is also less of a bulk method and nearer to a cut-and-fill approach (very small drifts, small equipment and captive sub-levels) so the costs are likely to be higher than the Open Stoping data in Figure 20.1. A cost of $80 per tonne is estimated for this study as indicated in the figure by the red square.

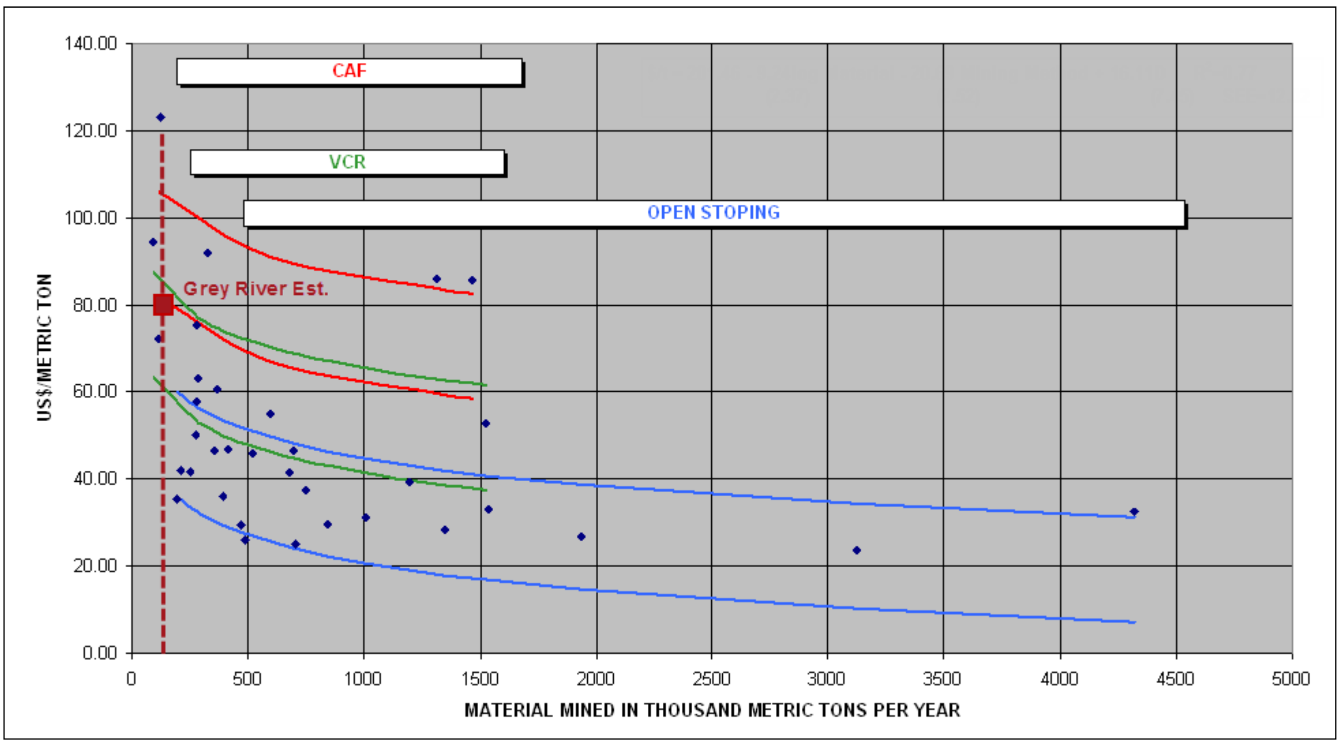


Figure 20.1: Canadian underground mining cost model data (MineCost 2009)

20.1.2 Processing Costs

Estimated operating costs for the concentrator include for power, water, consumables and labour. Unit costs are based on the basic design criteria using estimated labour and power rates factored for the planned capacity of the Grey River Project. Power supply is presently planned via diesel-powered generators located on site. Power costs are thus estimated for an estimated load of 410 kW (installed) at a diversity factor of 0.6 with no load factor and a power cost of $0.28/kWh for diesel-generated power. Labour costs include three operators, one supervisor and two maintenance personnel seven days per week. Consumables and spares are accounted for at 5% of initial capital cost per annum. The process plant cost breakdown is provided in Table 20-1.

Table 20-1: Process Plant Cost Breakdown

Item	$/t Milled
Diesel Power (410 kW installed)	4.10
Water	0.40
Labour	4.00
Consumables and Spares	3.00
Total	**11.50**

20.1.3 General and Administrative Costs

General and administration costs include general site staff (manager, clerks, accounting, purchasing and human resources), office and camp costs, and annual safety, security and environmental costs. These are estimated to be $2.2 million per annum or about $15 per tonne milled.

20.1.4 Refining and Transportation Costs

The APT product produced from the Grey River site is a saleable product and so further refining charges are not considered in this study. There will be costs for transporting this product to an APT plant in either Europe or the American mid-west. Three potential transport options are possible from the Grey River property situated on the southern coast of Newfoundland and with access to the St. Lawrence shipping route; short-range coastal transport to an Atlantic seaboard port, followed by rail transport overland to the American mid-west; shipping through the Great Lakes to the mid-west; or, transatlantic shipping to potential customers in Europe. Annual shipments from the mine would be less than 1000 tonnes of concentrate so sealed 23-tonne metal shipping containers would best suit the low tonnage, high value product. These containers also facilitate transporting by ship, rail and even by truck near the final destination.

For costing purposes a containerized cargo shipping rate between the Eastern US and continental Europe of $122 per tonne is used (Infomine Cost Database, 2011). This translates to a cost of $0.60 per tonne milled. In addition there will be costs associated with wharfage fees, drayage to loading, fuel surcharges, any overland trucking and insurance. A total shipping cost of $1.00 per tonne milled is assumed.

20.2 Capital Cost Estimates

20.2.1 Site Infrastructure

The estimated site infrastructure costs to develop the Grey River Project are presented in Table 20-2.

Table 20-2: Site Infrastructure Capital Cost Estimate (LOM)

Item	Cost (USD)
Access Roads	$3,000,000
Rock Storages	$750,000
Dock Upgrades	$500,000
Main Power System	$500,000
Camp Facility	$3,000,000
Generators	$350,000
Compressors	$180,000
Tailings Area	$4,000,000
Contingency (20%)	$2,456,000
Total	**$14,736,000**

20.2.2 Mine

The total life-of-mine development is shown in Figure 20.2 and consists of a main ramp, level access, ventilation infrastructure (access and raise), re-muck bays for truck loading and sills in both the stopes and in waste. All of the sill development is considered as an operating expense and all other development is capitalized.

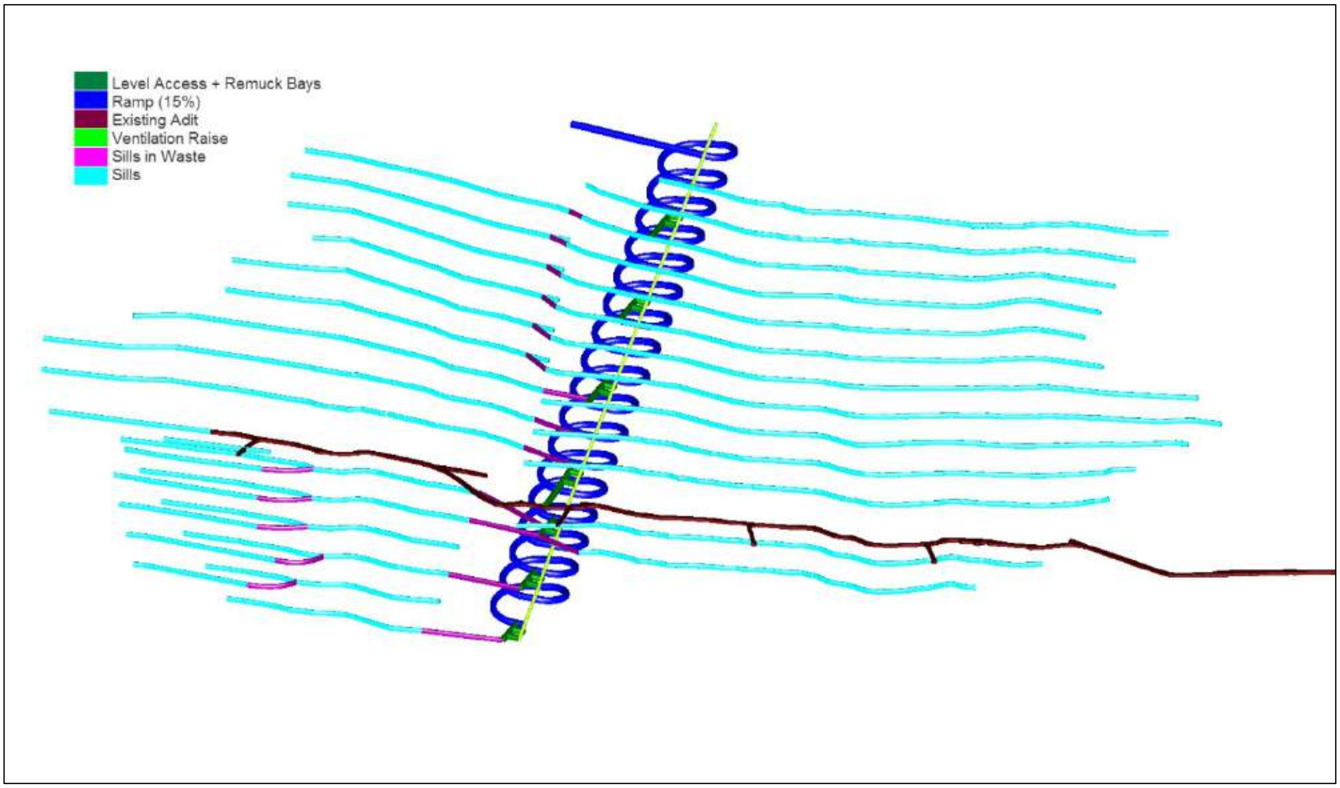


Figure 20.2: Isometric view looking towards the east of the LOM mine development

Table 20-3 summarizes the total life-of-mine (LOM) capital development and cost. About $7 million of this total is expensed in the pre-production phase and the remainder is expensed over the following seven years as sustaining capital. Rehabilitation of the existing adit drift is assumed to be completed during the proposed exploration Phase 1 described below in Section 25.1.1.

Table 20-3: Mine Development Capital Cost Estimate (LOM)

Item	Length (meters)	Unit Cost	Cost
Portal	1	$75,000	$75,000
Decline	2,555	$4,500	$11,497,500
Vent Raise	383	$2,500	$957,500
Vent Access	71	$4,000	$284,000
Level Access	279	$4,000	$1,116,000
Re-muck Bays	71	$4,000	$284,000
Paste Line	800	$400	$320,000
Contingency (15%)			$2,180,100
Total			**$16,714,100**

The mine development unit costs are estimates only and actual quotations from mine contractors will be required at the next level of engineering study. These costs can vary greatly depending on mobilization costs and specific site requirements. A 30% increase in these unit rates would increase the mine development costs by $6 million.

The mine equipment costs are summarized in Table 20-4. The unit costs are estimates from Golder's existing database and no actual quotations were obtained for this study.

Table 20-4: Mine Equipment Capital Cost Estimate (LOM)

Item	Quantity	Unit Cost	Cost (USD)
LHD (1.5 m^3)	2	$300,000	$ 600,000
LHD (3 m^3)	1	$650,000	$650,000
Trucks (20-t)	1	$550,000	$550,000
Drills	2	$250,000	$500,000
Jumbo/Long Tom	2	$400,000	$800,000
Scissorlifts	2	$250,000	$500,000
Grader	1	$235,000	$235,000
Jeeps	3	$90,000	$270,000
Electrical System	1	$500,000	$500,000
Pumps	4	$30,000	$120,000
Fans	1	$300,000	$300,000
Refuge Chambers	2	$150,000	$300,000
Contingency (20%)			$1,005,000
Total			**$6,330,000**

20.2.3 Process Plant

The capital cost for the process plant is based on a 400 tpd sorting/gravity plant based on estimates for the list of mechanical equipment as presented in Table 20-5. The mechanical list is generated from the flowsheets and priced to +/-30% from results of recent studies for similar process options.

Table 20-5: Process Plant Capital Cost Estimate

Item	Cost (USD)
Crushing & Screening	$1,365,397
Concentrator Building	$85,895
Sorting	$672,338
Spiral Plant	$224,112
Product Handling	$86,799
Slimes Handling	$282,089
Conveyors & Material Handling	$483,409
Plant Services	$96,810
Plant Piping System	$51,045
Plant Electrical, Instrumentation & Control	$704,172
Site Preparation, Bulk Earthworks	$366,605
Infrastructure General Services	$18,786
Water Treatment Plant	$37,573
Infrastructure and Buildings	$350,593
Stores Workshop and Offices	$141,281
Plant Maintenance Equipment and Tools	$33,964
Main Laboratory	$214,303
Plant Substation	$106,210
First Fill & Spares	$105,144
Subtotal	$4,999,412
EPCM	$749,911
Contingency (20%)	**$1,150,000**
Total	**$6,900,000**

Paste Plant

Generic high-density paste plant designs incorporate mechanical thickening of the full-fraction tailings stream to approximately 78% solids, followed by mixing of the high-density paste with binder (Portland cement) and pumping the mixture to the underground openings by positive displacement pumps. The paste plant design and cost estimate presented here is for a 20 tonne per hour (tph) full paste production. The cost is based on a similar 80 tph unit. The total estimated design and construction cost for the paste plant including a 20% contingency is $3,360,000. Appendix B contains a preliminary flowsheet for the proposed paste plant.

20.2.4 Project Capital Cost Schedule

The LOM capital cost schedule is presented in Table 20-6. Almost 60% of the capital spending occurs before Year 1 as the plant and site infrastructure is constructed prior to production. This schedule assumes that the main underground decline is developed to approximately the 200 m elevation (about 60 m vertical depth) in order to commence mining in the upper stope regions. Ramp development is then done each year in order to advance the mining downwards. The majority of mine equipment is also purchased within two years. Since the mine life is relatively short no replacement equipment is accounted for in this study. The tailings facility is staged so that an expansion portion is constructed in Year 4.

Table 20-6: LOM Capital Cost Schedule

Item	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mine Development	16,714	3,929	2,352	1,568	1,568	1,568	1,568	1,568	1,712	880	-	-
Mine Equipment	6,330	4,059	1,741	331	100	50	50	-	-	-	-	-
Plants*	10,260	9,405	855	-	-	-	-	-	-	-	-	-
Infrastructure	14,736	12,122	368	-	-	2,246	-	-	-	-	-	-
Closure	2,000	-	-	-	-	-	-	-	-	-	-	2,000
Total Capital	**50,040**	**29,515**	**5,316**	**1,899**	**1,668**	**3,864**	**1,618**	**1,568**	**1,711**	**880**	**-**	**2,000**

* Includes the paste plant ($3.36 million)

Of the total LOM capital cost of $50 million, $32 million is considered pre-production spending and the remaining $18 million is sustaining capital as summarized in Table 20-7.

Table 20-7: Total LOM Capital Cost Breakdown

Type	Cost
Pre-production Capital	$32,173,265
Sustaining Capital	$17,866,835
Total Capital	**$50,040,100**

21.0 ECONOMIC ANALYSIS

21.1 Methodology

A discounted cash flow method is used to evaluate the economics of the Grey River Project using the base case economic parameters and underground mining resource as defined above. A discount rate of 5% is used for the analysis. The cash flow is generated on a pre-tax basis at this early stage of the project. In future studies federal and provincial taxes will have to be considered as well as any royalties that may apply.

21.2 Basis of Analysis

Future annual cash flows have been estimated based on estimates for production rate, mined grade, mill recovery, metal price and capital and operating cost estimates as presented earlier. Table 21-1 summarizes the base case parameters used in the cash flow analysis.

Table 21-1: Summary of Base Case Project Parameters used in the Cash Flow Analysis

Parameter	Value
Underground Mining Resource	1,268,306
Mined Grade	0.524% WO_3
Mining & Milling Rate	400 tpd, 146,000 tpa
Plant Recovery	85%
Total LOM Capital Costs	$50 million
Pre-production Capital Cost	$32 million
Total Operating Costs	$107.50 per tonne
Metal Price	$355 per MTU ($16 per lb)
Discount Rate	5%

21.3 Results of Cash Flow Analysis

The preliminary base case cash flow is presented in Appendix D. This analysis demonstrates that the base case project yields a positive pre-tax undiscounted cash flow of $15.5 million. The Net Present Value (NPV) for the base case is $2.9 million using a discount rate of 5%. Net revenue is sufficient to cover operating and capital expenses in each year of full production.

21.4 Sensitivity Analysis

The Grey River deposit is most sensitive to revenue parameters in the cash flow; metal price, grade and recovery. This is seen in Figure 21.1 that presents a “spider” chart for the key parameters and the impact of changes to these parameters on the NPV.

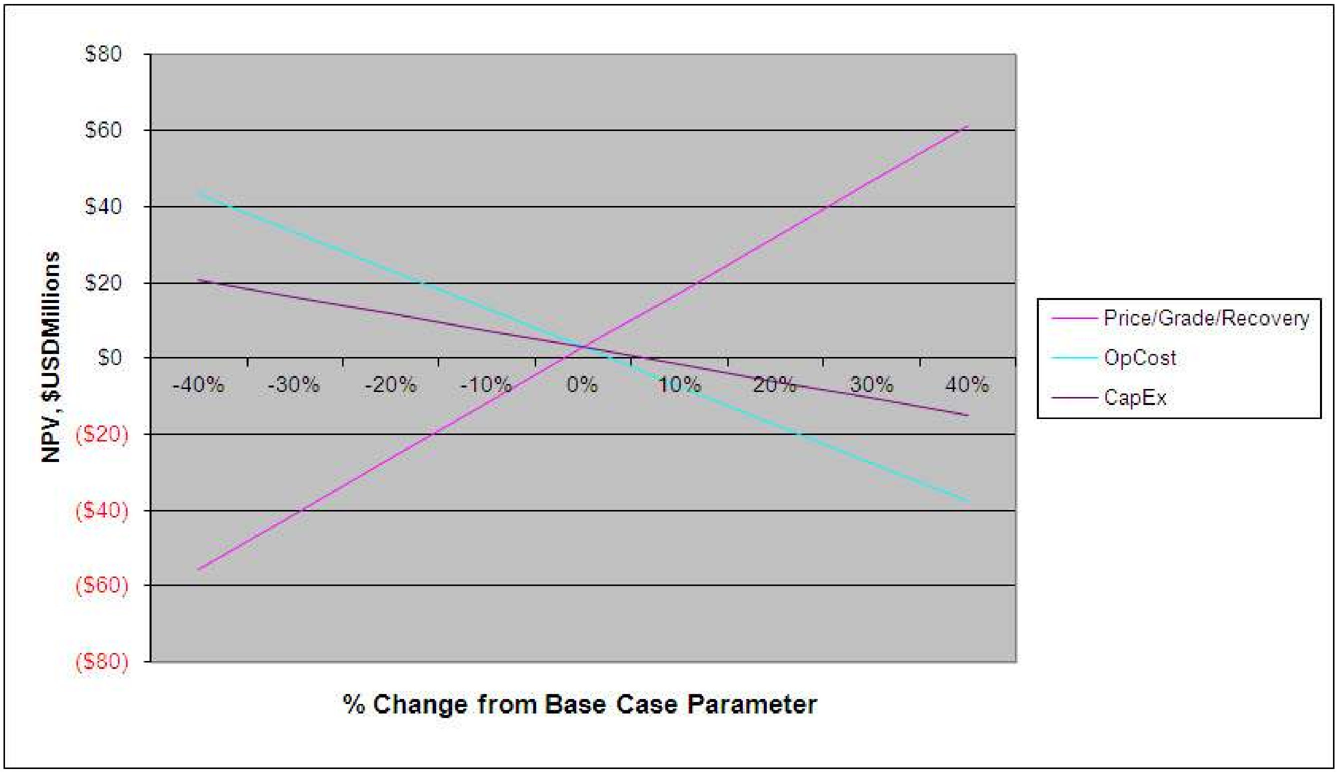


Figure 21.1: Sensitivity of the project NPV to changes to the key base case parameters

The current metal price is approximately 30% higher than the value used for the base case, which is nearer to the 3-year historic average. Figure 21.2 presents the sensitivity of the project NPV to metal price.

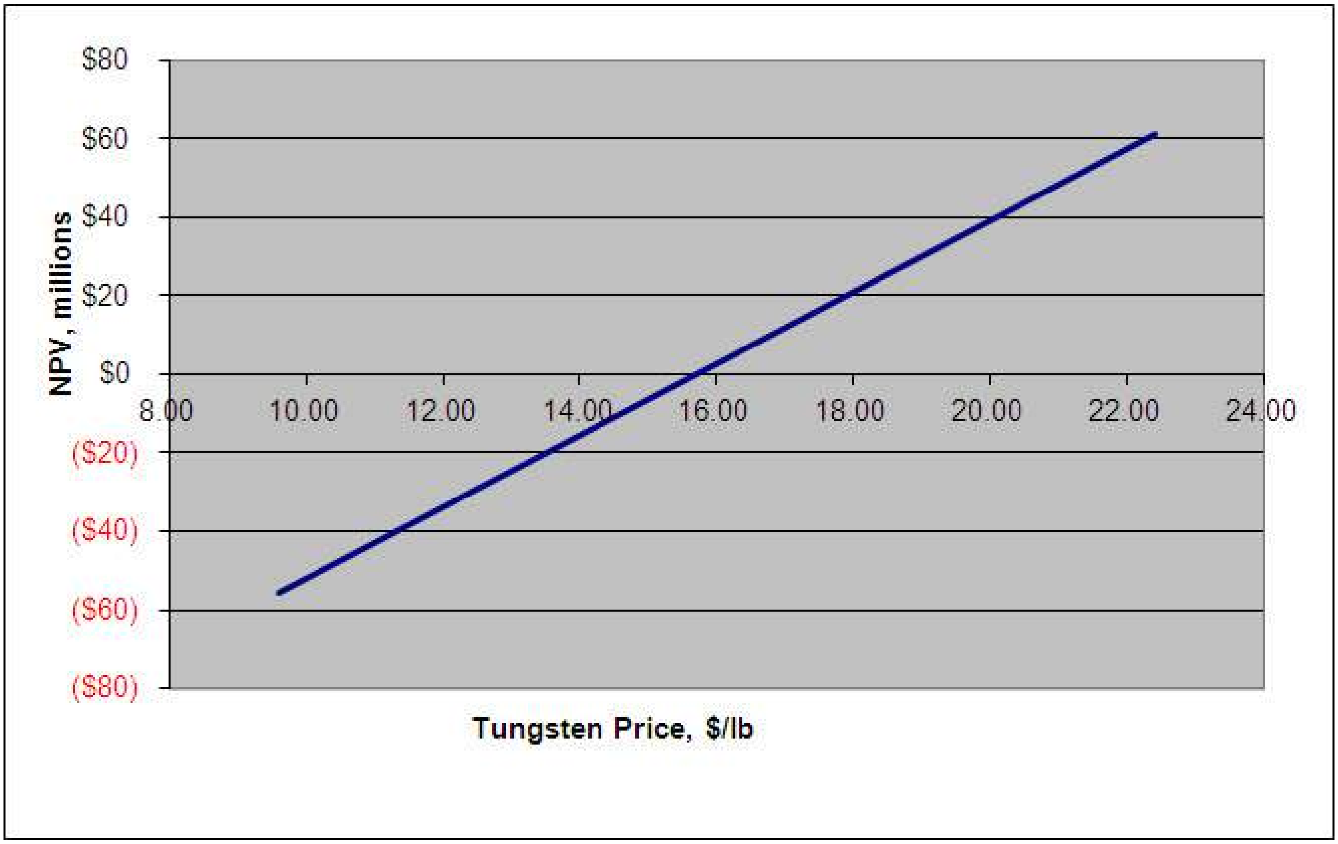


Figure 21.2: The sensitivity of project NPV to the metal price (base case at $16/lb)

As evidenced by Figure 21.2 a 30% increase in metal price (to $21/lb) generates a pre-tax NPV of $47 million. The cashflow for this case is provided in Appendix D.

21.5 Project Payback

The project payback period using the base case parameters is almost 8 years and the total mine life is just over 9 years. At current metal prices (about 30% higher than the base case value at December 2011) the payback period would be reduced to just over 4 years.

21.6 Project Risks and Opportunities

The following project risks are identified for the Grey River Project:

- Highly sensitive to metal price. This is both a risk and an opportunity depending on future metal price.
- Highly sensitive to dilution in the mining process; a 10% drop in mined grade would result in a negative NPV at base case parameters. The underground stope mining will have to be very efficient with high quality control.
- The 85% plant recovery used in this study is untested and requires confirmation through a metallurgical test program. Similar to above, a 10% drop in recovery to 77% generates a negative NPV at base case parameters. There is also opportunity that plant recovery could be higher than estimated here which would add value to the project.
- The operating costs, in dollars per tonne mined/milled, for narrow vein, low production underground mines are very sensitive to production rate since small reductions in the tonnage increase unit costs and cut-off grade.
- There have been few studies on environmental and social impacts and tailings disposal options and these pose risks to project development.

22.0 ADJACENT PROPERTIES

Tenajon Resource Corp owns the Moly Brook Molybdenum Property which is adjacent to Playfair's Grey River project. The Moly Brook Property is a molybdenum project that was first discovered in 1995 by the now defunct Royal Oak Mines Inc. while exploring for gold. In May 2009, based on 43 drill holes, Tenajon Resources announced a resource estimate for its 100% owned Moly Brook property located in Newfoundland, Canada. The property contains an Indicated resource of 124.6 million lbs of molybdenum and an Inferred resource of 38.6 million lbs at a 0.04% cut-off. 120.0 million pounds of the Indicated Resources and 32.1 million pounds of the Inferred Resources are contained within a pit shell which has an estimated strip ratio of 2.03:1.

23.0 OTHER RELEVANT DATA AND INFORMATION

This Section is not applicable.

24.0 INTERPRETATION AND CONCLUSIONS

24.1 Resources

The property is located adjacent to the fishing community of Grey River on the south coast of Newfoundland. The town of Grey River is situated at approximately latitude 47°34'N and longitude 57 °6'W. The Grey River Tungsten property consists of 154 contiguous mining claims grouped into one mineral license (015686M) held by Playfair through a purchase agreement with South Coast Ventures.

The project area is underlain by the Silurian-Devonian Burgeo Intrusive Suite and an east – west trending belt of Precambrian metamorphic rocks referred to as the Grey River Enclave which consists of amphibolites, quartz-mica schists, pelites and gneisses. The faults in the metamorphic rocks can be grouped into two main sets: an east-west set parallel to the schistosity and a south-east set cross-cutting the schistosity. A third set occurs only in the granites. Arising from this set of faults is a prominent fissure system of tensional origin striking north to northeast. These tension fissures act as the structural control for the tungsten veins.

The Grey River tungsten veins are typical fluorite-rich, wolframite-quartz greisen vein deposits. Wolframite is the dominant tungsten-bearing mineral although a number of small scheelite occurrences are known. The principal vein is the Number 10 Vein which strike from 10° to 30° north, with a steep dip of 65° to 75° to the west. The width of the vein ranges from 0.25 to 2.5 meter and average of 1.13 meter.

The Number 6 Vein is potentially the north extension of the Number 10 Vein displaced 150 meters east along an East-West trending fault system.

Number 10a Vein capture a portion of the high grade mineralization on the hanging wall of Number 10 Vein however its true orientation is not well know at present. A low core angle on some of the quartz veinlets suggests a possible east-west direction.

A bulk density of 2.81 g/cc has been used for the tonnage calculation based on 21 samples collected by Playfair.

The database consist of historical diamond drill data, trench, underground samples collected by ASARCO between 1957 and 1970. Playfair diamond drilled the property in 2006 and again in 2008 to confirm ASARCO's drill data and to increase the resource on the Number 6 vein and below the adit level on the Number 10 Vein. The back and face samples were de-clustered to remove any spatial bias using a pseudo polygonal method.

DGL performed data verification through a site visit, as well as the collection of independent character samples, and a database audit. Minor clerical errors in the Playfair database were corrected prior to mineral resource estimation.

There is a marked difference in the grade distribution between the historical data and the newer drill results with the Playfair assays generally returning lower WO_3 values. The difference is currently assumed to be related to the analytical procedure but needs to be evaluated by Playfair.

A mineral resource has been estimated for the Number 10 Vein on the Grey River tungsten property using data supplied by Playfair. This data includes drill hole information as well as historical assay data for back, face, raise, trench and grab samples.

Both Inverse Distance Squared and Nearest Neighbour interpolation methods were used. No significant discrepancies exist between these methods.

All indicated blocks in the resource were downgraded due to the lack of original assay certificates for the underground sampling, the difference in grade between the historical data and the newer Playfair data and the difficulty in capturing a representative sample with the small historical EX drilling.

An Inferred mineral inventory of 1.2 million tonnes at 0.730% WO3 using a 0.2% cut-off and excluding mineralization grading less than 0.2% WO_3 over a 1.0 minimum mining width has been estimated for the combined Number 10, Number 6 and Number 10a Vein. The mineral inventory was provided to Golder for further economic assessment.

24.2 Mining

The Grey River deposit is generally narrow-vein and steeply dipping with vein dip ranging from 70 to 80 degrees and is considered amenable to a blasthole open stoping mining method with pastefill. At about 2,400 tonnes per vertical meter of underground mining resource a 400 tpd operation is proposed at this level of engineering study. Small narrow vein mining equipment and techniques will be required to mine the deposit efficiently, minimize dilution and maximize recovery. To achieve higher production rates the deposit strike length needs to be longer but that would also necessitate additional decline, level and ventilation development thus increasing pre-production capital costs.

The underground mining resource (diluted and mine recovered) is estimated to be 1,268,306 tonnes at a grade of 0.524% WO_3 using a cut-off of 0.35% WO_3 and a minimum mining width of 2 m. Using the base case economic parameters the pre-tax cash flow is estimated to be positive at $15.5 million over a mine life of about 9 years. The pre-tax NPV for the base case using a 5% discount rate is $2.9 million. The project would generate a NPV(5%) of $47 million at an IRR of 27% at the current metal price of around $440 per MTU ($21 per lb) (at December 2011).

All of the resources used to develop the underground mining resource in this study are in the Inferred category. The Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary economic assessment will be realized.

24.3 Processing

Grey River is proposed as a low-tonnage, high grade operation, with a relatively free-milling ore, shown to be amenable to gravity separation methods, producing a potential concentrate of 60% with a tungsten recovery of 75%.

Individual gravity separation tests indicate that the quality of the sample preparation was insufficient to produce consistent metallurgical results. The test work indicates that testing was undertaken on a ‘bulk sample’, however, the indicated grade was relatively high compared to the current estimated feed grade. The potential mine feed is highly diluted compared to the sample tested by SGS and so metallurgical results from those reports are potentially optimistic compared to results achieved from a fully diluted sample.

A typical gravity/flotation-based plant processing material such as at Grey River would generally obtain between 85% to 92% recovery to a 65% to 70% concentrate. For this study a recovery of 85% to a 65% concentrate grade was assumed.

24.4 Environmental

The assessment of environmental considerations is preliminary at this stage of the project and will require further study. The specific requirements for environmental assessment will ultimately be determined by the scope of the project, known, or assumed, environmental sensitivities, and public and stakeholder response to the project information in the registration document.

25.0 RECOMMENDATIONS

25.1 Exploration Recommendations

The Number 10 Vein at Grey River is one of only a few deposits in Canada with demonstrated tungsten resources, partial underground development and two stages of metallurgical test data. From the available data the vein appears to be continuous between the surface trenches and the exposures within the adit. However, due to the nuggety nature of the mineralization, as well as the relatively wide-spaced drilling, there are gaps in the data that must be filled in order to change the resource categories.

25.1.1 Phase 1

Phase one is designed to validate the assays in advance of the drill program. The re-assay of the pulps can be done at anytime. Re-habilitating the adit level is required by a contractor and needs to be scheduled in advance of the arrival of the Playfair geological team. This adit rehab is also needed for the potential mining plan.

DGL recommends that Playfair re-run 12 (15 of the assays intersecting the vein) drill core pulps from the 2006 and 2008 program using the Fusion XRF or INAA since there is a possibility that the ICP-MS or ICP-ES understated the tungsten grade.

Since the accuracy of the historical assay in comparison to the newer results is not known, DGL recommends that Playfair embark on a campaign of verification of historical underground samples and drill core. DGL suggests replicating 35 underground channel samples or 5% of the underground dataset. If the grade compares well with the existing samples, then that would be sufficient to validate the remaining samples. If the grades do not compare, but a good correlation exists with the older samples, an adjustment could be mathematically applied via a regression equation. The intent is to minimize the risk to the resource and add a level of confidence that would allow re-classification of the current resource to Indicated. A budget estimate for this work is presented in Table 25-1.

Table 25-1: Budget Estimate for Validating Underground Samples

Item	Amounts
Assay cost - 47 @ $60 per sample	$2,820
Cost for rehabilitating the existing adit drift (1.9 km) and installing services (air and water)	$1,700,000
Personnel - Geologist and 2 helpers (4 weeks)	$44,000
Accommodation in Grey River.	$15,000
Helicopter support for Playfair (3 trips)	$18,000
Sub-total	$1,779,820
Contingency (20%)	$355,964
Total	**$2,135,784**

25.1.2 Phase 2

A series of close-spaced holes within the current Inferred category is suggested so that the nearby Indicated categories can be expanded. A surface drill rig capable of HQ (or thin wall N) core drilling is recommended to create data points in the area above the face and back samples in the South Vein. This program can be completed in one campaign and the results used to assess the viability of additional exploration.

Six holes are needed at the +50 m elevation level in an area immediately above the face/back sample locations in the South Vein. HQ core can be drilled from surface to intersect the vein in these locations but each hole will be in the order of 300 m in length.

Alternatively, these holes can be drilled from the three closest cross-cuts as soon as the adit has been rehabilitated. The cross-cut locations will shorten the hole lengths although this is at the expense of an optimum core angle with the vein.

Six holes should be drilled at the +100 m elevation level within the inferred category. As with the initial six holes these pierce points will be spaced about 50 m apart at this elevation with 5 holes intersecting the Number 10 Vein and one hole intersecting the Number 6 Vein. Holes should be approximately 230 meter in length to reach that elevation.

Four 150 meter long holes should test the +150 m elevation level. Two holes intersecting the Number 6 Vein and two holes intersecting the Number 10 Vein.

Four short holes on the +200 m level testing the southern extent of the Number 10 vein. Each hole should be about 100 meter in length.

The budget for all 20 surface holes is given in Table 25-2 and the drilling plan is shown in Figure 25.1.

Table 25-2: Proposed Budget for Additional Fill-in Drilling on the Number 10 Vein

Item	Amounts
Drilling 20 HQ size holes, total of 4,380 meter @ $120/m	$525,600
Report Writing, maps	$10,000
Equipment rental, & DDH collar surveying	$25,000
Personnel	$75,000
Assaying and sample preparation	$25,000
Field accommodation	$30,000
Helicopter support	$100,000
Sub-total	$790,600
Contingency	$80,000
Total	**$870,600**

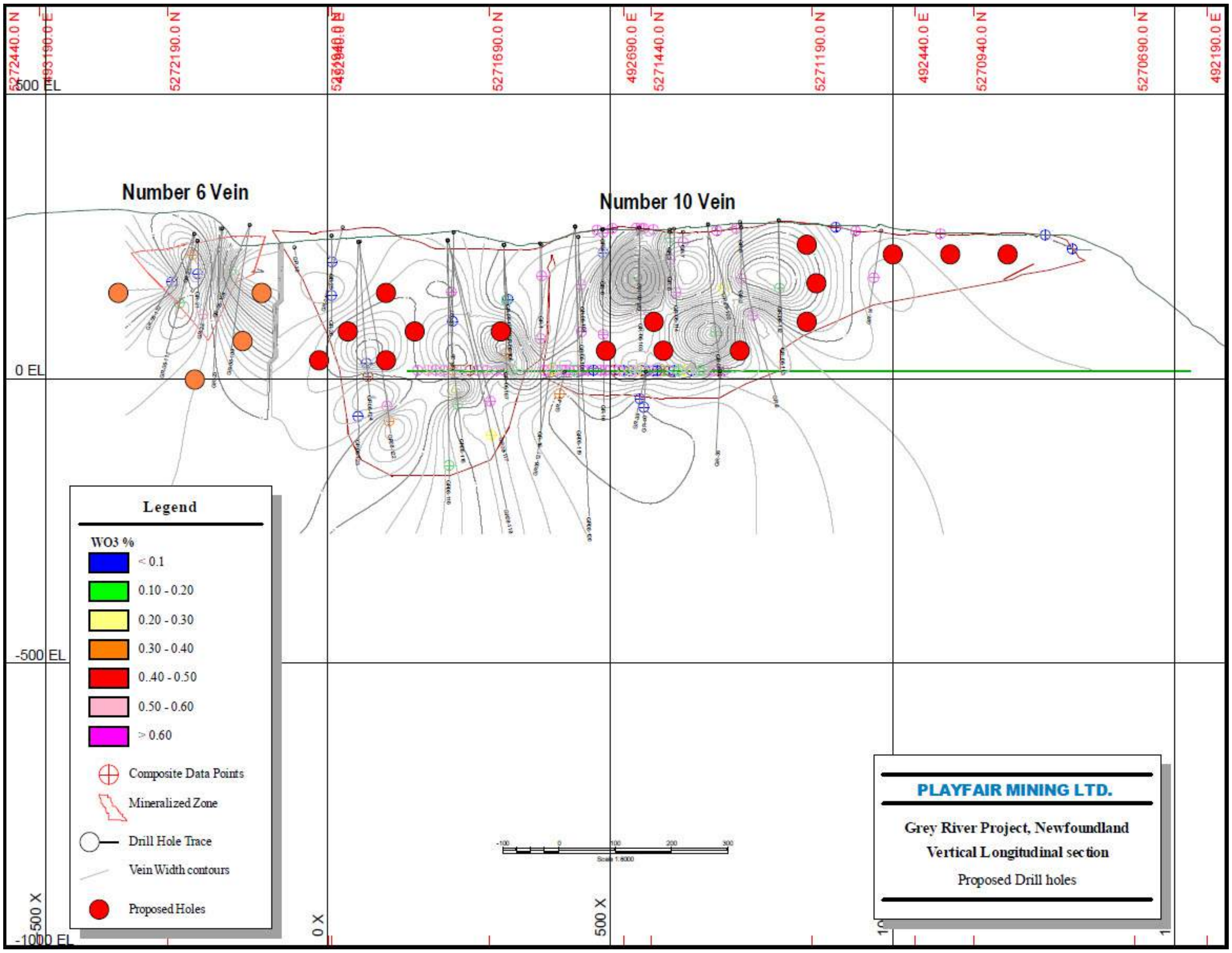


Figure 25.1: Proposed drilling plan for the Grey River project

Other targets should be examined to assess the potential for increasing the grade and/or tonnes of the Number 10 Vein. In particular, the area below the adit level should be drilled to verify the continuity of the tungsten mineralization in the vein. This program can be best accomplished after the adit has been re-habilitated using drill stations set up in the cross-cuts. No budget is proposed for this program due to uncertainty in the cost estimates for the different evaluation methods.

Additional recommendations for future exploration programs include:

- The insertion of blanks and duplicate samples should be added to the QA/QC protocol in future sampling programs.
- Field generated specific gravity determinations should continue.

- Geotechnical information should be routinely collected to create a data set that will be of use in future mining efforts. Core photographs should be taken and catalogued.
- A minimum of two ASARCO holes (10% of the data) should twinned with a new holes to quantify the effects of core re-drilling/grinding in the original standard drill holes (in addition to the proposed holes above).

25.2 Mining

Further work is required to develop more accurate cost estimates for both operating and capital costs. The project economics are quite sensitive to mine operating costs and a higher confidence in this value will be needed. In addition, the mine operating costs for small, narrow vein deposits are very sensitive to production rate. Further study will be needed to confirm the proposed rate of 400 tpd and develop a more detailed mine plan and production schedule. Other, more selective methods are possible but they could increase costs or limit production rates. The deposit is also very sensitive to dilution and additional studies should be completed on the relationship between mining cost, cut-off grade, dilution and productivity.

Limited geotechnical data is available for the Grey River deposit at this stage. More detailed geotechnical investigations and design exercises must be done for the mine workings, proposed stope sizes and crown pillar dimensions at the next level of engineering study.

In order to develop mineral reserves for the Grey River Project as part of a pre-feasibility study the majority of Inferred mineral resources, and all of the underground mining resource as defined here, will have to be upgraded to the Indicated category.

25.3 Processing

Concentrate specification needs to be addressed in future test work. Furthermore, WO_3 concentrates destined for APT plants have a target size distribution specification. Further metallurgical work is strongly recommended in order to; establish a firm specification on the head grade for the feed; maximize recovery to an acceptable WO_3 concentrate; and, move beyond the scoping stage and simulate the unit processes of the proposed flow sheet on a representative sample of the feed material. This will help establish firm criteria for the metallurgical performance to produce an acceptable concentrate.

A projected recovery of 85% WO_3 is used in this study but is provisional and should be confirmed by test work in the next phase of work.

25.4 Environmental

The following are recommended for future work in relation to environmental and socio-economic issues:

- Fish presence and habitat characterization should be determined for areas that might be affected by activities conducted on the plateau (i.e., Project, not exploration activities). The results of these surveys will assist in the Project planning. The overall site plan showing all water bodies (streams and ponds), roads and other infrastructure should be updated regularly as the project design advances.

- It is recommended that fish presence and fish habitat characterization be determined for areas that might be affected by activities conducted in the marine environment or with wharf construction in marine areas.
- It is recommended that consideration be given to setting up a baseline water quality sampling program that will carry into the development of the Project.

Other issues will be better defined when additional Project details are available, at which time liaison with appropriate regulators can be done to identify data gathering requirements.

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27.0 CERTIFICATES OF QUALIFIED PERSONS

27.1 Certificate for Joseph Rosaire Pierre Desautels, P.Geo.

I, Joseph Rosaire Pierre Desautels of Barrie, Ontario, do hereby certify that as an author of Sections of the technical report titled "**Grey River Project Preliminary Economic Assessment**" dated March 14, 2012, I hereby make the following statements:

- I am a Principal Resource Geologist with Desautels Geoscience Ltd. with a business address at 290 Harvie Road, Barrie, Ontario, L4N 8H1.
- I am a graduate of Ottawa University (B.Sc. Hons., 1978).
- I am a member in good standing of the Association of Professional Geoscientists of Ontario (Registration #1362).
- I have practiced my profession in the mining industry continuously since graduation.
- I visited the property from October 04 to 05, 2011.
- I have read the definition of "qualified person" set out in NI 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to resource modelling includes 31 years' experience in the mining sector covering database, mine geology, grade control, and resource modelling. I was involved in numerous projects around the world in both base metals and precious metals deposits.
- I am responsible for the preparation of parts of Section 1.0 to 3.0 and all of Sections 4.0 to 12.0, 14.0, 22.0 24.1 and 25.1 of this technical report titled "**Grey River Project Preliminary Economic Assessment**", dated March 14, 2012.
- I have no prior involvement with the property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 14th day of March, 2012 at Barrie, Ontario.

Pierre Desautels, P.Geo.

PROFESSIONAL GEOSCIENTIST
PIERRE DESAUTELS
PRACTISING MEMBER
1362
ONTARIO

27.2 Certificate of David Sprott, P.Eng.

I, David Sprott, of Mission, British Columbia, do hereby certify that as an author of Sections of the report titled "**Grey River Project Preliminary Economic Assessment"**, dated March 14, 2012, I hereby make the following statements:

- I am an Associate and Senior Mining Engineer with Golder Associates Ltd. with a business address at 500-4260 Still Creek Drive, Burnaby, B.C., V5C 6C6.
- I am a graduate of Queen's University, Kingston, Mining Engineering (BSc 1983, MSc 1984).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #19021) and the Association of Professional Engineers of Ontario (License #90533134).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to this deposit type includes 9 years working as an engineer at an underground gold mine that used similar mining methods.
- I am responsible for the preparation of parts of Section 1.0 to 3.0 and 16.5, and all of Sections 15.0, 16.0, 17.0, 18.0, 20.0 (except 20.1.2 and 20.2.3), 21.0, 24.2 and 25.2 of this technical report titled "**Grey River Project Preliminary Economic Assessment"**, dated March 14, 2012.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 14th day of March 2012 at Burnaby, British Columbia.

D. Sprott

PROFESSIONAL PROVINCE OF D. L. SPROTT # 19021 BRITISH COLUMBIA ENGINEER

David Sprott, P.Eng.

27.3 Certificate of Andrew Bamber, P.Eng.

I, Andrew Bamber, of Vancouver, British Columbia, do hereby certify that as an author of Sections of the report titled "**Grey River Project Preliminary Economic Assessment"**, dated March 14, 2012, I hereby make the following statements:

- I am employed as a Partner/ Principal Engineer with Minesense Technologies Ltd. with a business address at 122 – 1857 West 4th Avenue, Vancouver, BC, Canada, V6J 1M4.
- I am a graduate of the University of Cape Town, BSc. (Hons.) Mechanical Engineering, 1993, and the University of British Columbia, MASc. Mining and Mineral Process Engineering, 2004.
- I am a member in good standing of the South African Institute of Mechanical Engineers; the South African Institution of Certificated Mechanical and Electrical Engineers; the Canadian Institute of Mining and Metallurgy (CIM), and a Professional Engineer registered with the Engineering Council of SA License # 990013.
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Playfair Grey River Project includes over 14 years experience in mining and mineral processing projects in Southern Africa, Canada and Central Asia. I have been a principal in several pre-feasibility and feasibility studies, including the Kroondal 'K2' Platinum Project, the Mimosa Phase III Platinum Expansion, the Voskhod Chrome Project in Kazakhstan and the Pipe II Scoping Study for INCO Thompson.
- I am responsible for the preparation of parts of Section 1.0 and 16.5 and Sections 13.0, 20.1.2, 20.2.3, 24.3 and 25.3 of this technical report titled "**Grey River Project Preliminary Economic Assessment"**, dated March 14, 2012.
- I have no prior involvement with the Property that is the subject of the Technical Report.

- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 14th day of March, 2012 at Vancouver, British Columbia.

Andrew Bamber, P.Eng.

28.0 CLOSURE

GOLDER ASSOCIATES LTD.

Andrew Lyon, P.Eng. (BC)
Mining Engineer

David Sprott, P.Eng. (BC)
Associate, Senior Mining Engineer

DS/AL/rs/aw

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APPENDIX A

ASARCO Analytical Procedure

APPENDIX 1

COLORIMETRIC THIOCYANATE METHOD OF DETERMINATION OF TUNGSTEN IN ORES

This method is designed for the rapid control determination of tungsten in ores and concentrates from 0.05% to major concentrations of tungsten.

The sample is fused with a sodium peroxide-sodium carbonate mixture, water leached, and diluted to volume. A suitable aliquot of the clear solution is acidified (about 9 N) with sulfuric acid and hydrochloric acid. The tungstate ion is reduced with stannous chloride, potassium thiocyanate added, and the color measured spectrophotometrically.

Since many of the ions sensitive to thiocyanate are separated by the sample decomposition technique, there is no serious interferences except vanadate. Up to 5 mg of molybdate may be present in the aliquot (Note 1).

REAGENTS

Stannous Chloride, 2 M - Dissolve 452 g of $SnCl_2 2H_2O$ in 500 ml of concentrated HCl by warming in a covered 800 ml beaker. Transfer to a liter flask and dilute to the mark with HCl. The solution is stable for one month if stoppered tightly.

Potassium Thiocyanate, 2 M - Dissolve 194 g of KCNS in 500 ml of water by warming. Cool and dilute to one liter. The solution is stable for one month if stored in a cool, dark place.

Standard Tungsten Solution - Dissolve 1.7940 g of reagent-grade $Na_2WO_4 2H_2O$ in water. Add one pellett of NaOH, dissolve, and dilute to one liter. 1 ml = 1.00 mg of tungsten. With a pipette, transfer 10.00 ml of this stock solution to a 100-ml volumetric flask and dilute to the mark. 1 ml = 0.100 mg of tungsten.

PREPARATION OF ABSORBANCE-CONCENTRATION CURVE

With pipettes, transfer 1,2,5,7, and 10-ml aliquots of the diluted tungsten stock solution to 100-ml Pyrex volumetric flasks. The aliquots represent 0.10, 0.20, 0.50, 0.70, and 1.0 mg respectively of tungsten. Dilute to 25 ml and continue according to the procedure below, starting with "Add 10 ml of concentrated H_2SO_4...". Measure the % transmission in a filter photometer in a 2-cm cell at 400 mu using a reagent blank carried through all the steps of the procedure to set the photometer to 100% transmittance. Plot the absorbance versus mg of tungsten on semi-logarithmic paper (Note 2). The system follows Beer's Law.

PROCEDURE

A. DECOMPOSITION - Transfer a weighed sample (0.2 to 0.5 g) containing not more than 150 mg of tungsten to a 30-ml iron crucible containing 3 g of Na_2O_2 and 2 g of Na_2CO_3. Mix well and fuse over a burner (Note 3). Cool until solid, then with tongs place the crucible on its side in a 250-ml beaker. Cover and carefully add 50 ml of water. When disintegration is complete, remove and rinse the crucible. Police the crucible. Add 10 ml of ethanol (95%), cautiously boil 3 or 4 minutes, cool, and transfer to a 200 ml volumetric flask (Note 4). Cool and dilute to volume, mix well, and allow to settle for 10 minutes. Decant the supernatant liquor through a dry filter and collect about 50 ml of the filtrate.

B. COLOR DEVELOPMENT - With a pipette, transfer an aliquot, containing 0.1 to 1.0 mg of tungsten, of the filtered sample solution to a 100 ml pyrex volumetric flask. Dilute, if necessary, to 25 ml and add 10 ml of concentrated H_2SO_4 and 20 ml of concentrated HCl. Add 10 ml of $SnCl_2$ solution (2 M), mix, and digest on a steam bath for 5 minutes. Remove from heat and immediately stopper tightly with a rubber stopper (Note 5). Chill in an ice water bath to $10^{o}C$ or less (Note 6). Remove the stopper and quickly add 10 ml of KCNS solution (2 M), dilute to the mark, and mix well. Return to the ice water bath for 2 or 3 minutes. Remove, and after 5 minutes measure the absorbance a 2 cm cell in a filter photometer at 400 mu against a reagent blank. From the calibration curve, read milligrams of tungsten present in the aliquot sample.

CALCULATIONS

$$\%WO_3 = \frac{A \times 25.2}{B \times C}$$

where:

A = mg of W from calibration curve,

B = aliquot (ml), and

C = sample weight (g)

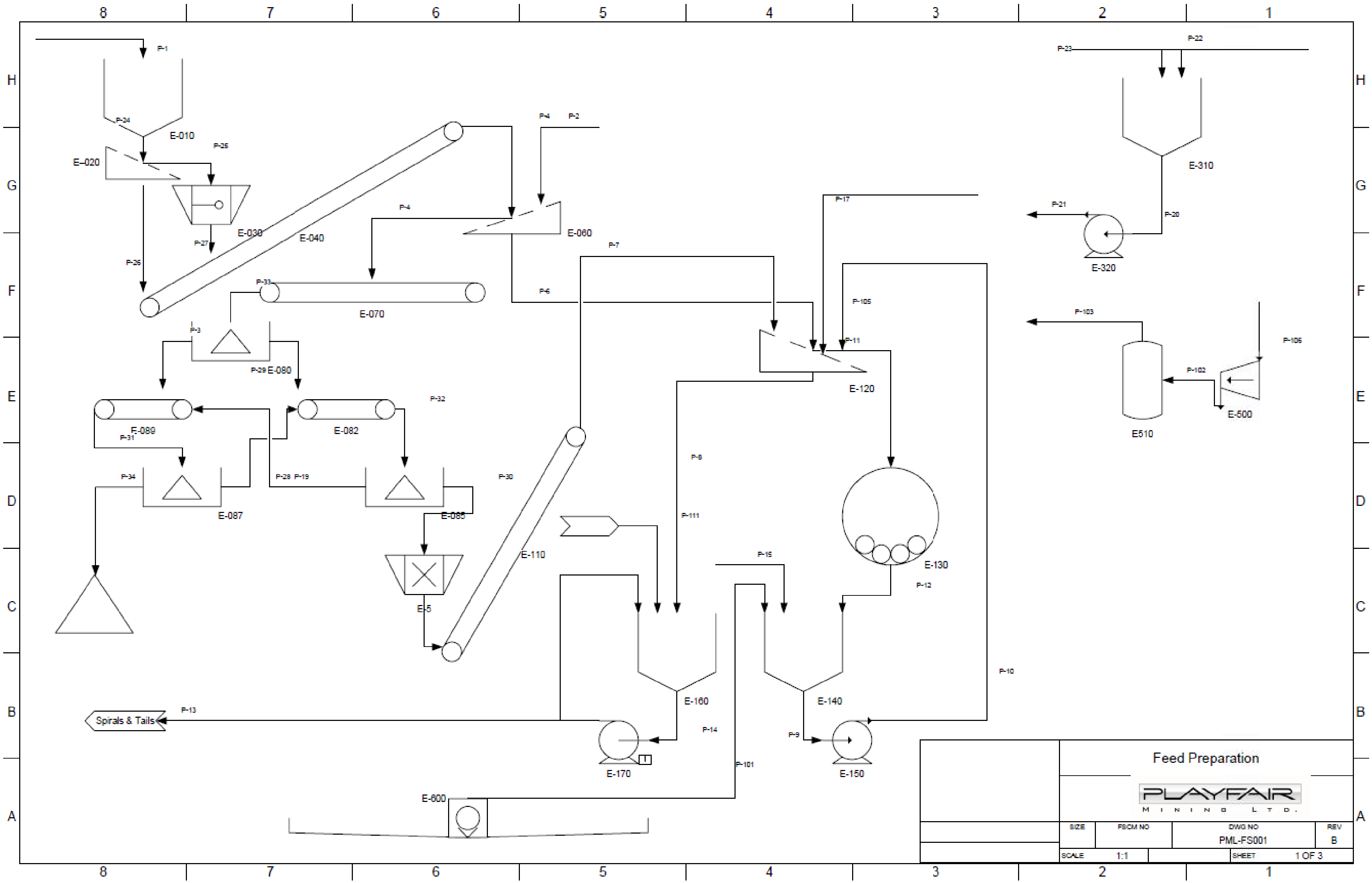
NOTES:

1. Vanadium reacts almost identically to tungsten under conditions of $SnCl_2$ reduction. Separations can be made by the usual chloroform-cupferron extration if tungsten is complexed with NaF. Fortunately, vanadium is a rarity in tungsten ores. Molybdenum thiocyanate fades very rapidly in 9 N acid medium. The absorbance of 5 mg of molybdenum is equivalent to about 0.05 mg of tungsten.

APPENDIX B

Plant Flow Sheets

Feed Preparation
Spiral Plant, Tailings, Concentrate Handling
Paste Plant



E-010
E–020
E-030
E-040
E-060
E-070
E-080
E-082
E-085
E-087
E-089
E-110
E-120
E-130
E-140
E-150
E-160
E-170
E-5
E-600
E-310
E-320
E-500
E510
Spirals & Tails
P-1
P-2
P-3
P-4
P-6
P-7
P-8
P-9
P-10
P-11
P-12
P-13
P-14
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P-28
P-29
P-30
P-31
P-32
P-33
P-34
P-101
P-102
P-103
P-105
P-106
P-111
Feed Preparation
PLAYFAIR
MINING LTD.
SIZE
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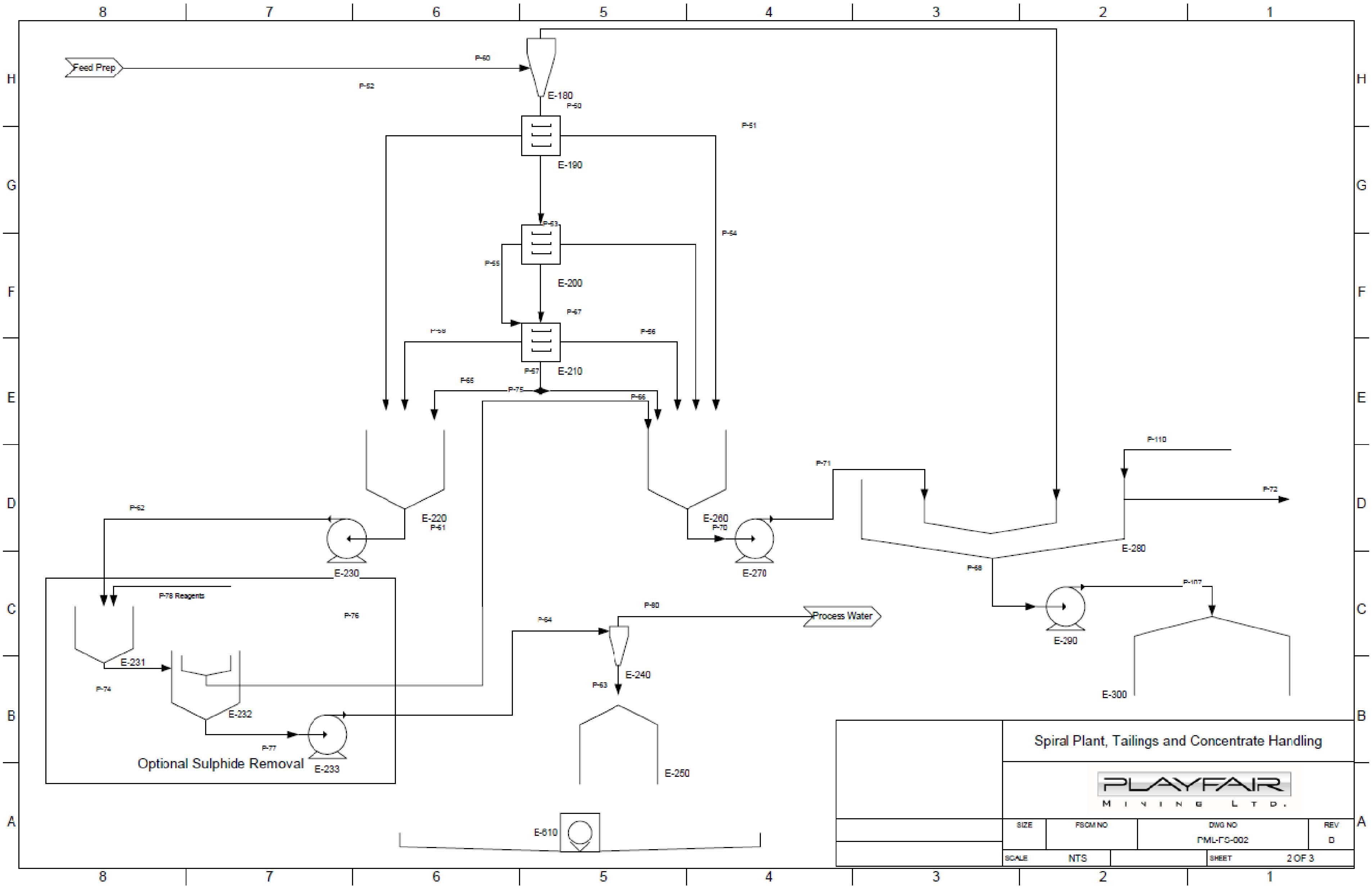
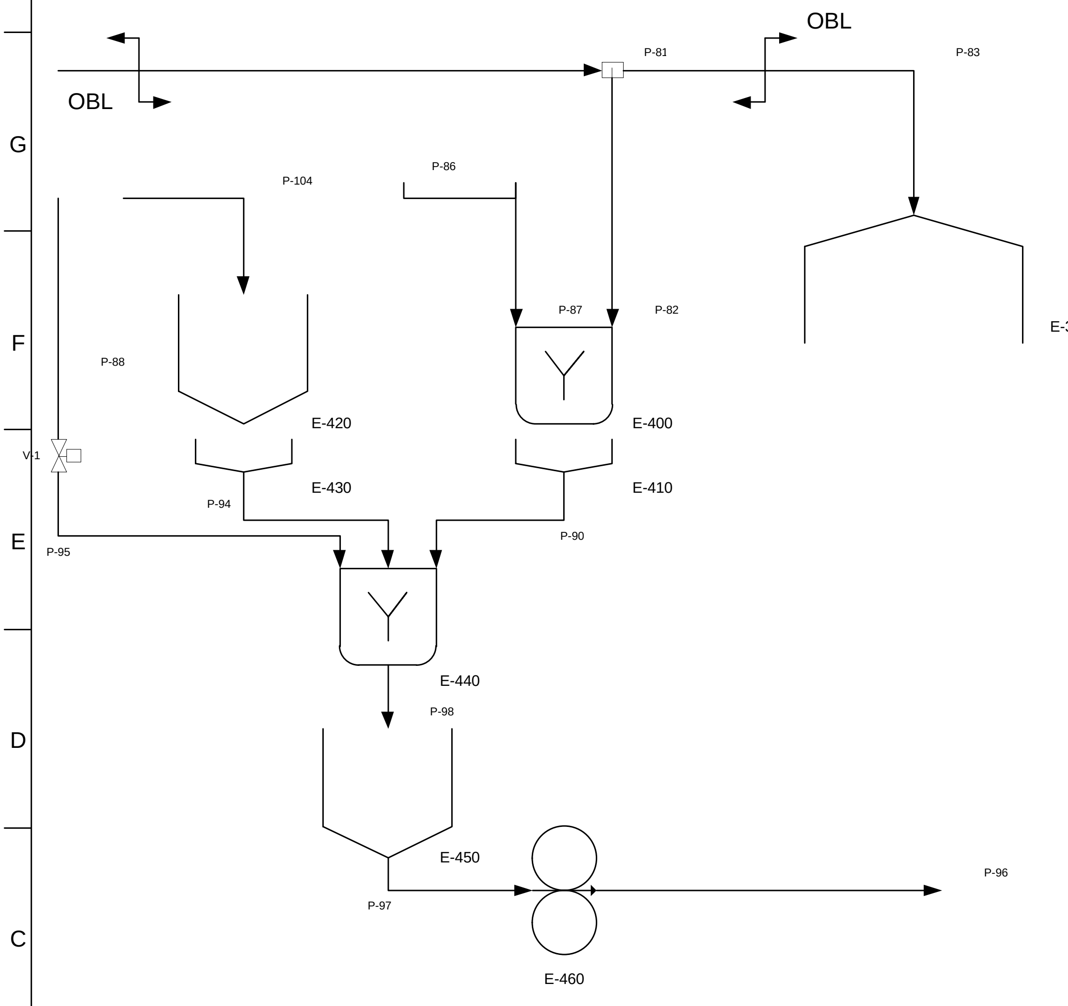
DWG NO
PML-FS001
REV
B
SCALE
1:1
SHEET
1 OF 3


Feed Prep
E-180
E-190
E-200
E-210
E-220
E-230
E-231
E-232
E-233
Optional Sulphide Removal
P-78 Reagents
E-240
E-250
E-260
E-270
E-280
E-290
E-300
E-610
Process Water
Spiral Plant, Tailings and Concentrate Handling
PLAYFAIR
MINING LTD.
SIZE
FSCM NO
DWG NO
PML-FS-002
REV
D
SCALE
NTS
SHEET
2 OF 3

Grey River Tungsten Equipment List - Concentrator					Rev B
Equipment Number	**Description**	**Type**	**Manufacturer**	**kW**	**Comment**
E-010	Feed hopper	5t			
E–020	Vibrating grizzly	1000 x 1200		2 x 1.9	40mm splay
E-030	Jaw crusher	430 x 250	Metso VB92	37	30mm CSS
E-040	Conveyor	10m L x 300mm W		5.5	1.25m/s
E-060	Screen	1500 x 1800		2 x 2.2	10mm PU deck
E-070	Conveyor	10m L x 300mm W		5.5	1.25m/s
E-080	Rougher sorter	ISS-X	Steinert	5.5	Sort -40 + 10mm
E-082	Product conveyor	10m L x 300mm W		5.5	1.25m/s
E-085	Waste conveyor	10m L x 300mm W		5.5	1.25m/s
E-086	Scavenger sorter	ISS-X	Steinert	5.5	Sort -40 + 10mm
E-087	Cleaner sorter	ISS-X	Steinert	5.5	Sort -40 + 10mm
E-089	Recycle conveyor	10m L x 300mm W		5.5	1.25m/s
E-090	Product hopper	5t			
E-095	Vibrating feeder	600x900	Vibramech	2x 2.2	
E-100	VSI		REMCO	30	6mm CSS
E-110	Conveyor	10m L x 300mm W			1.25m/s
E-120	Mill product screen	1800 x 2200	Vibramech	2x 2.2	1mm PU deck
E-130	Rod mill	1.2m x 1.8m EGL	Metso	200	Grate discharge
E-140	Mill dishcarge sump	3m3			
E-150	Mill discharge pump	2 x 1 1/2	Wier Envirotech	4	
E-160	Mill product sump	3m3			
E-170	Spiral feed pump	2 x 1 1/2	Wier Envirotech	4	
E-180	Desliming cyclone	150mm	Krebs		
E-190	Rougher spirals	SC 18/5 HM Spirals	Multotec		4x5 turn double start
E-200	Scavenger spirals	SC 18/5 HM Spirals	Multotec		4x5 turn double start
E-210	Scavenger table/spiral	Single deck	Wilfley	1.1	
E-220	Spiral concs sump	3m3			
E-230	Spiral concs pump	2x 1 1/2	Wier Envirotech	4	
E-231	Spiral concs conditioning tank	3m3	Aeromix	2.2	Optional
E232	Pyrite flotation	RCS 5	Metso	22	Optional
E-233	Pyrite flotation u/flow pump	2 x 1 1/2	Wier Envirotech	4	Optional
E-240	Product dewatering cyclone	150mm	Linatex spigot type		
E-250	Product stockpile	5t			
E-260	Spiral tails sump	3m3			
E-270	Talings underflow pump	2 x 1 1/2	Wier Envirotech	4	
E-280	Tailings thickener	5m BL Size 10	Metso	5.5	

E-290	Thickener u/flow pump	2 x 1 1/2	Wier Envirotech	4	
E-300	Tailings dam				
E-310	Process water tank	50m3			
E-320	Process water pump	2 x 1 1/2	Wier Envirotech	4	
E-500	Compressor	GA30	Atlas Copco	37	
E-510	Plant air receiver		Atlas Copco		
E-600	Spillage pump	3"DTV	Wier Envirotech	4	Mobile pump
E-610	Spillage pump	3"DTV	Wier Envirotech	4	Mobile pump
Total					



20 tph Pastefill Plant

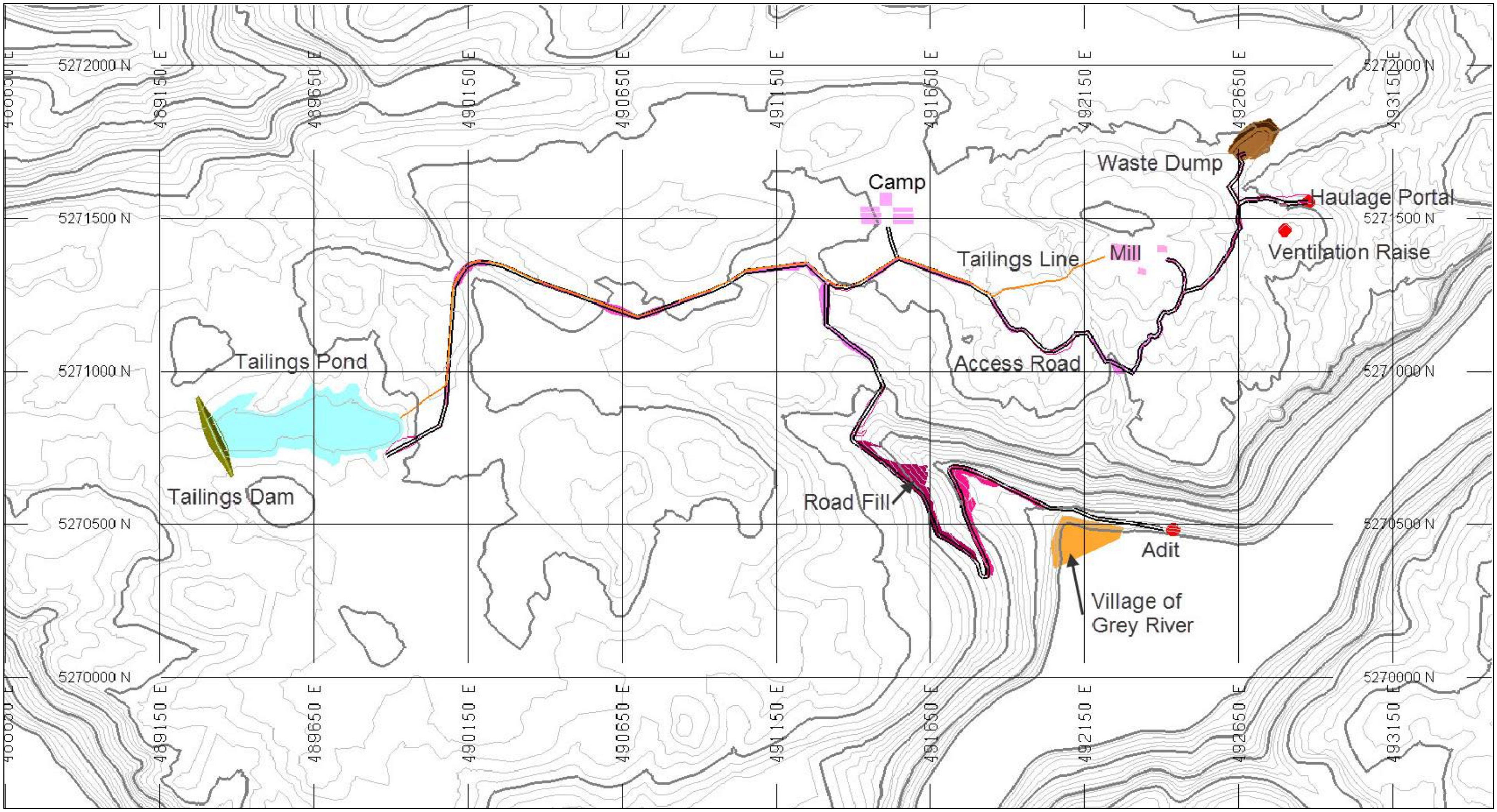
PLAYFAIR MINING LTD.

SIZE	FSCM NO	DWG NO	REV
		PML-FS-003	A
SCALE NTS		SHEET	3 OF 4

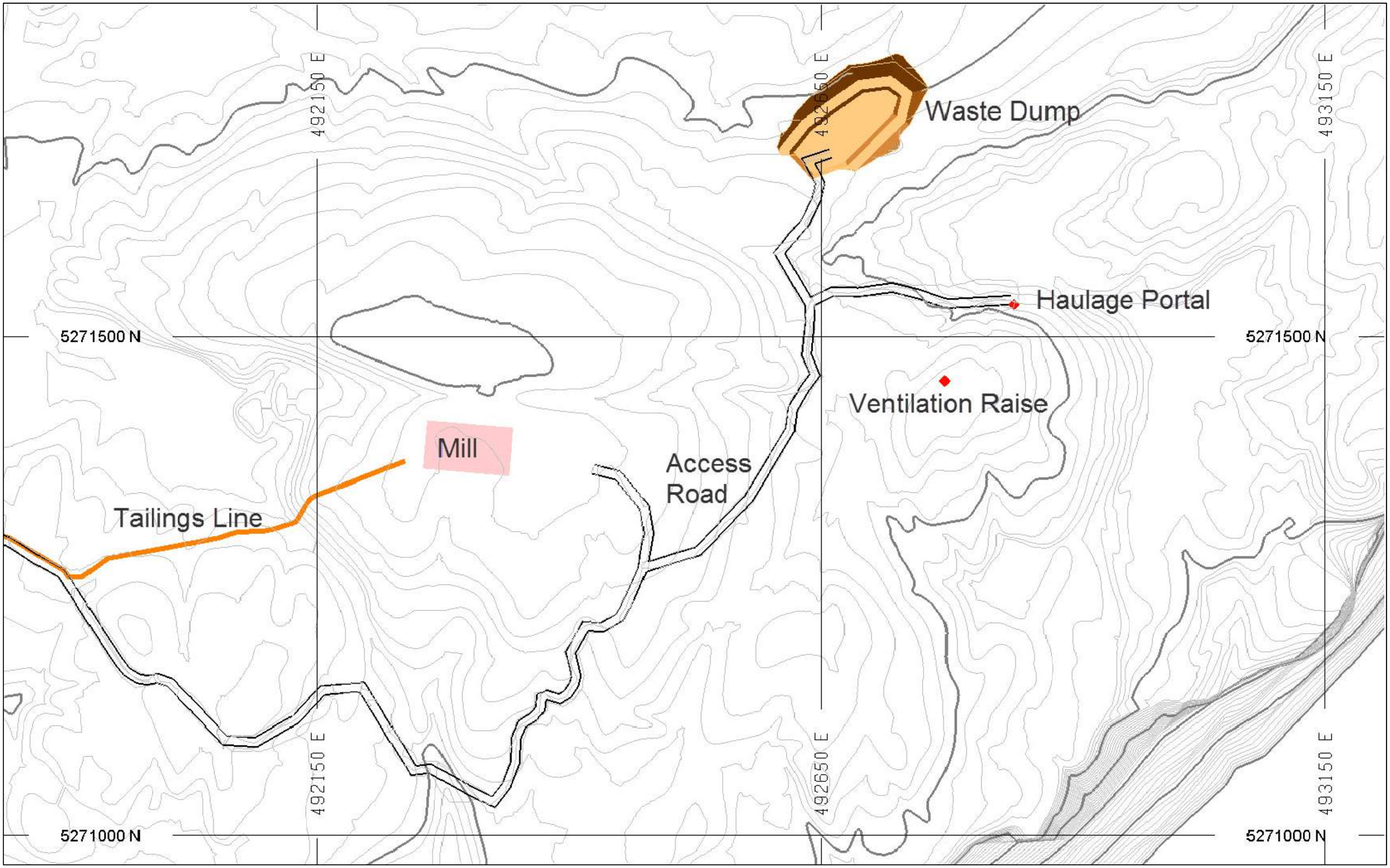
Grey River Tungsten Equipment List - Paste Plant					Rev A
Equipment Number	***Description***	***Type***	***Manufacturer***	***kW***	***Comment***
E-400	Paste thickener/mixer	7.9m x 11m h	PPSM	3	20 tph @ 78% solids
E-410	Weigh hopper				
E-420	Cement hopper	40m3			
E-430	Weigh hopper				
E-440	Paste mixer	2m3	PPSM	1.1	60 tph @ 78% solids
E-450	Paste storage hopper	4m3			
E-460	Paste delivery pump	8" mono	Netszch	22	60tph max @ 10MPa

APPENDIX C

Preliminary Site Layout Plans



Waste Dump
Camp
Haulage Portal
Tailings Line
Mill
Ventilation Raise
Tailings Pond
Access Road
Tailings Dam
Road Fill
Adit
Village of
Grey River
5272000 N
5271500 N
5271000 N
5270500 N
5270000 N
489150 E
489650 E
490150 E
490650 E
491150 E
491650 E
492150 E
492650 E
493150 E



Waste Dump
Haulage Portal
Ventilation Raise
Mill
Access Road
Tailings Line
5271500 N
5271500 N
5271000 N
5271000 N
492150 E
492650 E
493150 E
492150 E
492650 E
493150 E

APPENDIX D

Preliminary Pre-tax Cash Flow

Base Case - $16/lb

Spot Price - $21/lb

Project Cashflow - Grey River Project													
Project 11-1439-0003	**$USD**												
		Year -1	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Total**
Production													
Throughput, tpd	400												
Operating days per year	365												
Tonnes Milled			73,000	146,000	146,000	146,000	146,000	146,000	146,000	146,000	146,000	27,306	**1,268,306**
Grade	%WO3		0.524	0.505	0.473	0.473	0.525	0.522	0.533	0.530	0.613	0.630	
Mill Recovery	85%		85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	
Recovered pounds conc.	lbs		716,837	1,382,996	1,295,273	1,295,273	1,435,593	1,429,204	1,458,262	1,450,009	1,676,610	322,439	**12,462,496**
	Tonnes		325	627	588	588	651	648	661	658	761	146	**5,653**
Net Revenue	$16.00 USD/lb		$11,469,388	$22,127,933	$20,724,362	$20,724,362	$22,969,489	$22,867,259	$23,332,197	$23,200,145	$26,825,765	$5,159,031	$199,399,931
Operating Cost													
Mining	$80.00		$5,840,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$2,184,507	$101,464,507
Milling	$11.50		$839,500	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$314,023	$14,585,523
G&A	$15.00		$1,095,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$409,595	$19,024,595
Shipping&Insurance	$1.00		$73,000	$146,000	$146,000	$146,000	$146,000	$146,000	$146,000	$146,000	$146,000	$27,306	$1,268,306
Total Operating Cost	$107.50	$0	$7,847,500	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$2,935,431	$136,342,931
Capital													
Total Capital	$50,040,100	$29,515,195	$5,316,140	$1,898,660	$1,668,410	$3,863,760	$1,618,160	$1,568,160	$1,711,785	$879,830	$0	$2,000,000	$50,040,100
Pre-production capital	$32,173,265	$29,515,195	$2,658,070										
Sustaining Capital	$17,866,835		$2,658,070	$1,898,660	$1,668,410	$3,863,760	$1,618,160	$1,568,160	$1,711,785	$879,830	$0	$2,000,000	$15,208,765
Salvage	$2,502,005											$2,502,005	$2,502,005
Net Cashflow (Pre-tax)	$15,518,905	($29,515,195)	($1,694,252)	$4,534,273	$3,360,952	$1,165,602	$5,656,329	$5,604,099	$5,925,412	$6,625,315	$11,130,765	$2,725,605	$15,518,905
Pre-tax NPV	5.0%	$2,861,000											
IRR		7%											

Project Cashflow - Grey River Project

Project 11-1439-0003 **$USD**

		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Production													
Throughput, tpd	400												
Operating days per year	365												
Tonnes Milled			73,000	146,000	146,000	146,000	146,000	146,000	146,000	146,000	146,000	27,306	**1,268,306**
Grade	%WO3		0.524	0.505	0.473	0.473	0.525	0.522	0.533	0.530	0.613	0.630	
Mill Recovery	85%		85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	
Recovered pounds conc.	lbs		716,837	1,382,996	1,295,273	1,295,273	1,435,593	1,429,204	1,458,262	1,450,009	1,676,610	322,439	**12,462,496**
	Tonnes		325	627	588	588	651	648	661	658	761	146	**5,653**
Net Revenue	$21 USD/lb		$14,910,204	$28,766,313	$26,941,671	$26,941,671	$29,860,336	$29,727,437	$30,331,856	$30,160,188	$34,873,495	$6,706,741	$259,219,910
Operating Cost													
Mining	$80.00		$5,840,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$11,680,000	$2,184,507	$101,464,507
Milling	$11.50		$839,500	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$1,679,000	$314,023	$14,585,523
G&A	$15.00		$1,095,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$2,190,000	$409,595	$19,024,595
Shipping&Insurance	$1.00		$73,000	$146,000	$146,000	$146,000	$146,000	$146,000	$146,000	$146,000	$146,000	$27,306	$1,268,306
Total Operating Cost	$107.50	$0	$7,847,500	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$15,695,000	$2,935,431	$136,342,931
Capital													
Total Capital	$50,040,100	$29,515,195	$5,316,140	$1,898,660	$1,668,410	$3,863,760	$1,618,160	$1,568,160	$1,711,785	$879,830	$0	$2,000,000	$50,040,100
Pre-production capital	$32,173,265	$29,515,195	$2,658,070										
Sustaining Capital	$17,866,835		$2,658,070	$1,898,660	$1,668,410	$3,863,760	$1,618,160	$1,568,160	$1,711,785	$879,830	$0	$2,000,000	$15,208,765
Salvage	$2,502,005											$2,502,005	$2,502,005
Net Cashflow (Pre-tax)	$75,338,884	($29,515,195)	$1,746,564	$11,172,653	$9,578,261	$7,382,911	$12,547,176	$12,464,277	$12,925,071	$13,585,358	$19,178,495	$4,273,315	$75,338,884
Pre-tax NPV	5.0%	$46,790,000											
IRR		27%											

Africa	+ 27 11 254 4800
Asia	+ 86 21 6258 5522
Australasia	+ 61 3 8862 3500
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solutions@golder.com
www.golder.com

Golder Associates Ltd.
500 - 4260 Still Creek Drive
Burnaby, British Columbia, V5C 6C6
Canada
T: +1 (604) 296 4200