DALRADIAN
RESOURCES

Curraghinalt Gold Deposit, Northern Ireland

Mineral Resource Estimate Update
NI 43-101 Technical Report

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IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 (NI 43-101) Technical Report, in accordance with Form 43-101F1, for Dalradian Resources Inc. by T. Maunula & Associates Consulting Inc. in order to update the Mineral Resources Estimate for the Curraghinalt Gold Deposit.

The Curraghinalt Deposit was the subject of a Preliminary Economic Assessment (PEA) in 2012 (Hennessey et al 2012b) based on a previous mineral resource (2012a). An update to the PEA is being planned to reflect the change in the Mineral Resource, as well as using revised economic parameters.

This report is intended to be used by Dalradian Resources Inc. (Dalradian Resources) subject to the terms and conditions of its agreement with T. Maunula & Associates Consulting Inc. That agreement permits Dalradian Resources to file this report as a National Instrument 43-101 Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report, by any third party, is at that party's sole risk.

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Carbon-in-Leach CIL

CDN Resource Laboratories Ltd.	CDN Resource
Coefficient of Variation	CV
Crown Estate Commissioners	CEC
Cubic feet per minute	CFM
Dalradian Gold Limited	Dalradian Gold
Dalradian Gold Limited	DGL
Dalradian Resources Inc.	Dalradian Resources
Department for Regional Development	DRD
Department of Agriculture and Rural Development	DARD
Department of Culture, Arts and Leisure	DCAL
Department of Enterprise, Trade and Investment	DETI
Department of Health, Social Services and Public Safety	DHSSPS
Department of the Environment	DOE
Distributed Control System	DCS
Dollars per tonne	$/t
Ennex International plc	Ennex
Environmental Impact Assessment	EIA
Environmental Statement	ES
Fisheries Conservancy Board for Northern Ireland	FCBNI
G&T Metallurgical Services Ltd	G&T
Galantas Gold Corporation	Galantas
General and Administrative	G&A
Geological Survey of Northern Ireland	GSNI
Global Positioning System	GPS
Golder Associates UK	Golder
Gram per litre	g/L
Grams per centimetre cubic	g/cc
Grams per tonne	g/t
Health and Safety Executive Northern Ireland	HSENI
Heavy Media Separation	HMS
Horsepower	HP
Hours per day	h/d
Industrial Pollution and Radiochemical Inspectorate	IPRI
Internal Rate of Return	IRR
International Finance Corporation	IFC
Irish National Accreditation Board	INAB
Kilogram per tonne	kg/t
Kilometre square	km^2
Kilowatt hour per tonne	kWh/t

Kilowatt	kW
Less than	<
Life-of-Mine	LOM
Load-Haul-Dump	LHD
Megapascal	MPa
Metal Leaching / Acid Rock Drainage	ML/ARD
Metre cubic per hour	m^3/h
Metres above sea level	masl
Metres cubic per second	m^3/s
Micrometre (micron)	µm
Millimetre per year	mm/y
Millimetre	mm
Million ounces	Moz
Million tonnes	Mt
More than	>
Motor Control Centre	MCC
Net Present Value	NPV
Net Smelter Return	NSR
Nickelodeon Minerals Inc.	Nickelodeon
Non-Acid Generating	NAG
Northern Ireland Electricity Ltd.	NIE
Northern Ireland Housing Executive	NIHE
Northern Ireland Tourist Board	NITB
Northern Ireland Water	NIW
OMAC Laboratories	OMAC
Ordinary Kriging	OK
Ounces per year	oz/y
Parts per million	ppm
Percent	%
Potentially Acid Generating	PAG
Preliminary Economic Assessment	PEA
Programmable Logic Controllers	PLCs
Qualified Persons	QP
Quality Assurance/Quality Control	QA/QC
Right-of-Way	ROW
Rio Tinto Finance & Exploration	RioFinex
Rock Quality Designation	RQD
Rocklabs Ltd. (New Zealand)	Rocklabs
Run-of-Mine	ROM

Semi-Autogenous Grinding	SAG
SGS Mineral Services	Lakefield
SLR Consulting	SLR
Special Areas of Conservation	SACs
Specific Gravity	SG
Standard Deviation	SD
Standard Reference Material	SRM
Strongbow Exploration Inc.	Strongbow
Tailings Storage Facility	TSF
Three-dimensional	3D
T. Maunula & Associates Consulting Inc.	TMAC
Tonne (metric ton)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per metre cubic	t/m^3
Tonnes per year	t/y
Tournigan Gold Corp	Tournigan
Two-dimensional	2D
Tyrone Volcanic Group	TVG
Underground	UG
United Kingdom official currency	GBP
United Kingdom	UK
United States dollars	US$
Versatile Time Domain Electromagnetic	VTEM
Voice over Internet Protocol	VoIP
Volcanogenic Massive Sulphide	VMS
Volt	V
Weak Acid Dissociable	WAD
Weighted Average Cost of Capital	WACC
Yards	yd^3

1 SUMMARY

1.1 Scope of Work

At the request of Dalradian Resources Inc. (Dalradian Resources), T. Maunula & Associates Consulting Inc. (TMAC) has prepared a Mineral Resource estimate for the Curraghinalt gold deposit within that Company's Northern Ireland Property in County Tyrone and County Londonderry, Northern Ireland. The effective date of the Mineral Resource estimate is January 20, 2014. The Curraghinalt gold deposit was the subject of a Preliminary Economic Assessment (PEA) in 2012 (Hennessey et al., 2012b) based on the previous mineral resource (2012a). The findings from the PEA are repeated in Sections 15 to 19, and Sections 21 and 22 of this report for completeness. The 2012 mineral resource upon which the PEA is based is summarized in Sections 14.1 to 14.5. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

This Technical Report, prepared in accordance with the reporting standards and definitions required under Canadian National Instrument (NI) 43-101, summarizes the results of that study.

Dalradian Resources indirectly hold the Northern Ireland Property through a wholly-owned subsidiary, Dalradian Gold Limited (Dalradian Gold), in whose name the licences and option agreements are registered. Where the distinction is considered minor, the two companies are referred to herein simply as Dalradian.

1.2 Property Description and Location

The Northern Ireland Property (formerly the Tyrone project) is located in County Tyrone and County Londonderry, Northern Ireland. The Curraghinalt gold deposit, located near the centre of the property, in County Tyrone, is approximately 127 km west of Belfast by road and 15 km northeast of the town of Omagh. Access to the Curraghinalt deposit is via a number of paved highways and local roads.

The Department of Enterprise, Trade and Investment (DETI) has granted to Dalradian, Prospecting Licences for base metals on four contiguous areas referred to as DG1, DG2, DG3 and DG4. The Crown Estate Commissioners (CEC) has entered into Option Agreements with Dalradian for gold and silver over the same four areas.

The current DETI Prospecting Licences for DG1 and DG2 (named DG1/14 and DG2/14) expire December 31, 2015, at which point they can be extended for another two years. The Prospecting Licences for DG3 and DG4 (named DG3/11 and DG4/11) have a renewal term expiring April 23, 2015 at which point they can be extended for another two years. CEC Option Agreements for DG1 and DG2 have a renewal term expiring December 31, 2015. The Option Agreements for DG3 and DG4 have a renewal term expiring April 23, 2015. The CEC Option Agreements have a two-year term and can be renewed indefinitely at the CEC's discretion.

The mineral resource estimate presented in this report is located entirely on the property covered by licence DG1. A net smelter return (NSR) royalty of 2% is payable to Minco plc on a portion of DG1. As provided in the option agreements, a 4% royalty will be payable to the CEC upon production of silver and/or gold on the Northern Ireland Property

1.3 Geology and Mineralization

The Northern Ireland Property is host to the Curraghinalt gold deposit, an orogenic, high grade, lode gold deposit located in DG1. Mineralization is found in a stacked sulphidic, quartz-carbonate vein system that strikes approximately west-northwest and dips approximately 55° to 75° to the northeast. The current Mineral Resource includes 12 main vein zones, each of which is anchored by a shear vein (D vein) and may include numerous adjacent extensional veinlets (C veins). The known strike length of the vein systems is 1,900 m and D veins have been intersected at a vertical depth of over 1,000 m below surface.

Additionally, on licence DG2, there are known occurrences of gold mineralization in silicified rhyolite breccias, and porphyry-style copper mineralization. It has been postulated that the DG2 licence is also a potential host for volcanogenic massive sulphide (VMS) style mineralization due to stratigraphic correlations with mineral belts in Newfoundland and Scandinavia.

The Curraghinalt deposit is located 3 km to the north of the northeast-southwest striking Omagh Thrust Fault. This major fault has thrust Dalradian Supergroup rocks from the northwest over the Ordovician-aged Tyrone Volcanic Group (TVG) rocks, part of the Tyrone Igneous Complex, located to the south. The Dalradian metasediments on the northern side of the thrust strike northeast-southwest and dip to the northwest

The Dalradian metasedimentary formations present in the deposit area are, from the southeast (at the Omagh Thrust) to the northwest and, from the youngest to the oldest, the Mullaghcarn, Glengawna and Glenelly Formations. The Mullaghcarn Formation, which hosts the Curraghinalt deposit, consists of metasemi-pelites, meta-psammites, and metapelites.

The upper part of the TVG comprises bimodal sequences of basaltic and andesitic to rhyolitic submarine and subaerial lavas, volcaniclastics with chert horizons and intercalations of graptolitic shales. These lithologies are preceded by submarine basaltic andesite pillow lavas and associated intrusives. A series of porphyry bodies and a series of calc-alkaline granitic intrusions intrude the volcano-sedimentary sequence.

1.4 History

The DG1 and DG2 licence areas were initially acquired in 1981 by Ulster Base Metals Limited (later known as Ulster Minerals) an entity, which later became a wholly-owned subsidiary of Ennex International plc (Ennex). Ennex conducted exploration on the property between 1986 and 1999. Ennex sold its interest in Ulster Minerals to Nickelodeon Minerals Inc. (Nickelodeon) in January 2000. In August 2000, the name of Nickelodeon was changed to Strongbow Resources Inc. and subsequently to Strongbow Exploration Inc. (Strongbow).

In February 2003, Tournigan entered into an option agreement with Strongbow to earn an interest of up to 100% in the Curraghinalt deposit. At the same time, Tournigan entered into a similar option agreement with Strongbow in respect of its "Tyrone Project" in the TVG. Tournigan established Dalradian Gold as a wholly-owned subsidiary through which it would earn its interests in the Curraghinalt and Tyrone properties which were subsequently converted to licences UM-1/02 and UM-2/02.

In the following year (February 2004), Tournigan entered into a letter agreement with Strongbow for the outright purchase by Tournigan of all of the issued and outstanding shares of Ulster Minerals. A net smelter royalty of 2% held by Ennex was transferred to Minco plc. Full transfer of ownership in Ulster Minerals to Tournigan was completed in December 2004.

During this period, Tournigan also applied for and received the licences which later became DG3 and DG4 and which lie to the northwest of licences UM-1/02 and UM-2/02. As well, licences UM-1/02 and UM-2/02 were converted in name to licences DG1 and DG2 and their internal boundary was moved to reflect the location of the Omagh Thrust, the boundary between the Dalradian metasediments and the TVG.

In September 2009, Dalradian Resources entered into an agreement with Tournigan to purchase Dalradian Gold and all of its Northern Ireland assets.

Prior to the work of Ennex/Ulster Base Metals, gold was recognized in the gravels of the Moyola River as early as 1652 and, in the 1930s, an English company reported plans for alluvial gold mining in a prospectus, but very little work appears to have occurred. Other companies held the licences prior to Ulster Base Metals and the Geological Survey of Northern Ireland (GSNI) completed a report on the gold potential of the area.

Between 1983 and 1997, Ennex conducted a significant amount of exploration drilling at Curraghinalt and other targets in the area such as Cashel Rock. Ennex also completed a program of underground development on two of the veins at Curraghinalt. Nickelodeon and Strongbow completed very little work on the property.

Due to the relatively advanced stage of historical exploration completed at the Curraghinalt deposit by the previous operators of the property, in 2003 Tournigan initially moved directly to infill and deeper drilling and only undertook a small amount of additional exploration. Some geophysical surveys had been carried out on a regional scale to delineate drill targets along strike and some prospecting, sampling and mapping, sufficient to retain the other licences, was completed. Tournigan also compiled an extensive database of available historical exploration information and data.

The drill program allowed Tournigan to report a mineral resource estimate for Curraghinalt in 2007 consisting of both Indicated and Inferred resources (Mukhopadhyay, 2007). Micon performed this estimate. After completion of the resource estimate, and before halting exploration at the deposit, Tournigan completed four more holes to intersect deeper mineralization, which were not available to be used in completion of the resource estimate at the time.

1.5 Exploration

After acquisition of the property by Dalradian Resources, an updated mineral resource estimate was prepared that included the four additional holes drilled by Tournigan after the 2007 mineral resource (Hennessey and Mukhopadhyay, 2010). This was published in a Technical Report supporting the listing of Dalradian Resources.

Since acquisition of the Northern Ireland property, Dalradian Resources has embarked upon a campaign of drilling largely concentrated on the Curraghinalt deposit. A mineral resource estimate was prepared in late 2011(Hennessey et al., 2012a) followed by a PEA (Hennessey et al., 2012b). Since the 2011 resource, Dalradian Resources has completed an additional 83 holes for 28,353 m at Curraghinalt. The Company has also completed 1,034 line-km of helicopter-borne electromagnetic and magnetic geophysical surveys and commenced an expansion of the soil geochemistry grid.

1.6 Mineral Resources

1.6.1 Current Mineral Resource

Since freezing the database for the 2011 mineral resource estimate (Hennessey et al., 2012a), diamond drilling continued at Curraghinalt. In total, an additional 28,353 m were drilled in 83 holes, and were used in the current mineral resource. Additional sampling of 100 historical drill holes (12,346 m) was also completed.

The Curraghinalt deposit represents an orogenic gold deposit with parallel veins and sub-veins or veinlets which have been traced by trenching and drilling over a strike length of approximately two kilometers. Twenty-one D veins were interpreted and comprise the primary zones of mineralization in this updated resource model.

The updated mineral resource (which is the basis for this technical report) for Curraghinalt deposit is tabulated in Table 1-1 at a cut-off grade of 5.00 g/t Au. The effective date of the updated resource is January 20, 2014. This resource is exclusive of the underground development which is estimated at 1,700 tonnes at 15.73 g/t Au or 881 oz.

Table 1-1: Curraghinalt Deposit Resource Statement (Cut-off Grade 5.00 g/t Au)

Resource Class	Tonnage (Kt)	Au g/t	Ag g/t	Cu %	Contained Au (Koz)
Measured	23.3	20.15	7.54	0.02	15.1
Indicated	2,976.2	10.34	3.85	0.08	989.0
Measured + Indicated	2,999.5	10.41	3.88	0.08	1,004.1
Inferred	8,005.7	9.67	3.94	0.07	2,487.7

No minimum width constraint was applied before reporting this updated resource. The interpretation method of including a minimum of two metres of downhole length to define a vein resulted in an average horizontal thickness of 2.57 m.

To the best knowledge of the authors, the stated mineral resources are not materially affected by any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant issues. No known mining, metallurgical, infrastructure, or other factors materially affect this mineral resource estimate.

The Curraghinalt mineral resource estimate is compliant with the current CIM standards and definitions as required under NI 43-101 and is, therefore, reportable as a mineral resource by Dalradian Resources.

1.6.2 Previous Mineral Resource

Information in Sections 14.1 to 14.5, and summarized here, is taken from the Preliminary Economic Assessment (PEA) published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November, 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report.

In order to prepare the 2011 Micon mineral resource estimate, the mineralized area was again divided into different zones with statistical analysis carried out separately on all zones. Resources were estimated by the inverse distance interpolation method with selected checks on larger veins by ordinary kriging. Vein thickness and metal accumulation (grade times thickness) were interpolated into the model with the block grade determined by division of the metal accumulation by interpolated thickness.

The resource was classified following the standards and definitions of the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM). The mineral resource was reported at an economic cut-off grade of 5 g/t Au over a minimum horizontal thickness of 1 m. All mineralized blocks in the block model which were greater than 0.10 m in thickness were diluted to 1 m at zero grade (for the diluting material) and reported if they then met the cut-off grade.

The results of the 2011 mineral resource estimate, as originally reported in Hennessey et al., (2012a) are presented in Table 1-2. This estimate is based on exploration data available as of October 10, 2011 and is current as of November 30, 2011.

Table 1-2: 2011 Classified Mineral Resource Estimate at 5 g/t Au Cut-off Grade and 0.1 m Minimum Width, Diluted to a Minimum Width of 1.0 m

Resource Category	Tonnage (Mt)	Grade (g/t Au)	Contained Metal	
			(t)	(Moz)
Measured	0.02	21.51	0.44	0.01
Indicated	1.11	12.84	14.20	0.46
Measured + Indicated	1.13	13.00	14.65	0.47
Inferred	5.45	12.74	69.44	2.23

After the public disclosure of the Curraghinalt mineral resource estimate on November 30, 2011, and its publication in Hennessey et al. (2012a), silver and copper grades were estimated into the block model. This was done to allow for the contribution of silver to the cash flow for the PEA and to model copper's effect on cyanide consumption. Silver and copper grades were estimated using the same techniques as described for gold above. In the PEA base case, no copper concentrate is produced and no revenue from copper is in the cash flow. Silver accounts for approximately 1% of revenue forecast in the economic analysis.

The amended Curraghinalt deposit mineral resource estimate showing copper and silver grades is set out in Table 1-3. No changes in tonnages or gold grades have occurred. In order to make it comparable to Table 1-2, Table 1-3 is also reported at a cut-off grade of 5.0 g/t Au. No gold equivalent calculations were made.

Table 1-3: Curraghinalt Mineral Resource Estimate Amended to Include Silver and Copper

Category	Notes	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Cu (%)
Measured		0.02	21.51	17.56	0.49
Indicated		1.11	12.84	4.05	0.17
Measured + Indicated		1.13	13.00	4.29	0.18
Inferred	With Cu*	5.38	12.77	5.48	0.12
	Without Cu*	0.08	10.37	2.00	-

Notes: * Copper assay data were not available for the 752 and D veins and no copper grades were estimated there.
1. Original vein thickness blocks of less than 0.10 m were eliminated from the resource tabulation. There were minor differences between a 0.00-m and 0.10-m minimum thickness estimates which disappear with rounding. For horizontal vein thicknesses of more than 0.10 m, but less than 1.0 m, the grades were diluted to a minimum horizontal width of 1.0 m at zero grade prior to reporting.
2. Mineral resources are reported at a cut-off grade of 5 g/t Au after dilution.
3. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
4. The mineral resource estimate was prepared by Dibya Kanti Mukhopadhyay, MAusIMM (CP), under the overall direction of B. Terrence Hennessey, P.Geo. Mr. Hennessey has taken responsibility for the estimate in this report. The Curraghinalt resource estimate is compliant with the current standards and definitions required under NI 43-101.

1.7 Mining

Information in this section is taken from the Preliminary Economic Assessment (PEA) published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

Mineralized veins at the Curraghinalt deposit will be extracted using the Longitudinal Sublevel Retreat Underground Mining method with both paste and waste backfill. This backfill method was chosen in order to maximize the extraction of the high value resources, reduce the surface footprint required for tailings disposal, and allow savings in the volume and transportation cost of waste rock from underground development.

Access to the Curraghinalt deposit will be via a main decline from the surface. The main decline will be connected by cross-cuts to sublevels providing access to the mineralized veins. Sills, excavated along the mineralized veins on each sublevel, provide access to the mining stopes where production drilling, blasting, and extraction of the mineralized material take place.

The extraction of mineralized material from underground will be carried out with rubber tired mechanized equipment to maximize production and provide flexibility to the underground operations. Some key parameters are:

- mining method is longitudinal sublevel retreat
- decline or ramp dimensions are 4.5 m H x 4.5 m W at 15% grade
- bypasses at the decline or ramp at 4.5 m H x 4.5 m W x 3.0 m L
- safety bays along the decline and ramp at 3.0 m H x 3.0 m W x 3.0 m L
- sumps at 3.0 m H x 4.0 m W x 3.0 m L
- cross-cut dimension at 4.0 m H x 4.0 m W
- sill or development in ore at 3.0 m H x 3.0 m W
- refuge station/lunch room at 4.0 m H x 4.0 m W x 10.0 m L
- drift to ventilation shafts and remuck bays at 4.0 m H x 4.0 m W
- ventilation raise and ore or waste-pass at 3 m diameter
- level intervals at 20 vertical metres from the floor of the top sill to the floor of the bottom sill.

The mineral resource considered in the mine plan for Curraghinalt was estimated based on the following parameters:

- tonnages above the 5.0 g/t Au cut-off grade
- non-recoverable crown pillar extending 20 m below and parallel to the topography
- exclusion from the mine plan of resources below the -490 m level
- mining recovery of 95%
- dilution of mineral resource to minimum mining width of 1.8 m for stopes and 3 m for sills.

Table 1-4 summarizes the mineral resource considered in the mine plan, diluted to 1.8 m mining width at 95% recovery with external dilution.

Table 1-4: Measured, Indicated, and Inferred Mineral Resources Considered in the Mine Plan

Description	Tonnage (Mt)	Avg. Au (g/t)	Avg. Ag (g/t)	Avg. Cu (%)
Measured	0.02	16.46	14.19	0.40
Indicated	1.42	8.76	2.77	0.12
Total Measured & Indicated	**1.44**	**8.88**	**2.95**	**0.12**
Total Inferred	**7.98**	**7.91**	**3.38**	**0.08**

Sills at 3.0 m H x 3.0 m W in the mineralization veins will be developed when the cross-cut intercepts the veins. The sills will be developed in both directions along the strike length of the mineralized zone, providing two production faces for each vein on each level.

Mining in a production level will commence from one end of the veins, retreating to the level's crosscut. The availability of numerous production headings when a cross-cut intercepts the veins within the level provides multiple production headings to meet the 1,700 t/d production target.

The main decline into the mineralized zones can act as fresh air intake into the mine. Exhausted and contaminated air can be channelled through one of the two ventilation raises or directed into the existing exploration adit on 170 m elevation to the surface.

Ventilation from one level to another will be provided by the ventilation raises located in the cross-cuts. Ventilation drifts connect the ventilation raises to the cross-cuts to supply fresh or exhaust air from the underground workings.

Figure 1-1 presents typical schematic of a longitudinal sublevel retreat mining method with multiple mining levels. The total meterage of development in the LOM plan for the Curraghinalt deposit is presented in Table 1-5.

Figure 1-1: Isometric View of Curraghinalt Project Mine Layout

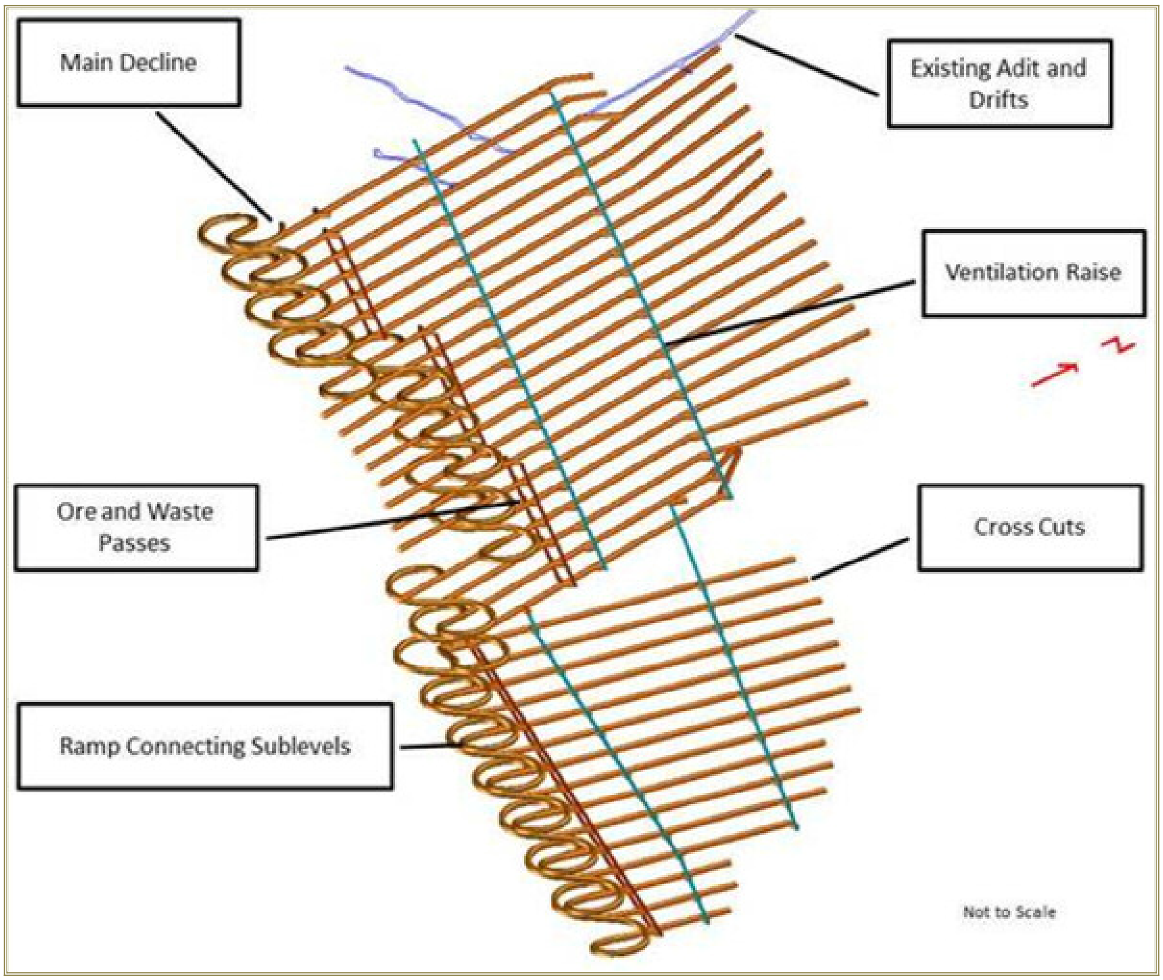


Table 1-5: Total Underground Development for the Curraghinalt Project

Description	Total Length (m)
Ramp	5,325
Ramp By Passes	157
Ramp Safety Bays	120
Sump in Ramp	120
Cross Cuts (including refuge stations and service drifts)	16,614
Sumps in Cross Cut	137
Sill	207,420
Sill (Through Waste)	7,152
Vent, Ore and Waste Pass	2,990
Total	**240,035**

Table 1-6 summarizes the proposed equipment fleet required to develop and extract 1,700 t/d of mineralized material from the Curraghinalt deposit. The table presents the average number of equipment units over the life-of-mine (LOM).

Table 1-6: Underground Mobile Equipment Fleet

Description	Units (Avg. LOM)
Stoping Drill (e.g., Boart Stopemate)	4
Stoping Drill (Electric-hydraulic)	1
Sill Narrow Vein Jumbo	5
Development Jumbo (Double Boom)	1
Bolter	1
LHD at 4.0 m^3	1
LHD at 3.0 m^3	4
Trucks -30-ton	3
Explosive Truck	1
Scissor Truck	1
Fuel and Lube Truck	1
Mechanic Light Pickup	1
Electrician Light Pickup	1

Description	Units (Avg. LOM)
Surveying Light Pickup	1
Light Pick-Up (Service and Supervision)	3
Man Carrier	1
UG Grader	1

A summary of the underground labour force requirements is presented in Table 1-7. Additionally, the Project's general and administrative (G&A) and technical management will require staff as outlined in Table 1-8.

Table 1-7: Estimate Mine Labour Requirement

Description	Labour Force (Avg. LOM)
Stoping Production Driller	12
Sill Narrow Vein Jumbo Driller	13
Development Jumbo (Double Boom) Driller	2
Bolter Driller	3
Trucks Drivers	9
Backfill Crew	4
Grader Operator	3
Blaster – Production	4
Blaster – Development	6
Mechanics and Electricians	18
Surveyor	4
General Miner/Helper	4
Sampler	6
Total	**56**

Table 1-8: General and Administrative Labour Force

Description	Labour Force (Avg. LOM)
General Administration	
General Manager/Mine Superintendent	1
Health & Safety and Training Officer	6
Mine Technical & Engineering	
Chief/Senior Mining Engineer	1
Intermediate Mining Engineer	3
Intermediate Draft Person	4
Senior Surveyor	1
Chief/Senior Geologist	1
Intermediate Geologist	3

Description	Labour Force (Avg. LOM)
Senior Mechanical Engineer	1
Mine Technicians	4
Salaried Staff – Mining	
Administrative Assistant	1
Mine General Foreman	1
Mine Supervisor/Shift Boss (Production)	6
Maintenance Supervisor/Shift Boss	3
Total	**36**

1.8 Metallurgy and Processing

Information in this section is taken from the Preliminary Economic Assessment (PEA) published by Micon in 2012 (Hennessey et al., 2012b) and is supplemented with additional metallurgical testwork information current to the effective date of this report. The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

1.8.1 Mineralogy and Metallurgical Testing

Mineralogical work has indicated that gold mineralization at Curraghinalt occurs in quartz-pyrite veins and is associated with variable abundances of carbonate, chalcopyrite and tennantite-tetrahedrite. In general, carbonate, chalcopyrite, and tennantite-tetrahedrite are paragenetically later than quartz and pyrite, and fill fractures in the latter. Gold occurs mainly as the native metal and more rarely as electrum (>20 wt% Ag), and is found primarily along fractures in pyrite, as inclusions in pyrite, and at pyrite grain contacts with carbonate and quartz.

The PEA base case proposes a conventional crushing, milling, and carbon-in-leach gold extraction flowsheet. Comminution tests have returned Bond ball millwork indices in the range 15.2 kWh/t to 15.4 kWh/t. The abrasion index on composite sample 12-1B returned a value of 0.1278 g. Cyanidation testwork has generally returned very high metal extractions—typically 95% or better for gold and about 80% for silver. A grind of approximately 85% passing 200 mesh (75 µm) and 48 h leach at NaCN 1g/L was generally effective in the earlier testwork.

Additional testwork has investigated options for the pre-treatment of the mill feed using heavy media separation (HMS), the production of gravity and/or flotation concentrates, and the possible sale or intensive leaching of these concentrates. However, evaluation of these suggests the most effective flowsheet is the whole-ore leach reflected in the PEA base case.

1.8.2 Process Flowsheets Considered

The four process flowsheet options considered and economically evaluated for this PEA study are:

- Option A: Whole Ore Leach
- Option B: Gravity concentration, sale of gravity concentrate followed by Bulk Flotation Concentrate, Leach of concentrate
- Option C: Gravity concentration followed by Bulk Flotation Concentrate – Sale of both concentrates
- Option D: Copper Flotation, with sale of the copper concentrate and Pyrite Flotation Concentrate with leach of the Pyrite concentrate.

Operating and capital costs for each of these four options were developed, as well as revenue estimates. All four options had positive outcomes, but following a preliminary internal review, Options B and C were rejected due to their higher operating costs and lower revenues. Additional and ongoing testwork suggests that Options B and C should be reconsidered in a future update of the PEA. Detailed estimates were developed for Options A and D and have been included in the PEA.

Options A, and D both involve a similar set of processing steps including crushing, grinding, and cyanidation. Option D includes 2 flotation steps producing a copper concentrate for sale and a pyrite concentrate for on-site cyanide leaching.

A summary of the Project production for options A and D are presented in Table 1-9.

Table 1-9: Production Summary for the Dalradian Resources Tyrone Project Milling Plant

Item	Option A Whole Ore Leach	Option D Copper Flotation and Pyrite Flotation Concentrate Leach
Tonnes Processed Annually, Avg. (t/y)	443,179	443,179
Feed Grades (LOM)		
Au (g/t)	8.06	8.06
Ag (g/t)	3.31	3.31
Cu (%)	0.08	0.08
Saleable Cu Concentrate Production, Avg. (t/y)	-	5,585
Gold Production, LOM Avg. (oz/y)	114,841	102,622
Silver Production, LOM Avg. (oz/y)	47,162	38,681
Key Operating Parameters		
Operating Days Per Year	365	365
Mill Feed Rate (t/d)	1,700	1,700
Chemical and Metallurgical Parameters		
Cu Concentrate Grade (Cu %)	-	8.4

Figure 1-2 and Figure 1-3 show the schematic flowsheets for Options A and D, respectively.

Figure 1-2: Option A Whole Ore Leach

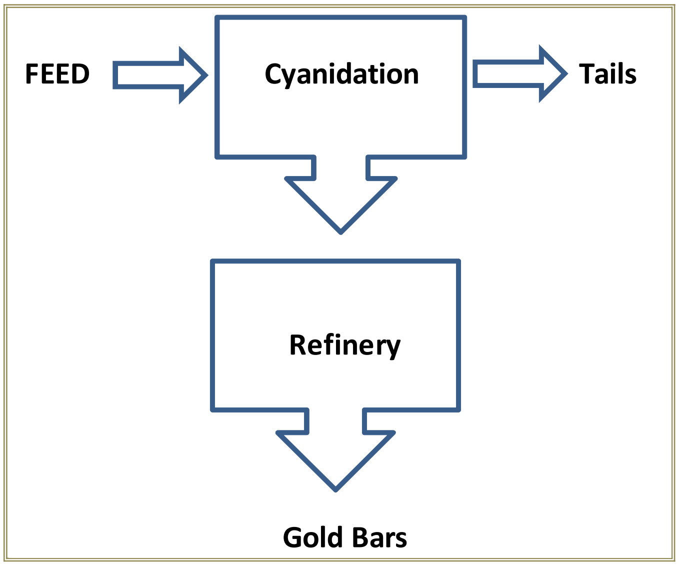
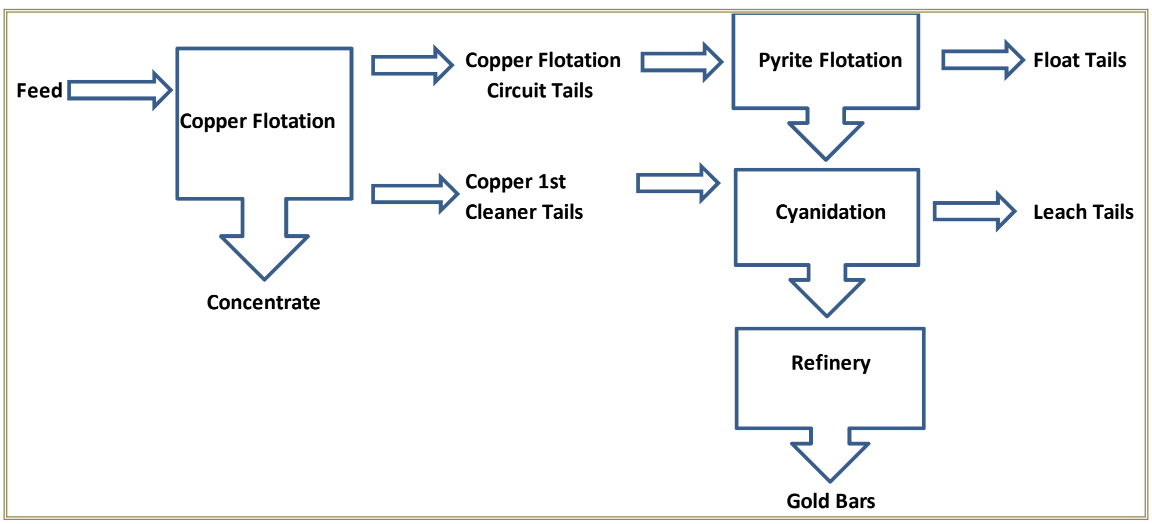


Figure 1-3: Option D Copper Flotation and Pyrite Flotation Concentrate Leach



1.8.3 Process Description (Option A)

The ore is delivered by mine trucks to the plant and fed over a grizzly with 400-mm openings to the run-of-mine (ROM) ore feed hopper. The ore is extracted from the feed hopper by an apron feeder and fed to a jaw crusher. The crushed ore is discharged onto a conveyor that will transport the ore to the plant. The crusher product is 80% passing 150 mm.

The crushed ore reports to the grinding circuit. The primary grinding semi-autogenous grinding (SAG) mill discharge product typically has a size passing of 850 µm.

The secondary grinding ball mill will operate in closed circuit with a cyclone, with the target cyclone overflow 80% size passing 141 µm.

A six CIL circuit is installed based on a retention time of 24 hours and a new carbon concentration of 20 g/L. Slurry pH is adjusted to 10.5 with the addition of lime. The first leach tank is aerated, and lead nitrate is added. The oxidation of sulphides in this step reduces cyanide consumption considerably as well as enhancing leach kinetics. The leach circuit feed is pumped to the train of six CIL tanks in series where it is contacted with activated carbon. Each of the tanks is provided with an in-tank screen to allow the slurry to flow by gravity through the CIL train, while retaining the carbon in the tanks. The carbon is transported counter-current to the slurry flow by means of carbon advance pumps. The loaded carbon from the first CIL tank is collected on a screen and is sent to the carbon stripping circuit. The slurry leaving the last CIL tank is passed over a safety screen to capture any carbon particles before it is sent to the cyanide destruction circuit.

A carbon stripping/carbon reactivation circuit is installed to treat the loaded carbon, to recover gold and silver, and to recycle the carbon to the CIL circuit.

The loaded carbon is acid washed and fed to a Zadra elution circuit to strip the gold from the carbon. The pregnant solution is then sent to two electrowinning cells to recover the gold from solution. The carbon from the stripping circuit is fed to a diesel-fired rotary kiln to reactivate the carbon, which is passed over a sizing screen prior to being recycled to the CIL circuit.

The stainless steel wool cathodes carrying the gold from the electrowinning cells are fed to an induction furnace to produce gold doré bars for sale.

A cyanide destruction circuit is installed to treat the slurry discharge from the CIL circuit.

The CIL discharge slurry is pumped to two agitated tanks in series, and SO_2 and air are added to destroy the cyanide before the slurry is pumped to the tailings thickener. Thickened tailings at 60% solids are pumped to the pastefill plant or tailings disposal area.

1.8.4 Process Description (Option D)

The flowsheet is similar to Option A except that:

- The ball mill ore slurry discharge, at 50% solids, is fed to a conditioning tank, which then overflows to the copper flotation circuit. The copper circuit's first cleaner tailings report to the cyanide leach circuit. Tails from the circuit are pumped to the bulk flotation circuit.
- The concentrate from the rougher cells is fed to the cleaner cells from which the final copper concentrate is thickened to 70% solids in conventional thickener and filtered using pressure filters. This product is then sold to a copper smelter.
- Tails from copper flotation are subjected to a bulk sulphide flotation to recover all remaining sulphides, pyrite, etc., as well as gold and silver bearing minerals. This flotation concentrate is thickened and pumped to a CIL circuit for recovery of gold and silver.

1.8.5 Process Manpower

Option D includes the process operations of Option A, but also includes two additional flotation circuits. Therefore, Option D (76 personnel) will require nine more than Option A (67 personnel) (Table 1-10).

Table 1-10: Summary of Estimated Processing Plant Operations Personnel

Description	Option A Whole Ore Leach	Option D Copper Float and Pyrite Flotation Concentrate Leach
Plant Superintendent	1	1
Senior Metallurgist	1	1
Metallurgist	0	1
Chief Chemist	1	1
Sample Prep. Technician	4	4
Assay Technician	2	2
Plant Shift Supervisor	4	4
Crusher Operator	4	4
Plant Operator	4	4
Control Room Operator	0	4
Operator Helper	4	4
Tailings Operator	2	2
Load out Operator	0	4
Cyanidation Plant Operator	4	4
Day Labour Crew	4	4
Maintenance Superintendent	1	1
Maintenance Supervisor	1	1
Instrument Technician	2	2
Electrician	4	4
Plant Mechanic	4	4
Security Personnel	5	5
Senior Accountant	1	1
Accountants/buyer	1	1
HR and Community Relations	1	1
Environmental Technician	1	1
Supervisor	1	1
Bus Driver	1	1
Secretaries/Administrative Assistant	2	2
Site Surface Maintenance Crew	4	4
Equipment Operator	2	2
Site Administrator	1	1
Total	**67**	**76**

1.9 Infrastructure

Information in this section is taken from the Preliminary Economic Assessment (PEA) published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

1.9.1 Access and Site Roads

The property is readily accessible by a network of existing all-weather paved highways and local roads. Micon considers that ultimately the property should be accessed from the south (off the B46 Gortin-Greencastle road).

Site roads will be built to connect the onsite facilities: gatehouse, administration complex, process plant, mine dry, warehouse, ore and waste storage pads, TSF, etc.

1.9.2 Power Supply

The base case electrical power requirement is 70 kWh/t, which results in a maximum demand of approximately 6 MW for a 1,700 t/d operation.

Power will be supplied by connecting to the national high voltage electricity grid through a 33 kV overhead transmission line that will be built from the 110/33 kV substation located at Strabane, for approximately 27 km to the site. A substation will be built on site, which will transform the electrical power from 33 kV to 11 kV for distribution around the site.

Emergency power will be provide by a 1.12 MW, 400 V standby diesel generator located on site.

1.9.3 Water Supply

Micon has assumed that supply of the initial and make-up water for the process plant will be through the construction of a rainwater catchment system. Rainfall in the Project area is in the order of 1,300 mm/y to 1,400 mm/y, and Micon anticipates the utilization of the proposed TSF to capture and store rainwater.

Nevertheless, further studies to assess potential water sources should be carried out during further development of the Project.

1.9.4 Tailings Storage Facility

In 2011, Golder Associates UK (Golder) was retained by Dalradian Resources to carry out a Tailings Management Facility Site Selection Study aimed at identifying potential sites to store approximately 2.92 Mt of tailings. Golder, in its December 2011 report, identified seven potential sites and recommended three of them for further investigation. Golder also recommended the use of conventional tailings disposal technology and to carry out the construction work in two separate phases.

1.10 Environmental, Permitting, and Social

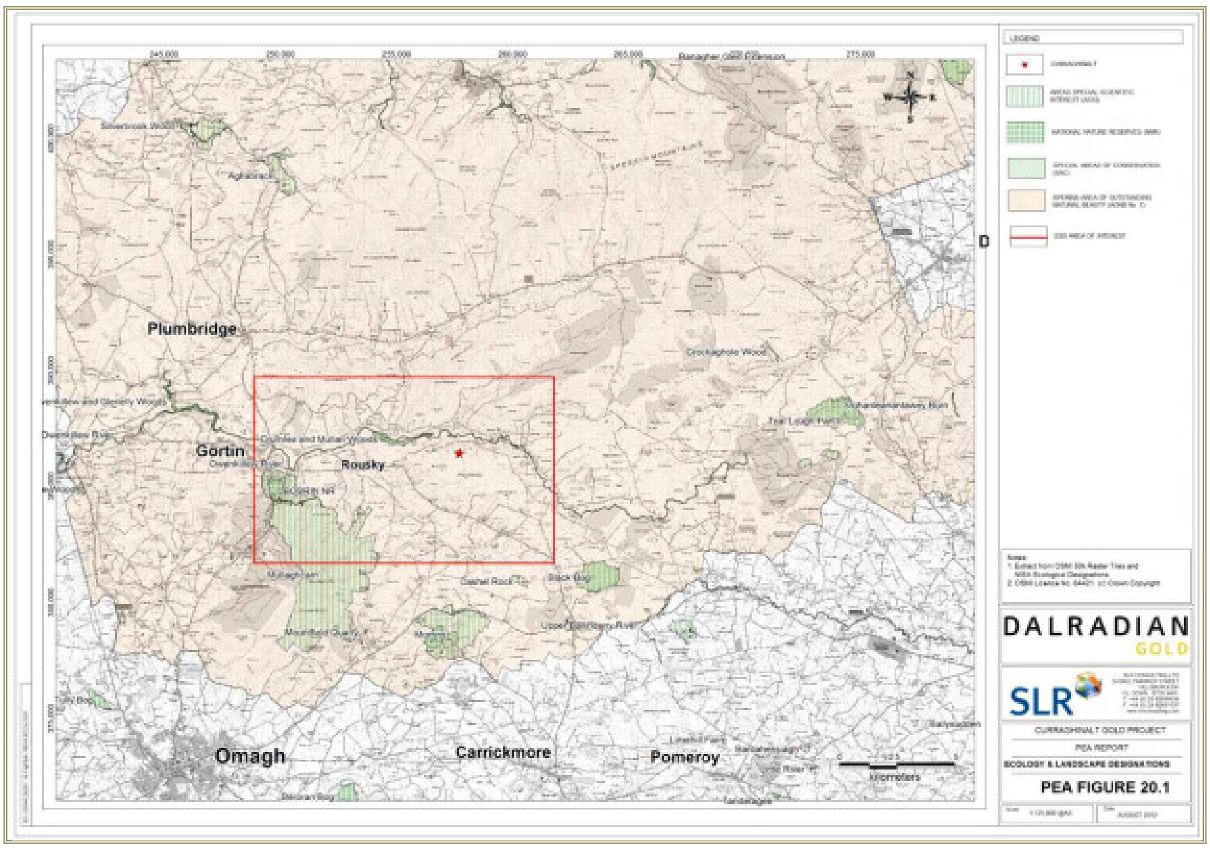
1.10.1 Studies and Issues

TMAC has not prepared a liability assessment of the property and cannot provide a legal opinion on the status of permits. The Sperrin Mountains are designated an Area of Outstanding Natural Beauty. There are also a number of protected and special interest areas around the Project (Figure 1-4).

The nearest Areas of Special Scientific Interest (ASSIs) to the Project are Owenkillew River, Mullaghcarn / Mountfield Quarry, Murrins, Cashel Rock, Boorin Wood, and Black Bog. The nearest Special Areas of Conservation (SACs) include Drumlea and Mullan Woods, Owenkillew River, and Black Bog.

Within the Owenkillew River SAC/ASSI are four Annex II listed species: freshwater pearl mussel (*Margaritifera margaritifera*), river otter (*Lutra lutra*), brook lamprey (*Lampetra planeri*) and Atlantic salmon (*Salmo salar*).

Figure 1-4: Ecological and Landscape Designations



Source: SLR, August 2012

1.10.2 Waste and Water Management

SLR and SRK undertook geochemical investigations to determine the potential for acid generation and metals leaching in the proposed operation. A total of seventy-two samples plus four quality control duplicates were collected in 2011 and 2013to represent rock types likely to be encountered during the mining operation. The testwork undertaken included acid base accounting (ABA), net acid generation and acid rain leach procedure (ARLP) tests aimed to determine whether and which of the mined materials would be acid generating and/or metal leaching.

Based on the ABA laboratory results, it was concluded that 12 samples were considered to be acid generating, 57 were classified as non-acid generating (NAG) and the results of 7 samples were considered to be inconclusive. Potentially acid generating (PAG) waste rock will be backfilled underground and ultimately flooded after closure, which will minimize future oxidation and metal release. The remaining waste rock will be stored in surface dumps, contoured, and eventually capped and revegetated in character with the surrounding landscape.

Metals leaching tests showed cadmium and cobalt were leached in concentrations below the limits of detection, and copper, lead and nickel were leached in concentrations compliant with International Finance Corporation (IFC) and EU guidelines. Zinc concentrations for ore sample (SRK 3253) exceeded by 0.001 mg/L the most stringent guideline in Directive 2000/60/EC.

Sulphate was leached only from six Ssp and seven Sps samples, whereas arsenic was released by all samples and lithology groups. One sample exceeded IFC guidelines for arsenic concentrations, and 27 samples exceeded the most stringent guideline in Directive 2000/60/EC.

Further testing of waste rock and tailings conducted during subsequent stages will provide additional information to assist with segregation and management of the various rock types during operations.

An estimated 40% of tailings will be backfilled underground. The remaining tailings will be disposed of at the surface in a conventional tailings impoundment, which will be lined with synthetic or impermeable material, or both. It is recommended that the Project consider paste or dry-stack tailings to minimize the footprint and increase the available options for siting the tailings and progressive reclamation to minimize visual impacts.

Mine water will be collected in underground sumps and pumped to surface for use as makeup water in the process plant. This water may need to be treated prior to use in the process, depending on the concentrations of suspended sediments and blasting residues.

The process plant is designed to include an INCO SO_2/air cyanide destruction circuit to meet the limit of 10 ppm weak acid dissociable (WAD) cyanide at the point of discharge to the tailings pond, as required under Directive 2006/21/EC, Article 13(6). Slurry tailings will be pumped to the tailings impoundment and pond water will be recycled back to the process plant via a reclaim barge and pipeline. The impoundment will need to be designed to minimize seepages. Monitoring wells will then be required at the tailings impoundment to monitor seepages and could be used for seepage collection and pump back to the pond if necessary.

Post-closure, the adit will be plugged, and underground workings allowed to flood. For the tailings impoundment at closure, it is assumed tailings pond water will be recycled back through the plant, and treated as necessary until the pond water-quality is sufficient for direct release of any annual surplus water. Groundwater and surface waters will need to be monitored post-closure.

1.10.3 Social and Environmental Management

SLR identified social and environmental design criteria should to be implemented to meet the conservation and visual landscapes objectives of the surrounding area. These include:

- tree and shrub plantings for screening and reclamation should use native species and conform to the surrounding landscape
- plant vegetation for screening before construction to allow vegetation to establish and minimize visual impacts during development
- minimize visual impacts from the tailings pond and stockpiles by progressive reclamation and use vegetation for screening around the perimeters
- design buildings in colours, styles, and size similar to those in the area, and use earth-sheltered buildings if the building will exceed the size of a typical farm shed
- use earth berms and stone walls to screen development and maintain the character of the surrounding landscape
- haul roads should be designed to follow existing field boundaries and landscape contours
- retain existing vegetation in accordance with recommendation in BS5837:2005, *Trees in Relation to Construction*.

Following good industry practice, it is assumed a social, environmental, health and safety management system will be implemented to meet the Company's commitments to protect the environment and the health and safety of the workers and surrounding communities. The management system and plans should be designed to monitor and maintain permit compliance and a social licence to operate.

1.10.4 Permitting Requirements

For exploration work, formal notice of intention to enter land to carry out work must be given, and the agreement of landowners sought, before entering any property. Compensation is generally payable to the landowner for any damage caused during exploration.

Project development is subject to legislative requirements from the European Union, England (UK), and Northern Ireland. Applicable environmental legislation is listed in Table 1-11. In addition, a number of international conventions will apply to the Project and need to be considered in further planning.

Table 1-11: Key Applicable Environmental Legislation

Name of Legislation	Jurisdiction	Date Adopted
Assessment of Effects of Certain Public and Private Projects on the Environment (EIA Directive) (Directive 97/11/EC of 3 March 1997 amending Directive 85/337/EEC)	European Union	1997
The Planning (Environmental Impact Assessment) Regulations 2012 SR 2012 No 59	Northern Ireland	2012
Environmental Protection Act	United Kingdom	1990
Climate Change Act	United Kingdom	2008
Directive 2006/21/EC Management of Waste from Extractive Industries and Amending Directive 2004/35/EC	European Union	2006
Waste Directive (Directive 2008/98/EC)	European Union	2008
Ambient Air Quality and Cleaner Air for Europe (Directive 2008/50/EC)	European Union	2008
Industrial Emissions (Integrated Pollution Prevention and Control) (Directive 2010/75/EU)	European Union	2010
Assessment and Management of Environmental Noise (Directive 2002/49/EC)	European Union	2002
Habitats Directive	European Union	1992
Birds Directive (Directive 2009/147/EC)	European Union	2009

At this time, it is uncertain how long it will take to permit the Project. Once decisions are made on the final location of infrastructure and facilities, land acquisition will also need to be negotiated.

1.10.5 Closure

The objective of the mine closure plan is to remove and close down activities in a manner that ensures public safety and to reclaim the land to a usable state consistent with the surrounding land-use objectives. In the case of the Curraghinalt project, the land will be restored to productive use for farming and/or heathlands. The visual landscape objective is to minimize disruption of the outstanding natural beauty of the area during operations and restore this value at closure.

Closure will consist of plugging and securing underground openings, removing from the site to licensed facilities any hazardous and contaminated materials, decommissioning and demolition of facilities and buildings, and reclaiming waste rock and tailings storage facilities. It has been assumed that the electrical substation is an infrastructural asset that will be left in place post-closure, owned by the utility.

When possible, equipment and machinery will be sold or recycled. Buildings will be demolished and reclaimed, recycled, or disposed. Concrete foundations will be broken up and removed from site, reclaimed if possible, or buried at a certified waste disposal facility.

Tailings facility seepage and surface waters will be monitored. Seepages will be collected and pumped back to the pond if the quality is not acceptable for release. Tailings pond water will be monitored and pumped back to the plant for treatment if necessary until the pond water is acceptable for discharge to the environment. A spillway will then be constructed in the dam and the dams and beach areas capped with overburden and revegetated. Similarly, any NAG waste rock stored on the surface will be capped with overburden and revegetated. It is assumed that to minimize oxidation any PAG waste rock will be used in backfill and flooded after mine closure

Reclamation costs are estimated at $7.5 million in Year 16. Using a real discount rate of 2% it is assumed for the PEA that the present value (approximately 71%) of this amount will be required to be posted as a financial guarantee at the start of the Project. A financial guarantee is required to meet requirements of Directive 2006/21/EC, Management of Waste from Extractive Industries.

1.11 Capital and Operating Cost Estimates

Information in this section is taken from the Preliminary Economic Assessment (PEA) published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

1.11.1 Capital Costs

Capital expenditures and capitalized development costs for the base case are summarized as initial and sustaining costs in Table 1-12. The estimates are expressed in second quarter 2012 Canadian dollars, without escalation unless otherwise noted. The expected accuracy of the estimates is ± 30%. Expressed as US dollars, initial and sustaining capital amounts to US$192.1 million and US$109.9 million, respectively.

Table 1-12: Capital Cost Summary

Area	Initial Capital Cost ($ '000s)	Sustaining Capital Cost ($ '000s)
Capitalized Development	15,745	64,925
Mining Equipment	14,770	33,058
Processing	50,743	-
Infrastructure	49,833	16,301
Indirect Costs	12,927	-
Owner's Costs	17,386	-
Contingency	38,341	-
Total	**199,745**	**114,284**

1.11.2 Operating Costs

Estimated cash operating costs over the life of the Project are summarized in Table 1-13.

Table 1-13: Summary of LOM Operating Costs

Area	LOM Cost ($ '000s)	Unit Cost ($/t ore treated)
Production Drilling and Blasting	44,600	4.73
Sill	399,183	42.36
Sill (through waste)	13,764	1.46
Diesel Fuel	22,869	2.43
Backfill	29,280	3.11
Manpower	99,948	10.61
Ventilation and Dewatering	2,279	0.24
Exploration	911	0.10
Major Equipment Maintenance	85,618	9.08
Miscellaneous and Sundries	7,395	0.78
Mining G&A	43,653	4.63
Stockpile Rehandle	310	0.03
Subtotal Mining	**749,812**	**79.56**
Labour – Metallurgy	5,180	0.53
Laboratory	6,963	0.71
Production	27,417	2.81
Maintenance	14,037	1.44
Power	35,561	3.65
Maintenance	18,346	1.88
Crushing	759	0.08
Grinding	33,760	3.46
Reagents	34,835	3.57
Miscellaneous	13,376	1.37
Subtotal Processing	190,235	20.19
Labour	21,903	2.25
Equipment Maintenance	6,646	0.68
Mobile Equipment Operation	1,472	0.15
Environmental and Social	7,594	0.78
G&A (other)	43,770	4.49
Subtotal G&A	81,385	**8.64**
Total Operating Costs	**1,021,431**	**108.38**

1.12 Economic Analysis and Sensitivity Studies

Information in this section is taken from the Preliminary Economic Assessment (PEA) published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

1.12.1 Basis of Evaluation

Micon has prepared its assessment of the Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback, and other measures of Project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to determine the viability of the proposed mine and process plant to exploit the Curraghinalt deposit. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions was then examined.

1.12.2 Macroeconomic Assumptions

Unless otherwise stated, all results are expressed in Canadian dollars. Cost estimates and other inputs to the cash flow model for the Project have been prepared using constant, second quarter 2012 money terms, without provision for escalation or inflation. Trailing 36-month average exchange rates of C$1.04/US$1 and C$1.62/GBP1 are applied in the base case.

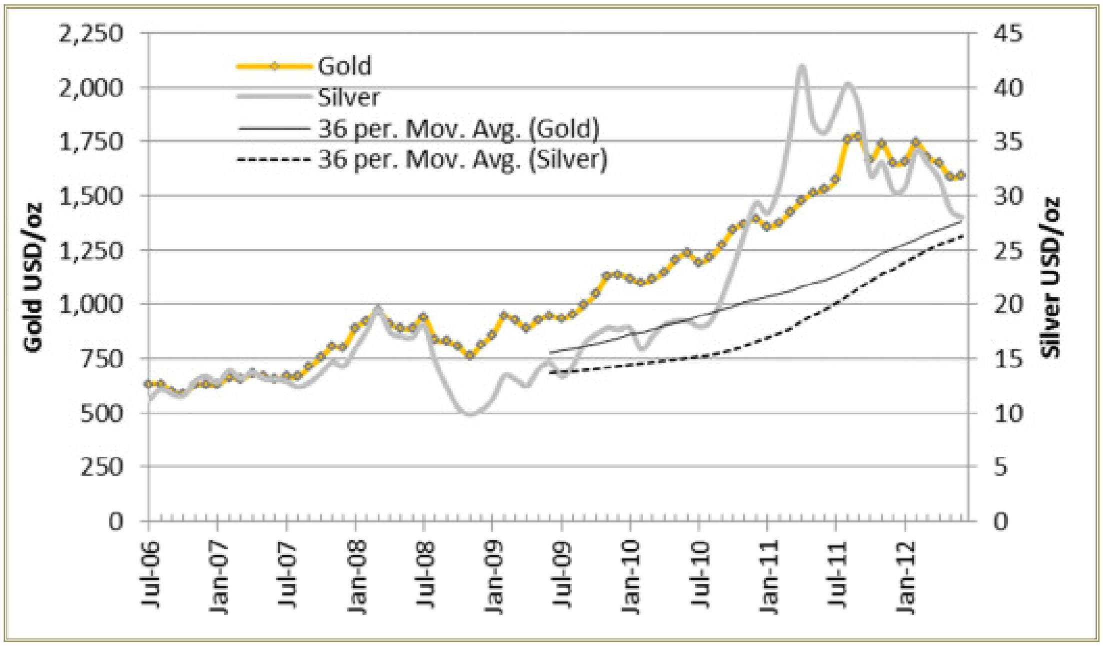
Applying the Capital Asset Pricing Model (CAPM) gives an estimated cost of equity for the Curraghinalt Project of between 5% and 11%, as shown in Table 1-14. Micon has taken a figure of 8% (i.e., in the middle of this range) as its base case, and provides the results at alternative rates of discount for comparative purposes.

Table 1-14: Estimated Cost of Equity

Range	Lower	Middle	Upper
Risk-Free Rate (%)	1.5	2.0	2.5
Market Premium for equity (%)	5.0	5.0	5.0
Beta	0.7	1.2	1.7
Cost of equity (%)	5.0	8.0	11.0

Figure 1-5 shows the monthly average gold and silver prices over the past six years, together with the three-year trailing averages. At the end of June 2012, the three-year trailing averages for each metal were $1,378/oz. Au and $26.28/oz. Ag, and these metal prices were selected for the base case. These prices were applied consistently throughout the operating period.

Figure 1-5: Monthly Average Gold and Silver Prices since July 2006



Source: Kitco.com

Silver contributes approximately 0.6% of the projected total revenue for the base case, so the impact of changing the silver price forecast is minimal.

For comparison, Micon also evaluated the sensitivity of the Project to using recent (one month), and 1-, 2-, 3-, 5-, and 10-year price averages. The prices used in each of these cases are shown in Table 1-15. As part of its sensitivity analysis, Micon also tested a range of prices 30% above and below base case values.

Table 1-15: Metal Price Averages

Item	Units	1 Month (Jun. 2012)	1-Year Average	2-Year Average	3-Year Avg. (Base Case)	5-Year Average	10-Year Average
Gold	US$/oz.	1,597	1,672	1,521	1,378	1,166	814
Silver	US$/oz.	28.05	33.16	30.98	26.28	21.44	14.66

United Kingdom corporation tax payable on the Project has been forecast using current rates, being 20% on the first GBP 300,000 per year, 22% on the increment up to GBP 1.50 million, and 24% on the balance. Depreciation allowances of 18% are assumed to be taken annually on all project capital, on a declining balance basis.

A royalty of 6% of NSR value has been provided for in the cash flow model.

Refining charges are estimated at US$6/oz. Au, and US$0.50/oz. Ag, based on similar projects. Doré transport charges of US$5,000 per shipment, and cash-in-transit insurance, calculated as a percentage of shipment value, are also provided.

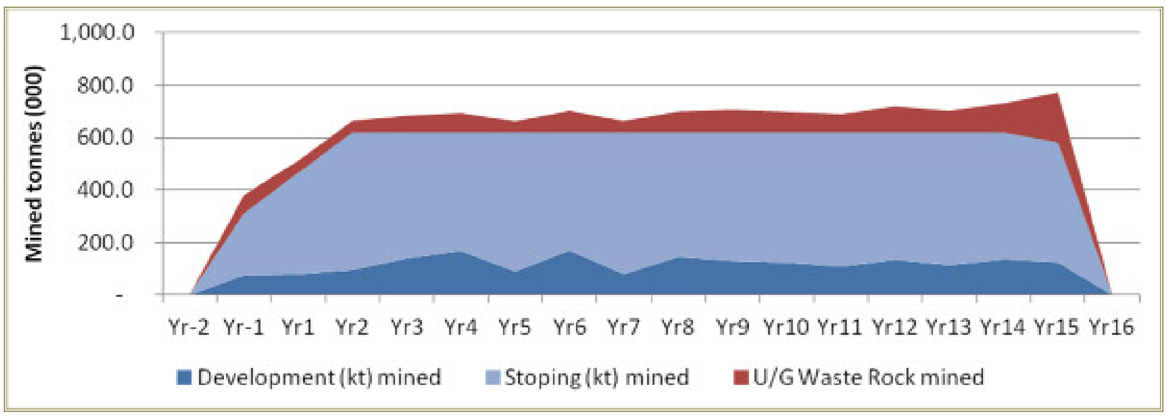
1.12.3 Technical Assumptions

The technical parameters, production forecasts, and estimates described elsewhere in this Report are reflected in the base case cash flow model. These inputs to the model are summarized below. The units of measure used in the study are metric, except where, by convention, gold and silver content, production, and sales are stated in troy ounces.

Mine Production Schedule

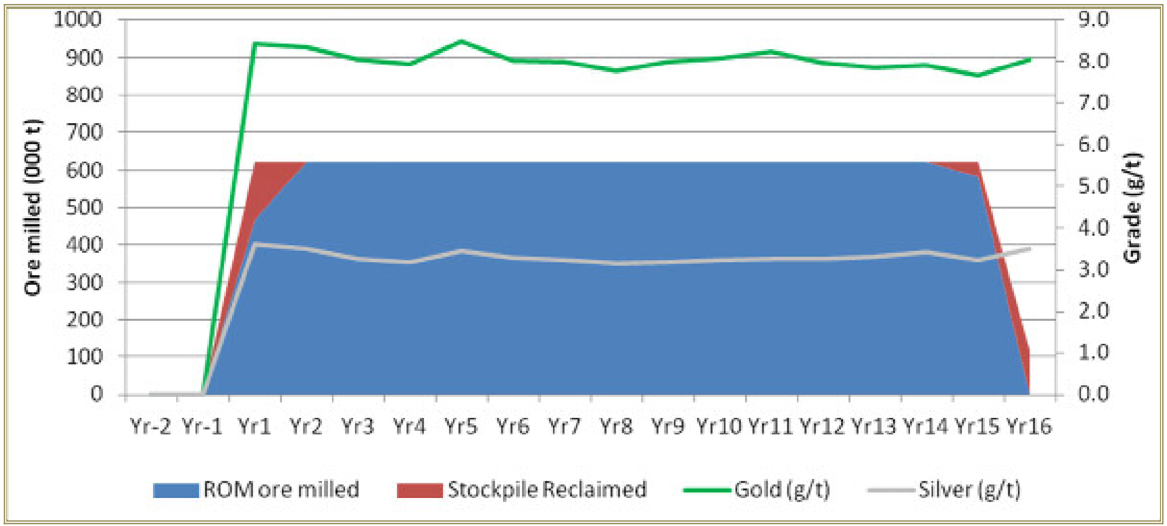
Figure 1-6 shows the annual tonnage of development and stoping material mined, as well as the waste rock tonnage, all of which are held reasonably constant over the LOM.

Figure 1-6 Annual Mining Schedule



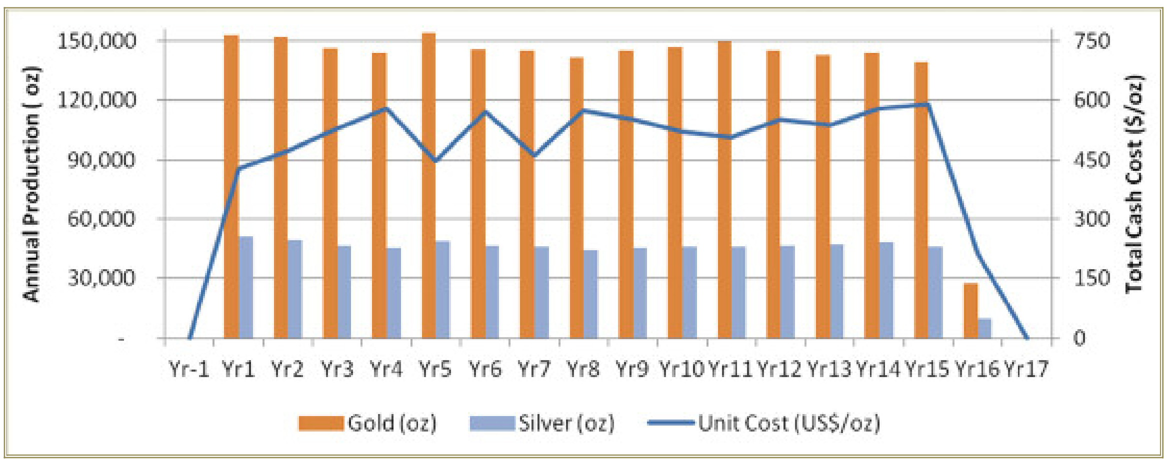
In Figure 1-7, the head grades for gold and silver in the mill feed are shown to remain within very narrow ranges over the LOM. Tonnages milled reflect the drawdown of stockpiled material during the mill start-up, and again at the end of the LOM.

Figure 1-7: Annual Processing Schedule



As a consequence of steady tonnage, grade, and recovery from mill feed, annual production of gold and silver remain steady over the LOM (Figure 1-8). This chart also shows the annual total cash cost per ounce of gold sales. Total cash costs, including refining charges and royalties, and net of silver credits, average $532/oz Au over the LOM. In Years 5 and 7, costs fall below this level, owing to a reduction in development costs in those periods.

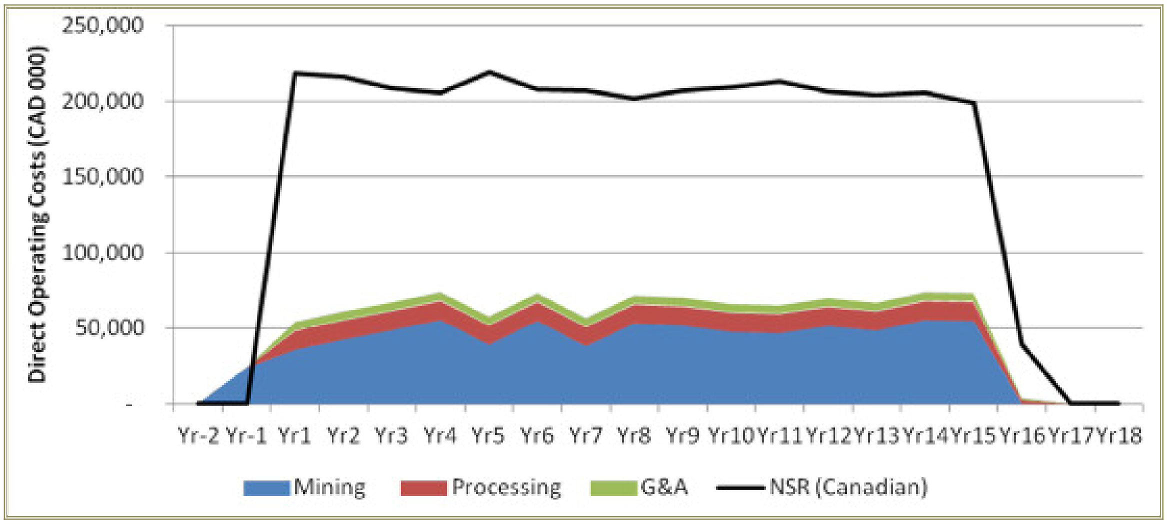
Figure 1-8: Annual Production Schedule



Operating Costs

Direct operating costs average $108.38/t milled over the LOM comprised $79.56/t mining, $20.19/t processing, and $8.64/t general and administrative costs. Figure 1-9 shows these expenditures over the LOM, compared to the net sales revenue, showing the strong margin maintained over the LOM.

Figure 1-9: Direct Operating Costs



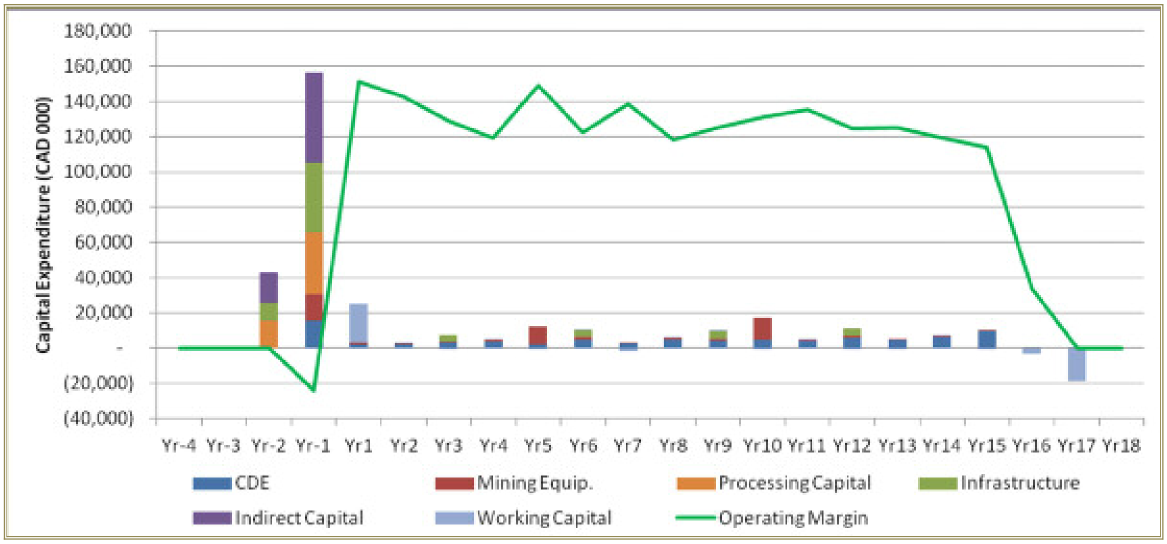
Capital Costs

Preproduction capital expenditures are estimated to total $199.75 million, including $30.5 million for mining equipment and pre-production development, $50.7 million processing, $49.8 million infrastructure, $12.9 million indirect costs, $12.0 million in owner's costs, a $5.4 million provision for closure and rehabilitation, and contingencies totalling $38.3 million.

Working capital has been estimated to include 15 days product inventory in the milling/leach circuit, and 15 days receivables from despatch of doré. Stores provision is for 60 days of consumables and spares inventory, less 30 days accounts payable. An average of $22.3 million of working capital is required over the LOM.

Figure 1-10 compares annual capital expenditures over the preproduction and LOM with the project's cash operating margin.

Figure 1-10: Capital Expenditures



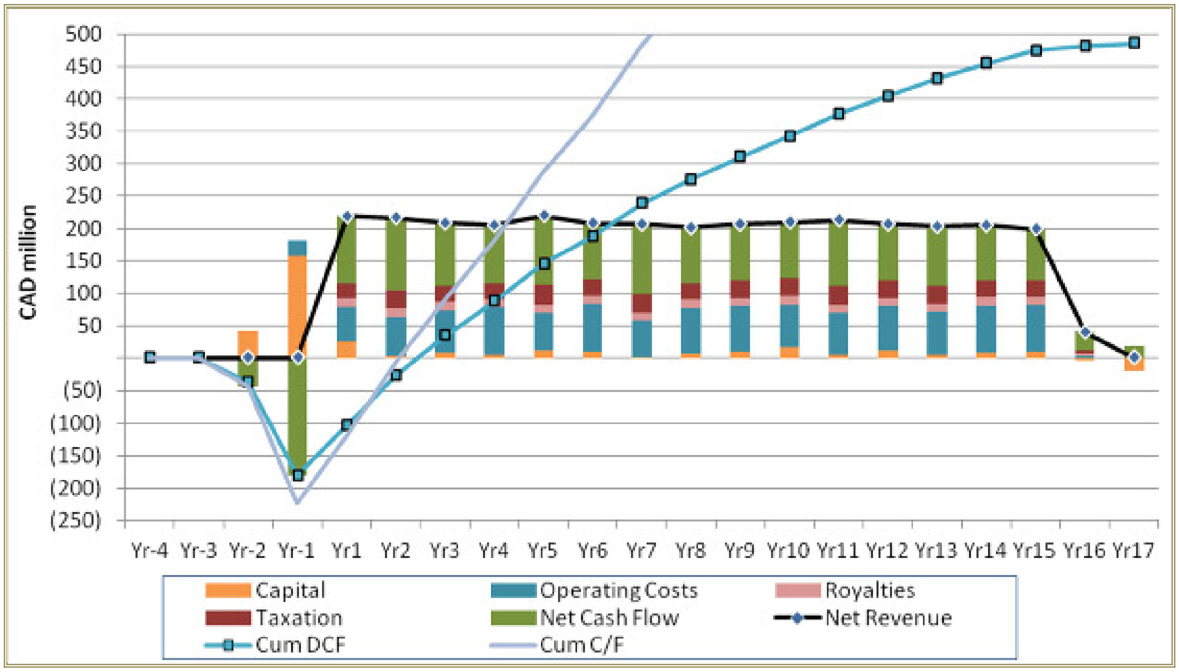
Base Case Cash Flow

The LOM base case project cash flow is presented in Table 1-16 and Figure 1-11.

Table 1-16: LOM Cash Flow Summary

	LOM Total ($000)	$/t Milled	US$/oz
Gold Revenue	3,186,233	338.08	1,378.03
Mining Costs	749,812	79.56	324.29
Processing Costs	190,235	20.19	82.28
G&A Costs	81,385	8.64	35.20
S/T Direct Site Operating Costs	1,021,431	108.38	441.76
Silver Credit	(19,044)	(2.02)	(8.24)
Refining and Transport Charges	37,575	3.99	16.25
S/T Cash Operating Costs	1,039,962	110.35	449.77
Royalty	190,062	20.17	82.20
Total Cash Costs	**1,230,024**	**130.51**	**531.98**
EBITDA	1,956,209	207.57	846.05
Capital Expenditure	314,029	33.32	135.82
Net Cash Flow (before tax)	1,642,180	174.25	710.23
Taxation	402,895	42.75	174.25
Net Cash Flow (after tax)	1,239,286	131.50	535.98

Figure 1-11: LOM Cash Flow



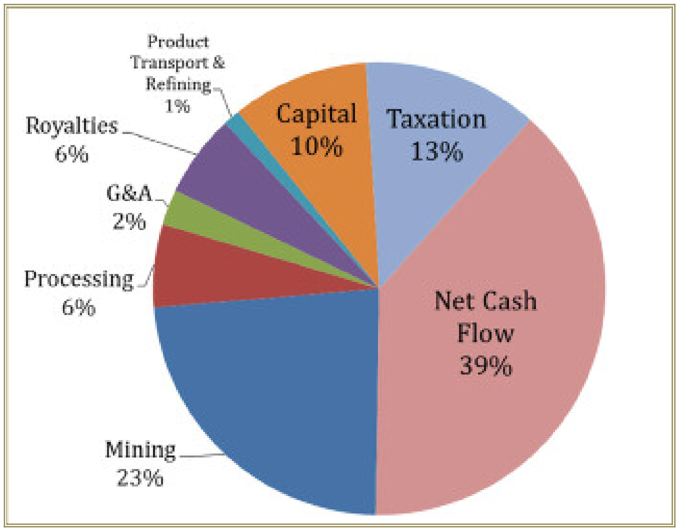
The Project demonstrates an undiscounted payback of 2.1 years, or approximately 2.4 years when discounted at 8.0%, leaving a production tail of just over 12 years. Annual cash flows are presented in Table 1-17.

Table 1-17: Base Case LOM Annual Cash Flow

Production Schedule			LOM	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17
Underground Mine Production			TOTAL																			
Development		000 t	1,890	-	74	79	95	138	166	89	167	80	143	128	122	107	133	112	134	123	-	-
Stoping		000 t	7,534	-	236	387	526	483	455	531	453	541	478	492	498	514	488	508	487	459	-	-
Resources mined		000 t	9,425	-	310	465	621	621	621	621	621	621	621	621	621	621	621	621	621	582	-	-
U/G Waste Rock mined		000 t	1,260	-	70	47	43	64	74	42	83	43	80	86	79	70	98	83	110	189	-	-
Processing Plant Throughput		000 t	9,425	-	-	621	621	621	621	621	621	621	621	621	621	621	621	621	621	621	117	-
Gold grade		g/t	8.06	-	-	8.43	8.36	8.04	7.93	8.47	8.03	8.00	7.79	8.01	8.09	8.23	7.98	7.87	7.92	7.68	8.06	-
Silver grade		g/t	3.31	-	-	3.62	3.50	3.28	3.20	3.45	3.29	3.24	3.15	3.19	3.24	3.26	3.28	3.33	3.42	3.24	3.52	-
Payable Metal (imperial)																						
Gold		oz	2,223,236	-	-	153,180	151,849	146,124	144,149	153,943	145,927	145,353	141,575	145,451	146,980	149,522	145,007	142,965	143,993	139,603	27,613	-
Silver		oz	715,655	-	-	51,406	49,788	46,556	45,509	49,011	46,767	46,084	44,811	45,403	46,009	46,297	46,576	47,305	48,662	46,041	9,429	-
Exchange Rate		CAD/USD	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04
Gold price		US$/oz	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03
Silver price		US$/oz	26.280	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28
Cash Flow Forecast			LOM TOTAL	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17
	CAD/t ore	US$/oz	CAD 000																			
Gross Revenue (Gold)	338.08	1,378.03	3,186,233	-	-	219,530	217,622	209,418	206,588	220,624	209,135	208,313	202,899	208,454	210,645	214,288	207,817	204,890	206,364	200,073	39,574	-
Total Cash Costs	130.51	531.98	1,230,024	-	23,919	67,967	74,818	80,450	87,040	71,608	86,521	69,773	84,514	83,429	79,431	78,828	83,055	79,788	86,873	85,946	6,066	-
Mining	79.56	324.29	749,812	-	23,919	35,762	42,708	48,847	55,612	39,267	54,943	38,235	53,329	51,864	47,725	46,872	51,564	48,522	55,534	54,992	117	-
Processing	20.19	82.28	190,235	-	-	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	2,362	-
G&A	8.64	35.20	81,385	-	-	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	1,010	-
Sub-total Direct Costs	*108.38*	*441.76*	*1,021,431*	-	*23,919*	*53,645*	*60,592*	*66,731*	*73,495*	*57,150*	*72,826*	*56,118*	*71,212*	*69,747*	*65,608*	*64,755*	*69,447*	*66,406*	*73,418*	*72,875*	*3,489*	-
By product credit (Silver, net)	(2.02)	(8.24)	(19,044)	-	-	(1,368)	(1,325)	(1,239)	(1,211)	(1,304)	(1,245)	(1,226)	(1,192)	(1,208)	(1,224)	(1,232)	(1,239)	(1,259)	(1,295)	(1,225)	(251)	-
Transport & Refining charges (Gold)	3.99	16.25	37,575	-	-	2,592	2,568	2,467	2,434	2,603	2,464	2,456	2,393	2,458	2,484	2,525	2,451	2,417	2,437	2,360	467	-
Royalties	20.17	82.20	190,062	-	-	13,098	12,983	12,491	12,322	13,160	12,475	12,425	12,102	12,432	12,563	12,780	12,396	12,224	12,313	11,936	2,361	-
Operating Margin	207.57	846.05	**1,956,209**	-	(23,919)	151,563	142,805	128,968	119,548	149,016	122,615	138,540	118,385	125,025	131,214	135,460	124,762	125,102	119,491	114,127	33,508	-
Capital Costs	33.32	135.82	314,029	42,831	156,914	2,971	2,601	7,689	4,814	12,328	10,229	3,051	5,615	9,645	17,203	4,665	11,201	5,286	6,719	10,267	-	-
Mine Development	8.56	34.89	80,670	-	15,745	1,896	1,877	3,154	3,940	2,301	4,549	2,406	4,656	4,490	4,657	4,206	5,902	4,826	6,259	9,807	-	-
Mining Equip.	5.07	20.69	47,827	-	14,770	1,075	725	460	874	10,027	1,605	645	959	1,080	12,546	460	1,224	460	460	460	-	-
Processing Capital	5.38	21.95	50,743	15,223	35,520	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure	7.02	28.60	66,134	10,430	39,403	-	-	4,075	-	-	4,075	-	-	4,075	-	-	4,075	-	-	-	-	-
Indirect Capital	7.28	29.69	68,654	17,178	51,476	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Change in Working Cap	-	-	-	-	-	22,153	323	(159)	330	(208)	367	(1,457)	815	330	(158)	224	(126)	(494)	709	(513)	(3,400)	(18,735)
Pre-tax c/flow	**174.25**	**710.23**	**1,642,181**	**(42,831)**	**(180,833)**	**126,439**	**139,880**	**121,438**	**114,403**	**136,896**	**112,018**	**136,946**	**111,955**	**115,050**	**114,170**	**130,570**	**113,688**	**120,311**	**112,063**	**104,374**	**36,907**	**18,735**
Tax payable	42.75	174.25	402,895	-	-	21,865	26,955	24,695	23,267	30,975	24,958	29,338	24,962	26,844	28,309	29,432	27,033	27,299	26,146	24,864	5,954	-
C/flow after tax	**131.50**	**535.98**	**1,239,286**	**(42,831)**	**(180,833)**	**104,574**	**112,925**	**96,742**	**91,136**	**105,921**	**87,060**	**107,609**	**86,993**	**88,206**	**85,861**	**101,139**	**86,655**	**93,013**	**85,917**	**79,509**	**30,953**	**18,735**
Cumulative C/Flow				(42,831)	(223,663)	(119,089)	(6,164)	90,579	181,715	287,636	374,697	482,305	569,298	657,504	743,365	844,504	931,159	1,024,171	1,110,088	1,189,598	1,220,551	1,239,286
Discounted C/Flow (8%)			485,330	(36,721)	(143,551)	76,865	76,855	60,964	53,177	57,226	43,552	49,844	37,310	35,028	31,571	34,434	27,317	27,149	23,221	19,897	7,172	4,020
Cumulative DCF				(36,721)	(180,271)	(103,406)	(26,551)	34,413	87,590	144,816	188,368	238,212	275,521	310,549	342,120	376,554	403,871	431,021	454,241	474,139	481,311	485,330
Max funding reqmt to positive cashflow			(270,652)	(42,831)	(223,663)	**(270,652)**	(148,969)	(38,389)	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Over the LOM, gross revenues for gold and silver totalling $3,205.8 million are distributed as shown in Figure 1-12. This diagram illustrates the robust nature of the net cash flows forecast to be generated by the project, both on a before- and after-tax basis.

Figure 1-12: Distribution of Gold and Silver Revenues



Base Case Evaluation

The base case evaluates to an IRR of 51.7% before taxes and 41.9% after tax.

At a discount rate of 8.0%, the net present value (NPV_8) of the cash flow is $664.6 million (US$639 million) before tax and $485.3 million (US$ 467 million) after tax. Table 1-18 presents the results in Canadian dollars at comparative annual discount rates of 5%, 8% and 11%. Stated in US dollars, at annual discount rates of 5% and 11%, the after-tax NPV of the Project is US$ 655 million and US$336 million, respectively.

Table 1-18: Base Case Cash Flow Evaluation ($ '000s)

	LOM Total	Discounted at 5%/y	Base Case Discounted at 8%/y	Discounted at 11%/y	IRR (%)
Gross Revenue (Au)	3,186,233	1,903,535	1,442,827	1,117,046	-
Mining Costs	749,812	448,187	340,171	263,940	-
Processing Costs	190,235	113,237	85,651	66,180	-
G&A Costs	81,385	48,444	36,642	28,312	-
Transport & Refining, Net of Ag Credit	18,531	11,074	8,392	6,494	-
Royalty	190,062	113,548	86,066	66,633	-
Total Cash Costs	**1,230,024**	**734,490**	**556,923**	**431,559**	**-**
EBITDA	1,956,209	1,169,046	885,904	685,487	-
Capital Expenditure	314,029	239,989	209,694	185,946	-
Working Capital	-	9,887	11,562	11,893	-

	LOM Total	Discounted at 5%/y	Base Case Discounted at 8%/y	Discounted at 11%/y	IRR (%)
Net Cash Flow (before tax)	1,642,181	919,169	664,648	487,648	51.7
Taxation	402,895	238,160	179,317	137,895	-
Net Cash Flow (after tax)	1,239,286	681,009	485,330	349,753	41.9

This preliminary economic assessment is preliminary in nature; it 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 the preliminary economic assessment will be realized.

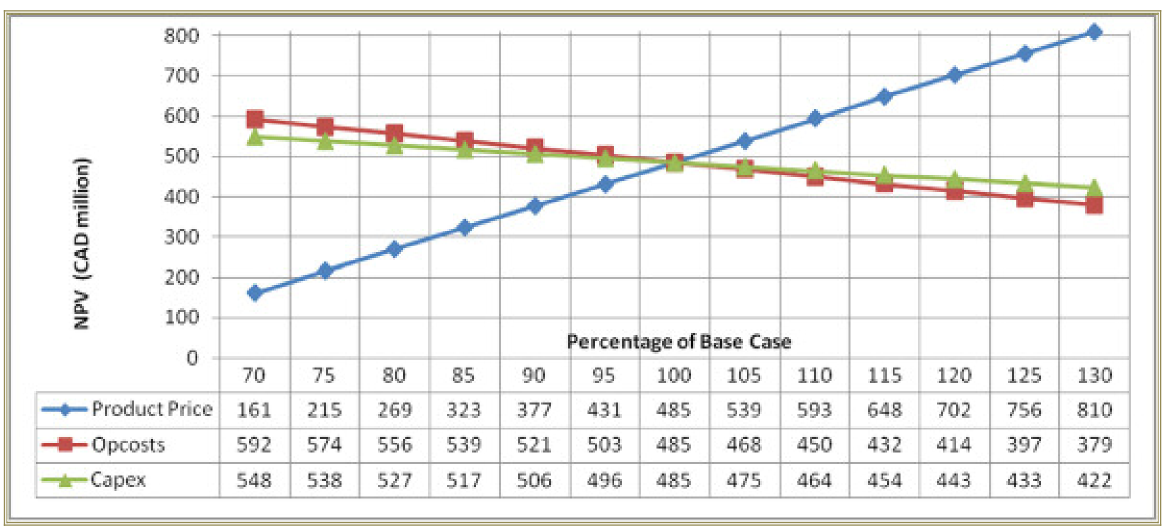
1.12.4 Sensitivity Study

Capital, Operating Costs, and Revenue Sensitivity

The sensitivity of project returns to changes in all revenue factors (including grades, recoveries, prices and exchange rate assumptions) together with capital and operating costs was tested over a range of 30% above and below base case values. The results show that the Project is most sensitive to revenue factors, with an adverse change of 30% reducing after-tax NPV_8 from $485 million to $161 million. The impact of changing operating costs is lower, with a 30% adverse change reducing NPV_8 to $379 million. The project is least sensitive to capital costs, with a 30% increase in capital reducing NPV_8 to $422 million. Thus, NPV_8 remains positive within the expected range of accuracy of the estimates.

In Micon's analysis, applying an increase of more than 85% in both capital and operating costs simultaneously would be required to reduce NPV_8 to near zero, further demonstrating the robust nature of the Project economics. Figure 1-13 shows the results of changes in each factor separately.

Figure 1-13: Sensitivity Diagram



Metal Price Sensitivity

The sensitivity of the Project to variation in gold price was tested using 1 month, and, 1, 2, 3, 5 and 10-year trailing averages applied over the LOM, as shown in Figure 1-14 and in Table 1-19. At US$1,200/oz. Au, the Project has an IRR of 39.1% pre-tax and 31.8% after tax.

Figure 1-14: Sensitivity to Metal Prices

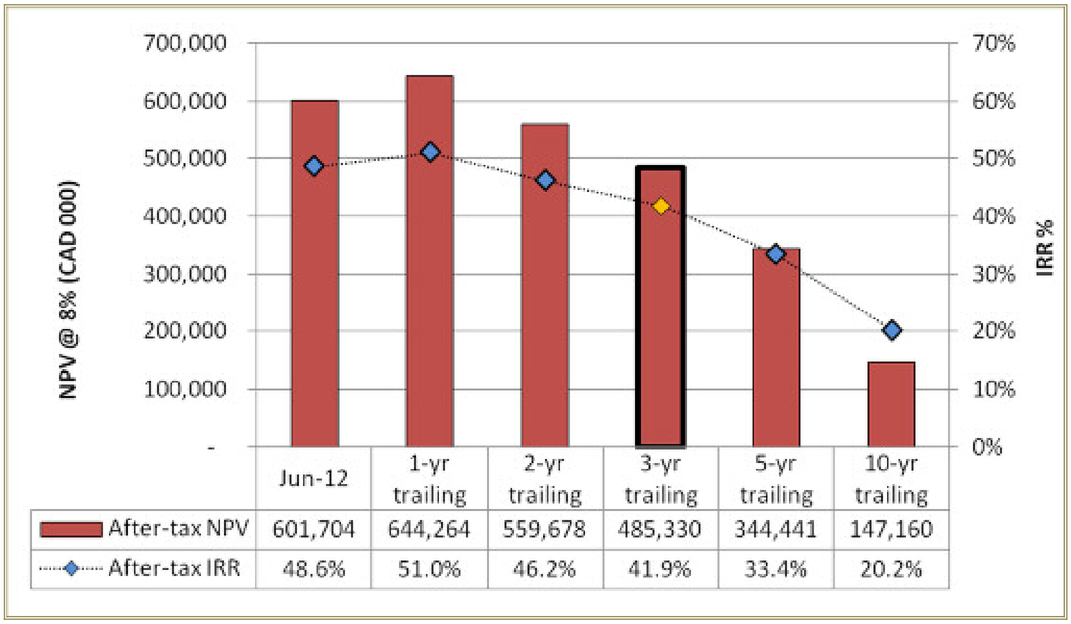


Table 1-19: Sensitivity to Metal Prices

Pricing Period	Unit	1-Month Average	1-Year Average	2-Year Average	3-Year Average	5-Year Average	10-Year Average
Pre-tax IRR	%	60.1	63.1	57.1	51.7	41.1	24.9
After-tax IRR	%	48.6	51.0	46.2	41.9	33.4	20.2
Pre-tax NPV_8	C$ '000s	818,095	874,215	762,681	664,648	478,875	218,744
After-tax NPV_8		601,704	644,264	559,678	485,330	344,441	147,160
Pre-tax NPV_8	US$ '000s	786,630	840,591	733,348	639,084	460,456	210,330
After-tax NPV_8		578,561	619,485	538,152	466,664	331,194	141,500

Sensitivity to Process Flowsheet

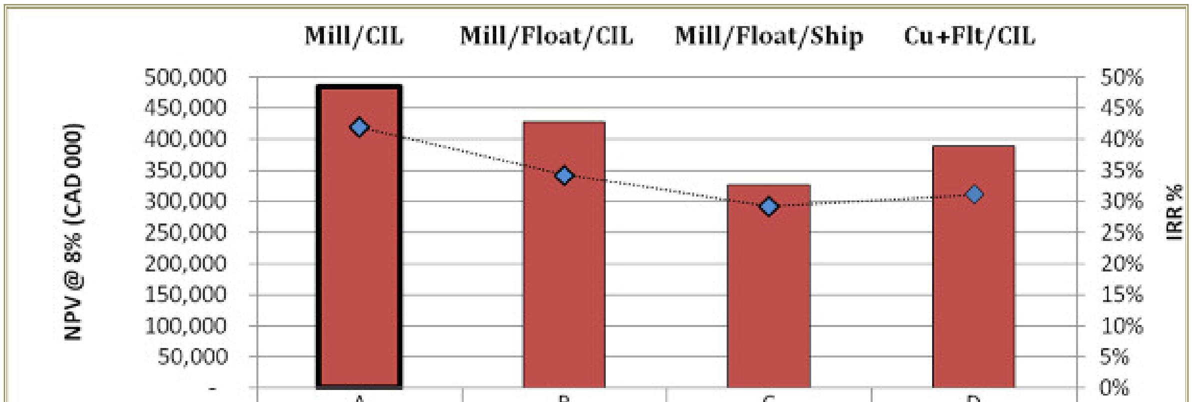
Micon conducted a trade-off study to determine the optimum process flowsheet for the Curraghinalt deposit. The four flowsheets considered were:

- crushing/milling followed by CIL and gold recovery
- crushing/milling/gravity concentration followed by flotation of a concentrate for CIL/gold recovery

- crushing/milling/gravity concentration followed by flotation of a concentrate for sale
- crushing/milling followed by flotation of (i) a copper concentrate for sale and (ii) a bulk sulphide concentrate for CIL/gold recovery.

A comparison of the cash flows for each of these options suggests that Option A, Milling/CIL provides the best overall economic return, and so this was selected as the base case for this study. Figure 1-15 shows the NPV and IRR for each of the four options.

Figure 1-15: Sensitivity to Process Flowsheet



	A	B	C	D
After-tax NPV	485,330	427,006	324,494	388,505
After-tax IRR	41.9%	34.2%	29.1%	31.1%

1.13 Conclusions

Some information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is currently being planned.

The Curraghinalt deposit represents an orogenic gold deposit with parallel veins and sub-veins or veinlets, which have been traced by trenching and drilling over a strike length of approximately 2 km. Twenty-one D veins comprise the primary zones of mineralization in the 2014 resource model.

The mineral resource for the Curraghinalt Deposit is shown in Table 1-1 at a cut-off grade of 5.00 g/t Au. The Measured and Indicated Resource is 3.0 million tonnes at 10.41 g/t Au and the Inferred Resource 8.0 million tonnes at 9.67 g/t Au. The effective date of the resource is January 20, 2014.

No minimum width constraint was applied before reporting the 2014 resource. The interpretation method of including a minimum of 2 m of downhole length to define a vein resulted in an average horizontal thickness of 2.57 m.

Mineral resources are reported at a cut-off grade of 5 g/t Au. TMAC cautions that mineral resources that are not mineral reserves do not have demonstrated economic viability.

The D veins remain open at depth and along strike. Also, additional material may result from the interpretation and interpolation of the C veins which are not included in the 2014 resource model.

Dalradian Resources has designed a program of exploration and preliminary engineering for the Curraghinalt gold deposit as well as the remainder of the mineral licence areas at the Tyrone project. Details for the exploration and development program are included in Section 26.

Mining of the Curraghinalt deposit will be undertaken using a Longitudinal Sublevel Retreat Underground Mining method with both paste- and waste rock backfill to maximize the extraction of the high value resources at the Curraghinalt Project, reduce the surface footprint required for tailings disposal and allow savings in the volume and transportation cost of waste rock from underground development. Stopes will have a minimum width of 1.80 m and development of sills along the veins will be restricted to 3.0 m width to minimize dilution. The sills will be developed in both directions along the strike length of the mineralized zone, providing two production faces for each vein on each level. Levels will be mined at 20 m intervals. Haulage ramps will have dimensions 4.5 m x 4.5 m to accommodate 30-ton trucks.

The PEA considers four possible process flowsheets for the extraction of gold from the Curraghinalt resource. Of these, the whole-ore leach (Option A) was considered the most effective and has been used as the base case for project evaluation. Nevertheless, Micon considers additional testwork, including optimizing grind, reagent strengths and retention times are required, and pilot studies are required before flowsheet development and design specifications can be finalized.

Micon considers the relatively low grade and elevated concentrations of penalty elements of the copper concentrate produced in Option D will present a challenge with regard to shipping and marketing.

The PEA has been prepared using the metallurgical testwork results available at the time. Subsequent and ongoing testing by ALS Metallurgy suggests that further investigation of the Option B and C flowsheets is worth pursuing as an alternative to whole ore cyanidation.

Infrastructure required for the Project has been identified and is provided for in the evaluation. A site-specific layout has not been developed, though, pending discussion with affected landowners over the siting of these works.

The provision of electricity to the mine will require a new overhead power line, probably from Strabane, approximately 27 km from the mine. A two-year permitting and construction period is expected following a decision to develop the mine.

The PEA assumes that collection of rainfall in the tailings storage facility will provide process make-up water, but further work is required to properly assess water supply sources available to the Project.

Future development of the Project will need to take into account its location within an Area of Outstanding Natural Beauty, and proximity to Areas of Special Scientific Interest and Special Areas of Conservation.

A small proportion of the waste rock samples tested were acid generating, so it will be necessary for such material to be segregated and stowed as underground backfill. On closure of the mine, the entrance will be plugged and flooding of the workings will then minimize oxidation.

Based on its economic evaluation of the base case and sensitivity studies, Micon concludes that this PEA demonstrates the viability of the Project as proposed, and that further development is warranted.

1.14 Recommendations

Some information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is currently being planned.

Dalradian Resources has proposed a multifaceted exploration and development program for advancing the Curraghinalt deposit. The proposed work is intended to further explore the extensions of the mineralization, infill gaps in the current resource, continue development work to increase confidence in the resource, test proposed mining methods, and further refine the process flowsheet. Additional exploration work is planned to advance other targets in the Dalradian Supergroup, but is not considered in the proposed budget presented here.

Dalradian has proposed the following breakdown in expenditures:

Exploration $5,000,000

Underground Development Program $12,000,000

Total $17,000,000

Snowden made recommendations regarding the collection and analysis of geotechnical data in its January 2012 report. Micon concurs with those recommendations. In addition, Micon recommends that during geotechnical data collection the rock mass should be accurately characterized and described. This allows the rock mass attributes, parameters and ratings to be assigned at a later stage enabling the classification to be made in any of the proposed rock mass classification systems (e.g., RMR, Q-system or MRMR).

Regarding development of the mine design, Micon recommends that:

- further optimization of the mining method be performed to evaluate potential additional economic benefits to the Project by considering and combining a higher selectivity, non-mechanized mining method with the proposed Longitudinal Sublevel Retreat
- further studies be performed on the backfill system for the Project and that a systematic laboratory test program be performed on the mine tailings to determine the optimum cement addition and cement type to be used as the binding agent for the paste backfill

- non-PAG waste rock produced during underground exploration and mine development activities may be utilized as bulk fill materials to meet the construction needs, where appropriate. The structural fill material will be imported from local suppliers, as required.

Based on a review of reports leading up to the metallurgical testwork currently underway, it is recommended that future testwork should include the following:

- locked-cycle flotation testing of the bulk sulphide concentrate circuit
- cyanidation tests of the bulk sulphide concentrate (rougher and cleaner products), both with and without regrinding, pre-aeration, and lead nitrite addition
- additional BBWi testing
- SAG amenability testwork
- further evaluation of the effect of grind size on leach extractions for the whole ore cyanidation option
- evaluation of alternative flotation reagents, especially collector 3418A
- carbon-in-Leach (CIL) testwork
- testwork with higher cyanide concentrations
- confirmatory testwork on a variety of representative feed samples once a preferred flowsheet is established.

Golder recommended three sites for further investigation as potential tailings storage facilities. Although these sites were selected on the basis of conventional (slurry) tailings disposal, Micon suggests that paste or dry stack tailings be considered to minimize the footprint and increase the available options for siting the tailings and employ progressive reclamation to minimize visual impacts.

Potential sources of process water should be investigated, including a study of groundwater in wells close to the proposed underground mine, as recommended by SLR, as well as further study of the proposed collection of rainfall at the TSF.

In this PEA, the estimated unit cost for electrical power is based on best available information. As the Project develops, Dalradian Resources should obtain project-specific pricing.

Negotiations with landowners should take place as the Project moves forward so that a site-specific layout of the plant and infrastructure can be developed during the next stages of project engineering.

2 INTRODUCTION AND TERMS OF REFERENCE

This National Instrument 43-101 (NI 43-101) Technical Report has been prepared by T. Maunula & Associates Consulting Inc. (TMAC) to provide an update of the Mineral Resource Estimate of the gold mineralization for the Curraghinalt deposit (Curraghinalt) on the Northern Ireland Property, Counties Tyrone and Londonderry, Northern Ireland (the “Project” or the “property”).

In 2012, Dalradian Resources commissioned Micon International Ltd. (Micon) to complete a Preliminary Economic Assessment (PEA) in accordance with NI 43-101 for the Project. Sections 15 to 19 and Sections 21 to 22 of the current Technical Report are repeated from the PEA (Hennessey et al., 2012b) as the current mineral resource does not invalidate the key elements of the 2012 PEA. The PEA was based upon an estimate of the Curraghinalt mineral resources originally prepared by Micon in November 2011 (Hennessey et al., 2012a) which is repeated in Section 14.1.

Sections 1 to 14, 20, 23, 25 and 26 have been updated to the effective date of this report.

This Technical Report was prepared in accordance with the reporting standards and definitions required under NI 43-101.

2.1 Curraghinalt Personal Inspections

For the current Mineral Resource Estimate, author T. Maunula, P. Geo visited the property on August 12 to 16, 2013 and December 14 to 15, 2013. The site visit included review of current drill holes at the core storage/logging facilities in Omagh, visiting the project site near Gortin and inspection of the underground workings.

For the PEA, authors B. Foo, P. Eng, C. Jacobs, C.Eng. MIMMM, and A. Villeneuve, P. Eng. visited the property on April 18 to 19, 2012. In addition to an inspection of the underground workings, the team visited the surface of the project site and potential tailings storage sites in the district, Dalradian Resources offices in Gortin, and core storage/logging facility and offices in Omagh.

Previously, the Northern Ireland Property was visited by B. Terrence Hennessey, P.Geo., Vice President of Micon and based in the firm’s head office in Toronto, Ontario Canada, on November 6 and 7, 2009. Mr. Hennessey conducted a second site visit to the Curraghinalt project from October 3 to 5, 2010.

Messrs. Maunula, Hennessey, Foo, Damjanović, Villeneuve, and Jacobs are all Qualified Persons (QP) as defined in NI 43-101.

3 RELIANCE ON OTHER EXPERTS

TMAC and Micon have reviewed and analyzed exploration data provided by Dalradian Resources, its consultants and previous operators of the property, and has drawn its own conclusions therefrom, augmented by its direct field examination. Micon and TMAC have not carried out any independent exploration work, drilled any holes or carried out any significant program of sampling and assaying. However, precious metal-bearing veins are visible in an adit and drifts developed underground and have also been found on surface in a nearby gravel pit.

While exercising all reasonable diligence in checking, confirming, and testing it, TMAC and Micon have relied upon the data presented by Dalradian Resources, and any previous operators of the project, in formulating its opinion.

The various agreements under which Dalradian Resources holds title to the mineral lands for this project have not been thoroughly investigated or confirmed by TMAC (refer to Section 4.2).

TMAC has relied on the expertise of SLR Consulting Ltd. in its reporting of social, environmental, and permitting issues.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The property is located in County Tyrone and County Londonderry, Northern Ireland. The Curraghinalt deposit is found on the property, approximately 75 km due west of Belfast and 15 km northeast of the town of Omagh. The property is accessible by paved road, a distance of approximately 127 km from Belfast (Figure 4-1).

Figure 4-1: General Location Map

4.2 Description

Dalradian's property in Northern Ireland measure approximately 84,000 ha comprising four contiguous areas (DG1, DG2, DG3, and DG4), to which the Company has title. There are two elements comprising the titles—base metal mineral prospecting licences (Prospecting Licences), and mining lease option agreements (Option Agreements) for gold and silver – which are controlled by two separate government bodies, as described in more detail below. Dalradian does not hold any other titles.

Dalradian holds, through its wholly-owned subsidiary Dalradian Gold Limited (DGL), a 100% interest, subject to royalties described below, in Prospecting Licences and Option Agreements in counties Tyrone and Londonderry, Northern Ireland, United Kingdom. The Department of Enterprise, Trade and Investment (DETI) has granted to DGL Prospecting Licences for base metals on four contiguous areas referred to as DG1, DG2, DG3 and DG4. The Crown Estate Commissioners (CEC) has entered into Option Agreements with DGL for gold and silver over the same four areas.

The current DETI Prospecting Licences for DG1 and DG2 (named DG1/14 and DG2/14) expire December 31, 2015, at which point they can be extended for another two years. The Prospecting Licences for DG3 and DG4 (named DG3/11 and DG4/11) have a renewal term expiring April 23, 2015 at which point they can be extended for another two years. Every six years (i.e., after two 2-year extensions), DGL must reapply for the Prospecting Licences before the Licences expire. Reapplication for the Prospecting Licences for DG1 and DG2 will be required in 2019 and for DG3 and DG4 in 2016.

CEC Option Agreements for DG1 and DG2 have a renewal term expiring December 31, 2015. The Option Agreements for DG3 and DG4 have a renewal term expiring April 23, 2015. The CEC Option Agreements have a two-year term and can be renewed indefinitely at the CEC's discretion.

The four pieces of property are often referred to simply as DG1, DG2, DG3, and DG4, or DG-1, DG-2, DG-3, and DG-4 (with or without hyphens). In this report they shall will be referred to as DG1, DG2, DG3, and DG4, although some older figures provided herein show them as DG-1, DG-2, DG-3, and DG-4, as well as DG-01, DG-02, DG-03, and DG-04, or as TG-03 (DG3) and TG-04 (DG3 and (DG4).

DETI uses the Irish National Grid system of easting and northing for reference. The Northern Ireland Property is located at approximately 257700 mE and 386000 mN.

The current licences and options, including sizes and expiry dates, are presented in Table 4-1. The DETI licences and the CEC option agreements expire on the same dates, and so only four records are shown in Table 4-1.

Table 4-1: Licence Details

Licensee	Area	Licence Number	Area (km^2)	Minerals	Date of Issue	First Extension Granted	Maximum Date of Expiry
Dalradian Gold Limited	Curraghinalt	DG1/14	167.5	All(1)	01/01/2014	NA	31/12/2019
Dalradian Gold Limited	Mountfield	DG2/14	184.5	All(1)	01/01/2014	NA	31/12/2019
Dalradian Gold Limited	Newtownstewart East	DG3/11	248	All(1)	24/04/2011	23/04/2013	23/04/2017
Dalradian Gold Limited	Sawel-Dart	DG4/11	244	All(1)	24/04/2011	23/04/2013	23/04/2017

Note: (1) Concurrent DETI licences / CEC option agreements.

The mineral resource estimate presented in this report is located entirely on the property covered by licence DG1. A net smelter return (NSR) royalty of 2% is payable to Minco plc on a portion of DG1. As provided in the option agreements, a 4% royalty will be payable to the CEC upon production of silver and/or gold on the Northern Ireland Property.

Table 4-2 lists the required annual expenditures made on each of the four licences. Tournigan held the licences from 2002 to late 2009, at which time they were transferred to Dalradian Resources. The licences outline the annual general work program to be undertaken on the four licences.

Prospecting licences (and CEC option agreements) in Northern Ireland are issued for two-year periods and may be renewed (or extended), subject to relevant conditions being met and satisfied, on two occasions for a period of two further years, for a total of six years. After the end of each two-year period, a progress work report is submitted within three months of the Licence anniversary date to the licensing authorities in order to support renewal, along with a letter indicating the intention to continue work on the license into the following two-year period. Included in the reports is a summary of the work performed for the year and an audited summary of the spending. At the end of each six-year period, a full reissuing application process must be undertaken, whereby a full six-year work report is submitted to the licensing bodies along with a new application for another six-year period. Renewals, extensions, and reissuing are not automatic.

Table 4-2: Licence Expenditures

Licence Number	Reporting Stage	From (day/month/year)	To (day/month/year)	Expenditure (GBP)
DG1/14	Reissued (further 6 years)	01/01/2014	31/12/2014	40,000
		01/01/2015	31/12/2015	40,000
DG2/14	Reissued (further 6 years)	01/01/2014	31/12/2014	40,000
		01/01/2015	31/12/2015	40,000
DG3/11	Reissued (further 6 years)	24/04/2011	23/04/2012	No Requirement
		24/04/2012	23/04/2013	No Requirement
DG3/11	First Extension	24/04/2013	23/04/2014	50,000
		24/04/2014	23/04/2015	50,000
DG4/11	Reissued (further 6 years)	24/04/2011	23/04/2012	No Requirement
		24/04/2012	23/04/2013	No Requirement
DG4/11	First Extension	24/04/2013	23/04/2014	50,000
		24/04/2014	23/04/2015	50,000

Licensees are required to give notice of their intention to enter land to carry out work, and must seek the agreement of landowners before entering their property.

DETI is required to consult with other departments and with public bodies concerning its intention to issue a licence, and is also required to place notices in the Belfast Gazette and at least one local newspaper. This is primarily to allow the owners of surface land within the area under application the opportunity to make their views known.

DETI notes that a draft licence and a “letter of offer” are provided to applicants once all comments have been considered. The letter of offer may contain a number of conditions, although DETI notes that, at the prospecting stage, it is usually sufficient for the applicant to inform all listed contacts of its plans and progress. When the conditions set out in the letter of offer are accepted and the terms of the draft licence agreed upon, the licence is executed by DETI and the company.

DETI states that planning permission is not required for early stage exploration, although the Planning Service of the Department of the Environment must be informed of the planned work, including the nature and scale, time and location of the company's activities, and drill hole locations.

4.3 Location of Mineralized Zones

Previous exploration has identified the Curraghinalt gold vein system. Ennex International plc developed an adit between 1987 and 1989 to access the deposit. The location of the Curraghinalt gold deposit relative to the boundaries of DG1 and DG2 can be seen in Figure 4-2. Details of the locations of the mineralized zones, the portal, and adit are shown in Figure 4-3.

Figure 4-2: Curraghinalt Gold Deposit Location

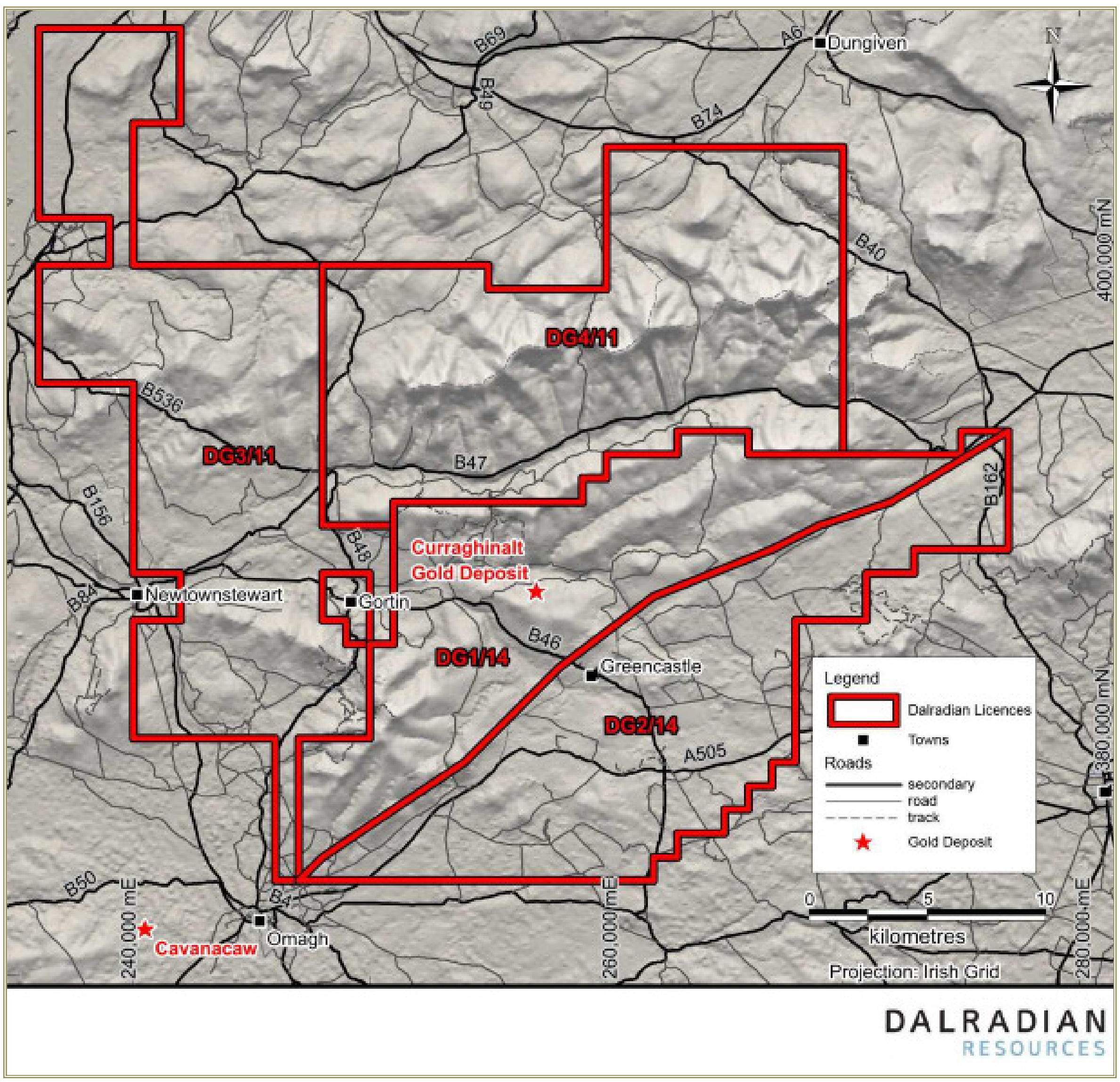
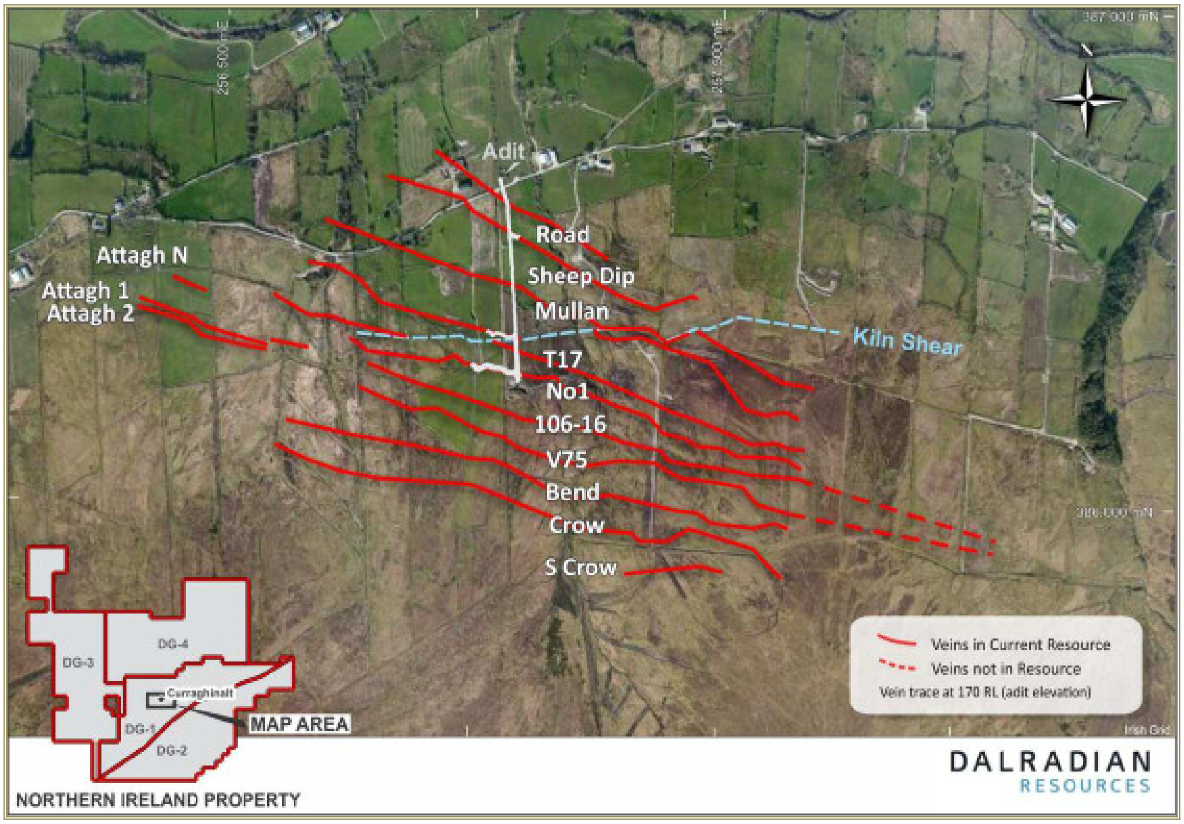


Figure 4-3: Locations of Principal Veins and Adit on DG1



5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

Access to the property is via a number of highways and local roads, including the B48 from Omagh to Gortin, and the B46 from Gortin to Greencastle. Local county roads, private roads, and farm tracks provide generally good access within the property. Figure 5-1 shows the network of local roads and rivers around the property area.

Local climate conditions are temperate, with an average annual temperature of 9°C, and average daily temperatures varying between 4.1°C and 14.7°C throughout the year. Average annual precipitation is 852 mm, the majority of which falls in the winter months between September and January (average >80 mm per month). Snowfall is usually restricted to areas of higher elevation and occurs on 10 days or less per year. Exploration activities can generally be conducted year-round.

The town of Omagh (population 50,000) provides lodging and local labour, as do smaller local villages. Few experienced mining personnel are available locally, although there is a small mining industry in Northern Ireland (salt and gold), and the Irish Republic has a number of underground base metal mines. There is a large quarrying industry in Northern Ireland. The principal economic activities in the area of the licences are sheep farming and, to a lesser extent, the raising of beef cattle.

Belfast is the capital city of Northern Ireland and supports a population of approximately 300,000 inhabitants. From Belfast, Omagh and the project can be reached by paved road, more than half of which is dual carriageway (limited access highway). The over-road distance is approximately 110 km, and requires less than 1.5 hours in good weather. A domestic and an international airport serve Belfast, together offering frequent daily flights to the rest of the UK and Europe.

The village of Gortin (pronounced Gorchin) is located a few kilometres from the Curraghinalt gold deposit at the western edge of licence DG1 (Figure 5-1). Dalradian Gold has a field office there, as well as storage facilities, all of which are rented. Gortin is centrally located within the licence areas and well situated to support the exploration program. Dalradian Gold also leases an office and core storage facility in Omagh near the road leading to Gortin. Geology and administration staff are located there, as well as the principal core logging and storage facility.

A principal power substation is located at Plumbridge, approximately 22 km north of Omagh, and the main 110 kV power line runs just outside Omagh. Local water resources are abundant.

The topography consists of rolling hills and broad valleys (Figure 5-2). Glacial deposits and peat cover much of the area, resulting in mixed forest and heathlands, as well as farmland in the valleys. Relief ranges between around 100 masl in the major river valleys to a maximum height of approximately 550 masl.

Figure 5-1: Northern Ireland Property Licence Map, Major Road Access, and Drainages

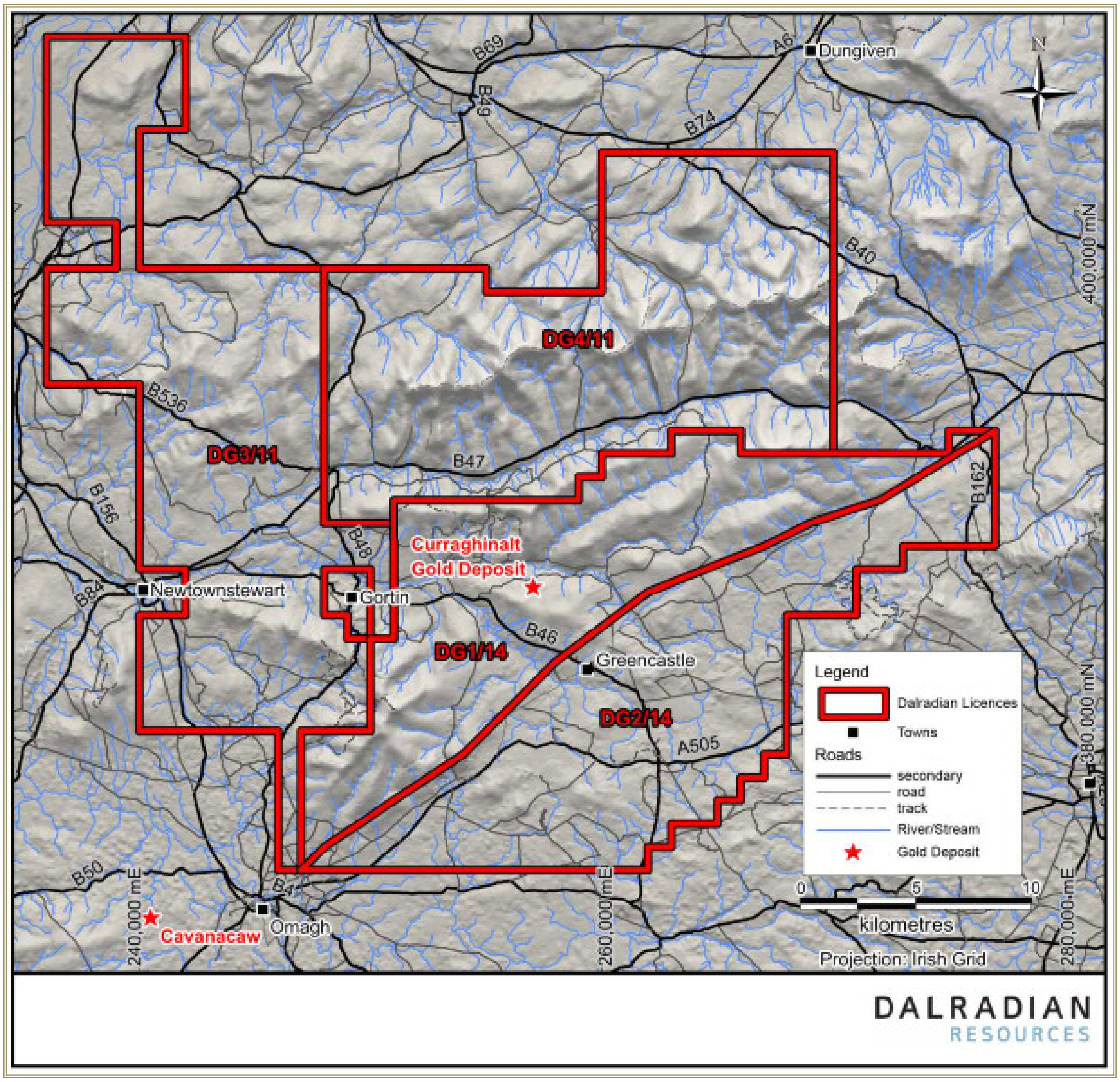


Figure 5-2: General View of the Curraghinalt Deposit Area Looking South

6 HISTORY

6.1 Acquisition History

The property containing the Curraghinalt deposit was initially acquired by Ulster Base Metals (which later became Ulster Minerals) in 1981, an entity which later became a wholly-owned subsidiary of Ennex International plc (Ennex). Ennex conducted exploration on the property between 1982 and 1999. Ennex sold its interest in Ulster Minerals to Nickelodeon in January 2000. In August 2000, the name of Nickelodeon was changed to Strongbow Resources Inc., and subsequently to Strongbow Exploration Inc. (Strongbow).

In February 2003, Tournigan Gold Corp (Tournigan) entered into an option agreement with Strongbow to earn an interest of up to 100% in the Curraghinalt deposit, located within a prospecting licence known as UM-11/96. Terms included staged exploration expenditures of C$4.0 million over a period of seven years, the delivery of a bankable feasibility study, and issuing shares to Strongbow at a price based on a 90-day trading average. At the same time, Tournigan entered into a similar option agreement with Strongbow for its Tyrone project, located within prospecting licence UM-12/96. Tournigan established Dalradian Gold as a wholly-owned subsidiary through which it would earn its interests in the Curraghinalt (UM-11/96) and Tyrone (UM-12/96) properties.

In the following year (February 2004), Tournigan entered into a letter agreement with Strongbow for the outright purchase by Tournigan of all of the issued and outstanding shares of Ulster Minerals through the issue of 5 million shares of Tournigan. The earlier option agreements were terminated and replaced by the letter of agreement. A NSR of 2% held by Ennex was transferred to Minco plc. Full transfer of ownership in Ulster Minerals to Tournigan was completed in December 2004.

Tournigan then applied to the licensing authorities, and received licences TG-3 and TG-4 (for both minerals and precious metals), to the northwest of UM-11/96 and UM-12/96.

Ulster Minerals licences UM-11/96 and UM-12/96 were later renamed UM-1 and UM-2, and ultimately DG1 and DG2. During the renaming and re-registering process, the internal boundary between DG1 and DG2 was reoriented from east–west to a position that reflects the approximate location of the Omagh Thrust Fault (refer to geological descriptions in Section 7 and Figure 7-2). Details of the current licences and option agreements are provided in Section 4.

In October 2009, Dalradian Resources completed a purchase and sale agreement with Tournigan to acquire all of the issued and outstanding shares of Dalradian Gold, which included the licences, mineral rights, and surface rights (including easements) in the Area of Interest. The Area of Interest is defined in the agreement as Mineral Prospecting Licences DG1, DG2, TG-3, and TG-4; the latter two being renamed to DG3 and DG4 by DETI after acquisition (refer to Section 4).

6.2 Exploration History

Gold was recognized in the gravels of the Moyola River to the east of the property in 1652, and in the 1930s, an English company reported plans for alluvial gold mining in a prospectus. Documented exploration in the area dates back to the early 1970s, when companies such as AMAX Exploration of the UK, Consolidated Goldfields, Selection Trust, and RioFinex completed grassroots exploration campaigns over the areas covered by DG1, DG2, DG3, and DG4. Following the 1975 report titled "The Geology and Metalliferous Mineral Potential of the Sperrin Mountains Area" by the GSNI, the ground covered by the licences comprising the property received renewed interest by a number of companies. Licence DG1 has been the focus of most of the historical exploration on the property, as outlined in Table 6-1.

Table 6-1: Historical Exploration of DG1

Company	Year	Work Completed	Area
AMAX Exploration of UK Inc.	1971–1972	Soil sampling	
Glencar Explorations Ltd.	1977–1978	Soil sampling, panning	
Ennex	1983–1987	Detailed prospecting, geochemistry, geophysics	Curraghinalt
	1983–1987	68 trenches (2,856 m)	
	1983–1987	63 diamond drill holes (6,387 m)	
Dungannon	1983–1984	Stream and soil sampling, panning, and geological mapping	DE5 Licence: included Golan Burn
Dungannon/Celtic Gold	1985	Detailed soil sampling, mapping prospecting Percussion overburden drilling (Pionjar)–107 sites; RC Drilling – 50 holes	DE5 Licence: included Garvagh, Slievebeg
Dungannon/Celtic Gold	1986–1987	Detailed soil sampling, mapping Prospecting VLF surveys; RC drill holes – 19 holes Diamond drill holes – 55 holes	DE5 Licence Garvagh Garvagh
Ennex	August 1987–March 1989	Underground development program (797 m)	Curraghinalt
Ennex	May 1995–March 1996	59 diamond drill holes (4,980 m)	Curraghinalt
Ennex	June 1996–June 1997	50 diamond drill infill holes (5,400 m)	Curraghinalt
Nickelodeon	2000	Due diligence underground channel samples	
Strongbow	2000–2003	226 mobile metal ions(MMI) geochemistry samples	Glenlark
Strongbow	2000–2003	Ground IP geophysical survey	Glenlark
Strongbow	2000–2003	Trench T10	Glenlark
Tournigan	2003–2007	22,910 soil samples, geophysical survey, prospecting	DG1
		26 diamond drill holes (4,391 m)	Curraghinalt
		7 drill holes	Glenlark
Tournigan	2007–2009	Resource Estimate (2007)	Curraghinalt
		4 deep diamond drill holes	Curraghinalt

The four phases of exploration at Curraghinalt, conducted by Ennex between 1983 and 1997, are summarized as follows:

- Phase 1 (1983 to July 1987):
  - detailed prospecting, geochemistry, and geophysics
  - 68 trenches (2,856 m) and 63 diamond drill holes (6,387 m)
- Phase 2 (August 1987 to March 1989):
  - underground development program, including development of an adit (412 m), lateral drifting (325 m), and raising (60 m)
  - lateral development using a Dosco SL 120 road header
- Phase 3 (May 1995 to March 1996):
  - detailed and reconnaissance drilling to test previously inadequately drilled veins
  - reconnaissance drilling of veins to the southwest of previously-drilled areas (total 59 holes (4,980 m))
- Phase 4 (June 1996 to May 1997):
  - infill drilling on 25 m to 30 m centres in the main vein areas (drilling of 50 holes (5,400 m)).

Between 1997, when Ennex transferred its interest to Nickelodeon, and late 2002, when the agreement was signed between Strongbow and Tournigan, little work was done at Curraghinalt.

The Tournigan exploration at Curraghinalt can be broken into several phases.

- Phase 1, 2003 to January 2005:
  - 22,910 soil samples collected
  - Small geophysical survey conducted
  - Mapping and prospecting on the DG1 Licence area
  - 26 diamond drill holes (4,391 m) drilled at Curraghinalt
  - 7 diamond drill holes drilled at Glenlark
- Phase 2, January 2005 to 2007:
  - 2 diamond drill holes drilled in the area of the Crows Foot-Bend
  - 24 diamond holes drilled (infill drilling) on the Southeast Extension target
- Phase 3, 2007 to 2009:
  - Resource estimate completed by Micon (November 29, 2007)
  - 4 deep diamond drills holes completed. After completion of the 2007 drill program, Tournigan ceased all exploration activity at the Curraghinalt deposit; except for some prospecting on TG-3 and TG-4 in 2008, the property remained inactive until its acquisition by Dalradian Resources in 2009.

Exploration programs on the ground currently covered by Licence DG2 initially targeted base metals; later both gold and base metals were sought. Historical exploration on DG2 is summarized in Table 6-2.

Table 6-2: Historical Exploration DG2

Company	Year	Type of Work
Consolidated Gold Fields	1970	Soil, stream and prospecting surveys
Selection Trust Exploration	1971–1972	Stream surveys, soil surveys, IP and EM surveys, trenching
Rio Tinto Finance & Exploration (RioFinex)	1972	Soil and stream surveys
Rio Tinto Finance & Exploration (RioFinex)	1973	Soil and stream surveys, magnetic and IP surveys, panning, trenching
Rio Tinto Finance & Exploration (RioFinex)	1974	Magnetic, IP, prospecting, drilling, pits, soil reconnaissance, and follow–up surveys
Glencar Explorations Ltd	1977–1978	Panning, soil surveys
Ulster Base Metals Limited	1982	Prospecting, VLF survey
Ulster Base Metals Limited	1983	VLF and magnetic survey, soil and deep overburden surveys, prospecting
Ulster Base Metals Limited	1984	Prospecting
Ulster Base Metals Limited	1985	Prospecting, deep overburden surveys , magnetic, IP and VLF surveys
Ennex International PLC	1986	Drilling, prospecting, deep overburden survey, IP and magnetic surveys, panning
Ennex International PLC	1987	Trenching, drilling, prospecting, deep overburden surveys, IP, VLF and magnetic surveys
Ennex International PLC	1988	Deep overburden surveys, magnetic and IP surveys
Ennex International PLC	1989	IP and VLF surveys, prospecting
Ennex International PLC	1997	Deep overburden surveys
Strongbow Resources	2001	Soil (MMI) at Crosh
Tournigan Gold Corporation	2004	Prospecting

The principal target of interest for Ennex on DG2 was the Cashel Rock showing, where a gold-mineralized silicified rhyolite breccia outcrop is exposed at surface. At this location, 15 shallow drill holes targeted the zone just below surface. The results and example sections were presented in Hennessey and Mukhopadhyay (2010). They are not relevant to the mineral resource estimate for the Curraghinalt deposit presented in this report, and are not repeated here.

Licences DG 3 and DG4 have also been the subject of regional-scale exploration programs (Table 6-3 and Table 6-4); however, there has not been follow-up drilling of any targets on these licences.

Table 6-3: Historical Exploration of DG3

Company Name	Year	Work Completed
AMAX Exploration of UK, Inc.	1971–1972	Soil surveys
Glencar Exploration Ltd.	1974	Stream surveys
Glencar Exploration Ltd.	1975	Soil and stream surveys
Glencar Exploration Ltd.	1977–1978	Soil surveys and panning
Ulster Base Metals Ltd.	1982	Prospecting
Ulster Base Metals Ltd.	1982–1983	Panning
Dungannon Explorations Ltd.	1983	Soil, stream, and deep overburden surveys, panning
Dungannon Explorations Ltd.	1984	Deep overburden surveys
Ulster Base Metals Ltd.	1985	Deep overburden and VLF surveys, panning, prospecting

Company Name	Year	Work Completed
Dungannon Explorations Ltd.	1985	Deep overburden surveys
Ennex International Plc.	1986	Prospecting and panning
Dungannon Explorations Ltd.	1986	Soil and deep overburden surveys, prospecting, panning
Ennex International Plc.	1987	IP, VLF and deep overburden surveys, prospecting
Dungannon Explorations Ltd.	1987	Stream and soil surveys
Celtic Gold Plc.	1987	Soil and deep overburden surveys, prospecting, panning
Ennex International Plc.	1988	Deep overburden surveys, prospecting
Celtic Gold Plc.	1988	Deep overburden, stream, and soil surveys, trenching, prospecting, panning
Ennex International Plc.	1989	Magnetic surveys, prospecting
Celtic Gold Plc.	1989	Stream
Celtic Gold Plc.	1996	Deep overburden surveys, prospecting
Brancote Mining Ltd.	1997	Stream surveys, panning, prospecting
Billiton UK Resources	1997	Magnetic survey
Ennex International Plc.	1997	Deep overburden survey
Brancote Mining Ltd.	1998	Magnetic, IP, Stream, deep overburden and soil surveys, prospecting, panning
Brancote Mining Ltd.	1999	Magnetic surveys, prospecting, panning
Tournigan Gold Corporation	2004	Prospecting

Table 6-4: Historical Exploration DG4

Company Name	Year	Work Completed
Glencar Explorations Ltd.	1977–1978	Soil surveys, panning
Rio Tinto Finance & Exploration (RioFinex)	1982	Stream surveys, panning
Dungannon Exploration Ltd.	1983	Deep overburden, soil and stream surveys, panning
Rio Tinto Finance & Exploration (RioFinex)	1983	Stream surveys, panning
Dungannon Exploration Ltd.	1984	Deep overburden surveys
Dungannon Exploration Ltd.	1985	Soil surveys
Ulster Base Metals Ltd.	1985	Deep overburden surveys, panning, prospecting
Dungannon Exploration Ltd.	1985	Deep overburden surveys
Ennex International Plc. International Plc.	1986	Deep overburden surveys, soils, panning, and prospecting
Ennex International Plc.	1987	Prospecting
Ennex International Plc.	1988	Deep overburden and VLF surveys, panning, prospecting
Celtic Gold Plc.	1988	Stream and soil surveys, panning, prospecting
Ennex International Plc.	1989	Magnetic surveys and prospecting
Celtic Gold Plc.	1989	Stream surveys
Ennex International Plc.	1995	Soil surveys
Brancote Mining Ltd.	1997	Stream and soil surveys, panning, prospecting
Biliton UK Resources	1997	Magnetic surveys
Ennex International Plc.	1997	Deep overburden surveys
Brancote Mining Ltd.	1998	Stream, soil, and magnetic surveys, panning, prospecting
Tournigan Gold Corporation	2004	Soil surveys, prospecting

6.3 Historical Mineral Resource Estimates

In May 1997, a polygonal resource estimate was prepared on behalf of Ennex by CSA Group (CSA, 1997). The estimate was prepared on a minimum mining width of 1.25 m and at a cut-off grade of 6 g/t Au. It is an historical estimate and is provided for information purposes only. It predates and is not compliant with NI 43-101, and should not be relied on.

Tully prepared a mineral resource estimate in 2005 (Tully, 2005); Micon completed a mineral resource estimate on the Curraghinalt deposit for Tournigan in 2007 (Mukhopadhyay, 2007), and again in 2010 for Dalradian Resources (Hennessey and Mukhopadhyay, 2010). The 2005, 2007, and 2010 estimates were NI 43-101-compliant, and are considered to be historical estimates under the current version of the instrument (June 30, 2011). The 2005, 2007, and 2010 estimates can be found on SEDAR (www.sedar.com), filed under Tournigan and Dalradian Resources.

Micon also completed a mineral resource estimate in 2011 (Hennessey et al., 2012a) upon which Micon prepared a PEA (Hennessey et al., 2012b).

6.4 Historical Production

There is no evidence of any historical mineral production from the property. There is no record that the mineralized material removed by Ennex during bulk sampling in 1987 was ever processed.

7 GEOLOGICAL SETTING

7.1 Regional Geology

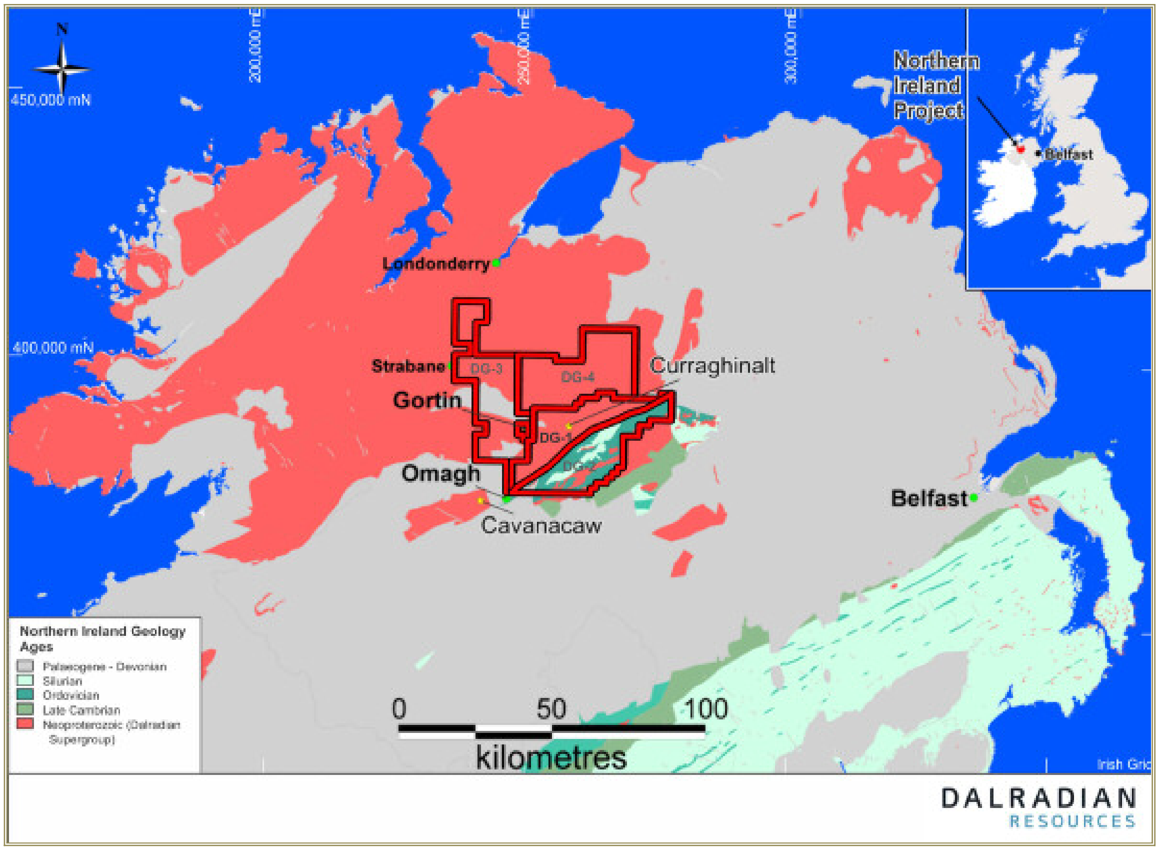
The following is taken primary from Mitchell (2004). The Caledonian orogenic belt of the British and Irish Caledonides resulted from the progressive closure of the Iapetus Ocean and Tornquist Sea during the early Palaeozoic. Assembly and docking of the terranes that form the basement in Northern Ireland commenced in mid-Ordovician time and continued for 80 Ma through the Silurian and finished in the Early Devonian. Final closure was accommodated by sinistral strike slip movement on terrane bounding faults. Northern Ireland covers three of the seven suspect terranes that together constitute the Caledonian Orogen in Ireland. From north to south, these are referred to as the Central Highlands (Grampian) Terrane, Midland Valley Terrane and the Southern Uplands-Down-Longford Terrane.

Dalradian's Northern Ireland Property straddles two of these terranes, the Central Highlands to the north (DG1, DG3, and DG4) and the Midland Valley to the south (DG2). The Central Highland Terrane consists of Moinian (Mesoproterozoic) and Dalradian (Neoproterozoic-Cambrian) rocks and Caledonian igneous intrusions. The southern margin of the terrane is marked by the concealed Fair Head-Clew Bay Line, which is interpreted as the southwesterly extension or major splay of the Highland Boundary Fault in Scotland. The associated regional magnetic lineament that extends southwestwards to Clew Bay in County Galway is located 10 km north of the Variscan (Carboniferous) Omagh Thrust.

The Midland Valley Terrane in Northern Ireland comprises Upper Paleozoic, Mesozoic and Paleogene rocks. However, in County Tyrone a late Ordovician to early Silurian succession is exposed with part of an early Ordovician ophiolite and island arc volcanic complex (Tyrone Igneous Complex) at its base. At the core of the Tyrone Igneous Complex is the fault bounded Central Inlier. This consists of schist and gneiss of Moinian affinity and originally formed part of the Central Highlands Terrane.

Figure 7-1 shows the regional geology of Northern Ireland.

Figure 7-1: Regional Geology of Northern Ireland

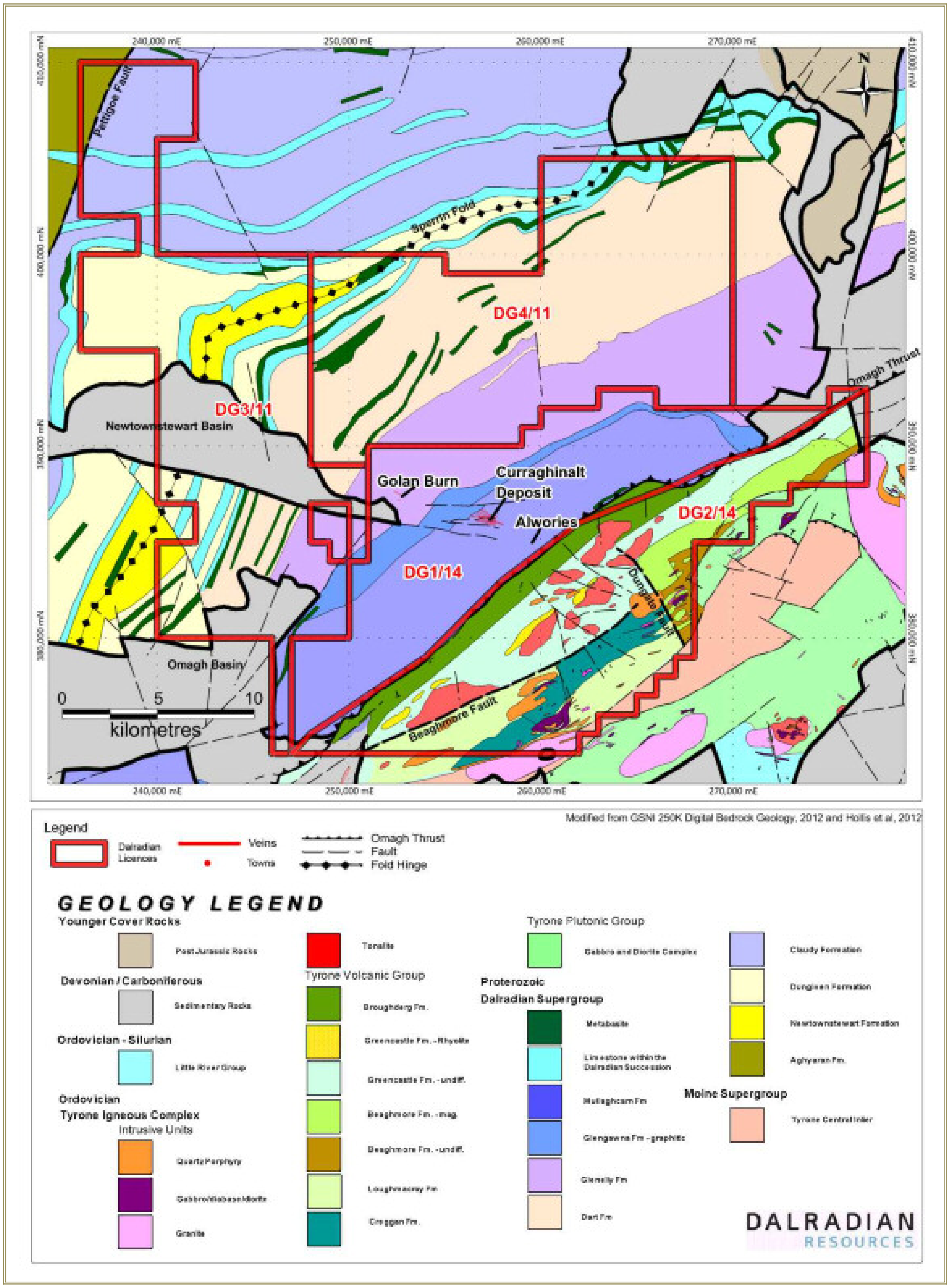


7.2 Property Geology

7.2.1 Dalradian Supergroup – Licences DG1, DG3, and DG4

On the Northern Ireland Property, Neoproterozoic aged rocks of the Dalradian Supergroup underlie DG1, DG3, and DG4 and form the Sperrin Mountains (Figure 7-2). The Dalradian Supergroup is divided into the Argyll Group and Southern Highland Group, both comprised of predominantly clastic marine sediments deposited in a rift basin. The oldest rocks on the property belong to the Newtownstewart Formation (Argyll Group) which is exposed in the core of the recumbent Sperrin Fold and is flanked by Dungiven Limestone Formation (Table 7-1) on DG3 and DG4. The Southern Highland Group is interpreted to flank the Argyll Group on both limbs of the Sperrin Fold although the stratigraphy differs markedly between the north and south limbs. Mitchell (2004) notes that "an absence of distinctive marker horizons allied to lateral facies changes makes correlation difficult between formations and results in the different nomenclature north and south of the fold axis".

Figure 7-2: Property Geology



The Southern Highland Group comprises a thick sequence of turbiditic arenite and pelitic metasediments with rare volcaniclastic (green bed) and calcareous schist units (Table 7-1). Progressing southeastward onto DG1, the Southern Highland Group is exposed and is divided from northwest to southeast into the Dart, Glenelly, Glengawna and Mullaghcarn Formations. The mineralized quartz-carbonate veins of the Curraghinalt deposit are hosted by the Mullaghcarn Formation.

Table 7-1: Stratigraphy of the Dalradian Supergroup

Group	Formation	Lithology
Southern Highland	Mullaghcarn	Semipelite, psammite, pelite
	Glengawna	Black graphitic pelites; psammite; semipelite
	Glenelly	Volcaniclastic semi-pelite; semipelite; psammite
	Dart	Schistose amphibolite; feldspathic and calcareous semipelite
Argyll	Dungiven	Limestone; pelite; semipelite; psammite; quartzite; basaltic pillow lavas; volcaniclastic sediments
	Newtonstewart	Quartzose psammite and thin pelite interbeds

Dart Formation

At the base of the Dart Formation, in contact with the underlying Dungiven Limestone Formation is the Glenga Amphibolite Member, which is interpreted as a resedimented volcaniclastic siltstone and sandstone. The remainder of the formation consists of conglomerate, psammite, schistose semipelite, and a volcaniclastic member.

Glenelly Formation

The Glenelly Formation comprises silvery to greenish grey schistose pelite and semipelite with miner psammite and limestone. Plagioclase porphyroblasts are ubiquitous in the rocks of this formation with more localised occurrences of small euhedral garnet and randomly distributed needles of tourmaline. Also present is a volcaniclastic member and a limestone and calcareous semipelite member.

Mullaghcarn Formation

The Mullaghcarn Formation consists predominantly of semi-pelites and psammites with subordinate pelitic horizons and chloritic semi-pelites. This unit is host to the Curraghinalt deposit and the Alwories prospect. Although not subdivided on the GSNI maps (Figure 7-2) because of lack of outcrop, a variation in magnetic intensity is apparent in the regional Tellus geophysical data suggesting internal variations are present.

The southern boundary of the Dalradian Supergroup is marked by the Omagh Thrust, a moderately northwest dipping thrust fault active as late as the Carboniferous (Figure 7-2).

Deformation and Metamorphism of Dalradian Supergroup

The following is summarized from Mitchell (2004). At least four phases of deformation are recognized in Dalradian rocks on the property:

1. D1 – manifested are barely discernible folds and cleavages
2. D2 – dominant deformation of the Grampian Orogeny and is associated with the formation of major regional southeast-facing recumbent anticlines including the Sperrin Fold
3. D3 – deformation in the south Sperrin mountains resulted in minor southeasterly-verging folds and low-angle, northwards inclined thrust faults such as the Omagh Thrust Fault which transposed Dalradian rocks to the south south-east over the early Ordovician Tyrone Igneous Complex
4. Post-D3 – structures mainly take the form of localised sets of kink bands and late stage brittle fractures.

The Dalradian Supergroup in Northern Ireland reflects a thermal and pressure gradient increasing from lower greenschist facies in the north to lower amphibolite facies in the south, close to the Omagh Thrust Fault.

7.2.2 Tyrone Igneous Complex – Licence DG2

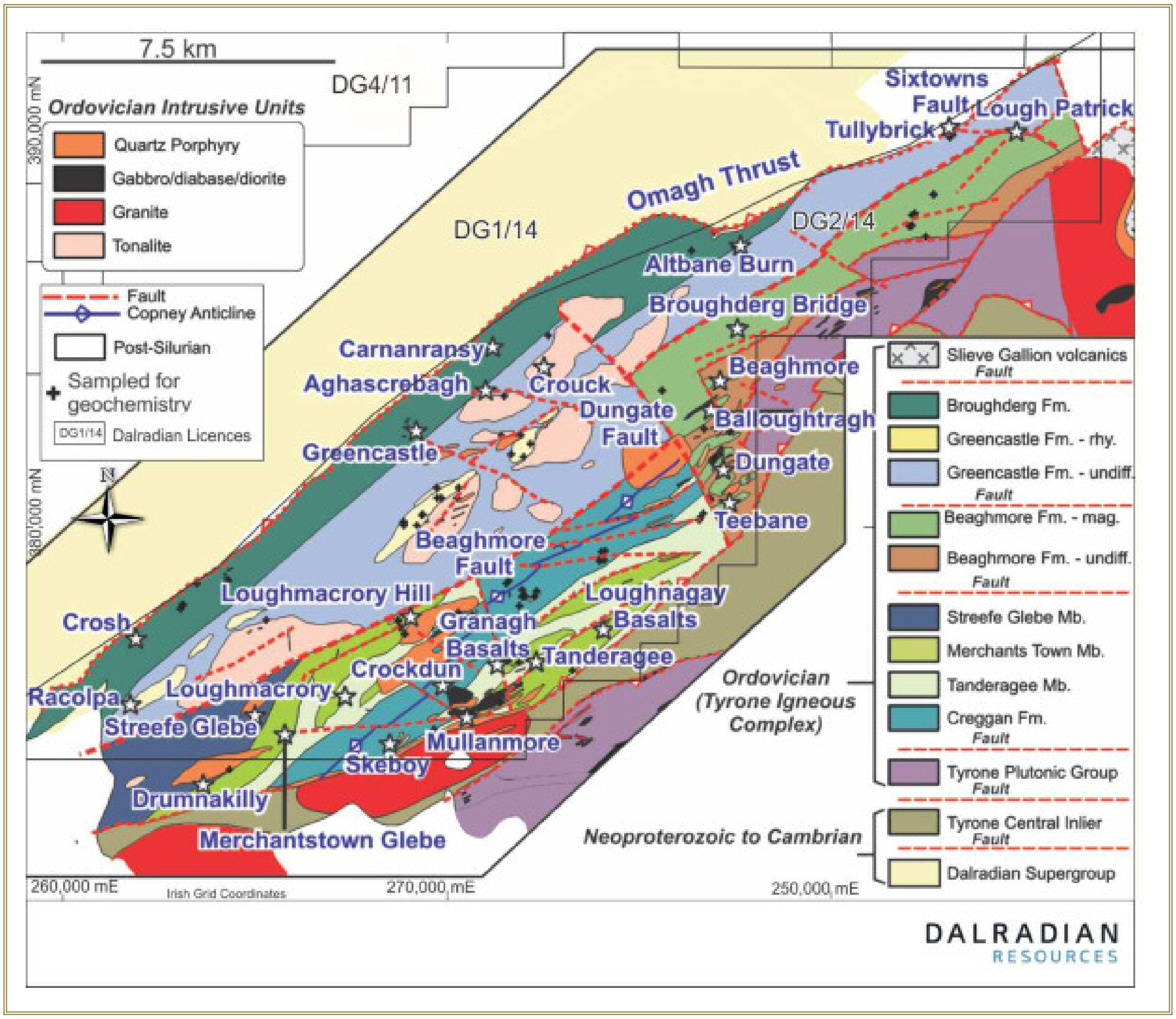
The following is taken from Hollis (2011).

Licence DG2 is largely covered by the Tyrone Igneous Complex (TIC), which is exposed over approximately 350 km^2, within the Midland Valley Terrane and is one of the largest areas of ophiolitic and arc-related rocks exposed along the northern margin of Iapetus within the British and Irish Caledonides. It is broadly divisible into the ophiolitic Tyrone Plutonic Group and the younger arc-related Tyrone Volcanic Group (TVG). The northwestern edge of the Tyrone Igneous Complex is bounded by the Omagh Thrust, which has emplaced Neoproterozoic Dalradian Supergroup metasedimentary rocks above the TVG (Figure 7-2). Within the central regions of the complex (to the southeast of DG2), the structurally underlying metamorphic basement (Tyrone Central Inlier) is exposed. A suite of granitic to tonalitic plutons (c. 470 Ma to 464 Ma) intrudes the Tyrone Igneous Complex and Tyrone Central Inlier (Cooper et al., 2011).

The Tyrone Plutonic Group is interpreted to represent the uppermost portion of a dismembered suprasubduction zone ophiolite and is characterized by layered, isotropic and pegmatitic gabbros, sheeted diabase dikes and the occurrence of rare pillow lavas (Cooper et al., 2011 and references therein). Layered olivine gabbro has provided a U-Pb zircon age of 479.6 ± 1.1 Ma (Cooper et al., 2011). Accretion to the Tyrone Central Inlier must have occurred prior to the intrusion of a 470.3 Ma ± 1.9 Ma tonalite, which contains inherited Proterozoic zircons and roof pendants of ophiolitic material (Cooper et al., 2011).

The TVG forms the upper part of the Tyrone Igneous Complex and comprises mafic to intermediate pillowed and sheeted lavas, tuffs, rhyolite, banded chert, ferruginous jasperoid (ironstone) and argillaceous sedimentary rocks (Mitchell, 2004). The TVG (473 Ma ± 0.8 Ma , Cooper, 2008) is interpreted to have formed within a peri-Laurentian island arc/back-arc, which was accreted to the Tyrone Central Inlier following emplacement of the Tyrone Plutonic Group (Draut et al., 2009; Cooper et al., 2011).

Figure 7-3: Tyrone Igneous Complex Geology



Source: after Hollis et al., 2012

Hollis et al., (2012) have revised the stratigraphy of the TVG based on mapping and geophysics (Figure 7-3, Table 7-2) and the following is summarized from this work. The lower part of the TVG is restricted to south of the Beaghmore fault (southwestern and eastern blocks) and is dominated by basaltic to andesitic lavas and volcaniclastic rocks, with subsidiary agglomerate, layered chert, ferruginous jasperoid (ironstone), finely laminated argillaceous sedimentary rocks, and rare rhyolite breccia, deformed into the SWNE– trending upright Copney anticline. All units in the lower TVG have been subjected to varying degrees of hydrothermal alteration and are characterized by regional subgreenschist- to greenschist-facies metamorphic assemblages. Abundant sills of undeformed quartz ± feldspar porphyritic dacite cut all stratigraphic levels of the Tyrone Volcanic Group.

Table 7-2: Stratigraphy of Tyrone Volcanic Group

Formation	Subdivisions	Description	Mafic Units	Intermediate - Felsic Units	Intrusives
OMAGH THRUST					
Broughderg Formation	Undivided Mountfield Basalts Undivided	Intermediate to felsic crystal (& lesser lapilli) tuff/schist, vesicular basalt, rhyolite, argillaceous sedimentary rocks (e.g. graphitic pelite), layered chert & ironstone (silica-magnetite). Rare basaltic tuff and andesite.	OIB-like, weak arc signature *(R)* Alk *(S)*	Calc-alkalic II *(Q)*	Tonalite Q+/-F. Dacite Porphyry
Greencastle Formation 471.9 ± 0.5 Ma 469.4 ± 0.4 Ma 473.0 ± 0.8 Ma	Undivided Rhyolite Undivided	Volcaniclastic crystal tuff, syndepositional flow-banded & brecciated rhyolite, with lesser diorite, lapilli tuff & rare immature sandstone (feldspathic arenite). Rhyolitic agglomerate (lapilli tuff) contains fragments of rhyolite, felsic tuff, rare diorite & tonalite in a chloritized groundmass. Hornblende-phyric tuffs (at Lough Patrick & Greencastle) are associated with subvolcanic intrusives of hornblende-porphyritic dacite.		Calc-alkalic II *(P)*	Diorite Tonalite: 465.7 ± 1.1 Ma 469.3 ± 0.3 Ma Hb. Dacite Porphyry Q+/-F. Dacite Porphyry
BEAGHMORE FAULT					
Beaghmore Formation	Bonnely Bush Member	Pillowed, massive and sheet-flow (often plagioclase-phyric) lava, crystal tuff and agglomerate. A drillhole at Broughderg Bridge intersected approximately 6 m of andesitic lava underlain by interbedded rhyolite and crystal tuff and basaltic/andesitic lava which becomes more porphyritic at depth.	Fe-Ti eMORB *(O)* & IAT *(N)*	Tholeiitic felsic *(L)* Calc-alkalic I *(M)*	Gabbro Quartz Diabase Diorite Q+/-F. Dacite Porphyry
	Beaghbeg Member	Quartzo-feldspathic volcaniclastic crystal, ash & lithic tuff, interbedded with rhyolitic agglomerate/ lapilli tuff, ferruginous jasperoid (ironstone) and vesicular basalt. Rare rhyolite breccia (with scoria and chert).	Fe-Ti eMORB *(O)*	Tholeiitic felsic *(L)* Calc-alkalic I *(M)*	
DUNGATE FAULT					
Loughmacrory Formation	Streefe Glebe Member	Grey, foliated plagioclase and pyroxene bearing crystal tuff with rare occurrences of mafic lava.	Unknown affinity	Calc-alkalic I *(K)*	Gabbro/Diabase Diorite Hb. Andesite Porphyry Quartz-porphyritic Microtonalite Microgranite Q+/-F. Dacite Porphyry 465.0 ± 1.7 Ma
	Merchantstown Glebe Member	Chloritized pillowed, massive and sheet-flow basalt/basaltic andesite, with subordinate quartzo-feldspathic & pyroxene-plagioclase porphyritic crystal tuff, & rare volcanic agglomerate with chert fragments.	Fe-Ti eMORB, weak arc signature *(I)*	Calc-alkalic I *(J)*	
	Tanderagee Member	Interbedded crystal (& lithic) tuff, andesitic lava, rare pillowed basalt, volcanic agglomerate, layered grey and graphitic black chert, ironstone (ferruginous jasperoid), siltstone & rhyolite breccia.	Fe-Ti eMORB *(G,H)* & CAB *(F)*	Calc-alkalic I *(E)*	
Creggan Formation		Upper: Vesicular, massive & sheet-flow basalt, locally associated with basaltic agglomerate & subordinate weakly-foliated chloritized crystal tuff. Lower: Pillow lavas of fine- (locally plagioclase-phyric) to medium-grained ophitic basalt. Pale pink to red chert and fine-grained hyaloclasite breccias outcrop between pillow structures. Thin, chloritized and feldspathic crystal tuff beds and white/cream or grey-and-green-banded chert. Rare basaltic agglomerate.	CAB *(D)* & Fe-Ti eMORB, weak arc signature *(C)* Fe-Ti eMORB *(A,B)*		Diabase Diorite Q+/-F. Dacite Porphyry
NOT EXPOSED					

Minimum Stratigraphic Thickness (km): 0, 2, 4, 6, 8, 10

Source: from Hollis et al., 2012

North of the Beaghmore fault, the Greencastle and Broughderg formations of the upper TVG are exposed as a conformable sequence dipping between 35° and 60°NW. Dalradian metasedimentary rocks overlie the succession along its western edge, separated by the low-angle Omagh Thrust, which dips around 30°NW (Alsop and Hutton, 1993). The crosscutting nature of the Omagh Thrust provides a relatively complete section through the upper part of the Tyrone Volcanic Group, which has been metamorphosed to chlorite-grade greenschist facies. Further south, sub-greenschist facies metamorphic assemblages are preserved around Formil. Hydrothermal alteration and associated Zn-Pb-Cu(Au) mineralization are widespread within the Greencastle and Broughderg formations. Mineralization is characterized by pyrite-sphalerite–galena and chalcopyrite in locally silicified, sericitic and/or chloritic tuff/rhyolite (Clifford et al., 1992). Between Racolpa and Broughderg, bodies of tonalite and sills of quartz ± feldspar porphyry intrude both formations. The Greencastle formation is a relatively thick succession dominated by chloritic, locally sericitized and siliceous quartzofeldspathic crystal tuff, flow-banded and brecciated rhyolite, rhyolitic lapilli tuff, lesser diorite, rare arkosic sandstone, and localized occurrences of hornblende-phyric tuff. The overlying Broughderg formation is a diverse succession of intermediate to felsic crystal and lesser lapilli tuff/schist, rhyolite (e.g., around Crosh), vesicular basalt, argillaceous sedimentary rocks, layered chert, and black ironstone (silica-magnetite) with bedded pyrite.

A late suite of I-type, calc-alkalic, tonalitic to granitic plutons intrudes the Tyrone Igneous Complex and Tyrone Central Inlier (Draut et al., 2009). Recent U-Pb zircon geochronology indicates these were intruded between c. 470 and 464 Ma (Cooper et al., 2011). Strong LILE- and LREE-enrichment, coupled with zircon inheritance and strongly negative ε_{Nd} values, suggest that assimilation of Dalradian affinity metasedimentary rocks was an integral part of their petrogenesis (Draut et al., 2009; Cooper et al., 2011).

A gently northwest-dipping cleavage intensifies northwards in the volcanics towards the Omagh Thrust, and is correlated with the S3 fabric in the Dalradian Supergroup. The Laght Hill Tonalite has variable relationships with the fabric in the volcanics – early stage tonalite porphyry bodies are deformed by it, but the main body itself cuts the fabric and contains xenoliths that contain the fabric. This suggests that magmatic activity outlasted the overthrusting of the volcanics by the Dalradian (Hollis, 2012).

Hollis et al., (2014) suggest the Tyrone Igneous Complex of Northern Ireland represents a possible broad correlative of the Buchans-Robert's Arm belt of Newfoundland, host to some of the most metal rich VMS deposits globally. Stratigraphic horizons prospective for VMS mineralization in the Tyrone Igneous Complex are associated with rift-related magmatism, hydrothermal alteration, synvolcanic faults, and high-level subvolcanic intrusions (gabbro, diorite, and/or tonalite). Locally intense hydrothermal alteration is characterized by Na-depletion, elevated SiO_2, MgO, Ba/Sr, Bi, Sb, chlorite–carbonate–pyrite alteration index (CCPI). On the property, stratigraphic horizons favorable for VMS mineralization occur in the Greencastle Formation and in the Broughderg Formation, all of which contain occurrences of base and precious metal mineralization (Hollis et al., 2013).

7.2.3 Carboniferous

Two Carboniferous basins are present within the licence area; the Omagh Basin comprises the Omagh Sandstone Group to south and the Newtonstewart Outlier comprising the Owenkillew Sandstone Group to the north (Figure 7-2).

Omagh Sandstone Group

The Omagh Sandstone Group rests unconformably on Dalradian rocks. The basal unit is up to 100 m thick and is composed of non-fossiliferous red sandstone with calcrete nodules, and quartz pebble conglomerates (Mitchell, 2004). Much of the remaining sequence is dominated by channel sandstone and siltstone that contain Courceyan to early Chadian miospores. However, thin algal limestones with evaporite replacement textures occur locally. Some of the limestones contain rare brachiopods. The exact thickness of this group is difficult to estimate based on the amount of uplift, folding, and erosion that has taken place (Mitchell, 2004).

Owenkillew Sandstone Group

The Owenkillew Sandstone Group rests unconformably on the Dalradian and comprises approximately 1,500 m of predominantly non-marine strata present within a half graben. Lithologies include greenish grey and purplish red sandstone and siltstone, with thin beds of algal laminated limestone (Mitchell, 2004). Mudstones containing miospores have indicated an early Chadian age. The group is thought to have formed in an inter-cratonic basin with current indicators suggesting the sediment source to the north (Mitchell, 2004).

7.3 Mineralization

7.3.1 The Curraghinalt Deposit

High-grade gold mineralization at Curraghinalt occurs as a series of west-northwest trending, steeply dipping, subparallel stacked veins and arrays of narrow extension veinlets.

Both sets of veins are hosted by a sequence of highly strained metasediments of the Mullaghcarn Formation (Figure 7-4). The veins range from a few centimetres wide to over 3 m wide. The present day knowledge of the vein system has been deduced primarily from the underground exposure in the adit, trenching and drill hole data. The known strike extent of the veins is at least 1,900 m. Surface exposures of the vein system are limited to the Curraghinalt and Attagh Burns (creeks). The veins included in the present resource occur over a width of approximately 800 m from north to south and more veins are known to occur to the south. On average, the veins dip between 55° and 75° to the north and have been traced to approximately 1,000 m below surface The vein system remains open at depth.

In 2007 the operators retained Dave Coller, P.Geo., Euro.Geol., a structural and economic geological consultant, to prepare an initial review of the Curraghinalt vein system. Coller (2007) recognized that the west-northwest trending vein system at Curraghinalt consists of complexes and zones, which in turn comprise multiple veins and vein branches (Table 7-3).

Figure 7-4: Curraghinalt Vein System

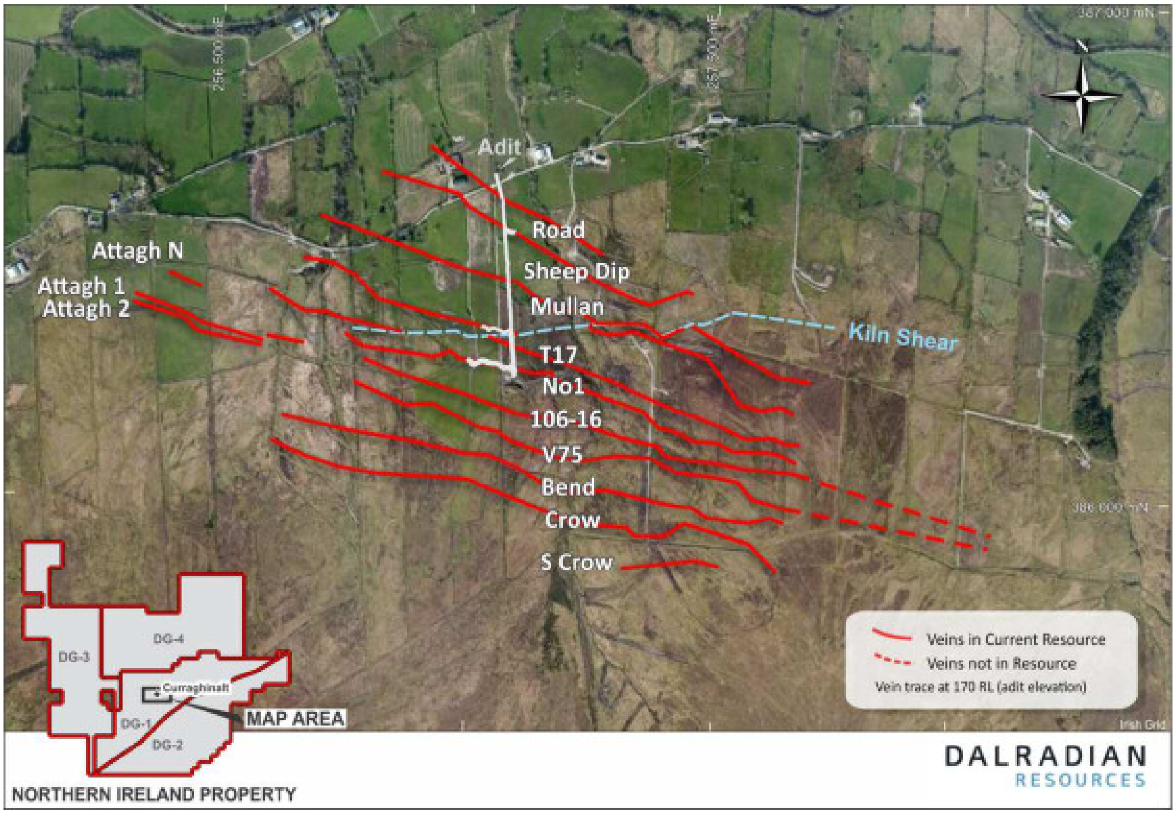


Table 7-3: Definitions for Geometry of the Vein System

Type	Characteristic	Representation	Comments
Vein complex	Series of veins which probably all link in 3D, with one main vein or several en echelon high grade veins and many branches.	Envelope encompassing all the vein branches.	Previously defined as single veins. Several vein zones and linked branches are potentially economic.
Vein zone	Large single or closely spaced branching veins regarded as a single vein.	Width of zone containing veins defined as one vein with average grade as on drill sections.	In detail, internal vein segments and associated veinlets have separate Au assays but would be mined as a single vein.
Vein branch	Veins branch connections between veins zones within a vein complex [sic].	Separate veins in drill sections.	Larger branches often high grade and may be mined by linkage to main vein zone.
Other veins	Major veins that may link or occur between vein complexes.	Separate significant vein zones not named.	Could be defined as separate vein complexes with more drilling.

In 2012 Miron Berezowski, consulting geologist, independently observed the same relationship between individual veins and vein zones and complexes. Additionally, Mr. Berezowski proposed the following nomenclature for the two sets of mineralized veins that have been recognized at Curraghinalt:

- Shear (D) veins – west-northwest trending, steeply dipping, subparallel stacked veins
- Extensional (C) veinlets – arrays of narrow extension veinlets.

Single or multiple D veins form vein zones while vein complexes are anchored by a vein zone and are flanked by C vein arrays.

The D or shear veins are thought to be hosted in west northwesterly trending shear zones which dip moderately to steeply to the northwest and are linearly continuous along strike. D veins are often laminated and include slivers of wall rock, evidence of incremental development. Additionally, D veins are typically brecciated. This cataclastic deformation attests to the reactivation of the shear zones.

The C veinlets are southeast trending, steeply dipping extension veins (tension gashes) which are oriented obliquely to the D veins. They show evidence of open space filling, are never brecciated, and do not have sheared margins.

The Curraghinalt vein system is cut by an east-west, steeply north-dipping, 4 m to 7 m wide ductile shear zone called the Kiln Shear. The Kiln Shear also shows evidence of brittle reactivation as indicated by the presence of gouge zones along the contact between the highly strained ductile rocks within the shear zone and the Dalradian metasedimentary wallrocks. The Kiln Shear clearly disrupts and displaces the vein zones (D veins) with an apparent dextral component on horizontal sections and a reverse component on vertical sections. In the hangingwall or north side of the Kiln Shear, D veins dip steeply, typically 75° N, and are considerably steeper than in the footwall or south side of the Kiln Shear where the average dip is 55° N. This implies either a block rotation or distributed control by shears with a similar dip to the Kiln Shear in the hangingwall or footwall panel.

Vein zones are entrained within the Kiln Shear and previous workers (Boland, 1997) have suggested that the shear has controlled vein emplacement or at least served to produce wider mineralized segments.

Underground (Earls, 1987, Boland 1997)) noted that veins are offset sinisterly by east-northeast and north-northeast trending normal faults that dip northwest. Additionally, low angle, north-dipping thrust faults are also evident in the main access adit. The displacement on all faults observed underground is interpreted to be a few to 10 m.

Petrographic work by Clarke (2004) has documented that the gold mineralization at Curraghinalt occurs in quartz-iron-carbonate pyrite veins and is associated with variable abundances of carbonate, chalcopyrite, and tennantite-tetrahedrite. Gold is commonly in the form of native Au and more rarely as electrum (>20 wt% Ag), and occurs primarily along fractures in pyrite, as inclusions in pyrite, and at pyrite grain contacts with carbonate and quartz. Most native gold grains are associated paragenetically with carbonate, chalcopyrite, tennantite-tetrahedrite, and telluride minerals infilling fractures in pyrite. The seven veins studied at the time have similar mineralogy. Native gold was observed in samples from all veins and grains range in size from 2 μm to 150 μm.

The vein swarm has been traced along strike for approximately 1,950 m, across strike for approximately 800 m and down dip for over 1,000 m by prospecting, trenching, and drilling. The deposit remains open in all directions. Twelve 'main' veins are included in the current resource estimate. These are:

1. Road
2. V75
3. Sheep Dip
4. Bend
5. Mullan
6. Crow
7. T17
8. Attagh1
9. No. 1
10. Attagh2
11. 106-16
12. AttaghN

7.3.2 Tyrone Volcanic Group

The TVG hosts a number of gold and gold plus base metal prospects, which are described in Section 9. Hollis et al., (2014) have identified stratigraphic horizons associated with rift-related magmatism, hydrothermal alteration, synvolcanic faults, and high-level subvolcanic intrusions, which are prospective for VMS mineralization. Hollis et al., (2014) suggest that the TVG is broadly correlative with the Buchans-Robert's Arm belt of Newfoundland, which is host to numerous VMS deposits.

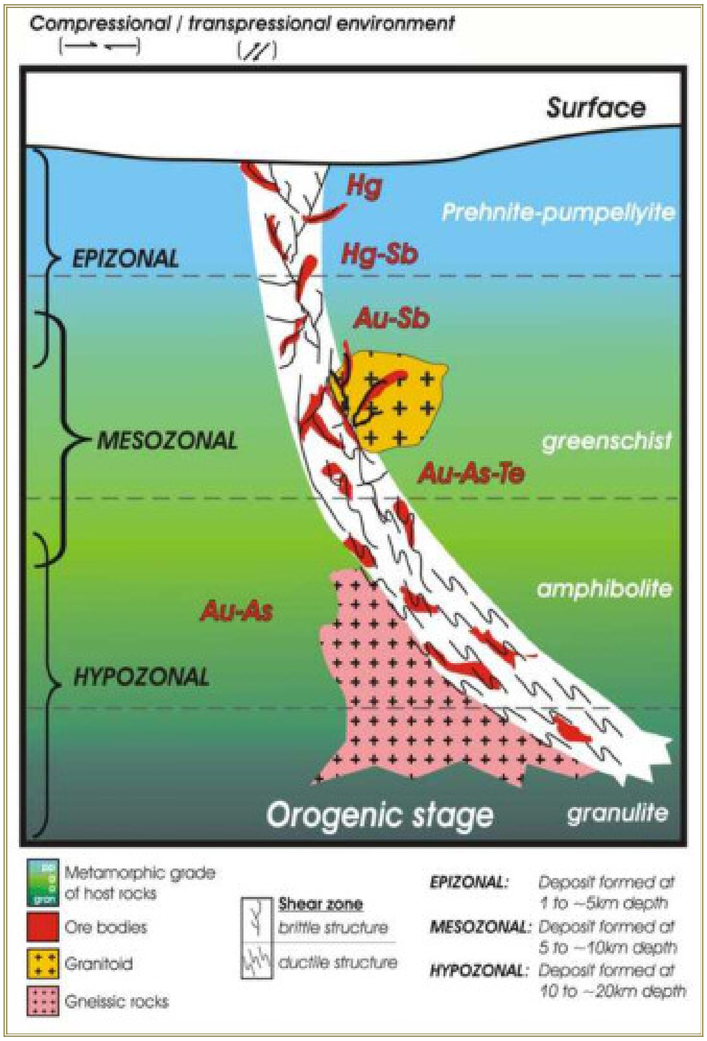
8 DEPOSIT TYPES

8.1 Orogenic Gold Deposit Model

An orogenic gold deposit model best describes the Curraghinalt vein system (Figure 8-1). The following description of the model is taken from Goldfarb (2005):

> *The majority of orogenic gold deposits are located adjacent to first-order, deep-crustal fault zones, which show complex structural histories and may extend along strike for hundreds of kilometers with widths of as much as a few thousand meters. Fluid migration along such zones was driven by episodes of major pressure fluctuations during seismic events. Ores formed as vein fill of second- and third-order shears and faults, particularly at jogs or changes in strike along the crustal fault zones. Mineralization styles vary from stockworks and breccias in shallow, brittle regimes, through laminated crack-seal veins and sigmoidal vein arrays in brittle-ductile crustal regions, to replacement- and disseminated-type orebodies in deeper, ductile environments (i.e., a continuum model). Most orogenic gold deposits occur in greenschist facies rocks, but significant orebodies can be present in lower and higher-grade rocks. Deposits typically formed on retrograde portions of pressure-temperature-time paths and thus are discordant to metamorphic features within host rocks. Spatial association between gold ores and granitoids of all compositions reflects a locally favourable structural trap, except in the case of the intrusion-related gold deposits where there is a clearer genetic association.*
>
> *World-class orebodies are generally 2 to 10 km long, approximately 1 km wide, and are mined downdip to depths of 2 to 3 km. Most orogenic gold deposits contain 2 to 5 percent sulfide minerals and have gold/silver ratios from 5 to 10. Arsenopyrite and pyrite are the dominant sulfide minerals, whereas pyrrhotite is more important in higher temperature ores and base metals are not highly anomalous.*

Figure 8-1: Orogenic Gold Deposit Model



Source: after Goldfarb, 2005

8.2 Volcanic Massive Sulphide Deposit Model

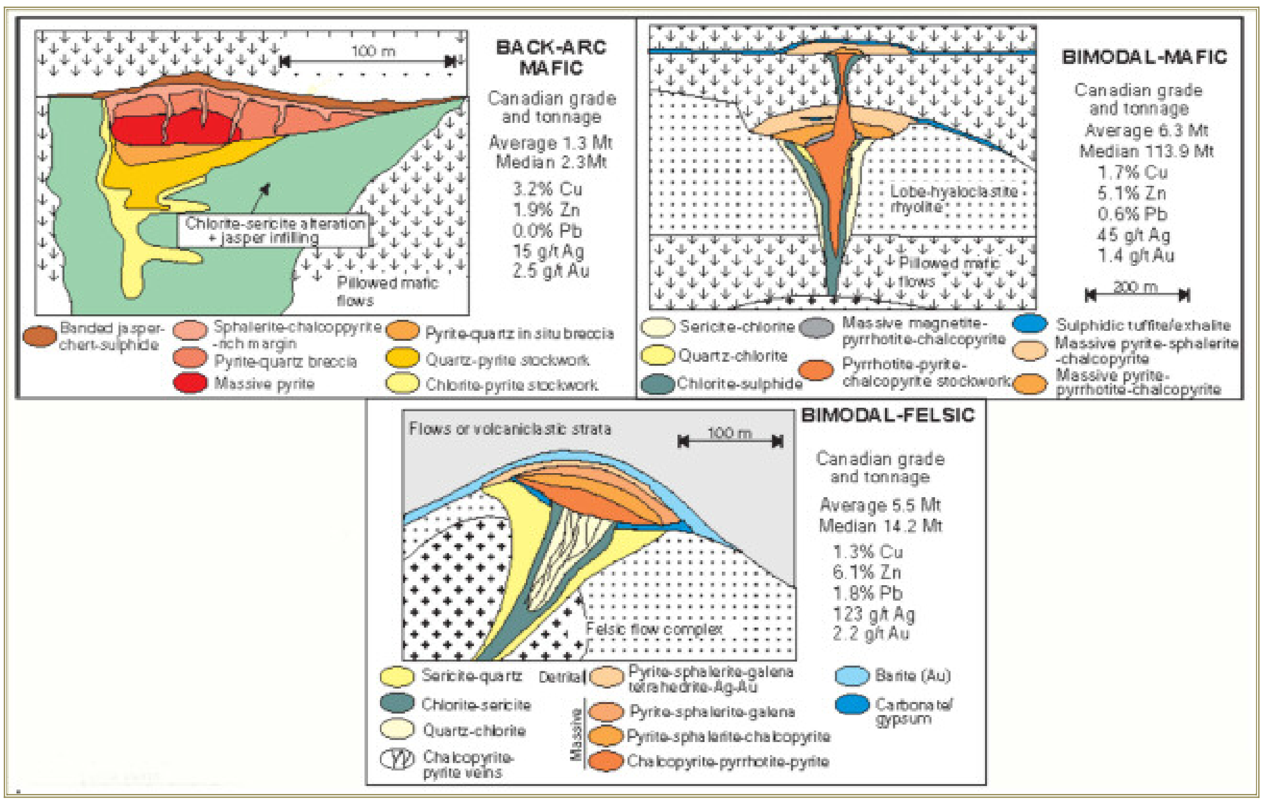
The volcanic stratigraphy underlying DG2 is a potential host to volcanic massive sulphide (VMS) deposits. VMS deposits are syngenetic, stratabound, and in part stratiform accumulations of massive to semi-massive sulphide that form seafloor hydrothermal systems at or near the seafloor (Gibson et al., 2007; Galley et al., 2007). The deposits consist of two parts: a concordant massive sulphide lens (>60% sulphide minerals), and discordant vein-type sulphide mineralization, commonly called the stringer or stockwork zone, located within an envelope of altered footwall volcanic and or sedimentary rocks (Gibson et al., 2007).

Recently, VMS deposits have been classified by host lithologies that define a distinctive time–stratigraphic event (Barrie and Hannington, 1999; Franklin et al., 2005). These five different groups are:

1. bimodal-mafic dominated volcanic – Cu rich
2. mafic back-arc (ophiolite associated) – Cu rich
3. pelitic mafic back-arc
4. bimodal felsic-dominated volcanic – Zn rich
5. siliciclastic – felsic.

The order of the lithologic groups above reflects a change from the most primitive VMS environments, represented by ophiolite settings, through oceanic-rifted arc, evolved rifted arcs, continental back-arc, to sedimented back-arc. Hollis et al. (2014) have identified the Lower TVG as having formed in a bimodal-mafic arc to back-arc, and the Upper TVG as having characteristics of the bimodal-felsic model (Figure 8-2).

Figure 8-2: Volcanogenic Massive Sulphide Deposit Model



Source: Galley et al., 2007

Gold-rich VMS deposits are viewed as a subtype where the gold content exceeds the associated combined copper, lead, and zinc grades.

9 EXPLORATION

Since 2010, Dalradian Resources has completed 135 drill holes at Curraghinalt (DG1), and 28 regional drill holes. In addition, airborne and ground geophysical surveys, prospecting, mapping, and geochemical surveys have been completed. Drilling is discussed in Section 10.

9.1 2010–2011

Dalradian assumed ownership of the project at the end of 2009 and began a drill program at Curraghinalt in March 2010. The drill programs are described in Section 10.

In 2011, Patterson, Grant and Watson Limited (PGW, 2011) reprocessed the Tellus airborne geophysical survey data covering Northern Ireland. Additionally all available historical ground IP, magnetic and VLF/EM-R data was also reprocessed over the four licence areas. Twenty-three exploration targets within the four licences were generated. These targets were evaluated based on published geology and a more detailed data compilation.

A regional exploration programme followed, composed of prospecting on all four licences in the first and second quarters of 2011.

A total of 929 samples were collected, 143 of which assayed greater than 0.25 g/t Au. A summary of the samples is provided in Table 9-1. The locations of the prospecting samples are shown in Figure 9-1.

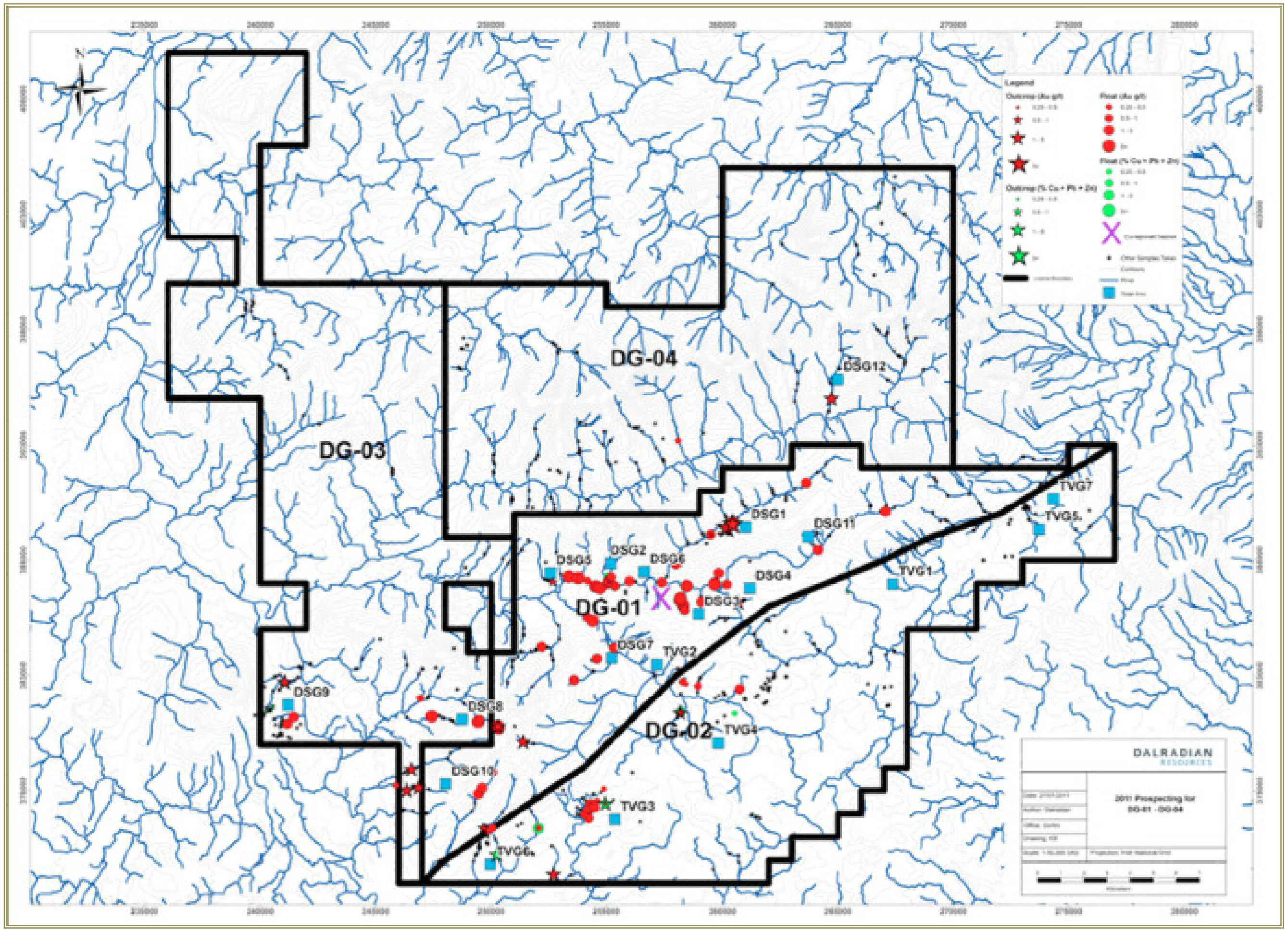
Table 9-1: Summary of 2011 Prospecting Samples

Licence Area	Total Samples	Outcrop Samples	Float Samples	Gold Value Range (g/t Au)	Significance
DG1	316	139	177	0.01 – 44.96	88 samples over 0.25 g/t Au
DG2	270	144	126	0.01 – 5.48	31 samples over 0.25 g/t Au
DG3	184	86	98	0.01 – 14.08	22 samples over 0.25 g/t Au
DG4	159	97	62	0.01 – 2.07	2 samples over 0.25 g/t Au

Integration and re-evaluation of previous exploration and the new regional prospecting information identified 19 out of the 23 areas for detailed follow-up and drill testing. These are shown in Figure 9-2.

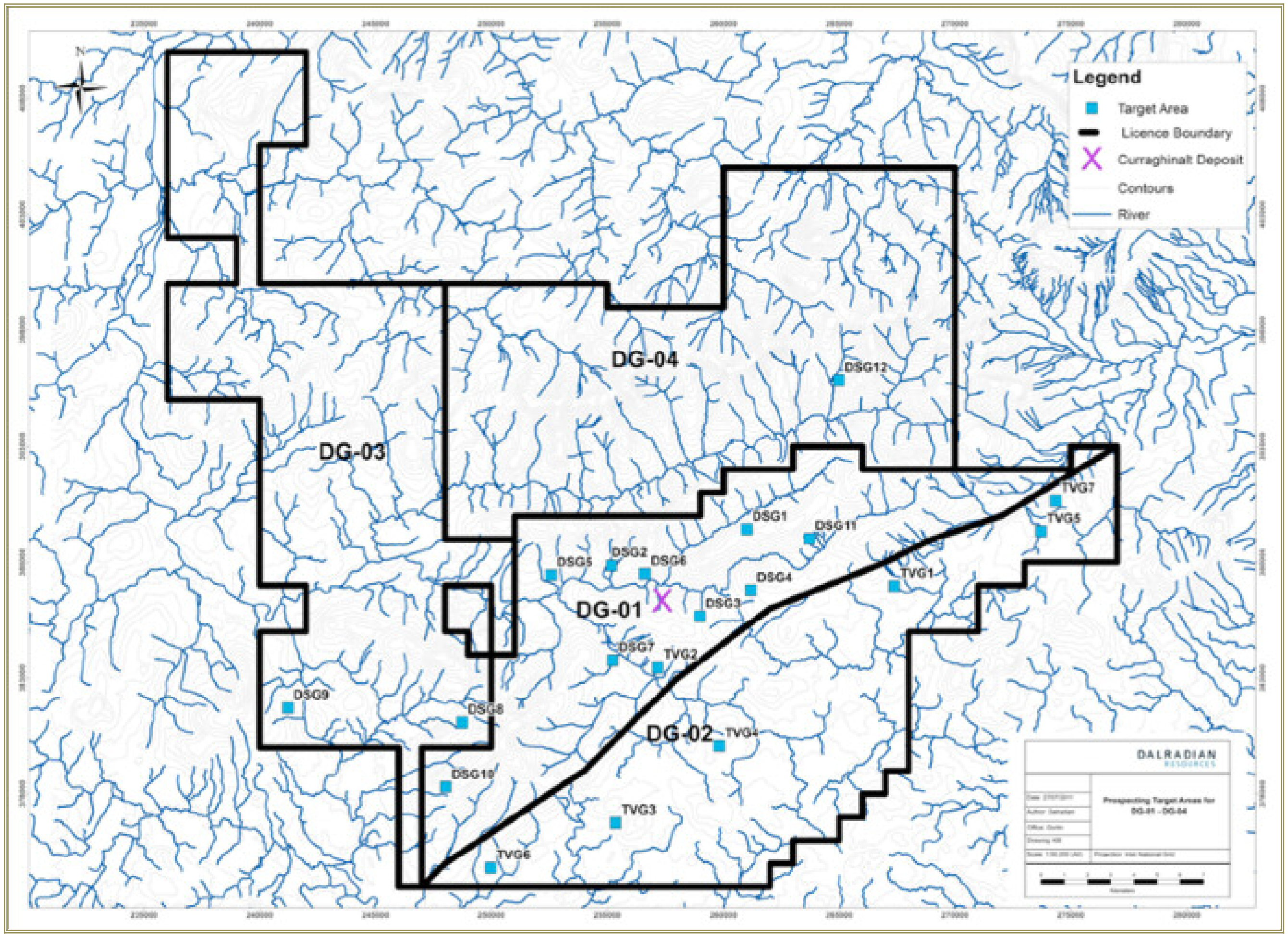
These targets can be split into two distinct groups: those within the TVG, on licence DG2, and those within the Dalradian Supergroup (DSG) in DG1, DG3, and DG4.

Figure 9-1: Location of Prospecting Samples, 2011



Source: Dalradian Resources, 2011

Figure 9-2: Exploration Targets



Source: Dalradian Resources, 2011

The TVG is an environment favourable for the formation of VMS deposits. It is interpreted to have a similar origin as the Buchans VMS camp in Newfoundland, and contains an abundance of float with VMS-style mineralization. Gold and base metal mineralization is most prevalent within the upper part of the volcanic sequence. Historical prospecting results include siliceous tuffs assaying 16.1% Pb and 1.5 g/t Au, and chloritic tuffs assaying 8.8% Zn, 1.2% Pb, and 1.7 g/t Au.

In the Dalradian Supergroup, the highest priority exploration targets are within the Curraghinalt structural corridor, an area of approximately 8 km of strike length and 3 km in width, centred on the Curraghinalt orogenic gold deposit.

The targets in the TVG and DSG are described in Table 9-2 and Table 9-3, and their locations are shown in Figure 9-1 and Figure 9-2.

Following the target identification, a program of regional scout drilling was started, with two targets having been drilled. Two drill holes were completed at TVG1 (Broughderg), and one drill hole was completed at TVG7 (Tullybrick). There were no significant intercepts in these three drill holes. In late 2011, Dalradian Resources suspended the scout-drilling program in order to gather additional geological information to better define drill targets.

Table 9-2: Tyrone Volcanic Group Exploration Targets

Target Name	Area	Target defined by	Significant Samples	Sample Type	Geology	Historical Drilling
TVG1	750 m x 350 m	Historical drilling, trenching, magnetic high, Au soil geochemistry, mineralized outcrop and float	1.5 m at 4.36 g/t Au	Trench	Auriferous chert-magnetite horizon. Interpreted to be exhalative unit	2 shallow drill holes
TVG2	2.9 km x 1.9 km	EM, Au and Zn–Pb–Cu soil geochemistry, mineralized outcrop and float	1.63 g/t Au and 4.3% Cu+Pb+Zn from outcrop	Prospecting	Outcrop of altered tuffs in poorly exposed area	15 shallow drill holes
TVG3	750 m x 500 m	Au soil geochemistry and mineralized float	5.48 g/t Au from quartz float	Prospecting	Rhyolite–tonalite contact with abundant angular quartz float with visible gold	None
TVG4	350 m x 300 m	Historical drilling and trenching, EM, Au soil geochemistry, and mineralized outcrop and float	Historical shallow drill hole: 3.63 m at 30.12 g/t Au	Drill hole	Rhyolite breccia with gold and base metals. Airborne geophysics shows new EM anomalies	15 shallow drill holes
TVG5	4.5 km x 700 m	EM, IP, magnetic geophysics, Zn–Cu soil geochemistry, and mineralized outcrop and float	Historical Prospecting: 4.54 g/t Au in ironstone	Prospecting	Mineralized ironstone overlying altered tuffs and basalts	None
TVG6	3.5 km x 2.2 km	EM, Au soil geochemistry, and mineralized outcrop and float	2.19 g/t Au and 2.99% Cu+Pb+Zn from outcrop	Prospecting	Auriferous rhyolite breccias and tuffs with galena and sphalerite	None
TVG7	1.2 km x 1 km	EM and IP, Zn–Cu soil geochemistry, and mineralized outcrop and float	Historical Prospecting: 1.87 g/t Au in float	Prospecting	Altered volcanic tuffs with quartz veins	None

Table 9-3: Dalradian Supergroup Exploration Targets

Target Name	Area	Target defined by	Significant Samples	Sample Type	Geology	Historical Drilling
DSG1	7 km x 600 m	EM, IP, and mineralized outcrop and float	Trench: 9.5 m at 5.64% Zn+Pb Historical prospecting: 141.2 g/t Au from float Recent prospecting: 33.94 g/t Au from float	Trench and Prospecting	Metasediment-hosted quartz vein and stratiform gold and base metal mineralization	12 shallow drill holes
DSG2	2 km x 50 m	Au soil geochemistry, mineralized subcrop and float	10.52 g/t Au from float.	Prospecting	Graphitic pelite-hosted breccia zone up to 50 m wide and 2 km long	None
DSG3	1.8 km x 375 m	Mineralized float	44.96 g/t Au and 32.80 g/t Au from float	Prospecting	Quartz float, 200 m east of drilled mineralization at Curraghinalt	None
DSG4	1.5 km x 700 m	Au soil geochemistry, mineralized outcrop and float	Channel: 0.88 m @ 39.43 g/t Au	Channel	Quartz vein, 2 km east along strike from Curraghinalt	None
DSG5	1 km x 600 m	Historical drilling, mineralized outcrop and float	Historical shallow drill hole: 0.6 m at 61.43 g/t Au	Drill hole	Quartz vein 4.5 km west along strike from Curraghinalt	44 shallow drill holes
DSG6	7.5 km x 600 m	Au soil geochemistry and mineralized float	22.4 g/t Au from quartz float 3.36 g/t Au from silicified metasediment float	Prospecting	Wide range of float styles in river valley	None
DSG7	2.5 km x 650 m	Au soil geochemistry and mineralized float	11.68 g/t Au from float	Prospecting	Quartz float in river valley	None
DSG8	2.3 km x 1.7 km	Au soil geochemistry, mineralized outcrop and float	14.08 g/t Au from quartz float with pyrite. 8.18 g/t Au from silicified metasediment float	Prospecting	Float train of silicified metasediments and quartz veins	None
DSG9	2 km x 1.5 km	Au soil geochemistry, mineralized outcrop and float	2.96 g/t Au from outcrop	Prospecting	Silicified quartzite with disseminated and fracture fill pyrite	None
DSG10	2.3 km x 0.9 km	Au soil geochemistry, mineralized outcrop and float	1.63 g/t Au from graphitic pelite outcrop 1.88 g/t Au from outcropping quartz vein	Prospecting	Silicified graphitic pelite and quartz veins	3 shallow drill holes
DSG11	4 km x 1 km	Au soil geochemistry, mineralized outcrop and float	2.35 g/t Au from float	Prospecting	Quartz breccias with pyrite in metasediments	None
DSG12	2 km x 1 km	Au soil geochemistry, mineralized outcrop and float	2.07 g/t Au from outcrop	Prospecting	Quartz vein with pyrite.	None

9.2 2012–2013

In April of 2012, Dalradian Resources commissioned a helicopter airborne 1,034.3 line-km Versatile Time Domain Electromagnetic (VTEM) and magnetic survey along 10 km of the Curraghinalt Trend. The area of the concessions surveyed is shown in Figure 9-3. The survey was flown at a 50 m line spacing.

The VTEM survey was unable to detect conductive horizons caused by the sulphide bearing veins that host the gold mineralization at Curraghinalt. However, the conductivity, apparent resistivity, and magnetic signatures were able to resolve stratigraphic subdivisions and contacts.

In addition to the airborne geophysical survey, Dalradian Resources initiated a detailed soil geochemistry survey on DG1 to extend the existing geochemistry grid (Figure 9-4). A coherent gold-in-soil anomaly was interpreted in the Alwories area (Anomaly A on Figure 9-4), and was followed up with drilling in the summer of 2012 (Section 10, Alwories).

Figure 9-3: Airborne Geophysical Survey Coverage

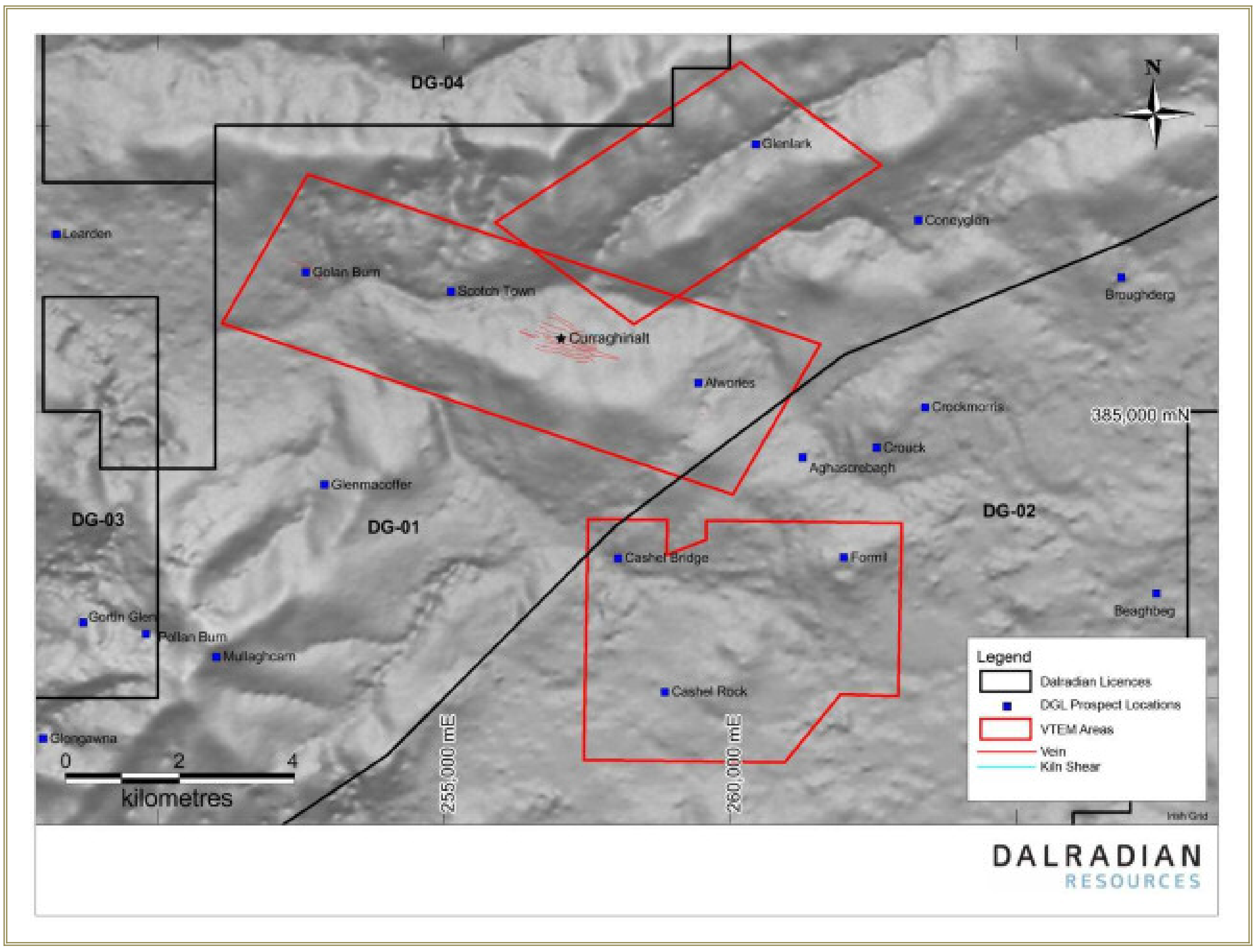
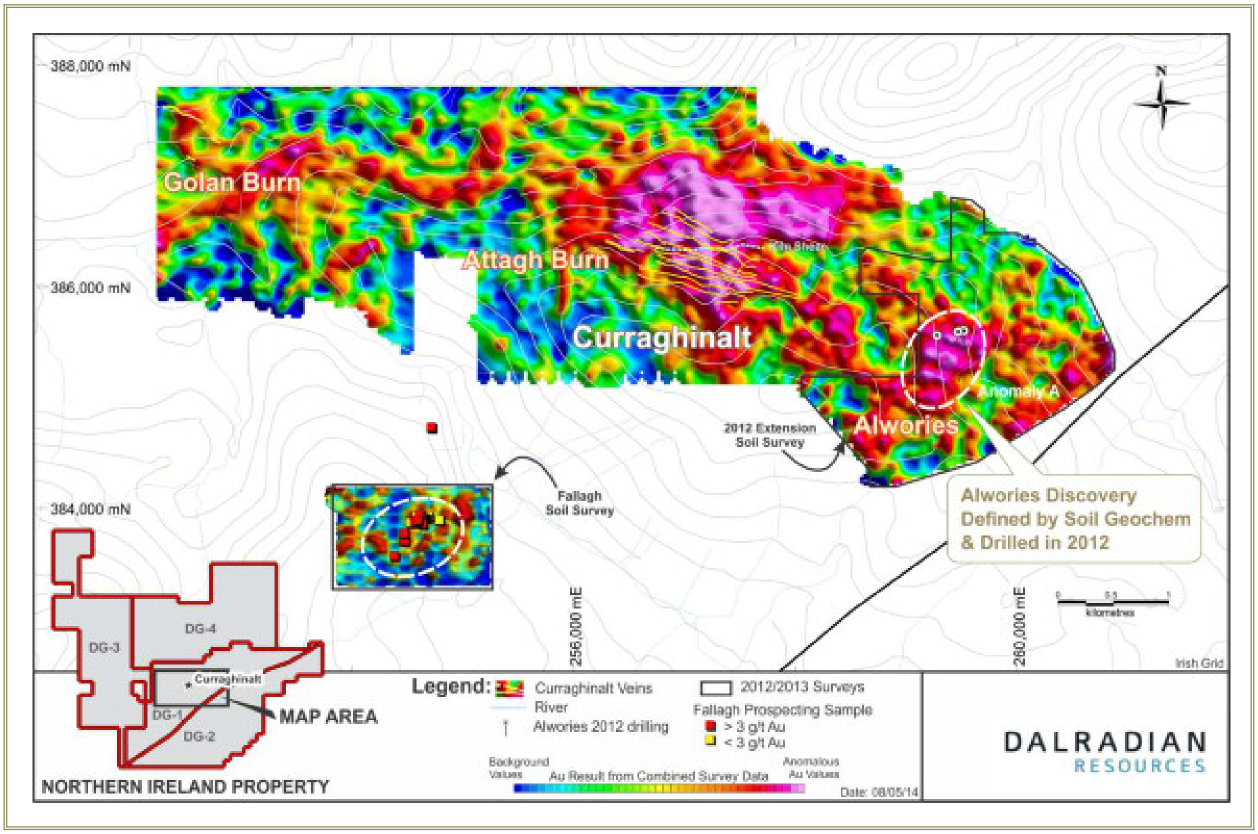


Figure 9-4: 2012 Curraghinalt East Soil Survey Grid



A second soil survey grid was initiated in the Fallagh area during late 2012 and into 2013 (Figure 9-4). Encouraging gold-in-soil results were followed up with Trench 13-FA-T01. Although bedrock mineralization was not intersected, a sample of quartz-carbonate vein in the till within the trench returned assay results of 14.65 g/t Au, and a float sample discovered near the trench returned 91.5 g/t Au.

Prospecting was carried out concurrently with the soil survey on DG1, and as a separate campaign on the other three licences. On DG1, 185 samples were collected, with 63 of them reporting above 1 g/t Au. On DG3, a total of 313 bedrock and float samples were collected, and a total of 30 samples returned results in excess of 0.5 g/t Au. The most anomalous results were returned from the known prospects of Bessy Bell, Pollan Burn/Gortin Glen and Rylagh/Erganagh. On DG4, A total of 168 bedrock and float samples were collected, but only three samples returned results in excess of 0.5 g/t Au.

In the Glenlark area, 52 panned concentrate samples were collected, following-up on three drill holes collared in the area.

Dalradian Resources' plan is to continue following up targets identified by prospecting with soil sampling surveys, and to then drill the most promising targets in the Dalradian Supergroup once the information from the geochemistry has been interpreted and drill targets identified.

10 DRILLING

10.1 Historical Drilling

Historical drilling on the Northern Ireland properties dates back to the 1980s, and was carried out in a number of campaigns until 2007. Most of the drilling was at the Curraghinalt deposit, including a number of underground holes completed while the Ennex adit was being excavated. Celtic Gold completed some drilling at the Golan Burn prospect. The remainder of the historical drilling targeted prospects on DG1 and DG2. Historical drill core sizes for surface drill holes at Curraghinalt were generally HQ (63.5 mm) and BQ (36.4 mm diameter). Occasionally HQ holes were reduced to NQ (47.6 mm) at depth. Most of the core from all historical operators is stored on site at Dalradian's core facility in Omagh, Northern Ireland, except for the Celtic Gold core from Golan Burn.

10.1.1 Curraghinalt

Ennex/Ulster Base Metals completed two drill programs at Curraghinalt. The early phase, between 1985 and 1989, included holes collared at surface as well as underground holes collared in the exploration adit and drifts. The second phase, between 1995 and 1997, consisted of holes collared at surface. Ennex drilled 187 holes from surface for a total of almost 18,000 m and 25 underground holes for a total of 634 m.

As operator of the Project, Tournigan completed 59 drill holes for a total of 12,565 m between 2003 and 2007. Each drill site was located with GPS, staked, and then photographed prior to the rig being moved in. The holes were down-hole surveyed by using an Encore Reflexit smart multi-shot tool instrument. Tully (2005) reports that a Tropari survey instrument was used for the first four holes in the Tournigan program. On completion of the drill hole, Celtic Surveys Ltd. surveyed the collar using the Irish National Grid to the nearest centimetre. All sites were cleared on completion and re-photographed.

Many of the historical holes drilled at Curraghinalt have been relogged and additionally sampled as part of the Historical Core Evaluation Program (HCEP) initiated by Dalradian in 2013. Part of this program involved verifying the accuracy of the historical collar, survey, and assay data in the Dalradian drill hole database.

10.1.2 Golan Burn

The Golan Burn prospect straddles the licence boundary between DG1 and DG3. Celtic Gold tested this prospect in 1987. In total, the company drilled 55 short diamond drill holes (3,623 m) and 69 short RC holes. The drill hole data for these holes remains to be verified and incorporated into Dalradian's drill hole database; however, historical results report the following intersection from auriferous quartz veins:

- Hole DG-13: 0.1 m at 17.6 g/t Au
- Hole DG-14: 2.8 m at 6.84 g/t Au
- Hole DG-18: 0.4 m at 9.3 g/t Au

- Hole DG-41: 0.6 m at 61.43 g/t Au.

10.1.3 Glenlark

Ennex drilled eight holes for a total of 441 m in 1988, and Tournigan completed seven holes for a total of 831 m in 2003. Localized trenching and drilling by Ennex intersected stratabound pyrite–sphalerite–galena–gold mineralization within a northerly-dipping but overturned sequence of mica-schists. Ennex drill hole DDH 90-200 intersected 1.93 m at 2.09 g/t Au and 3.7 g/t Ag, and 0.75 m at 8.19 g/t Au, 14.8 g/t Ag and 1.11% Pb+Zn (Ennex, 1988). The Tournigan drill program intersected 2.9 m of 2.8 g/t Au in Hole 03-GL-02.

10.1.4 Tyrone Volcanic Group (DG2) – Cashel Rock Area

Riofinex completed seven drill holes for a total of 920 m to test the coincident Cu–Pb–Zn soil and IP anomalies at the Cashel Rock target area. None of the holes encountered significant base metal mineralization. Subsequent to Riofinex's work, Ennex identified through geochemical surveying the Cashel Gold prospect, hosted in fine-grained, flow-banded silicified rhyolite, outcropping west of Leaghan (southwest of Cashel Rock). Ennex completed 453 m of trenching in order to define the surface extent of an auriferous silicified rhyolite and its associated stockwork. The trenching confirmed a mineralized zone within a horst type structure 100 m in length, averaging 15 m in width, truncated to the north and south by northwest-trending sinistral faults (Ennex, 1987). Trench 9 returned 10.8 m averaging 5.52 g/t Au in silicified rhyolites (Ennex, 1987).

Ennex then completed 23 holes for a total of 1,438 m near this prospect.

In general, long intercepts of very low anomalous gold values were intersected (up to 145 m grading 394 ppb Au), with some shorter, better grade zones (5.45 m at 4.3 g/t Au, 6.9 m at 1.3 g/t Au, and 3.63 m at 30.6 g/t Au) (Tully, 2005).

In addition, Ennex drilled 14 reconnaissance holes in the TVG rocks, including 4 at each Formil and Cashel Bridge, and 6 at Aghascrebagh-Crouk.

10.2 Dalradian Resources Drilling

10.2.1 Curraghinalt

Dalradian Resources began a new drilling campaign following the purchase of Dalradian Gold from Tournigan in late 2009. The drill program commenced in March 2010 with the mobilization of two Boyles 37 track-mounted drills with the capability of drilling to a depth of 500 m using HQ rods. These drills were supplied by Irish Drilling, who continued on the Project with three rigs until the spring of 2012. A second diamond drill contract was signed with Major Drilling, and two drills were mobilized to site in December 2010, with a third Major drill mobilized in August 2011. Between December 2010 and May 2013, approximately 48,954 m in 135 holes were drilled at the Curraghinalt deposit, while 28 holes (7,400 m) tested regional prospects.

The majority of the drilling with all rigs was HQ diameter. NQ diameter was sometimes used when a reduction from HQ was required due to ground conditions. Drilling was carried out on a single shift, 7 days/wk. Drilling was suspended in March 2012 and resumed in July 2012 until May 2013.

Prior to drilling, access agreements are negotiated with the owners of the surface lands and, generally, compensation is paid for use of the land and for disturbances caused by drilling. Before drilling, all drill sites are photographed. They are also photographed upon completion of a drill hole, and again after rehabilitation of the drill site.

Drill hole locations are marked out using a Global Positioning System (GPS), and later all drill collars are independently surveyed. Down-hole surveys are carried out on all drill holes using a Reflex multi-shot tool, with a reading taken every 6 m. This survey information is stored in the drill hole database. The vast majority of holes are drilled towards the south or the south–southwest in order to intercept the north–northeasterly dipping vein zones.

Figure 10-1 shows the location of all Dalradian drill holes at Curraghinalt.

All drill core is stored at the drill location during the shift, and brought to Gortin at the end of each day. The drill core is stored in a secure shed. For the initial part of the drill campaign, the drill core was logged, sampled, and photographed at the core facility in Gortin, and then moved to storage at one of two facilities located outside Gortin. Dalradian Resources currently has an office and core facility in Omagh, approximately 15 km from the Gortin facility. Presently, core is stored overnight in the secure facility in Gortin, and transported the following morning to Omagh, where it is logged, sampled, photographed, and stored.

For the 2011 resource estimate (Hennessey et al., 2012a), an additional 45 holes since the 2010 resource estimate were drilled totalling 20,561.9 m. The database was frozen at hole 11-CT-103; however, hole 11-CT-101 was still in progress, and not included in the 2011 estimate. This drilling was a combination of holes to the east and west of the previous resource and testing the structures at depth. In addition to the 45 holes completed, five holes, totalling 318.4 m, were drilled but were not included in the resource estimate, as these holes were abandoned at various depths due to drilling difficulties.

Figure 10-1: Curraghinalt Deposit, Dalradian Drill Hole Locations (May 2013)

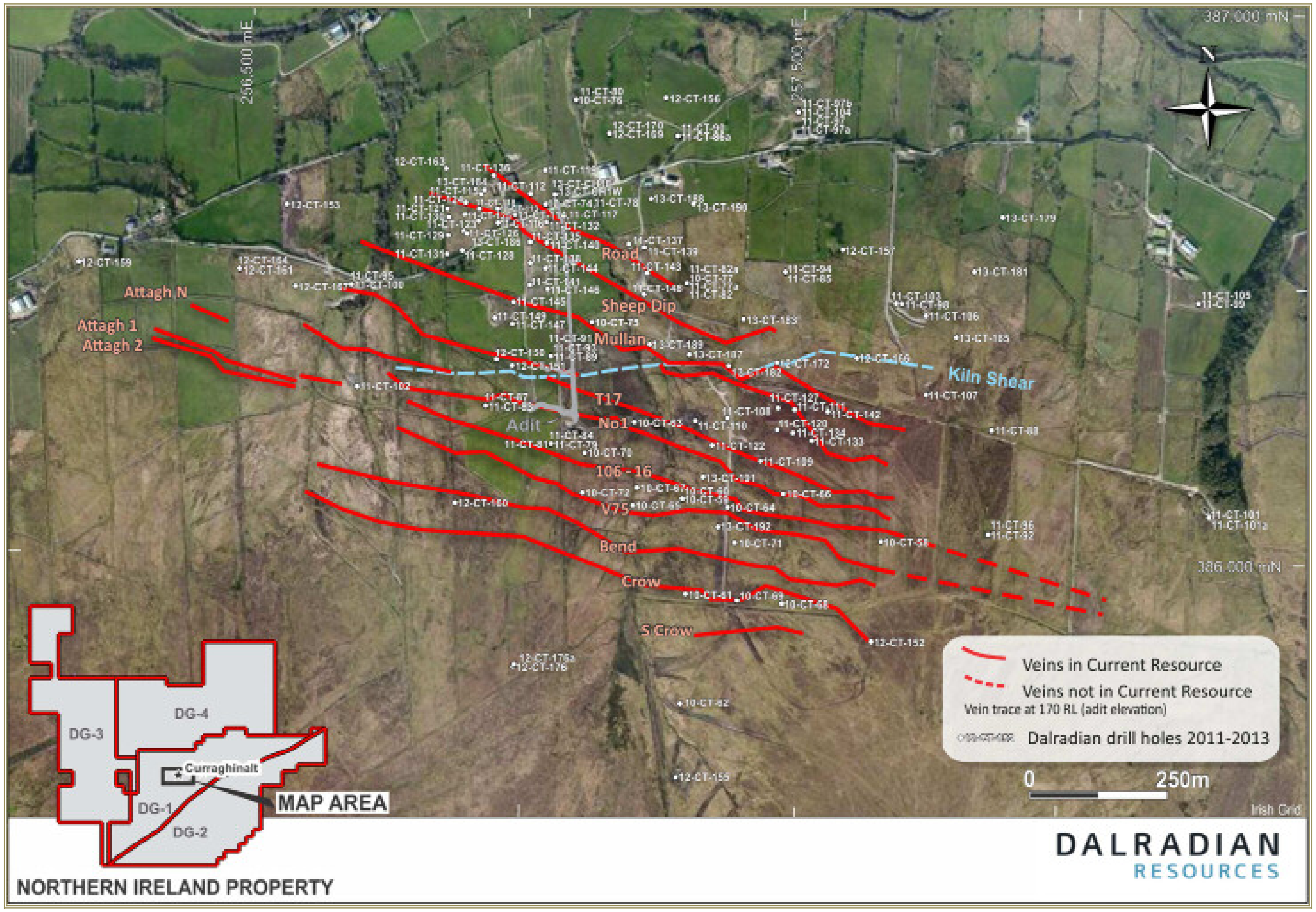
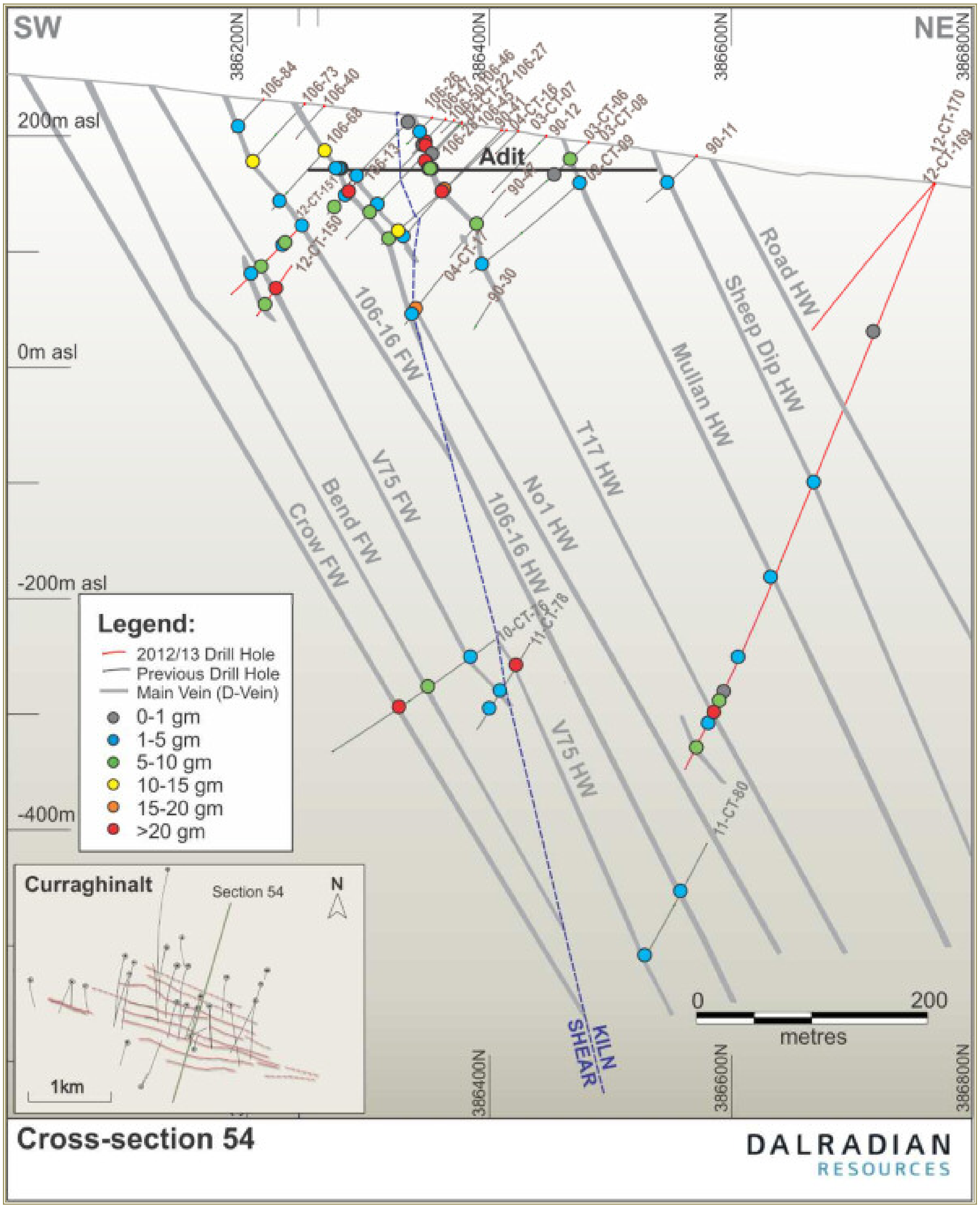


Figure 10-2: Oblique Section at approximately 257055 E, Looking West



After freezing the database for the 2011 mineral resource estimate, diamond drilling continued at Curraghinalt. In total, an additional 28,353 m were drilled in 83 holes, and were used in the current resource estimate (Section 14.1). The database was frozen at drill hole 13-CT-192. Additional sampling of 100 historical drill holes (12,346 m) was also conducted. The majority of these drill holes can be considered as infill holes, and significant results are shown in Table 10-1.

Table 10-1: Significant recent Drill Hole Results at Curraghinalt

Hole ID	From (m)	To (m)	Core Length (m)	Au Grade (g/t)	Vein
12-CT-150	200.90	201.70	0.80	58.83	Secondary Vein
12-CT-150	220.10	221.30	1.20	10.73	Bend
12-CT-150	305.50	308.80	3.35	13.18	Crow
12-CT-151	131.00	132.10	1.11	21.06	106-16
12-CT-151	279.10	280.00	0.92	16.18	Crow
12-CT-152	297.60	298.20	0.64	8.97	New Vein
12-CT-156	364.16	364.72	0.56	45.18	Sheep Dip
12-CT-156	442.14	446.14	4.00	7.38	"C" Vein
12-CT-156	534.00	534.75	0.75	11.77	Mullan
12-CT-156	586.70	590.67	3.97	38.00	T17
including	588.77	589.07	0.30	400.00	
12-CT-156	671.57	675.98	4.41	1.29	"C" Vein
12-CT-157	255.68	256.18	0.50	16.46	Mullan
12-CT-157	359.31	361.65	2.34	47.94	T17
12-CT-157	490.33	494.50	0.12	61.20	"C" Vein
12-CT-159	238.50	240.97	2.47	5.19	Attagh Burn
12-CT-160	31.62	32.00	0.38	16.05	Crow
12-CT-161	245.74	246.34	0.60	10.05	Attagh Burn
12-CT-161	256.57	258.27	1.70	17.55	Attagh Burn
12-CT-163	44.05	45.78	1.73	3.25	Sheep Dip
12-CT-163	263.12	263.28	0.10	60.60	T17
12-CT-163	412.12	413.62	1.50	13.78	106-16
12-CT-163	494.21	497.47	3.26	6.34	Bend
12-CT-163	546.08	549.31	3.24	4.89	Crow
12-CT-164	107.25	107.63	0.38	14.32	Attagh Burn
12-CT-164	127.88	129.15	1.27	5.28	Attagh Burn
12-CT-164	186.40	187.00	0.60	13.35	Attagh Burn
12-CT-166	20.62	21.23	0.61	10.70	"C" Veins
12-CT-166	142.60	142.86	0.26	5.37	Mullan
12-CT-166	199.80	201.12	1.32	17.61	T17
including	200.16	200.60	0.44	46.89	
12-CT-166	297.52	297.76	0.24	14.05	106-16
12-CT-166	316.51	316.81	0.30	28.30	V75
12-CT-166	379.09	379.40	0.31	12.55	Bend
12-CT-166	415.24	415.50	0.26	7.38	"C" Vein

Hole ID	From (m)	To (m)	Core Length (m)	Au Grade (g/t)	Vein
12-CT-166	444.50	444.67	0.17	12.00	Crow
12-CT-166	447.87	448.00	0.13	5.07	"C" Vein
12-CT-166	450.22	450.48	0.26	43.10	Crow
12-CT-167	151.77	152.12	0.35	12.47	Attagh Burn
12-CT-167	201.93	204.40	2.47	18.99	Attagh Burn
12-CT-167	203.90	204.10	0.20	158.00	
12-CT-167	221.92	222.19	0.27	15.35	Attagh Burn
12-CT-170	298.38	298.68	0.30	18.65	Not classified
12-CT-170	395.08	396.70	1.62	7.82	Not classified
12-CT-170	405.12	405.43	0.31	14.15	Not classified
12-CT-170	484.78	485.24	0.46	11.25	Not classified
12-CT-170	495.60	496.36	0.76	33.60	Not classified
12-CT-170	507.53	507.64	0.11	9.45	Not classified
12-CT-170	530.13	530.80	0.67	13.34	Not classified
12-CT-170	593.59	593.88	0.29	13.40	Not classified
12-CT-170	597.65	597.81	0.16	7.08	Not classified
12-CT-170	692.36	693.36	1.00	13.71	Not classified
12-CT-172	48.12	49.27	1.15	7.19	C-veins
12-CT-172	107.60	108.23	0.63	25.56	Mullan
12-CT-172	120.90	122.50	1.60	12.81	T-17 zone
12-CT-172	157.78	157.94	0.16	41.9	T-17 zone
12-CT-174	215.55	215.67	0.12	82.9	Alwories
12-CT-174	220.70	221.43	0.73	32.81	Alwories
12-CT-175	873.47	874.45	0.98	6.84	Not classified
12-CT-175	1107.08	1108.62	1.54	30.13	Not classified
12-CT-176a	256.75	262.10	5.35	6.60	Crow
12-CT-176a	364.10	365.25	1.15	15.16	Crow
13-CT-181	289.86	290.05	0.19	40.9	Mullan
13-CT-181	464.48	466.12	1.64	6.21	106-16
13-CT-181	470.46	471.34	0.88	10.94	106-16
13-CT-181	729.46	731.25	1.79	20.66	Unnamed footwall vein
13-CT-181	844.08	845.02	0.94	13.64	Unnamed footwall vein
13-CT-182	33.71	35.90	2.19	8.8	Mullan
13-CT-182	40.93	41.09	0.16	69.3	Mullan
13-CT-182	191.52	192.39	0.87	28.44	No. 1
13-CT-182	206.36	206.66	0.30	19.3	
13-CT-182	218.55	219.76	1.21	41.22	106-16
13-CT-182	324.16	324.91	0.75	11.19	Bend
13-CT-183	103.95	104.05	0.10	31.5	Mullan
13-CT-183	251.28	253.14	1.86	8.93	No. 1
13-CT-183	265.40	267.42	2.02	5.67	No. 1
13-CT-184	392.96	393.72	0.76	24.46	106-16

Hole ID	From (m)	To (m)	Core Length (m)	Au Grade (g/t)	Vein
13-CT-184	499.56	499.88	0.32	17.15	Bend
13-CT-186	460.86	461.15	0.29	17.85	Bend
13-CT-186	490.45	490.95	0.50	20.57	Crow
13-CT-187	41.32	46.20	4.88	8.61	Mullan
including	41.32	43.40	2.08	19.92	
13-CT-187	69.79	70.10	0.31	43.22	C-veins
13-CT-187	133.57	134.88	1.31	6.39	T-17
13-CT-187	158.13	160.74	2.61	8.81	C-veins
13-CT-187	174.25	174.42	0.17	22.2	No. 1
13-CT-187	226.13	228.08	1.95	12.25	106-16
13-CT-187	276.43	276.58	0.15	38	Bend
13-CT-187	301.77	303.45	1.58	10.7	Crow
including	302.29	302.62	0.33	40.63	
13-CT-187	327.53	328.32	0.79	12.32	Unnamed footwall vein
13-CT-188	321.80	322.16	0.36	24.38	T17
13-CT-188	345.58	350.45	4.87	8.35	No. 1
including	350.03	350.21	0.18	54.7	"C" Vein
including	350.21	350.45	0.24	85.5	"C" Vein
13-CT-188	380.06	382.39	2.33	6.26	No. 1
including	380.06	381.46	1.40	9.66	
13-CT-188	552.93	554.82	1.89	6.61	Bend
13-CT-189	129.72	132.20	2.48	21.33	"C" Veins No. 1 zone
including	129.93	130.07	0.14	197.5	"C" Vein
including	132.00	132.20	0.20	124.5	"C" Vein
13-CT-189	154.11	156.18	2.07	7.97	No. 1
including	154.76	155.00	0.24	50.5	
13-CT-189	217.64	218.20	0.56	16.49	106-16
13-CT-190	190.81	192.69	1.88	5.54	"C" Vein Array
including	191.65	191.87	0.22	34.6	"C" Vein
13-CT-190	284.46	285.90	1.44	9.9	Mullan
13-CT-190	333.00	333.81	0.81	28.16	No. 1
13-CT-190	359.82	361.35	1.53	17.14	106-16 HW
13-CT-190	410.63	413.42	2.79	8.58	106-16 FW
including	410.63	411.00	0.37	58.61	
13-CT-190	515.35	517.81	2.46	8.99	Bend
including	515.35	516.52	1.17	17.14	
13-CT-190	555.48	555.64	0.16	59.7	Crow

10.2.2 Regional Drilling

Twenty-five holes have been drilled by Dalradian at regional targets since 2010.

In 2011, four holes were drilled on DG2, two at the Broughderg prospect, one at Cashel Rock, and one at Tullybrick. 11-CR-01 intersected the auriferous horizon down dip from the earlier Ennex era drilling, with 0.39 m at 9.44 g/t Au. Additionally, one hole was drilled at Scotchtown on DG1 in 2011.

In 2012, 12 drill holes were drilled at the Alwories Prospect following up a 800 m x 500 m gold-in-soil target. Alwories is approximately 1,700 m along strike from Curraghinalt. Significant results are presented in Table 10-2.

Table 10-2: Significant Drill Results from Alwories Prospect

Hole ID	From (m)	To (m)	Core Length (m)	Au Grade (g/t)
12-CT-162	123.50	123.60	0.10	3.90
12-CT-162	139.75	140.43	0.68	4.70
12-CT-162	165.00	165.18	0.18	1.10
12-CT-162	186.00	188.22	2.22	14.10
12-CT-162	229.08	230.55	1.47	23.60
12-CT-162	239.70	239.85	0.15	4.40
12-CT-162	279.28	279.45	0.17	1.40
12-CT-162	303.50	304.60	1.10	7.60
12-CT-173	236.50	241.72	4.72	14.82
including	237.72	239.94	2.22	0.07
12-CT-173	236.50	237.72	1.22	25.14
12-CT-173	239.94	241.22	1.28	30.58
12-CT-173	416.22	416.32	0.10	62.9
12-CT-173	541.06	541.35	0.29	78.5
12-CT-174	196.15	196.35	0.20	23.2
12-CT-174	215.55	215.67	0.12	82.9
12-CT-174	220.70	221.43	0.73	32.81

Additionally, three holes at Glenlark, one at Scotchtown, three at Glenmacoffer, and one at Cashel Rock were drilled.

10.3 Core Logging

Prior to sampling, the drill core is washed and logged by the site geologist. After September 2012, all core is aligned during the washing stage by geological technicians. This process results in a recovery log, and core annotated with metre marks. Drill core was logged to record lithology, structure, alteration, rock quality designation (RQD), and mineralization.

Core logging was carried out by geologists assigned to the Project by Aurum (until July 2011) and then by Dalradian Resources staff (after July 2011). Core logging procedures have been continually modified and improved upon since this time.

The length of core brought from the drill site is confirmed against the drill report. Logging commences with the calculation of core recovery. Geology is then marked, with particular attention paid to the identification of mineralized zones. Structural data collected include the orientation of mineralized quartz veins against the longitudinal axis of the core. Alteration and geotechnical information (RQD) are noted. In 2012, the company also started collecting magnetic susceptibility measurements, as well as near infrared spectral data at 3 m intervals.

In general, core recoveries vary from 90% to 97%, and RQDs are generally in the high 80s or better, except in certain wall rock zones (pelites and faults). In general, the core size drilled is HQ, which results in excellent recoveries, and thus the samples are considered to be representative of the veins. Some drill holes are reduced to NQ to allow for completion of a hole. The overall recovery difference between NQ and HQ core is quite similar.

Logging was carried out on paper, with data then being input into a series of digital data logs that were entered into the computer database with other drill hole data logs. In 2012, the company implemented a Datashed database to house all drill hole and surficial data. In July 2012, geologists began using LogChief software to record logging observations and sample intervals.

Tournigan initiated photography of all vein material in August 2007 Presently, Dalradian photographs all core, and the photographic record, along with copies of the paper drill logs, geotechnical logs, and assay certificates, are stored in a separate file for each drill hole. As a product of the HCEP program, about half of the historical core has been photographed and photography continues. Figure 10-3 shows an example photograph of mineralized drill core.

Figure 10-3: Quartz-Sulphide Vein in Drill Core

The quality assurance/quality control (QA/QC) procedures for the drilling and logging processes are shown in Table 10-3. This table is based on information provided by Dalradian Resources. TMAC's observation of core logging during the 2013 visit showed it to be carried out to industry standards.

Table 10-3: Drill Set-up and Core Logging Controls

Procedure	Comments
Drill Rig Set-up	Drillers to set-up rig
	Check location of small ditch to be dug from the pierce point for the drill return water
	Returns are directed to a large excavated sump or a series or two sumps in series, where the returns settle out. Clear water is then pumped from the sump back to the drill rig for reuse.
	Drill rig to be checked daily (early afternoon); vein material, in core box, to be taken directly to core logging shed
	Drillers to deliver drill core to the logging shed each evening unless collected by the geologist
Drill Core Logging	Approximate depths to be reported as core comes in for ongoing section correlation
	Sequence of core boxes to be checked for correct order; boxes to be checked for correct labelling
	Core markers to be checked every 3 m
	Core loss to be checked if markers are not evenly spaced
	Orientate drill core so that the general schistosity is consistent
	Complete RQD on every 3 m run down the core using detailed recording sheet
	RQD, joint frequency and orientation, roughness, joint infill, fractures, faults, etc. to be recorded
	Log lithology using pelite/semipelite/psammite system
	Record host rock mineralogy/alteration
	Record nature of hanging wall and footwall contacts
	Log veins; measure hanging wall and footwall contact angles if natural and not faulted
	Record mineralogy in detail; identify zoning/banding
	Separate vein into areas of similar mineralogy and percentage sulphides
	Log structure and alteration
	Enter data into digital data book on an ongoing basis
Surveying Drill Hole	Down-hole surveys are carried out with Reflex multi-shot tool. Two digital files are produced and converted into Microsoft Excel file format using proprietary software. These files are then reviewed by the geologists for errors, and then entered in the drill hole database.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 2010 to mid-2012

The Dalradian Resources sampling approach up to mid-2012 was as follows.

All sulphidic quartz vein material intersected in the drill core is sampled.

The geologist logging the hole selects core sample intervals. The maximum sample interval is currently 0.30 m in mineralized veins with a 0.10 m minimum width. The 0.30 m sample length was selected so that the entire sample can be pulverized at the laboratory, eliminating the need to split the crushed sample. Samples in weakly mineralized or unmineralized wall rock are usually significantly larger (i.e., up to 1 m).

Sample intervals tend to be split by the geology and or mineralization noted (i.e., sulphide-rich zones in veins are separated from quartz-rich zones), and wall rock samples are generally only taken if mineralized (i.e., stockwork veins are present).

During the logging process, the geologist marks the intervals to be sampled on both the core and the core box. The core is then oriented along the line of symmetry and sawn with a diamond-tipped saw. This process allows an accurate depth measurement to be assigned to each sample. Once an interval is cut, both halves of the core are placed back in the box, and the next interval is sawn.

A sample ticket book is used to record sample intervals and a brief mineralogical description. Each sample is assigned an identifying number as printed on each ticket. A plastic sample bag is then clearly numbered with the matching ticket number placed inside with the split core. A duplicate ticket is stapled to the core box at the start of the sample interval above the remaining half core in the core box.

Once bagged, all the samples split during each work shift are placed on the floor in numerical order. When all the individual samples have been counted, and the tag numbers verified, the samples are placed into large plastic bags. Each larger bag of samples is sealed with a wire tag and an address label. These bags are stored in the locked core shed until shipment to the laboratory by a commercial courier service.

All drill core is brought from the drills at the end of shift, and is stored in a rented lockable storage facility on the main street of Gortin, across the road from the field office. Until November 2011, core was sawn at a nearby farm, and unmineralized core boxes were stored in a concrete-floored barn at that location. Mineralized sample boxes were returned to the industrial building where they were kept under lock and key. In November 2011, Dalradian Resources leased a new office and core facility in nearby Omagh. The drill core is brought from the drills at the end of the shift and stored overnight in the secure facility on the main street in Gortin, as previously done. The following day, the core is brought to the Omagh facility, where it is logged, sawn, and stored. The Omagh facility is located in a secure fenced area, which is locked in the evenings and on weekends. The core storage and logging facility is kept locked when unoccupied. Unshipped samples are also stored at this location.

All samples were analyzed at OMAC Laboratories (OMAC), Athenry Road, Loughrea, County Galway, Republic of Ireland. In July 2011, the ALS Group acquired OMAC.

11.2 Mid-2012 to 2013

In September 2012, Dalradian revised their logging and sampling procedures.

All core is brought to the Omagh logging facility, where it is laid out and box numbers are verified. The core is then aligned such that the foliation trends from the upper left to the lower right. During the alignment process, the core is washed, visually inspected for mineralization, out of sequence pieces are identified, and meter marks verified. The core is then marked with blue crayon every 1 m to aid in recovery, RQD, and logging measurements.

Mineralized vein zones (D veins and their associated C veins) are sampled, with maximum 0.30 m and minimum 0.10 m sample lengths. Weakly mineralized zones are sampled at 1 m intervals. Sample intervals are demarcated with yellow crayon across the top of the aligned core. Sample "from" and "to" are written on the core box with permanent black marker and entered into the sample ticket book. Sample tags are stapled to the core box at the start of every sample, and QA/QC sample tags added as required.

Cutting lines for sawing are drawn in yellow along the top of core. When cutting, the core is turned 90° to this cut line before entering the saw. Once the core is split, the left hand side of the core is sent to be sampled, and the right hand side of the core with the cut line is kept in the box. Where core is oriented, the sample line is drawn above the orientation line so that the line remains preserved in the sample left in the box.

Where samples are unevenly mineralized, the geologist marks a specific cut line in yellow on the core, allowing representative mineralization to be captured in assaying and preserved in the remaining half of the core. The core boxes are marked in red where these "direct cut line" samples are and blue where orientation stubs occur, to make technicians aware to cut along the specific line.

Unmineralized zones may be left unsampled, based on review by the geologist.

Blank Material (Blanks) and Standard Reference Material (SRM) are inserted into the sample stream. Blanks are inserted once for every 20 samples and also after each high grade sample (>25% pyrite). Standards are also inserted once every 20 samples.

After cutting, samples are placed in a plastic bag and stapled shut. Sample tags are placed inside the bag, and sample numbers are written on the outside of the bag. Samples are prepared for shipping, and shipped to ALS Ltd, Loughrea (ALS Loughrea) in the Republic of Ireland. ALS Loughrea is accredited by the Irish National Accreditation Board (INAB) to undertake testing, including for ores and minerals (INAB P9 703), as detailed in the Schedule bearing the Registration Number 173T, in compliance with the International Standard ISO/IEC 17025:2005 2nd Edition "General Requirements for the Competence of Testing and Calibration Laboratories".

Sample pulps and rejects are returned to Dalradian's facility in Omagh.

11.3 Quality Control/Quality Assurance

Ennex, Tournigan and Dalradian Resources have conducted QA/QC programs on the Curraghinalt project. Details of the Ennex and Tournigan program are not outlined here but can be reviewed in prior technical reports (Hennessey et al., 2012a; Hennessey et al., 2012b).

11.3.1 Dalradian Resources QA/QC Program

The QA/QC program was under the supervision of Aurum from the restart of drilling in 2010 until July 2011, at which time Dalradian Resources staff assumed supervision.

The Dalradian Resources sampling and QA/QC procedures can be described as follows:

- Core sample intervals are selected by the geologist as described above.
- Sample intervals are generally split by geology as described in Section 11.1.
- Samples are cut by diamond saw down the ellipse of the vein. The same (left) half is always taken for the sample, and the other (right) half is left in the core tray for reference. Cross-cuts at sample extents are along the contact of the vein and wall rock, where this is possible on both vein contacts (with the interval measured to the centre of the core axis) or, if this is impossible, perpendicular across the core axis.
- To check for contamination during sample preparation at the laboratory, blanks are inserted where the geologist estimates the sample will be high grade. These are inserted so that approximately 10% of the submitted samples are blanks. The blank material is chunky (fist-sized) limestone from a local source that has been used extensively enough so that there is confidence that the material contains negligible amounts of gold.
- Certified SRMs (or “standards”) are inserted into the sample stream so that they make up approximately 10% of the samples submitted. These are 50 g sachets of pulverized, homogenized rock with gold and sulphide added. The accepted gold grade of each standard is determined by round-robin assaying at multiple laboratories. Up to five or six different standards are in circulation at any one time, and a current gold grade range from 0.835 to 30.04 g/t Au. The selection has been chosen to reflect the typical range of gold assays, as it is the accuracy within this grade range that is of interest. The inserted SRM is randomly selected.
- Before being shipped, the samples are sealed in bags, and an additional, signed and dated, tamper-proof security tag is placed around the wire tie. All samples and core that will be sampled is stored in a locked core shed.
- Specific gravity (SG) measurements are attempted on some vein samples, using the water submersion method.

Dalradian Resources' QA/QC program consists of assaying of Blanks and SRMs and a duplicate assay program at ALS Loughrea. No sample re-assaying has been conducted at a second reference laboratory. The details of samples assayed as part of Dalradian Resources' QA/QC program (as of October 2013) are given in Table 11-1. Table 11-2 summarizes the SRMs used by Dalradian Resources.

Table 11-1: Details of Dalradian Resources' QA/QC Samples

Type of QA/QC Samples	Number of Samples
Blanks	3283
SRMs	3217

Table 11-2: Dalradian Resources Certified SRM for Gold - Post January 2010

Low Grade Au (0-1ppm)			Medium Grade Au (1-5ppm)			High Grade Au (>5ppm)		
Standard ID	**Exp. Value**	**Inserts**	**Standard ID**	**Exp. Value**	**Inserts**	**Standard ID**	**Exp. Value**	**Inserts**
SF67	0.835	299	SG66	1.086	200	SL46	5.867	25
SF57	0.848	103	SH35	1.323	25	SL51	5.909	93
Total		**402**	**SH41**	**1.344**	**89**	**SL61**	**5.931**	**262**
			SH65	1.348	100	SN38	8.573	25
			SH55	1.375	81	SN60	8.595	563
			SJ63	2.632	132	SN50	8.685	90
			SJ53	2.637	172	CDN-GS-10A	9.78	20
			SJ39	2.641	24	SP59	18.12	376
			CDN-GS-5C	4.74	20	SP37	18.14	108
			Total		**843**	**SP49**	**18.34**	**104**
						SQ36	30.04	108
						SQ48	30.25	198
						Total		**1972**

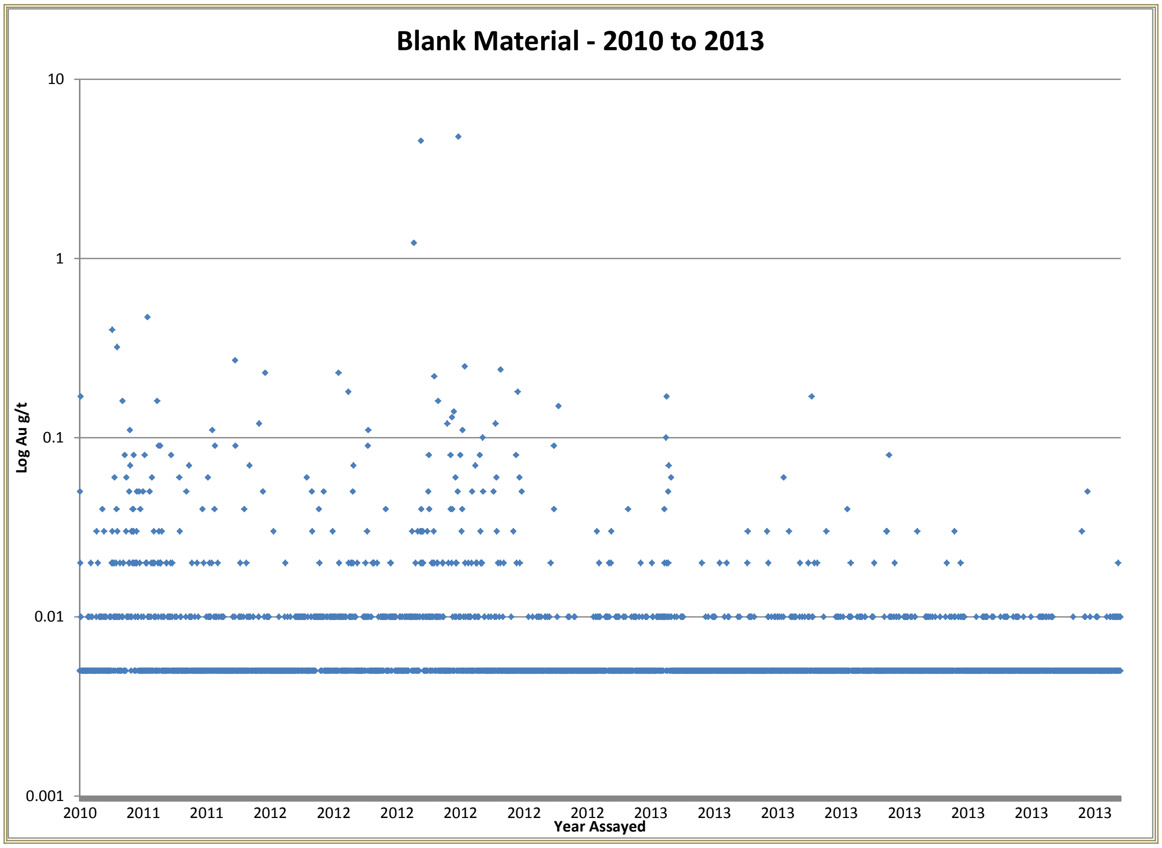
As part of their review of the QC data, TMAC and prior consultants (Hennessey, 2012a; Hennessey et al., 2012b) conducted a number of checks to assess precision, accuracy, and bias. These are described in the following sections.

11.3.2 Blank Material Results

Blank samples form part of all analytical QC procedures, and are used to assess the accuracy of the assay results and to identify any possible contamination and mixing during analysis. A total of 3,727 blanks were inserted along with other samples by Dalradian Resources. There were four blanks that had considerable error (greater than 1.0 g/t Au), and these were designated as outliers and may be attributable to a sample mix-up. An additional 40 samples assayed at values greater than 0.10 g/t Au. All the results were plotted in Figure 11-1.

The accuracy of the blanks is considered to be within an acceptable limit of error. Also, it is notable on Figure 11-1 that there was improvement in the Blanks during 2013 with fewer assayed over 0.10 g/t Au.

Figure 11-1: Blank Material Control Chart



11.3.3 Reference Standard Sample Results

Pre-Dalradian Resources Results

A total of six different types of reference material were used to check the accuracy of assay results at the Project. All standards were acquired from CDN Resource Laboratories Ltd. (CDN Resource). The average value for each standard, along with its tolerance, is given in Table 11-3. A total of 134 standard samples were submitted along with the drill core samples. The average value received for each of the standards is given in Table 11-4. TMAC has concluded that although outliers are present there is no significant or systematic bias identified by the SRMs.

Table 11-3: Certified Value of CDN Resource Standard Samples

	CDN-GS-11	CDN-GS-5A	CDN-GS-14	CDN-GS-12	CDN-GS-15	CDN-GS-20
Mean	3.4	5.1	7.47	9.98	15.31	20.6
±2 SD	0.27	0.27	0.31	0.37	0.58	0.67
SD % of Mean	8%	5%	4%	4%	4%	3%
Number of Samples	84	84	84	84	84	84

Table 11-4: Analyzed Value of CDN Resource Standard Samples

	CDN-GS-11	CDN-GS-5A	CDN-GS-14	CDN-GS-12	CDN-GS-15	CDN-GS-20
Mean	3.45	5.05	7.56	10.16	15.30	20.61
±2 SD	0.15	0.29	0.55	0.88	1.14	1.06
SD % of Mean	4%	6%	7%	9%	7%	5%
Number of Samples	19	20	29	20	27	19
Bias	1%	-1%	1%	2%	0%	0%

11.3.4 Dalradian Resources SRM

A total of 23 different types of reference materials were used to check the accuracy of assays. Twenty-one of the SRMs were acquired from Rocklabs Ltd. (New Zealand) and the remaining two from CDN Resources (Table 11-5).

Table 11-5: Certified and Analyzed Value of Rocklabs SRMs

SRM	Rocklabs SRM		ALS Analysis			Bias %
	Expected Value (Au g/t)	Expected SD	Sample Mean (Au g/t)	Sample SD	Sample No.	
SF57	0.85	0.03	0.82	0.03	103	-3.2
SF67	0.84	0.02	0.83	0.02	299	-0.4
SG66	1.09	0.03	1.09	0.02	200	0.2
SH35	1.32	0.04	1.28	0.04	25	-3.6
SH41	1.34	0.04	1.30	0.04	89	-3.4
SH55	1.38	0.05	1.36	0.04	81	-1.2
SH65	1.35	0.03	1.34	0.03	100	-0.9
SJ39	2.64	0.08	2.55	0.10	24	-3.3
SJ53	2.64	0.05	2.62	0.06	172	-0.6
SJ63	2.63	0.06	2.66	0.05	132	1.1
SL46	5.87	0.17	5.85	0.21	25	-0.3
SL51	5.91	0.14	5.89	0.18	93	-0.4
SL61	5.93	0.18	5.93	0.11	262	0.0
SN38	8.57	0.16	8.51	0.19	25	-0.7

SRM	Rocklabs SRM		ALS Analysis			Bias %
SN50	8.69	0.18	8.53	0.22	90	-1.8
SN60	8.60	0.22	8.58	0.16	563	-0.2
SP37	18.14	0.38	17.98	0.53	108	-0.9
SP49	18.34	0.34	18.23	0.40	104	-0.6
SP59	18.12	0.36	18.13	0.33	376	0.1
SQ36	30.04	0.60	29.55	0.84	108	-1.6
SQ48	30.25	0.51	30.31	0.69	198	0.2

It may be seen that there are no significant bias or accuracy issues in analyzing the standards. This indicates that the assay results would not have any significant bias. However, generally the SRM average was slightly lower than expected. The overall average difference was -0.5%. All the results for analyzed SRMs from Rocklabs, along with their certified values are plotted in Figure 11-2 through Figure 11-22.

There are 104 outliers identified by Dalradian Resources from the assay results of the SRMs that are beyond the acceptable limit of two SDs. No systematic bias was attributable to these outliers. Dalradian Resources regularly reviews the outliers and, if necessary, re-assays the complete sample batch.

Overall, TMAC considers that the Dalradian Resources results are within the acceptable range of error.

Figure 11-2: SRM SJ57

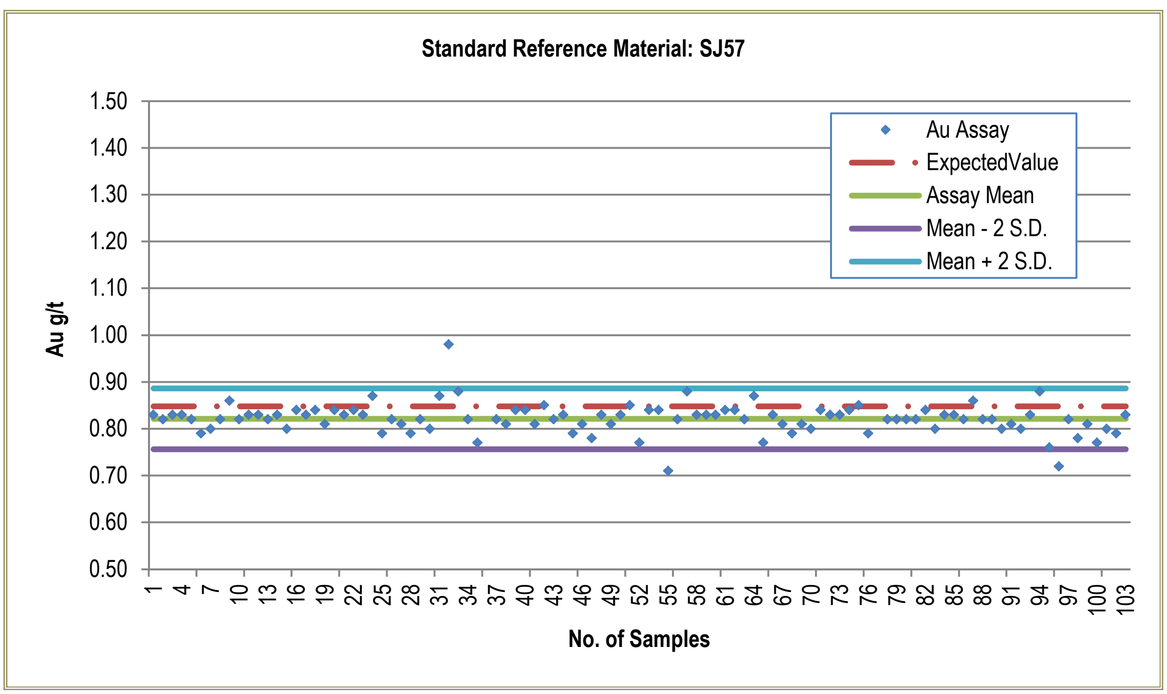


Figure 11-3: SRM SJ67

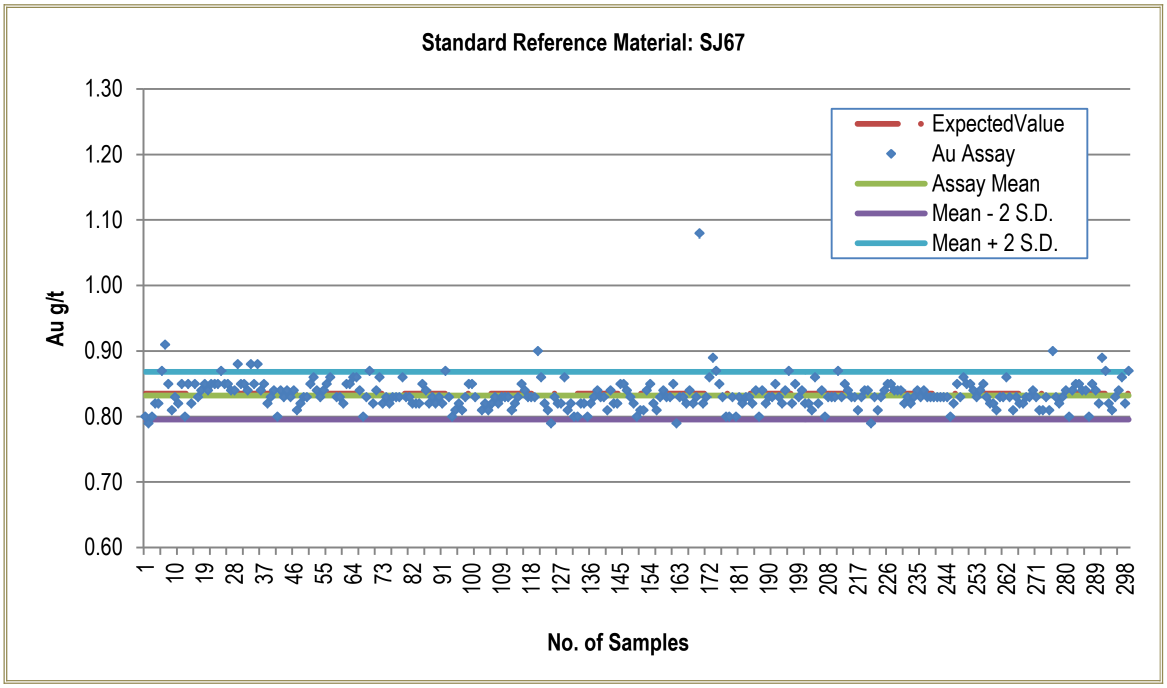


Figure 11-4: SRM SG66

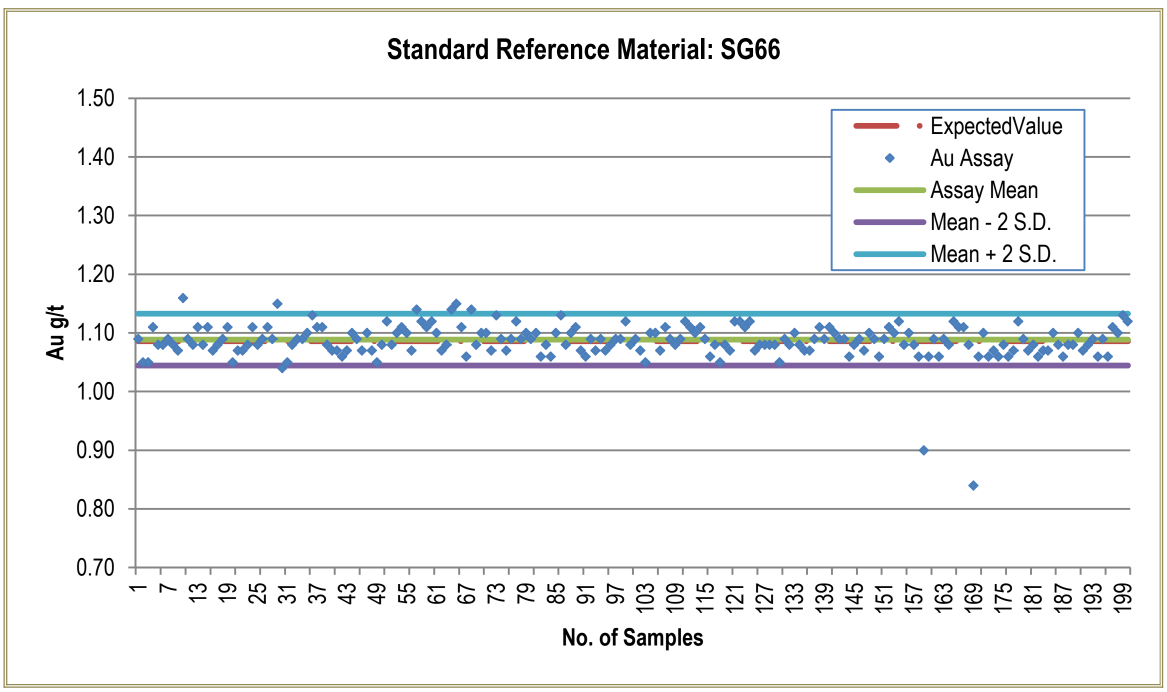


Figure 11-5: SRM SH35

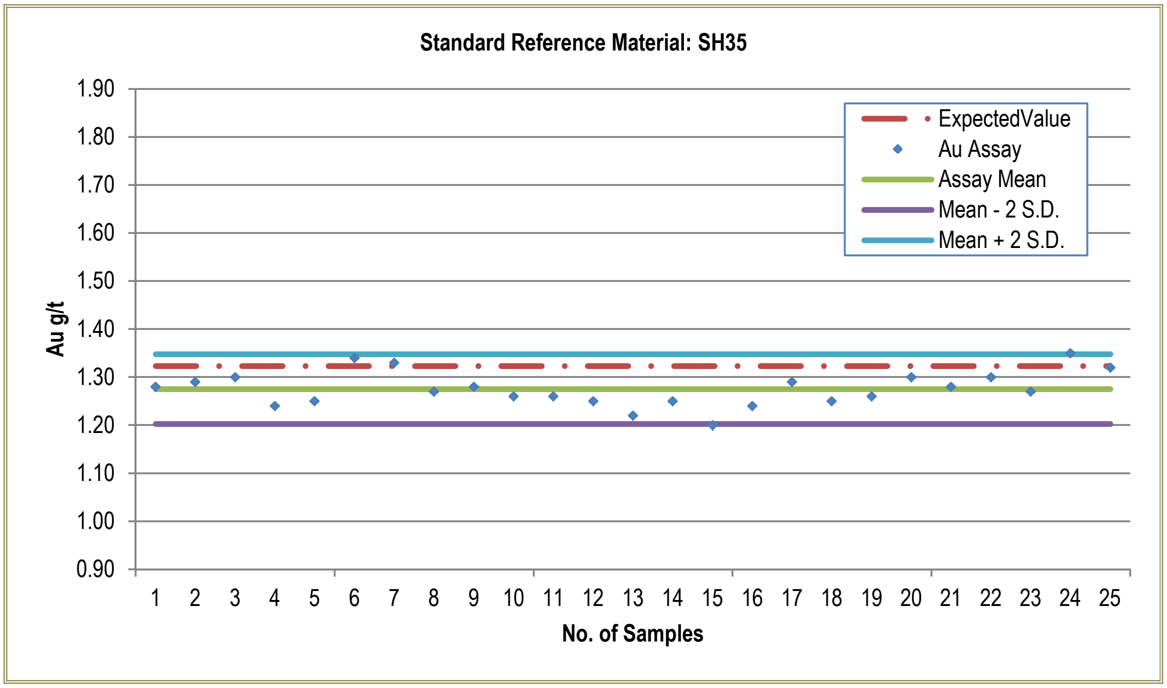


Figure 11-6: SRM SH41

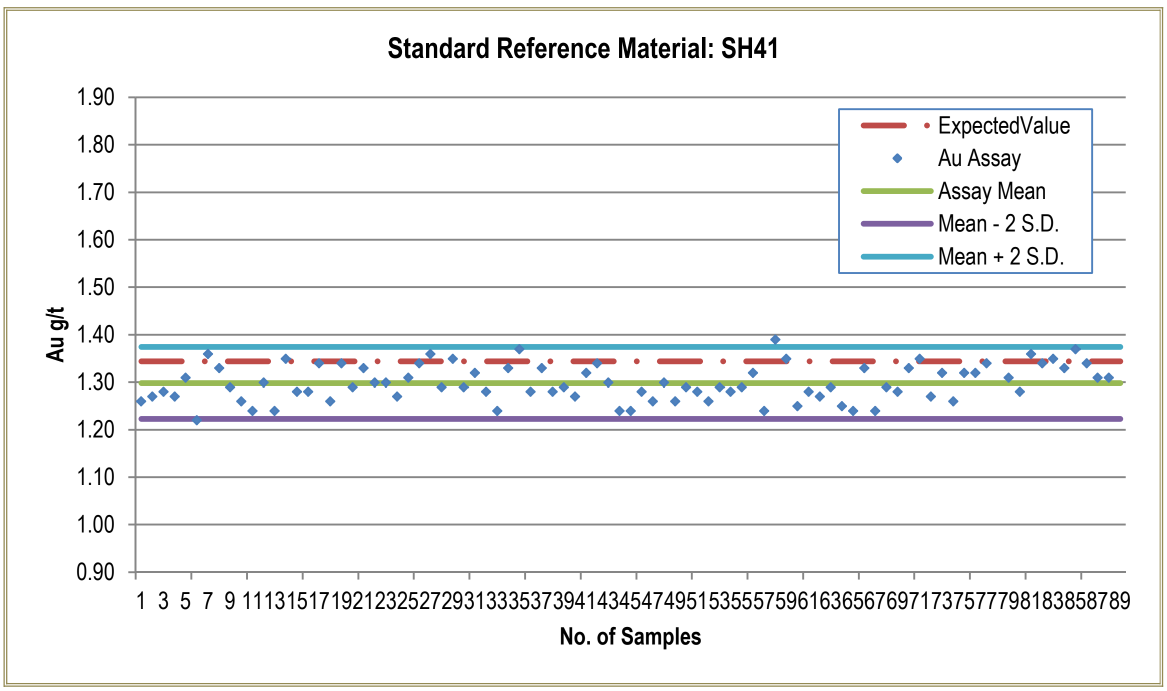


Figure 11-7: SRM SH55

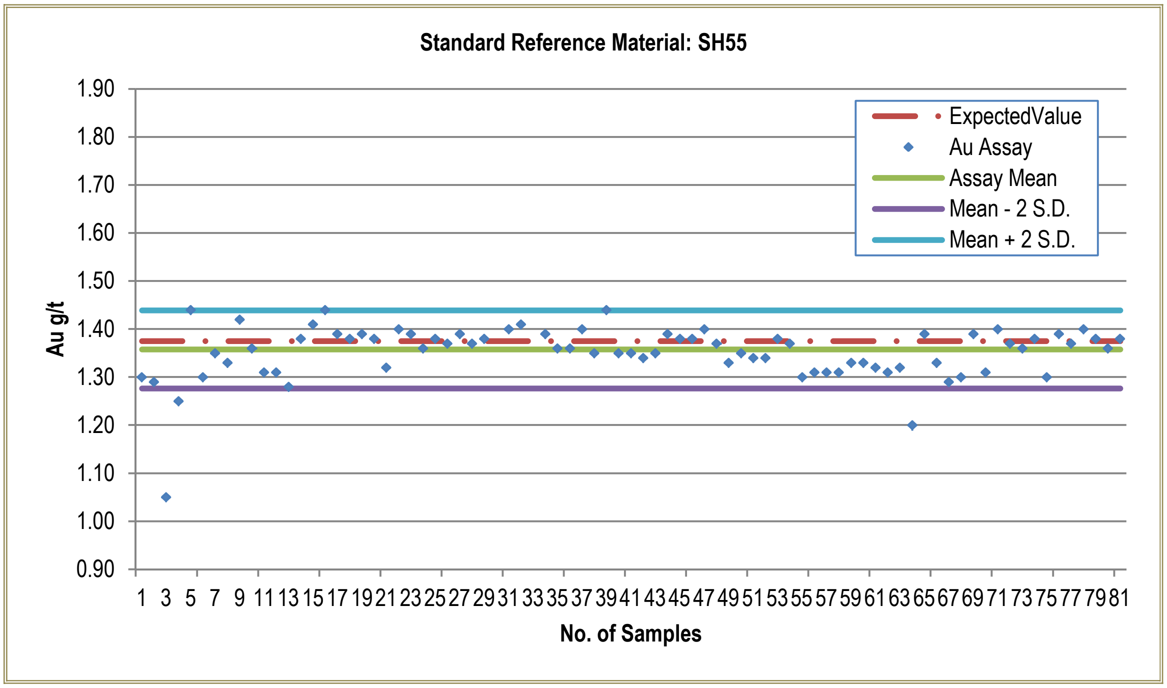


Figure 11-8: SRM SH65

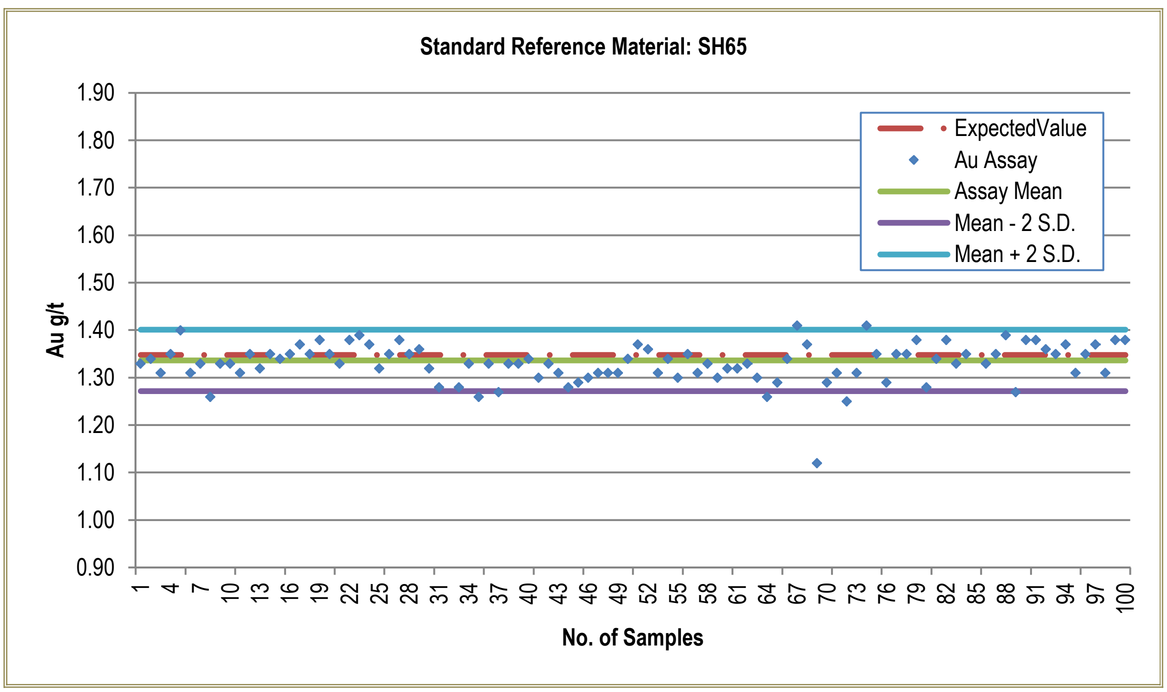


Figure 11-9: SRM SJ39

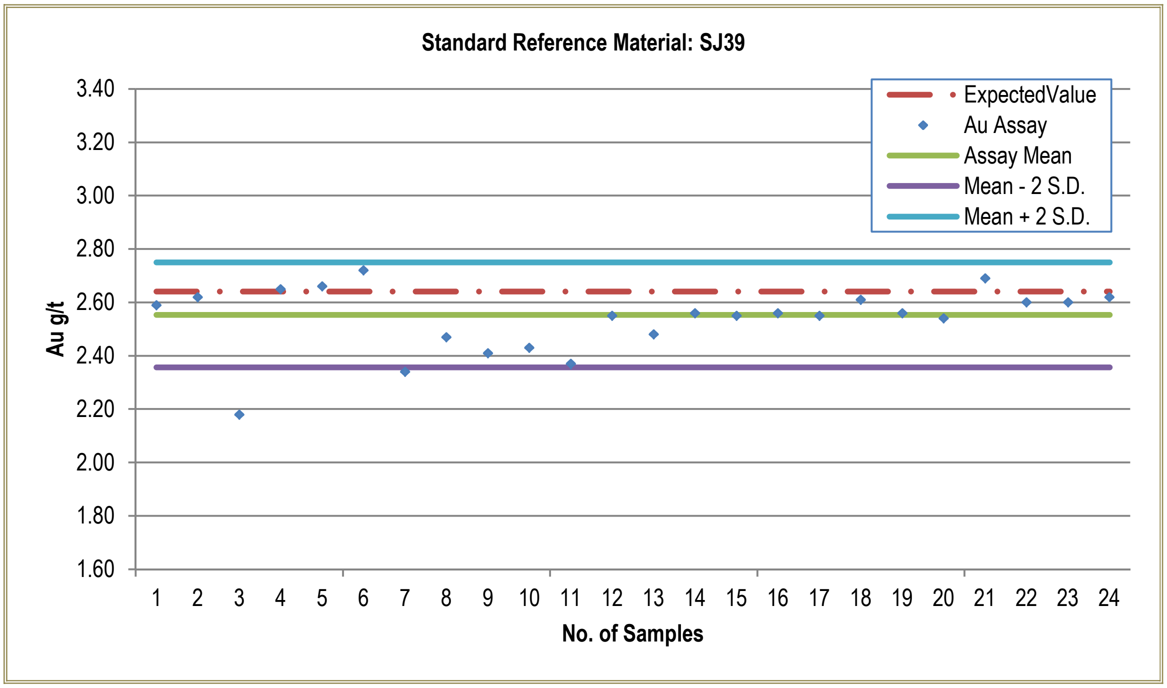


Figure 11-10: SRM SJ53

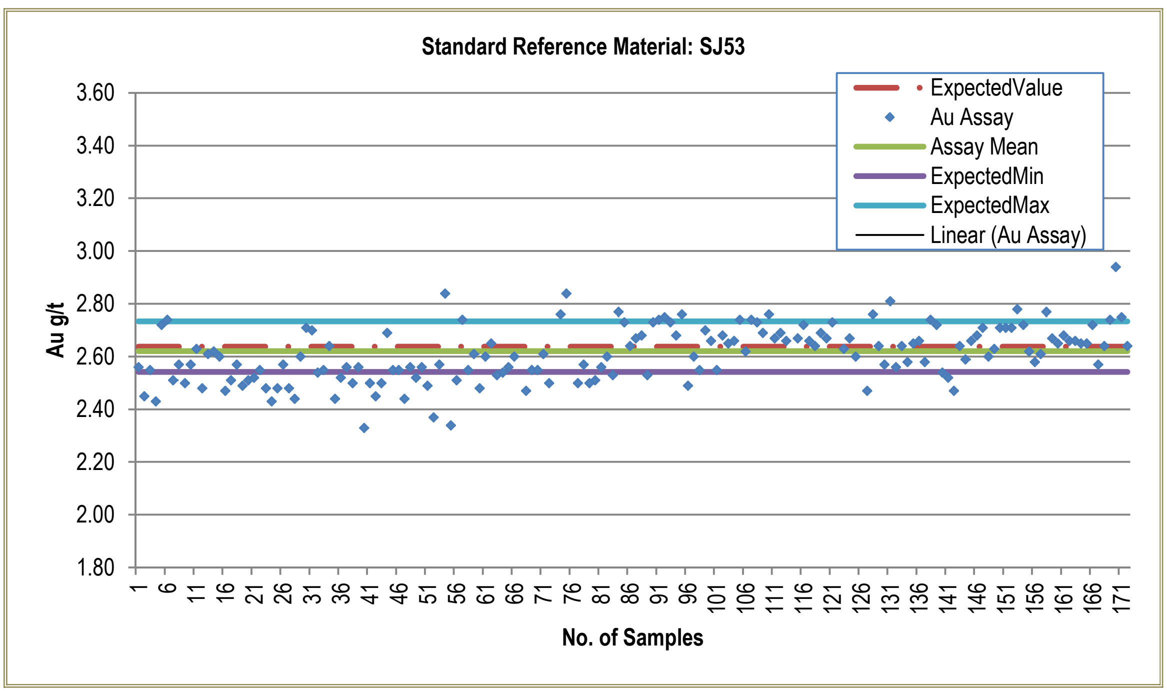


Figure 11-11: SRM SJ63

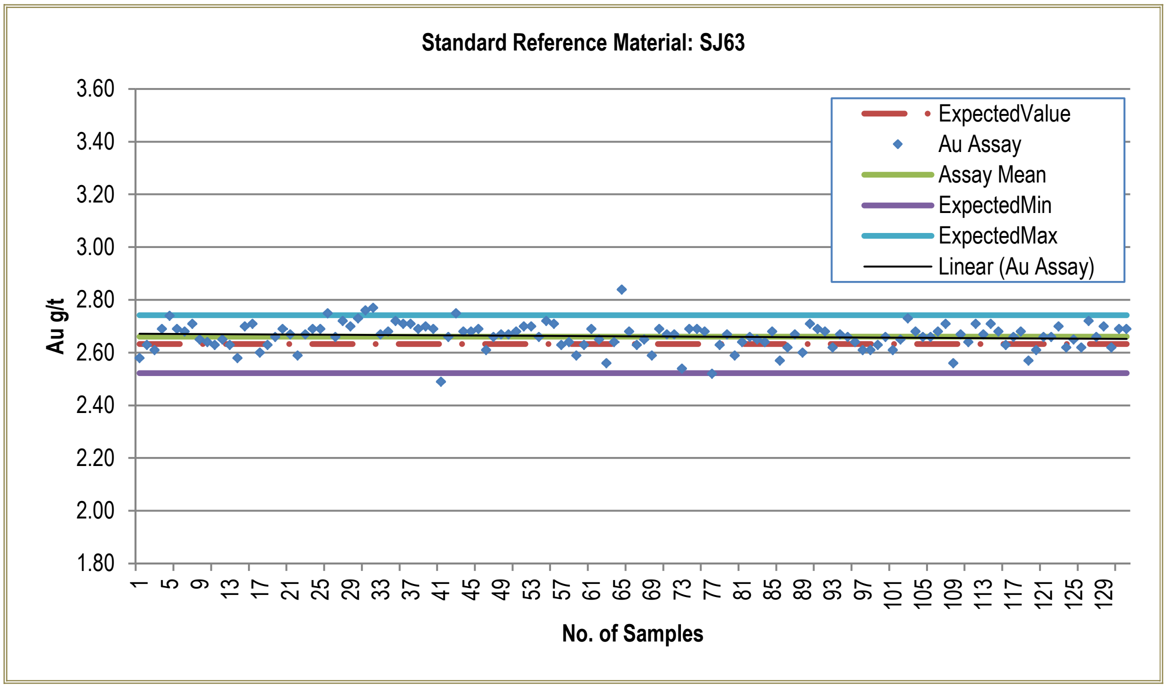


Figure 11-12: SRM SL46

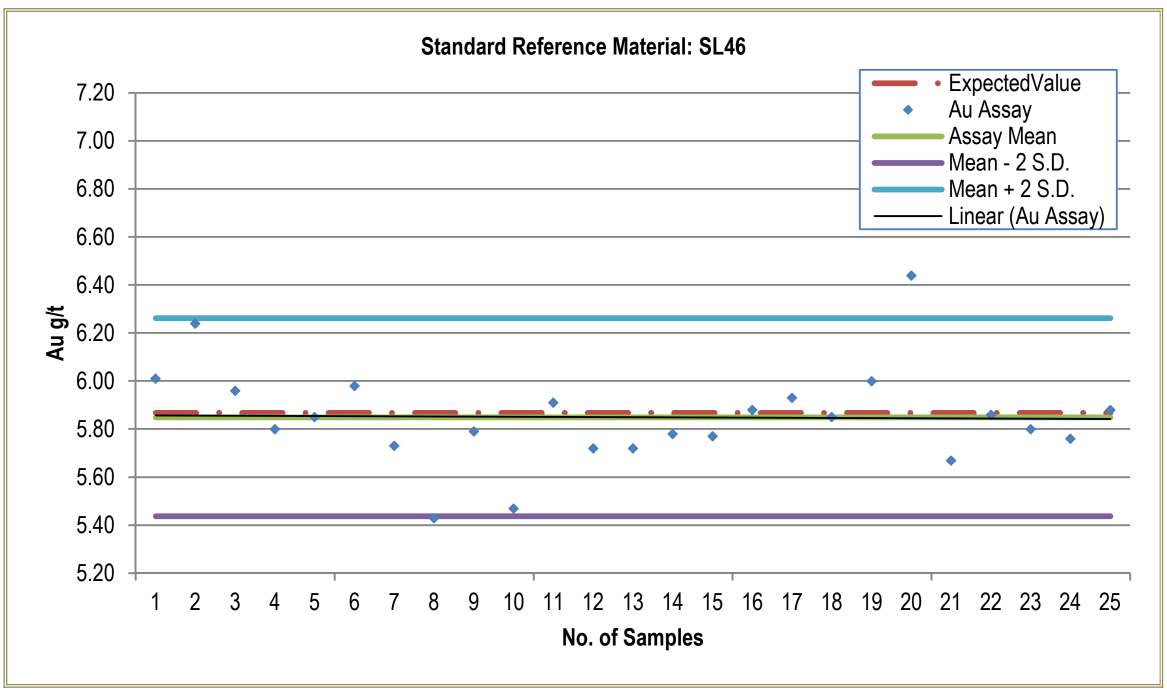


Figure 11-13: SRM SL51

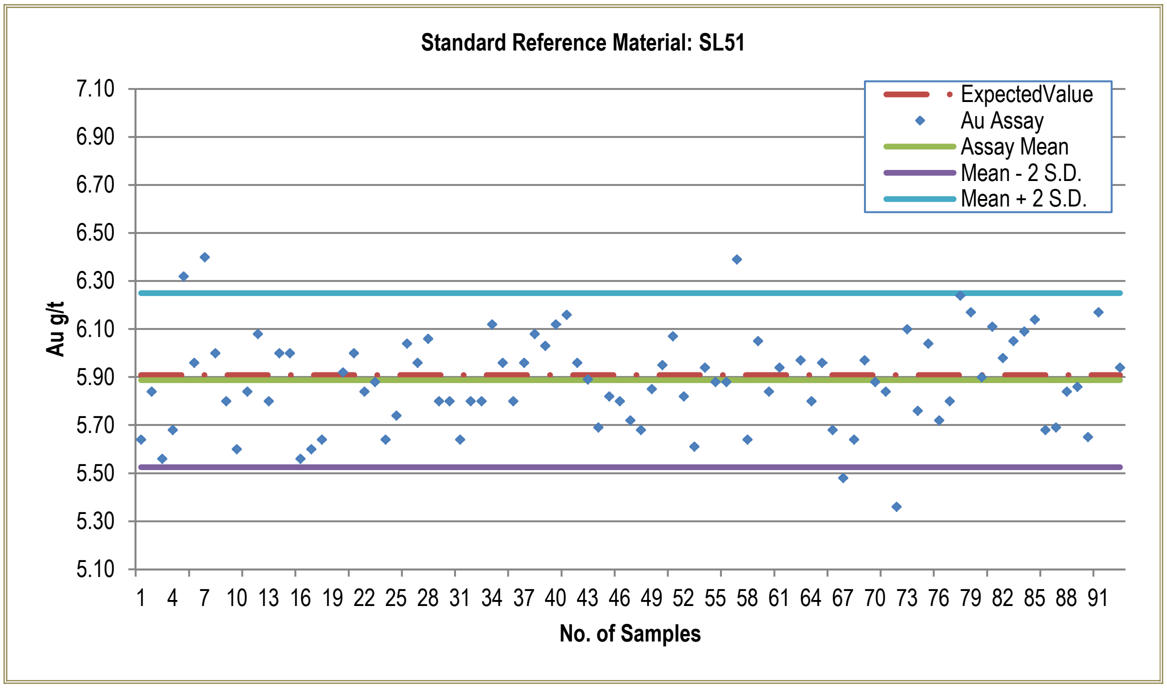


Figure 11-14: SRM SL61

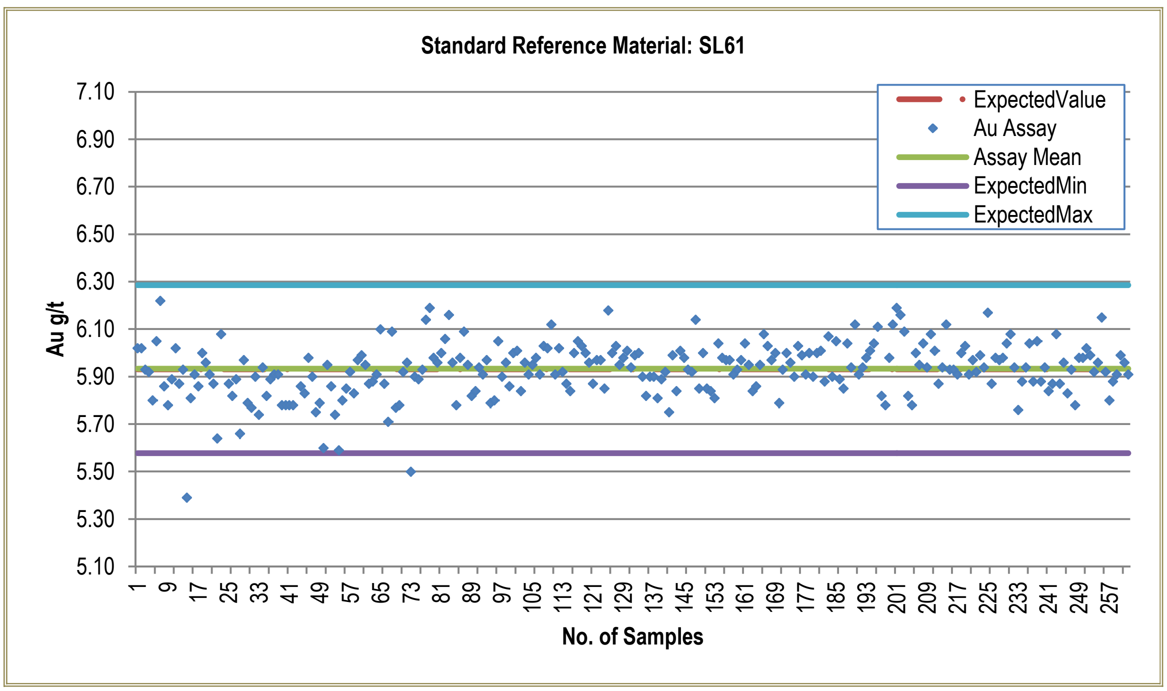


Figure 11-15: SRM SN38

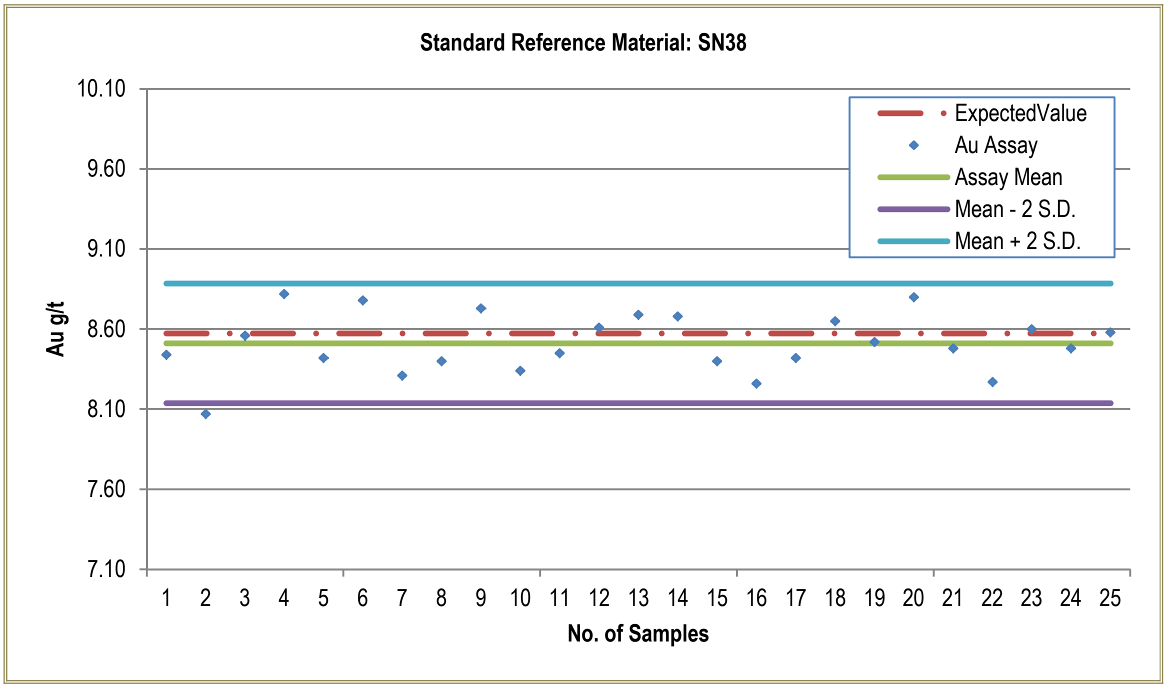


Figure 11-16: SRM SN50

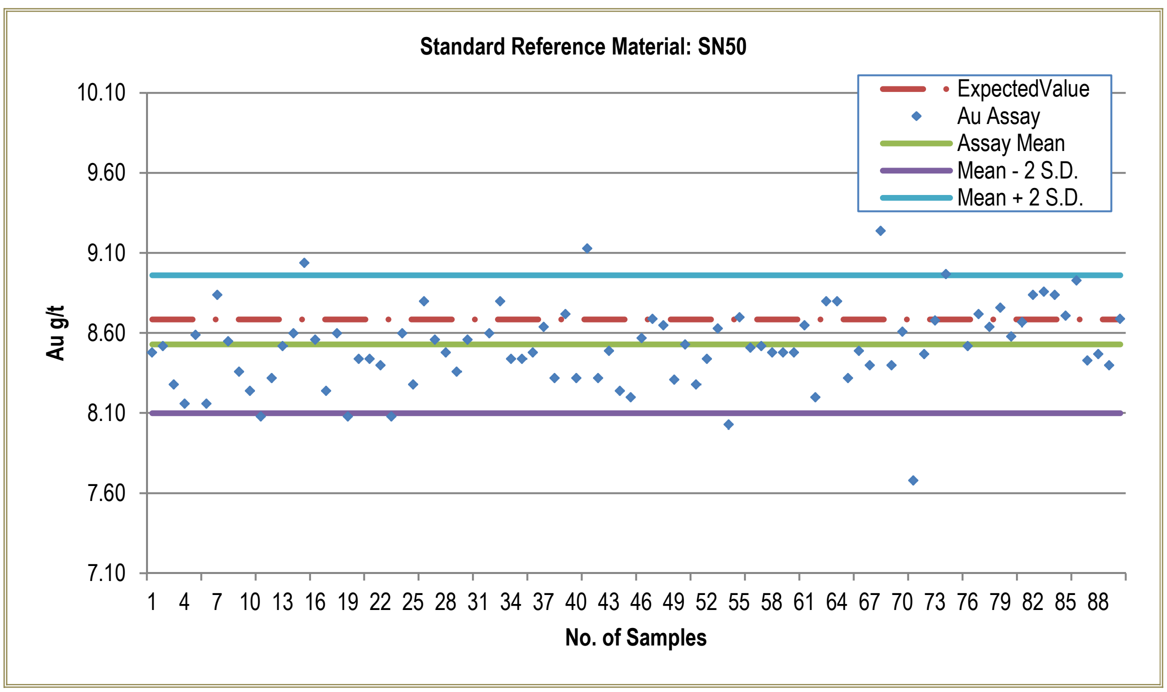


Figure 11-17: SRM SN60

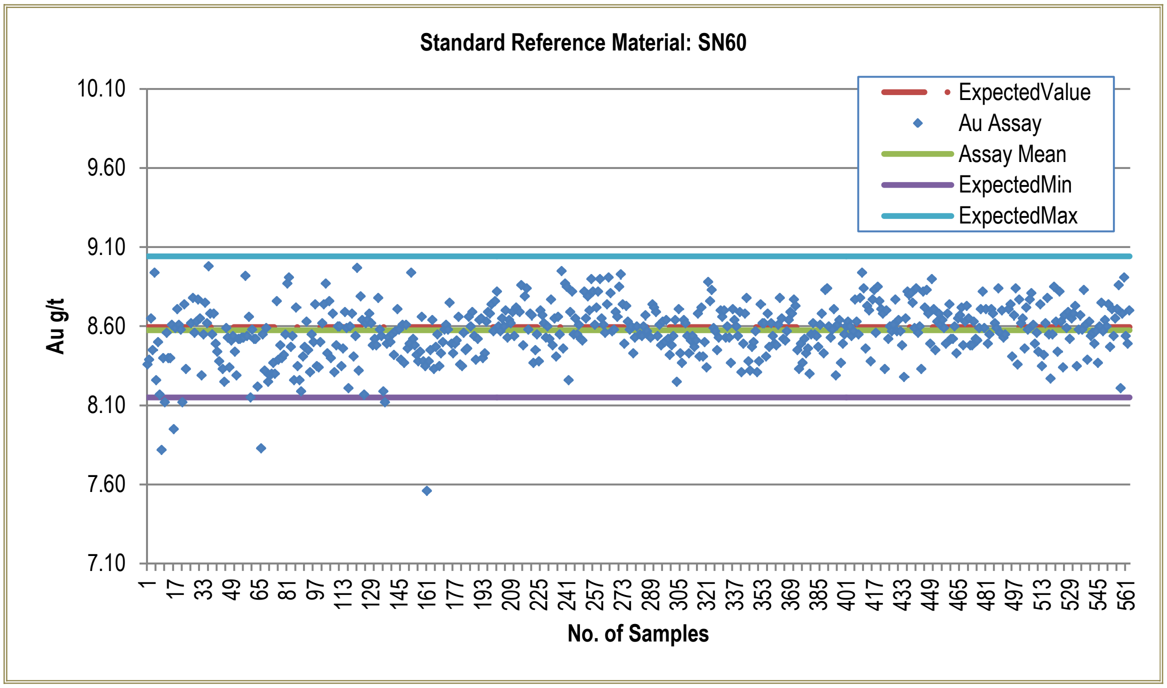


Figure 11-18: SRM SP37

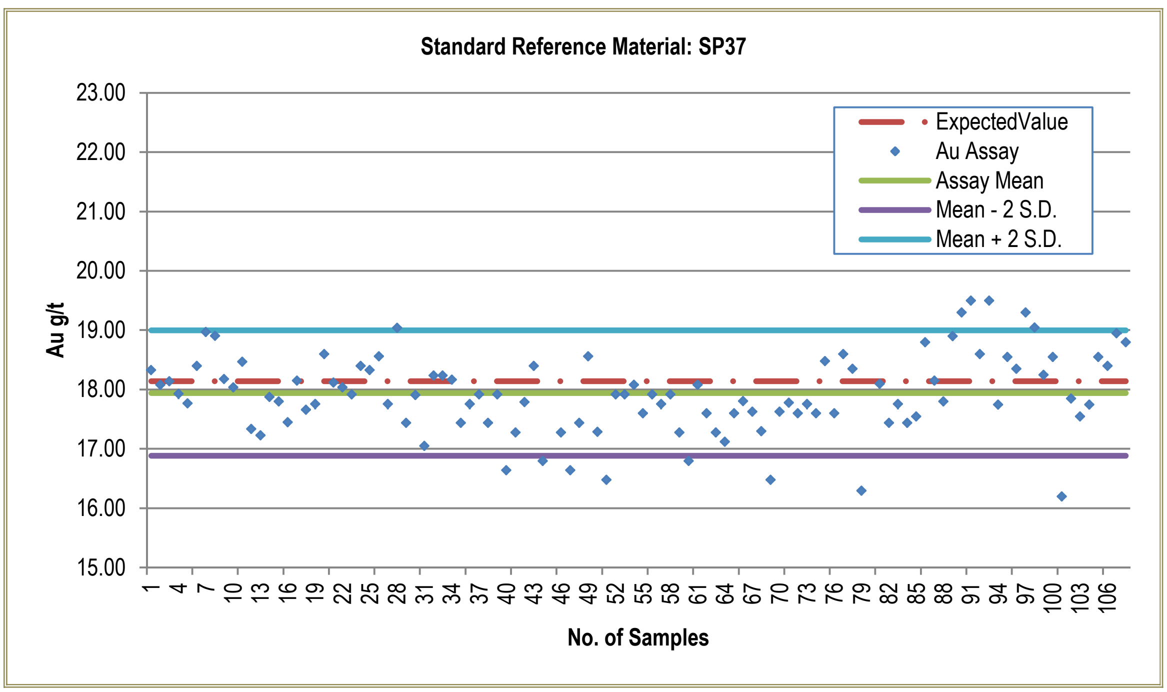


Figure 11-19: SRM SP49

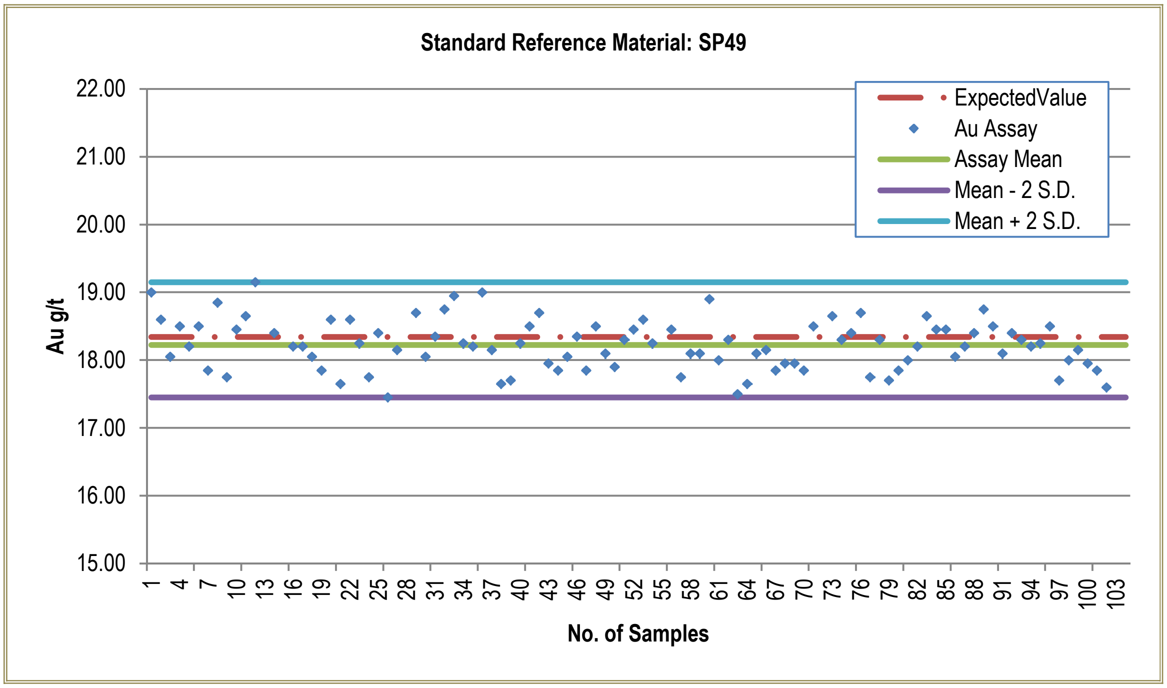


Figure 11-20: SRM SP59

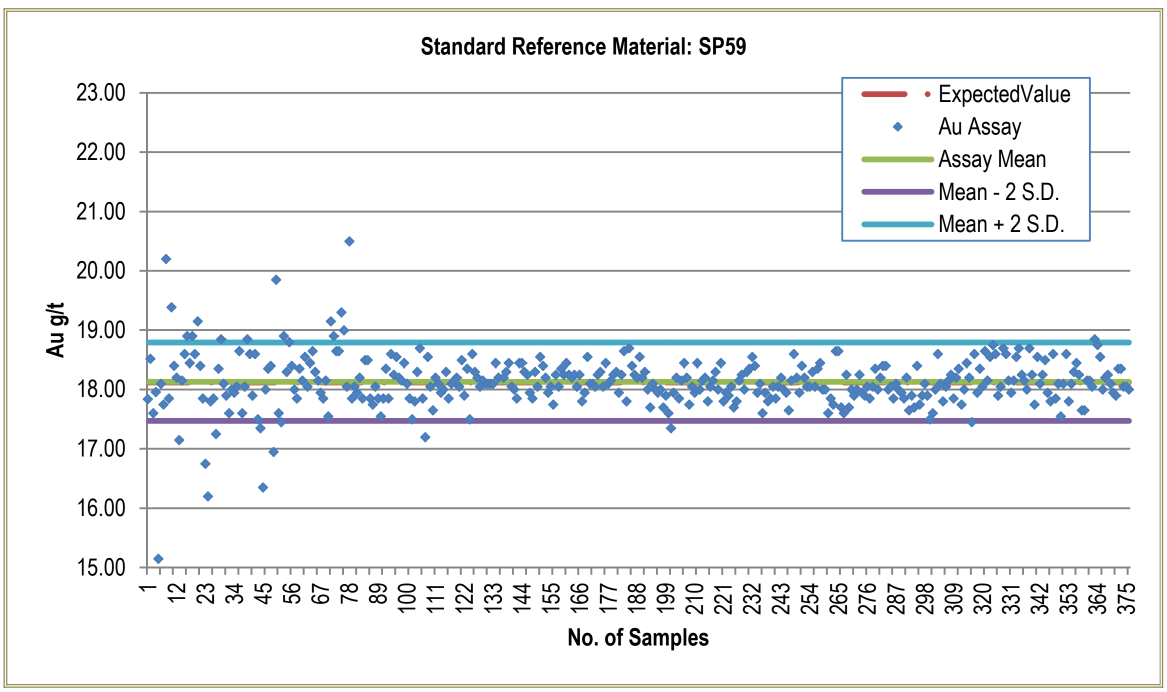


Figure 11-21: SRM SQ36

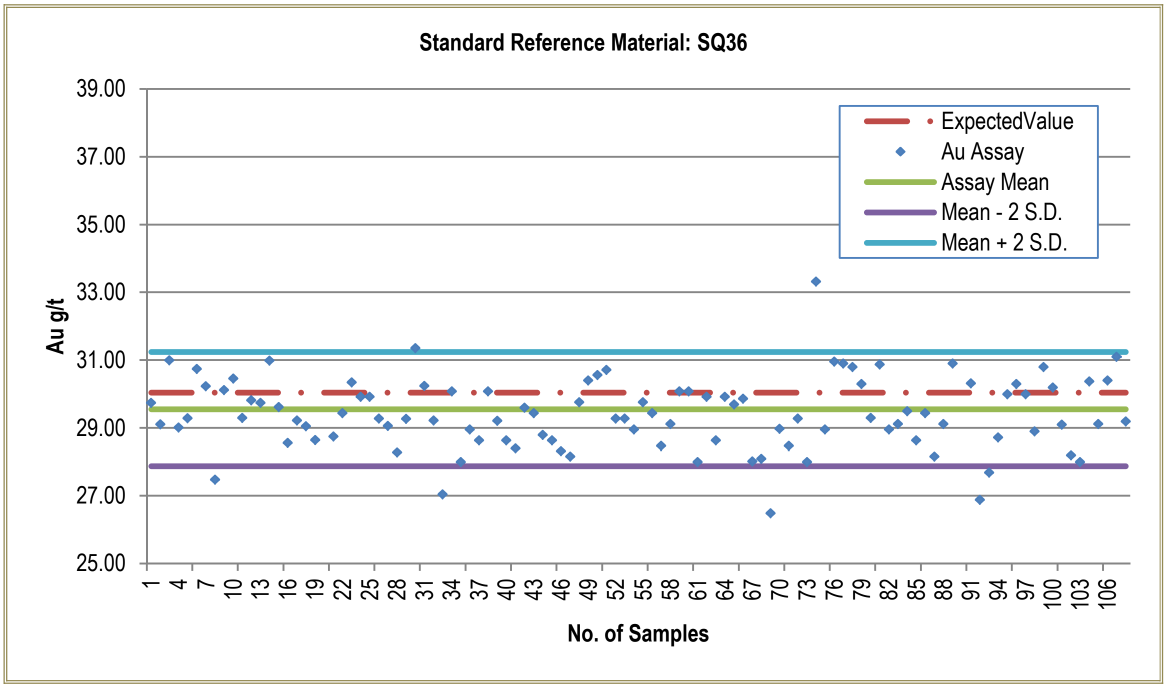
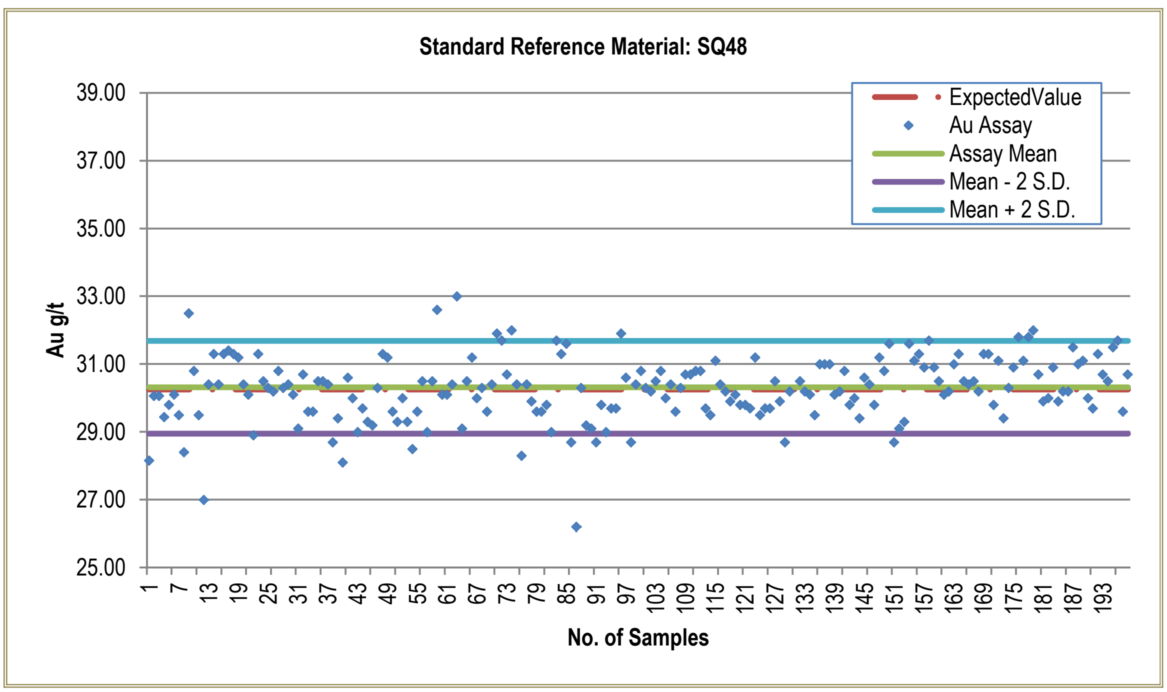


Figure 11-22: SRM SQ48



From a review of the different standard samples, it is evident ALR Loughrea has underperformed in the case of some SRMs. The bias is not systematic and does not appear to be correlated with a specific grade range. Current control charts should be maintained by Dalradian Resources and closely monitored. TMAC concludes that the QA/QC procedures conform to industry standards, and the data generated as a result are suitable for use in the present mineral resource estimate. No significant bias has been identified as a result of the above analysis which would warrant further study.

11.3.5 Duplicate Sample Results

The QC program prior to Dalradian Resources involvement did not include duplicate samples. Table 11-6 summarizes the status of the current duplicate samples in the Dalradian Resources' database.

Table 11-6: Duplicate Samples by QC Category

QC Category	Count	Au g/t Original	Au g/t Check
Lab Check	2693	3.883	3.879
Pulp Split	655	9.855	9.690
Resample	819	10.815	9.844

Scatter plots (Figure 11-23) were generated to understand the relationship between the original assays and the check assays. The plots clearly show good repeatability of original samples during re-assay for the lab check samples (Figure 11-23) and pulp splits (Figure 11-24). The resampled core duplicates (Figure 11-25) show greater variability with the check assays lower than the original assay.

Figure 11-23: Lab Duplicate Samples Scatter Plot

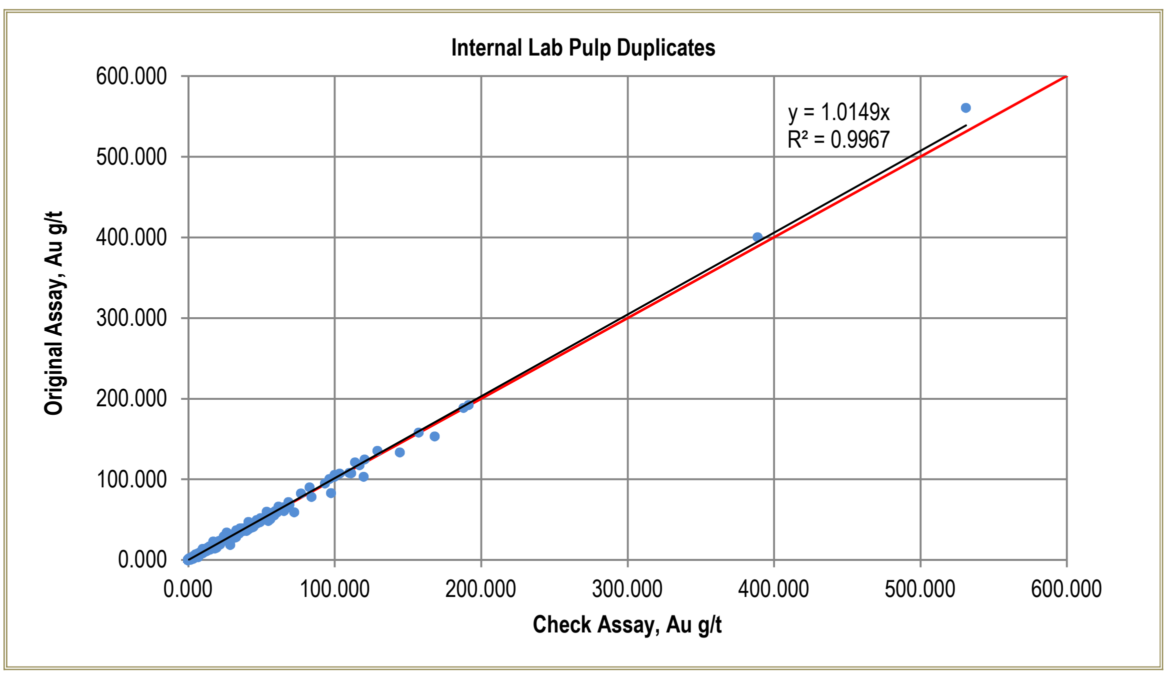


Figure 11-24: Pulp Duplicate Samples Scatter Plot

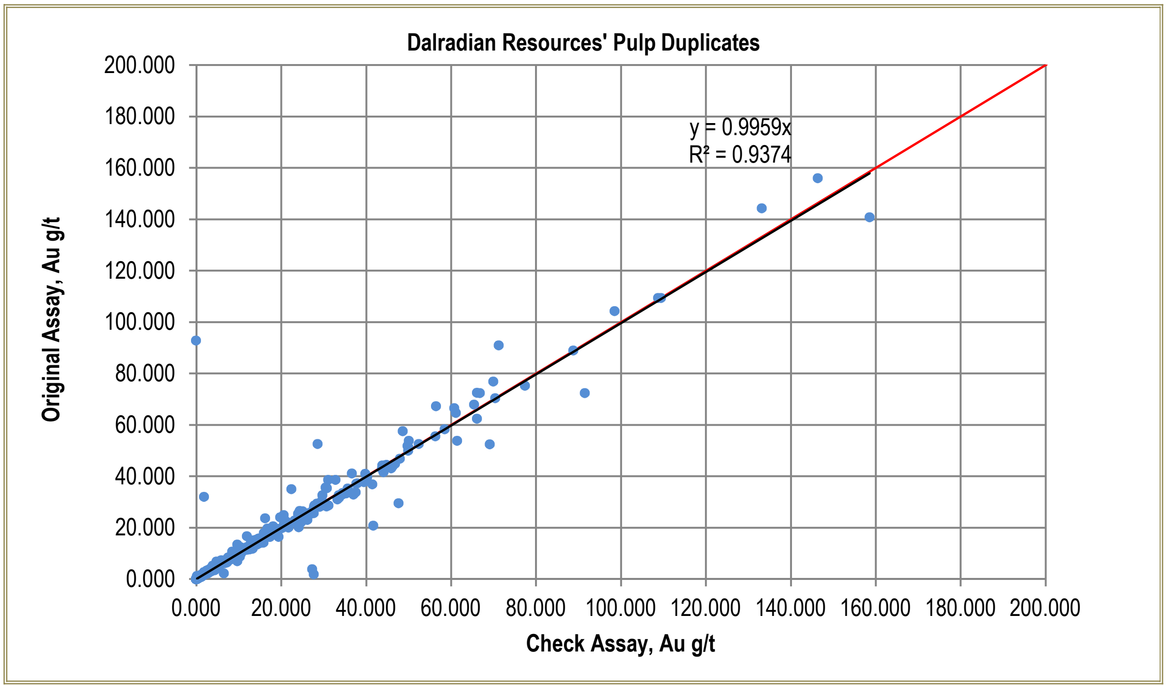
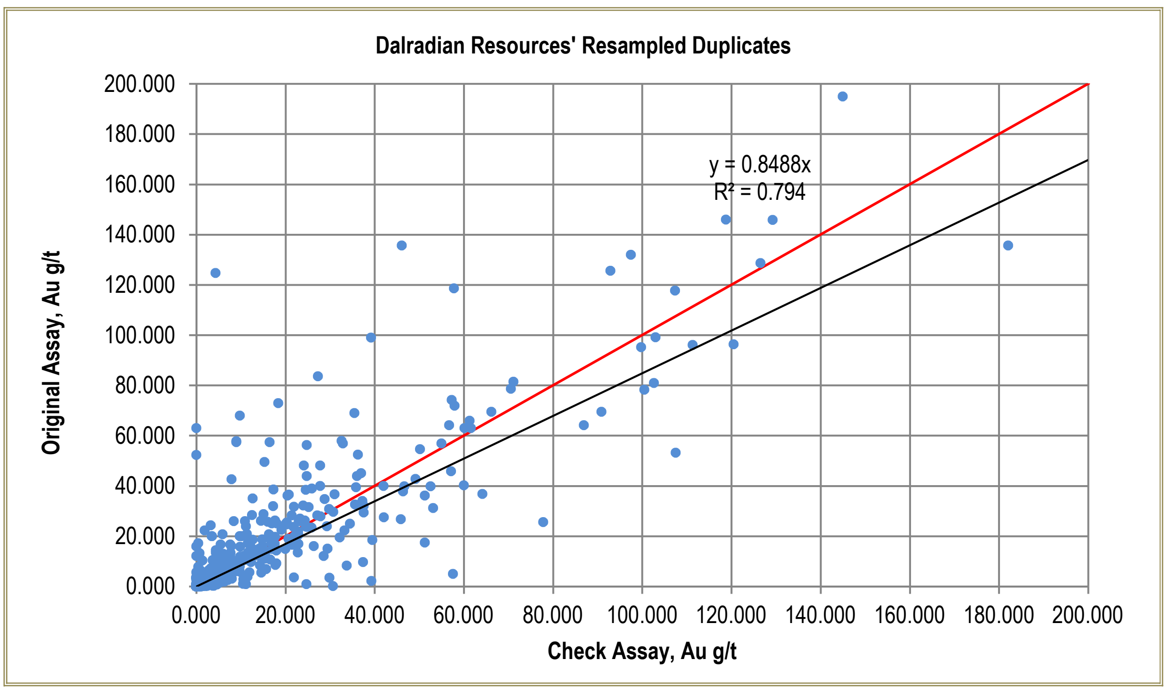


Figure 11-25: Resampled Duplicates Scatter Plot



TMAC concludes that the QA/QC procedures used conform to industry standards, and the data generated as a result are suitable for use in the present mineral resource estimate. No significant or systematic bias has been identified as a result of the above analysis which would warrant further study.

11.4 Specific Gravity Determinations

Tully (2005) provided a tabulation of SG determinations carried out by Tournigan in 2004. Two samples from drill core from each of the veins were sent to OMAC (now ALS Loughrea) for SG determinations by the water displacement method. Table 11-7 summarizes the SG determinations for the veins at that time.

Table 11-7: Specific Gravity Determinations

Sample Number	SG
Sheep Dip Vein	
SD41	3.01
SD43	2.54
Average	2.78
Attagh Burn Vein (ABB)	
ABB34	2.49
ABB45	3.65
Average	3.07
T17C Vein	
17C-50	3.65
17C-11	2.86
Average	3.05
T17HW Vein	
17HW-4	2.62
17HW-54	3.51
Average	3.07
T11F Vein	
T11F-14	2.90
T11F-28	2.97
Average	2.93
No. 1 Vein	
No1-21	3.16
No1-52	2.89
Average	3.03

A total of 2085 SG samples have been collected. Table 11-8 summarizes the SG determinations by vein codes. Two outliers were noted in the database and excluded from this tabulation. The one outlier was near 10, which may indicate a recording error. The other was near 1, which may be indicative of a less competent sample or also a recording error.

Table 11-8: Summary – Specific Gravity Determinations (g/cc)

Vein Code	Count	Average
310	11	2.70
320	20	2.69
325	9	2.74
330	38	2.76
335	13	2.79
340	61	2.79
345	15	2.95
350	27	2.86
355	94	2.80
360	12	2.72
365	102	2.78
370	4	2.82
375	37	2.79
385	49	2.74
395	72	2.80
397	6	2.89
Unknown	1515	2.74
Total	2083	2.75

The determinations were also classified as vein type, i.e., C- or D-veins. Within the mineralized D-Veins above 1 g/t Au, which were the basis for this resource update, the average specific gravity value was 2.85 g/cc. This is higher than the global average due to the sulphide association with higher gold grades. TMAC also determined that the vein sampling was not representative (spatially or by drill programs) so that an average values was selected rather than using the averages for each vein code. Continued sampling for SG determinations along with vein characterization is recommended.

Table 11-9: Mean Specific Gravity and Standard Deviation for Major Veins

Vein ID	No. of Samples	Min. (t/m³)	Max. (t/m³)	Avg. (t/m³)	SD (t/m³)	CV	No. of Samples	Vein Group	Density Used (t/m³)
10	22	2.43	3.40	2.70	0.22	0.08	22	10	2.70
20	18	2.48	3.18	2.81	0.19	0.07	18	20	2.81
30	22	2.43	3.68	2.88	0.35	0.12	22	30	2.88
40	78	2.47	4.12	2.85	0.32	0.11	98	40	2.84
403	8	2.41	3.02	2.73	0.21	0.08			
DL426B	12	2.69	3.36	2.84	0.21	0.08			
50	7	2.61	3.41	2.98	0.31	0.10	14	50	2.96
50E	5	2.68	3.49	2.96	0.28	0.09			
55	2	2.68	3.14	2.91	0.23	0.08			
60	104	2.43	3.99	2.86	0.25	0.09	121	60	2.85
604	10	2.66	2.88	2.79	0.08	0.03			
605	3	2.68	2.89	2.80	0.09	0.03			
606	3	2.76	3.00	2.85	0.11	0.04			
607	1	3.09	3.09	3.09	-				
70N	27	2.44	3.63	2.87	0.26	0.09	202	70	2.83
70S	97	2.45	7.39	2.83	0.49	0.17			
701	11	2.67	3.06	2.78	0.13	0.05			
703	2	2.52	3.20	2.86	0.34	0.12			
75	65	2.51	3.49	2.83	0.19	0.07			
80	32	2.67	3.27	2.82	0.15	0.05	37	80	2.81
DL8290	5	2.72	2.84	2.75	0.05	0.02			
90	49	2.53	3.59	2.81	0.20	0.07	49	90	2.81
D									2.57
G									2.57

Note: SD = Standard Deviation; CV = Coefficient of Variation

12 DATA VERIFICATION

12.1 Mineral Resource Data Verification

12.1.1 2011 Data Verification

The following is a summary of the data verification carried out at Curraghinalt for the 2011 mineral resource estimate (Hennessey, 2012a) in support of the 2012 PEA (Hennessey et al., 2012b) as outlined in Section 14.2 to 14.5. Five samples were collected in 2009 to confirm the presence of gold and copper mineralization at the Curraghinalt deposit. A sixth sample was collected from silicified and mineralized outcrop at the Cashel Rock showing on license DG2. The results of this sampling are set out in Table 12-1.

Table 12-1: Check Samples for 2011 Mineral Resource

Sample No.	Location	Type	Au (g/t)	Ag (g/t)	Cu (ppm)
75117	T17 Vein	Underground grab sample	15.1	6.7	1,790
75118	T17 Vein	Underground grab sample	4.2	4.0	2,150
75119	#1 Vein	Underground grab sample	21.9	5.1	54
75120	Hole CT55	Duplicate ¼ core sample	42.6	9.2	97
75121	Hole CT54a	Duplicate ¼ core sample	45.2	9.0	25
75122	Cashel Rock	Surface grab sample	8.71	10.9	555

The original assay results for the duplicate quarter core samples 75120 and 75121 were 44.96 g/t Au and 23.68 g/t Au. The samples collected confirmed the presence of gold mineralization at the expected grades on both the DG1 and DG2 licences.

The checks carried out in 2011 confirm the logging procedures utilized at the Curraghinalt site. All drill core is stored in wooden boxes with proper numbering to indicate the drill hole number and meterage. Random checks were carried out by Micon on the stored core and no discrepancies were identified.

The drill hole database at that time was maintained in Microsoft Access and work was carried out at Dalradian's offices in Omagh, Northern Ireland. Regular checks were performed there to ensure that there are no errors in data entry. Assay results were provided in Excel files from OMAC (now ALS), thereby allowing direct electronic data transfer. This eliminated any potential error arising from manual data entry.

Data verification protocols available in Datamine software were utilized to check the database for errors such as transposed or crossed from and to entries for sample intervals, or incorrect entries in data fields.

12.1.2 Data Verification for the 2011 Mineral Resource Appears Reliable to TMAC 2013 Data Verification

TMAC performed a verification of the compiled project database against digital logs provided at site. The validation of the data was completed on three drill holes. This data verification process examined three tables: collars, surveys, and assays. Drill holes were selected which had been drilled after the previous resource estimate (Hennessey et al., 2012a): 12-CT-166, 13-CT-184, and 13-CT-190.

In addition to this verification, validation was conducted during the import of the drill holes into MineSight. This included comparing EOH length with last sampled interval, overlapping From-To intervals, duplicate intervals and that data were within valid ranges (i.e., collar coordinates are within the established project or that grades are within the accepted grade ranges).

Dalradian has followed industry accepted procedures in the collection and validation of their data. Based on the verification completed by TMAC in this study, the database provided on January 20, 2014 is acceptable for use in the updated resource.

12.2 TMAC Site Visit

T. Maunula, P.Geo. visited the property from August 12 to 15, 2013. During this site visit, core was examined for the drill holes: 12-CT-166, 13-CT-184, and 13-CT-190. The logging tables reviewed in conjunction with the core were: assays, core loss, core recovery, structure, veins and the geology composite log. The logging was consistent with the remaining split core.

Dalradian has followed industry-accepted procedures in the logging and sampling of drill core and no material issues were identified by TMAC.

The collars were visited in the field but the collar sites had been reclaimed. The approximate position of the collar versus reclaimed site was confirmed by GPS.

Ten samples were collected by TMAC: nine from ¼ split drill core and one an underground grab sample. The assay results (Activation Laboratories Limited, October 2013) from these samples are reported in Table 12-2. The presence of gold was confirmed by the comparable grades obtained from the TMAC samples.

Table 12-2: TMAC Site Visit Samples

Site Visit Check Samples (ACTLABS A13-11879)				Dalradian			
Sample No.	Location	Type	Au (ppm)	Sample No.	From	To	Au (ppm)
F30900	13-CT-184	Duplicate ¼ core sample	0.08	F10943	318.00	318.72	0.10
F30901	13-CT-184	Duplicate ¼ core sample	4.57	F10944	318.72	318.93	9.80
F30902	13-CT-184	Duplicate ¼ core sample	0.38	F10945	318.93	319.09	0.57
F30903	12-CT-166	Duplicate ¼ core sample	0.02	D8114	316.00	316.51	0.51
F30904	12-CT-166	Duplicate ¼ core sample	27.20	D8115	316.51	316.81	28.30
F30905	12-CT-166	Duplicate ¼ core sample	0.10	D8116	316.81	317.00	0.67
F30906	12-CT-166	Duplicate ¼ core sample	2.88	D8193	349.92	350.02	6.58
F30907	13-CT-190	Duplicate ¼ core sample	12.40	F17994	333.35	333.59	14.50
F30908	13-CT-190	Duplicate ¼ core sample	32.90	F17995	333.59	333.81	56.90
F30909	QAQC	QC Sample – blank	0.09	-	-	-	-
F30910	QAQC	QC Sample – SRM SJ63	2.58	-	-	-	2.63
F30911	T17	Underground grab sample	3.83	-	-	-	-
F30912	QAQC	QC Sample – blank	0.01	-	-	-	-

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b) and is supplemented with additional metallurgical testwork information current to the effective date of this report. The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

Previously prepared reports dating from 1985 to 1999 describe metallurgical testwork completed on samples from the Sperrin Mountain and Curraghinalt Project. More recently, testwork was completed by SGS Mineral Services (Lakefield) in 2012, and ALS Metallurgy (previously operating as G&T Metallurgical Services Ltd), Kamloops, BC, Canada (ALS) has conducted ongoing work since 2012.

A Bond Ball Mill Work index (BBWi) test was carried out in 1985 and repeated in 2012, and an abrasion index (Ai) test was completed on a single composite. Completed gravity concentration testing indicates that the samples are amenable to this process. All samples tested responded well to cyanidation, but cyanide consumption rates were high. In 1986, Lakefield examined pre-aeration and lead nitrate addition, which resulted in improved recovery rates and lower cyanide consumption. Alternatively, the gravity recoverable concentrate could be suitable for sale to a smelter.

Flotation testwork undertaken suggests that a copper concentrate could be produced for sale to a smelter. A bulk flotation concentrate was also produced during testwork, which showed high gold recovery and amenability to cyanidation, or could possibly be suitable for sale to a smelter.

The testwork reports reviewed are referenced in Section 27.

It was observed that there was a wide variation in gold, silver, and copper head assays in the samples used for the testwork carried out prior to 2012. The exact origin of samples from the earlier studies was not clearly identified. This earlier testwork was primarily conducted on samples from the T-17, No. 1, and Sheep Dip veins. Metallurgical testing since early 2012 has been carried out on composites as follows:

- Composite 12-1A, consisting of approximately 200 kg of vein and wall rock material taken from drill core and underground workings. Assay of Composite 12-1A by ALS showed average grades of 10.6 g/t Au, 6.0 g/t Ag, and 0.33% Cu.
- Composite 12-1C, a smaller composite made by blending Composite 12-1A material with selected drill core material from Curraghinalt assayed 5.23 g/t Au, 4.0 g/t Ag, and 0.10% Cu.
- Composite 12-1D, consisted of a portion of Composite 12-1A diluted with barren wall rock and assayed 9.14 g/t Au, 7.0 g/t Ag, and 0.21% Cu.

13.1 Sample Characterization – Mineralogy

Mineralogical studies were conducted by Lakefield on a sample in 2011, and are currently being undertaken by ALS as part of an ongoing testwork program, which includes QEMSCAN and chemical assay.

Mineralogy work conducted in 2004 indicated that gold mineralization at Curraghinalt occurs in quartz-pyrite veins and is associated with variable abundances of carbonate, chalcopyrite, and tennantite-tetrahedrite. In general, carbonate, chalcopyrite, and tennantite-tetrahedrite are paragenetically later than quartz and pyrite, and fill fractures in the latter. Gold occurs mainly as the native metal, and more rarely as electrum (>20 wt% Ag), and is found primarily along fractures in pyrite, as inclusions in pyrite, and at pyrite grain contacts with carbonate and quartz. Most native gold grains are associated paragenetically with carbonate, chalcopyrite, tennantite-tetrahedrite, and telluride minerals. The report examined mineralized samples from all the veins, and concluded that the mineralogy was generally similar in all the veins found at Curraghinalt.

Results from ALS (2012) indicate that 94.0% of the copper-bearing minerals occur in the sample as chalcopyrite. There were minor amounts of chalcocite/covellite and tetrahedrite/tennantite, together accounting for less than 5% of the total copper.

Pyrrhotite was not identified in the sample analyzed in this study by ALS, nor was it identified in the 2011 Lakefield work. Pyrrhotite was identified in a single sample in the 1986 testwork. This may explain the lower levels of cyanide consumption during the ALS testwork compared to testwork conducted between 1985 and 1999. Also, the copper grades in the historical testwork were generally higher than the average of the deposit, which would explain the elevated cyanide consumption observed during those tests.

In the ALS assays of sample Composite 12-1A, copper occurrence, as reported through QEMSCAN, was 0.33%, with iron assaying 4.83%, and sulphur 3.32%.

13.2 Metallurgical Testwork Results

13.2.1 Comminution

ALS (2012) completed a single BBWi test on composite sample 12-1B that returned a value of 15.2 kWh/t. This agrees well with a BBWi test completed by Lakefield in 1986 on a composite sample that returned a value of 15.4 kWh/t.

ALS also completed a single Abrasion index on composite 12-1B that returned a value of 0.1278 g.

13.2.2 Gravity Concentration

The earlier testwork (1985 to 1989) examined the amenability of the samples to gravity concentration using a Wilfley table with cleaning on a Mozley concentrator. The gravity recovery of gold varied widely in these tests, with gold recovery results ranging from 26% to 52%. Testwork done in 1999 utilized a Knelson concentrator, and gold recovery ranged from 50% to 52%.

Gravity concentration testwork was carried out by ALS in 2012 and 2013 using a Knelson concentrator. Gold recoveries into the gravity concentrate of 81% and 76% into a concentrate of 6% and 5% mass pull was achieved from composite samples 12-1A and 12-1B, respectively. Additional testing achieved gold recoveries into the gravity concentrates of 61.5% and 67.9% with mass pulls of 3.1% and 3.4%, and gold recoveries of 29.4% and 24.2% with mass pulls of 0.2%. These results suggest a good correlation between mass pull and gold recovery into the gravity concentrate.

13.2.3 Heavy Media Separation

Lakefield carried out testwork in 2011 to investigate Heavy Media Separation (HMS) as a means of reducing the amount of feed material reporting to the grinding section of the plant by rejecting the waste portion of the plant feed that would come from the mine. The tests indicated that, using this pre-treatment of the plant feed, it is possible to reject 50% of the feed material into the waste stream; however, gold loss into the reject material was about 4%. As part of this testwork program, the sink portion was subjected to bulk flotation and cyanidation tests. HMS is not considered for this PEA Study.

13.2.4 Whole Ore Cyanidation

Cyanidation testwork has generally returned very high metal extractions, typically 95% or better for gold, and about 80% for silver. A grind of approximately 85% passing 200 mesh (75 µm) and 48 hours leach time at 1 g/L NaCN was found generally effective in the earlier testwork.

During the historical testwork, cyanide consumptions in direct cyanidation tests were variable but generally high. These tests suggested consumption rates of between 1 kg and 2.4 kg NaCN per tonne of feed.

Where solution assays were available, they showed high copper and thiocyanate (CNS) in solution, and copper sulphides were most likely the cause of the high cyanide consumption.

As part of their HMS testwork, Lakefield completed cyanide leach tests on samples of the rougher concentrate and rougher tailings from the sinks and float portions of the HMS test. Extractions were fairly consistent (≈90%), and indicated that there is likely a strong dependency on particle size (independent of grade).

Recent test results from ALS using leach feed with lower copper levels more closely matching the current mine production model have indicated lower cyanide consumption of 0.2 kg/t to 0.6 kg/t, as shown in Table 13-1.

Table 13-1: Gravity Recovery Tailings Cyanidation Testwork

Sample	Cyanide Consumption	Reference
Composite 12-1A	0.6 kg/t	ALS, June 2012
Composite 12-1B	0.2 kg/t	ALS, June 2012

13.2.5 Flotation

Several flotation tests have been conducted, both historically, and as part of the Lakefield and ALS work to produce a copper concentrate and a bulk flotation concentrate. Several cyanide-leaching tests of the bulk flotation concentrate were also completed.

During the recent Lakefield study, the sinks concentrate from the HMS test was submitted for flotation tests. Gold and silver rougher recoveries were 99% and 95%, respectively (relative to the flotation feed), into 42% of the mass. This testwork suggested that a relatively coarser grind is likely possible prior to flotation. The flotation results suggest that the gold occurrence is strongly associated with sulphides, which is consistent with the conclusions in the 2004 mineralogy report.

A copper flotation process option was considered, with the dual goals of producing a saleable copper concentrate and reducing cyanide consumption during the leaching process. Testing by ALS on composites 12-1A and 12-1C showed that, while a copper-gold concentrate could be produced, elevated levels of penalty elements in the concentrate would likely make it difficult to market.

The most recent flotation testing by ALS focused on producing a rougher concentrate, both with and without a gravity circuit prior to flotation. Gold recoveries into the flotation concentrate without a gravity circuit were 98.8% to 99.4% (Composite 12-1A), and 94.7% to 95.3% (Composite 12-1D), and 70.0% to 74.6% (Composite 12-1A) when a gravity circuit was used ahead of the flotation circuit. Multi-element assaying of the resulting concentrates showed low levels of penalty elements such as arsenic, antimony, bismuth, chlorine, and fluorine.

13.2.6 Concentrate Cyanidation

In 2013, ALS Metallurgy carried out a number of bottle roll tests to assess the cyanide leach characteristics of the rougher flotation concentrates produced from Composite 12-1A and the effects of regrind size, pre-aeration, and cyanide concentration. Leach recoveries ranged from 77.3% to 95.5%, with cyanide consumption ranging from 4.6 kg/t to 9.1 kg/t of concentrate. ALS noted that pre-aeration helped with reducing cyanide consumption, and improved recovery to some extent. Regrinding of the concentrate before leaching also improved gold recoveries in some samples.

13.3 Proposed Future Testwork

Based on a review of reports leading up to the testwork currently underway, it is recommended that future testwork should include the following:

- locked-cycle flotation testing of the bulk sulphide concentrate circuit
- cyanidation tests of the bulk sulphide concentrate (rougher and cleaner products), both with and without regrinding, pre-aeration, and lead nitrite addition
- additional BBWi testing
- SAG amenability testwork
- further evaluation of the effect of grind size on leach extractions for the whole ore cyanidation option

- evaluation of alternative flotation reagents, especially collector 3418A
- carbon-in-Leach (CIL) testwork
- testwork with higher cyanide concentrations
- confirmatory testwork on a variety of representative feed samples once a preferred flowsheet is established.

14 MINERAL RESOURCE ESTIMATES

14.1 Previous Mineral Resources

Information in Sections 14.1 to 14.5 is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

14.1.1 Micon Mineral Resource Estimate

The mineral resources used to prepare the PEA presented in this report were first published in Hennessey et al. (2012a) and have an effective date of November 30, 2011. A summary description of the methodology used to prepare the 2011 estimate is presented here.

Micon prepared the 2011 estimate of mineral resources for the Curraghinalt deposit using geological information and assay data from 301 drill holes and 277 underground channel samples. A total of 583 specific gravity measurements were also utilized. Primary, or raw, assay data were composited for gold and were analyzed to determine the basic statistical and geostatistical characteristics. This information has been used in several modelling algorithms, which have been compared and checked for validity.

Several mineralized veins have been identified at Curraghinalt and these have been drilled to different degrees. The 2011 resource estimate encompassed all the known mineralized veins, which, at the time, comprised 12 major veins, 4 secondary veins, and 19 veinlets, which bifurcate out of the major veins.

For the purpose of the 2011 resource definition and modelling, each of the identifiable veins was designated with a code.

The details of each mineralized vein, showing the codes used in the mineral resource model, are given in Table 14-1.

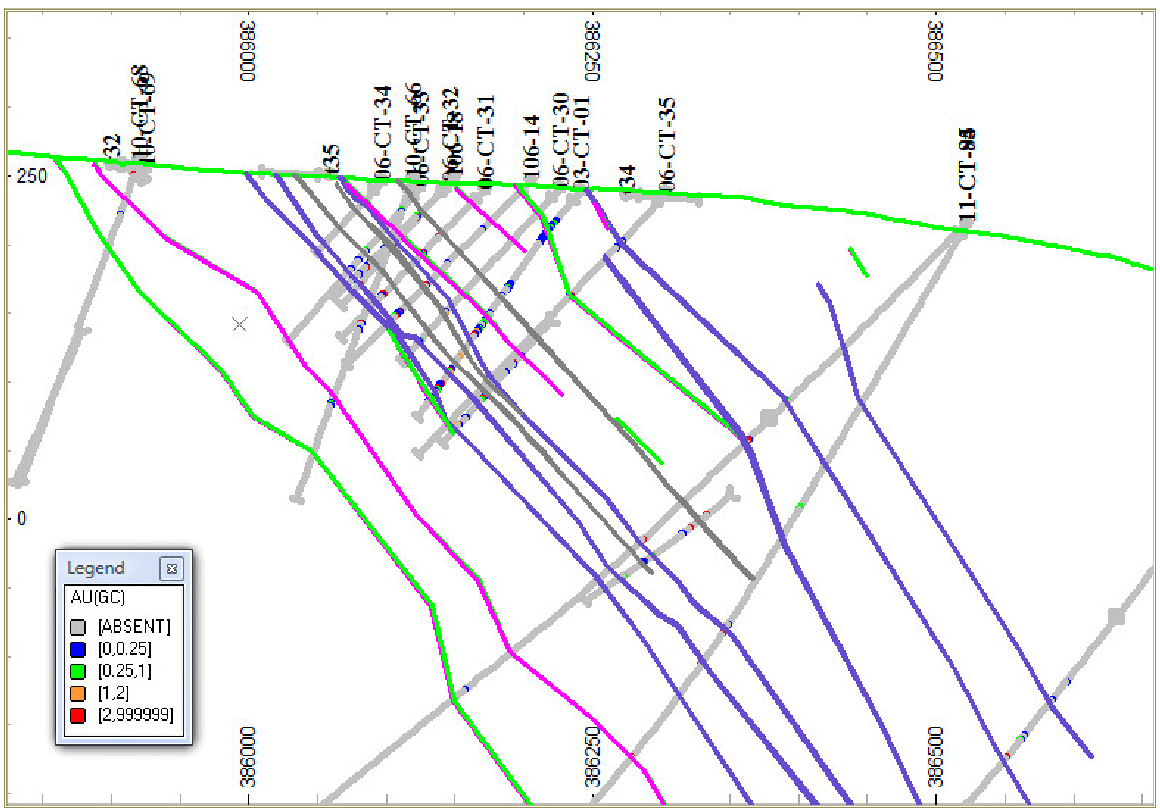
Table 14-1: Mineralized Veins and Codes

Primary Veins	Secondary Vein	Small Veinlets Associated with Primary Veins
10		
20		
30	35	302, 302E
40	45	403, 451E, DL426B
50, 50E	55	552

Primary Veins	Secondary Vein	Small Veinlets Associated with Primary Veins
60		61,62, 601,604,605,606,607
70S	75	70N, 701,703,708, 752
80		DL8290
90		
D		
G		

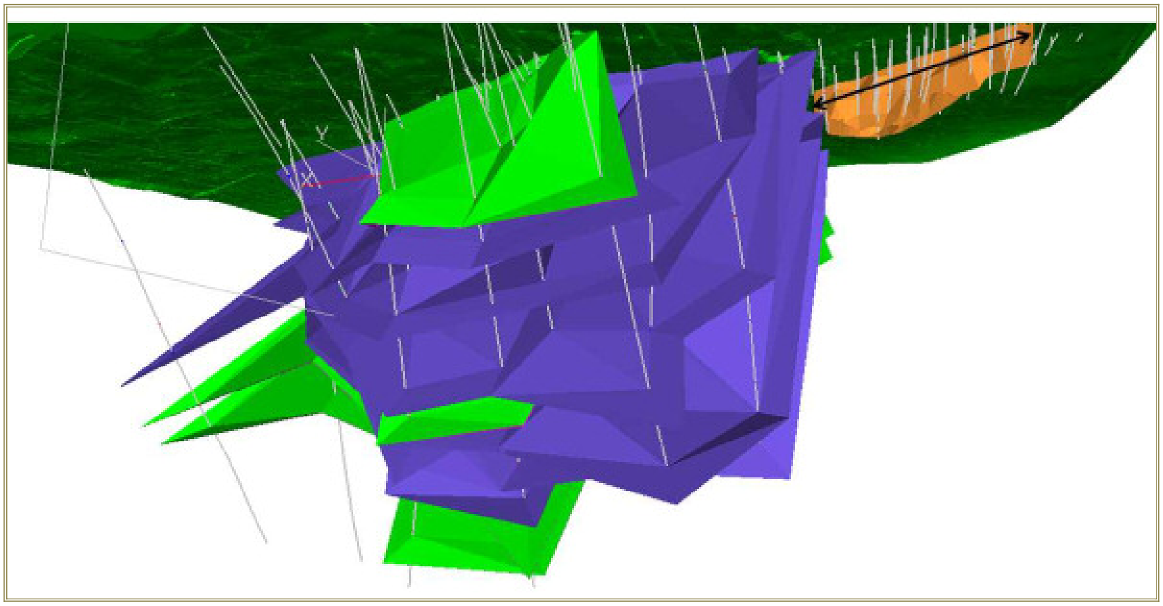
The hanging wall and footwall contacts of each of the veins were calculated from the identified zone intersection from each drill hole. Two-dimensional (2D) surfaces were then created for the hanging wall and footwall of each vein, with an additional extension 30 m to 40 m beyond the last contact point derived from the drill hole intersections. A vertical section derived from the hanging wall and footwall wireframes is shown in Figure 14-1. The disposition of the different veins is illustrated schematically in Figure 14-2.

Figure 14-1: Typical Geological Section for the Curraghinalt Area (Section Orientation is North-South Looking West at X = 257437.5 m)



Note: Vertical and horizontal scale shown by grid axes in metres.

Figure 14-2: Schematic 3D Disposition of Different Mineralized Veins for the Curraghinalt Area (View from Northeast Underneath)



Note: Scale shown by marked Line = 340.0 m.

Basic statistics were calculated for gold for each of the individual veins with statistics calculated for the raw, capped and composited samples. Linear metal accumulation was also calculated for each intersection by multiplying the total length of the samples by grade and statistics estimated for that. The classical statistical results may be seen in Tables 14.3 through 14.10 of Hennessey et al. (2012b).

Samples were selected within the mineralized wireframe for all of the veins. The statistical analysis and the resource estimation were carried out using only these samples.

A study on capping of high-grade gold assays was carried out. There are not enough samples for each sub-zone (the veinlets) to apply capping individually. All the samples from each veinlet were included with the samples from the major veins in order to assess capping for that particular vein.

While a few of the major veins do not have enough samples to create a probability plot, these plots were created for almost all of the major veins and a few of the sub-zones. The inflection point in the probability plot indicates the break in continuity of the distribution for each vein. Capping was applied since these inflections indicate the beginning of the outlier population for the respective veins. The details of grade capping including percentile at which capping has been applied, along with the number of samples affected by capping for each vein, are given in Table 14-2. The probability plots may be seen in Hennessey et al. (2012a or 2012b).

Table 14-2: Grade Capping Including Number of Samples Affected and Percentile at Which Capping is Applied

Vein Code	Total Number of Samples	Capping (g/t Au)	No. of Samples Affected by Capping	Percentile at which Capping is Applied
10	51	72	4	92
20	65	50	7	89
30	102	65	7	93
40	356	110	11	97
50	46	70	1	98
50E	49	70	0	100
60	357	110	9	97
70S	327	85	6	98
80	89	75	4	96
90	118	70	6	95
D	65	75	3	95
G	38	60	2	95
35	24	65	0	100
45	15	110	0	100
55	148	70	5	97
75	147	85	2	99
302	25	65	0	100
302E	12	65	0	100
403	69	110	1	99
DL426B	16	110	0	100
451E	13	110	0	100
552	27	70	0	100
61	30	110	0	100
62	38	110	3	92
601	25	110	0	100
604	46	110	2	96
605	28	110	0	100
606	30	110	0	100
607	55	110	0	100
70N	75	85	0	100
701	52	85	1	98
703	19	85	0	100
708	27	85	0	100
752	22	85	0	100
DL8290	8	75	0	100
40 (UG)	620	110	34	95
60 (UG)	191	110	1	99

The intersection defined by each drill hole for each zone represents the thickness of the quartz vein for that zone. In places, the width of the quartz vein is as low as 10 cm. It is not possible to carry out uniform length compositing with a length of 10 cm since it would generate a huge number of artificial samples, which is not supported by the theory of compositing. A larger composite length on the other hand would dilute the grade of the vein. Under such circumstances, it is better to group all of the samples from a single drill intersection through a vein into one composite. The issue of variable composite length may be overcome by estimating linear metal accumulation rather than grade for the purpose of interpolation.

14.1.2 2011 Mineral Resource Grade Interpolation Method

Estimation Parameters and Search Distances

The density of drilling in the principal zones of mineralization was generally 30 m by 30 m. This drill density, together with good understanding of the geology and mineralization, provides a reasonable level of confidence in the resource estimate. The method of compositing resulted in a single sample point per intersection. This results in a two-dimensional estimation protocol. The method of compositing has resulted in reduction of the nugget effect to a minimum. A very low nugget effect prompted the selection of the inverse distance method of estimation. The power was raised to five so that the variation of grade remains local.

The analysis of classical statistics and compositing has dictated the selection of the estimation parameters. Thirty-five different search ellipses were used in each interpolation. All search ellipses had a length of 30 m in the X and Z direction and 250 m in the Y direction. The rotation about the Z axis ranged from -5° to 25°, and the rotation about the X axis ranged from -20° to -50°. The search parameters remained the same for all search ellipses, and are shown in Table 14-3. The table also shows the enlargement factors for the different interpolation passes used. Individual details are available in Hennessey et al. (2012a or 2012b).

Table 14-3: Summary of Search Parameters

Parameter	Value
Minimum No. of Samples	3
Maximum No. of Samples	5
Search Enlargement Factor	2
Minimum No. of Samples	3
Maximum No. of Samples	5
Search Enlargement Factor	10
Minimum No. of Samples	1
Maximum No. of Samples	5

14.2 Block Model

The block model constructed for the Curraghinalt 2011 resource estimate (Hennessey et al., 2012a) utilized rectangular blocks measuring 10 m (X) by 2,000 m (Y) by 5 m (Z) in height. Although the dimension in the Y direction is unusually long, this was selected based on an understanding of the nature of the transverse sections of the mineralized zones. In order to better conform to the mineralization contacts, Datamine software uses a system of sub-blocking. The sub-blocking method used in Datamine would result in a single block in the Y-direction conforming to the width of the vein, and is consistent with the method of compositing. Blocks were permitted to split in the X and Z directions. This procedure minimizes the volume variance between the wireframe model and the block model. This block size was the most appropriate considering the morphology of the mineralization and the distribution of sample data. The parameters that describe the block model are summarized in Table 14-4.

Table 14-4: Block Model Parameters

Direction	Origin[1]	Block Dimension (m)	No. of Blocks	No. of Sub-Blocks
X	256300 E	10	210	8
Y	385500 N	2,000	1	1
Z	-630	5	210	8

Note: [1] Origin of X and Y axes based on Irish Transverse Mercator Grid; origin of Z axis is at -630 m, and is the altitude from mean sea level.

Block model grade interpolation was performed for gold using inverse distance to the power of 5 in different zones. Concentric search ellipses were used to avoid smearing of grade and for preservation of local variation. Estimations were carried out on gold for both length of the samples and linear metal accumulation. The grades were recalculated from linear metal accumulation by dividing it by the estimated length. This recalculated grade was used for resource reporting in order to carry through the procedure adopted during compositing.

14.3 Block Model Validation

Validation of the block model included the following:

- visual inspection
- comparison of individual block grades with the de-clustered sample grades at two different block sizes
- alternate estimation method.

14.3.1 Visual Inspection

The block model was compared with the drill hole intersections at uniform intervals, both vertically and along cross-section. No visible bias was observed. The cross-section, plan view on 170 m RL (the 170 m level, 60 m below surface) and a three-dimensional (3D) view of the block model are shown in Figure 14-3 through Figure 14-5.

14.3.2 Comparison of Individual Block Grades with De-clustered Sample Grade

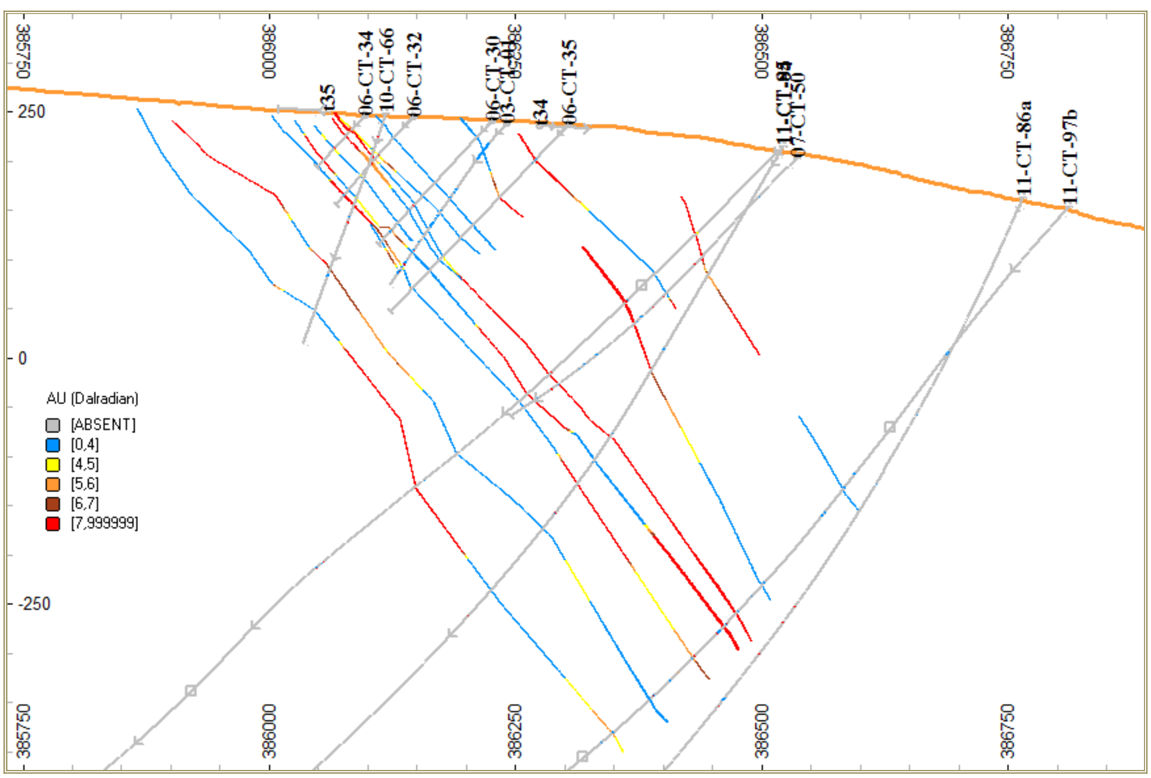
The block model grade interpolation protocol was investigated by inspecting plots of block model grades versus mean borehole sample composite grades that occur within each block. Mean borehole sample composite grades were calculated for each block using the de-clustering technique in which the weighted average grade of composites that fall within a block is calculated. This value is compared with the grade interpolated for the block. A successful grade interpolation protocol will result in block grade estimates that demonstrate a minimum amount of bias. De-clustering was carried out on the block dimensions shown in Table 14-4, also with block dimensions of 30 m (X) and 30 m (Z). This dimension was selected as it represents the drilling density.

Plots comparing composite grades and block model grades for all the veins are presented in Figure 14-6 and Figure 14-7. It is apparent from the plots that there is no significant bias. The de-clustering analysis demonstrates that the mineral resource model provides a reasonable estimate of Curraghinalt mineral resources.

14.3.3 Alternate Estimation Method

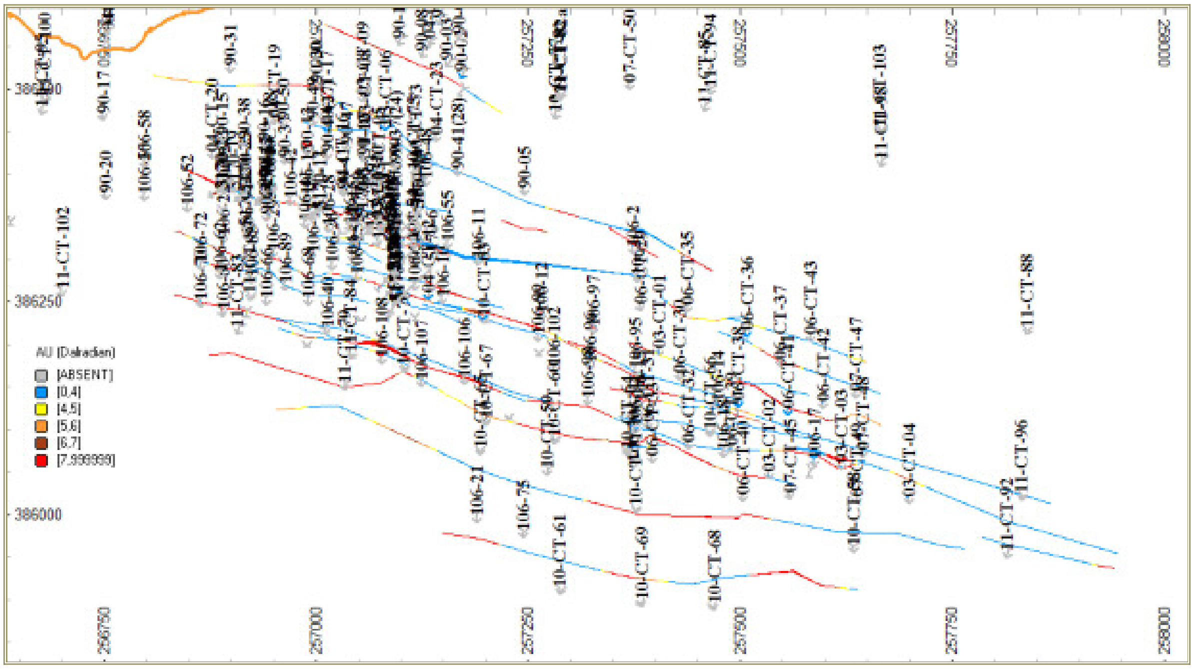
An experimental semi-variogram was calculated using the samples of vein 60 for linear metal accumulation. The variogram model developed, based on the experimental semi-variogram, was used to estimate all the veins using ordinary kriging (OK) (Figure 14-8). All other parameters including dimension and orientation of search ellipse and number of samples used for estimation were kept constant. A scatter plot was produced to compare the grade estimates produced by the two estimation methods, as shown in Figure 14-9. The plot indicates that there may be local variation based on the method of estimation but, globally, they will produce similar results. Based on these three validation checks, it was concluded that the 2011 resource model represents a reasonable grade estimate of the deposit when compared to the drill hole samples.

Figure 14-3: North-south Cross-section Showing Block Model Grades for Mineralized Veins for Curraghinalt Deposit (Section Looking West at X = 257437.5 m)



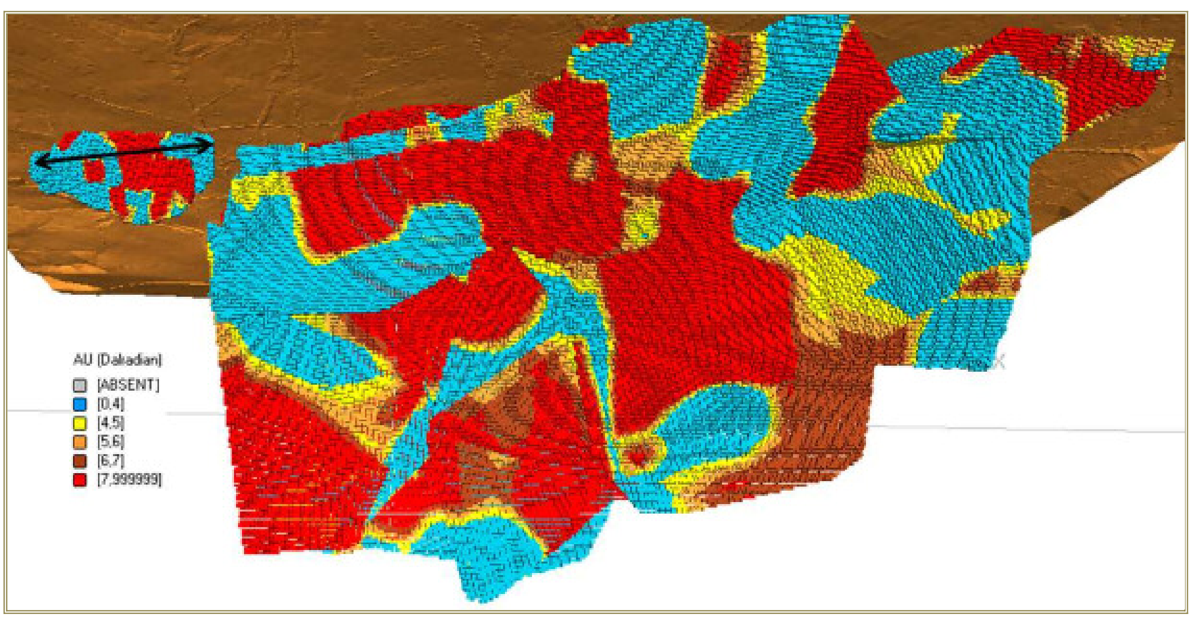
Note: Vertical and horizontal scale shown by grid axes in metres.

Figure 14-4: Plan View at 170 m RL of the Block Model Categorized into Different Grades for Curraghinalt Deposit



Note: Vertical and horizontal scale shown by grid axes in metres.

Figure 14-5: Schematic 3D Rendering of Block Model Categorized into Different Grades for Curraghinalt Deposit (View from Northeast Underneath)



Note: Scale shown by marked line = 340.0 m.

Figure 14-6: Comparison of De-clustered Drill Hole Data with Resource Model Block Grade for Curraghinalt Deposit (Block Dimension of 10 m x width of vein x 5 m)

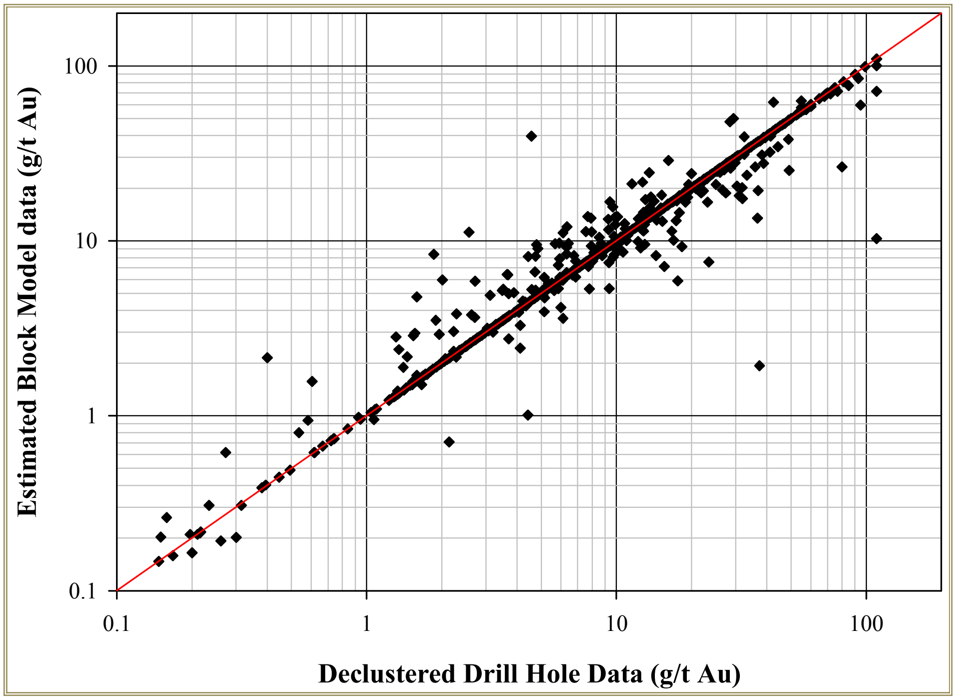


Figure 14-7: Comparison of De-clustered Drill Hole Data with Resource Model Block Grade for Curraghinalt Deposit (Block Dimension of 30 m x width of vein x 30 m)

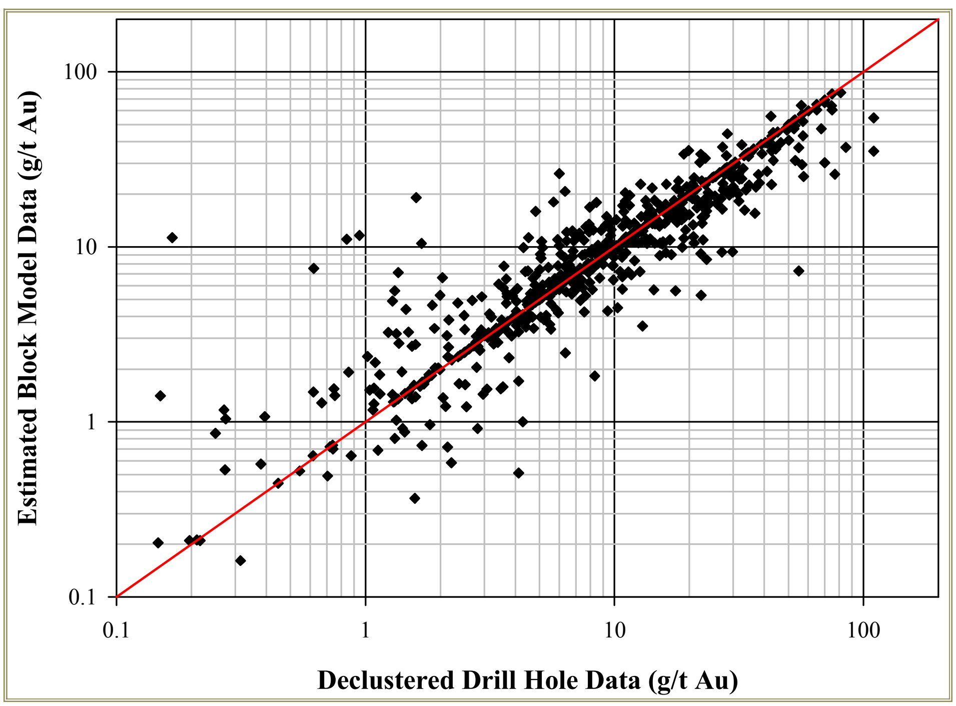


Figure 14-8: Omni-directional Semi-Variogram Model Using Data from Vein 60

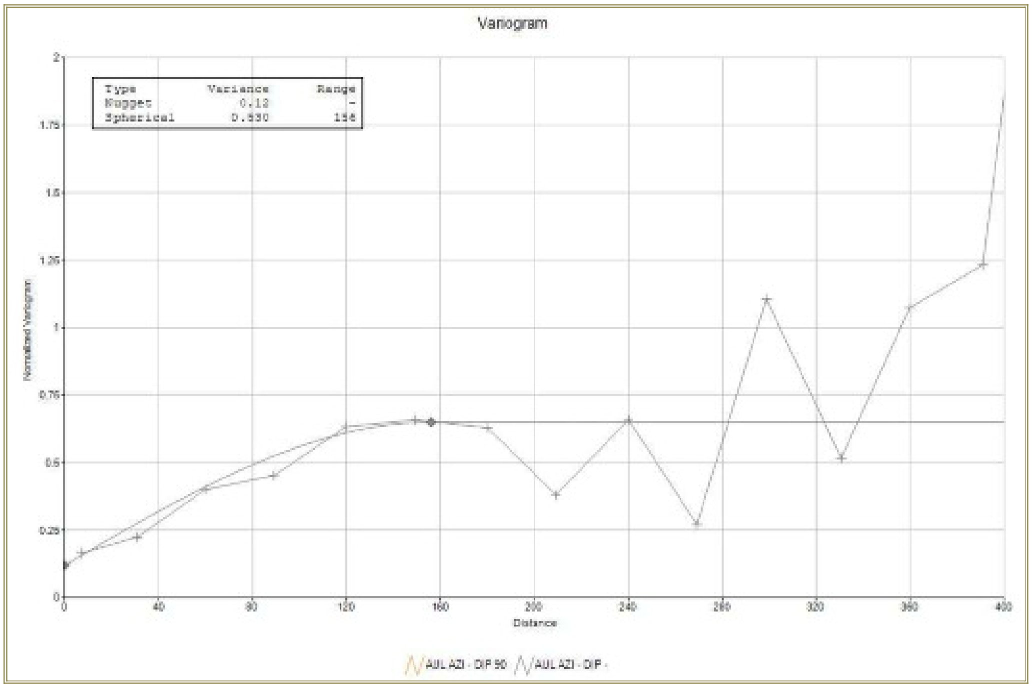
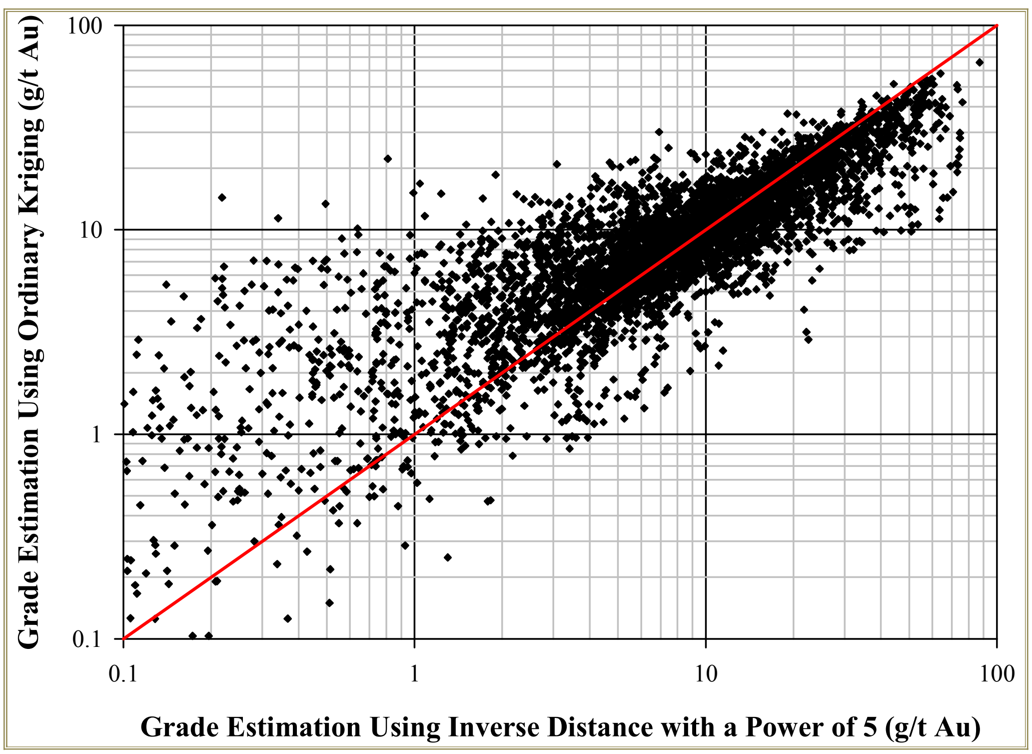


Figure 14-9: Comparison of Two Methods of Estimation



14.4 2011 Mineral Resource Classification

The 2011 mineral resources were estimated following the CIM guidelines. The following definitions were adopted for the classification of mineral resources.

- Measured mineral resources are defined as those portions of the deposit that have an underground channel sample within a 15-m radius.
- Indicated mineral resources are defined as those portions of the deposit estimated with a drill spacing generally defined by 30 m by 30 m, and with a high level of confidence on geological continuity of mineralization. The limit of Indicated resources was considered to be 30 m from the last intersecting drill hole with the drill spacing above.
- Inferred mineral resources are defined as those portions of the deposit for which grade is interpolated utilizing a wider drill spacing, in places with a 100-m radius, or with fewer intersections but with a high level of confidence on the geological continuity of the mineralization. The limit of the Inferred resource was considered to be 30 m from the last drill hole on the periphery.

Only Veins 40 and 60 were explored by one level of underground development. Measured mineral resources are only from these two veins. Most of the major veins have been classified as indicated and inferred resources. Veins 80, 90, D and G have been classed as inferred since there is insufficient drilling on each for them to be included in the indicated category. Sub-zones 55 and 75 have been classified as indicated and inferred resources respectively. All the other sub-zones have been classified as inferred resources. All the veinlets have been classified as inferred except Veins 61, 62, 604, and 701 where, at the upper levels, there is sufficient drilling to classify some portions as indicated resources.

The estimated mineral resources at different cut-off grades and vein thickness are shown in Table 14-5 and Table 14-6. It is evident from the grade tonnage curves (see Hennessey et al., 2012a or 2012b) and the tables that there is little resource associated with blocks of vein thickness less than 0.1 m.

Table 14-5: 2011 Mineral Resource Estimate at Different Grade and Thickness Cut-offs

Original Vein Thickness Cut-off	Grade Cut-off	Measured Resource				Indicated Resource				Measured + Indicated Resource				Inferred Resource			
		Tonnage	Grade	Contained Metal		Tonnage	Grade	Contained Metal		Tonnage	Grade	Contained Metal		Tonnage	Grade	Contained Metal	
(m)	(g/t Au)	(Mt)	(g/t Au)	(t)	(Moz)	(Mt)	(g/t Au)	(t)	(Moz)	(Mt)	(g/t Au)	(t)	(Moz)	(Mt)	(g/t Au)	(t)	(Moz)
0.0	0.0	0.03	14.54	0.46	0.01	2.11	7.57	15.99	0.51	2.14	7.67	16.45	0.53	11.59	7.03	81.47	2.62
0.0	1.0	0.03	17.67	0.46	0.01	1.70	9.33	15.88	0.51	1.73	9.45	16.34	0.53	9.45	8.55	80.80	2.60
0.0	2.0	0.02	18.96	0.46	0.01	1.52	10.25	15.61	0.50	1.55	10.39	16.07	0.52	8.22	9.61	78.96	2.54
0.0	3.0	0.02	19.87	0.45	0.01	1.37	11.11	15.24	0.49	1.39	11.25	15.69	0.50	7.25	10.57	76.56	2.46
0.0	4.0	0.02	20.73	0.45	0.01	1.22	12.08	14.70	0.47	1.24	12.23	15.15	0.49	6.29	11.64	73.24	2.35
0.0	5.0	0.02	21.51	0.44	0.01	1.11	12.84	14.20	0.46	1.13	13.00	14.65	0.47	5.46	12.73	69.48	2.23
0.0	6.0	0.02	22.35	0.44	0.01	0.97	13.86	13.45	0.43	0.99	14.03	13.89	0.45	4.77	13.78	65.73	2.11
0.0	7.0	0.02	23.24	0.43	0.01	0.81	15.28	12.45	0.40	0.83	15.46	12.88	0.41	4.04	15.11	60.97	1.96
0.0	8.0	0.02	24.04	0.43	0.01	0.69	16.68	11.52	0.37	0.71	16.86	11.94	0.38	3.45	16.39	56.55	1.82
0.0	9.0	0.02	24.82	0.42	0.01	0.59	18.15	10.62	0.34	0.60	18.34	11.04	0.36	3.05	17.43	53.16	1.71
0.0	10.0	0.02	25.65	0.41	0.01	0.52	19.31	9.97	0.32	0.53	19.50	10.38	0.33	2.59	18.85	48.82	1.57
0.1	0.0	0.03	14.59	0.46	0.01	2.11	7.57	15.99	0.51	2.14	7.68	16.45	0.53	11.44	7.10	81.29	2.61
0.1	1.0	0.03	17.67	0.46	0.01	1.70	9.33	15.88	0.51	1.73	9.46	16.34	0.53	9.38	8.60	80.66	2.59
0.1	2.0	0.02	18.96	0.46	0.01	1.52	10.26	15.61	0.50	1.55	10.39	16.07	0.52	8.19	9.64	78.87	2.54
0.1	3.0	0.02	19.87	0.45	0.01	1.37	11.11	15.24	0.49	1.39	11.25	15.69	0.50	7.23	10.58	76.51	2.46
0.1	4.0	0.02	20.73	0.45	0.01	1.22	12.08	14.70	0.47	1.24	12.23	15.15	0.49	6.29	11.64	73.22	2.35
0.1	5.0	0.02	21.51	0.44	0.01	1.11	12.84	14.20	0.46	1.13	13.00	14.65	0.47	5.45	12.74	69.44	2.23
0.1	6.0	0.02	22.35	0.44	0.01	0.97	13.86	13.45	0.43	0.99	14.03	13.89	0.45	4.77	13.78	65.73	2.11
0.1	7.0	0.02	23.24	0.43	0.01	0.81	15.28	12.45	0.40	0.83	15.46	12.88	0.41	4.04	15.11	60.97	1.96
0.1	8.0	0.02	24.04	0.43	0.01	0.69	16.68	11.52	0.37	0.71	16.86	11.94	0.38	3.45	16.39	56.55	1.82
0.1	9.0	0.02	24.82	0.42	0.01	0.59	18.15	10.62	0.34	0.60	18.34	11.04	0.36	3.05	17.43	53.16	1.71
0.1	10.0	0.02	25.65	0.41	0.01	0.52	19.31	9.97	0.32	0.53	19.50	10.38	0.33	2.59	18.85	48.82	1.57
1.0	0.0	0.02	18.05	0.40	0.01	1.21	8.61	10.42	0.34	1.23	8.78	10.83	0.35	4.03	9.33	37.63	1.21
1.0	1.0	0.02	20.13	0.40	0.01	1.04	10.02	10.39	0.33	1.06	10.21	10.79	0.35	3.39	11.07	37.51	1.21
1.0	2.0	0.02	21.14	0.40	0.01	0.97	10.62	10.28	0.33	0.99	10.83	10.68	0.34	3.19	11.67	37.21	1.20
1.0	3.0	0.02	21.71	0.40	0.01	0.90	11.19	10.12	0.33	0.92	11.40	10.52	0.34	3.05	12.09	36.86	1.19
1.0	4.0	0.02	22.28	0.40	0.01	0.82	11.99	9.84	0.32	0.84	12.21	10.23	0.33	2.83	12.74	36.11	1.16
1.0	5.0	0.02	22.89	0.39	0.01	0.77	12.49	9.60	0.31	0.79	12.72	10.00	0.32	2.64	13.35	35.24	1.13
1.0	6.0	0.02	23.53	0.39	0.01	0.69	13.31	9.15	0.29	0.70	13.56	9.54	0.31	2.42	14.05	34.07	1.10
1.0	7.0	0.02	24.23	0.39	0.01	0.58	14.65	8.43	0.27	0.59	14.91	8.82	0.28	2.08	15.31	31.80	1.02
1.0	8.0	0.02	24.86	0.38	0.01	0.48	16.03	7.73	0.25	0.50	16.30	8.11	0.26	1.76	16.72	29.38	0.94
1.0	9.0	0.01	25.53	0.38	0.01	0.40	17.59	7.03	0.23	0.41	17.87	7.40	0.24	1.55	17.83	27.61	0.89
1.0	10.0	0.01	26.20	0.37	0.01	0.35	18.73	6.56	0.21	0.36	19.03	6.93	0.22	1.25	19.88	24.75	0.80

Table 14-6: 2011 Mineral Resource Estimate at 5.0 g/t Au Cut-off Grade with Different Vein Thickness

Resource Category	Original Vein Thickness Cut-off (m)	Tonnage (Mt)	Grade (g/t Au)	Contained Metal	
				(t)	(Moz)
Measured	0.0	0.02	21.51	0.44	0.01
	0.1	0.02	21.51	0.44	0.01
	1.0	0.02	22.89	0.39	0.01
Indicated	0.0	1.11	12.84	14.20	0.46
	0.1	1.11	12.84	14.20	0.46
	1.0	0.77	12.49	9.60	0.31
Measured + Indicated	0.0	1.13	13.00	14.65	0.47
	0.1	1.13	13.00	14.65	0.47
	1.0	0.79	12.72	10.00	0.32
Inferred	0.0	5.46	12.73	69.48	2.23
	0.1	5.45	12.74	69.44	2.23
	1.0	2.64	13.35	35.24	1.13

14.5 2011 Mineral Resource Estimate

Micon considered the technical and economic criteria used to estimate a reasonable gold cut-off grade for reporting of the mineral resources. Using these criteria, Micon produced a preliminary estimate for a break-even cut-off grade for reporting mineralization at Curraghinalt considering all factors, including the accessibility and infrastructure. Original vein thickness blocks of less than 0.10 m were eliminated from the resource tabulation. There were minor differences between a 0.00-m and 0.10-m minimum thickness that disappear with rounding. For horizontal vein thicknesses of more than 0.10 m, but less than 1.0 m, the grade was diluted to a minimum horizontal width of 1.0 m prior to reporting.

It was concluded that 5 g/t Au is an appropriate cut-off grade for mineral resource reporting. The mineral resource estimate was updated with the surface topography (veins close to surface were wire framed and modelled beyond the surface). There are two underground drives, one on Vein 40 and the other on Vein 60. The total volume of mined out material is small compared to the volume of these veins and is well within the rounding error of the estimate. The resources as a result have not been depleted for these two workings.

Based on a cut-off grade of 5 g/t Au and the considerations set out above, the Measured mineral resource totalled 0.02 Mt at an average grade of 21.51 g/t Au and the Indicated mineral resources totalled 1.11 Mt at an average grade of 12.84 g/t Au. The Inferred mineral resources totalled 5.45 Mt at an average grade of 12.74 g/t Au. The mineral resources, as first published in Hennessey et al. (2012a), are summarized in Table 14-7.

Table 14-7: Classified 2011 Mineral Resource Estimate at 5 g/t Au Cut-off Grade and 0.1 m Minimum Width, Diluted to a Minimum Width of 1.0 m

Resource Category	Tonnage (Mt)	Grade (g/t Au)	Contained Metal	
			(t)	(Moz)
Measured	0.02	21.51	0.44	0.01
Indicated	1.11	12.84	14.20	0.46
Measured + Indicated	**1.13**	**13.00**	**14.65**	**0.47**
Inferred	5.45	12.74	69.44	2.23

After the public disclosure of the Curraghinalt mineral resource estimate on November 30, 2011, and its publication in Hennessey et al. (2012a), silver and copper grades were estimated into the block model. This was done to allow for the contribution of silver to the cash flow for the PEA and to model copper's effect on cyanide consumption. Silver and copper grades were estimated using the same techniques as described for gold above. No copper concentrate is produced and there is no copper revenue in the base case cash flow. Silver accounts for approximately 1% of revenue in the economic analysis.

The amended Curraghinalt deposit mineral resource estimate showing copper and silver grades is set out in Table 14-8. No changes in tonnages or gold grades have occurred. In order to make it comparable Table 14-8 is also reported at a cut-off grade of 5.0 g/t Au. No gold equivalent calculations were made.

Table 14-8: Curraghinalt 2011 Mineral Resource Estimate Amended to Include Silver and Copper

Category	Notes	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Cu (%)
Measured		0.02	21.51	17.56	0.49
Indicated		1.11	12.84	4.05	0.17
Measured + Indicated		1.13	13.00	4.29	0.18
Inferred	With Cu*	5.38	12.77	5.48	0.12
	Without Cu*	0.08	10.37	2.00	-

Notes: * Copper assay data were not available for the 752 and D veins and no copper grades were estimated there.
1. Original vein thickness blocks of less than 0.10 m were eliminated from the resource tabulation. There were minor differences between a 0.00-m and 0.10-m minimum thickness estimates which disappear with rounding. For horizontal vein thicknesses of more than 0.10 m, but less than 1.0 m, the grades were diluted to a minimum horizontal width of 1.0 m at zero grade prior to reporting.
2. Mineral resources are reported at a cut-off grade of 5 g/t Au after dilution.
3. Mineral resources are not mineral reserves, and do not have demonstrated economic viability.
4. The mineral resource estimate was prepared by Dibya Kanti Mukhopadhyay, MAusIMM (CP), under the overall direction of B. Terrence Hennessey, P.Geo. Mr. Hennessey has taken responsibility for the estimate. The Curraghinalt resource estimate was compliant with the current standards and definitions required under NI 43-101 at the time of estimation.

To the best knowledge of the author the stated mineral resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, unless stated elsewhere in this report. No known mining, metallurgical, infrastructure, or other factors materially affect this mineral resource estimate.

The mineral resource estimate presented herein was originally completed by Dibya Kanti Mukhopadhyay, MAusIMM (CP) under the overall direction of B. Terrence Hennessey, P.Geo who were both co-authors of Hennessey et al. (2012a). In the absence of Mr. Mukhopadhyay, Mr. Hennessey has accepted responsibility for the estimate for this PEA and report. It is compliant with the current standards and definitions required under NI 43-101 and therefore, reportable as a mineral resource by Dalradian Resources.

This mineral resource was estimated from a database that was frozen on October 10, 2011 and is current as of November 30, 2011.

14.6 Current Mineral Resource Estimate

TMAC prepared the current Curraghinalt resource estimate model using MineSight v9.00 desktop software. The model was prepared using interpreted mineralized domains built on minimum two-metre downhole lengths. The effective date of this Mineral Resource is January 20, 2014.

14.7 Drill Hole Database

The drill hole database was compiled by Dalradian Resources and subjected to spot checks and statistical analysis by TMAC. TMAC reports the database is acceptable for use in the resource estimation.

Dalradian Resources implemented a Datashed database to house all drill hole and surficial data. LogChief software is currently used to record logging observations and sampling intervals. The following exported tables were provided to TMAC in the file Curraghinalt_Database_20140120_Clean.xlsx: vwDHCollar, vwDHSurvey, vwDHAssays, tblDHCoreLoss, vwDHLithology, vwDHStructure_Interval, vwDHStructure_Point, vwDHVeins_C_N, vwDHVeins_D_DU_A_UNK_L, tblDHCoreRecovery, tblDHConductivity, tblDHMagSus, TrenchCollar, TrenchAssays and Trench Surveyed.

The vwDHAssays table contained the gold assays used in this resource estimate. The database store assay values below detection limit as negative values so these were converted to positive value and half the lower detection limit was recorded in the field Au_ppm_Positive.

The database provided contained information from 301 underground channel samples and 404 drill holes, totalling 1,304.14 m and 80,143.46 m, respectively. The drill holes were split between 379 surface drill holes and 25 underground drill holes. The average length was 4.3 m for channel samples and 198.4 m for drill holes. The maximum drill hole length was 1,494 m.

Surface trench data was not used in this resource estimate.

14.8 Geological Interpretation

3-D wireframe interpretations were developed by Dalradian Resources in section and reviewed by TMAC. A total of 21 veins were interpreted. These veins are classified as D veins and comprise the primary zones of mineralization. Hanging wall and footwall zones were coded separately. Table 14-9 summarizes the vein codes and corresponding vein name. Figure 14-10 and Figure 14-11 illustrate the veins in plan and section view, respectively. In addition, C Veins are coded in the database but were not included in the current resource estimate except where they are contiguous with D veins.

Table 14-9: Curraghinalt Vein Codes

Vein Name	Vein Code
RoadHW	310
RoadFW	315
SheepDipHW	320
SheepDipFW	325
MullanHW	330
MullanFW	335
T17HW	340
T17FW	345
No1HW	350
No1FW	355
106-16HW	360
106-16FW	365
V75HW	370
V75FW	375
BendHW	380
BendFW	385
CrowFW	395
SCrowFW	397
Attagh1	400
Attagh2	410
AttaghN	420
AttaghN2	430

The drill hole database contains numerous narrow, high grade intersections so the assay table was composited to 1 m length to facilitate the interpretation. A minimum of two samples were used for the interpretation. In some cases, where the vein was very narrow, this necessitated the inclusion of unmineralized intervals.

Figure 14-10: Plan View of Interpreted Veins

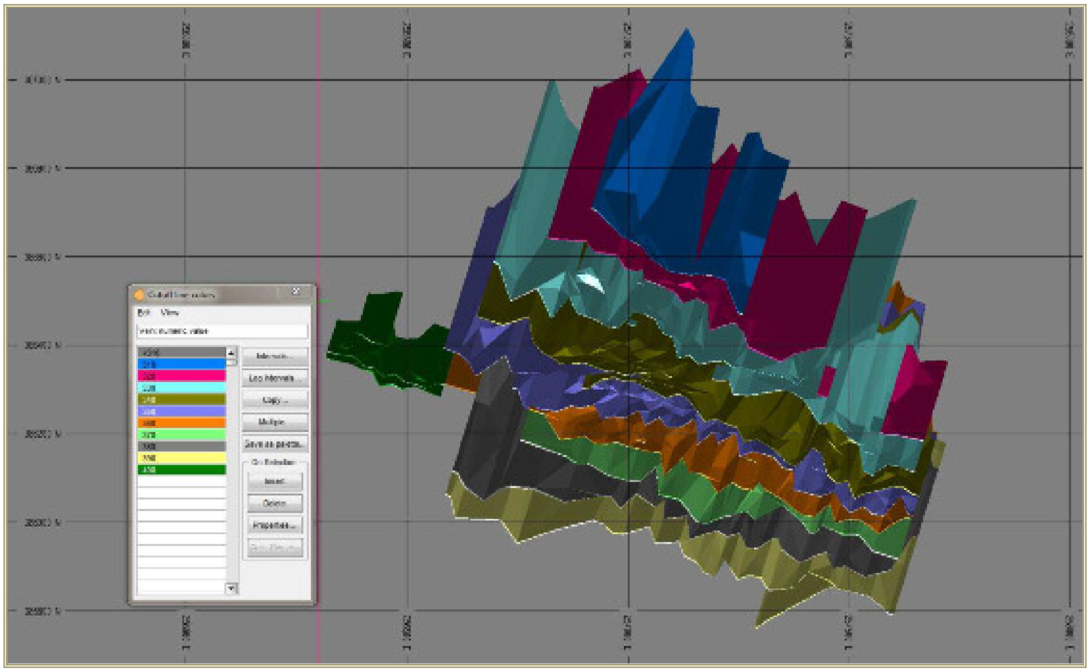
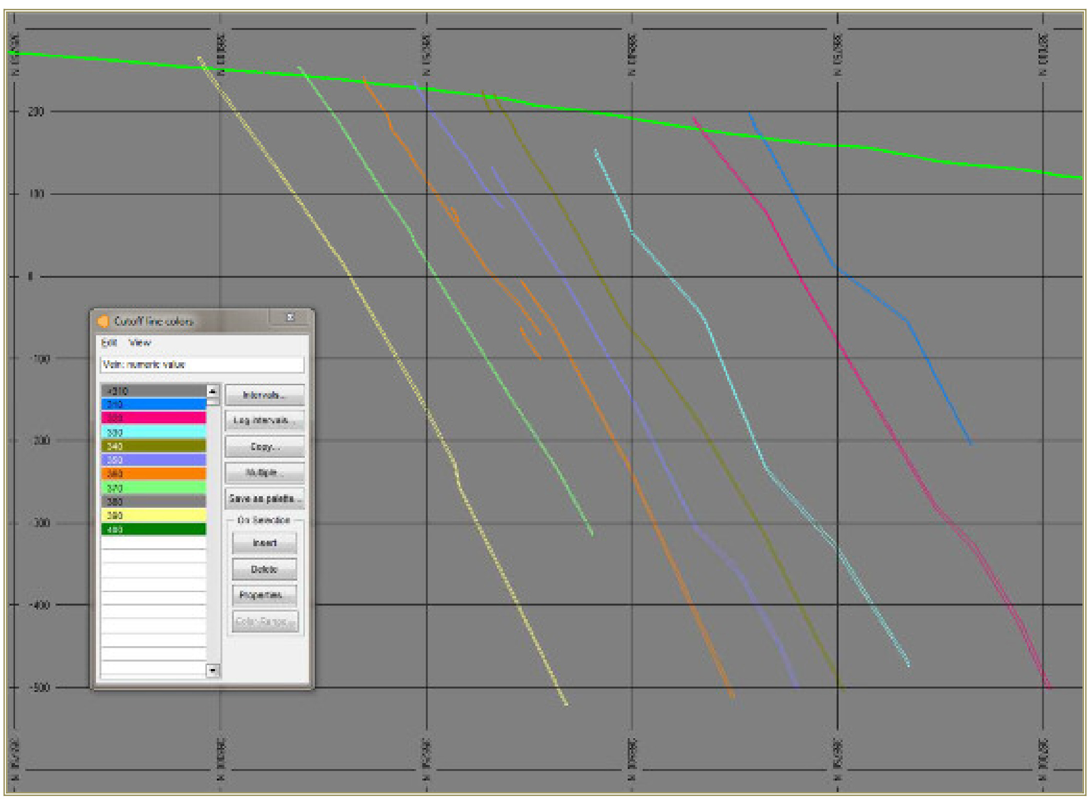


Figure 14-11: Section View Looking West of Curraghinalt Veins (256874N)



14.9 Exploratory Data Analysis

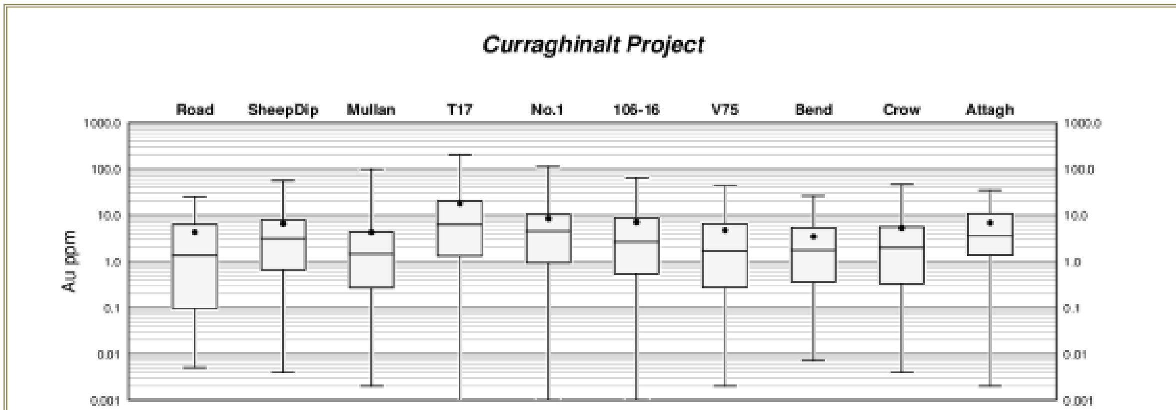
The resource estimate includes gold, silver, and copper. Five percent of the assays did not have a corresponding copper assay and 1% did not have a corresponding silver assay. The exploratory data analysis (EDA) was conducted for all veins and individually by vein wireframe codes.

Box plots for uncapped gold (Figure 14-12), silver (Figure 14-13), and copper (Figure 14-14) grades illustrate the grade distributions by vein. Overall, the veins demonstrate similar grade statistics, which supports grouping less sampled veins as required.

Figure 14-15, Figure 14-16, and Figure 14-17 illustrate the grade distributions through histogram and probability plots. The probability plot indicates that there is some mixing of gold grade populations (or effectively including internal dilution) as a result of the vein interpretation method. There appears to be a low-grade population below 1 g/t Au to 2 g/t Au and a higher-grade population above 100 g/t Au. The silver grade distribution reflects two populations: one low grade below 0.25 g/t Au and the other above. For copper, a single lognormal population is reflected in the probability plot.

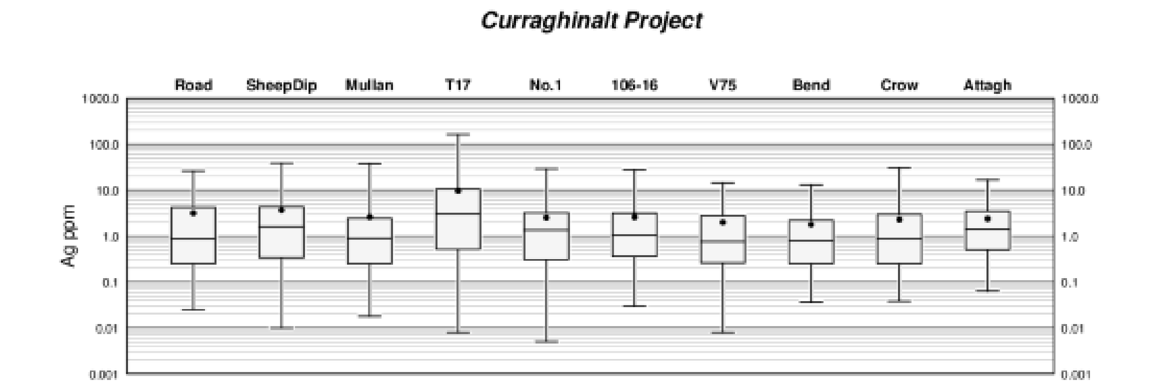
Figure 14-18 provides statistics on sample length and reports that about 95% of the samples are 1.0 m in length and 97.5% greater than 0.5 m. The gold grade distributions were filtered and reviewed for sample lengths less than and greater 0.5 m. It was found that high-grade samples were prevalent in the less than 0.5 m composite so these intervals were not excluded from the resource estimation. Therefore, the selection of composites was not filtered based on sample length for resource estimation in order to include these samples.

Figure 14-12: Gold Box Plot



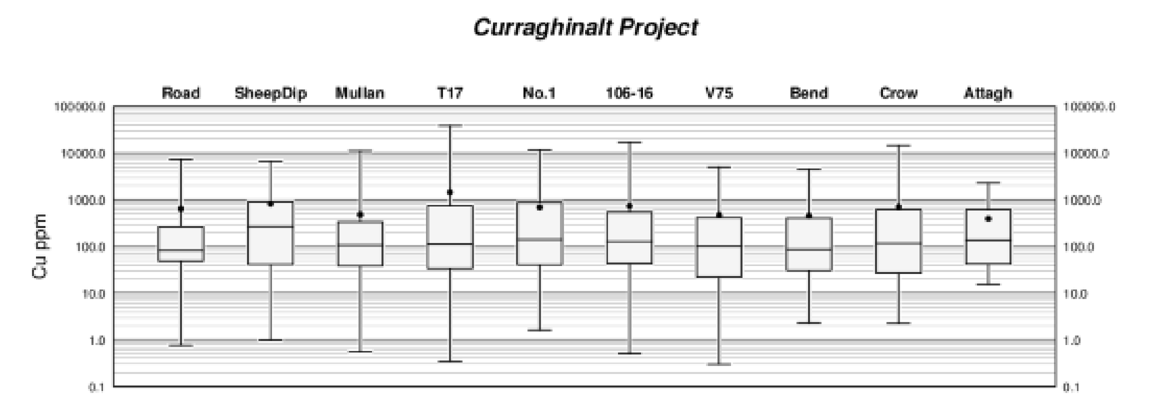
	Road	SheepDip	Mullan	T17	No.1	106-16	V75	Bend	Crow	Attagh	
Number of data	39	118	200	596	460	318	117	91	109	104	Number of data
Mean	4.378	6.639	4.364	18.237	8.423	7.134	4.822	3.469	5.37	6.874	Mean
Maximum	25.325	57.708	99.232	206.295	111.3	65.369	43.7	26.139	47.555	34.526	Maximum
Upper quartile	6.371	7.927	4.393	20.454	10.292	8.775	6.658	5.437	5.647	10.692	Upper quartile
Median	1.375	3.1	1.478	6.342	4.624	2.625	1.718	1.759	1.986	3.607	Median
Lower quartile	0.095	0.658	0.268	1.345	0.925	0.546	0.272	0.364	0.336	1.384	Lower quartile
Minimum	0.005	0.004	0.002	0.0	0.001	0.001	0.002	0.007	0.004	0.002	Minimum
Variance	40.6327	88.4398	89.4177	921.307	160.383	121.044	51.801	21.1562	65.685	56.9619	Variance
CV	1.456	1.417	2.167	1.664	1.504	1.542	1.493	1.326	1.509	1.098	CV
Skewness	1.923	2.598	6.09	3.167	3.664	2.862	2.532	2.4	2.448	1.622	Skewness

Figure 14-13: Silver Box Plot



	Road	SheepDip	Mullan	T17	No.1	106-16	V75	Bend	Crow	Attagh	
Number of data	39	118	200	496	353	308	117	91	108	78	Number of data
Mean	3.173	3.676	2.611	9.679	2.523	2.591	1.996	1.785	2.309	2.397	Mean
Maximum	26.441	38.602	37.822	166.3	29.495	27.65	14.415	13.084	30.558	16.738	Maximum
Upper quartile	4.236	4.371	2.438	10.456	3.215	3.13	2.748	2.224	2.958	3.44	Upper quartile
Median	0.858	1.591	0.892	2.98	1.35	1.049	0.765	0.79	0.87	1.421	Median
Lower quartile	0.25	0.341	0.25	0.521	0.304	0.363	0.259	0.25	0.25	0.5	Lower quartile
Minimum	0.025	0.01	0.018	0.008	0.005	0.03	0.008	0.036	0.038	0.063	Minimum
Variance	25.648	32.2035	22.1483	332.336	11.3092	16.8101	6.755	6.0982	16.3258	7.8242	Variance
CV	1.596	1.544	1.803	1.884	1.333	1.582	1.302	1.384	1.75	1.167	CV
Skewness	2.839	3.21	3.8	4.156	3.153	3.289	2.103	2.512	4.252	2.496	Skewness

Figure 14-14: Copper Box Plot



	Road	SheepDip	Mullan	T17	No.1	106-16	V75	Bend	Crow	Attagh	
Number of data	35	83	167	273	321	290	111	89	105	47	Number of data
Mean	640.064	825.399	482.691	1450.05	690.404	735.731	474.226	453.815	711.2	392.877	Mean
Maximum	7347.79	6590.74	11169.1	38461.9	11647.7	17375.0	5049.31	4533.39	14455.5	2246.91	Maximum
Upper quartile	266.772	901.974	333.768	745.126	880.135	546.011	416.344	403.595	613.348	616.002	Upper quartile
Median	81.44	289.49	106.001	112.32	142.986	127.549	100.28	86.5	116.699	137.86	Median
Lower quartile	49.118	41.08	38.47	33.761	40.605	43.14	22.478	30.395	27.112	42.42	Lower quartile
Minimum	0.765	0.979	0.55	0.35	1.62	0.5	0.3	2.3	2.25	15.58	Minimum
Variance	2.13929e+06	1.77993e+06	1.63542e+06	1.94611e+07	1.84724e+06	4.06095e+06	829328.0	772220.0	2.9069e+06	282162.0	Variance
CV	2.285	1.616	2.649	3.042	1.969	2.739	1.92	1.936	2.397	1.352	CV
Skewness	3.241	2.386	5.486	5.622	4.573	5.625	2.948	2.923	5.535	1.848	Skewness

Figure 14-15: Gold Histogram and Probability Plots

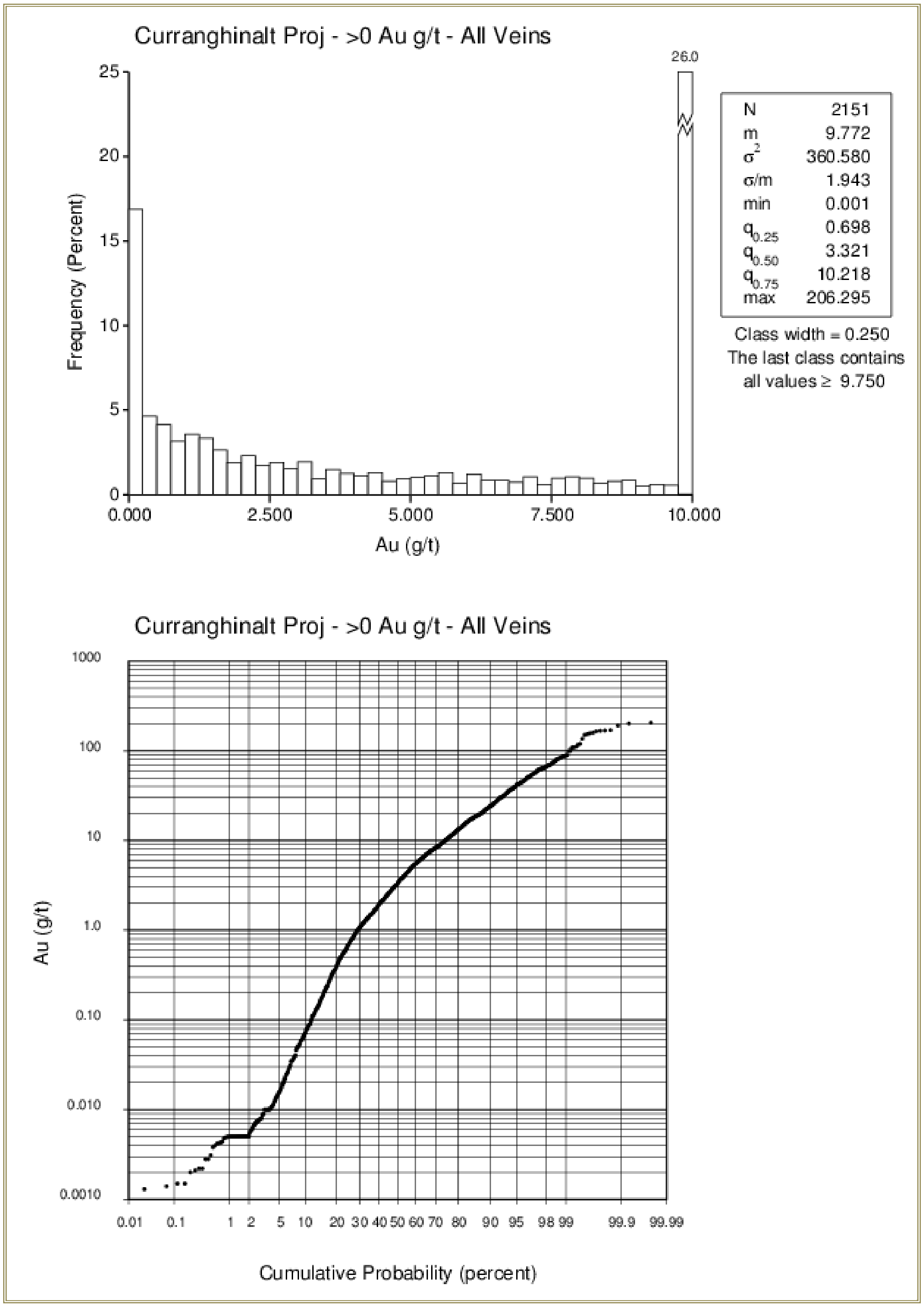


Figure 14-16: Silver Histogram and Probability Plots

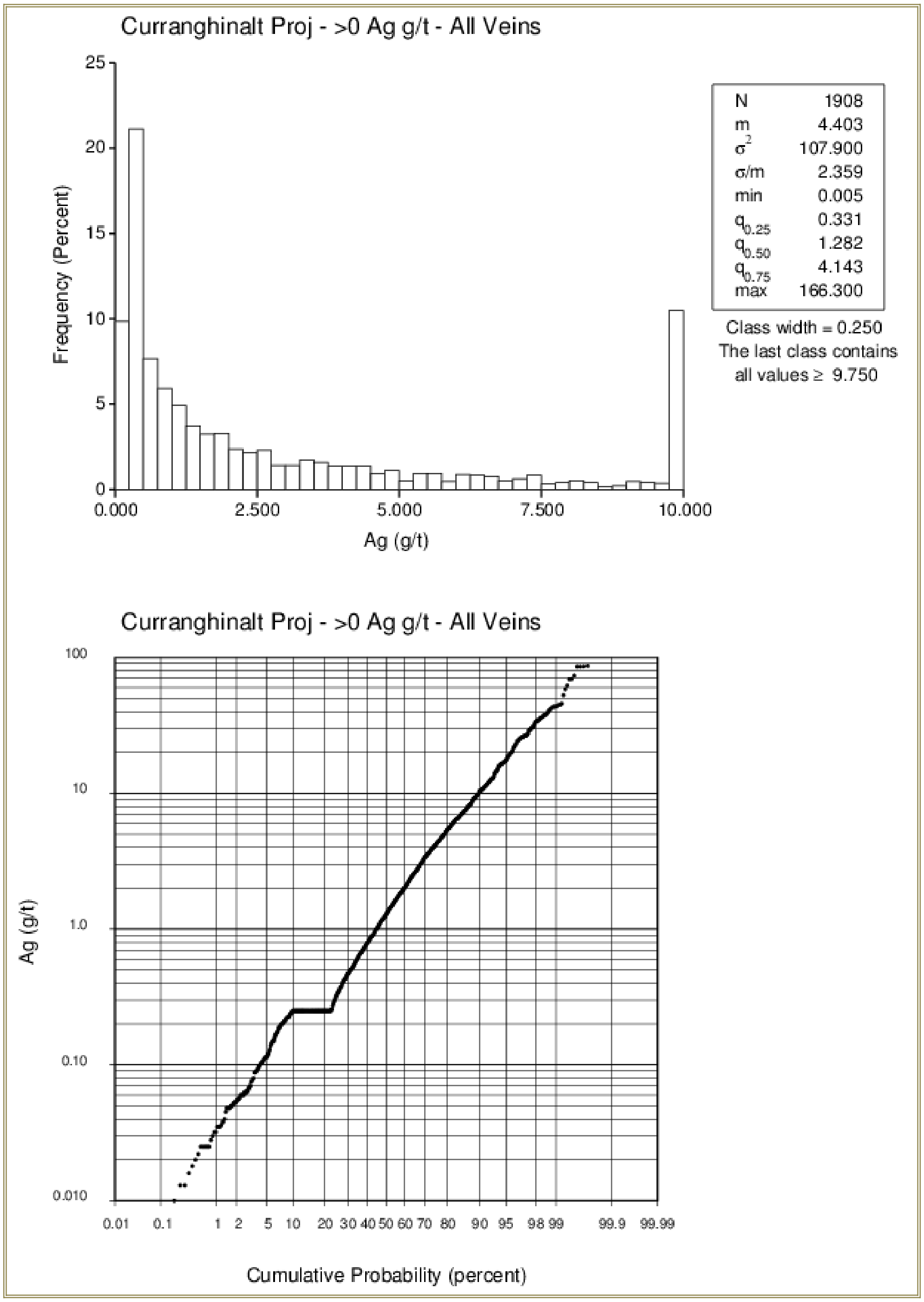


Figure 14-17: Copper Histogram and Probability Plots

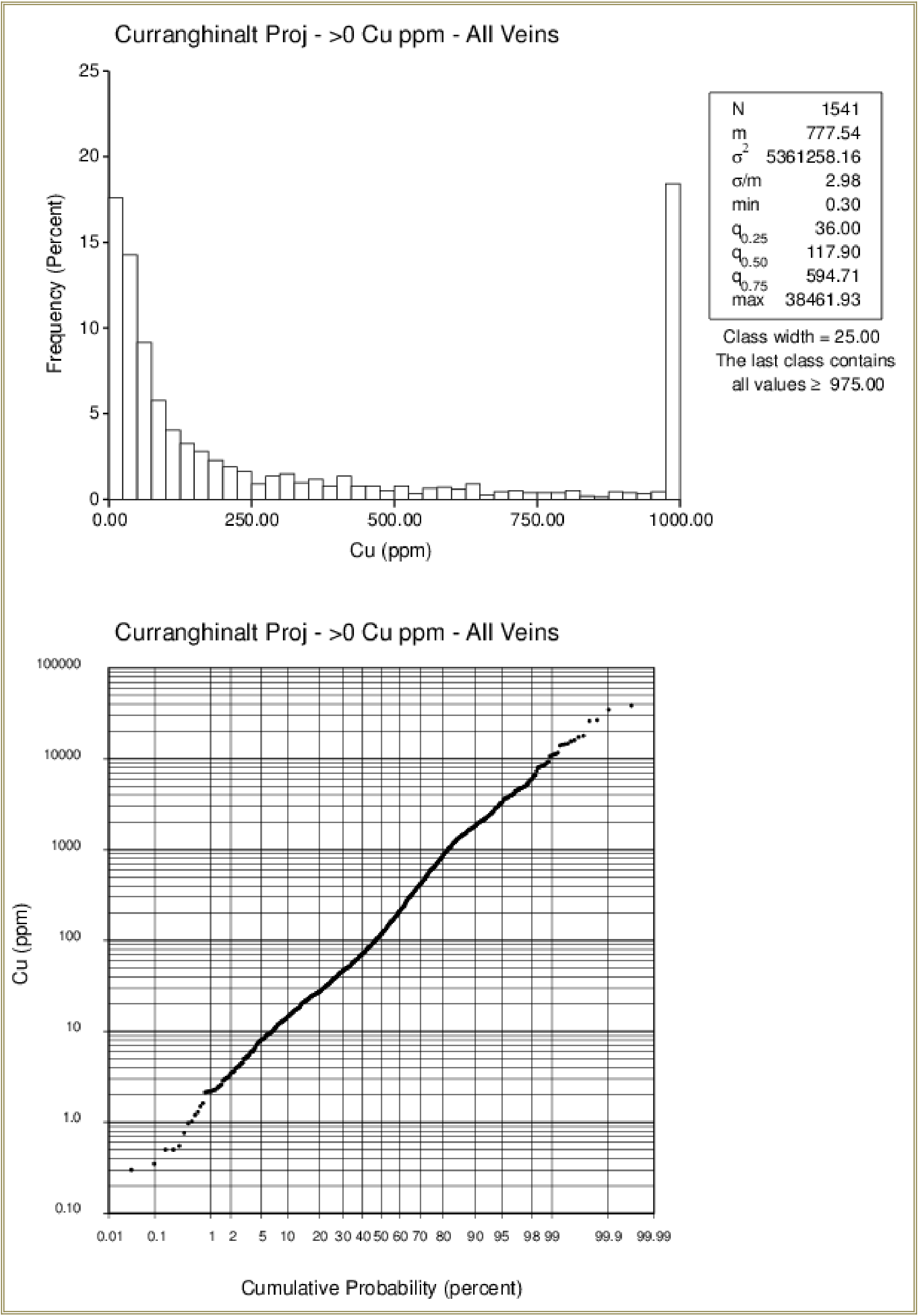
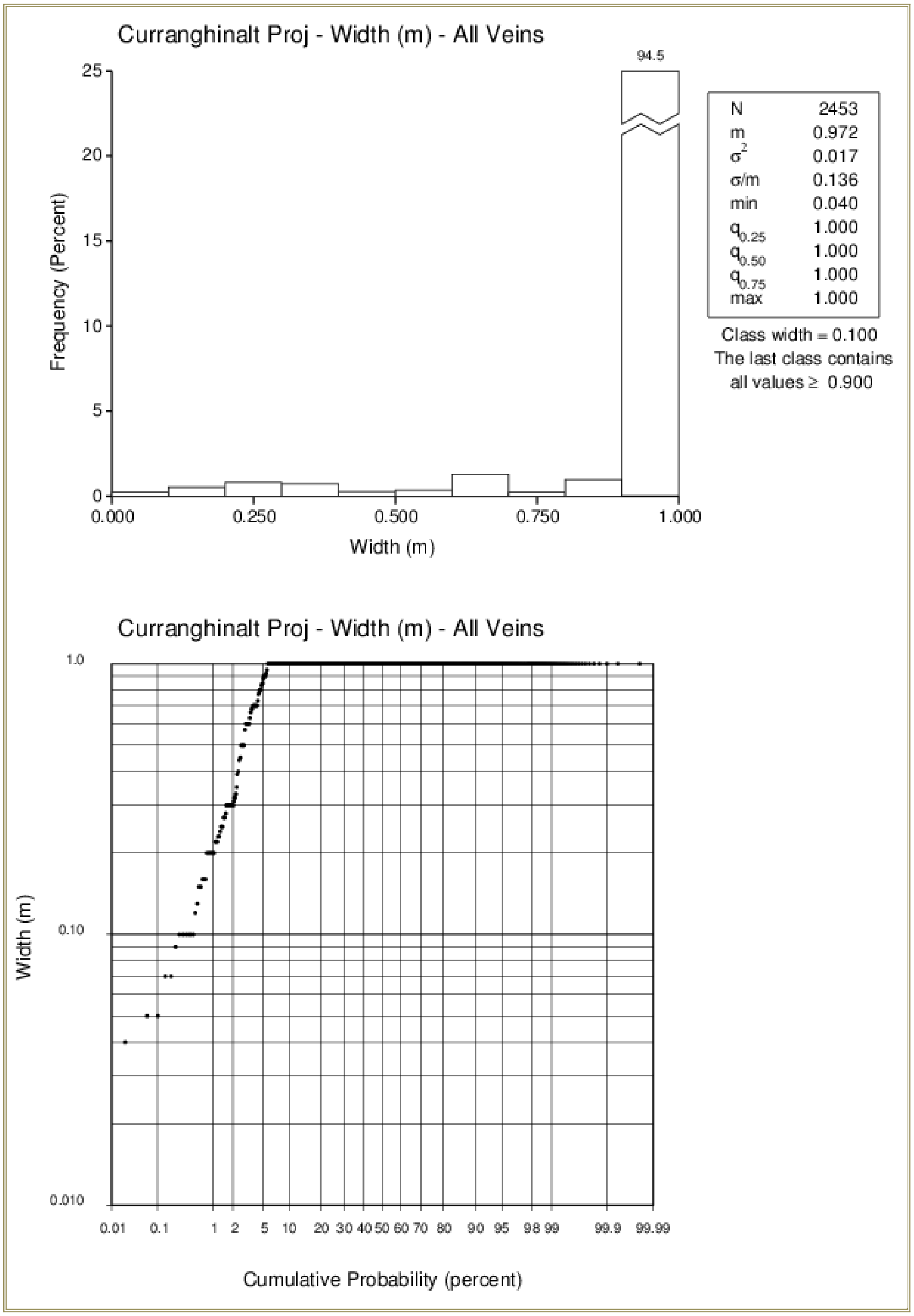


Figure 14-18: Sample Width Histogram and Probability Plots



14.10 Capping

In mineral deposits having skewed distributions (typically with coefficient of variation greater than 1.0), a few high-grade assays can represent a large portion of the metal content. Often there is little continuity demonstrated by these assays. In other words, it can be assumed they occur at random within the deposit.

The global CV for gold is 1.94 (Figure 14-15) and 2.36 and 2.98 for silver (Figure 14-16) and copper (Figure 14-17) respectively. Other methods such as the Parrish method and disintegration analysis also confirmed capping is required at Curraghinalt. The Parrish method (Parrish, 1997) is based on decile analysis of the metal distribution as related to the assay frequency distribution with the sample population. Disintegration analysis uses a 15% step function to denote the changes in an ordered (ranked) data to supplement the interpretation of statistical graphics, such as a probability plot.

Table 14-10, Table 14-11, and Table 14-12 summarize the capping analysis and provide the recommended cap grades. T17 has the most capped samples but 14 of the 19 samples are from high-grade channel samples.

Capping was applied after creating the one-metre composites.

Table 14-10: Gold Capping Analysis Summary (g/t)

Vein	Count	Maximum Value	Probability Plot	Disintegration Analysis	Parrish Method	Recommended Capped Grade	# Capped	Percentile (%)
Road	42	25.33	10.0	-	40.0	25.0	1	2
SheepDip	134	57.71	25.0	-	40.0	30.0	3	2
Mullan	226	99.23	30.0	20.0	30.0	30.0	4	2
T17	675	206.30	105.0	119.0	160.0	90.0	19	3
No. 1	511	111.30	55.0	52.0	-	52.0	9	2
106-16	350	65.37	50.0	40.0	55.0	50.0	5	1
V75	135	43.70	20.0	-	40.0	30.0	2	1
Bend	112	26.14	15.0	-	40.0	25.0	1	1
Crow	123	47.56	18.0	-	40.0	30.0	2	2
Attagh	139	34.53	20.0	-	40.0	30.0	2	1

Table 14-11: Silver Capping Analysis Summary (g/t)

Vein	Count	Maximum Value	Probability Plot	Disintegration Analysis	Parrish Method	Recommended Capped Grade	# Capped	Percentile (%)
Road	39	26.44	3.2	10.0	-	10.0	4	10
SheepDip	118	38.60	3.7	25.0	-	25.0	2	2
Mullan	200	37.82	2.6	18.0	-	18.0	3	2
T17	496	166.30	9.7	55.0	-	55.0	15	3
No. 1	353	29.50	2.5	15.0	-	15.0	5	1
106-16	308	27.65	2.6	20.0	-	20.0	4	1
V75	117	14.42	2.0	9.0	-	9.0	5	4

Vein	Count	Maximum Value	Probability Plot	Disintegration Analysis	Parrish Method	Recommended Capped Grade	# Capped	Percentile (%)
Bend	91	13.08	1.8	9.0	-	9.0	4	4
Crow	108	30.56	2.3	6.0	-	6.0	9	8
Attagh	78	16.74	2.4	10.0	-	10.0	2	3

Table 14-12: Copper Capping Analysis Summary (ppm)

Vein	Count	Maximum Value	Probability Plot	Disintegration Analysis	Parrish Method	Recommended Capped Grade	# Capped	Percentile (%)
Road	35	7,347.8	640.1	4,000.0	-	4,000.0	1	3
SheepDip	83	6,590.7	825.4	4,500.0	-	4,500.0	5	6
Mullan	187	11,169.1	482.7	6,000.0	-	6,000.0	3	2
T17	273	38,461.9	1450.1	6,000.0	-	6,000.0	13	5
No. 1	321	11,647.7	690.4	4,500.0	-	4,500.0	7	2
106-16	290	17,375.0	735.7	4,500.0	-	4,500.0	8	3
V75	111	5,049.3	474.2	3,000.0	-	3,000.0	3	3
Bend	89	4,533.4	453.8	2,000.0	-	2,000.0	5	6
Crow	105	14,455.5	711.2	5,000.0	-	5,000.0	2	2
Attagh	47	2,246.9	392.9	1,000.0	-	1,000.0	5	11

Table 14-13 summarizes the metal removed by capping at a 5.0 g/t Au cut-off. For Measured+Indicated resource, a total of 19% or 186,000 oz Au was removed. For Inferred resource, 13% or 316,000 oz was removed.

Table 14-13: Metal at Risk Reduction

Resource Class	Tonnes (Kt)	AUK (g/t)	AUK (koz)	AUCK (g/t)	AUCK (koz)	Difference (koz)
MEA+IND	2,999.5	10.41	1,004.1	8.48	818.2	-19%
INF	8,005.7	9.67	2,487.7	8.44	2,171.3	-13%

14.11 Variography

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. One of these is the correlogram, which 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 is indicative of short scale variability. A more gradual decrease moving away from the origin suggests more short scale continuity. A plot of 1-correlation is made so the result looks like the more familiar variogram plot.

The approach used to develop the variogram models employed Sage2001© software. Directional sample correlograms were calculated along horizontal azimuths of 0, 30, 60, 120, 150, 180, 210, 240, 270, 300, and 330 degrees. For each azimuth, sample correlograms were also calculated at dips of 30 and 60 degrees in addition to horizontally. Lastly, a correlogram was calculated in the vertical direction. Using the thirty-seven sample correlograms an algorithm determined the best-fit model nugget effect and two-nested structure variance contributions. After fitting the variance parameters, the algorithm then fitted an ellipsoid to the thirty-seven ranges from the directional models for each structure. The anisotropy of the correlation was given by the range along the major, semi-major, and minor axes of the ellipsoids and the orientations of these axes for each structure. The fitted variogram was then reviewed by TMAC and adjusted to reflect the mineralization.

An exponential model was used for the variogram. The traditional exponential range R is defined as Gam(3R) = 0.95 * Sill. The variogram structure is summarized in Table 14-14.

Table 14-14: Variogram Structure

Nugget	Sill	Rot. Z	Rot. X'	Rot. Y'	Range X' (m)	Range Y' (m)	Range Z' (m)
0.370	0.527	-20.00	26.00	43.00	2.13	4.32	21.17
	0.103	103.00	59.00	2.00	10.46	481.87	386.95

All conventions follow those of the Cartesian Coordinate System. For example, the system of axes is oriented so that:

1. the X axis runs east/west with values increasing to the east
2. the Y axis runs north/south with values increasing to the north.

Then, the Z axis will be vertical with values increasing upward. A positive dip angle is measured upwards from the horizontal, while a negative dip angle is measured downwards from the horizontal.

The order and direction of these rotations around the three axes are given by:

1. the first rotation is around the Z axis. The direction is given by the left hand rule
2. the second rotation is around the rotated X axis. The direction is given by the right hand rule
3. the third rotation is around the rotated Y axis. The direction is given by the right hand rule.

14.12 Resource Block Model

14.12.1 Model Limits

The current block model was based on a block size of 6 m Easting x 3 m Northing x 3 m Elevation. The model was rotated 17.091° in 2-dimensions at the rotation origin 256100 East, 386000 North.

MineSight v9.00 desktop software was used for resource estimation. MineSight maintains two coordinate systems for a rotated model: the project bounds and model limits. These are tabulated in Table 14-15 and Table 14-16. Figure 14-19 illustrates the positioning of the coordinate systems graphically.

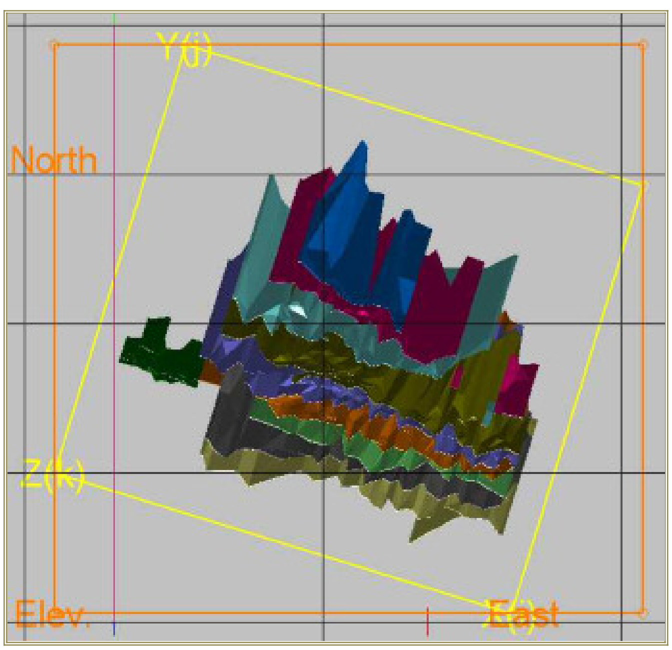
Table 14-15: MineSight Project Bounds

Description	Coordinate	Minimum	# of Blocks
Project Bounds	Easting	256100	258072
	Northing	385529	387434
	Elevation	-600	369

Table 14-16: MineSight Model Limits

Description	Coordinate	Minimum	Maximum	Block Size	# of Blocks
Model Limits	X	0	1602	6	267
	Y	0	1500	3	500
	Z	-600	369	3	323

Figure 14-19: Graphic Representation of MineSight Models



The block model was also sub-blocked six times in each direction. The smallest block dimension is 1.00 m X, 0.50 m Y, and 0.50 m Z.

Table 14-17 summarizes the fields in the block model MINE15.DAT.

Table 14-17: MineSight Block Model Description

Item	Description
TOPO	% below topographic surface
ORE	Vein code
ORE%	% of block within vein wireframe
AUCK	Kriged – Au Capped g/t
AUK	Kriged – Au g/t
CUCK	Kriged – Cu Capped ppm
CUK	Kriged – Cu ppm
AGCK	Kriged – Ag Capped g/t
AGK	Kriged – Ag g/t
AUCID	IDW3 – Au Capped g/t
CUCID	IDW3 – Cu Capped ppm
AGCID	IDW3 – Ag Capped g/t
AUCNN	NN – Au Capped g/t
CUCNN	NN – Cu Capped ppm
AGCNN	NN – Ag Capped g/t
CMPK	Number of composites used for OK estimation
DISOK	Distance to nearest composite used for OK estimation
AVGOK	Average distance to all samples used for OK estimation
MAXOK	Maximum distance to sample used for OK estimation
OCTOK	Not used
DDHOK	Number of drill holes used for OK estimation
KRVAR	Not used
DISNN	Distance to nearest composite used for NN
RLCSS	Resource classification
DENS	Density, 2.85 g/cc
TEMP1	Not used
TEMP2	Not used
TEMP3	Not used
AUCD5	IDW5 – Au Capped g/t
NCID	Number of composites used for IDW estimation

14.12.2 Specific Gravity

Specific gravity is discussed in Section 11.4. Based on the analysis by TMAC of the specific gravity available, an average value of 2.85 g/cc was used for all veins in the block model.

14.12.3 Grade Interpolation

Grade interpolation was conducted using ordinary kriging (OK), nearest neighbour (NN) and inversed distance weighting (IDW) methods. The OK grade interpolation used search ellipses as defined in Table 14-18. These parameters were based on the geological interpretation and variogram analysis. A two-pass strategy was used for each vein. Similar search ellipses were used for IDW grade estimation. For the NN estimation, only the first pass (larger search) was used.

The number of composites used in estimating a model grade followed a strategy that matched composite values and model blocks sharing the same vein code for both passes. For the first pass, there was no minimum number of drill holes as the second pass, which has a shorter search range in MineSight, overwrites the first past blocks. The second pass used a minimum of two drill holes to estimate the grade. Estimates used a maximum of eight composites for both passes.

The first pass ranges were 200 m x 100 m x 200 m and the second pass 100 m x 50 m x 100 m. The search rotation convention is GSLIB-MS, which uses a ZXY Left-Right-Left rotation.

Table 14-18: OK Search Strategy

Vein	Pass	Elevation (m)	Rot Z	Rot X	Rot Y
RoadHW	R01	> - 10	130	-66	0
	R02	> - 10	130	-66	0
	R03	≤ - 10	130	-50	0
	R04	≤ - 10	130	-50	0
Sheep Dip	D01	-	125	-62	0
	D02	-	125	-62	0
MullanHW	M01	-	116	-63	0
	M02	-	116	-63	0
MullanFW	M03	-	116	-54	0
	M04	-	116	-54	0
No1HW	N01	-	112	-60	0
	N02	-	112	-60	0
No1FW	N03	-	112	-55	0
	N04	-	112	-55	0
V75HW	V01	-	109	-53	0
	V02	-	109	-53	0
V75FW	V03	-	109	-58	0
	V04	-	109	-58	0
T17HW	T01	-	130	-64	0

Vein	Pass	Elevation (m)	Rot Z	Rot X	Rot Y
	T02	-	130	-64	0
T17FW	T03	-	130	-53	0
	T04	-	130	-53	0
106-16HW	Z01	-	109	-64	0
	Z02	-	109	-64	0
106-16FW	Z03	-	109	-53	0
	Z04	-	109	-53	0
BendHW/FW	B01	-	106	-60	0
	B02	-	106	-60	0
CrowFW	C01	> - 200	106	-52	0
	C02	> - 200	106	-52	0
	C03	≤ - 200	106	-57	0
	C04	≤ - 200	106	-57	0
SCrowFW	C05	-	106	-60	0
	C06	-	106	-60	0
AttaghN/1/2	A01	-	109	-67	0
	A02	-	109	-67	0

14.12.4 Model Verification and Validation

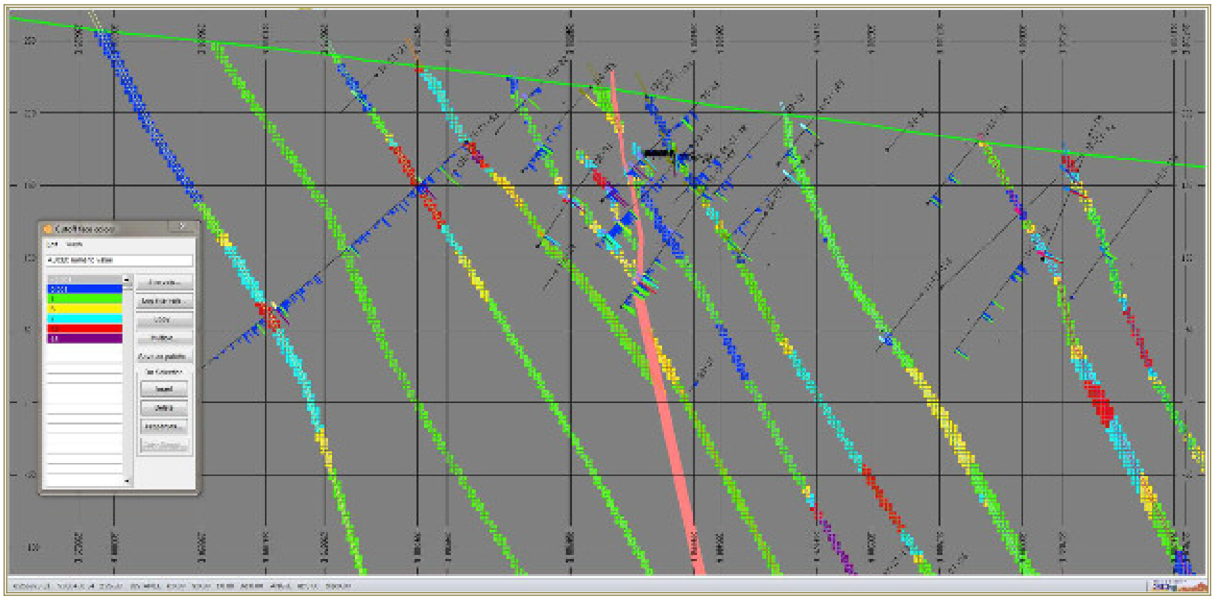
TMAC distinguishes between verification from validation as follows:

- Verification is a manual (e.g., visual inspection) or quasi-manual (e.g., spreadsheet) check of the actual procedure used
- Validation is a test for reasonableness using a parallel procedure, which may be either manual or a computer-based procedure (e.g., different interpolation methods).

Visual Checks

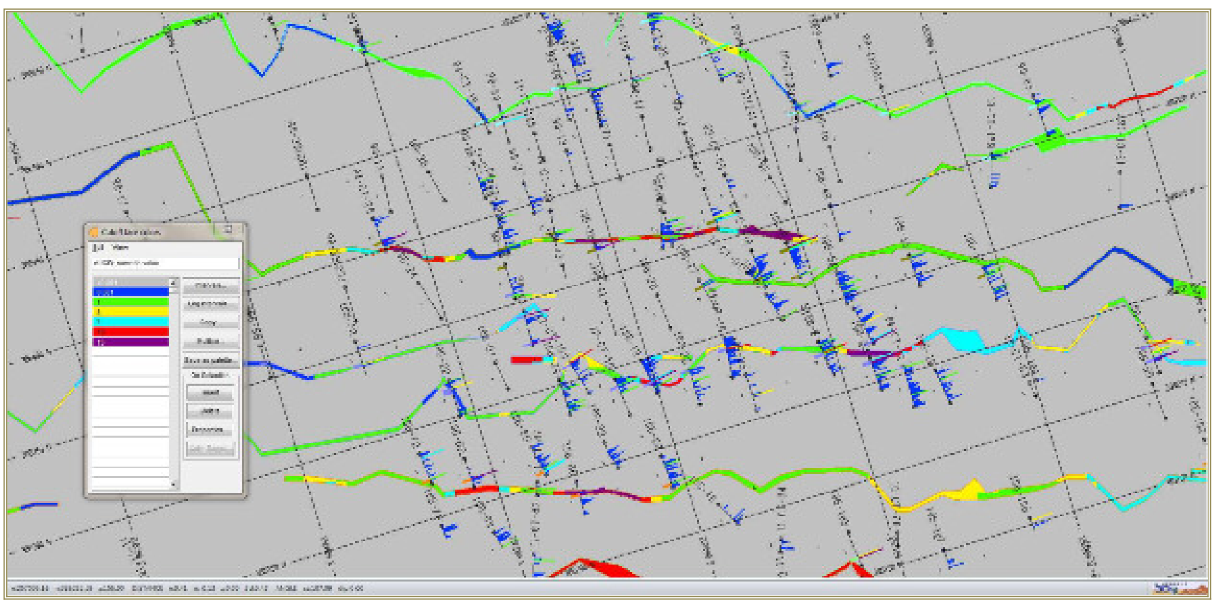
Interpolated block grades, resource classification, geological interpretation outlines, and drill hole composite intersections were verified on screen for plan and section views. Based on the visual inspection by TMAC conducted with Dalradian Resources, the block model grades appeared to honour the data well. Representative views are shown in Figure 14-20 and Figure 14-21. The interpolated block model grades exhibit satisfactory consistency with the drill hole composites.

Figure 14-20: Section View – AUCID Block Model Verified by Capped Au Composites



Note: Rotated MineSight Grid: Rot_NS50m. Section: Sectional az=17.09 132059.7

Figure 14-21: Plan View - AUCID Block Model Verified by Capped Au Composites



Note: Rotated MineSight Grid: Plan_RotN. Elevation 156.0 m. Looking North 17.09° East

Global Grade Comparison

TMAC verified the block model grade estimates for global grades by comparing the average gold grades (with no cut-off grade) from the IDW, OK, and NN estimates. The NN estimator produces a theoretically unbiased estimate of the average value when no cut-off is imposed and is a good basis for checking the performance of the different estimation methods. The results (Table 14-19) show no evidence of bias in the estimate. The IDW grade, which is the basis for the resource statement, lies between the grade estimate using OK and NN.

Table 14-19: Global Grade Verification

Estimation Method	Capped Au (g/t)
IDW	4.52
OK	4.62
NN	4.43

Histogram and probability plots were created of the block mode data and reviewed with respect to the corresponding composite data. Table 14-20 summarizes the composite versus block model statistics. Potential indications of grade bias reflect the poor sampling density of some veins, statistics generated by block count rather than tonnage weighted and because the blocks were not filtered based on level of confidence (resource classification).

Table 14-20: Statistical Grade Comparison, Capped Au g/t

Vein	Vein Code	Composite Data		NN Block Model		IDW3 Block Model		OK Block Model	
		Average	CV	Average	CV	Average	CV	Average	CV
Road	310	4.07	1.55	6.16	1.38	5.06	0.82	4.43	0.85
SheepDip	320	6.32	1.50	4.60	1.72	4.81	0.97	4.75	0.90
	325	1.81	1.45	1.45	1.46	1.36	0.79	1.49	0.47
Mullan	330	3.82	2.60	3.14	1.97	4.13	1.35	3.71	1.21
	335	3.88	1.68	5.31	1.22	5.53	0.72	5.11	0.60
T17	340	20.93	1.59	4.80	1.63	5.25	1.08	5.25	0.96
	345	5.11	2.11	9.38	1.86	7.16	1.42	6.87	1.25
No. 1	350	6.47	2.06	6.54	1.78	6.56	1.14	6.94	1.05
	355	7.83	1.54	5.30	1.52	5.43	0.88	5.51	0.66
106-16	360	3.46	1.52	3.31	1.62	3.20	1.07	3.23	0.94
	365	6.93	1.62	4.49	1.66	4.53	0.96	4.61	0.76
V75	370	5.94	0.93	4.99	0.78	6.12	0.45	5.91	0.30
	375	4.07	1.72	3.45	1.69	3.73	0.82	3.87	0.72
Bend	380	4.21	1.32	9.16	0.56	5.58	0.31	4.62	0.21
	385	2.77	1.57	2.29	1.52	2.48	0.76	2.47	0.67
Crow	395	4.71	1.64	4.46	1.49	4.47	1.01	3.85	0.94
	397	5.28	1.80	8.12	1.42	5.06	1.06	3.32	1.35
Attagh	400	5.16	1.45	5.10	1.71	5.49	1.06	5.86	0.68
	410	5.90	1.24	6.99	0.96	6.71	0.59	6.29	0.43
	420	0.98	1.79	0.91	1.83	1.37	0.42	1.18	0.35

Interpolation Method Validation

The IDW model was validated with the OK and NN models. Figure 14-22 illustrates the agreement between the OK and IDW gold grade models. The IDW model underestimates the low grade (<1 g/t Au) and overestimates the higher grade (>1 g/t Au) relative to the OK model.

Swath plots were generated to compare the AUCID block model with the AUCNN block model. Fifty metre swaths were created by Easting, Northing, and Elevation using the reblocked (to Parent Block) block model in model coordinates. The two models demonstrate good correlation. Except for swaths containing only a few blocks, the IDW model was slightly smoother than the NN model.

Figure 14-23, Figure 14-24, and Figure 14-25 are the swath plots by Easting, Northing, and Elevation respectively.

Figure 14-22: OK vs. IDW Model Scatterplot

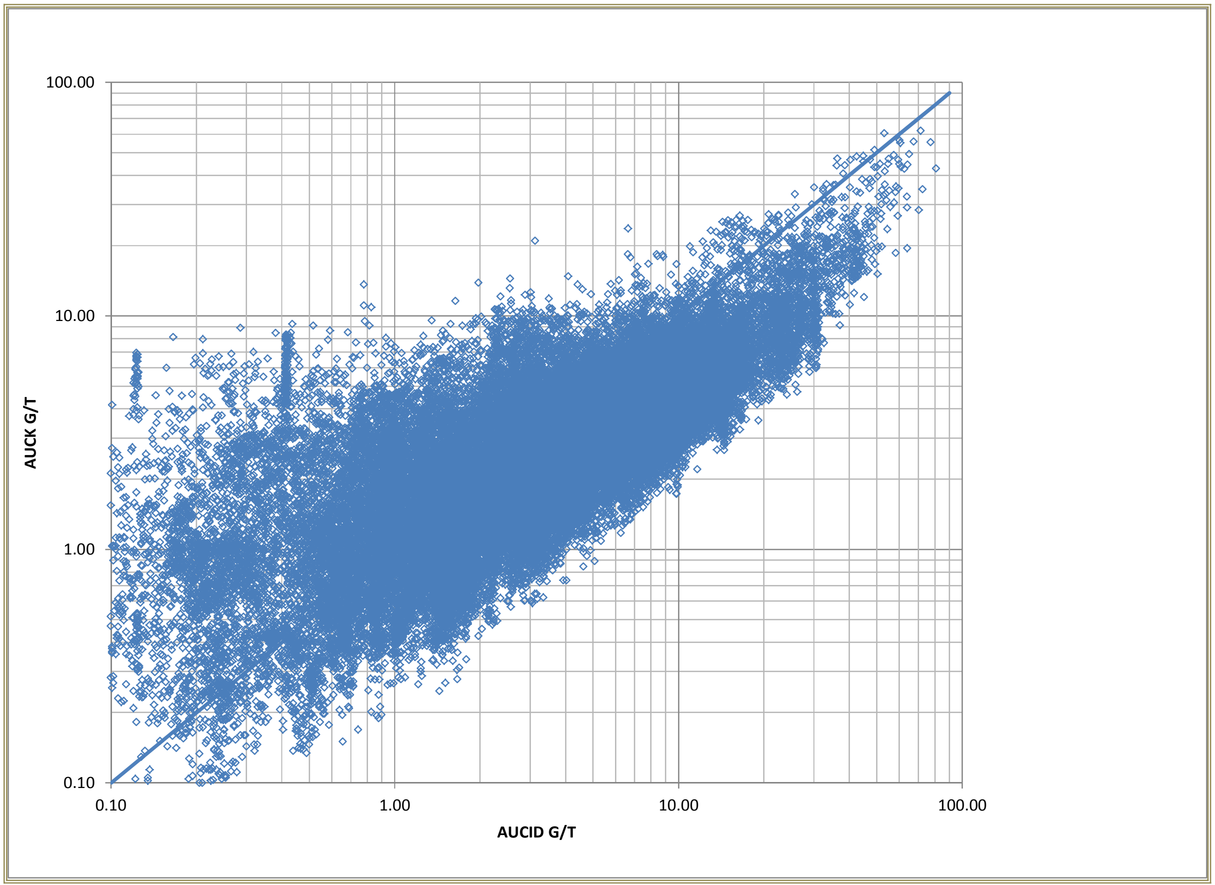


Figure 14-23: AUCID-AUCNN Swath Plot by Easting

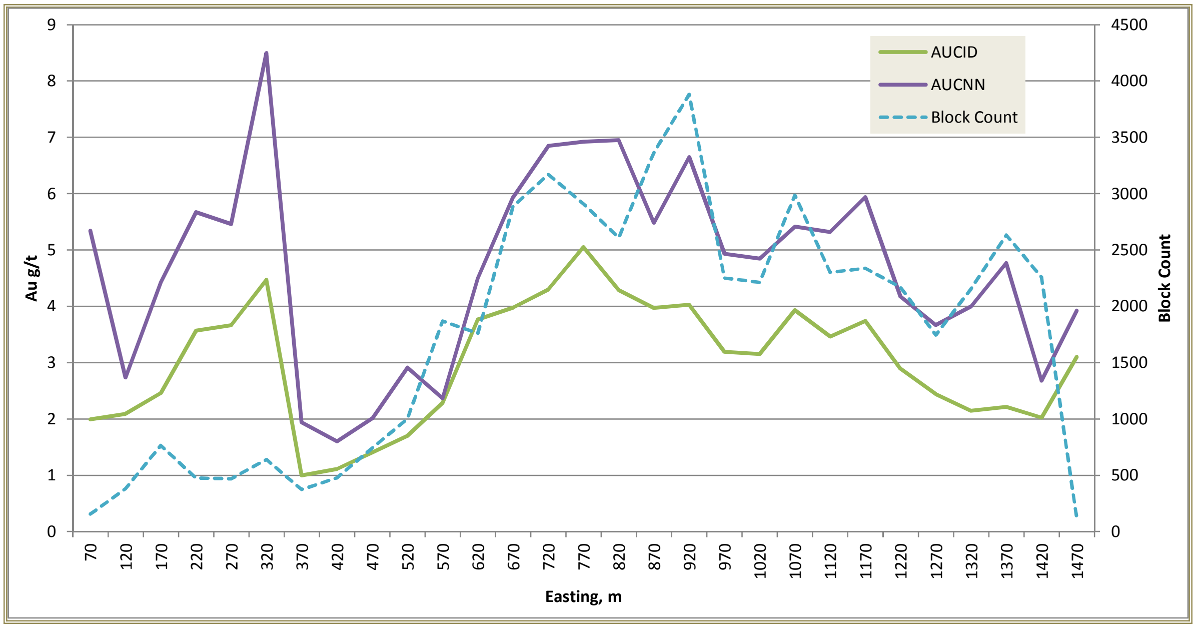


Figure 14-24: AUCID-AUCNN Swath Plot by Northing

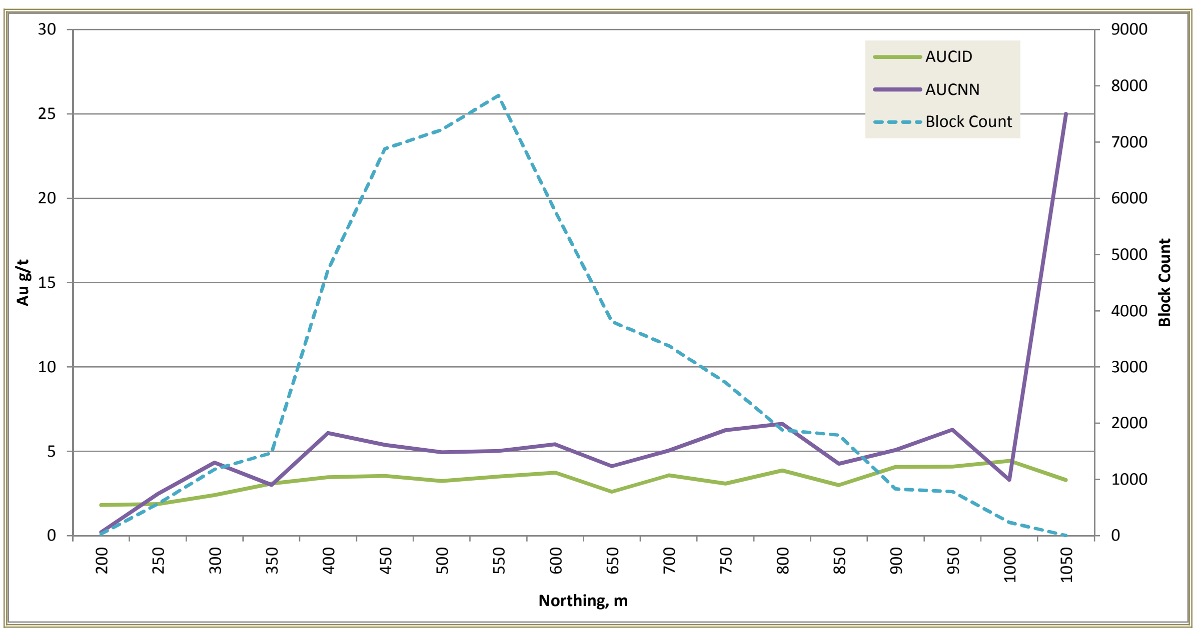
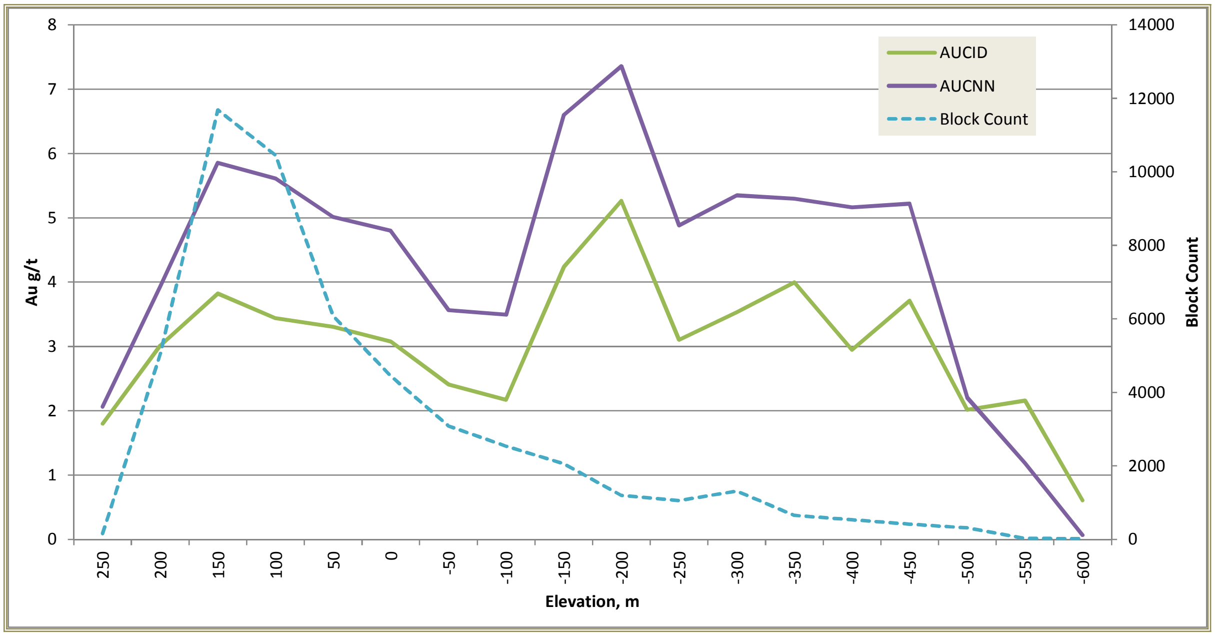


Figure 14-25: AUCID-AUCNN Swath Plot by Elevation



14.12.5 Adequacy of Resource Estimation Methods

The Curraghinalt deposit has been interpolated using industry accepted modeling techniques in MineSight v9.0 desktop software. This included geologic input, appropriate block model cell sizes, grade capping, assay compositing, and reasonable interpolation parameters. The results have been verified by visual review and statistical comparisons between the estimated block grades and the composites used to interpolate. The IDW model has been selected as the best representation of the grade distribution based on the current geological understanding and vein interpretation. The IDW model has been validated with alternate estimation methods: NN and OK. No biases have been identified in the model.

14.13 Mineral Resource

14.13.1 Mineral Resource Classification

Mineral resources were classified in accordance with the 2010 Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Mineral Reserves" incorporated by reference into National Instrument 43-101 "Standards of Disclosure for Mineral Projects." Mineral resources have an effective date of January 20, 2014.

Resources were classified based on using a minimum of two drill holes and two composites for grade estimation. Blocks within 6 m from underground development were classified as Measured resource, the nominal spacing (distance to nearest composite) was 10 m. Indicated resources were based on a maximum distance to nearest composite of 30 m with a nominal spacing of 20 m. Inferred resources were classified up to 125 m with a nominal spacing of 72 m.

No minimum width constraint was applied before reporting this resource. The interpretation method of including a minimum of two metres of downhole length to define a vein zone resulted in an average horizontal thickness of 2.57 m.

Mineral resources are reported at a cut-off grade of 5.00 g/t Au. TMAC cautions that mineral resources that are not mineral reserves do not have demonstrated economic viability.

The author is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, or other relevant issues that may affect the estimate of mineral resources.

Figure 14-26 provides a 3D perspective of the resource classification in the veins with drill hole traces. Figure 14-27 illustrates the resource classification in plan view at elevation 165.0 m.

Figure 14-26: 3D Perspective of Resource Classification Looking Southwest

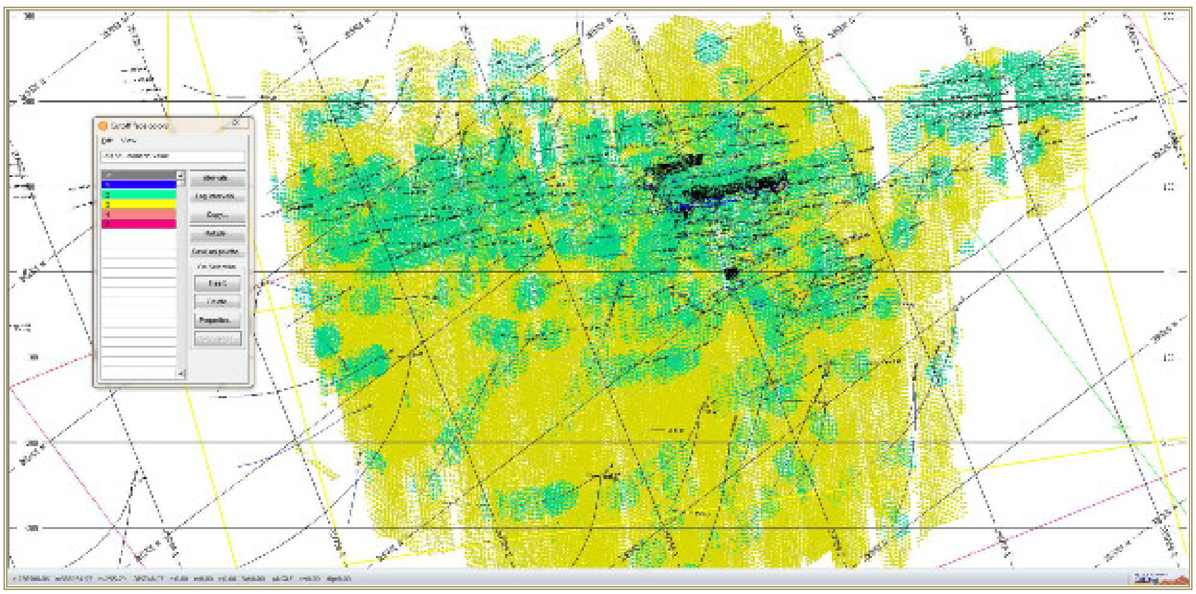
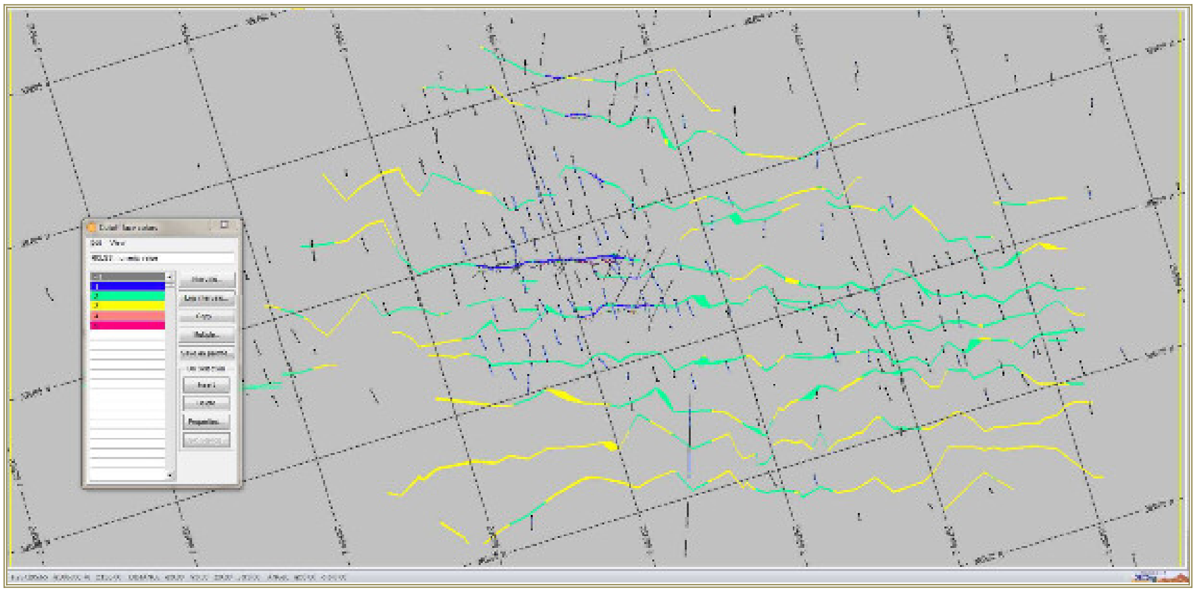


Figure 14-27: Plan View of Resource Classification



Note: Rotated MineSight Grid: Plan_RotN. Elevation 165.0 m. Looking North 17.09° East

14.13.2 Mineral Resource Statement

The mineral resource for Curraghinalt Deposit is tabulated in Table 14-21 at a cut-off grade of 5:00 g/t Au. The effective date of the resource is January 20, 2014. This resource is exclusive of the underground development, which is estimated at 1,700 tonnes at 15.73 g/t Au or 881 oz.

Table 14-21: Curraghinalt Deposit Resource Statement (Cut-off Grade 5.00 g/t Au)

Resource Class	Tonnage (Kt)	Au g/t	Ag g/t	Cu %	Contained Au (Koz)
Measured	23.3	20.15	7.54	0.02	15.1
Indicated	2,976.2	10.34	3.85	0.08	989.0
Measured + Indicated	2,999.5	10.41	3.88	0.08	1,004.1
Inferred	8,005.7	9.67	3.94	0.07	2,487.7

Grade-Tonnage curves for Measured+Indicated Resources and Inferred Resources are shown in Figure 14-28 and Figure 14-29.

Figure 14-28: Measured+Indicated Resources Grade-Tonnage Curve

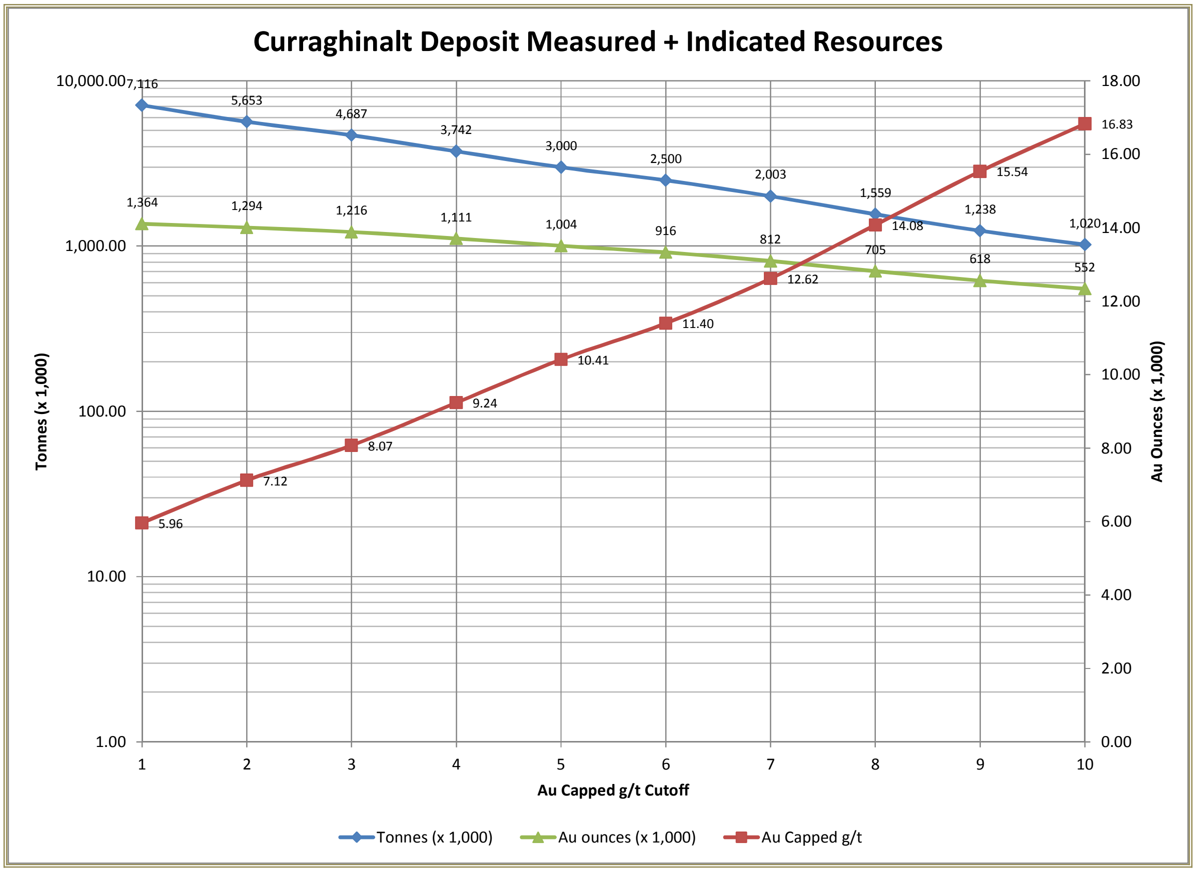
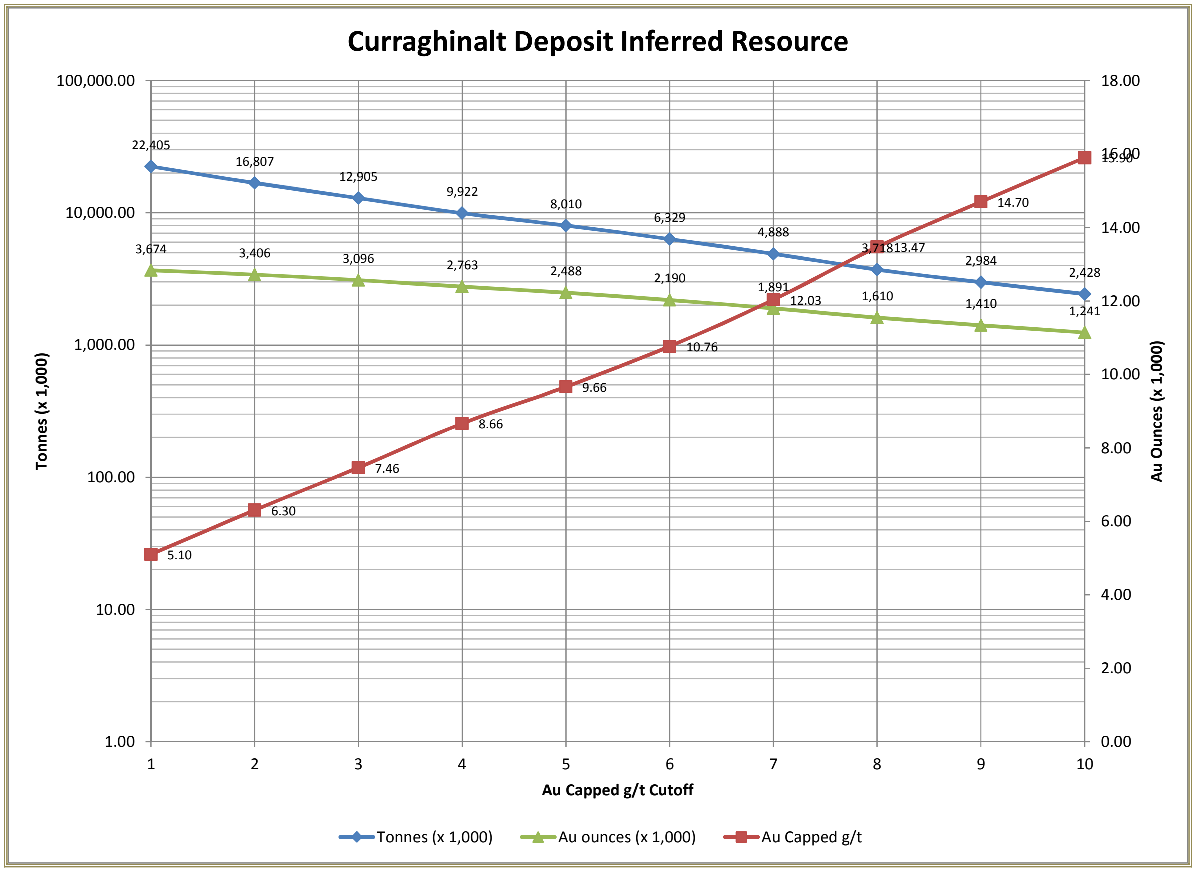


Figure 14-29: Inferred Resource Grade-Tonnage Curve



14.13.3 Mineral Resource Discussion

As reported in Table 14-7, the Micon PEA resource was 1.13 Mt at 13.0 g/t Au for Measured+Indicated. As a result of infill drilling, resampling of older core and reinterpretation of the veins, there is a significant increase to 3.0 Mt at 10.41 g/t Au in the TMAC resource. This has resulted in a gain of 114% in the contained ounces of gold, from the Micon contained metal for Measured+Indicated at 0.47 Moz Au to the TMAC resource of 1.0 M oz Au.

The resource grade has dropped from the Micon grade of 13.00 g/t Au for Measured+Indicated to the TMAC grade of 10.41 g/t Au. Two factors impacted this grade reduction: upgrade classification from Inferred to Indicated of lower grade material and internal dilution added as a result of the vein interpretation methodology.

Inferred Resource material increased by 12%. This was the result of similar factors: infill drilling, resampling of older core and reinterpretation of the veins.

The D veins remain open at depth and along strike. Also, additional material may result from the interpretation and interpolation of the C veins which are not included in this resource.

15 MINERAL RESERVE ESTIMATES

As no prefeasibility or feasibility studies have been completed to date, no mineral reserve estimates have been determined for the Northern Ireland project or the Curraghinalt deposit at this time.

16 MINING METHODS

Information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

16.1 Introduction

The mineralized veins at the Curraghinalt deposit will be extracted using the Longitudinal Sublevel Retreat Underground Mining method with both paste and waste backfill. This backfill method was chosen in order to maximize the extraction of the high value resources at the Curraghinalt Project, reduce the surface footprint required for tailings disposal, and allow savings in the volume and transportation cost of waste rock from underground development.

Access to the Curraghinalt deposit will be via a main decline from the surface. The main decline will be connected by cross-cuts to sublevels providing access to the mineralized veins. Sills, excavated along the mineralized veins on each sublevel, provide access to the mining stopes where production drilling, blasting, and extraction of the mineralized material take place.

The Curraghinalt deposit comprises a swarm of gold-bearing quartz veins. The veins range from a few centimetres wide to over 3 m wide and are aligned along a west-northwest trend. Average geological width is reported to be approximately 1.2 m. The strike length of the vein swarm at Curraghinalt has been traced by prospecting, trenching and drilling to a minimum length of 1,950 m and remains open to the east and west. The veins dip between 60º and 75º to the north and have been traced to a depth of approximately 840 m below surface (Hennessey et al. 2012a).

The extraction of mineralized material from underground will be carried out with rubber tired mechanized equipment to maximize the production and provide flexibility to the underground operations.

16.2 Mine Design Parameters

The following summarizes the preliminary key design parameters and assumptions for the proposed mining method and extraction of mineralized material at Curraghinalt:

- General Parameters:
  - pre-production period with mine development in waste and mineralized material duration is one year
  - production ramp-up at 50%, 75% and 100% from Year -1, 1, and 2, respectively. This is equivalent to 310,500 t/y in Year -1, 465,375 t/y in Year 1 followed by full production at 620,500 t/y by Year 2
  - total mine life (excluding pre-production, Year -1) is 15.0 years
  - mine production rate of 1,700 t/d at full production
  - 365 operating days per year
  - underground operators working three 8-hour shifts per day
  - specific gravity for waste material is 2.7 (assumed)
  - specific gravity for mineralized material is 2.85 (average)
- Underground mining:
  - mining method is longitudinal sublevel retreat
  - decline or ramp dimensions are 4.5 m H x 4.5 m W at 15% grade
  - bypasses at the decline or ramp at 4.5 m H x 4.5 m W x 3.0 m L
  - safety bays along the decline and ramp at 3.0 m H x 3.0 m W x 3.0 m L
  - sumps at 3.0 m H x 4.0 m W x 3.0 m L
  - cross-cut dimension at 4.0 m H x 4.0 m W
  - sill or development in ore at 3.0 m H x 3.0 m W
  - refuge station/lunch room at 4.0 m H x 4.0 m W x 10.0 m L
  - drift to ventilation shafts and remuck bays at 4.0 m H x 4.0 m W
  - ventilation raise and ore or waste-pass at 3 m diameter
  - level intervals at 20 vertical metres from the floor of the top sill to the floor of the bottom sill.

16.2.1 Geotechnical Parameters

In November 2011, Snowden performed a site visit to Curraghinalt to assess the geotechnical core logging process carried on the Project and performed a scan line mapping of the exploration adit and drifts totalling 65 m to obtain an overall preliminary assessment the site geotechnical conditions (Snowden, January 2012).

Snowden reviewed the geotechnical data collected by Dalradian Resources from 94 drill holes (31,344 m) which included some basic geotechnical parameters.

Snowden had noted the geotechnical data collected was not fully compliant with the industry norm but does not consider this significantly influential at the current level of study, although this issue should be addressed for higher level of studies. Snowden also provided recommendations to Dalradian Resources on methodologies to improve the quality of the data collected and a quality assurance process prior to data analysis.

Results of the scan line mapping and review of geotechnical data concluded that:

- rock mass in the exploration adit is fresh and unweathered at greater than 100 MPa
- single dominant joint set with mean orientation of 86°/49° (dip/dip direction)
- five faults have similar orientation of 52°/330° and another at 75°/35°
- overburden is clayey at 3 to 18 m (typically 9 m) across the site.
- weathered zone - 0 to 81 m. In most drill holes extends from 25 to 50 m.
- the average RQD is presented in Table 16-1 for the different lithologies.
- the percentage of the rock mass having joint sets of:
  - none – 0.4%
  - one – 45.9%
  - two – 51.4%
  - three and more – 2.3%.

Snowden indicated that in "a broad sense there is agreement as much of the core is recorded as being intercepted by only one joint set and the joint conditions were similar" between the observation made during mapping and the core.

Table 16-1: RQD and Joint Data by Lithology

Description		RQD %			Jointing J/m
		Min.	Avg.	Max.	
Psammite	PS	3	72	100	1.48
Psammite Weathered	PSW	0	39	84	1.55
Semi Pelite	SP	0	65	100	1.94
Semi Pelite Weathered	SPW	0	33	100	3.48
Pelite	PE	0	53	100	1.95
Pelite Weathered	PEW	0	20	38	2.64

Source: Snowden, January 2012

Observations of the geotechnical conditions of Curraghinalt Project made during Micon's site visit in April 2012 are consistent with observations made by Snowden in scan line mapping. The existing underground excavations appear stable with no major falls of ground or geotechnical stability issues, despite the excavations having been made in the late 1980s.

Based on the geotechnical information presented by Snowden, and for the purpose of the mine design for the PEA, the rock mass is considered to be of fair to good quality. A preliminary assessment of the geotechnical information provided by Snowden indicated "…the overall rock mass is fair based on the RQD with a typical rock strength of R3 (25 to 50 MPa) though the weathered material is poor. The average joint set is 0.5 m but as there is no bedding and the foliation appears welded this in effect quite a massive rock mass" (Snowden, January 2012).

Micon concurs with the recommendations made by Snowden regarding future geotechnical work on data collection, analysis and testing. This additional work will be required to better determine the rock mass classes for the deposit and for mine design purposes.

Micon recommends that, during the geotechnical data collection, the rock mass should be accurately characterized and described. This allows the rock mass attributes, parameters, and ratings to be assigned at a later stage enabling the classification to be made in any of the proposed rock mass classification systems (e.g., RMR, Q-system, or MRMR).

16.3 Cut-off Grade Determination

Within the mineral resource reported by Hennessey et al. (2012a), the selection of material for inclusion in the mine plan for the PEA mine plan was determined by a cut-off grade. The value of the mining and milling cut-off grade for underground mining at Curraghinalt Project is at 5.0 g/t Au.

This cut-off grade was determined considering the preliminary cost estimates for mining, milling and general administration for the deposit, taking into account expected recoveries and metal price.

16.4 Mineral Resources Considered in the Mine Plan

For the purposes of the PEA, the diluted and potentially extractable portions of the mineral resources that are above a 5.0 g/t Au cut-off grade have been termed "mineral resource considered in the mine plan." This term is used here for the purposes of distinguishing the mineral resources contained within the preliminary mine design and production plan from the mineral resource estimate tabulated in Section 14.

The mineral resource estimate given in Section 14 is derived from a geological block model created in Datamine software and was used as the basis for the PEA. The Datamine models were converted to Surpac format in order to prepare the PEA mine design and plan.

The mineral resource considered in the mine plan for Curraghinalt was based on the following estimation parameters:

- tonnages above the 5.0 g/t Au cut-off grade, after applying dilution factors to the grade to account for a minimum mining width of 1.8 m for stopes and 3 m for sills
- a non-recoverable crown pillar extending 20 m below and parallel to the topography
- exclusion from the mine plan of resources below the -490 m level
- mining recovery of 95%.

16.4.1 Mining Recovery and Dilution

The mining recovery and dilution values for Curraghinalt deposit were estimated based on the geological characteristics of the vein system, the use of the longhole retreat mining method and the dimensions of the selected narrow-vein mining equipment.

A mining recovery of 95% is assigned to account for mineralized material losses contributed by continuity of the deposit and mining activities in general.

The dilution values for the Curraghinalt Project were estimated in two stages, during the mineral resource stage and mine design stage.

At the mineral resource estimate stage, all the mineralized blocks in the block model with vein thickness of greater than 0.10 m, but less than 1.0 m, were diluted to 1.0 m. The diluting material is conservatively assumed to have zero grades. The mineral resource was then reported at an economic cut-off grade of 5.0 g/t Au over the minimum thickness of 1 m (Hennessey et al., 2012a).

In the mine design stage, all the mineralized blocks in the block model with thickness of greater than 1.0 m, but less than 1.8 m in thickness, were diluted to 1.8 m. The diluted material is considered as external dilution and is conservatively assumed to have zero grade. The 1.8 m width was considered as the minimum mining width for the stopes so, except where the veins have been shown to be wider than this, 1.8 m is the final mined stope width.

The design width of the sublevel sills is 3.0 m. This is the minimum mining width for rubberized equipment entry to develop the sill, perform production drilling and blasting and for blasted material to be extracted out of the stope. Again, additional dilution at zero grade was added to the mineralized zone to bring the width of the sills less than 3.0 m to 3.0 m. This diluted material is considered as external dilution.

The overall weighted average mining dilution is 33%. This dilution is additional to that already included in the resource estimate to account for a 1.0 m minimum width. A breakdown of the percentage dilution by resource classification is presented in Table 16-2.

Table 16-2: Percentage Dilution

Description	Percentage Dilution	Percent Grade Differences			Remarks/ Reference
		Au	Ag	Cu	
Measured	15%	-22%	-22%	-21%	Percentage Difference from "Mineral Resources Considered in the Mine Plan" to Mineral Resources Considered in the Mine Plan Diluted to 1.8 m at 95% Recovery with External Dilution"
Indicated	28%	-45%	-45%	-45%	
Total Measured & Indicated	**27%**	**-45%**	**-44%**	**-44%**	
Total Inferred	**35%**	**-61%**	**-61%**	**-61%**	
Overall Total Tonnage Dilution	**33%**				

The mining recovery and dilution values are considered to be within the range for the proposed mining method and backfill incorporated, level interval, regularity and continuity of the deposits, losses in fines during blasting and mucking, and potential material left in the stope to form pillars to stabilize the excavations.

16.4.2 Classification of Mineral Resources Considered in the Mine Plan

Table 16-3 displays the conversion of the 2012 Mineral Resources (Hennessey et al., 2012a) to the mineral resource considered in the mine plan diluted to a 1.8 m mining width at 95% recovery with external dilution.

The mineral resources considered in the mine plan for Curraghinalt Project are:

- Measured and Indicated resources
  - 1.44 Mt
  - 8.88 g/t Au average grade
  - 2.95 g/t Ag average grade
  - 0.12% of Cu average grade
- Inferred resource
  - 7.98 Mt
  - 7.91 g/t Au average grade
  - 3.38 g/t Ag average grade
  - 0.08% of Cu average grade.

These values include the losses in tonnages from the crown pillar, mineralized material at depth, minimum mining widths, recoveries, and dilution factors described in the previous sections.

Table 16-3: Measured, Indicated, and Inferred Mineral Resources Considered in the Mine Plan

Description	Tonnage (Mt)	Avg. Au (g/t)	Avg. Ag (g/t)	Avg. Cu (%)	Remarks/Reference
Mineral Resource					
Measured	0.02	21.51	17.56	0.490	Hennessey et al., 2012a
Indicated	1.11	12.84	4.05	0.170	
Total Measured & Indicated	**1.13**	**12.99**	**4.29**	**0.176**	
Total Inferred	**5.46**	**12.73**	**5.43**	**0.118**	
Mineral Resource In Crown Pillar and Below -490 m Elevation					
Measured	0.00	20.46	17.36	0.49	Resource with thickness 0.1 m to 1.0 m diluted to 1.0 m as reported in 2012 Mineral Resource
Indicated	0.08	12.64	4.01	0.17	
Total Measured & Indicated	**0.08**	**12.80**	**4.27**	**0.17**	
Total Inferred	**0.24**	**12.74**	**5.44**	**0.12**	
Mineral Resource Considered in the Mine Plan, prior to application of modifying factors					
Measured	0.02	20.10	17.29	0.49	Resource less Tonnages in Crown Pillar and below -490 m Elevation
Indicated	1.03	12.69	4.01	0.17	

Description	Tonnage (Mt)	Avg. Au (g/t)	Avg. Ag (g/t)	Avg. Cu (%)	Remarks/Reference
Total Measured & Indicated	**1.05**	**12.84**	**4.27**	**0.17**	
Total Inferred	**5.22**	**12.71**	**5.43**	**0.12**	
Mineral Resource Considered in the Mine Plan					
Measured	0.02	16.46	14.19	0.40	Mineral Resources Considered in the Mine Plan Diluted to 1.8 m at 95% Recovery with External Dilution
Indicated	1.42	8.76	2.77	0.12	
Total Measured & Indicated	**1.44**	**8.88**	**2.95**	**0.12**	
Total Inferred	**7.98**	**7.91**	**3.38**	**0.08**	

16.5 Mining Method

During the conceptual study prepared by Snowden (January 2012), seven underground mining methods and an open pit mining option were evaluated. The evaluated underground mining methods range from longhole stoping with and without backfill, cut and fill mining and shrinkage mining, incorporating of several backfill types.

The proposed underground mining method for the Curraghinalt Project for the PEA is Longitudinal Sublevel Retreat with a vertical level interval of 20 m from the floor of the top level to floor of the bottom level. This mining method and the height of the vertical level were selected based on:

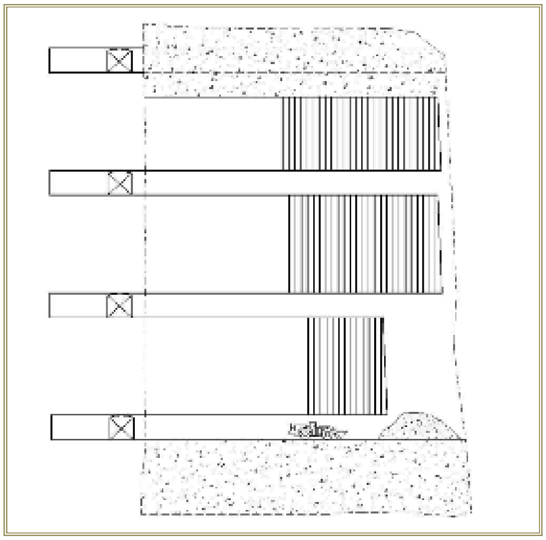
- preliminary geotechnical assessment of the ground conditions indicate massive rock which should be conducive to better drilling, blasting and dilution control
- availability of mechanized narrow vein mining equipment for the geometry of the deposit
- dilution control measures and knowledge in mining and milling
- increased technology and knowledge in open stoping mining methods in narrow vein deposits
- overall continuity and regularity of the veins.

Paste backfill will be incorporated into the mining method as the primary backfill type. Waste rock will also be used as backfill material in combination with the paste backfill.

A sill pillar may be left between levels in the open stope, or a sill mat can be constructed to isolate the fill material from one level to another. Depending on the regularity and rock mass condition, more than one level can be mined within the same mining sequence.

Figure 16-1 presents typical schematic of a longitudinal sublevel retreat mining method with multiple mining levels.

Figure 16-1: Typical Schematic of Longitudinal Sublevel Retreat with Multiple Mining Levels



Micon recommends that further optimization of the mining method be performed to evaluate potential additional economic benefits to the Project by considering and combining a higher selectivity non-mechanized mining method with the proposed Longitudinal Sublevel Retreat.

16.6 Underground Mine Plan and Development

Access to the underground mineable zones will be through a decline excavated from the surface to elevation 210 m. Mining of the underground resources will commence at elevation 250 m, followed by the remaining mineralized zones as mining progresses.

Figure 16-2 to Figure 16-4 display isometric and plan views of the proposed underground mine for the Curraghinalt Project.

The main access decline will be excavated at a grade of -15% with an arched roof for greater stability at 4.5 m H x 4.5 m W.

Underground ramp networks, with similar dimensions to the main decline, connect one level to another within the ramping system. The vertical distance between the levels is 20 m from the floor the top sill to the floor of the bottom sill.

The proposed ramp system excavation includes mobile equipment passing bays, a safety bay, and a sump between each level. Refuge stations or lunchrooms will be located in the cross-cuts at every 40 m interval (i.e., on every second level).

Cross-cuts at 4.0 m H x 4.0 m W will be excavated to connect the ramp to the mineralized zones and are configured to be approximately perpendicular to the mineralized zones.

A sump of 3.0 m H x 4.0 m W x 3.0 L will be excavated in every cross-cut to decant all the water generated from the mining activities or potentially generated from the ground prior to pumping and discharging to the surface water treatment facility.

Two drifts at 10 m each in length will be excavated to connect the cross-cuts to the ore and waste-passes. These drifts will be utilized as temporary remuck bays for waste and mineralized storage areas until the ore and waste-passes are developed.

Sills at 3.0 m H x 3.0 m W in the mineralization veins will be developed when the cross-cut intercepts the veins. The sills will be developed in both directions along the strike length of the mineralized zone, providing two production faces for each vein on each level.

Mining in a production level will commence from one end of the veins retreating to the level's crosscut. The availability of numerous production headings when a cross-cut intercepts the veins within the level provides multiple production headings to meet the 1,700 t/d production target.

Figure 16-2: Isometric View of Curraghinalt Project Mine Layout

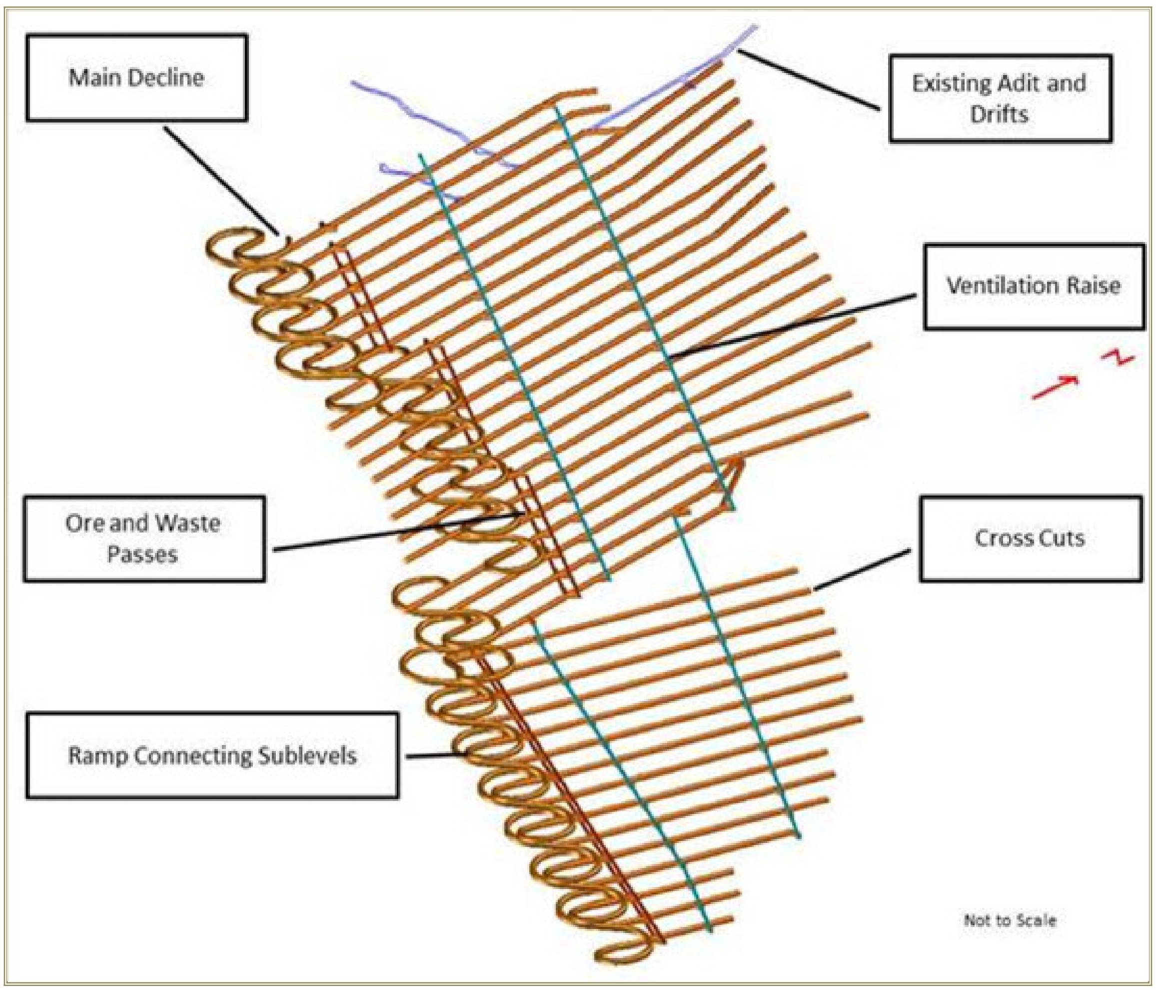


Figure 16-3: Isometric View of the Mine Layout with Mineralized Veins

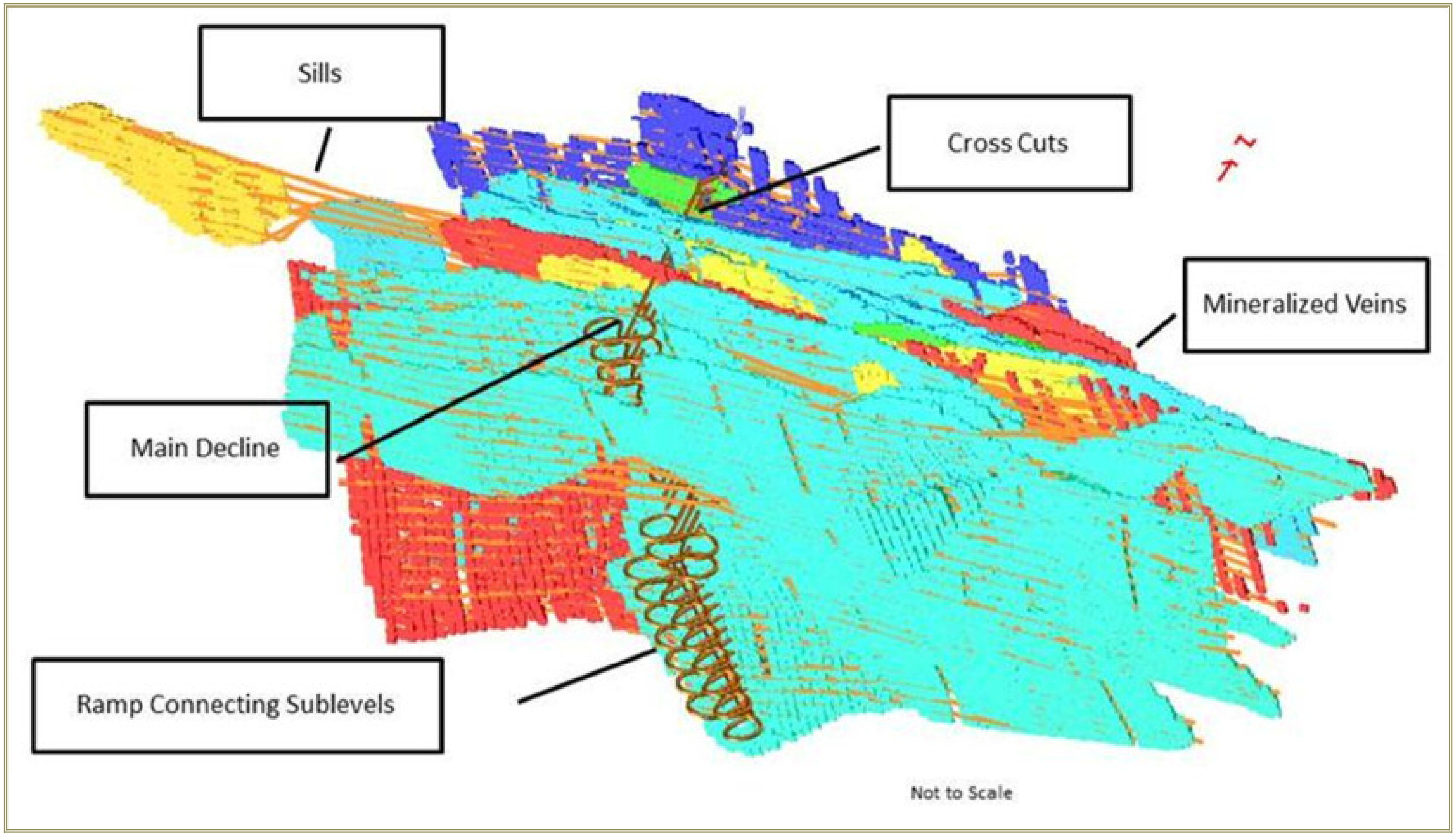
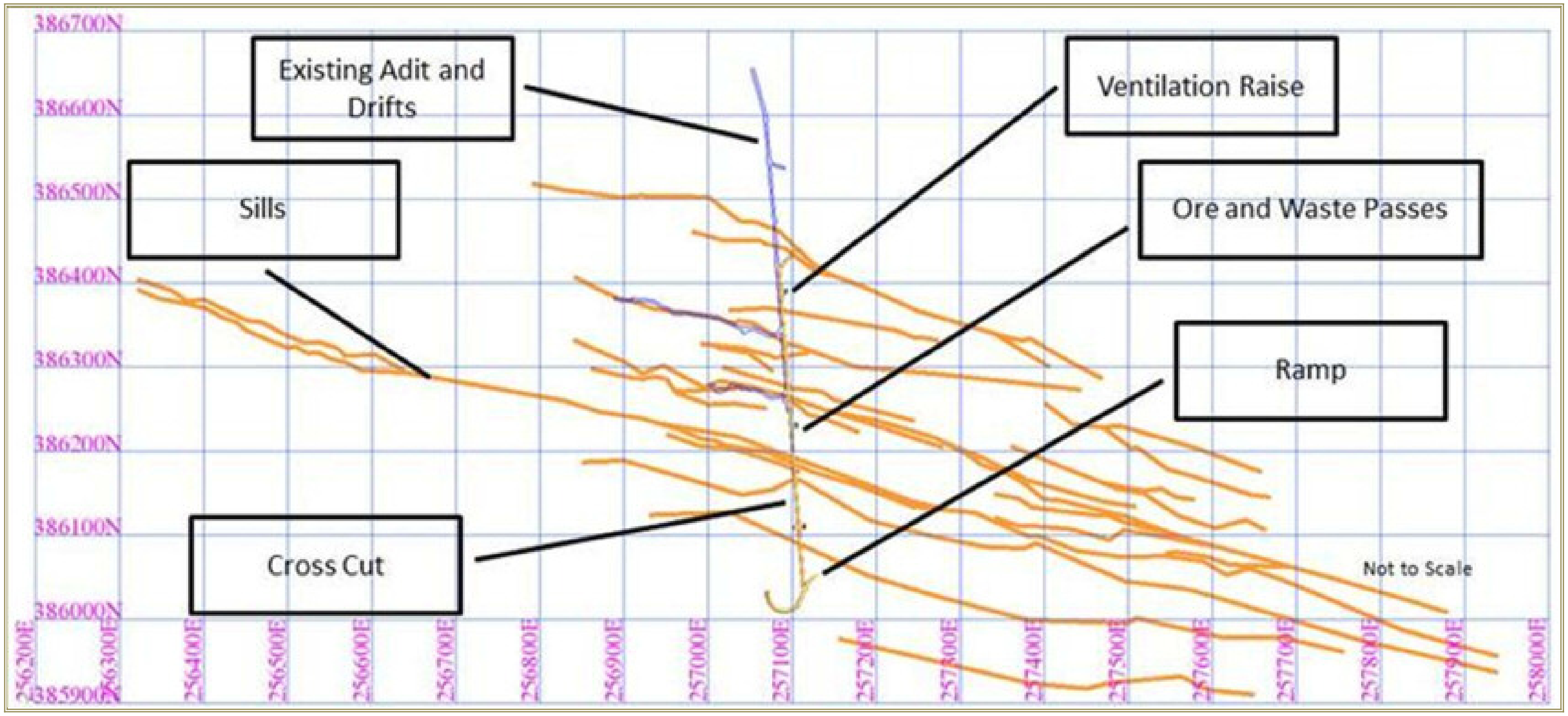


Figure 16-4: Typical Plan View of the Mine Layout (at 180 m Elevation)



The main decline into the mineralized zones can act as fresh air intake into the mine. Exhausted and contaminated air can be channelled through one of the two ventilation raises or directed into the existing exploration adit on 170 m elevation to the surface.

Ventilation from one level to another will be provided by the ventilation raises located in the cross-cuts. Ventilation drifts connect the ventilation raises to the cross-cuts to supply fresh or exhaust air from the underground workings.

The total amount of development in LOM for the Curraghinalt Project is presented in Table 16-4.

Table 16-4: Total Underground Development for the Curraghinalt Project

Description	Total Length (m)
Ramp	5,325
Ramp By Passes	157
Ramp Safety Bays	120
Sump in Ramp	120
Cross Cuts (including Refuge Stations & Service Drifts)	16,614
Sumps in Cross Cut	137
Sill	207,420
Sill (Through Waste)	7,152
Vent, Ore, and Waste Pass	2,990
Total	**240,035**

16.7 Mine Ventilation Requirement

The mine fresh air requirement estimation is based on the operating mining equipment fleet, their utilization and allowances for losses due to friction and short circuits. This forms the preliminary estimated fresh air requirement of approximately 118 m^3/s or 250,000 CFM for the mine (Table 16-5).

The estimate is based on the provision of 0.06 $m^3/s/kW$ of diesel powered equipment operating underground with some allowance for personnel working around some equipment for dust and fume control.

The ventilation demand estimated by Micon is preliminary in nature and considers all the diesel power mining equipment working within a deposit or mineralized zone. However, this may not be the case during operation because ore can be extracted from various zones.

Micon recommends that a detailed evaluation of the ventilation demand be investigated in further detail as part of a prefeasibility study to ensure that the fresh air requirement for underground mine operation at Curraghinalt meets the Northern Ireland regulation for ventilation in mining operations, as well as any other regulations related to the levels of dust and particulates, carbon monoxide and oxides of nitrogen in the vicinity of operators and in the undiluted equipment exhaust which may be applicable.

Table 16-5: Estimated Ventilation Requirement

Description	Units	Power (kW/unit)	Total	Utilization (%)	Power (kW)	Flow Rate (m^3/s)
Stoping Drill (Elec.-hyd.)	1	45	45	54	24.3	1.5
Sill Narrow Vein Jumbo	5	38	190	54	102.6	6.2
Development Jumbo (Double Boom)	1	110	110	54	59.4	3.6
Bolter	1	110	110	54	59.4	3.6
LHD at 4 m^3	1	150	150	54	81.0	4.9
LHD at 3.0 m^3	4	210	840	54	453.4	27.2
Trucks -30-ton	3	310	930	54	502.0	30.1
Explosive Truck	1	78	78	45	35.1	2.1
Scissor Truck	1	96	96	45	43.2	2.6
Fuel and Lube Truck	1	96	96	45	43.2	2.6
Mechanic Light Pickup	1	78	78	54	42.1	2.5
Electrician Light Pickup	1	78	78	54	42.1	2.5
Surveying Light Pickup	1	78	78	54	42.1	2.5
Light Pickup (Service/Supervision)	1	78	78	45	35.1	2.1
Man Carrier	1	96	96	22	21.6	1.3
UG Grader	1	103	103	45	46.3	2.8
Fume Dilution and Losses	20%	-	-	-	-	19.6
Total						**117.6**

16.8 Drilling and Blasting

Underground ramps and cross cuts will be excavated using double boomed electric-hydraulic jumbo drills.

Sill development in the mineralized areas will be excavated using single-boomed electric-hydraulic jumbo drills to maintain the mining width of 3.0 m and to reduce the amount of dilution.

Production drilling will be performed by a combination of pneumatic and electric-hydraulic longhole drills. Production drill holes in the wider sections of the veins will be drilled with 89 to 102 mm (3.5" to 4.0" diameter) drill holes and drilling will be performed with the electric-hydraulic longhole drill. Narrow vein production drill holes will be drilled by the smaller pneumatic longhole drill (Figure 16-5) with drill holes of 50.8 to 89 mm (2" to 3.5") diameter.

Ammonium Nitrate Fuel Oil (ANFO) will be the bulk explosive for all underground blasting and cartridge emulsion will be used during priming in production drill holes and in areas susceptible to water infiltration such as lifters in horizontal development.

Figure 16-5: Production Drilling in Narrow Vein

Source: Boart Longyear)

16.9 Haulage of Waste and Mineralized Material

Blasted waste and mineralized material from the underground mining activities will be transported to the designated disposal areas or processing facility by rubber tyre mechanized mining equipment.

Waste material generated during the initial stage of the underground development will be trucked to the surface until voids are available in the open stopes. Blasted waste and mineralized material from the cross-cuts and stopes will be mucked by Load-Haul-Dump (LHD) and hauled to the remuck bays or ore and waste-passes located in the cross-cuts.

The blasted muck from development faces will be cleaned with 4 m^3 (5.2 yd^3) LHD units to temporary re-muck bays or directly into waste-passes. Narrower development in sills will be mucked with a low profile 3 m^3 (3.9 yd^3) LHD units.

Material from the ore and waste-passes will be loaded into 30-ton low-profile diesel-powered trucks to be transported to the surface.

Oversize material will be blasted during the end of the shift in secured areas before being discharged into the ore or waste-passes or used as backfill material.

An estimated 1.26 Mt (653,000 m^3) of waste material will be generated during the LOM based on the mine plan and it is assumed at that 50% of this material will be utilized as backfill material underground.

16.10 Backfill System – Pastefill

The placement of tailings as backfill material at the Curraghinalt Project is an important component of the underground mining cycle. Paste backfill is an engineered material, which enhances the stability of excavated underground mine openings by providing engineered supporting pillars or as a working platform.

The tailings produced during gold extraction will be used as one of the major ingredients in the backfill material. Waste rock generated from underground development will be also be used as backfill material to minimized the amount of material transportation to the surface disposal areas.

Paste backfill will be prepared with dewatered tailings from the mill, mixed with cement and a binding agent and tailings slurry or water to control the pulp density.

The effective void volume for backfill placement is estimated based on the assumption that 90% or 3.0 Mm^3 of the voids in the stopes or mineralized areas are fillable. A total amount of 2.7 Mm^3 will be filled with paste backfill and the remaining 0.33 Mm^3 will be filled with the waste material generated underground.

Micon recommends that further studies be performed on the backfill system for the Project and that a systematic laboratory test program be performed on the mine tailings to determine the optimum cement addition and cement type to be used as the binding agent for the paste backfill.

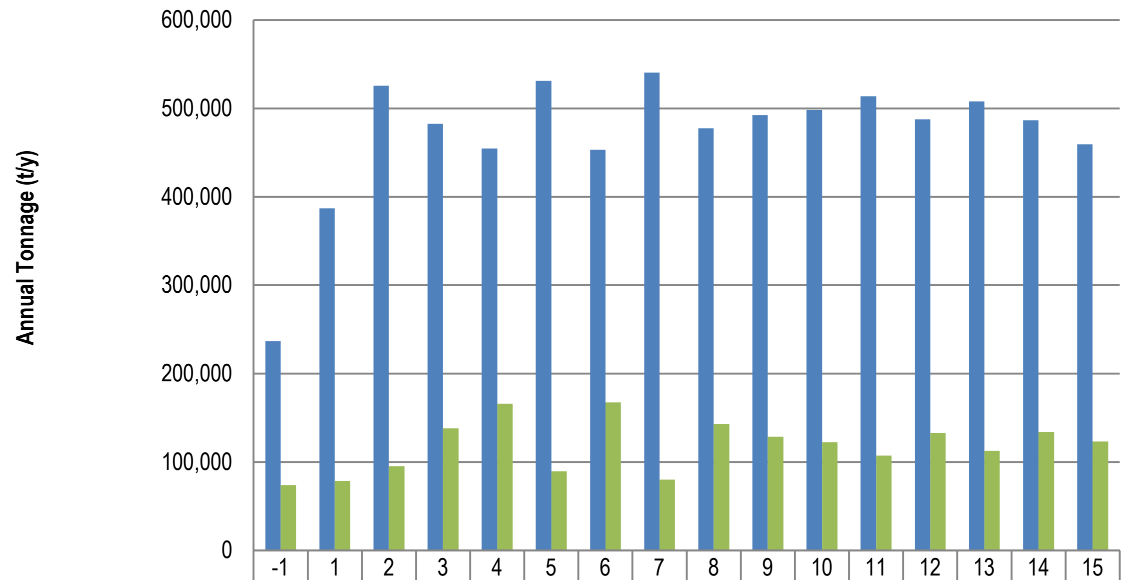
16.11 Mine Preproduction and Production Schedule

The overall mine production schedule is shown in Table 16-6. Figure 16-6 to Figure 16-8 display the underground production tonnage ramp up and annual gold and silver grades, respectively.

Table 16-6: Curraghinalt Project Annual Production Plan

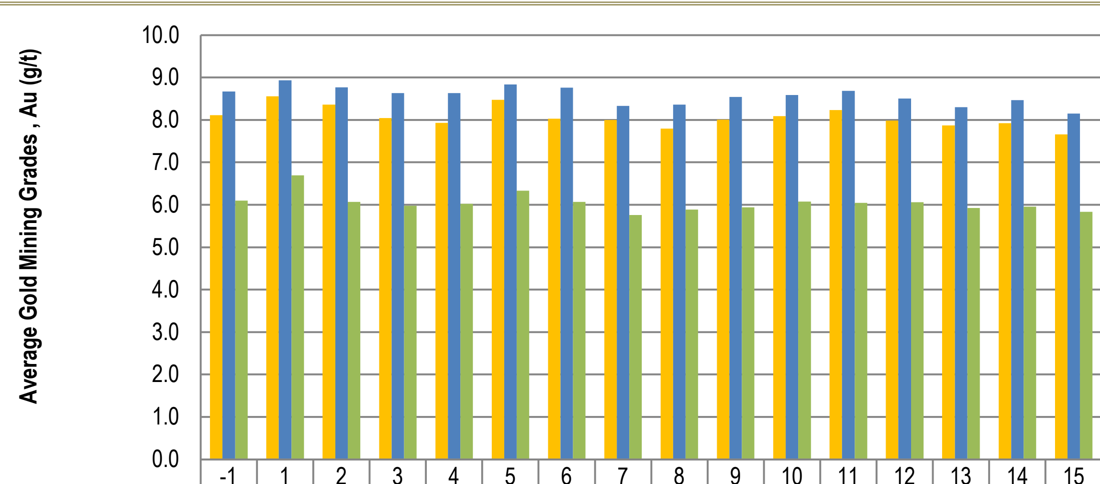
Description	LOM Total Tonnes/Grades	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Sublevel	7,534,046	236,459	386,791	525,524	482,681	454,786	531,030	453,227	540,662	477,534	492,158	498,164	513,510	487,576	508,063	486,633	459,249
Au (g/t)	8.56	8.67	8.93	8.77	8.63	8.63	8.83	8.76	8.33	8.36	8.54	8.58	8.68	8.50	8.30	8.47	8.15
Ag (g/t)	3.52	3.76	3.82	3.65	3.51	3.48	3.59	3.58	3.38	3.40	3.41	3.44	3.44	3.50	3.51	3.65	3.43
Cu (%)	0.09 %	0.08 %	0.09 %	0.09 %	0.09 %	0.09 %	0.09 %	0.09 %	0.08 %	0.08 %	0.09 %	0.09 %	0.09 %	0.09 %	0.09 %	0.09 %	0.08 %
Sill	1,890,460	73,791	78,584	94,976	137,819	165,714	89,470	167,273	79,838	142,966	128,342	122,336	106,990	132,924	112,437	133,867	123,132
Au (g/t)	6.03	6.10	6.70	6.07	5.98	6.02	6.33	6.07	5.76	5.89	5.94	6.08	6.05	6.06	5.92	5.95	5.84
Ag (g/t)	2.49	2.75	2.81	2.68	2.45	2.45	2.58	2.50	2.32	2.32	2.38	2.43	2.37	2.46	2.49	2.62	2.45
Cu (%)	0.06 %	0.06 %	0.07 %	0.07 %	0.07 %	0.07 %	0.06 %	0.06 %	0.06 %	0.06 %	0.06 %	0.06 %	0.06 %	0.06 %	0.06 %	0.06 %	0.06 %
Total Tonnage	9,424,506	310,250	465,375	620,500	620,500	620,500	620,500	620,500	620,500	620,500	620,500	620,500	620,500	620,500	620,500	620,500	582,381
Au (g/t)	8.06	8.11	8.55	8.36	8.04	7.93	8.47	8.03	8.00	7.79	8.01	8.09	8.23	7.98	7.87	7.92	7.66
Ag (g/t)	3.31	3.52	3.65	3.50	3.28	3.20	3.45	3.29	3.24	3.15	3.19	3.24	3.26	3.28	3.33	3.42	3.22
Cu (%)	0.08 %	0.07 %	0.09 %	0.09 %	0.09 %	0.09 %	0.09 %	0.08 %	0.08 %	0.08 %	0.08 %	0.08 %	0.08 %	0.08 %	0.08 %	0.08 %	0.08 %
Contained Au (t)	75.94	2.52	3.98	5.19	4.99	4.92	5.26	4.98	4.96	4.83	4.97	5.02	5.11	4.95	4.88	4.92	4.46
Contained Ag (t)	31.24	1.09	1.70	2.17	2.03	1.99	2.14	2.04	2.01	1.96	1.98	2.01	2.02	2.03	2.07	2.12	1.88
Contained Cu (t)	7,784.01	229.60	409.06	563.95	541.76	531.70	539.99	496.73	492.81	471.28	508.10	511.55	510.72	516.98	502.51	495.30	461.95

Figure 16-6: Annual Production Tonnage



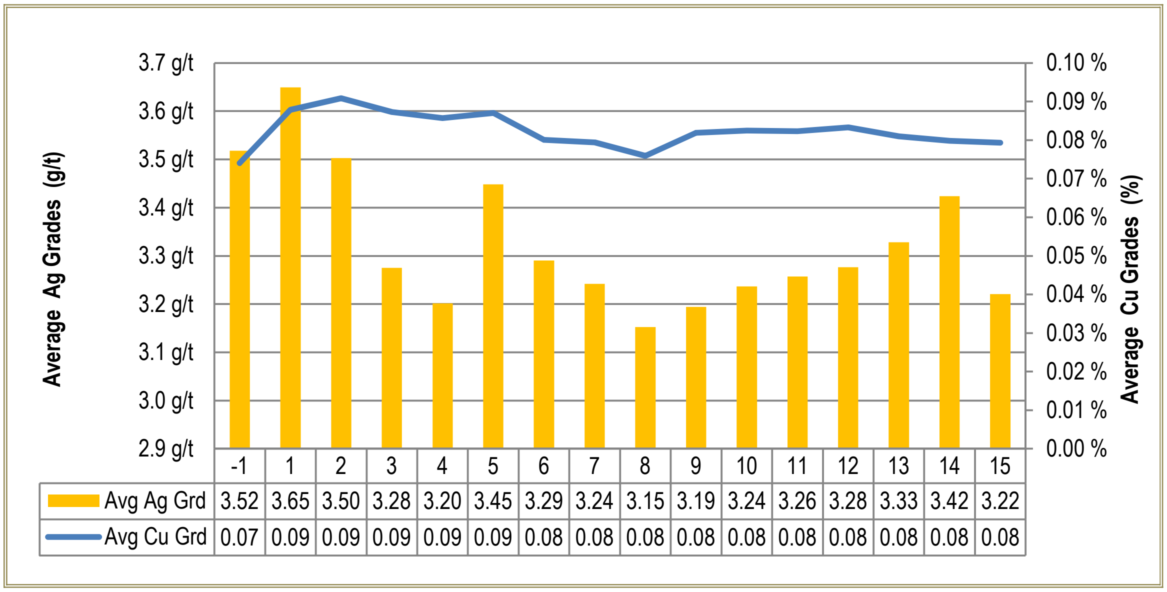
	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Tonnage from Sublevel	236,4	386,7	525,5	482,6	454,7	531,0	453,2	540,6	477,5	492,1	498,1	513,5	487,5	508,0	486,6	459,2
Tonnage from Sill	73,79	78,58	94,97	137,8	165,7	89,47	167,2	79,83	142,9	128,3	122,3	106,9	132,9	112,4	133,8	123,1

Figure 16-7: Annual Gold Grades



	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Avg Au Grd	8.11	8.55	8.36	8.04	7.93	8.47	8.03	8.00	7.79	8.01	8.09	8.23	7.98	7.87	7.92	7.66
Sublevel Au Grd	8.67	8.93	8.77	8.63	8.63	8.83	8.76	8.33	8.36	8.54	8.58	8.68	8.50	8.30	8.47	8.15
Sill Au Grd	6.10	6.70	6.07	5.98	6.02	6.33	6.07	5.76	5.89	5.94	6.08	6.05	6.06	5.92	5.95	5.84

Figure 16-8: Annual Silver and Copper Grades



16.12 Dilution Control

Dilution control during the mining operation will be a collaborative responsibility of all the departments on the Curraghinalt Project site.

The following highlights some of the measures that will have to be implemented during operation to reduce the amount of potential dilution during mining:

- sampling and mapping of the mineralized structures during the development of the sills by geological department
- interpretation of mineralized structures when the sills are developed by the geological department.
- accuracy of the mine engineering group in the drilling and blasting design from the interpreted mineralized structures
- accuracy of the surveying personal to identify the set-up location for production drill rings.
- drilling accuracy and deviation control by the drillers
- drill holes verifications - the drill holes drilled from the upper sill will allow the breakthrough location of each hole to be examined. This allows for re-drilling of any deviated drill holes before blasting to minimize unplanned dilution
- drill cutting sampling at the prescribed interval recommended by the geological department during production drill
- rapid turnaround time on sample preparation and assay reporting by the analytical team

- revision of the geological interpretation with the assay results from the drilling cutting and adjustment made according to the production drilling and blasting plan
- training for all the production drillers and parties involved.

Additional costs have been accounted for in the mine plan for auxiliary labour force to assist with the sampling of material underground.

Other dilution control measures can include the installation of an optical sensor-based sorting machine to separate the waste material from mineralized material prior to mineral processing.

16.13 Equipment Fleet

16.13.1 Mine Services

The underground services equipment estimate is presented in Table 16-7.

Table 16-7: Estimated Underground Service Equipment

Description	Units
Jackleg/Stoper	4
Compressor (2,500 CFM)	1
Surveying Equipment and Software	1
Ventilation Fans – 75 HP	2
Ventilation Fans – 50 HP	2
Ventilation Fans – 30 HP	4
Pumps – 30 HP	2
Pumps – 15 HP	4
General Electrical Equipment	1
Fuel Tank (15,000 L) and Facilities	1
Self-Contained Rescue Chamber	1

16.13.2 Underground Mobile Equipment

The basis of estimate for the mobile equipment is based on the assumption of the following productivity estimated for the proposed equipment, as shown in Table 16-8.

Table 16-9 summarizes the proposed equipment fleet required to develop and extract 1,700 t/d of mineralized material from the Curraghinalt deposit. The table presents the average number of equipment units over the LOM. The underground mobile and auxiliary equipment estimate is based on Project's demand at full capacity.

Table 16-8: Estimated Underground Service Equipment

Description	Unit	Productivity
Jumbo productivity in the ramp	m/y	1,157
Jumbo productivity in the cross-cuts	m/y	1,822
Production drill productivity	t/y	110,974
Narrow vein jumbo productivity	m/y	3,053
Bolter	m/y	1,313
LHD at 4 m^3 (ramp)	t/y	90,520
LHD at 4 m^3 (cross-cuts)	t/y	204,400
LHD at 3.0 m^3 (stope)	t/y	190,530
Trucks – 30-ton		Varies with depth

Table 16-9: Underground Mobile Equipment Fleet

Description	Units (Avg. LOM)
Stoping Drill (e.g., Boart Stopemate)	4
Stoping Drill (Electric-hydraulic)	1
Sill Narrow Vein Jumbo	5
Development Jumbo (Double Boom)	1
Bolter	1
LHD at 4.0 m^3	1
LHD at 3.0 m^3	4
Trucks – 30-ton	3
Explosive Truck	1
Scissor Truck	1
Fuel and Lube Truck	1
Mechanic Light Pickup	1
Electrician Light Pickup	1
Surveying Light Pickup	1
Light Pick-Up (Service and Supervision)	3
Man Carrier	1
UG Grader	1

16.14 Mining Labour Force

The mine will operate at 365 d/y at 3 shift/d for 8 h/shift with most of the major underground mining tasks be performed during the working days within a calendar week. Overtime expenses have been accounted in the labour force estimate for additional work that has to be performed during the weekend periods.

The workforce for the Project will comprise locally and regionally trained and skilled labour force. Individuals will be trained and certified for the operation of a range of mining equipment to increase operational efficiencies.

All the mining and engineering staff will be encouraged to domicile within the surrounding communities, close to the Project site.

The work schedule for geology and engineering staff will be 8 h/d, 5 d/wk. This labour force estimate depicts labour force on site and does not account for contractors, medical and general absenteeism, and vacations. The site labour force was estimated based on the Project tasks and responsibilities, working hours, shift rotations, site operating equipment, and operating capacity. A summary of the underground labour force requirements is presented in Table 16-10.

Table 16-10: Estimate Mine Labour Requirement

Description	Labour Force (Avg. LOM)
Stoping Production Driller	12
Sill Narrow Vein Jumbo Driller	13
Development Jumbo (Double Boom) Driller	2
Bolter Driller	3
Trucks Drivers	9
Backfill Crew	4
Grader Operator	3
Blaster – Production	4
Blaster – Development	6
Mechanics and Electricians	18
Surveyor	4
General Miner/Helper	4
Sampler	6
Total	**56**

In addition to the labour requirements listed above, the Project's G&A and technical management will require staff as outlined in Table 16-11.

Table 16-11: General and Administrative Labour Force

Description	Labour Force (Avg. LOM)
General Administration	
General Manager/Mine Superintendent	1
Health and Safety and Training Officer	6
Mine Technical and Engineering	
Chief/Senior Mining Engineer	1
Intermediate Mining Engineer	3
Intermediate Draft Person	4
Senior Surveyor	1
Chief/Senior Geologist	1
Intermediate Geologist	3
Senior Mechanical Engineer	1
Mine Technicians	4
Salaried Staff – Mining	
Administrative Assistant	1
Mine General Foreman	1
Mine Supervisor/Shift Boss (Production)	6
Maintenance Supervisor/Shift Boss	3
Total	**36**

17 RECOVERY METHODS

Information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

The gold at Curraghinalt is associated with chalcopyrite and pyrite. Generally, the gold is free milling and not refractory. The processing options considered for Curraghinalt take into account the association of gold with copper and pyrite.

The four process flowsheet options considered and economically evaluated for this PEA study are:

- Option A: Whole Ore Leach
- Option B: Gravity concentration, sale of gravity concentrate followed by Bulk Flotation Concentrate, Leach of concentrate
- Option C: Gravity concentration followed by Bulk Flotation Concentrate - Sale of both concentrates
- Option D: Copper Flotation, with sale of the copper concentrate and Pyrite Flotation Concentrate with leach of the Pyrite concentrate

These options are shown schematically in Figure 17-1 to Figure 17-4.

Figure 17-1: Option A Whole Ore Leach

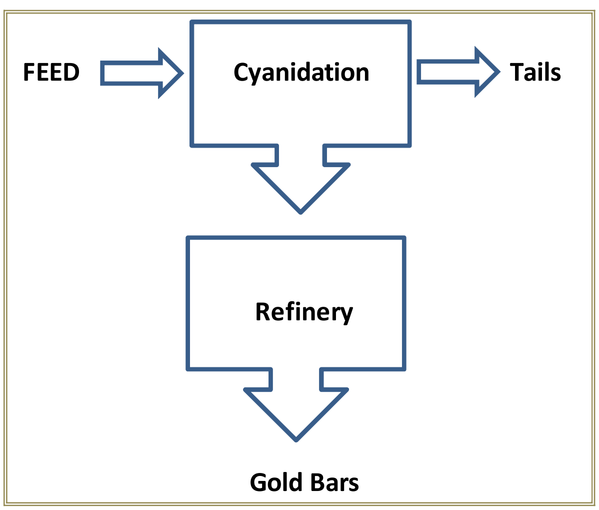


Figure 17-2: Option B Bulk Flotation Concentrate Leach Whole Ore Leach

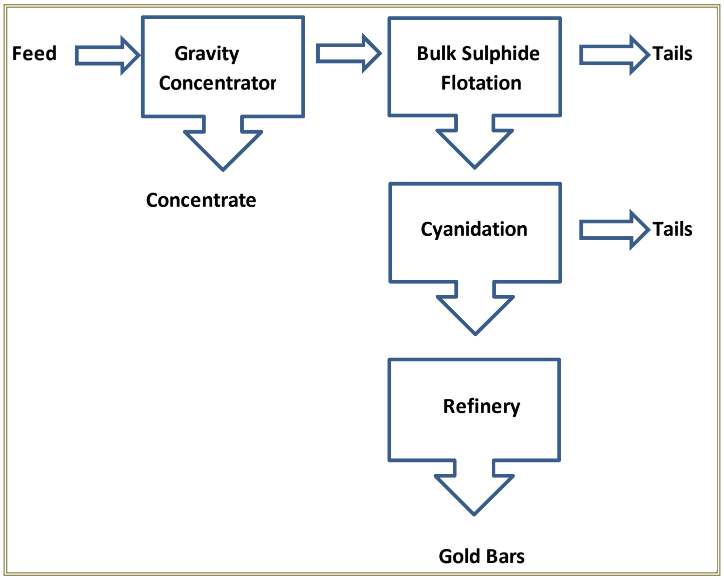


Figure 17-3: Option C Bulk Flotation Concentrate – No Leach

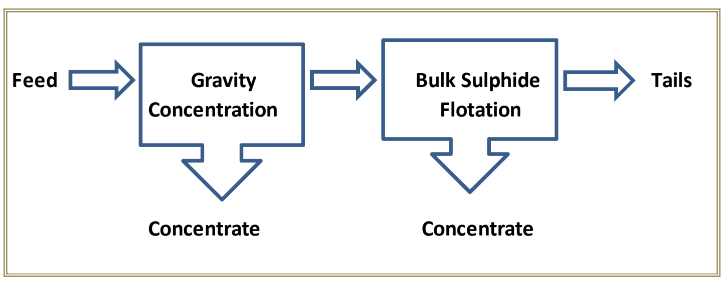
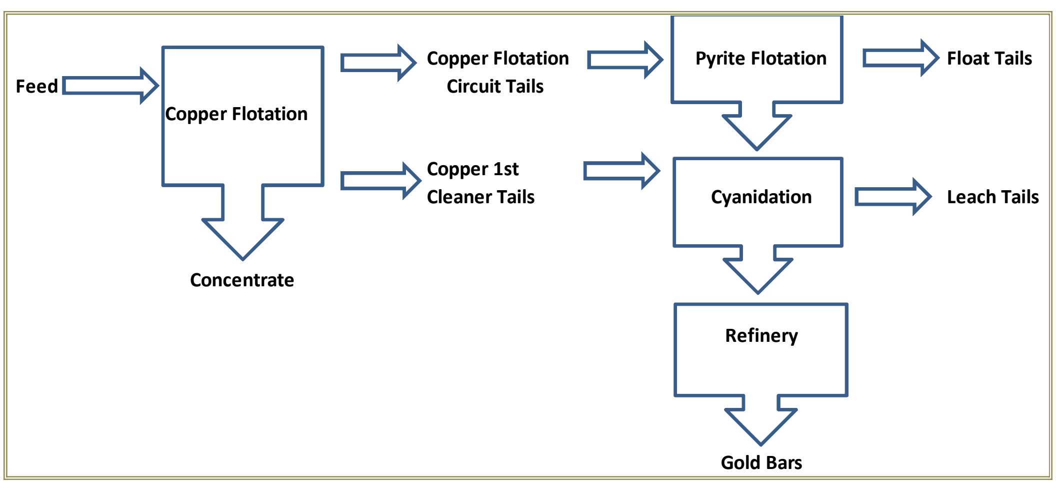


Figure 17-4: Option D Copper Flotation and Pyrite Flotation Concentrate Leach



Operating and capital costs for each of these four options were developed, as well as revenue estimates. All four options had positive outcomes but, following a preliminary internal review, Options B and C were rejected due to their higher operating costs and lower revenues. Detailed estimates were developed for Options A and D and have been included in the PEA.

Options A, and D both involve a similar set of processing steps including Crushing, Grinding, and Cyanidation. Option D includes two flotation steps: producing a copper concentrate for sale and a pyrite concentrate for on-site cyanide leaching.

The source of assay data for this report was recent and on-going testwork being conducted at ALS, and Lakefield testwork completed in 2012 ("An investigation into the recovery of gold and silver from the Tyrone Project prepared for Dalradian Resources Inc. Project 13471-001 Final Report April 23, 2012" prepared by SGS Lakefield.).

17.1 Design Basis

A summary of the project production and process design criteria are presented in Figure 17-1 and Figure 17-2, respectively. The process design criteria are based on the metallurgical testwork discussed in Section 13.

Table 17-1: Production Summary for the Dalradian Resources Tyrone Project Milling Plant

Item	Option A Whole Ore Leach	Option D Copper Flotation and Pyrite Flotation Concentrate Leach
Processed Annually, Average (t/y)	443,179	443,179
Feed Grades (LOM)		
Au (g/t)	8.06	8.06
Ag (g/t)	3.31	3.31
Cu (%)	0.08	0.08
Saleable Cu Concentrate Production, Avg. (t/y)	-	5,585
Gold Production, LOM Avg. (oz/y)	114,841	102,622
Silver Production, LOM Avg. (oz/y)	47,162	38,681
Key Operating Parameters		
Operating Days (d/y)	365	365
Mill Feed Rate (t/d)	1,700	1,700
Chemical and Metallurgical Parameters		
Copper Concentrate Grade (Cu %)	-	8.4

Table 17-2: Process Design Criteria

General Ore Characteristics	Unit	Option A Whole Ore Leach	Option D Copper Flotation and Pyrite Flotation Concentrate Leach
Annual Processing Rate (nominal)	t/y	621,000	621,000
Operating days/year	d/y	365	365
Daily Processing Rate	t/d	1,700	1,700
Average Ore Grades	g/t Au	8.06	8.06
	g/t Ag	3.31	3.31
	% Cu	0.08	0.08
Ore Specific Gravity		2.85	2.85
Crushing			
Crusher Rate - Nominal	t/h	71	71
Crusher Operating Time	h/d	14.4	14.4
Crusher Availability	%	80	80
Crushing Rate – Design	t/h	148	148
Crusher Type		Jaw	Jaw
Crushed Ore Product P_{80}	mm	150	150
Grinding			
Total Feed Tonnage to Grinding – Nominal	t/d	1,700	1,700
Mill Availability	%	92	92
Total Feed Tonnage to Grinding – Design	t/d	1,848	1,848
Feed Rate to Grinding – Design	t/h	77	77
SAG Mill Feed F_{80}	mm	150	150
SAG Mill Discharge P_{80}	µm	850	850
Abrasion Index	kWh/t	0.1278	0.1278
Bond Work Index	kWh/t	15.2	15.2
SAG Mill Unit Power Consumption	kWh/t	11.6	11.6
SAG Mill Installed Power	kW	896	896
Ball Mill Unit Power Consumption	kWh/t	8.6	8.6
Ball Mill Installed Power	kW	672	672
Ball Mill Discharge P_{80}	µm	141	141
Copper Flotation			
Feed Rate to Cu Flotation – Design	t/h	-	77
Cu Flotation Feed Density	% solids	-	40
Lime to Cu Flotation	kg/t	-	2.890
3418A Collector to Cu Flotation	kg/t	-	0.010
MIBC Frother to Cu Flotation	kg/t	-	0.047
Cu Concentrate Mass Pull	%	-	0.9
Cu Concentrate Production – Annual	t/y	-	5,585
Cu Concentrate Production – Design	t/d	-	17
Cu Concentrate Grade	%Cu	-	8

General Ore Characteristics	Unit	Option A Whole Ore Leach	Option D Copper Flotation and Pyrite Flotation Concentrate Leach
Cu Concentrate Recovery	%Cu	-	94
Cu Tailings – Design	t/d	-	1,785
Concentrate Dewatering			
Cu Concentrate Solids Feed Rate	t/h	-	0.7
Cu Concentrate Thickener % Solids	%	-	70
Cu Concentrate Filter Press % Solids	%	-	90
Flocculant Addition	kg/t	-	0.025
Bulk Sulphide Flotation			
Feed rate to Bulk Sulphide Flotation – Design	t/d	-	1,785
Mass Pull to Flotation Product	% of feed	-	6
Bulk Sulphide Concentrate – Design	t/d	-	111
Bulk sulphide production annual	t/y	-	40,565
Bulk sulphide tailings annual	t/y	-	562,173
3418A Collector Addition	kg/t	-	0.060
Cyanide Leaching and CIP			
Feed rate to Cyanide Leaching – Nominal	t/h	71	4.6
Cyanide Leaching Availability	%	90	90
Feed rate to Cyanide Leaching – Design	t/h	79	5.1
Leach Slurry Density	% solids	40	40
Leach Slurry Flowrate	m^3/h	144	9.3
Leach Residence Time	h	24	24
Number of Leach Tanks	#	6	6
Cyanide Concentration	g/L	2	2
Cyanide Consumption	kg/t	0.6	0.6
Lime Consumption	kg/t	1.84	0.6
Lead Nitrate Consumption	kg/t	0.5	0.5
Effluent Treatment			
Feed Rate to Cyanide Destruction – Nominal	t/h	71	4.6
Effluent Treatment Availability	%	90	90
Feed Rate to Cyanide Destruction – Design	t/h	79	5.1
Slurry Density	% solids	40	40
Slurry Flow Rate	m^3/h	79	9.3
Cyanide Destruction Residence Time	h	2	2
No. Cyanide Destruction Reactor Tanks		3	3
Tailings Thickener % Solids	%	60	60

17.2 Process Description

17.2.1 Crushing (Options A and D)

The ore is delivered by mine trucks to the plant and fed over a grizzly with 400 mm openings to the run-of-mine (ROM) ore feed hopper. The ore is extracted from the feed hopper by an apron feeder and fed to a jaw crusher. The crushed ore is discharged onto a conveyor that will transport the ore to the plant. The crusher product is 80% passing 150 mm.

17.2.2 Grinding (Options A and D)

The crushed ore reports to the grinding circuit. The primary grinding SAG mill discharge product typically has a size passing of 850 µm.

The secondary grinding ball mill will operate in closed circuit with a cyclone, with the target cyclone overflow 80% size passing 141 µm.

17.2.3 Copper Flotation (Option D)

The ball mill ore slurry discharge, at 50% solids, is fed to a conditioning tank, which then overflows to the copper flotation circuit. The conditioning tank also acts to buffer flotation from upsets in the grinding areas.

The copper concentrate is produced in a conventional copper flotation circuit. The copper circuit's first cleaner tailings report to the cyanide leach circuit. Tails from the circuit are pumped to the bulk flotation circuit.

Aerophine Collector 3418A, MIBC frother and lime (used as a pH modifier) are added to the copper flotation circuit.

The concentrate from the rougher cells is fed to the cleaner cells from which the final copper concentrate is sent to a pressure filter for dewatering.

17.2.4 Copper Concentrate Dewatering (Option D)

Copper concentrate is thickened to 70% solids in conventional thickener. Copper concentrate thickener overflow is recycled to the water reclaim tank. This reclaim water tank can serve as a headtank for the reclaim water pumps provide the bulk of process water to the mill. It also serves as the water supply for the emergency diesel powered fire pump. Copper concentrates are filtered using pressure filters. This product is then sold to a copper smelter.

17.2.5 Bulk Sulphide Flotation (Option D)

For Option D, tails from copper flotation are subjected to a bulk sulphide flotation to recover all remaining sulphides, pyrite, etc. as well as gold and silver bearing minerals.

The flotation concentrate is thickened and pumped to cyanidation circuit for recovery of gold, and silver.

17.2.6 Pre-treatment and Cyanidation (Options A and D)

A six-stage CIL circuit is installed based on a retention time of 24 hours and a new carbon concentration of 20 g/L. Slurry pH is adjusted to 10.5 with the addition of lime. The first leach tank is aerated, and lead nitrate is added. The oxidation of sulphides in this step reduces cyanide consumption considerably as well as enhances leaching kinetics. The leach circuit feed is pumped to the train of six CIL tanks in series where it is contacted with activated carbon. Each of the tanks is provided with an in-tank screen to allow the slurry to flow by gravity through the CIL train, while retaining the carbon in the tanks. The carbon is transported counter-currently to the slurry flow by means of carbon advance pumps. The loaded carbon from the first CIL tank is collected on a screen and is sent to the carbon stripping circuit. The slurry leaving the last CIL tank is passed over a safety screen to capture any carbon particles before it is sent to the cyanide destruction circuit.

17.2.7 Carbon Stripping / Carbon Reactivation / Gold Room (Options A and D)

A carbon stripping/carbon reactivation circuit is installed to treat the loaded carbon to recover the gold and to recycle the carbon to the CIL circuit.

The loaded carbon is acid washed and then it is fed to a ZADRA elution circuit to strip the gold from the carbon. The pregnant solution is then sent to two electrowinning cells to recover the gold from solution. The carbon from the stripping circuit is fed to a diesel-fired rotary kiln to reactivate the carbon that is passed over a sizing screen prior to being recycled to the CIL circuit.

The stainless steel wool cathodes carrying the gold from the electrowinning cells are fed to an induction furnace to produce gold doré bars for sale.

17.2.8 Cyanide Destruction (Options A and D)

A cyanide destruction circuit is installed to treat the slurry discharge from the CIL circuit.

The CIL discharge slurry is pumped to two agitated tanks in series, and SO_2 and air are added to destroy the cyanide before the slurry is pumped to the tailings thickener. Thickened tailings at 60% solids are pumped to the pastefill plant or tailings disposal area.

17.3 Labour Force

Labour force requirements are based on a continuous operation, 7 d/wk and 365 d/y.

Table 17-3 shows the labour force requirements for the base case (Option A) and Option D.

Option D includes the process operations of Option A, but also includes two additional flotation circuits. Thus, Option D will require nine more personnel than Option A.

Table 17-3: Summary of Estimated Processing Plant Operations Personnel

Description	Option A Whole Ore Leach	Option D Copper Float and Pyrite Flotation Concentrate Leach
Plant Superintendent	1	1
Senior Metallurgist	1	1
Metallurgist	0	1
Chief Chemist	1	1
Sample Preparation Technician	4	4
Assay Technician	2	2
Plant Shift Supervisor	4	4
Crusher Operator	4	4
Plant Operator	4	4
Control Room Operator	0	4
Operator Helper	4	4
Tailings Operator	2	2
Load out Operator	0	4
Cyanidation Plant Operator	4	4
Day Labour Crew	4	4
Maintenance Superintendent	1	1
Maintenance Supervisor	1	1
Instrument Technician	2	2
Electrician	4	4
Plant Mechanic	4	4
Security Personnel	5	5
Senior Accountant	1	1
Accountants/buyer	1	1
HR and Community Relations	1	1
Environmental Technician	1	1
Supervisor	1	1
Bus Driver	1	1
Secretaries / Administrative Assistants	2	2
Site Surface Maintenance Crew	4	4
Equipment Operator	2	2
Site Administrator	1	1
Total	**67**	**76**

17.4 Processing Conclusions

The capital and operating costs for the concentrator have been estimated on the basis of available data. The PEA considers four possible process flowsheets for the extraction of gold from the Curraghinalt resource. Of these, the whole-ore leach (Option A) was considered the most effective and has been used as the base case for project evaluation. It is recommended that this option should be explored further in future testwork, including optimizing grind, reagent strengths and retention times, and pilot studies are required before flowsheet development and design specifications can be finalized

Micon considers the relatively low grade and elevated concentrations of penalty elements of the copper concentrate produced in Option D will present a challenge with regard to shipping and marketing. Dalradian Resources will need to confirm off-take terms with a smelter, and also investigate any deleterious content of the copper concentrate.

The PEA has been prepared using the metallurgical testwork results available at the time. Subsequent and ongoing testing by ALS Metallurgy suggests that further investigation of the Option B and C flowsheets is worth pursuing as an alternative to whole ore cyanidation.

18 PROJECT INFRASTRUCTURE

Information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

18.1 Site Access Road

The property is readily accessible by a network of existing all-weather paved highways and local roads; more specifically, these include the two-lane paved road B48 from Omagh to the village of Gortin, and from there, the B46 road to Greencastle.

The existing adit is accessed from a narrow road on the north side of the Curraghinalt deposit. However, Micon has determined that the property should ultimately be accessed from the south (off the B46), which would have the significant benefit of reducing the potential for congestion associated with the northern access.

18.2 Site Roads

A site road system will be built to connect the on-site facilities: these include the gatehouse, administration complex, process plant, mine dry, warehouse, ore and waste storage pads, and tailings storage facility (TSF).

18.3 Power Supply

The total power requirement for the base case is 70 kWh/t, which results in a maximum demand of approximately 6 MW for a 1,700 t/d operation. Power will be supplied by connecting to the national high voltage electricity grid.

Due to the required demand, a predominantly high capacity 33 kV overhead transmission line will be built from the 110/33 kV substation located at Strabane, for the approximately 27 km distance to the site. It is expected that it should follow the existing right-of-way (ROW) to Plumbridge, then parallel the existing 11 kV line to Gortin, and from there run directly to the site. There is another 110/kV substation located at Omagh, which is equidistant, but according to Northern Ireland Electricity Ltd. (NIE), does not have enough capacity. In addition, NIE has also indicated that it is planning to install a new 33 kV switchboard at the Strabane site within the next 18 months. Another option would be to tie in to the existing 110 kV transmission line located between Strabane and Omagh and build a 33 kV substation at the site itself. It would take NIE approximately two years to complete the power line construction, once it receives a firm commitment from Dalradian.

A substation will be built on site which will transform the electrical power from 33 kV to 11 kV for distribution around the site.

In case of interruption of the main power supply, a 1.12 MW, 400 V standby diesel generator located on-site will automatically be activated through the use of a motor control centre (MCC) in order to supply emergency power.

18.4 Water Supply

Micon considers that options for make-up water for the process plant are fairly limited. Although the Owenkillew River is located near the edge of the property on the North side, it is a protected habitat with Areas of Special Scientific Interest (ASSI), making its usage unlikely; even if it were possible to use, it would require a long lead time and substantial efforts in order to obtain the necessary permits from the governmental agencies.

A hydrogeology survey report produced by SLR indicates that the use of water wells on the property is not a likely option either. The report states that, according to the GSNI, the metasediments of the Mullaghcarn, Glengawna and Glenelly Formations are all considered to have limited potential for containing significant bedrock aquifer. Most supply wells in these metasediments are known to have only a low yield and the occurrence of moderate yields are unusual. However, it is important to mention that SLR recommends undertaking a survey of local wells around the adit area.

Micon considers that, currently, the most likely viable option is to construct a rainwater catchment system to supply the make-up water for the initial process plant start-up. The SLR report states that rainfall in the area is in the order of 1,300 mm/y to 1,400 mm/y, with a considerable net residual residue for runoff water, probably in the order of 1,000 mm/y. Micon proposes using the proposed 25 ha TSF to capture the rainwater, and possibly to store pumped runoff water if necessary.

In Micon's opinion, more work is required in order to properly assess water supply sources, and Micon recommends that additional investigations be carried out during further development of the Project.

18.5 Buildings

As previously noted, the site will accommodate a process plant, a two-storey administration complex that will also house the technical department (total available space 1,060 m²), laboratory, a single-storey mine dry complex incorporating mine supervisors' offices (covering an area of 816 m²), a maintenance shop for underground and surface equipment (with an area of 896 m²), a heated warehouse (756 m²), and a gatehouse.

18.6 Ancillary Facilities

Water tanks for industrial use will be built, including one for fire suppression water. The fire suppression water will be distributed to the protected area through a network of underground water pipes.

A fuel tank farm with adjacent fuelling station will primarily supply underground equipment needs.

Sewage will be collected from the various facilities and pumped into a sewage treatment plant.

Communication facilities will consist of a redundant fibre optic communication backbone system that will link and manage the data transmissions of the distributed control system (DCS), third-party programmable logic controllers (PLCs), motor controls, fire detection system, and computers around the mine site. It is expected that the site can be connected to the local phone system network. If this not possible, a Voice over Internet Protocol (VoIP) telephone system will be required. In any case, a transceiver or cellular radio tower will be installed in order to optimize the use of cellular telephones, as current coverage in the area is inadequate.

For the duration of the construction work, portable temporary office and facility trailers will be used.

18.7 Construction Fill Materials

In order to reduce capital costs and land disturbance, Micon recommends using non-PAG waste rock produced by further underground exploration and mine development activities as bulk fill materials to meet construction needs, where appropriate. Structural fill material will be imported from local suppliers as required.

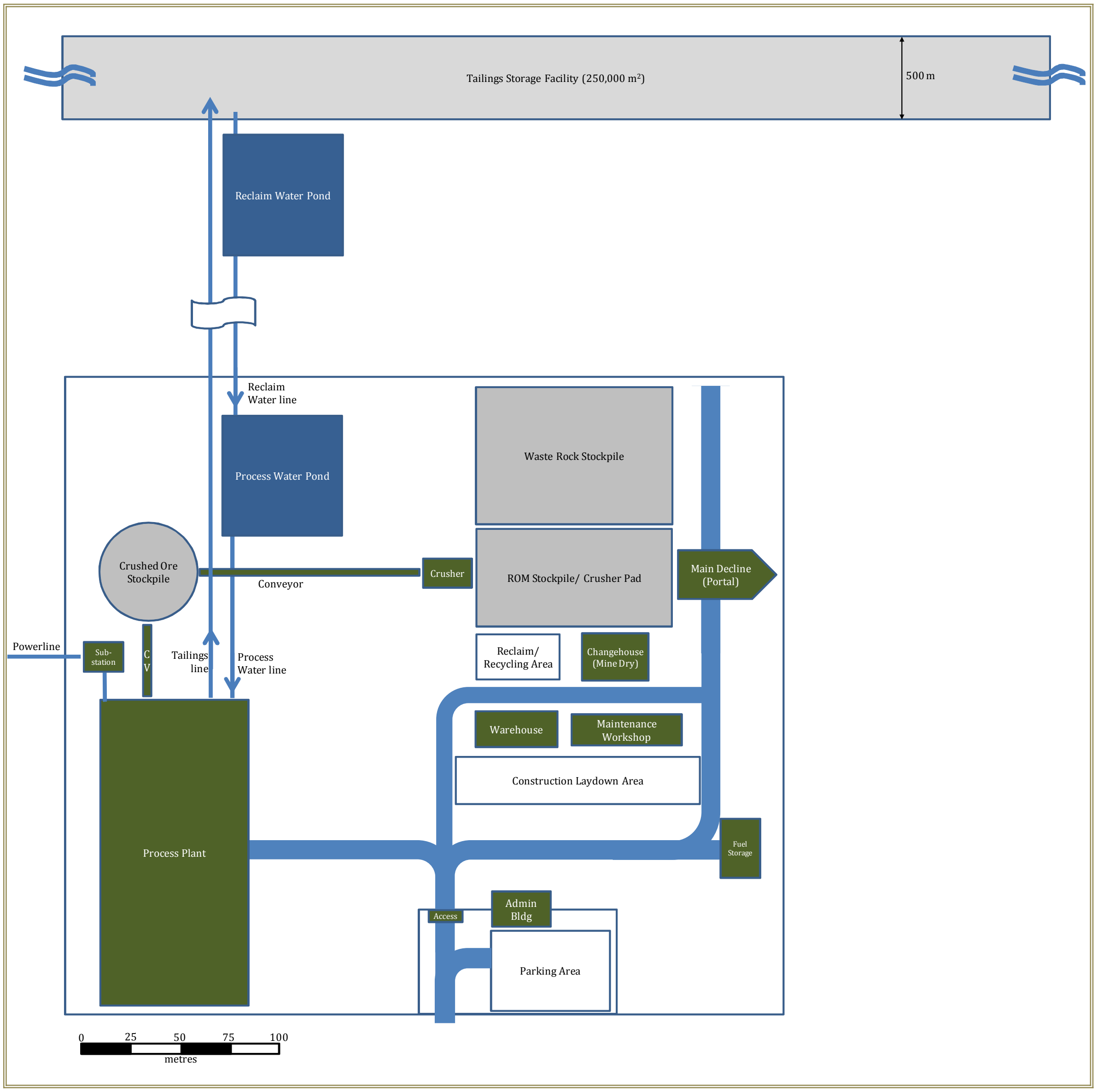
18.8 Tailings Storage Facility

In 2011, Golder was retained by Dalradian to carry out a Tailings Management Facility Site Selection Study aimed at identifying potential sites to store approximately 2.92 Mt of tailings. In its December 2011 report, Golder identified seven potential sites and recommended three of them (in order of preference, Sites 3, 7, and 6) for further investigation. Golder also recommended the use of conventional tailings disposal technology, and that the construction work be carried out in two separate phases (Phase I and Phase II).

18.9 Schematic Site Layout

At this stage of Project development, there is a high degree of flexibility in the siting of the main ramp portal, ventilation shaft collars, process plant, and other surface infrastructure. Therefore, Micon has not made specific recommendations as to the location of these works, pending discussion with affected landowners. Accordingly, the layout shown in Figure 18-1 is schematic only, in the expectation that negotiation with landowners will take place as the Project moves forward, so that a site-specific design can be developed during the next stages of Project engineering.

Figure 18-1: Schematic Surface Layout



19 MARKET STUDIES AND CONTRACTS

Information in this section is taken from the (PEA) published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

In evaluating the Project, Micon made assumptions about the terms on which products from the Project might be sold which, based on its experience on similar projects elsewhere, it believes to be reasonable. Micon is not aware of any Project-specific contract or off-take terms having been negotiated for sales from the Curraghinalt Project.

Prior to entering production, significant contracts will be required for:

- the supply of electrical power to the site
- the supply of fuels, explosives, cement, lime, sodium cyanide, and other reagents and consumables for the mining and processing activities.

Micon has used publicly available information to derive its estimate of the cost of power required by the Project from the relevant utilities, and has used knowledge and experience gained on other projects to derive other significant input unit costs. Micon is not aware of any Project-specific contracts or terms having been negotiated for supplies to the Curraghinalt Project.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies and Issues

20.1.1 Introduction

This section provides a summary of environmental studies completed by SLR Consulting (SLR), SRK, and Golder, and an indication of the permitting and management plans needed to take the Project into production. Dalradian Resources has provided existing permit information, but this does not constitute a legal opinion on the status of existing permits.

Landscape and Ecology

The Sperrin Mountains are designated an Area of Outstanding Natural Beauty. There are also several protected and special interest areas around the Project (Figure 22-1).

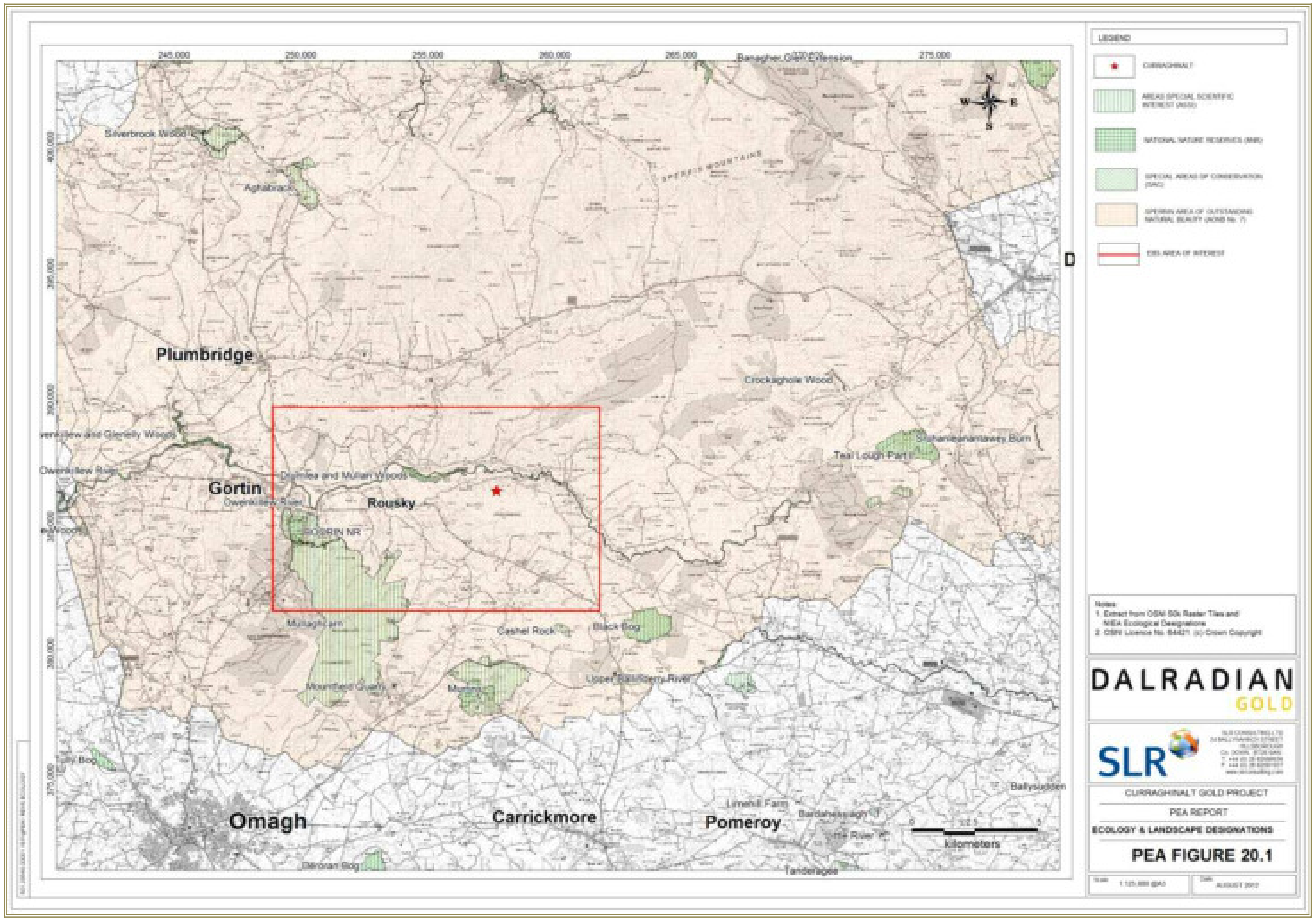
The nearest Areas of Special Scientific Interest (ASSIs) to the Project include the Owenkillew River, Mullaghcarn/Mountfield Quarry, Murrins, Cashel Rock, Boorin Wood, and Black Bog. The nearest Special Areas of Conservation (SACs) include Drumlea and Mullan Woods, Owenkillew River, and Black Bog.

Within the Owenkillew River SAC are four Annex II listed species: freshwater pearl mussel (*Margaritifera margaritifera*), river otter (*Lutra lutra*), brook lamprey (*Lampetra planeri*) and Atlantic salmon (*Salmo salar*).

Environmental Baseline Studies

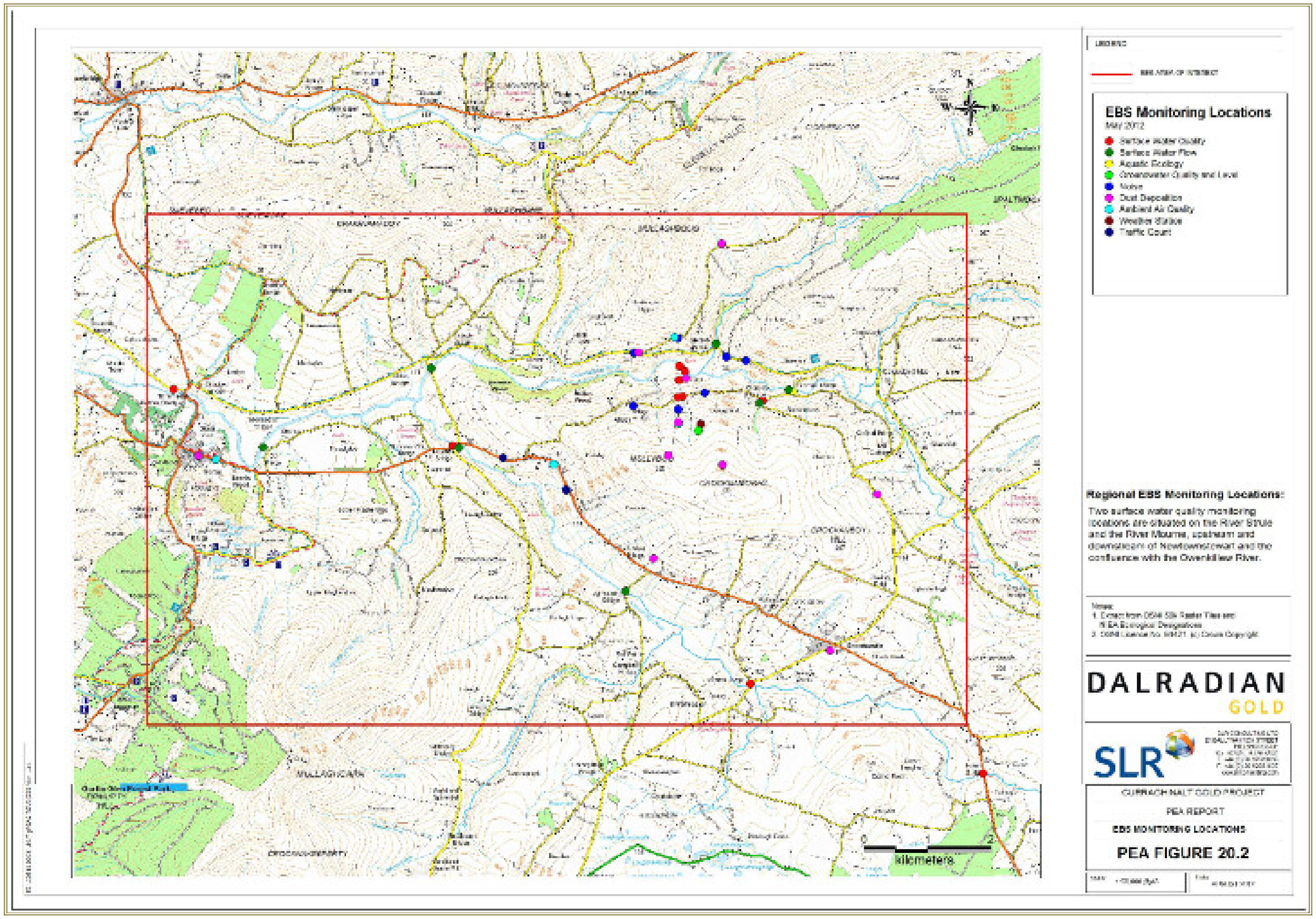
SRL initiated environmental baseline studies for Dalradian Resources in June 2010, and have collected data on meteorology, hydrology, hydrogeology, water quality, sediment quality, acid–rock-generating potential of the mineral and waste rock, flora, terrestrial and aquatic fauna, air quality, traffic, visual resources, cultural heritage resources, and the local socioeconomics. Initial monitoring locations for physical and biological components are shown in Figure 20-2.

Figure 20-1: Ecological and Landscape Designations



Source: SLR, August 2012

Figure 20-2: Environmental Monitoring Locations



Source: SLR, August 2012

Surface Water

The following section is abstracted from the SLR Draft EBS Hydrology Report (SLR, 2013a). Surface water flow is monitored at eight stations on the Owenkillew catchment area, which includes the Owenkillew, Owenreagh and Glenlark Rivers, and the Curraghinalt and Glenealy Burns. Hydrology data indicate that the flow regime in the rivers is extremely flashy, with steep stream and river hydrographs. The flow monitoring indicates that there is little precipitation storage in the catchment, and therefore rainfall runs off quickly to the streams and rivers, during and immediately after precipitation events.

Surface-water quality is monitored at fifteen sites. Water quality is good for aquatic life; there are naturally occurring elevated concentrations of copper, iron, and zinc, as well as ammonia, nitrite, and phosphorous, likely associated with runoff from local farming activity. The marginally elevated concentrations of heavy metals in surface water are likely a result of mineralization in this region. To date he background copper concentration detected in surface water bodies near the adit ranges from 2.3 µg/L in the Owenkillew River, to 12.8 µg/L in nearby tributaries of the Owenkillew River; the maximum background copper concentration in drainage from the existing adit is 10.0 µg/L.

Groundwater

There are three principal groundwater bodies near the Curraghinalt Project (SLR, 2013b):

1. A superficial aquifer in the fluvioglacial sands and gravels typically located within the base of the valleys, is likely to be in hydraulic continuity with the Owenkillew River
2. Groundwater in the weathered bedrock, which typically comprises the upper 4 m to 5 m of the rock mass
3. Deeper groundwater in the unweathered bedrock, the flow of which is limited to fractures and faults within this rock mass, the bulk of the rock being effectively impermeable.

There is also likely to be a superficial aquifer in the peat deposits south of, and above the existing underground workings, above approximately 200 masl.

Significant groundwater flow within the bedrock is restricted to the upper weathered zone and/or fracture zones within the bedrock. Groundwater yields are typically low unless a major fracture zone is encountered in the bedrock. An inspection of the adit currently on the site revealed that the majority of the rock mass was effectively dry, with little or no water entering the tunnel.

Fisheries

The Owenkillew River is one of the principal catchments of the Foyle system in the northwest of the island of Ireland, which straddles the international boundary between Northern Ireland and the Republic of Ireland. The Owenreagh River is one of the larger tributaries of the Owenkillew (Paul Johnston Associates, 2014).

The Owenkillew hosts a substantial stock of Atlantic salmon, which supports a popular recreational fishery. The Owenreagh is a key tributary of the main river, although less significant as a recreational fishery due to its smaller size and lower incidence of deeper pools.

The Owenkillew River is designated as an SAC and ASSI with freshwater pearl mussel as the species providing the primary reason for the site's selection. The Atlantic salmon is noted as a qualifying feature, but not a primary reason for selection as an SAC.

Soils

Soils in the Project area range from predominantly blanket peats on upper slopes, to humic gleys on lower slopes in the valleys, and podzols on the valley floors (SLR, 2011a).

The upper slopes in the Project area have been assessed as blanket bog, with lesser proportions of wet heath / acid grassland mosaic. Historical peat cutting by hand is evident across a large part of the blanket bog. Peat cutting has removed approximately the top 1 m. Active hand cutting of peat has left bare peat; otherwise, vegetation has regenerated in areas cut over in the past. Further degradation of the peats has occurred through burning, drainage and damage caused by livestock grazing. Peat thickness varies from <5 cm to over 3 m. On higher elevations and flatter areas, the peat typically has a greater depth than on steeper slopes and lower elevations (SLR, 2014).

The lower elevations within the Project area are mainly grasslands used for farming with taller shrubs and forested areas bordering farms and along the river valley floor (Figure 20-3).

Figure 20-3: Landscape of the Project Area

20.2 Waste and Water Management

20.2.1 Waste Characterization and Waste Management

SLR undertook geochemical investigations to determine the potential for acid generation in the proposed operation. In December 2011, 35 samples were collected to represent rock types likely to be encountered during the mining operation. The testwork included acid base accounting (ABA), net acid generation and acid rain leach procedure (ARLP) tests aimed to determine whether and which of the mined materials would be acid-generating or metal-leaching, or both.

Based on the ABA laboratory results, SRL concluded that 5 sampled rocks were considered acid generating, 23 were classified as NAG, and the results of the remaining 7 samples were considered to be inconclusive. Rock types that were PAG included the following: oxidized semi-pelites with quartz veins and chlorite or pyrite; semi-pelite/psammite with chlorite stubs >2 mm; and psammite with predominant quartz vein (SLR, 2012b).

Leach tests indicated that the potential contaminants of concern are aluminium, arsenic, iron, manganese, copper, iron, molybdenum, nickel, and lead. The rock type that was most problematic was the semi-pelite/pelite with potassium feldspar and hematite veinlets, under pH3 conditions (SLR, 2012b).

SRK carried out additional testwork in 2013 in support of the permitting process for the proposed underground exploration development program (SRK, 2013).

A series of short-term (static) geochemical tests were undertaken to allow preliminary assessment of acid rock drainage and metals leaching (ARDML) characteristics of rock that may be extracted during mining. For the purposes of the ARDML assessment, 37 samples plus 4 quality-control duplicates were collected. The samples are considered representative of the major waste-rock material types that will be generated at the Dalradian mine. Samples were collected from drill core generated during the exploration programme. Tests undertaken on the waste rock were ABA, multi-element assay of solids, deionised water leach, NAG and NAG leachate, and mineralogical assessment.

A total of 37 core samples and 4 duplicate samples were collected from around the existing underground workings, in the planned adit extensions, and in rock up to 800 m from the existing workings.

Based on EU Directive 2006/21/EC, relating to mine waste, 7 samples were identified as PAG and 34 were identified as non-acid generating (NAG) during the ABA testwork. The 7 PAG samples were distributed throughout the sample area, and belonged to two lithology groups, 3 samples were from the semi-pelite (Ssp) lithology group and 4 were from the psammite-pelite (Sps) lithology group. Based on the samples assessed, the pelite (Spe) group does not exhibit acid-generating potential.

To confirm the results of the ABA data, NAG testing was undertaken on all samples; however, at the time of this report only the 4 high sulphide sample results were available for comment. The NAG pH values were generally above 7 s.u., which indicates there was no net acid generation. However, one sample (SRK3285) had a NAG pH of 4.6 s.u. indicating the potential for acid generation. The NAG results did not support the findings from the ABA testwork, possibly as a result of the presence of iron carbonates within the samples. All samples were classified as NAG due to NAG pH results >4.5, and low total net acid generation values (<1 for a majority of samples).

Metals-leaching tests showed cadmium and cobalt were leached in concentrations below the limits of detection, and copper, lead and nickel were leached in concentrations compliant with International Finance Corporation (IFC) and EU guidelines. Zinc concentrations for one sample (SRK 3253) exceeded by 0.001 mg/L the most stringent guideline in Directive 2000/60/EC.

Sulphate was leached only from 6 Ssp and 7 Sps samples, whereas arsenic was released by all samples and lithology groups. One sample exceeded IFC guidelines for arsenic concentrations, and 27 samples exceeded the most stringent guideline in Directive 2000/60/EC.

Further testing of waste rock and tailings conducted during subsequent stages will provide additional information to assist with segregation and management of the various rock types during operations.

An estimated 40% of tailings will be backfilled underground. The remaining tailings will be disposed of at the surface in a conventional tailings impoundment, which will be lined with synthetic or impermeable material, or both. It is recommended that the Project consider paste or dry-stack tailings to minimize the footprint, and increase the available options for siting the tailings and progressive reclamation to minimize visual impacts.

PAG waste rock will be backfilled underground and ultimately flooded after closure, which will minimize future oxidation and metal release. The remaining waste rock will be stored in surface dumps, which will be lined if required, contoured, and eventually capped and revegetated in character with the surrounding landscape.

20.2.2 Water Management

Temporary abstraction permits granted by the Northern Ireland Environmental Agency (NIEA) were in place for the exploration-drilling program to provide for drilling makeup water. There are no other groundwater abstraction licences in the immediate Project area. A discharge consent has been obtained from the NIEA for site drainage from the existing exploration adit during any future underground exploration works.

Mine water will be collected in underground sumps and pumped to surface for use as makeup water in the process plant. This water may be treated prior to use in the process, depending on the concentrations of suspended sediments and blasting residues.

The process plant is designed to include an INCO SO_2/air cyanide destruction circuit to meet the limit of 10 ppm weak acid dissociable (WAD) cyanide at the point of discharge to the tailings pond, as required under Directive 2006/21/EC, Article 13(6). Cyanide concentrations will then degrade and attenuate within the tailings pond through various processes including complexation, precipitation, adsorption, oxidation, reaction with sulphur, volatilization, biodegradation, and hydrolysis (www.cyanidecode.org). Slurry tailings will be pumped to the tailings impoundment and pond water will be recycled back to the process plant via a reclaim barge and pipeline. The impoundment will be designed to minimize seepages. Monitoring wells will be placed hydraulically down-gradient of the tailings impoundment to monitor and collect seepage, which would be pumped back to the pond if necessary. The tailings impoundment will be fenced to prevent livestock access and other measures may be needed to deter birds, depending on the pond water quality.

Any additional makeup water will be obtained from abstraction of groundwater or surface water, depending on results from more water-balance studies to be completed during the subsequent stages of the Project.

Post-closure, the adit will be plugged, and underground workings allowed to flood. For the tailings impoundment at closure, it is assumed tailings pond water will be recycled back through the plant, and treated as necessary until the pond water-quality is sufficient for direct release of any annual surplus water. Groundwater and surface waters will need to be monitored post-closure.

20.2.3 Social and Environmental Management

A number of specific social and environmental design criteria should be implemented to meet the conservation and visual landscapes objectives of the surrounding area. Based on their Landscape and Visual Baseline Study SLR (2011b) made the following recommendations:

- tree and shrub plantings for screening and reclamation should use native species and conform to the surrounding landscape
- plant vegetation for screening before construction to allow vegetation to establish and minimize visual impacts during development
- minimize visual impacts from the tailings pond and stockpiles by progressive reclamation and use vegetation for screening around the perimeters
- design buildings in colours, styles, and size similar to those in the area, and use earth-sheltered buildings if the building will exceed the size of a typical farm shed
- use earth berms and stone walls to screen development and maintain the character of the surrounding landscape
- haul roads should be designed to follow existing field boundaries and landscape contours
- retain existing vegetation in accordance with recommendation in BS5837:2005, *Trees in Relation to Construction*.

Following good industry practice, it is assumed a social, environmental, health, and safety management system will be implemented to meet the Company's commitments to protect the environment and the health and safety of the workers and surrounding communities. The management system and plans should be designed to monitor and maintain permit compliance and a social licence to operate.

20.3 Permitting Requirements

20.3.1 Permits for Exploration

DGL has a mining lease option agreement with the CEC for gold and silver on DG1, DG2, DG3, and DG4. In addition DGL also have four mineral prospecting licenses issued by the Department of Trade and Industry (DETI)—DG1/14, DG2/14, DG3/08, and DG4/08—to prospect for other minerals in County Tyrone and

Londonderry. The Licence and Prospecting Licences permit DGL to undertake exploration activities subject to a number of conditions.

Exploration activities within the Project area are generally permitted development and do not require planning permission. However, DGL is required to notify the Department of the Environment (DOE) Strategic Planning Division in writing prior to exploration works taking place; DOE must be informed of Project details, including the location of boreholes, target minerals, details of all plant and operations, the anticipated timescale, and confirmation that the works will not be undertaken within a designated area such as an ASSI. Appropriate notifications have been sent to the DOE for DGL's exploration works.

In addition, the Health and Safety Executive Northern Ireland (HSENI) is also provided with notification of exploration boreholes, in accordance with the 1995 Borehole Regulations. Appropriate notifications have been sent to the HSENI for DGL's exploration works.

The proposed extension of the existing exploration adit at Curraghinalt does not qualify as permitted development owing to the scale, time frame, and nature of the proposed works. Planning permission for these works has been obtained from the DOE Strategic Planning Division.

Dalradian Resources reports that, to date, it has negotiated, or is in the process of negotiating with all the landowners, land access and compensation agreements for drilling programs. Additional access agreements will be required when drilling takes place to the east and west of the known mineralization. Dalradian Resources indicates that it expects to be able to successfully negotiate access to the required land to undertake any future drill programs at the Curraghinalt Project.

20.3.2 Permit Requirements for Development

Project development is subject to legislative requirements from the European Union, England (UK), and Northern Ireland. Applicable environmental legislation is listed in

Table 20-1. In addition, a number of international conventions will apply to this Project and need to be considered in further planning.

An initial meeting with the Planning Service was held February 22, 2011 to introduce the Project and the Project Scoping Report to the authorities, and to initiate the above-noted environmental baseline studies. The Project requires consultation and preparation of an Environmental Statement (ES), since it could potentially have significant socio-environmental effects. Baseline information and Project engineering is being completed, and will be used for preparing the ES. At this time, it is uncertain how long it will take to permit the Project.

Once decisions are made on the final location of infrastructure and facilities, land acquisition will also need to be negotiated.

Table 20-1: Key Applicable Environmental Legislation

Name of Legislation	Jurisdiction	Date Adopted
Assessment of Effects of Certain Public and Private Projects on the Environment (EIA Directive) (Directive 97/11/EC of 3 March 1997 amending Directive 85/337/EEC)	European Union	1997
The Planning (Environmental Impact Assessment) Regulations 2012 SR 2012 No 59	Northern Ireland	2012
Environmental Protection Act	United Kingdom	1990
Climate Change Act	United Kingdom	2008
Directive 2006/21/EC Management of Waste from Extractive Industries and Amending Directive 2004/35/EC	European Union	2006
Waste Directive (Directive 2008/98/EC)	European Union	2008
Ambient Air Quality and Cleaner Air for Europe (Directive 2008/50/EC)	European Union	2008
Industrial Emissions (Integrated Pollution Prevention and Control) (Directive 2010/75/EU)	European Union	2010
Assessment and Management of Environmental Noise (Directive 2002/49/EC)	European Union	2002
Habitats Directive	European Union	1992
Birds Directive (Directive 2009/147/EC)	European Union	2009

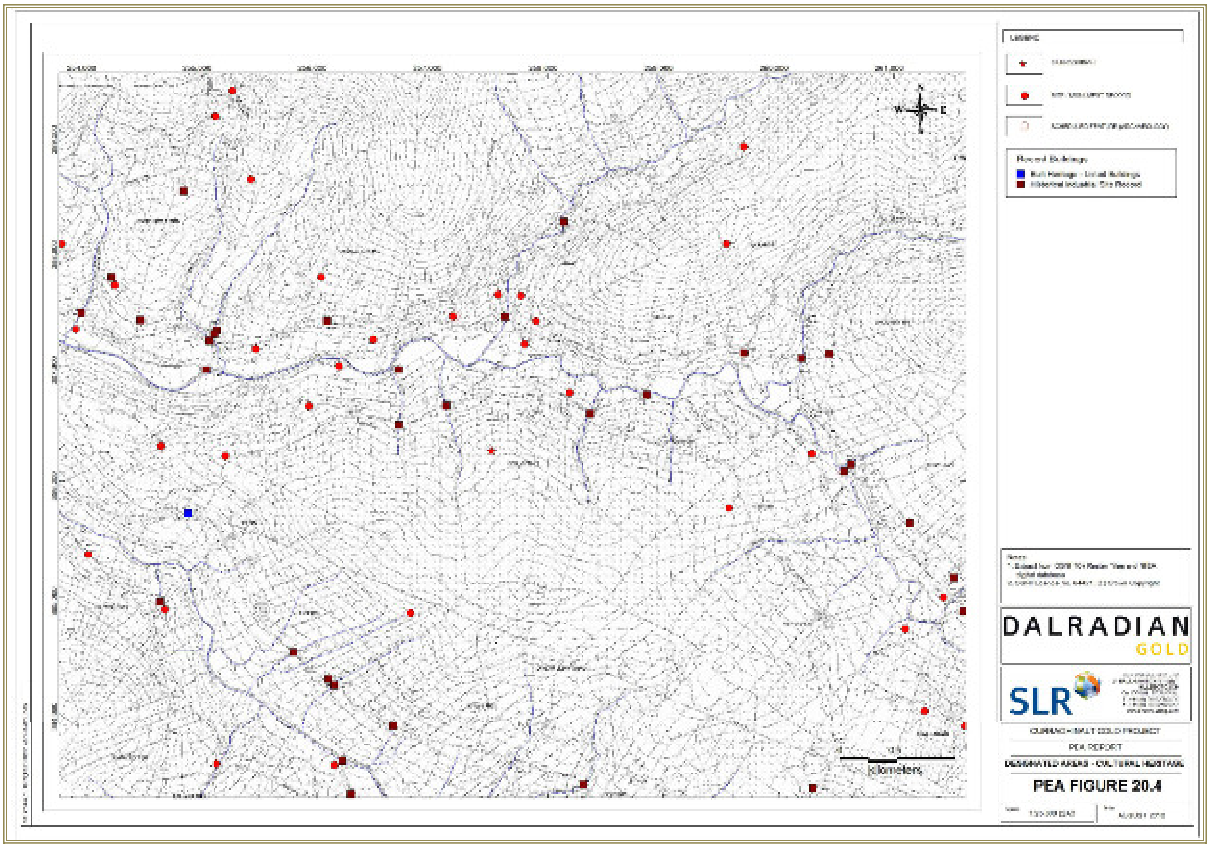
20.4 Social and Community Aspects, Stakeholder Consultation

Curraghinalt is located on the south side of the Owenkillew River, approximately 7.5 km east of Gortin. There are a number of groups of residential houses and farm holdings along the B46 road south of the Crocknamoghil hill, including Rousky, Casorna and Teebane West. There are also rural residences and farms north of the Project along Camcosy Road, south of the Owenkillew River, and along Gorticashel road, north of the river. Agriculture is the predominant land use in the local area.

Cultural heritage resources will need to be considered in the final siting of Project components. A number of sites of cultural heritage are designated around the Project area, including two Industrial Heritage Sites (river crossings) and a site monument located in the near vicinity of the exploration adit. Many other sites in the surrounding area located generally along roads, valley bottoms, or at viewpoints (Figure 20-4; SLR, 2011a).

The largest urban centre is Omagh southwest of the Project site. The socioeconomic impact assessment is not yet complete, but for the purposes of the PEA, it is anticipated that Omagh would be the closest centre for obtaining minor supplies and services for the Project, and that Belfast would likely be the major supply and services centre. It has also been assumed that there will be sufficient local housing for the majority of the operations workforce.

Figure 20-4: Cultural Heritage Sites



Source: SLR, August 2012

Dalradian Resources is consulting with various agencies and associations listed below, and community members, as part of the environmental assessment and planning process. Initial stakeholders include:

- The Crown Estate Commissioners
- Department of the Environment, including The Planning Service
- Northern Ireland Environment Agency (NIEA)
- Rivers Agency
- Omagh District Council and neighbouring local authorities, including Environmental Health Officers
- Department of Agriculture and Rural Development (DARD), including Rivers, Quality, Veterinary, Forest and Countryside
- Department for Regional Development (DRD) – Roads Service
- Department of Enterprise, Trade and Investment (DETI) – Northern Ireland Tourist Board (NITB), GSNI, Health and Safety Executive NI (HSE NI)
- Department of Health, Social Services and Public Safety (DHSSPS)
- Department of Culture, Arts and Leisure (DCAL)

- Northern Ireland Electricity (NIE)
- Northern Ireland Water
- Northern Ireland Housing Executive
- Fisheries Conservancy Board for Northern Ireland (FCBNI)
- Loughs Agency
- Industrial Pollution and Radiochemical Inspectorate (IPRI).

20.5 Mine Closure Requirements

The objective of the mine closure plan is to remove and close down activities in a manner that ensures public safety, and to reclaim the land to a usable state consistent with the surrounding land-use objectives. The visual landscape objective is to minimize disruption of the outstanding natural beauty of the area during operations, and restore this value at closure.

Closure will consist of plugging and securing underground openings, removing from the site to licensed facilities any hazardous and contaminated materials, decommissioning and demolition of facilities and buildings, and reclaiming waste rock and tailings facilities. It has been assumed that the electrical substation is an infrastructure asset that will be left in place post-closure and owned by the utility.

An adit plug will be engineered and constructed at closure to prevent access to the underground workings and allow the underground workings to flood. Mine flooding will slow down oxidation of remaining sulphides exposed within the drifts, and so reduce acid production and help prevent contaminants in discharges.

Unused fuel and reagents will be removed from site and either sold or disposed of at an approved hazardous waste facility. Used oil and oil filters, fuel storage tanks, and contaminated soil will be removed from site and disposed of at an approved hazardous waste facility.

When possible, equipment and machinery will be sold or recycled. Buildings will be demolished and reclaimed, recycled, or disposed of. Concrete foundations will be broken up and removed from site, reclaimed if possible, or buried at a certified waste disposal facility.

Tailings facility seepage and surface waters will be monitored. Seepages will be collected and pumped back to the pond if the quality is not acceptable for release. Tailings pond water will be monitored and pumped back to the plant for treatment if necessary, until the pond water is acceptable for discharge to the environment. Once acceptable, a spillway will be constructed in the dam and the dams and beach areas capped with overburden and revegetated. Similarly, any NAG waste rock stored on the surface will be capped with overburden and revegetated. It is assumed that to minimize oxidation any PAG waste rock will be used in backfill and flooded after mine closure.

Reclamation costs are estimated at $7.5 million in Year 16. Using a real discount rate of 2%, it is assumed for the PEA that the present value (approximately 71%) of this amount will need to be posted as a financial guarantee at the start of the Project. A financial guarantee is required to meet requirements of Directive 2006/21/EC, Management of Waste from Extractive Industries.

21 CAPITAL AND OPERATING COSTS

Information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

21.1 Capital Costs

Capital expenditures and capitalized development costs for the base case are summarized as initial and sustaining costs in Table 21-1. The estimates are expressed in second quarter 2012 Canadian dollars, without escalation, unless otherwise noted. The expected accuracy of the estimates is ±30%.

Table 21-1: Capital Cost Summary

Area	Initial Capital Cost ($ '000s)	Sustaining Capital Cost ($ '000s)
Capitalized Development	15,745	64,925
Mining Equipment	14,770	33,058
Processing	50,743	-
Infrastructure	49,833	16,301
Indirect Costs	12,927	-
Owner's Costs	17,386	-
Contingency	38,341	-
Total	**199,745**	**114,284**

Expressed as US dollars, initial and sustaining capital amounts to US$192.1 million and US$109.9 million, respectively.

21.1.1 Mine Development

The initial capital cost estimate for mine development comprises haulage access, cross-cuts, and ventilation systems development, as well as pre-production operating costs which are capitalized. Sustaining capital costs include only direct costs for haulage access, cross-cuts and ventilation raises. Table 21-2 shows a breakdown of these development costs.

Table 21-2: Capital Cost Summary – Mine Development

Area	Initial Capital Cost ($ '000s)	Sustaining Capital Cost ($ '000s)
Ramp	846	16,085
Ramp's bypasses	17	320
Ramp's safety bays	13	245
Sumps in ramp	13	245
Cross cuts	2,185	45,341
Sumps in cross cuts	21	247
Vent, ore, and waste raise/pass	21	2,444
Other pre-production mining	12,630	-
Total	**15,745**	**64,925**

21.1.2 Mining Equipment

The initial capital cost estimate for mining comprises fleet purchase costs, as shown in Table 21-3. Sustaining capital comprises the replacement of this equipment at appropriate intervals over the life of the mine.

Table 21-3: Capital Estimate – Mining Equipment

Area	Initial Capital Cost ($ '000s)	Sustaining Capital Cost ($ '000s)
Drilling equipment, including jumbos	4,798	11,208
LHD (3 and 4 m^3)	2,229	7,212
Truck (30-ton payload)	1,864	4,660
Ancillary vehicles	1,120	1,830
Pickup trucks	270	810
Fans	270	203
Pumps	90	68
Other equipment	944	4,817
Portal construction	185	0
Pastefill plant	3,000	2,250
Total	**14,770**	**33,058**

21.1.3 Processing Plant

The capital estimate for the processing plant breaks down as shown in Table 21-4. Ongoing maintenance is covered by operating costs, so there is no sustaining capital forecast.

Table 21-4: Process Plant Capital Estimate

Area	Initial Capital Cost ($ '000s)	Sustaining Capital Cost ($ '000s)
Crushing plant	7,382	-
Grinding, incl. Conc. grinding	19,572	-
Dewatering	5,368	-
Common systems	13,421	-
Leach/Goldwin Area	5,000	-
Total	**50,743**	**-**

21.1.4 Infrastructure

The project's initial infrastructural requirements are estimated to be as shown in Table 21-5. Capital for the establishment of the tailings storage facility is an average of the estimated cost for the three most highly recommended sites. Sustaining capital comprises a phased expansion of this tailings storage facility, in line with the recommendation by the consultant.

Table 21-5: Infrastructure Capital Estimate

Area	Initial Capital Cost ($ '000s)	Sustaining Capital Cost ($ '000s)
Access Road	200	-
Site Preparation	2,000	-
Power Supply	4,053	-
Power Distribution	5,270	-
Power – Emergency Supply	673	-
Water Supply	2,500	-
Administration Complex	1,280	-
Mine Dry Complex	2,552	-
Maintenance Shop	3,157	-
Warehouse and Laydown Area	1,760	-
Fuel Tank Farm and Fuelling Station	850	-
Sewage System	680	-
Fire Water System	560	-
Communications	290	-
Guard House	250	-
Fencing	190	-
Solid Waste Disposal and Recycling	40	-
Tailings Storage Facility	23,528	16,301
Total	**49,833**	**16,301**

21.1.5 Indirect Capital, Owner's Cost and Contingency

The indirect capital cost estimate is shown in Table 21-6.

Table 21-6: Indirect Capital Cost Estimate

Area	Initial Capital Cost ($ '000s)	Sustaining Capital Cost ($ '000s)
Vendor Engineering and Representatives	1,015	-
Construction Equipment	1,522	-
EPCM	7,612	-
Equipment Spares	1,510	-
General Site Costs	1,269	-
Subtotal Indirect Costs	**12,927**	**-**
General (including First Fills)	6,180	-
Insurance	507	-
Commissioning/Training	2,537	-
Owners Site costs	2,768	-
Mine Rehabilitation and Closure	5,393	-
Subtotal Owner's Costs	**17,386**	**-**
Contingency	38,341	-
Total	**68,654**	**-**

Indirect costs include EPCM costs, estimated to be 15% of the direct capital cost estimate for the process plant. Provision is also made for vendor's representatives during construction and commissioning, as well as the temporary hire of construction equipment (e.g., mobile cranes) and general site costs.

Owner's costs include first fills of reagents and consumables in the plant, construction insurance, commissioning, recruitment/training costs, and owner's site costs (including site management and supervision).

Mine rehabilitation costs are estimated to be $7.5 million, incurred following closure of the mine in Year 16. This cost is discounted back to the present at 2% annually to find the amount of the environmental bonding expected to be required for the project. The discounted amount of $5.4 million is then reflected in the cash flow as a cost incurred in two tranches, 30% prior to construction and 70% prior to production start-up.

A total contingency of $38.3 million includes $7.6 million for mining, $15.2 million in the plant, $12.5 million for infrastructure, and $3.0 million for indirect costs. Overall, the contingency equates to approximately 24% of the initial capital cost including indirects.

21.2 Operating Costs

21.2.1 Mine Operating Costs

Mine operating costs are inclusive of supervision, operating and maintenance labour, spares and consumables for the owner's fleet of mobile production, haulage and support equipment, ventilation, dewatering and backfill. Provision is made for rehandling of stockpiled ore, where appropriate.

Development carried out in waste rock for haulage ramps, ventilation raises and cross-cuts will utilize the same fleet of equipment. However, the direct costs of this work are treated as sustaining capital expenditure and so are excluded from operating expenses.

21.2.2 Processing Operating Costs

Processing costs include crushing ROM material, milling, leaching of the pulp, carbon adsorption, stripping, electrowinning and smelting of gold doré, cyanide destruction and disposal of barren tailings. Reagents and process consumables, power, operating and maintenance labour and spare parts, and laboratory costs are considered.

21.2.3 General and Administrative Costs

General and Administrative (G&A) costs include site management, supervisory and technical staff, office running costs, environmental management, and other costs, including overheads.

Estimated cash operating costs over the life of the project are summarized in Table 21-7.

Table 21-7: Summary of LOM Operating Costs

Area	LOM Cost ($ '000s)	Unit Cost ($/t ore treated)
Production Drilling and Blasting	44,600	4.73
Sill	399,183	42.36
Sill (through waste)	13,764	1.46
Diesel Fuel	22,869	2.43
Backfill	29,280	3.11
Manpower	99,948	10.61
Ventilation and Dewatering	2,279	0.24
Exploration	911	0.10
Major Equipment Maintenance	85,618	9.08
Miscellaneous and Sundries	7,395	0.78
Mining G&A	43,653	4.63
Stockpile rehandle	310	0.03
Subtotal Mining	**749,812**	**79.56**
Labour – Metallurgy	5,180	0.53
Laboratory	6,963	0.71

Area	LOM Cost ($ '000s)	Unit Cost ($/t ore treated)
Production	27,417	2.81
Maintenance	14,037	1.44
Power	35,561	3.65
Maintenance	18,346	1.88
Crushing	759	0.08
Grinding	33,760	3.46
Reagents	34,835	3.57
Miscellaneous	13,376	1.37
Subtotal Processing	**190,235**	**20.19**
Labour	21,903	2.25
Equipment Maintenance	6,646	0.68
Mobile Equipment Operation	1,472	0.15
Environmental and Social	7,594	0.78
G&A (other)	43,770	4.49
Subtotal G&A	**81,385**	**8.64**
Total Operating Costs	**1,021,431**	**108.38**

22 ECONOMIC ANALYSIS

Information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

22.1 Basis of Evaluation

Micon has prepared its assessment of the Project on the basis of a discounted cash flow model, from which NPV, IRR, payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to determine the viability of the proposed mine and process plant to exploit the Curraghinalt deposit. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

22.2 Macroeconomic Assumptions

22.2.1 Exchange Rate and Inflation

Unless otherwise stated, all results are expressed in Canadian dollars. Cost estimates and other inputs to the cash flow model for the project have been prepared using constant, second quarter 2012 money terms, i.e., without provision for escalation or inflation. Using trailing 36-month averages, exchange rates of C$1.04/US$ and C$1.62/GBP are applied in the base case.

22.2.2 Weighted Average Cost of Capital

In order to find the NPV of the cash flows forecast for the project, an appropriate discount factor must be applied which represents the weighted average cost of capital (WACC) imposed on the project by the capital markets. The cash flow projections used for the valuation have been prepared on an all-equity basis. This being the case, WACC is equal to the market cost of equity, and can be determined using the Capital Asset Pricing Model (CAPM):

$$E(R_i) = R_f + \beta_i(E(R_m) - R_f)$$

where $E(R_i)$ is the expected return, or the cost of equity. R_f is the risk-free rate (usually taken to be the real rate on long-term government bonds), $E(R_m)$-R_f is the market premium for equity (commonly estimated to be around

5%), and beta (β) is the volatility of the returns for the relevant sector of the market compared to the market as a whole.

Figure 22-1 illustrates the real return on Canadian long bonds computed by the Bank of Canada, taken as a proxy for the risk-free interest rate. Recently, this has dropped from around 2.0% to less than 0.5%. Nevertheless, it is generally accepted that using a long-term average rate will give a more reliable estimate of the cost of equity. Micon has therefore used a value of 2.0% for the base case risk free rate, close to the real rate of return averaged over 10 years.

Figure 22-1: Real Return on Canadian Long Bonds

Source: Bank of Canada

Taking beta across this sector of the equity market to be in the range 0.7 (for instance, some large gold producers) to 1.7 (for diversified base metal groups) with a mid-range of 1.2 (typical for the mining sector), CAPM gives an estimated cost of equity for the Curraghinalt project of between 5% and 11%, as shown in Table 22-1. Micon has taken a figure of 8% (i.e., in the middle of this range) as its base case, and provides the results at alternative rates of discount for comparative purposes.

Table 22-1: Estimated Cost of Equity

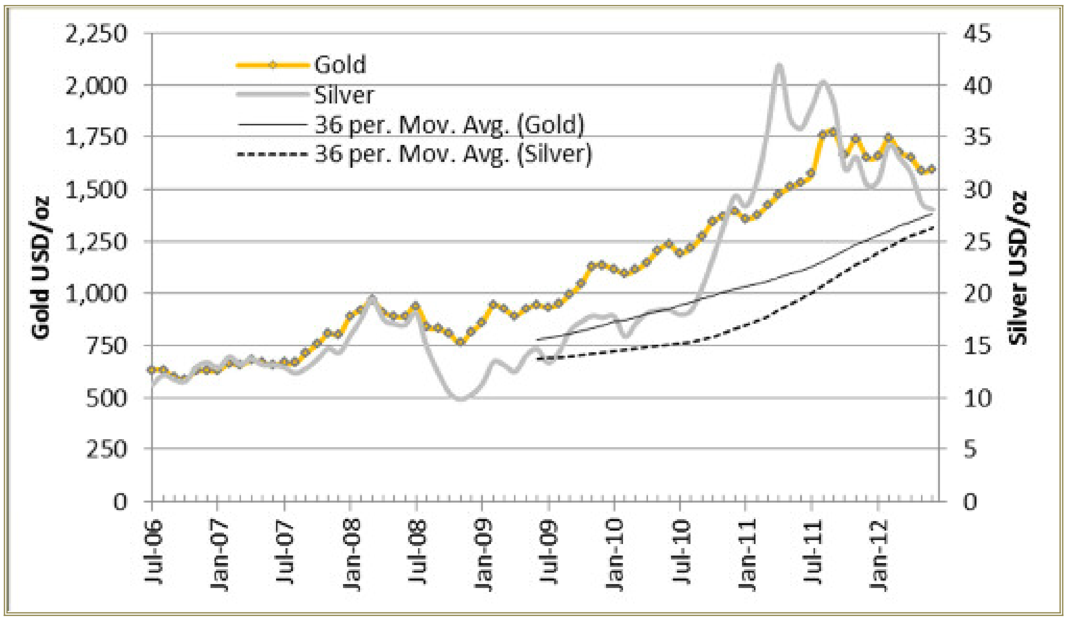
Range	Lower	Middle	Upper
Risk Free Rate (%)	1.5	2.0	2.5
Market Premium for Equity (%)	5.0	5.0	5.0
Beta	0.7	1.2	1.7
Cost of Equity (%)	5.0	8.0	11.0

22.2.3 Expected Metal Prices

Figure 22-2 shows the monthly average gold and silver prices over the past six years, together with the 3-year trailing averages. At the end of June, 2012, the three-year trailing averages for each metal were US$1378/oz. Au and US$26.28/oz. Ag, and these metal prices were selected for the base case. These prices were applied consistently throughout the operating period.

Silver contributes approximately 0.6% of the projected total revenue for the base case, so the impact of changing the silver price forecast is minimal.

Figure 22-2: Monthly Average Gold and Silver Prices since July 2006



Source: Kitco.com

For comparison, Micon also evaluated the sensitivity of the project to using recent (1 month), and 1-, 2-, 3-, 5-, and 10-year price averages. The prices used in each of these cases are shown in Table 22-2. As part of its sensitivity analysis, Micon also tested a range of prices 30% above and below base case values.

Table 22-2: Metal Price Averages

Item	Unit	1-Month Jun-2012	1-Year Average	2-Year Average	3-Year Avg. Base Case	5-Year Average	10-Year Average
Gold	US$/oz.	1,597	1,672	1,521	1,378	1,166	814
Silver	US$/oz.	28.05	33.16	30.98	26.28	21.44	14.66

22.2.4 *Taxation Regime*

United Kingdom corporation tax payable on the project has been forecast using current rates, being 20% on the first GBP 300,000 per year, 22% on the increment up to GBP 1.50 million, and 24% on the balance.

Depreciation allowances of 18% are assumed to be taken annually on all project capital, on a declining balance basis.

22.2.5 *Royalty*

A royalty of 6% of NSR value has been provided for in the cash flow model.

22.2.6 *Selling Expenses*

Refining charges are estimated at US$6.00/oz. Au and US$0.50/oz. Ag, based on similar projects. Doré transport charges of US$5,000 per shipment and cash-in-transit insurance, calculated as a percentage of shipment value, are also provided.

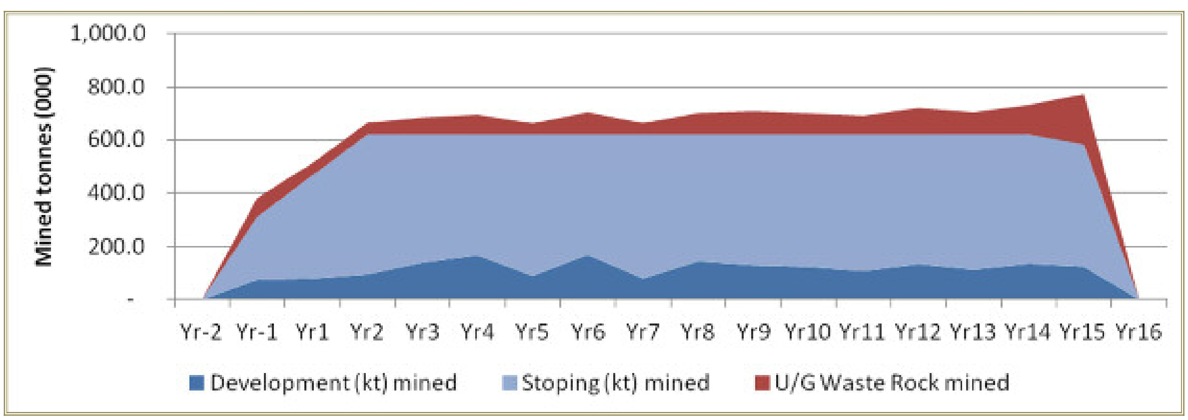
22.3 Technical Assumptions

The technical parameters, production forecasts and estimates described elsewhere in this report are reflected in the base case cash flow model. These inputs to the model are summarized below. The measures used in the study are metric except where, by convention, gold and silver content, production, and sales are stated in troy ounces.

22.3.1 *Mine Production Schedule*

Figure 22-3 shows the annual tonnage of development and stoping material mined, as well as the waste rock tonnage, all of which are held reasonably constant over the LOM.

Figure 22-3: Annual Mining Schedule



In Figure 22-4, the head grades for gold and silver in the mill feed are shown to remain within very narrow ranges over the LOM. Tonnages milled reflect the drawdown of stockpiled material during the mill start-up and again at the end of the LOM.

As a consequence of steady tonnage, grade and recovery from mill feed, annual production of gold and silver remain steady over the LOM (Figure 22-5). This chart also shows the annual total cash cost per ounce of gold sales. Total cash costs, including refining charges and royalties and net of silver credits, average US$532/oz. Au over the LOM. In Year 5 and Year 7, costs fall below this level owing to a reduction in development costs in those periods.

Figure 22-4: Annual Processing Schedule

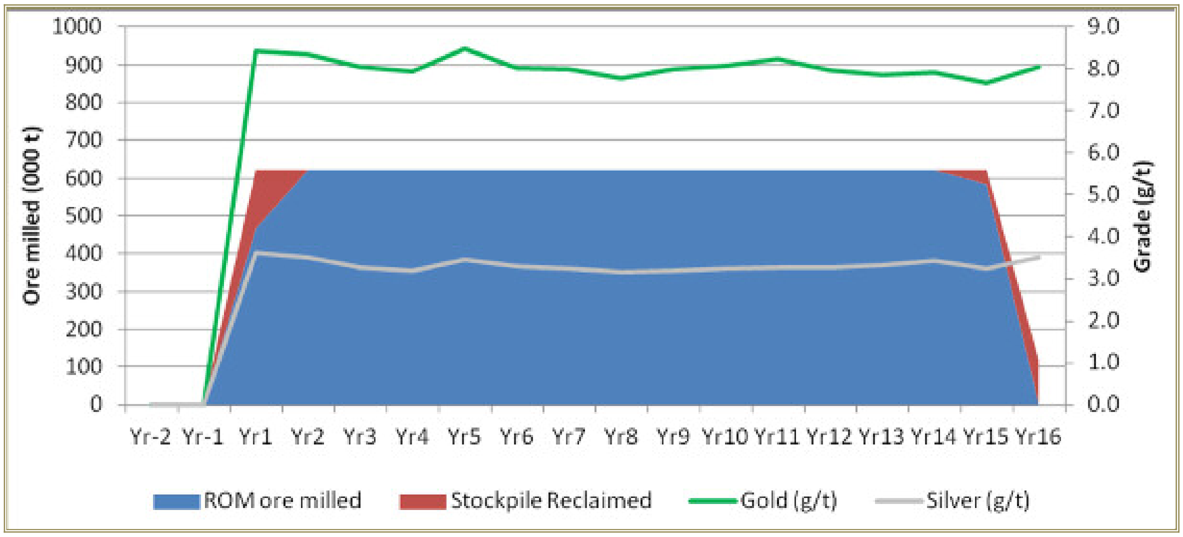
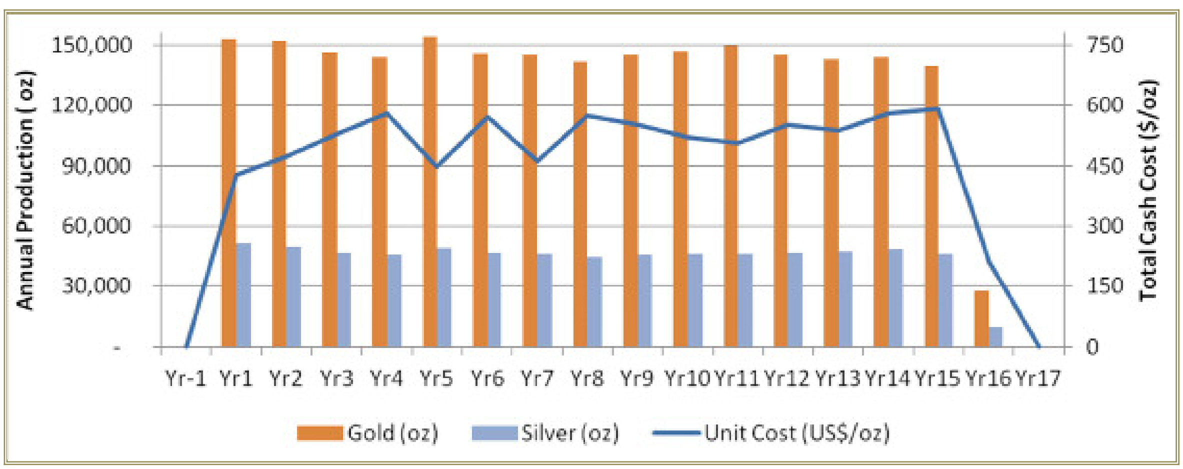


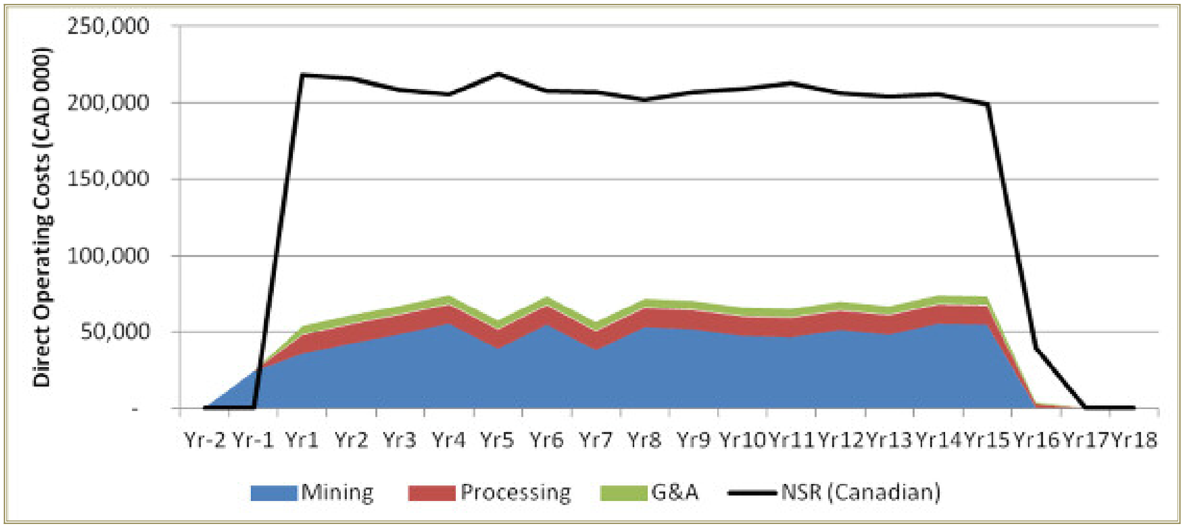
Figure 22-5: Annual Production Schedule



22.3.2 *Operating Costs*

Direct operating costs average $108.38/t milled over the LOM, comprises $79.56/t mining, $20.19/t processing, and $8.64/t G&A costs. Figure 22-6 shows these expenditures over the LOM, compared to the net sales revenue, showing the strong margin maintained over the LOM.

Figure 22-6: Direct Operating Costs



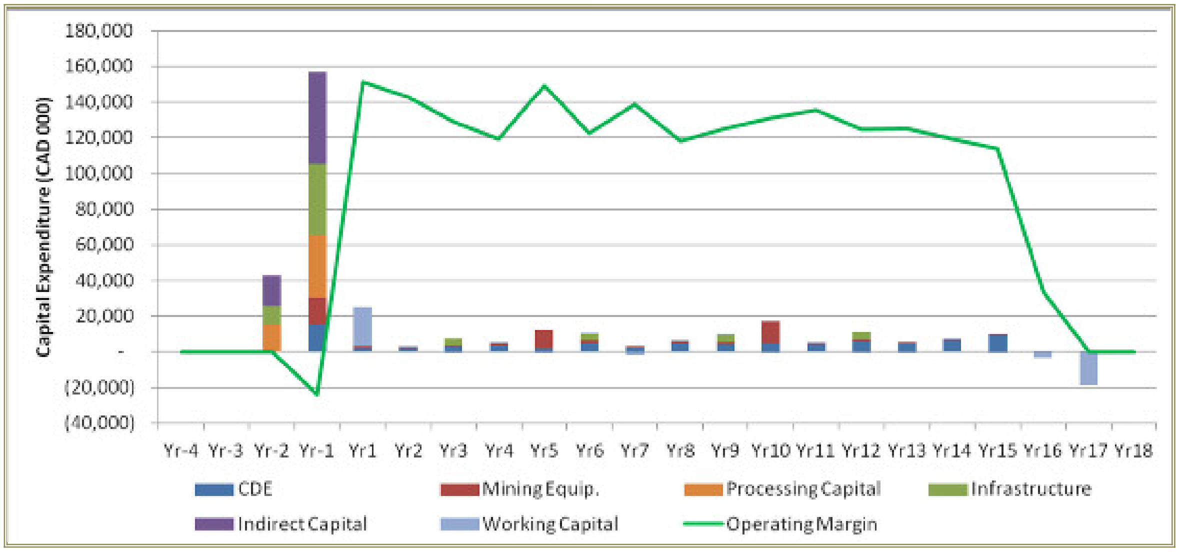
22.3.3 *Capital Costs*

Pre-production capital expenditures are estimated to total $199.75 million, including $30.5 million for mining equipment and pre-production development, $50.7 million processing, $49.8 million infrastructure, $12.9 million indirect costs, $12.0 million in owner's costs, a $5.4 million provision for closure and rehabilitation, and contingencies totalling $38.3 million

Working capital has been estimated to include 15 days product inventory in the milling/leach circuit, and 15 days receivables from despatch of doré. Stores provision is for 60 days of consumables and spares inventory, less 30 days accounts payable. An average of $22.3 million of working capital is required over the LOM.

Figure 22-7 compares annual capital expenditures over the preproduction and LOM with the project's cash operating margin.

Figure 22-7: Capital Expenditures



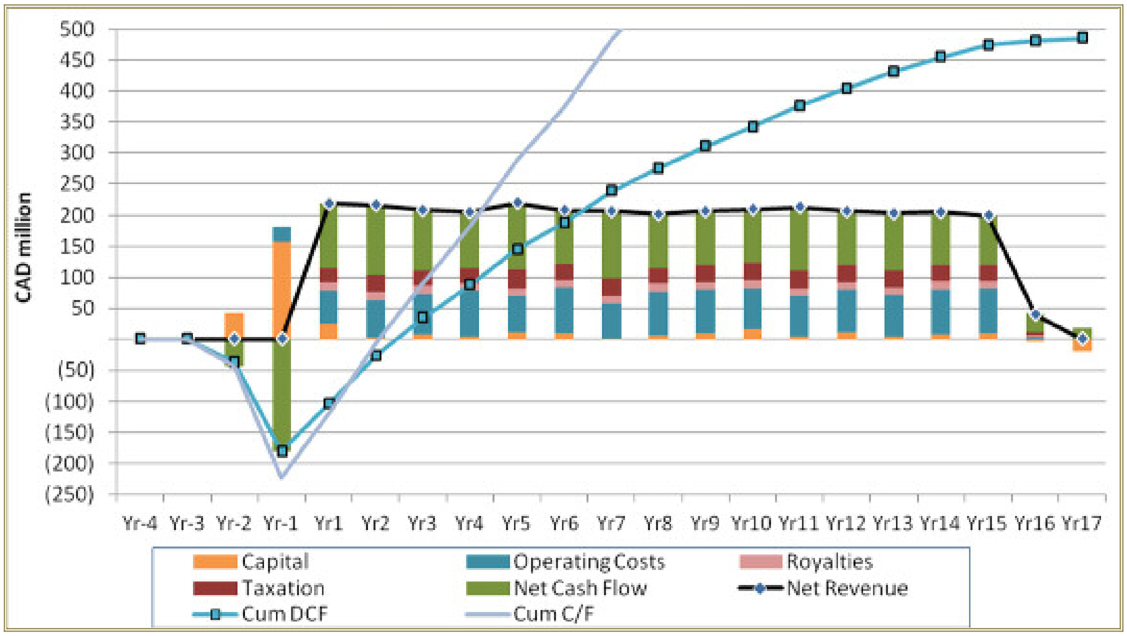
22.3.4 Base Case Cash Flow

The LOM base case project cash flow is presented in Table 22-3 and Figure 22-8.

Table 22-3: LOM Cash Flow Summary

	LOM Total ($ '000s)	$/t Milled	US$/oz
Gold Revenue	**3,186,233**	**338.08**	**1,378.03**
Mining Costs	749,812	79.56	324.29
Processing Costs	190,235	20.19	82.28
G&A Costs	81,385	8.64	35.20
S/T Direct Site Operating Costs	**1,021,431**	**108.38**	**441.76**
Silver Credit	(19,044)	(2.02)	(8.24)
Refining and Transport Charges	37,575	3.99	16.25
S/T Cash Operating Costs	**1,039,962**	**110.35**	**449.77**
Royalty	190,062	20.17	82.20
Total Cash Costs	1,230,024	**130.51**	**531.98**
EBITDA	**1,956,209**	**207.57**	**846.05**
Capital Expenditure	314,029	33.32	135.82
Net Cash Flow (before tax)	**1,642,180**	**174.25**	**710.23**
Taxation	402,895	42.75	174.25
Net Cash Flow (after tax)	**1,239,286**	**131.50**	**535.98**

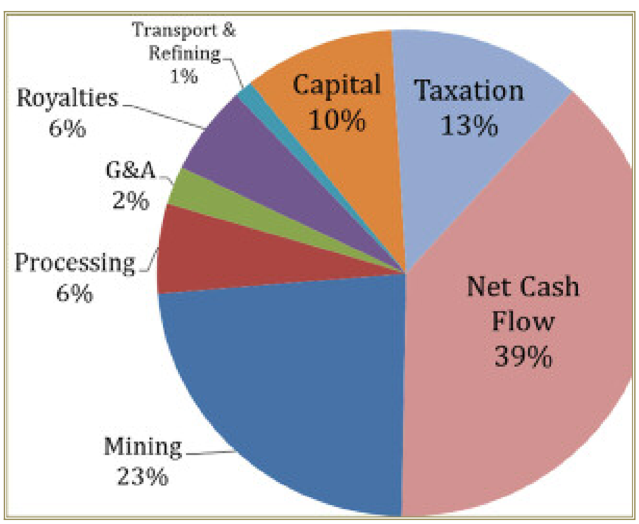
Figure 22-8: LOM Cash Flows



The project demonstrates an undiscounted pay back of 2.1 years, or approximately 2.4 years when discounted at 8.0%, leaving a production tail of just over 12 years.

Over the LOM, gross revenues for gold and silver totalling $3,205.8 million are distributed as shown in Figure 22-9. This diagram illustrates the robust nature of the net cash flows forecast to be generated by the project, both on a before- and after-tax basis.

Figure 22-9: Distribution of Gold and Silver Revenues



Annual cash flows are presented in Table 22-4.

Table 22-4: Base Case LOM Annual Cash Flow

Production Schedule			LOM	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17
Underground Mine Production			TOTAL																			
Development		000 t	1,890	-	74	79	95	138	166	89	167	80	143	128	122	107	133	112	134	123	-	-
Stoping		000 t	7,534	-	236	387	526	483	455	531	453	541	478	492	498	514	488	508	487	459	-	-
Resources mined		000 t	9,425	**-**	**310**	**465**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**582**	**-**	**-**
U/G Waste Rock mined		000 t	1,260	**-**	**70**	**47**	**43**	**64**	**74**	**42**	**83**	**43**	**80**	**86**	**79**	**70**	**98**	**83**	**110**	**189**	**-**	**-**
Processing Plant Throughput		000 t	**9,425**	**-**	**-**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**621**	**117**	**-**
Gold grade		g/t	8.06	-	-	8.43	8.36	8.04	7.93	8.47	8.03	8.00	7.79	8.01	8.09	8.23	7.98	7.87	7.92	7.68	8.06	-
Silver grade		g/t	3.31	-	-	3.62	3.50	3.28	3.20	3.45	3.29	3.24	3.15	3.19	3.24	3.26	3.28	3.33	3.42	3.24	3.52	-
Payable Metal (imperial)																						
Gold		oz	2,223,236	-	-	153,180	151,849	146,124	144,149	153,943	145,927	145,353	141,575	145,451	146,980	149,522	145,007	142,965	143,993	139,603	27,613	-
Silver		oz	715,655	-	-	51,406	49,788	46,556	45,509	49,011	46,767	46,084	44,811	45,403	46,009	46,297	46,576	47,305	48,662	46,041	9,429	-
Exchange Rate		CAD/USD	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04
Gold price		US$/oz	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03	1,378.03
Silver price		US$/oz	26.280	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28	26.28
Cash Flow Forecast			**LOM TOTAL**	**Yr-2**	**Yr-1**	**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**Yr9**	**Yr10**	**Yr11**	**Yr12**	**Yr13**	**Yr14**	**Yr15**	**Yr16**	**Yr17**
	CAD/t ore	**US$/oz**	**CAD 000**																			
Gross Revenue (Gold)	**338.08**	**1,378.03**	**3,186,233**	**-**	**-**	**219,530**	**217,622**	**209,418**	**206,588**	**220,624**	**209,135**	**208,313**	**202,899**	**208,454**	**210,645**	**214,288**	**207,817**	**204,890**	**206,364**	**200,073**	**39,574**	**-**
Total Cash Costs	**130.51**	**531.98**	**1,230,024**	**-**	**23,919**	**67,967**	**74,818**	**80,450**	**87,040**	**71,608**	**86,521**	**69,773**	**84,514**	**83,429**	**79,431**	**78,828**	**83,055**	**79,788**	**86,873**	**85,946**	**6,066**	**-**
Mining	79.56	324.29	749,812	-	23,919	35,762	42,708	48,847	55,612	39,267	54,943	38,235	53,329	51,864	47,725	46,872	51,564	48,522	55,534	54,992	117	-
Processing	20.19	82.28	190,235	-	-	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	12,525	2,362	-
G&A	8.64	35.20	81,385	-	-	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	5,358	1,010	-
Sub-total Direct Costs	*108.38*	*441.76*	*1,021,431*	-	*23,919*	*53,645*	*60,592*	*66,731*	*73,495*	*57,150*	*72,826*	*56,118*	*71,212*	*69,747*	*65,608*	*64,755*	*69,447*	*66,406*	*73,418*	*72,875*	*3,489*	-
By product credit (Silver, net)	(2.02)	(8.24)	(19,044)	-	-	(1,368)	(1,325)	(1,239)	(1,211)	(1,304)	(1,245)	(1,226)	(1,192)	(1,208)	(1,224)	(1,232)	(1,239)	(1,259)	(1,295)	(1,225)	(251)	-
Transport & Refining charges (Gold)	3.99	16.25	37,575	-	-	2,592	2,568	2,467	2,434	2,603	2,464	2,456	2,393	2,458	2,484	2,525	2,451	2,417	2,437	2,360	467	-
Royalties	20.17	82.20	190,062	-	-	13,098	12,983	12,491	12,322	13,160	12,475	12,425	12,102	12,432	12,563	12,780	12,396	12,224	12,313	11,936	2,361	-
Operating Margin	207.57	846.05	**1,956,209**	-	(23,919)	151,563	142,805	128,968	119,548	149,016	122,615	138,540	118,385	125,025	131,214	135,460	124,762	125,102	119,491	114,127	33,508	-
Capital Costs	**33.32**	**135.82**	**314,029**	**42,831**	**156,914**	**2,971**	**2,601**	**7,689**	**4,814**	**12,328**	**10,229**	**3,051**	**5,615**	**9,645**	**17,203**	**4,665**	**11,201**	**5,286**	**6,719**	**10,267**	**-**	**-**
Mine Development	8.56	34.89	80,670	-	15,745	1,896	1,877	3,154	3,940	2,301	4,549	2,406	4,656	4,490	4,657	4,206	5,902	4,826	6,259	9,807	-	-
Mining Equip.	5.07	20.69	47,827	-	14,770	1,075	725	460	874	10,027	1,605	645	959	1,080	12,546	460	1,224	460	460	460	-	-
Processing Capital	5.38	21.95	50,743	15,223	35,520	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure	7.02	28.60	66,134	10,430	39,403	-	-	4,075	-	-	4,075	-	-	4,075	-	-	4,075	-	-	-	-	-
Indirect Capital	7.28	29.69	68,654	17,178	51,476	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Change in Working Cap	-	-	-	-	-	22,153	323	(159)	330	(208)	367	(1,457)	815	330	(158)	224	(126)	(494)	709	(513)	(3,400)	(18,735)
Pre-tax c/flow	**174.25**	**710.23**	**1,642,181**	**(42,831)**	**(180,833)**	**126,439**	**139,880**	**121,438**	**114,403**	**136,896**	**112,018**	**136,946**	**111,955**	**115,050**	**114,170**	**130,570**	**113,688**	**120,311**	**112,063**	**104,374**	**36,907**	**18,735**
Tax payable	42.75	174.25	402,895	-	-	21,865	26,955	24,695	23,267	30,975	24,958	29,338	24,962	26,844	28,309	29,432	27,033	27,299	26,146	24,864	5,954	-
C/flow after tax	**131.50**	**535.98**	**1,239,286**	**(42,831)**	**(180,833)**	**104,574**	**112,925**	**96,742**	**91,136**	**105,921**	**87,060**	**107,609**	**86,993**	**88,206**	**85,861**	**101,139**	**86,655**	**93,013**	**85,917**	**79,509**	**30,953**	**18,735**
Cumulative C/Flow				(42,831)	(223,663)	(119,089)	(6,164)	90,579	181,715	287,636	374,697	482,305	569,298	657,504	743,365	844,504	931,159	1,024,171	1,110,088	1,189,598	1,220,551	1,239,286
Discounted C/Flow (8%)			485,330	(36,721)	(143,551)	76,865	76,855	60,964	53,177	57,226	43,552	49,844	37,310	35,028	31,571	34,434	27,317	27,149	23,221	19,897	7,172	4,020
Cumulative DCF				(36,721)	(180,271)	(103,406)	(26,551)	34,413	87,590	144,816	188,368	238,212	275,521	310,549	342,120	376,554	403,871	431,021	454,241	474,139	481,311	485,330
Max funding reqmt to positive cashflow			(270,652)	(42,831)	(223,663)	**(270,652)**	(148,969)	(38,389)	-	-	-	-	-	-	-	-	-	-	-	-	-	-

22.3.5 Base Case Evaluation

The base case evaluates to an IRR of 51.7% before taxes and 41.9% after tax. At a discount rate of 8.0%, the net present value (NPV_8) of the cash flow is $664.6 million (US$639 million) before tax and $485.3 million (US$467 million) after tax.

Table 22-5 presents the results in Canadian dollars at comparative annual discount rates of 5%, 8%, and 11%. Stated in US dollars, at annual discount rates of 5% and 11%, the after-tax NPV of the project is US$655 million and US$336 million, respectively.

Table 22-5: Base Case Cash Flow Evaluation ($ '000s)

	LOM Total	Discounted at 5%/y	Base Case Discounted at 8%/y	Discounted at 11%/y	IRR (%)
Gross Revenue (Au)	3,186,233	1,903,535	1,442,827	1,117,046	
Mining costs	749,812	448,187	340,171	263,940	
Processing costs	190,235	113,237	85,651	66,180	
G&A Costs	81,385	48,444	36,642	28,312	
Transport and Refining, Net of Ag Credit	18,531	11,074	8,392	6,494	
Royalty	190,062	113,548	86,066	66,633	
Total Cash Costs	**1,230,024**	**734,490**	**556,923**	**431,559**	
EBITDA	1,956,209	1,169,046	885,904	685,487	
Capital Expenditure	314,029	239,989	209,694	185,946	
Working Capital	-	9,887	11,562	11,893	
Net Cash Flow (before tax)	1,642,181	919,169	664,648	487,648	51.7
Taxation	402,895	238,160	179,317	137,895	
Net Cash Flow (after tax)	1,239,286	681,009	485,330	349,753	41.9

This preliminary economic assessment is preliminary in nature; it 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 the preliminary economic assessment will be realized.

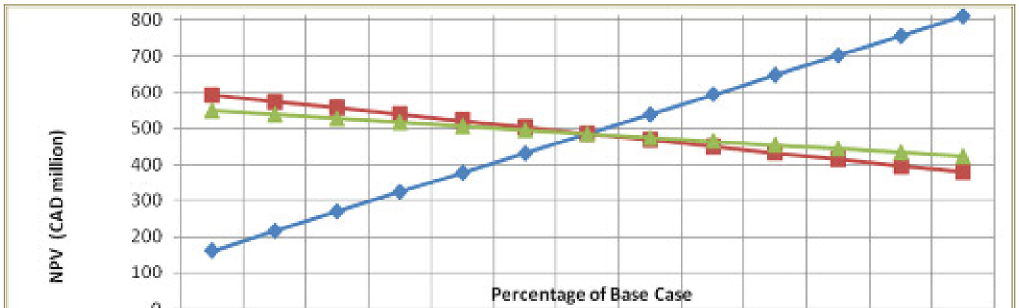
22.4 Sensitivity Study

22.4.1 Capital, Operating Costs, and Revenue Sensitivity

The sensitivity of project returns to changes in all revenue factors (including grades, recoveries, prices, and exchange rate assumptions) together with capital and operating costs was tested over a range of 30% above and below base case values. The results show that the project is most sensitive to revenue factors, with an adverse change of 30% reducing after-tax NPV_8 from $485 million to $161 million. The impact of changing operating costs is lower, with a 30% adverse change reducing NPV_8 to $379 million. The project is least sensitive to capital costs, with a 30% increase in capital reducing NPV_8 to \$422 million. Thus, NPV_8 remains positive within the expected range of accuracy of the estimates.

In Micon's analysis, applying an increase of more than 85% in both capital and operating costs simultaneously would be required to reduce NPV_8 to near zero, further demonstrating the robust nature of the project economics. Figure 22-10 shows the results of changes in each factor separately.

Figure 22-10: Sensitivity Diagram

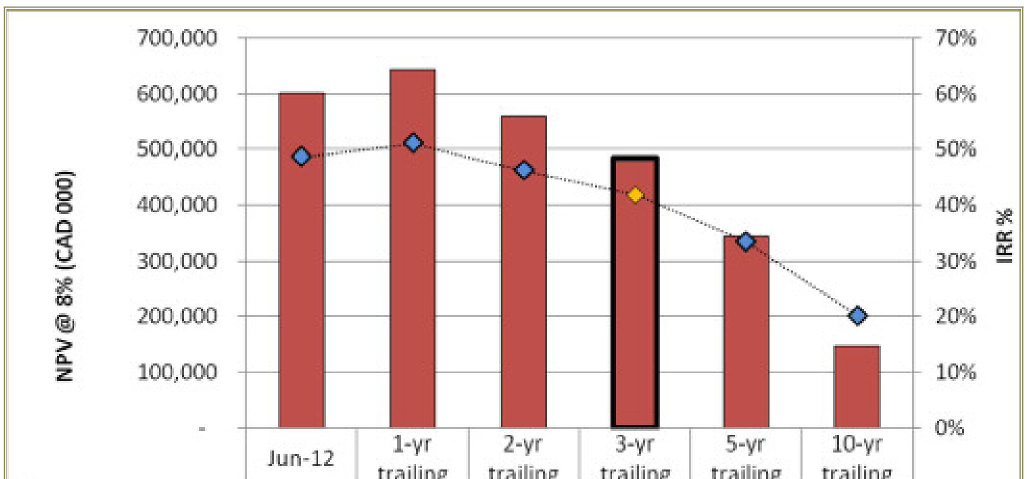


	70	75	80	85	90	95	100	105	110	115	120	125	130
Product Price	161	215	269	323	377	431	485	539	593	648	702	756	810
Opcosts	592	574	556	539	521	503	485	468	450	432	414	397	379
Capex	548	538	527	517	506	496	485	475	464	454	443	433	422

22.4.2 Metal Price Sensitivity

The sensitivity of the project to variation in gold price was tested using 1 month, and, 1-, 2-, 3-, 5-, and 10-year trailing averages applied over the LOM, as shown in Figure 22-11 and in Table 22-6.

Figure 22-11: Sensitivity to Metal Prices



	Jun-12	1-yr trailing	2-yr trailing	3-yr trailing	5-yr trailing	10-yr trailing
After-tax NPV	601,704	644,264	559,678	485,330	344,441	147,160
After-tax IRR	48.6%	51.0%	46.2%	41.9%	33.4%	20.2%

Table 22-6: Sensitivity to Metal Prices

Pricing Period	Unit	1-Month Average	1-Year Average	2-Year Average	3-Year Average	5-Year Average	10-Year Average
Pre-tax IRR	%	60.1	63.1	57.1	51.7	41.1	24.9
After-tax IRR	%	48.6	51.0	46.2	41.9	33.4	20.2
Pre-tax NPV_8	C$ '000s	818,095	874,215	762,681	664,648	478,875	218,744
After-tax NPV_8		601,704	644,264	559,678	485,330	344,441	147,160
Pre-tax NPV_8	*US$ '000s*	*786,630*	*840,591*	*733,348*	*639,084*	*460,456*	*210,330*
After-tax NPV_8		*578,561*	*619,485*	*538,152*	*466,664*	*331,194*	*141,500*

At US$1,200/oz Au, the project has an IRR of 39.1% pre-tax and 31.8% after tax.

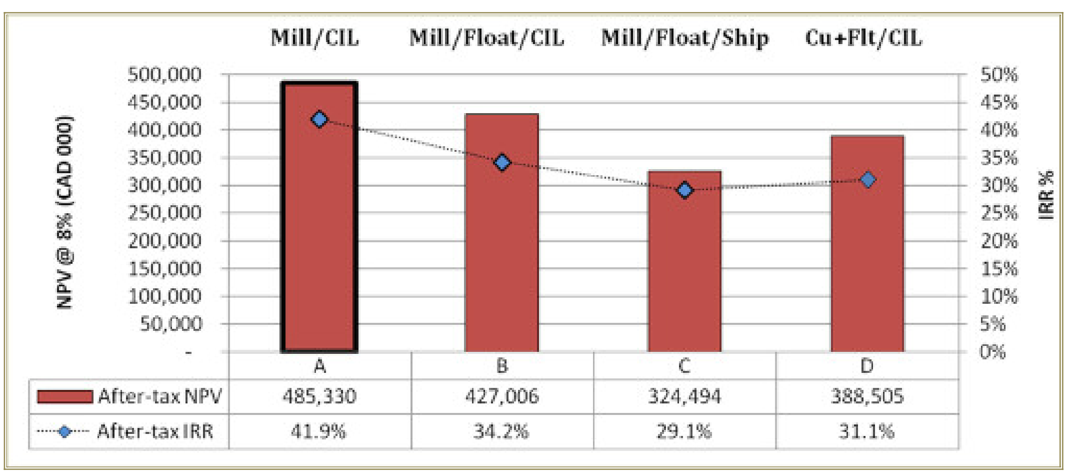
22.4.3 Sensitivity to Process Flowsheet

Micon conducted a trade-off study to determine the optimum process flowsheet for the Curraghinalt deposit. The four flowsheets considered were:

- Crushing/milling followed by CIL and gold recovery
- Crushing/milling/gravity concentration followed by flotation of a concentrate for CIL/gold recovery
- Crushing/milling/gravity concentration followed by flotation of a concentrate for sale
- Crushing/milling followed by flotation of (i) a copper concentrate for sale and (ii) a bulk sulphide concentrate for CIL/gold recovery.

A comparison of the cash flows for each of these options suggests that Option A, Milling/CIL provides the best overall economic return, and so this was selected as the base case for this study. Figure 22-12 shows the NPV and IRR for each of the four options.

Figure 22-12: Sensitivity to Process Flowsheet



22.5 Conclusion

Micon concludes that this study demonstrates the viability of the project as proposed, and that further development is warranted.

23 ADJACENT PROPERTIES

To the southwest of the Curraghinalt deposit is the Cavanacaw deposit, which is located on the 189 km^2 licence OM 1/09 held by Galantas Gold Corporation (Galantas). The processing plant was reported by Galantas as commissioned in January 2007. An NI 43-101-compliant resource estimate was prepared by Galantas (Phelps and Mawson, 2013) dated June 12, 2013, and filed on SEDAR on July 23, 2013. Galantas reported contained gold as follows:

- Measured – 3,300 oz Au
- Indicated – 92,000 oz Au
- Inferred – 231,000 oz Au.

The property is producing at a small scale, with production in 2013 reported as 1,349 oz. Au, 2,622 oz. Ag, and 36.3 t Pb.

TMAC has not verified this information and it is not necessarily indicative of the mineralization on the Dalradian Resources' property.

24 OTHER RELEVANT DATA AND INFORMATION

TMAC is not aware of any additional information or information necessary in order to make this report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 2014 Mineral Resource

The Curraghinalt deposit represents an orogenic gold deposit with parallel veins and sub-veins or veinlets which have been traced by trenching and drilling over a strike length of approximately two kilometers. Twenty-one D veins were interpeted and comprise the primary zones of mineralization in this updated resource model.

The mineral resource for Curraghinalt Deposit is tabulated in Table 25-1 at a cut-off grade of 5.00 g/t Au. The effective date of the resource is January 20, 2014. This resource is exclusive of the underground development which is estimated at 1,700 tonnes at 15.73 g/t Au or 881 oz.

Table 25-1: Curraghinalt Deposit Resource Statement (Cut-off Grade 5.00 g/t Au)

Resource Class	Tonnage (Kt)	Au g/t	Ag g/t	Cu %	Contained Au (Koz)
Measured	23.3	20.15	7.54	0.02	15.1
Indicated	2,976.2	10.34	3.85	0.08	989.0
Measured + Indicated	2,999.5	10.41	3.88	0.08	1,004.1
Inferred	8,005.7	9.67	3.94	0.07	2,487.7

No minimum width constraint was applied before reporting this resource. The interpretation method of including a minimum of two metres of downhole length to define a vein resulted in an average horizontal thickness of 2.57 m.

Mineral resources are reported at a cut-off grade of 5.00 g/t Au. TMAC cautions that mineral resources that are not mineral reserves do not have demonstrated economic viability.

The D veins remain open at depth and along strike. Also additional material may result from the interpretation and interpolation of the C veins which are not included in this resource.

25.2 2012 PEA Conclusions

Information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

Based on its economic evaluation of the base case and sensitivity studies, Micon concludes that the PEA demonstrates the viability of the Curraghinalt project as proposed, and that further development is warranted.

The mineral resource estimate on which the PEA is based is open along strike and at depth, and there is good potential for further exploration to expand the resource and to increase confidence in the existing resource.

The key findings of the PEA are:

- pre-tax IRR of 51.7% (after-tax 41.9%) based on a 36-month trailing average gold price of US$1,378/oz. (after-tax IRR of 31.8% based on US$1,200/oz. Au price)
- project payback of 2 years from first gold production
- after-tax NPV of US$467 million based on a 8% discount rate and a realized gold price of US$1,378/oz. (US$655 million using a 5% discount rate)
- initial capital expenditures of approximately US$192 million prior to production start-up (including contingencies of $36.9 million), with sustaining capital of US$110 million for a total LOM capital spend of US$302 million
- A 15-year mine life with average LOM total cash costs of US$532/oz., or US$125/t milled, including royalties, refining costs and by-product credits of US$8.24/oz. Au
- LOM gold production of 2.223 Moz
- average mined grade of 8.1 g/t Au. Processing at a rate of 1,700 t/d and producing approximately 145,000 oz. Au/y using a conventional flowsheet of crushing, grinding, cyanidation and conventional tailings disposal
- underground mining using mechanized longhole methods with ramp access and truck haulage
- the mine plan considers 89% of the November 2011 Micon resource estimate, of which 83% is Inferred.

The PEA is preliminary in nature. It 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. There is no certainty that the results of the PEA will be realized.

25.2.1 Mining

Mining of the Curraghinalt deposit will be undertaken using a Longitudinal Sublevel Retreat Underground Mining method with both paste- and waste rock backfill to maximize the extraction of the high value resources at the Curraghinalt Project, reduce the surface footprint required for tailings disposal, and allow savings in the volume and transportation cost of waste rock from underground development. Stopes will have a minimum width of 1.80 m and development of sills along the veins will be restricted to 3.0 m width to minimize dilution. The sills will be developed in both directions along the strike length of the mineralized zone, providing two production faces for each vein on each level. Levels will be mined at 20 m intervals. Haulage ramps will have dimensions 4.5 m x 4.5 m to accommodate 30-ton trucks.

25.2.2 Mineral Processing

The PEA considers four possible process flowsheets for the extraction of gold from the Curraghinalt resource. Of these, Option A, which consists of crushing, grinding, whole-ore cyanidation and conventional tailings disposal, was considered the most effective and has been used as the base case for project evaluation.

Nevertheless, Micon considers additional testwork, including optimizing grind, reagent strengths and retention times are required, and pilot studies are required before flowsheet development and design specifications can be finalized.

Total gold recoveries, based on existing metallurgical testwork, are expected to be approximately 92%.

25.2.3 Infrastructure and Capital Costs

Infrastructure required for the project has been identified and is provided for in the evaluation. A site-specific layout has not been developed, though, pending discussion with affected landowners over the siting of these works.

Initial capital expenditures total approximately US$192 million, inclusive of a US$37 million contingency as set out in Table 25-2. LOM sustaining capital totals approximately US$110 million. Sustaining capital consists of capitalized waste development after the initial production start-up, major equipment replacement and tailings expansions. Mining sub-level development cost is included in the operating cost.

Table 25-2: Curraghinalt Initial Capital Expenditures

Year (US$ '000s)	Start-Up -2	Start-Up -1	Start-Up Capital
Preproduction	-	15,139	15,139
Mining Equipment	-	14,202	14,202
Processing Capital	14,638	34,154	48,792
Infrastructure	10,029	37,887	47,916
Indirect Capital	16,517	49,497	66,014
Total	**41,184**	**150,879**	**192,063**

The provision of electricity to the mine will require a new overhead power line, probably from Strabane, approximately 27 km from the mine. A two-year permitting and construction period is expected following a decision to develop the mine.

The PEA assumes that collection of rainfall in the tailings storage facility will provide process make-up water, but further work is required to properly assess water supply sources available to the project.

Future development of the project will need to take into account its location within an Area of Outstanding Natural Beauty, and proximity to Areas of Special Scientific Interest and Special Areas of Conservation.

A small proportion of the waste rock samples tested were acid generating, so it will be necessary for such material to be segregated and stowed as underground backfill. On closure of the mine, the mine entrance will be plugged and flooding of the workings will then minimize further oxidation.

26 RECOMMENDATIONS

Some information in this section is taken from the PEA published by Micon in 2012 (Hennessey et al., 2012b). The PEA was based on the November 2011 Mineral Resource (Hennessey et al., 2012a) and has not been updated to incorporate the 2014 Mineral Resource Estimate presented in this report. An update to the PEA, incorporating the 2014 Mineral Resource Estimate as well as the results of an options analysis for mineral processing, tailings management, mine design and infrastructure location, is being planned.

26.1 Proposed Exploration and Development Program

Dalradian Resources has proposed a multifaceted exploration and development program for advancing the Curraghinalt deposit. The proposed work is intended to further explore the extensions of the mineralization, infill gaps in the current resource, continue development work to increase confidence in the resource, test proposed mining methods, and further refine the process flowsheet. Additional exploration work is planned to advance other targets in the Dalradian Supergroup, but is not considered in the proposed budget, presented here.

26.1.1 Curraghinalt Resource Drilling and Update

The Curraghinalt deposit is open along and across strike, and to depth. There is scope for identifying additional veins both to the north and south of the known mineralization, as well as extending the known gold-bearing veins along strike to the east and west and to depth. A program of continued step-out and infill drilling is currently being planned to continue to grow the resource, and to convert existing resources from the Inferred to Indicated category.

Additional work is proposed to better understand the C Vein mineralization. This work could be comprised of additional drilling and a structural study.

The collection of specific gravity data should be an ongoing component of all new sampling programs.

Work should be initiated to refine the D vein interpretation by reconciling the interpretation in plan and section.

26.1.2 Curraghinalt Development

With the completion of this PEA, Dalradian has applied for and received planning permission to extend the underground workings. In the timeframe covered by the current budget 1,000 m of underground development is proposed. The purpose of this work will be to confirm the grade and tonnage and continuity of the veins in an underground setting, further evaluate underground ground conditions, test proposed mining methods, and access additional material for further metallurgical testing. Dalradian Resources also plans to continue with the metallurgical testing of the Options A, B, and C flowsheets and evaluate the tailings properties of those two flowsheets.

Work will continue in the gathering of environmental baseline data to support an ES.

Geotechnical and Mine Design

In its January 2012 report on the property, Snowden made recommendations regarding the collection and analysis of geotechnical data. Micon concurs with those recommendations. Also, Micon recommends that during geotechnical data collection the rock mass should be accurately characterized and described. This allows the rock mass attributes, parameters, and ratings to be assigned at a later stage enabling the classification to be made in any of the proposed rock mass classification systems (e.g., RMR, Q-system or MRMR).

Regarding development of the mine design, Micon recommends the following:

- Further optimization of the mining method be performed to evaluate potential additional economic benefits to the project by considering and combining a higher selectivity, non-mechanized mining method with the proposed Longitudinal Sublevel Retreat.
- Further studies be performed on the backfill system for the Project and that a systematic laboratory test program be performed on the mine tailings to determine the optimum cement addition and cement type to be used as the binding agent for the paste backfill.
- Non-PAG waste rock produced during underground exploration and mine development activities may be utilized as bulk fill materials to meet the construction needs, where appropriate. The structural fill material will be imported from local suppliers, as required.
- The mine design proposed in the 2012 PEA should be updated with the 2014 mineral resources.

Metallurgical

Future metallurgical testwork should be considered upon review of the final testwork report currently being prepared by G&T.

Micon considers the elevated level of penalty elements in copper concentrate produced during the ALS Metallurgy testwork, as largely ruling out option D for mineral processing at Curraghinalt. Production of a rougher concentrate for sale to a smelter (Option B), or treatment onsite (Option C), may be a viable alternative to whole ore cyanidation, but additional work is required to assess the full costs and benefits of these options.

Based on a review of reports leading up to the current testwork underway, it is recommended that future testwork should include the following:

- locked-cycle flotation testing of the bulk sulphide concentrate circuit
- cyanidation tests of the bulk sulphide concentrate (rougher and cleaner products), both with and without regrinding, pre-aeration, and lead nitrite addition
- additional BBWi testing
- SAG amenability testwork

- further evaluation of the effect of grind size on leach extractions for the whole ore cyanidation option
- evaluation of alternative flotation reagents, especially collector 3418A
- carbon-in-leach (CIL) testwork
- testwork with higher cyanide concentrations
- confirmatory testwork on a variety of representative feed samples once a preferred flowsheet is established.

In further development of the project, Micon recommends that Dalradian Resources obtain offers of specific terms for the purchase of doré.

Tailings and Water

Golder recommended three sites for further investigation as potential tailings storage facilities. Although these sites were selected based on conventional (slurry) tailings disposal, Micon suggests that paste or dry stack tailings be considered to minimize the footprint and increase the available options for siting the tailings and employ progressive reclamation to minimize visual impacts.

Potential sources of process water should be investigated including a study of groundwater in wells close to the proposed underground mine, as recommended by SLR, as well as further study of the proposed collection of rainfall at the tailings storage facility.

Other Items

In this PEA, the estimated unit cost for electrical power is based on best available information. As the project develops, Dalradian Resources should obtain project-specific pricing.

Negotiations with landowners should take place as the project moves forward so that a site-specific layout of the plant and infrastructure can be developed during the next stages of project engineering.

26.1.3 Budget

Dalradian has proposed the following breakdown in expenditures:

Exploration $5,000,000

Underground Development Program $12,000,000

*Total ....$17,000,000 (*Excludes corporate G&A)

TMAC has reviewed Dalradian Resources' proposed budget for the Curraghinalt deposit to complete the programs described above and finds it to be reasonable and warranted in light of the data presented and observations made in this report. TMAC recommends that Dalradian Resources proceed with the program.

The budget presented is a base case scenario and may be modified. This budget does not include additional exploration programs planned for Northern Ireland and elsewhere, nor does it include general corporate costs.

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28 CERTIFICATE OF AUTHORS

28.1 Tim Maunula, P.Geo.

I, Tim Maunula, P.Geo., of Chatham, Ontario, a QP of this technical report titled "Mineral Resource Estimate Update, National Instrument (NI) 43-101 Technical Report for the Curraghinalt Gold Deposit, Northern Ireland" dated May 30, 2014 (the "Technical Report"), do hereby certify the following statements:

I am Principal Geologist of T. Maunula & Associates Consulting Inc., 15 Valencia Drive, Chatham, ON, N7L 0A9, Canada.

I graduated with a H.B.Sc. degree in Geology from Lakehead University in 1979. In addition, I have obtained a Citation in Geostatistics from the University of Alberta in 2004.

I am a member of the Association of Professional Geoscientists of Ontario (Registration Number 1115). I am a member in good standing of The Canadian Institute of Mining, Metallurgy and Petroleum.

I have worked as a Geologist for a total of 35 years since my graduation from university.

I have read the definition of QP set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.

I am responsible for the preparation of Sections 2 to 12, 14.6 to 14.13, 20, 23, 24, 27 and portions of Sections 1, 25 and 26 of this technical report titled "Mineral Resource Estimate Update, NI 43-101 Technical Report for the Curraghinalt Gold Deposit, Northern Ireland" with an effective date of January 20, 2014 (the "Technical Report") relating to the Dalradian property.

I have visited the property on two occasions, August 12 to 16, 2013 and December 14 to 15, 2013.

As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 30th Day of May 2014 in Chatham, ON.

"Original Document Signed and Sealed"

Tim Maunula, P.Geo.

28.2 B. Terrence Hennessey, P.Geo.

As co-author of this report on certain mineral properties of Dalradian Resources Inc. in Northern Ireland, I, B. Terrence Hennessey, P.Geo., do hereby certify that:

1. I am employed by, and carried out this assignment for:

Micon International Limited
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2. I hold the following academic qualifications:

B.Sc. (Geology)	McMaster University	1978

3. I am a registered Professional Geoscientist with the Association of Professional Geoscientists of Ontario (membership # 0038); as well, I am a member in good standing of several other technical associations and societies, including:

The Australasian Institute of Mining and Metallurgy (Member)
The Canadian Institute of Mining, Metallurgy and Petroleum (Member)
Society of Economic Geologists (Fellow).

4. I have worked as a geologist in the minerals industry for over 35 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 1 year with a government geological survey, 7 years as an exploration geologist looking for iron ore, gold, base metal and tin deposits, more than 10 years as a mine geologist in both open pit and underground mines and over 17 years as a consulting geologist working in precious, ferrous and base metals as well as industrial minerals.

6. I visited Northern Ireland and the Tyrone property during the period November 6 and 7, 2009 and October 3 to 5, 2010 to review exploration results and examine drill core. The property had not previously been visited by me.

7. I am responsible for the preparation of Sections 14.1 to 14.5, and the portions of Section 1 summarized therefrom, of the Technical Report titled "Curraghinalt Gold Deposit, Northern Ireland, Mineral Resource Estimate Update, NI 43-101 Technical Report" and dated May 30, 2014.

8. I am independent of the parties involved in the transaction for which this report is required, as defined in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date of the PEA: July 25, 2012

Dated this 30th day of May, 2014

"B. Terrence Hennessey" {signed and sealed}

B. Terrence Hennessey, P.Geo.

Vice President

28.3 Barnard Foo, P.Eng.

As co-author of this report on certain mineral properties of Dalradian Resources Inc. in Northern Ireland, I, Barnard Foo, do hereby certify that:

1. I am employed by, and carried out this assignment for:

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2. I hold the following academic qualifications:

- Laurentian University, B.Eng., Mining Engineering 1998
- University of British Columbia, M. Eng., Rock Mechanics 2007
- University of Northern British Columbia, Executive MBA 2010

3. I am a registered Professional Engineers of Ontario (Membership # 100052925)

4. I have worked as a mining engineer in the minerals industry for 16 years.

5. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 4 years as a mining engineer in cassiterite, base and precious metal deposits, 5 years in underground and open pit geotechnical engineering and 7 years in mine design and mining project evaluations for the mineral industry.

6. I visited the Curraghinalt property on April 18-19, 2012.

7. I am responsible for the preparation of Sections 15, 16, 21.1.1, 21.1.2, 21.2.1 and the portions of Sections 1, 25 and 26, summarized therefrom, of the Technical Report titled “Curraghinalt Gold Deposit, Northern Ireland, Mineral Resource Estimate Update, NI 43-101 Technical Report” and dated May 30, 2014.

8. I am independent of the parties involved in the transaction for which this report is required, as defined in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date of the PEA: July 25, 2012

Dated this 30th day of May, 2014

"Barnard Foo" {signed and sealed}

Barnard Foo, M.Eng., P.Eng., MBA.

Senior Mining Engineer

28.4 Bogdan Damjanović, P.Eng.

As co-author of this report entitled “Mineral Resource Estimate Update, National Instrument (NI) 43-101 Technical Report for the Curraghinalt Gold Deposit, Northern Ireland”, with an effective date of January 20, 2014 (the “Technical Report”), I, Bogdan Damjanović, do hereby certify that:

1. I am employed as a senior metallurgist by, and carried out this assignment for:

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2. I hold the following academic qualifications:

B.A.Sc. Engineering, University of Toronto, 1992.

3. I am a Professional Engineer registered with the Professional Engineers Ontario. (registration number 90420456). Also, I am a professional member in good standing of: The Canadian Institute of Mining, Metallurgy and Petroleum (Member);

4. I have worked in the minerals industry for 20 years; my work experience includes 8 years as a metallurgist on gold, copper/nickel and lead/zinc/gold deposits; and the remainder as an independent consultant when I have worked on a variety of precious and base metal deposits;

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101;

6. I did not visit the Curraghinalt property;

7. I am responsible for the preparation of Section 13, 17, 21.1.3, 21.2.2 and the portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report;

8. I am independent of Dalradian Resources Inc., as defined in Section 1.5 of NI 43-101;

9. I have had previous involvement with the property; the nature of my prior involvement was as co-author of a report entitled “A Preliminary Economic Assessment of the Curraghinalt Gold Deposit, Tyrone Project, Northern Ireland”, with an effective date of 25 July, 2012

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;

11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date of PEA 25 July, 2014

Dated this 30th day of May, 2014

"Bogdan Damjanović" {signed and sealed}

Bogdan Damjanović, P.Eng.

Senior Metallurgist

28.5 André Villeneuve, P.Eng.

As co-author of this report entitled "Curraghinalt Gold Deposit, Northern Ireland, Mineral Resource Estimate Update, NI 43-101 Technical Report", dated May, 30, 2014 (the "Technical Report"), I, Andre Villeneuve, do hereby certify that:

1. I am an Associate Mining Engineer with the firm of:

   Micon International Ltd.
   Suite 205, 700 West Pender Street,
   Vancouver, British Columbia;

2. I hold the following academic qualifications:

   B.Sc. in Mining Engineering , University of Montreal 1983;

3. I am a Professional Engineer registered with the APEG of British Columbia (registration number 19287);

4. I have worked in the minerals industry for 29 years; I have extensive senior level management and executive experience with all the various phases of a mineral development project from advanced exploration stage through mine development and construction to mine start-up and production for gold, silver, base metals and coal. I have been involved since 1989 on a consulting basis on numerous mining projects and operations in the Americas, Africa and Australasia.;

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101;

6. I visited the Curraghinalt property on April 18-19, 2012;

7. I am responsible for the preparation of Sections 18.1 to 18.9, 21.1.4, as well as co-author of summaries therefrom in Section 1, 25 and 26 of the Technical Report.

8. I am independent of Dalradian Resources Inc., as defined in Section 1.5 of NI 43-101;

9. I have had no previous involvement with the property;

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;

11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 30th day of May, 2014

Andre Villeneuve, P.Eng. {signed and sealed}

Andre Villeneuve, P.Eng.

Associate Mining Engineer

28.6 Christopher Jacobs, CEng MIMMM

As co-author of this report entitled "Curraghinalt Gold Deposit, Northern Ireland, Mineral Resource Estimate Update, NI 43-101 Technical Report" and dated May 30, 2014 (the "Technical Report"), I, Christopher Jacobs, do hereby certify that:

1. I am employed as a mineral economist by, and carried out this assignment for:

   Micon International Limited,
   Suite 900 - 390 Bay Street,
   Toronto, ON,
   M5H 2Y2
   tel. (416) 362-5135
   email: cjacobs@micon-international.com.

2. I hold the following academic qualifications:

   B.Sc. (Hons) Geochemistry, University of Reading, 1980;

   M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.

3. I am a Chartered Engineer registered with the Engineering Council of the U.K. (registration number 369178). Also, I am a professional member in good standing of: The Institute of Materials, Minerals and Mining; and The Canadian Institute of Mining, Metallurgy and Petroleum (Member);

4. I have worked in the minerals industry for 30 years; my work experience includes 10 years as an exploration and mining geologist on gold, platinum, copper/nickel and chromite deposits; 10 years as a technical/operations manager in both open pit and underground mines; 3 years as strategic (mine) planning manager and the remainder as an independent consultant when I have worked on a variety of precious and base metal deposits;

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101;

6. I visited the Curraghinalt property on April 18-19, 2012;

7. I am responsible for the preparation of Sections 19, 21.1.5, 21.2.3 and 22, and the portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.

8. I am independent of Dalradian Resources Inc., as defined in Section 1.5 of NI 43-101;

9. I have had no previous involvement with the property;

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;

11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date of the PEA: July 25, 2012

Dated this 30th May, 2014

"Christopher Jacobs" {signed and sealed}

Christopher Jacobs, CEng MIMMM