RESOURCE ESTIMATE, PRELIMINARY ECONOMIC ASSESSMENT & DETAILED FEASIBILITY STUDY

ON THE

OMAGH GOLD PROJECT

COUNTY TYRONE,

NORTHERN IRELAND

26th July 2014

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NI.43-101 Report, 2014 Revision

Contents

1. SUMMARY

Galantas Gold Corporation (with its wholly owned subsidiary Omagh Minerals Ltd - OML) has prepared an updated mineral resource estimate for the Cavanacaw gold deposit, in accordance with the reporting standards and definitions of the Pan-European Reporting Code (PERC). This report (Galantas 2014) provides a summary of the project geology and exploration programme, a revised mineral resource estimate and the results of economic studies.

INDEPENDENCE

This report (Galantas 2014) has not been prepared independently of Galantas Gold Corporation. It has been prepared under the overall supervision of R. Phelps C.Eng. MIOM3, (President & CEO of Galantas Gold Corporation) a Qualified Person for the purposes of Canadian National Instrument 43-101. Parts of the report have been drawn from prior independent reports where that information has been assessed as reasonable in the context in which it is used.

REGULATORY CONTEXT

The economic study, summarised in this report, includes use of Measured and Indicated resources with a minor portion of Inferred resources estimated for two veins (Joshua and Kearney veins). The Inferred resources are contiguous with or lie within scheduled mining areas. The use of Inferred resources, in a restricted qualifying manner, is permitted by the PERC code in regard to economic studies but is excluded within Canadian National Instrument 43-101, except within a "Preliminary Economic Assessment (PEA)". In compliance with the disclosure requirements of Canadian National Instrument (NI.) 43-101, it has been determined that the economic study including associated inferred resources is deemed a Preliminary Economic Assessment. PERC is an approved code is respect of NI. 43-101. As part of PERC requirements, a comparative study is required which does not include inferred resources. In the circumstances of this report, this second economic study meets the requirements of a feasibility study.

1.1 SUMMARY DESCRIPTION

The Cavanacaw Mine is located approximately 5 kilometres from of Omagh, County Tyrone, Northern Ireland. It is situated on freehold land owned by Omagh Minerals Ltd (OML). OML holds a Mining License for gold and silver from the Crown Estate Commissioners (CEC) and Exploration Licenses from CEC and the Department of Enterprise, Trade and Investment Northern Ireland (DETI), the latter over approximately 439 sq. kms. Additionally OML holds Exploration Licenses in the Republic Of Ireland, contiguous with its licenses in Northern Ireland.

The mine has good access by public road and is approximately a 1.5 hour drive west of Belfast. The mine is located on rough agricultural land.

ACA Howe International Ltd (ACA Howe), in an independent Technical Report On The Omagh Gold Project, Counties Tyrone and Fermanagh, Northern Ireland - Parker and Pearson August 10th 2012 (Howe 2012) gave the following historical context, geological description and description of data gathering techniques employed by Galantas :-

The occurrence of gold in the Sperrin Mountains in Northern Ireland has been known for centuries but no mining operations have taken place prior to that at Cavanacaw. Following the discovery of vein gold *at Curraghinalt by Ennex International in the mid 1980's, Riofinex North Ltd (Riofinex) commenced exploration of an area of similar rocks located south-west of Omagh which led to the discovery of the gold bearing Kearney vein structure and the surrounding swarm of veins at Cavanacaw. The deposit was evaluated by stripping of overburden and carrying out intensive channel sampling of the exposed vein and by diamond drilling.*

In 1990, the Riofinex project was transferred to Omagh Minerals (OML) who commissioned metallurgical, mining and environmental studies.

In 1997, European Gold Resources Inc (EGR) acquired Omagh Minerals (OML) who re-excavated the open cut on the Kearney structure and carried out selective mining trials at the southern end of the Kearney structure to extract high grade ore and produce gold bullion and jewellery under the Galantas brand name.

In 2003, EGR commissioned ACA Howe to prepare a technical report in compliance with Canadian NI 43-101 and to carry out a compilation of exploration data over the Lack inlier. The study identified twenty-four exploration targets. Follow-up on these targets resulted in the discovery of gold mineralisation at Cornavarrow Burn East, where a shear zone containing disseminated pyrite and galena included a 1.5m section returning 1.15 g/t gold.

European Gold Resources Inc was re-named Galantas Gold Corporation in 2004. Subsequent to a financing in the spring of 2005, Galantas initiated mine development by engaging technical staff, updating engineering design, procuring both mobile plant and processing plant equipment and removing further overburden. Construction of the ore processing plant commenced in November 2005 and mining development commenced in early 2006.

The mineral resources on which the Omagh Gold Project is based are hosted by a system of mineralised veins and shear structures within which more than a dozen individual deposits have been identified over a 4 sq. kms area. The most intensively studied area is the Kearney Structure, which has been diamond drill tested over its approximately 850 m length and shown to persist to at least 300 m below surface.

A resource and reserve estimate carried out by ACA Howe in 1995 estimated a total of 1.9 million tonnes at 7.06 g/t Au of Indicated resources and Probable reserves. That historical estimate is not in accordance with the Canadian Institute of Mining and Metallurgy and Petroleum CIM Standards on Minerals Resources and Reserve Definitions ("CIM Standards") and therefore does not conform to sections 1.3 and 1.4 of NI 43-101.

A CIM compliant resource estimate by ACA Howe in 2008 estimated Measured resources at the Kearney vein at 78,000 t at 6.35 g.t Au, Indicated resources at 350,000 at 6.74 g/t Au and Inferred resources at 730,000 t at 9.27 g/t Au. Open Pit mining at the Kearney vein commenced in 2006. By May 2012, mining was largely restricted to the northern end of the pit, mining in other parts having reached its economic limits as dictated by stripping ratio, by the property boundary and public road to the east, and by rock stockpiles to the west.

The Cavanacaw deposit lies within the Caledonian orogeny which extends through Scandinavia, the British Isles, Newfoundland and the Appalachians. It is hosted by rocks of Neoproterozoic age of the Dalradian Supergroup, which host similar orogenic vein deposits at Curraghinalt 27 km northeast and *at Cononish in Scotland. The mineralised veins strike either north-south or northwest-southeast and are steeply dipping. Mineralisation consists of quartz veins up to (and over) a metre wide with disseminated to massive auriferous sulphides, predominately pyrite and galena with some accessory arsenopyrite and chalcopyrite. The quartz veins are commonly accompanied by clay gouge and by an envelope of sericitised pelites.*

A large number of regional targets have been identified by past exploration on prospecting license OM1/09.

Diamond core drilling was mostly HQ3 (61.1mms) size and used triple tube core barrels to ensure good recovery. Core handling, logging and sampling were carried out to best industry standards. Core recovery within the mineralised veins generally exceeded 50% but in narrow veins sometimes fell below this value if clay gouge was encountered. There is no statistical correlation between core recovery and gold grade and ACA Howe therefore concluded that poor core recovery is not serious as to invalidate the use of drill core samples for resource estimation.

Drill intersections included some exceptional values including 7.6m at 8.44 g/t Au in hole 103 at Joshua vein and 3.5m at 11.2 g/t Au in hole 90B at Kearney vein (both intersections are true widths). Channel samples were collected by diamond saw at 10cm intervals across the vein. Drill core samples were determined by mineralisation and lithological type and were confined to the vein and immediate wall rock. Sampling of the orientated core was performed by diamond saw to produce one half core for retention and the other for assay.

*Analysis of all samples generated from channels and drill core was undertaken by OMAC Laboratories of Loughrea, County Galway, Ireland, which is accredited to ISO 17025. Sample preparation, gold fire assay with AA finish and ICP analysis for silver and 19 other elements followed industry standard methods. OMAC's internal QA/QC procedures using blanks, standards and duplicates were monitored by Galantas and ACA Howe (*NB : Howe up until June 2012 and Galantas thereafter*) and indicated that the assay data have a high level of accuracy and precision and that sample preparation resulted in no significant contamination. Quarter core samples returned somewhat erratic results when compared to original half core samples, due to the erratic distribution of gold/sulphide mineralisation in the core, which is exacerbated by the short sample length and small sample size of the quarter core. This problem could be mitigated by increasing the sample length but ACA Howe believes that this would not be justified since it would result in loss of definition of the gold distribution.*

*The authors carried out checks during site visits and confirmed that best practice logging and processing procedures were being implemented, witnessed core cutting and sampling, verified channel sampling locations and reviewed internal reports. The data supplied to ACA Howe by Galantas and by third parties appear reliable in the light of checks carried out by ACA Howe and the review of QA/QC practices. In view of these checks, ACA Howe is of the opinion that the data cited in this report (*NB. ACA Howe refers here to the Howe 2012 report*) are reliable and adequate for use in the resource estimate.*

The author of the Galantas 2014 report confirms that logging and processing procedures and QA/QC procedures have not altered, continue to reflect best practice and are reliable and adequate for use in the resource estimate.

1.2 MINERAL RESOURCE ESTIMATE

Galantas has prepared an updated estimate of mineral resources for the Kearney Vein system and Joshua's Vein system and for several other veins in the project area. Kearney and Joshua were the main focus of exploration since the Howe 2012 report, which had a cut-off date of June 1st 2012. The Galantas 2013 report had a cut-off date for channel samples and drilling of 18th May 2013. A summary plan view of the mine site showing drill locations associated with four previous drilling phases is shown in Figure 1. Figure 2 displays a schematic summary cross section of the main veins highlighting key features and intersects.

FIGURE 1 : OVERVIEW OF ALL CAVANACAW DRILL HOLE LOCATIONS

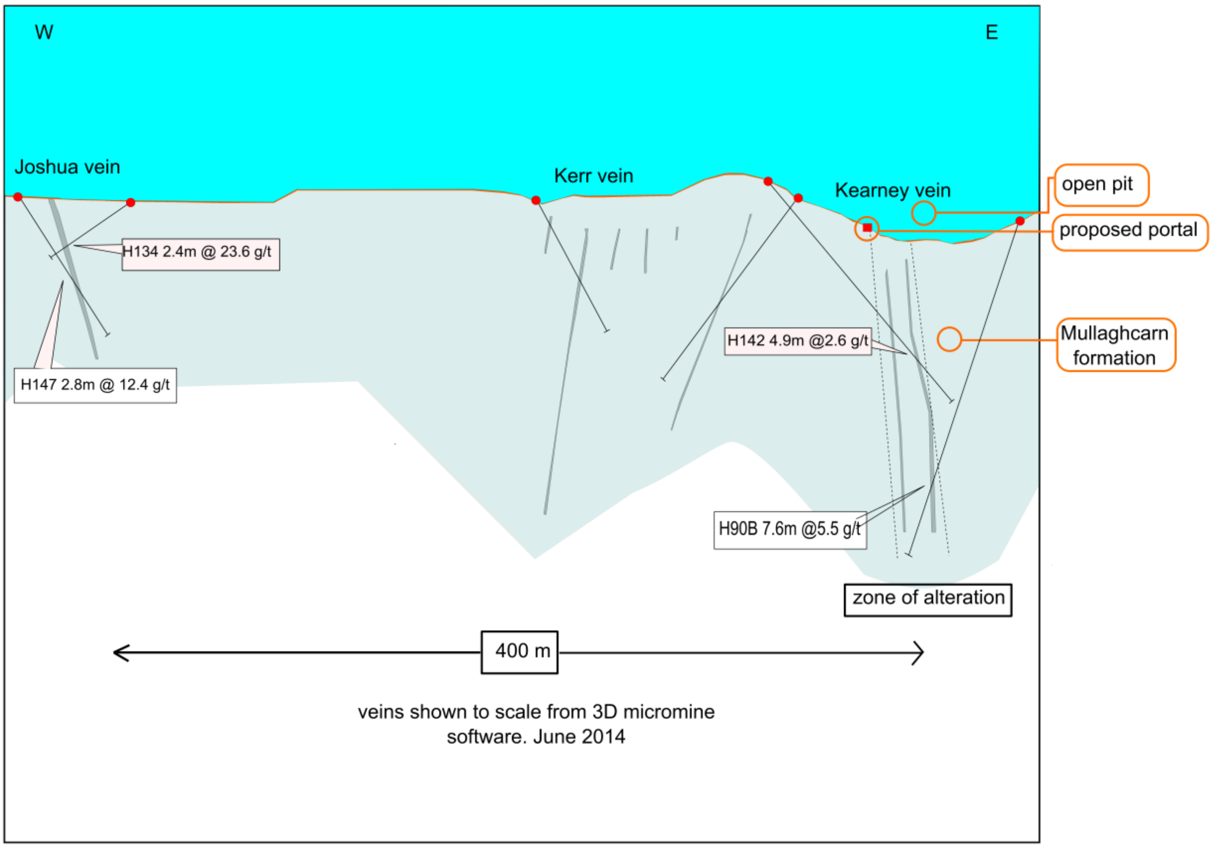


FIGURE 2: SCHEMATIC CROSS SECTION OF THE MAIN VEINS

Galantas followed the same procedures as carried out previously by ACA Howe in their Howe 2012 report. Mineralised zones were interpreted and Micromine software was used to create three dimensional wireframes. Sample data was selected and statistical analysis performed to assess the validity of this data for use in resource estimation. Following the generation of mineralisation domains, raw data was composited in order to standardise sample support. Further statistical and geostatistical analysis was performed on composite data to assess grade characteristics and continuity. The orientation and range of continuity selected for this study followed the same criteria assessed within Howe 2012 report, earlier deemed to be appropriate.

Following selection of orientation and range of grade continuity, wireframe constrained block models were created and grade interpolation into each block model was undertaken using the inverse distance weighting algorithm. Upon completion of block estimation, the resulting models were validated. Density values (based upon testing in the OML on-site laboratory) were entered into the block model, to calculate CIM compliant grade and tonnage estimates. A cut-off grade of 2.5 g/t gold was used in Howe 2012 and the Galantas 2013 report.

The cut off grade used in this report (Galantas 2014) is 2 g/t gold and has been revised in light of the adjusted cost estimates. The May 2013 resource estimate (to CIM code) for all veins at Cavanacaw is as follows :-

	PREVIOUS ESTIMATE GALANTAS 2013 CUT-OFF 2.5 g/t Au		
CATEGORY	**TONNES**	**GRADE (Au g/t)**	**Au Ozs**
MEASURED	77,919	5.87	20,772
INDICATED	651,582	5.85	121,761
INFERRED	1,403,746	6.54	295,599

TABLE 1: TOTAL RESOURCE ESTIMATE GALANTAS 2013

Note: Rounded numbers, gold grades capped at 75g/t. Diluted minimum vein width 0.9m

The increase in resources identified in the Galantas 2013 report when compared to the Howe 2012 resource report is mainly due to the increased amount of drilling carried out since June 2012. The drilling was designed, for the most part, to target Measured and Indicated resources.

The revised resource estimate (Galantas 2014) to PERC Code for all veins is:

	REVISED ESTIMATE GALANTAS 2014 CUT-OFF 2 g/t Au			Increase over
RESOURCE CATEGORY	**TONNES**	**GRADE (Au g/t)**	**Au Ozs**	**2013 report**
MEASURED	138,241	7.24	32,202	55%
INDICATED	679,992	6.78	147,784	21.4%
INFERRED	1,373,879	7.71	341,123	15.4%

TABLE 2: TOTAL RESOURCE ESTIMATE GALANTAS 2014

Minerals Resources that are not Mineral Reserves do not have demonstrated economic viability.

Were the CIM Code used for the resource estimation, the estimate would be the same. The increase in resources identified in the Galantas 2014 report is due mainly to drilling carried out subsequent to the Galantas 2013 report, and the process of re-stringing historical channels to drill core intersects.

Overall there has been a 60% increase in resources since the last independent estimate (Howe 2012) of resources, from 326,000 ounces of gold (Howe 2012) to 521,109 ounces of gold (Galantas 2014).

1.3 MINING STUDY RESOURCE UTILISATION

The mineral resource estimate used within mining studies is calculated to include all dilution and mining recovery estimates applicable to produce feed to the processing plant, and does not include processing plant metallurgical recovery. No mining reserve estimate has been estimated due to the independence requirement of NI.43-101.

1.4 MINE PLAN

The Howe 2012 report contained the following commentary regarding an internal Galantas cost study for an underground mine designed to exploit the deeper resources at Kearney and Joshua veins that are not amenable to open pit mining :-

The mining method proposed by Galantas is “Shrinkage Stoping with Backfill”, or “Cut and Fill” in areas not suited to Shrinkage. Underground access will be via a cut and cover ramp installed within the back-filled open pit and a spiral ramp developed from the base of the pit. Rubber-tyred diesel loaders, trucks *and development jumbo rigs are envisaged with jackleg operations within the production stopes. The proposed operation is anticipated to provide employment for approximately 130 persons.*

The existing plant comprises a three stage crushing system, two ball mills and flotation cells, which produce a sulphide concentrate with average gold grade of approximately 100 g/t that is shipped in bags to a smelter in Canada under a long term contract with Xstrata which is expected to continue.

The design of the new plant is based upon an up-rated version of the existing plant. Where components of the existing plant are compatible, they have been integrated into the new plant design.

The description of OML's proposed mine design as above has been reviewed and is materially current. The Preliminary Economic Assessment as (PEA) reviewed by Howe 2012 has been updated to reflect savings in capital and operating costs and also reflect a lower gold price than 2012.

OML owns the freehold land upon which the existing open pit mine has been excavated. Plans have been prepared which demonstrate that OML owns sufficient land for an underground operation, including land for tailings disposal.

The underground mine, up-rated processing plant and the export of a limited quantity of country rock from the underground mine will require planning permits to be issued through the Planning Service, Department of the Environment for Northern Ireland. OML submitted a detailed Environmental Impact Assessment with a planning application on 6th July 2012. Since that date, neighbour and statutory consultations have taken place. Several statutory consultees have written with comments encouraging approval. Notable are positive comments by Roads Service and Omagh District Council. Consultations continue with statutory consultees and Galantas is confident any remaining issues can be satisfactorily addressed to create a positive economic benefit for the local community whilst preserving strong environmental control.

Galantas notes two recent environmental studies on the operating mine site. The first of these studies prove conclusively that the country rock found at the mine is not acid forming and that some of the rocks are indicated to be potentially acid neutralising. The sampling was carried out by independent, environmental monitoring company Pentland Macdonald Ltd of Belfast. They undertook the collection of a representative set of 100 samples, with analysis taking place at the SGS Minerals Services Ltd laboratory in Cornwall. This extensive study is consistent with the results of earlier studies, which also showed no acid generation potential.

In a second report, a detailed Northern Ireland Environment Agency (NIEA) water study (June 2013) has declared Galantas subsidiary, Omagh Minerals Ltd, operator of the Omagh gold-mine, fully compliant with its water outlet requirements.

The NIEA study, which is the second one of its type on the gold-mine property with similar results, backs up routine sampling data with more detailed continuously recorded information and also demonstrates that no acidic drainage from the mine takes place.

1.5 ECONOMIC SUMMARY

The economic study has been carried out on only Joshua and Kearney veins and forms the basis for the Mining Study. The study (Preliminary Economic Assessment) includes a limited quantity of Inferred resource that is closely associated with the proposed mining of Measured and Indicated resources.

The resources on Joshua and Kearney veins used within the study are detailed below. The Percentage of each mineral classification scheduled for mining in each year is tabulated below (Table 4).

	Measured	Indicated	Inferred
Year	**Ounces Mined (%)**	**Ounces Mined (%)**	**Ounces Mined (%)**
1	2.3%	81.3%	16.5%
2	6.7%	67.6%	25.7%
3	9.4%	55.4%	35.3%
4	14.5%	55.5%	30.0%
5	12.0%	43.7%	44.2%
6	0.5%	38.4%	61.2%
Total	**9.0%**	**54.3%**	**36.7%**

TABLE 4: MINING SCHEDULE PER RESOURCE CLASSIFICATION

The total of scheduled Measured and Indicated ounces utilised within the mining study is 104,627 ounces. The total of Inferred resources scheduled in the economic study is 60,635 ounces. Total Inferred resource estimated on the Joshua and Kearney orebodies are 293,918 ounces of gold. The amount of Inferred resources included in the economic estimate amounts to 20.6% of the total Inferred resources estimated on these veins. Were Inferred resources excluded within the mining plan, approximately 1 year would be removed from the estimate of mine life and annual output would be reduced.

In compliance with NI 43-101 2.3.3(a) " ***the preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have 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*.**"

The Life of Mine Capital Cost Estimate shown in Table 5 has costs in United Kingdom Sterling (UK£).

LOM Capital Expenditure	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM
Capital Excluding Leasable Equipment	£1,679,432	£4,149,604	£422,355	£390,534	£0	£0	**£6,641,926**
Capital Leasable Equipment	£1,273,469	£1,334,177	£0	£0	£0	£0	**£2,607,646**
Contingency 15%	£442,935	£822,567	£63,353	£58,580	£0	£0	**£1,387,436**
Working Capital	£1,000,000	£0	£0	£0	£0	£0	**£1,000,000**
GRAND TOTAL	**£4,395,836**	**£6,306,349**	**£485,708**	**£449,115**	**£0**	**£0**	**£11,637,007**

TABLE 5: LIFE OF MINE CAPITAL EXPENDITURE SUMMARY

The operating costs and revenue based on gold prices of £800, £750 and £700, an ounce, are detailed in Tables 6-8.

Gold Price £800/oz	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM
Operating Costs	£5,693,338	£10,430,904	£11,964,071	£11,261,136	£10,830,431	£8,459,001	**£58,638,882**
Revenue	£5,711,798	£16,637,139	£21,615,178	£20,520,831	£21,092,405	£12,524,609	**£98,101,960**
Cashflow	£18,460	£6,206,235	£9,651,107	£9,259,695	£10,261,974	£4,065,608	**£39,463,079**

TABLE 6: OPERATING CASH FLOW AT AN AVERAGE GOLD PRICE OF £800 PER OUNCE

Gold Price £750/oz	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM
Operating Costs	£5,693,338	£10,430,904	£11,964,071	£11,261,136	£10,830,431	£8,459,001	**£58,638,882**
Revenue	£5,354,810	£15,597,318	£20,264,230	£19,238,279	£19,774,130	£11,741,821	**£91,970,588**
Cashflow	-£338,528	£5,166,414	£8,300,158	£7,977,143	£8,943,699	£3,282,820	**£33,331,706**

TABLE 7: OPERATING CASH FLOW AT AN AVERAGE GOLD PRICE OF £750 PER OUNCE

Gold Price £700/oz	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM
Operating Costs	£5,693,338	£10,430,904	£11,964,071	£11,261,136	£10,830,431	£8,459,001	**£58,638,882**
Revenue	£4,997,823	£14,557,497	£18,913,281	£17,955,727	£18,455,855	£10,959,033	**£85,839,215**
Cashflow	-£695,515	£4,126,592	£6,949,210	£6,694,591	£7,625,424	£2,500,032	**£27,200,334**

TABLE 8: OPERATING CASH FLOW AT AN AVERAGE GOLD PRICE OF £700 PER OUNCE

At a gold price of UK£750 / ounce, the net cash flow (pre-tax operating surplus less capital expenditure) provides an Internal Rate of Return of 72% and, at an 8% discount rate, a net present value of approximately UK£14.5m and a cash cost of production of UK£394 per ounce (USD$662 at $1.68/UK£). The study scheduled approximately 36% of the combined resources identified on the Kearney and Joshua veins.

2. INTRODUCTION

This report (Galantas 2014) is an updated mineral resource estimate for the Cavanacaw gold deposit, prepared in accordance with PERC (Pan European Reporting Standard). The report also summarises the project's geology and exploration potential. The economic study, summarised in this report, includes use of a minor portion of Inferred resources estimated for two veins (Joshua and Kearney veins). The Inferred resources are contiguous with or lie within scheduled mining areas. The use of Inferred resources, in a restricted qualifying manner, is permitted by the PERC code but is excluded within the CIM code / NI.43-101. For Canadian public disclosure, an economic study that uses any portion of inferred resources can only be considered a preliminary economic assessment, although PERC is an approved code in respect of NI. 43-101. A comparable economic study is shown which excludes the use of any inferred resources and is a feasibility study under NI.43-101. A comparison of terms associated with the CIM and PERC codes is tabulated in Appendix 1.

The report is not independent and is prepared under the supervision of R.Phelps C.Eng. MIOM3, (President & CEO, Galantas Gold Corporation), a Qualified Person for the purposes of NI 43-101.

The updated mineral resource estimate is based upon exploration drilling, conducted since May 2013. An interim, independent report (Howe 2012), commissioned from ACA Howe International Ltd, reported upon data generated to June 1st 2012 by the drilling & sampling campaign, which continued thereafter. The Galantas 2013 and 2014 reports include, in addition, data generated to 18th May 2013 and 31st March 2014, respectively.

This report is written in compliance with Canadian National Instrument (NI) 43-101 and in conformity with the requirements of the Ontario Securities Commission and utilises National Instrument 43-101 – Standards of Disclosure for Minerals Projects, Form 43-101F1 and Companion Policy 43-101CP.

3. RELIANCE ON THIRD PARTIES

Some data has been sourced from independent third parties, for instance, in the case of laboratory analyses and this data has been relied upon. Quality control checks were put in place to monitor the accuracy of third party data and those checks are deemed to provide an acceptable degree of repeatability.

The author has reviewed and relied upon independent reports, by ACA Howe International Ltd, calculating historical reserves and resources, evaluating exploration targets and reporting geophysical evaluations and field work. Please note that italicised parts of this document have been drawn from previous Howe reports. The author has also reviewed and relied upon internal Galantas Gold Corporation reports, third party reports commissioned by OML and a report on Title to Lands by Elliott Duffy Garrett (Solicitors to OML).

Additional information relied upon has been sourced from Galantas personnel, published topographic and geological maps, government authorities and government agencies. First-hand information known to the author has also been included.

To the best of the author's knowledge, having taken reasonable care, the information contained in this report is in accordance with the facts and makes no omission likely to affect the import of such information. While exercising all reasonable supervision and diligence in checking and confirming the data, the author has relied upon the information within the OML database and notes this is also reviewed for the greater part by independent persons.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 MINERAL LEGISLATION AND LICENSING

Two licensing (option) regimes are in place in Northern Ireland, relating to OML's operations. One is administered by the Crown Estate Commissioners (CEC), for gold and silver. The second is administered by the Department of Enterprise, Trade and Investment (DETI) of the devolved Government of Northern Ireland and covers base metals and other minerals.

DETI provides the following description of minerals licensing at www.detini.gov.uk :- "The Mineral Development Act (Northern Ireland) 1969 ("the 1969 Act") vested most minerals in the Department and enables it to grant prospecting licences and mining licences for exploration and development of minerals. This licensing system is based on the provisions of the 1969 Act and on subsequent subordinate legislation. The provisions relating to prospecting for minerals are quite separate and distinct from those relating to the development of minerals.

There is no automatic continuity between exploration and development work.

The legislation covers all minerals with three main exceptions (the scheduled substances):
(i) Gold and Silver belong to the Crown Estates and were not vested in the Department,
(ii) the few mineral deposits (mainly salt) which were being worked at the time of the 1969 Act were not vested in the Department, and,
(iii) 'common' substances including crushed rock, sand and gravel and brick clays are excluded."

Prospecting licences, from DETI and Options (formerly Crown Exploration Licences), from CEC, require agreed work programmes and can run for up to six years in two year increments. Generally drilling and other forms of exploration do not require planning consent but are regulated by statutory rules in Section 16 of the Planning (General Development) Order (Northern Ireland) 1993. Bonding arrangements are required and are in place. Mining operations need a separate Mining License and Planning Consent is required to enable the application to be made. In Northern Ireland, DETI collects royalties for base metals, where appropriate and precious metals royalties are payable to the Crown Estate.

In the Republic Of Ireland, Minerals Licences are administered by the Department of Communications, Energy and Natural Resources, through its Exploration and Mining Division. A six year programme is agreed at the onset of the license and the licence is renewable. Royalties, where a Mining Lease has been granted, are fixed by individual agreement. Further information is available on www.dcenr.gov.ie

Freehold title to the lands owned by Omagh Minerals Ltd has been reviewed by Elliott Duffy Garrett (EDG), OML's Belfast based lawyers. The author is satisfied from past reports on title by EDG, which he has personally seen, that OML has title to its land in all material respects but the author is not an expert in such matters and relies upon the advice of EDG. For completeness, the author notes a mortgage debenture in respect of a loan from G&F Phelps Ltd (a company related to the author).

4.2 LOCATION

The Cavanacaw Mine is located 5 kms west / south-west of Omagh, at approximately latitude 54^0 35' 00" north and longitude 7^0 22' 50" west. Related to the Irish National Grid, which is used for topographic and exploration data, this is the equivalent of IH 40046E and 70748N.

4.3 OMAGH MINERALS LTD LICENCES

Galantas Gold Corporations owns, through OML, exclusive exploration rights for gold, silver, base metals and other minerals, over the Northern Ireland licence areas shown in Figure 3 and the Republic of Ireland licence areas Figure 4.

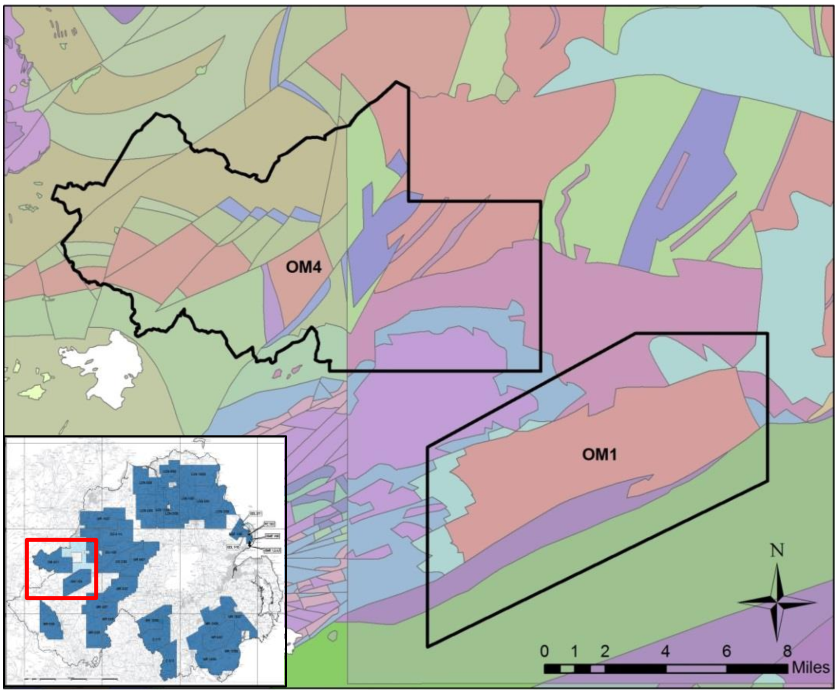


FIGURE 3: OMAGH MINERALS EXPLORATION LICENCES IN NORTHERN IRELAND. INSET MAP SHOWS AREAS UNDER LICENCE, PRODUCED BY DETI IN 2012.

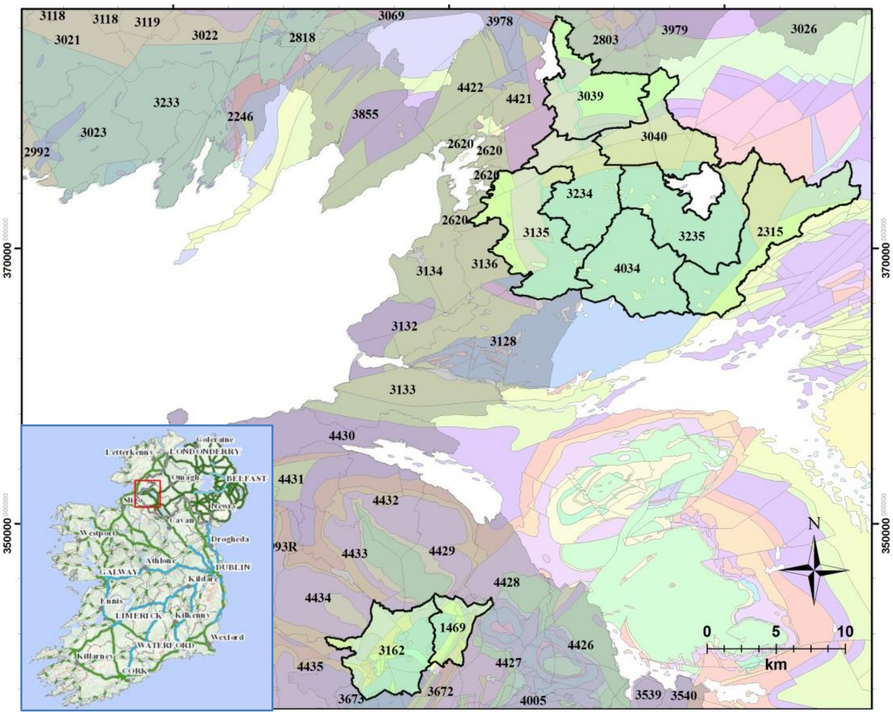


FIGURE 4: OMAGH MINERALS LTD EXPLORATION LICENCES IN THE REPUBLIC OF IRELAND

The current expiry of the Mines Royal Option Agreements are coincident with the Exploration License issued by DETI these were renewed on July 18th 2013 in respect of OM1/09 and is due on December 31st 2014, in respect of OM4/11. The Crown Estate has granted a Mines Royal Mining Lease to OML expiring June 22nd 2015 for the area shown in Figure 5, covering most of the OML freehold land but excluding the Elkins veins. Figure 5 is drawn from the "Howe 2012" report but the detail is unchanged.

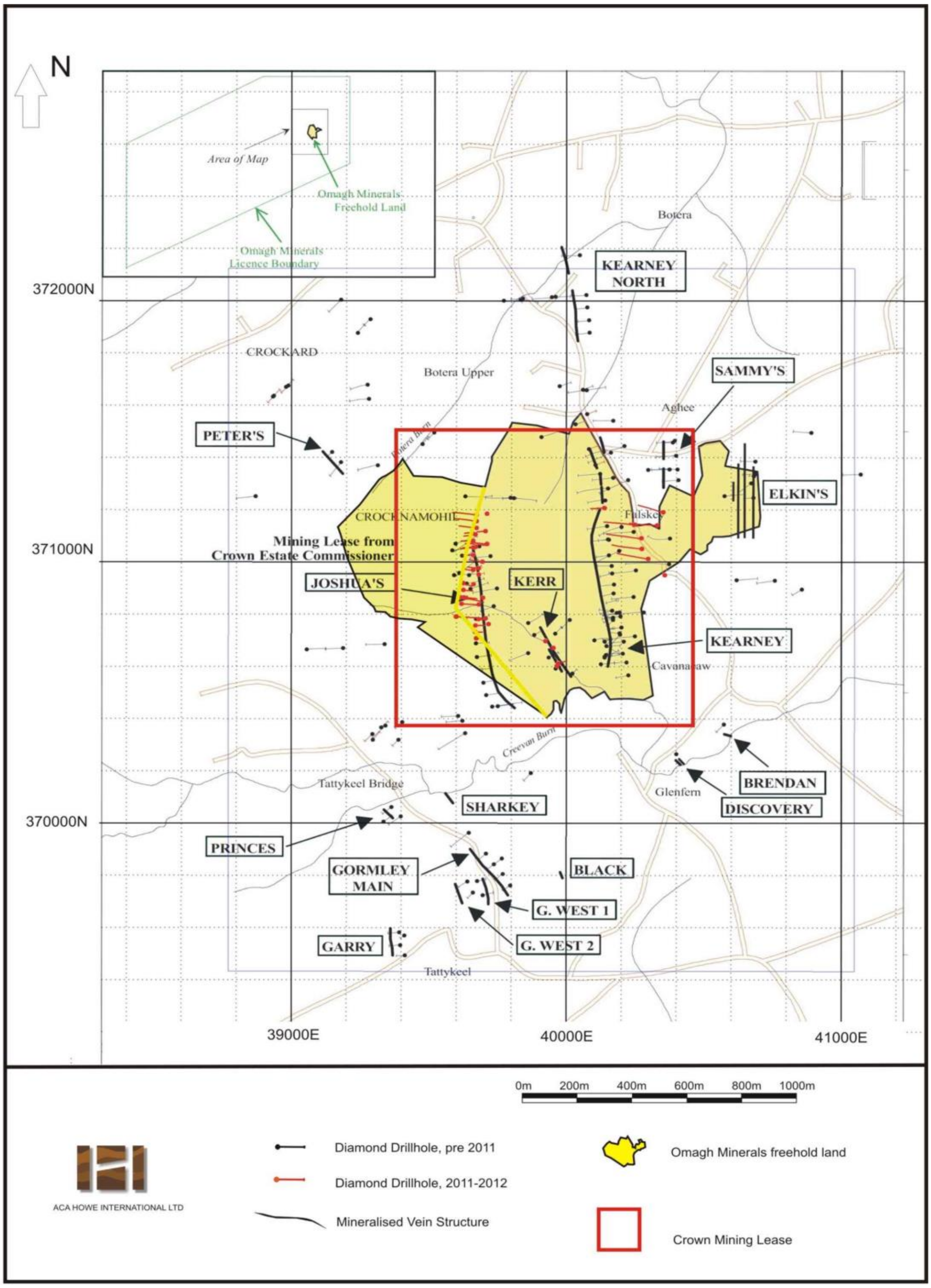


FIGURE 5: CAVANACAW PROJECT SHOWING FREEHOLD AND CROWN LEASE

4.4 PERMITS

The Department of the Environment for Northern Ireland (DoE NI) granted planning permission for open pit mining of gold and silver and associated minerals on certain areas of OML land in May 1995. A number of conditions were attached. The permission remains valid and in operation.

A further planning permission granted in 2012 by Planning Service, DoE NI, which permitted phased continuous restoration of the site and confirmed disposal of waste rock by road transport, was quashed by judicial review on the grounds of procedural failing by the Planning Service. The application awaits re-determination.

A planning application for an underground mine, uprated processing plant and the export of a limited quantity of country rock was submitted to the Planning Service, DoE NI, with a detailed Environmental Impact Assessment, on 6th July 2012. Since that date, neighbour and statutory consultations have taken place. Several statutory consultees have written with satisfied comments. Notable are positive comments by Roads Service and Omagh District Council. Consultations continue with statutory consultees and OML management is confident any remaining issues can be satisfactorily addressed.

OML holds Discharge Consents for mine waters from DoE NI, via the Northern Ireland Environment Agency (NIEA). OML and NIEA monitor flows from the mine and the author notes that the results are routinely within the limits imposed. A detailed compliance check on the consent conditions relating to ground waters and surface waters from the premises of Omagh Minerals was passed by NIEA and recorded as such on the 14th September 2011. A similar study, carried out in June 2013 also concluded compliance in all respects.

Other operating permits, such as that issued by the Industrial Pollution and Radiochemical Inspectorate (IPRI), are in operation and OML is adhering to the requirements.

A study of potential for acid drainage, reported in January 2013, concluded that the country rock found at the mine is not acid forming and that some of the rocks are indicated to be potentially acid neutralising. The sampling was carried out by an independent, environmental monitoring company, Pentland Macdonald Ltd of Belfast. They undertook the collection of a representative set of 100 samples, with analysis taking place at the SGS Minerals Services Ltd laboratory in Cornwall. This extensive study is consistent with the results of earlier work, which also showed no acid generation potential.

Restoration requirements exist under agreements made with regulating authorities. The Crown Estate hold a restoration Bond from OML to ensure the requirements for site restoration are met.

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

The county town of Omagh and the mine area are easily accessible by paved road from Belfast. The road distance is less than 100 kms and takes approximately 1.5 hours. Belfast is served by two airports with domestic and international flights. Situated some 5 kms from Omagh, the mine site is accessible by public paved roads. Some local roads have been recently improved, at OML expense, with additional passing bays, in order to improve a surplus rock haulage route. The mine site contains a concreted road to the processing plant and various unpaved roads. One upland section of the mine site is only reachable on foot.

Two power lines (33 kV and 11kV) traverse close to or on the site, however, power to the mill is generated with diesel on site.

The principal Prospecting Licence & Option (OM1/09) is situated on the south-western fringe of the Sperrin Mountains in glaciated terrain. Topography ranges from 140 m to 160 metres above sea level with rounded hills up to 330 m. Glacially derived till in thicknesses up to 18 m, provides generally low quality grazing, except where techniques such as drainage and fertility have been carried out to improve grazing quality. Farming, which is the principal local economic activity is dominated by small / medium sized operations that rely on raising cattle and sheep. Upland hills and hollows in the landscape, support peat bogs which have a history of small scale cutting for domestic fuel use. There has been some urbanisation of housing closer to the county town of Omagh, although it is understood that planning policy has in recent years restricted outwards growth further from the town. There are some small coniferous plantations for commercial forestry and one is situated on the mine site. Wind farms have been approved in the region, including one on an upland area within the western part of OM1/09, part of the mine site might be suitable for such a purpose.

The climate is temperate with about 1500 mm of rainfall per annum. The usual pattern of mild winters has been disrupted in recent years by severe falls in temperature. The mine has experienced some production difficulties during very cold temperatures but the disruption has been short lived.

Omagh is a County town and hosts schools, colleges and is a local administrative centre. The standard of education locally is good, housing costs are modest compared to many areas of the UK and unemployment is a local issue following the closure of several large employers. Operators of mobile plant are available in the local workforce. There is local knowledge of crushing and screening and local manufacturing of such equipment. Operators of the flotation plant have been trained. A number of skilled, small and medium size, engineering companies exist in the local region and the out-sourcing of a wide range of engineering and maintenance work is available.

OML has acquired its freehold land over a number of years, its latest acquisition being a parcel of 52 acres in January 2012 covering part of the Joshua vein. OML holds an estimated 220 acres of freehold land, which it believes is sufficient to operate its planned underground mine.

6. HISTORY

Howe 2012 contains the following description, which the author has reviewed :-
The occurrence of gold has been known for several centuries but no mining operations have taken place prior to Cavanacaw. A regional study of mineralisation by the Geological Survey of Northern Ireland (Arthurs, 1976) encouraged a new phase of mineral exploration in the Dalradian meta-sedimentary rocks in the 1980’s resulting in the discovery of vein hosted gold prospects associated with shear zones in Dalradian rocks at Curraghinalt (Earls et al.,1989; Clifford et al., 1992), Cavanacaw (Cliff and Wolfenden, 1992) and Golan Burn (Woodham etal., 1989).

6.1 PROJECT HISTORY

Following the Curraghinalt gold discovery, Riofinex commenced exploration on the Lack Inlier, a geologically uplifted block of Dalradian metasediments. The Kearney structure was discovered comparatively early in the exploration programme, the author has been told that it was concentrated upon partly because of ease of access.

A number of exploration and resource definition methodologies were employed. These included geological mapping, which is not readily achieved given the paucity of local rock exposures. Stream sediment was panned for gold and sulphide evidence. Loose boulders were sampled. Soil samples and deep overburden samples (Pionjar) were also taken. Induced polarisation geophysical work was deployed and core drilling (size NQ) carried out. The Kearney vein was stripped and an intense channel sampling programme resumed, backed by vein mapping. A resource was assessed and a mining project scoped. Environmental baseline studies were commenced.

In 1990 Omagh Minerals acquired the project from Riofinex and engineering studies from Kilborn Engineering Ltd, Knight Piesold and Lakefield Research. Wardell Armstrong carried out an Environmental Impact Assessment, which was completed in late 1992. The Crown estate Commissioners entered into a Mining Lease with OML, conditional on planning consent.

Following a Public Enquiry in 1993 and 1994, conditional planning consent was granted in 1995. The planning conditions were fulfilled in 2001, crystallising the consent.

Further engineering studies were carried out by Kilborn in 1995.

In 1997, European Gold Resources of Ontario acquired OML. OML excavated a section of the Kearney structure, to the north of the Riofinex trench, and mapped and sampled in a similar manner to Riofinex. ACA Howe carried out stream sediment sampling they also digitised and consolidated the resulting geochemical data.

In 1998, Lakefield Research completed further metallurgical and environmental studies.

In 2000 and 2001, OML carried out selective mining trials and produced a high grade, sulphidic ore. Following specialist laboratory treatment to separately recover the gold, the bullion was made into 18 ct jewellery with accreditation of the Irish gold source and test marketed under the Galantas brand name.

In 2003 EGR commissioned ACA Howe to analyse Landsat satellite imagery over the whole of the Lack Inlier and to integrate with other exploration data using MapInfo software. Resulting reconnaissance sampling, mapping, data compilation and interpretation was carried out subsequently (ACA Howe 2004A). Twenty-four exploration targets were identified.

European Gold Resources was renamed Galantas Gold Corporation (Galantas) in 2004.

Following a financing in early 2005, Galantas commenced mine development.

During the summer of 2005, Galantas contracted Geotech Airborne Ltd to carry out an airborne time domain electromagnetic (VTEM) and magnetic survey over the Lack Inlier. The results identified new geophysical targets and helped prioritise existing targets.

In December 2005, ACA Howe studied the resource potential of all targets and ranked them. Eight vein structures, including Kearney and Joshua veins were ranked as having good potential for upgrading of the reserves and resources previously enumerated.

Galantas started to build the ore processing plant in November 2005 and commenced mining development in early 2006.

6.2 HISTORICAL ESTIMATES OF MINERAL RESOURCES AND RESERVES.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

ACA Howe in their Howe 2012 report summarised their numerous reports, reserve and resource estimates from 1995 to 2008 as follows :-

Historical data used to calculate resources and reserves comprised saw-cut outcrop channel sampling, drill core sampling and selective mining trials by Riofinex and Omagh Minerals. Prior to 2007, three resource/reserve estimation studies (1995, 2004 and 2008) and a bulk mining trial (2003) were undertaken and are summarised below.

The following comparison of Howe's historic estimates is drawn from Howe's 2012 report.

6.2.1 INTRODUCTION

The mineral resources on which the Lack Gold project is based are hosted by a system of mineralised veins and shear structures within which more than a dozen individual deposits have been identified over a 4 sq. km area. The deposits can be grouped as:

- *the well-established Kearney Structure;*
- *several other nearby structures which have been variously explored, and whose reserves will be upgraded as part of on-going mine development to provide for continuation and later expansion of open pit operations, and;*
- *a large number of essentially untested gold occurrences, geological targets and geochemical anomalies distributed along the 20 km length and 5 km breadth of the Lack Inlier.*

The most intensively studied area is the Kearney Structure which has been diamond drill tested over its approximately 850m length and shown to persist to at least 300m below surface. It was investigated at surface by both Riofinex and Omagh by means of stripping and detailed bedrock sampling. The Kearney Structure has hitherto been the main focus of project studies both by Riofinex and Omagh and has been the main focus of open pit mining since 2006.

6.2.2 1995 RESOURCE/RESERVE ESTIMATE

In 1995, ACA Howe undertook a resource estimate for the Kearney Vein Zone and other named veins (details of these resource estimations are contained in the ACA Howe 2003 and 2005 reports, to which the reader is referred) using the polygonal sectional estimation method, now largely abandoned in the mining industry in favour of more robust linear and geostatistical methods of interpolation. At that time, the accepted standard for reserve and resource classification was the "Australasian code for reporting of identified mineral resources and ore reserves", developed by the Joint Committee of the Australasian Institute of Mining and Metallurgy and Australian Mining Industry Council (Joint Ore Reserve Committee = JORC code).

Accordingly, ACA Howe estimated JORC compliant proven and probable reserves using channel sampling data only, totalling 367,310 tonnes grading 7.52g/t Au (89,000ozs) over a width of 4.43m for the 850m strike length of a proposed open pit designed by Kilborn Engineering, and to a depth of 37m (the limit of the proposed Kilborn open pit). In addition, a further Indicated resource of 1,183,680 tonnes grading 7.02g/t Au (270,000ozs) over a width of 4.43m was estimated using historical drill hole data, based on extrapolation from the base of the proposed pit to a depth of 137m and along strike for a distance of 850 metres.

Data derived from limited trenching and drilling, partly defining other named veins in the Cavanacaw vein swarm were used to calculate an additional JORC compliant Indicated resource of 328,820 tonnes grading 6.72g/t Au (71,000ozs). Geochemical and geophysical data were used to extrapolate from these zones for the estimation of an additional Inferred resource of 135,500 tonnes at a grade of 4.38g/t Au (46,000ozs)

Estimated gold resources and reserves for the Omagh Project, calculated in 1995 totalled 427,000 ozs, as listed in Table 9 below.

TABLE 9: 1995 HISTORICAL RESERVE/RESOURCE ESTIMATE				
Vein	(t)	Au (g/t)	Au (ozs)	Class
Kearney, 0-37m depth	367,310	7.52	88,806	*Probable reserve*
Kearney, 37-137m depth	1,183,680	7.02	267,154	IND
subtotal, Kearney	**1,550,990**	**7.14**	**355,960**	
Elkins	97,600	3.50	11,000	IND
Kerr	6,950	6.30	1,400	IND
Joshua's	108,450	6.90	24,000	IND
Gormley	103,370	9.52	31,600	IND
Garry's	7,450	5.42	1,300	IND
Princes	5,000	10.10	1,600	IND
subtotal, other veins	**328,820**	**6.72**	**70,900**	**IND**
Total	**1,879,810**	**7.06**	**426,860**	

TABLE 9: 1995 HISTORICAL RESERVE/RESOURCE ESTIMATE

***** It should be noted that the above referenced Historical Reserves and Resources are not in accordance with the Canadian Institute of Mining and Metallurgy and Petroleum CIM Standards on Mineral Resources and Reserve Definitions ("CIM Standards") and therefore do not conform to sections 1.3 and 1.4 of NI 43-101***.

6.2.3 CONFORMITY OF 1995 RESOURCES TO CIM CLASSIFICATION

Although justified under the reporting code of the time, extrapolation of surface channel data over the entire 850m strike length of the Kearney deposit, into areas containing very little or no sample data does not meet the criteria for defining Indicated resources under current CIM guidelines (see Section 16.8, Howe 2012) since those parts of the resource informed by extrapolated grade data and not based on actual grade data are not reliably informed. Similarly, resources below the proposed pit floor at the time, estimated to a depth of 137m based on sparse drill hole data at spacing of between 75m and 200m does not meet the criteria for reporting Indicated resources under current CIM guidelines since the sample spacing is too wide to demonstrate grade continuity to the required level of confidence. In addition, the recognition of a sub-parallel, north plunging structure (the Lack Shear) which effectively cuts off the mineralisation to the south of the Kearney pit (drill tested by sterilisation holes during the 2006 drilling campaign) suggests that the extrapolation of resources down to 137m depth over the whole deposit resulted in a significant overestimation of contained resource tonnage.

Resources at the other named veins were classified as Indicated under the reporting code of the time, whereas the 2008 and 2012 estimates largely classify these resources as Inferred by virtue of the drill spacing (50m by 50m and 100m by 100m), and lack of demonstrated continuity, adhering to current CIM guidelines.

6.2.4 2004 RESOURCE AND RESERVE STUDY

In June 2004, ACA Howe commenced a re-analysis of the data to comply with the more rigid requirements of CIM/Canadian National Policy 43-101 for the definition of mineral reserves and resources (ACA Howe International Limited, 2004B). All the historical trench and drill data were reinterpreted and remodelled in Micromine software. Variograms showed that the natural area of influence for intersections is 20 metres. The most dependable data are the very closely spaced, saw-cut channel sample results from the Kearney deposit. Accordingly, the Kearney trench results were extrapolated for that distance along strike and down dip for Measured resources and for a further 17 metres down dip for Indicated resources. Using a 3 g/t Au cut-off and a density of 2.93 Measured and Indicated resources were calculated as shown in Table 10, below.

TABLE 10: MEASURED AND INDICATED RESOURCES ON KEARNEY STRUCTURE (2004)								
Cut-off 3 g/t Au, density 2.93 t/m^3								
Resource Category	**Grade g/t Au**	**Depth M**	**Trenched strike +20m N and S M**	**Measured Resource tonnes**	**Indicated Resource Tonnes**	**Total Meas. + Ind tonnes**	**Implied average width m**	**Grams Au Meas. + Ind.**
Measured	11.03	0 to 20	441	56,414	-	-	2.18	-
Indicated	11.03	20 to 37	441	-	58,363	-	2.66	-
Total Meas. + Ind.	**11.03**	**0 to 37**	**441**	-	-	114,777	2.40	1,265,990

TABLE 10: MEASURED AND INDICATED RESOURCES ON KEARNEY STRUCTURE (2004)

This partial estimation of the Kearney deposit resources, confirmed that higher grades could be maintained in a mining operation. Proportions of these Measured and Indicated resources could then be converted to proven and probable reserves respectively, following the development of a final mining plan.

6.2.5 BULK MINING TRIALS, 2003

The ACA Howe report of 2003 describes selective mining trials of high grade ore and gold recovery for jewellery manufacture and test marketing.

An 80 metre long section in the south end of the Kearney vein, which had been stripped and sampled in the late 1980's by Riofinex, was chosen for mining trials by Omagh Minerals in 2000 and 2001. The Riofinex sampling had been done in great detail with 533 samples taken on lines one metre apart and all assayed in independent laboratories. Using a cut-off grade of 1.0 g/t Au, this sampling had shown an average grade for the 80 metre section of 15.79 g/t Au and 23.57g/t Ag. Approximately 200 tonnes of visually identified, high grade, sulphidic ore were selectively extracted by Omagh Minerals, from 5 metre by 6 metre mining panels, by a closely supervised 4-man crew using a small excavator and hand sorting of sulphidic ore blocks. The ore was put into strong industrial bags for storage and shipping. The rejects of this operation which were surveyed as 2870 tonnes were stockpiled nearby.

Four lots of the high grade ore, amounting to just over 101 tonnes in total, were processed in two independent laboratories. Assay results showed an overall grade of 53.41 g/t Au. This is more than three times the gold grade shown by Riofinex channel sample results above a 1g/t Au cut-off. Analytical results and other details for the 101 tonnes processed are detailed in Table 11:

TABLE 11: GOLD AND SILVER CONTENT OF SELECTIVELY MINED HIGH GRADE ORE						
Lot Number	**Dry Wt**	**Gold Content**		**Silver Content**		**Processing Facility and gold recovery %**
	tonnes	**g/t**	**oz/t**	**g/t**	**oz/t**	
1	26.000	66.35	2.13	57.40	1.84	Reminex pilot plant, ONA Group, Maroc. 90.17%
2	25.688	50.90	1.77	38.00	1.22	Mintek Laboratory, Randburg, South Africa. 79%
3	25.016	40.80	1.31	32.80	1.05	Mintek, as above. 79%
4	24.650	50.70	1.63	74.30	2.38	Mintek, as above. 79%
Total	**101.354**	**53.41**	**1.71**	**50.52**	**1.62**	

TABLE 11: GOLD AND SILVER CONTENT OF SELECTIVELY MINED HIGH GRADE ORE

The results showed that, using selective mining techniques, it should have been possible to produce ore from the Kearney vein at a mill head grade markedly higher than the 7.52 g/t Au estimated in the 1995 reserve statement by ACA Howe.

However, the author (Galantas 2013 & 2014) notes that this was unachievable in practice.

6.2.6 2008 RESOURCE ESTIMATE

In 2008, ACA Howe undertook a resource estimate for the Kearney Vein Zone and other named veins.

The 2008 estimate was based on all data generated from channel sampling and drilling programmes carried out by Riofinex and Galantas up to that time. The 2008 estimate, using Micromine software, was based on a block model with sub-block cell dimensions of 1.5 metres (X), 0.5 metres (Y), 0.5 metres (Z) which was coded to reflect surface topography and geology. Gold grades were estimated from 0.3 metre length-weighted composites into the interpreted mineralised blocks. The estimates were calculated using Inverse Distance Squared and Cubed (IDW2 and IDW3) using parameters established from analysis of the variography within each domain. Based on the variographic analysis, search ellipses were created to enable a four-pass approach to interpolate gold grades into the blocks. A density factor of 2.984 grams/cc was assigned to all mineralised veins except Elkins, for which a density factor of 3.636 was used, based on measurements of specific gravity performed by Galantas. For resource classification, 4 trenches or drill holes with 4 composites were required within the search ellipsoid for classification as Measured, 2 drill holes with 3 composites were required for Indicated, the remainder being Inferred.

The 2008 resource estimate for the Kearney deposit and other named veins is summarised in the following table (Table 12) with resources classified in accordance with CIM Definition Standards on Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Resource Definitions and adopted by the CIM council on December 11, 2005.

TABLE 12: RESOURCE ESTIMATE, 2008					
Vein	SG	Tonnage	Au (g/t)	Au (ozs)	Classification
Kearney	2.984	78,000	6.35	16,000	MEASURED
Kearney	2.984	350,000	6.74	76,000	INDICATED
Kearney	2.984	730,000	9.27	218,000	INFERRED
Elkins	3.636	113,000	3.30	12,000	INDICATED
Elkins	3.636	29,000	3.82	3,600	INFERRED
Kerr	2.984	60,000	4.03	7,800	INFERRED
Joshua's	2.984	160,000	3.96	20,400	INFERRED
Gormley	2.984	115,000	6.57	24,300	INFERRED
Garry's	2.984	40,000	1.27	1,600	INFERRED
Princes	2.984	10,000	38.93	12,500	INFERRED
Sammy's	2.984	30,000	4.26	4,100	INFERRED
Kearney North	2.984	55,000	1.97	3,500	INFERRED

TABLE 12: RESOURCE ESTIMATE, 2008

In 2012, ACA Howe was commissioned to calculate an interim resource estimate that took into account depletion of resources by mining post 2008 and additional drilling and trenching. It used a cut-off grade of 2.5 g/t gold diluted at a minimum mining width of 0.9 m. Utilising Micromine software, it took into account and described in detail drilling and channel sampling carried out in the 2011 exploration programme, with a cut-off date of June 1st 2012 for results. It was a NI 43-101 report compliant to CIM standards and enumerated the following estimates in Table 13.

TABLE 13: ACA HOWE 2012 RESOURCE ESTIMATE				
ZONE	CATEGORY	CUT-OFF 2.5 g/t Au		
		TONNES	Grade (Au g/t)	Au ozs
KEARNEY	INDICATED	270,900	7.94	69,000
KEARNEY	INFERRED	490,000	8.54	135,000
JOSHUA	MEASURED	13,000	6.48	2,800
JOSHUA	INDICATED	66,800	6.27	13,000
JOSHUA	INFERRED	173,000	8.48	47,000
ELKINS	INDICATED	68,500	4.24	9,000
ELKINS	INFERRED	20,000	5.84	3,800
KERR	MEASURED	2,250	6.75	500
KERR	INDICATED	5,400	5.03	900
KERR	INFERRED	26,000	4.58	4,000
GORMLEYS	INFERRED	75,000	8.78	21,000
GARRY'S	INFERRED	0	0	0
PRINCES	INFERRED	10,000	38.11	13,000
SAMMY'S	INFERRED	27,000	6.07	5,000
KEARNEY NORTH	INFERRED	18,000	3.47	2,000
TOTAL	MEASURED	15,250	6.52	3,300
	INDICATED	411,600	7.01	92,000
	INFERRED	839,000	8.53	231,000

TABLE 13: ACA HOWE 2012 RESOURCE ESTIMATE

6.3 OPEN PIT MINING, 2006-2012

Open pit mining (other than bulk sampling) commenced in 2006. By May 2012, mining was largely restricted to the northern end of the pit, mining in other parts of the pit having reached economic limits as dictated by stripping ratio, by the property boundary and road to the east, and by rock stockpiles to the west.

The movement of the rock stockpile was prevented when OML's planning permission to do so was quashed by judicial review, following procedural failings of Planning Service, DoE NI, sterilising surface access to the deeper part of the northern section of the Kearney open pit.

7 GEOLOGICAL SETTING AND MINERALISATION

The following review, from Howe 2012, and as amended, describes the geological setting and mineralisation.

The geological setting and the gold mineralisation is described from a combination of information identified in References and Sources and from first hand observations and interpretations by ACA Howe (ACA Howe International Limited, 2003, 2005, 2006 and 2008). The location and the geological setting are shown in Figure 6. The veins of the Kearney swarm are depicted in Figure 5.

7.1 REGIONAL GEOLOGY AND GOLD DEPOSITS

The region forms part of the Caledonian orogen which extends through Scandinavia, the British Isles, Newfoundland and the Appalachians.

The principal host rocks of gold mineralisation in the region belong to the Neoproterozoic age Dalradian Supergroup which comprises a thick sequence of clastic marine sediments, with minor volcanic units, deposited in a passive-margin rift basin between c.800 million years ago and the early Cambrian during the breakup of the Late Precambrian supercontinent Rodinia, and the formation of the Iapetus Ocean of that geological time.

The Dalradian rocks can be correlated with successions in the Scottish Highlands, the Republic of Ireland (Cos. Donegal, Mayo and Galway) and perhaps the Fleur de Lys Supergroup in Newfoundland (Kennedy, 1975) and the Eleonore Bay Supergroup in eastern Greenland (Soper, 1994). Deposition took place along the eastern side of the palaeocontinent of Laurentia where extensive passive margin sedimentary sequences were formed in response to continental rifting and ocean widening, lasting until the early Ordovician (Strachan et al. 2002).

The Dalradian rocks consist of a metamorphosed clastic sedimentary package of biotite to garnet grade semi-pelites, (siltstone) psammites (impure sandstone) and chloritic-sericitic pelites (shale). The Dalradian terrane is structurally bounded to the south by the Highland Boundary Fault in Scotland and its western extension, the Omagh Thrust, in Ireland. Rocks immediately beneath the Omagh Thrust comprise Ordovician volcanics exposed in the Central Tyrone Inlier northeast of the Galantas licence area.

The Dalradian rocks of the Sperrins are interpreted to lie on the lower limb of a gently to moderately northwest-dipping major recumbent overturned tight isoclinal fold. This fold is referred to as either the Sperrins Overfold or the Sperrins Nappe.

The Galantas licence area mostly overlies rocks of the Upper Dalradian, Southern Highland Group, exposed in the Lack inlier, including the Glengawna Formation and the Mullagharn Formation. The Glengawna Formation contains a distinctive assemblage of psammites, talcose schists and graphitic pelites. The Cavanacaw deposit is hosted by the Mullaghcarn Formation that is composed of fine grained clastic meta-sedimentary rocks (psammite, semi-pelite and chlorite-rich pelite). Garnets are sometimes present, but are commonly replaced by chlorite or hematite.

Mineral exploration during the past 30 years has identified a number of significant deposits in the Caledonian orogenic belt including Curraghinalt and Cavanacaw in Northern Ireland and Cononish in Scotland. The strike extensions of the Caledonian belt into Scandinavia and North America are known to host a number of major mineral deposits in a similar geological environment. These include the Silurian hosted, shear-zone gold deposit of Kolsvik (Bindal) in Norway, the Upper Proterozoic, sandstone and porphyry hosted, high-sulphidation, epithermal gold deposit of Hope Brook in Newfoundland and the Ridgeway gold deposit in the Upper Proterozoic Slate Belt of South Carolina.

The mineralisation present is subject to two dominant structural controls, the north-south Omagh Lineament and the east-southeast trending Curraghinalt lateral ramp in the footwall of the northeast trending Omagh thrust (Parnell et al., 2000).

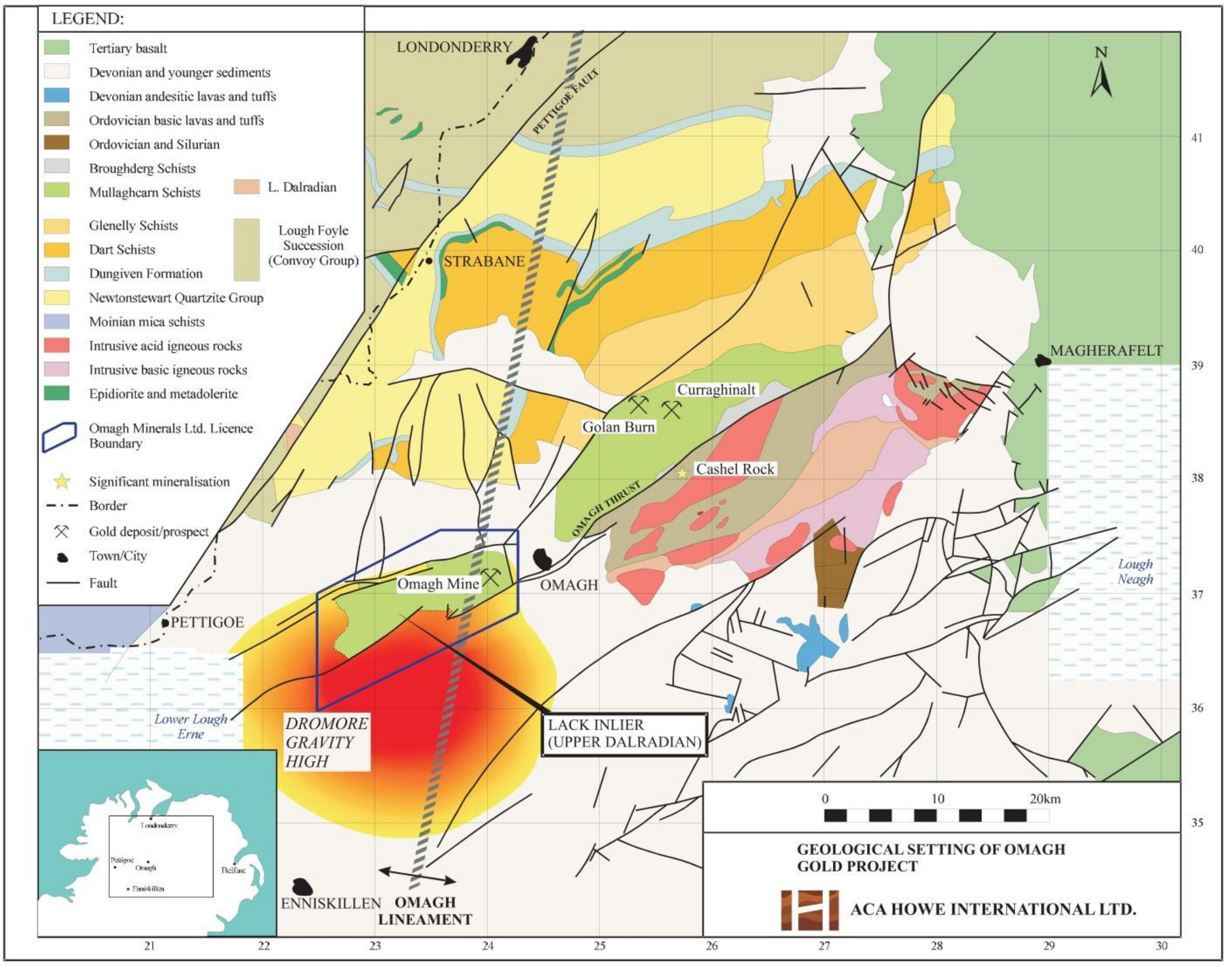


FIGURE 6: GEOLOGICAL SETTING OF OMAGH GOLD PROJECT

7.2 LOCAL GEOLOGY

As mapped at 1/50,000 scale by the Geological Survey of Northern Ireland, the Lack inlier is composed of undifferentiated mixed semipelite, schistose psammite and pelitic schistose slates of the Mullaghcarn Formation of the Southern Highland Group of the Dalradian Supergroup. In the southwest part of the inlier, there are several small Dalradian, schistose amphibolite bodies described as a metamorphosed sequence of basic volcaniclastic and igneous rocks. The schistosity in the Dalradian dips at various angles from 20 to 65 degrees in various strike directions but generally to the north-northwest. Minor fold axes are indicated, plunging at 6 to 40 degrees towards west-southwest in the western half of the inlier. One plunge symbol in the north - central area plunges east-northeast at 20 degrees.

The Dalradian of the eastern half of the Lack inlier, where most of the exploration work has been done, consists mainly of a series of quartz-feldspar-muscovite-chlorite schists (using the IUGS definition of that term) of varying composition with schistosity dipping at variable but generally low angles to the north-northwest. As indicated in work by Riofinex and the recent airborne electromagnetic survey, carbonaceous schists are prominently developed all along the northern boundary of the inlier and along a six-kilometre strike length at the eastern end of the southern boundary. In this area, the contact with the Lower Palaeozoic sediments is the Omagh Thrust, the plane of which dips to the north-northwest. The airborne electromagnetic data and Pionjar surveys indicate a number of internal lenses or layers of black schists within the Dalradian.

A few kilometres to the east-northeast, off the Omagh Minerals licence area, rocks of the Ordovician age, Tyrone Volcanic Group are mapped in thrust and fault contact with the Dalradian, Devonian and Carboniferous rocks.

The Dalradian of the Lack Inlier is in faulted contact with Carboniferous sedimentary rocks on its northern, southern and eastern boundaries. A small part of the southern boundary of the Dalradian inlier is mapped as an unconformity below the Upper Carboniferous Greenan Sandstone Formation. The western boundary is an unconformity below Lower Carboniferous Courceyan and Chadian sedimentary rocks cut and displaced by several faults with east-north-easterly trends, which penetrate the Dalradian inlier.

Tertiary age, dolerite dykes of north-westerly trend are mapped cutting the Dalradian and Lower Palaeozoic sedimentary rocks. A Tertiary olivine basalt dyke occupies part of the east-northeast trending, transcurrent, sinistral displacement, Cool Fault system which bounds the Dalradian inlier on the north side.

The Dalradian and Lower Palaeozoic rocks are largely but patchily covered by several metres of Quaternary glacial till and less extensive hill peat up to a few metres thick. Steep narrow gorges in till expose bedrock in some places.

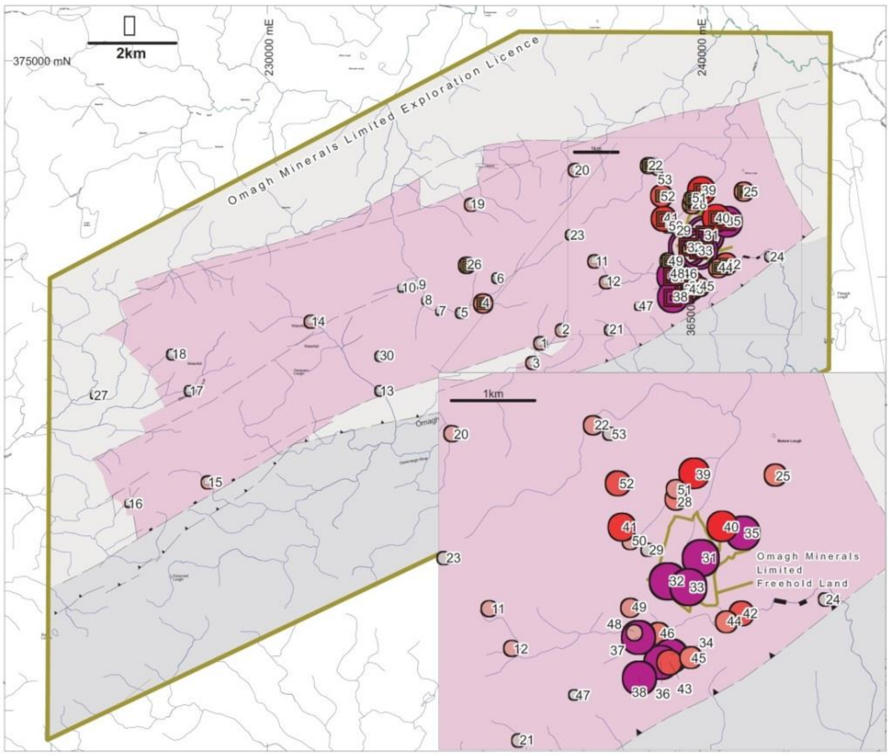
A major positive gravity anomaly known as the Dromore High is centred 10 kilometres south of the centre of the Lack inlier (Figure 6). A northern lobate "ridge" of this gravity anomaly trends east-northeastwards, coincident with the centre of the Dalradian inlier. Although the reason for the anomaly remains unknown, the most likely explanation in this environment is an unexposed, late Caledonian, granodioritic body which may be of significance as a heat source in the genesis of gold mineralisation.

The airborne geophysical data of 2005 is useful in the interpretation of geology in unmapped or overburden-masked areas. For example, the lithologies of the Mullaghcarn Formation of the Dalradian, are not differentiated on the published 1/50,000 scale geological map. However, the airborne electromagnetic geophysical surveys of 2005 enable conductive members of the formation (probably black, carbonaceous, sulphidic schists) to be outlined in a few areas. The mapped Dalradian amphibolites are clearly indicated by a prominent, regional strike-parallel, magnetic high anomaly in the vertical magnetic gradient map and other geologically significant magnetic strike lines can also be interpreted from this data. Mapped and unmapped Tertiary dykes are indicated by the magnetic data.

The northerly trending Omagh Lineament, one of three major, parallel, basement lineaments in the region, crosses the eastern part of the Lack inlier, in the area underlain by the northerly trending Kearney Vein swarm (Figure 6). This long-lived feature may have a zone of influence several kilometres wide. Earls et al. (1996b) concluded that the Omagh Lineament has a significant control on the location and orientation of the Cavanacaw mineralised veins, based on the distribution of gold and arsenic anomalies and the north-northeasterly or north-south orientation of mineralised veins in the vicinity of the Lack Inlier.

The Kearney vein swarm comprises 16 named vein structures in an area of about 6 square kilometres listed in order of importance as: Kearney, Joshua's, Kerr, Gormley Main, Elkin's, Gormley West 2, Princes, Garry, Kearney North, Sammy's, Peter's, Brendan, Gormley West 1, Discovery, Black and Sharkey (Figure 5). The largest of these is the Kearney vein with strike length of 850 metres (1000 metres including an IP anomaly) and widths up to 6.6 metres or more, dipping eastwards at 70 degrees. The maximum vertical extent proved by drilling is 200m. Since the Howe 2012 report, the maximum vertical extent has been deepened to 337m and remains open at depth. The Howe 2012 Report continues in its description :-

The Cornavarrow Burn showings are named Cornavarrow Burn East Showing and Cornavarrow Burn West Showing. These are located some 5 kilometres to the west of Kearney. The small West showing was relocated in 2003 but the Riofinex gold values were not confirmed by sample assay results. The poorly exposed East showing in the south bank and bed of the burn was discovered in 2003 and comprises 6.5 metres horizontal width of structurally complex mineralisation with 0.13 to 1.15 g/t Au and anomalous Ag and Pb and visible galena, possibly dipping northeast at 85 degrees but that may be the internal dip of a constituent quartz vein. It includes a pod of massive, dark, tough, silicified, quartz - sericite - graphitic pelite - pyrite - galena mineralisation, 1.5 metres in horizontal width, possibly dipping west at 20 degrees. It is not possible to discern the structure precisely in the available outcrop. Numerous other targets exist for undiscovered gold mineralisation throughout the licence area (Figure 7).



Target Key *

No	Score	No	Score	No	Score
31	10	45	5	6	2
32	10	46	5	13	2
33	10	22	4	17	2
34	9	26	4	18	2
35	9	49	4	21	2
36	9	51	4	23	2
37	9	1	3	24	2
38	9	2	3	29	2
39	8	3	3	30	2
40	8	11	3	53	2
41	7	12	3	7	1
42	6	14	3	8	1
43	6	15	3	9	1
52	6	19	3	10	1
4	5	20	3	16	1
25	5	48	3	27	1
28	5	50	3	47	1
44	5	5	2		

Map Legend

Simplified Geology
- Dalradian (Lack Inlier)
- Carboniferous
- Devonian/Carboniferous

Target Score *
- 10 (3)
- 9 (5)
- 8 (2)
- 7 (1)
- 6 (3)
- 5 (6)
- 4 (4)
- 3 (11)
- 2 (11)
- 1 (7)

* (Likelihood of target contributing to "ore" in near term).

FIGURE 7: LACK INLIER, GEOLOGY AND EXPLORATION TARGETS

7.3 MINERALISATION

Gold mineralisation is known in the Cavanacaw vein swarm (Figure 5) and in two showings in the Cornavarrow Burn some 5 kilometres to the west. Numerous other mainly geochemical targets for undiscovered gold mineralisation exist throughout the licence area (Figure 7) which is largely covered by glacial till and, in the higher areas, by hill peat.

Prior to 2005 the Kearney vein swarm had been explored by various methods including over 140 diamond drill holes.

The Kearney structure, as revealed by trenching, is a very complex zone of quartz-sulphide mineralisation and associated alteration, along which there has clearly been pre- to post-mineralisation movement, resulting in an irregular lattice-work of mineralised veins.

Quartz veins may swell from stringers to a width of over a metre, over a distance of several metres. The veins are commonly fringed by varying widths of clay gouge. Wallrock alteration in the form of sericitisation and bleaching may extend several metres into quartz-feldspar schist host rocks, depending on the degree of fracturing.

The more limited drilling and trenching on the other structures showed them to be broadly similar in terms of overall mineralogy and grade of mineralisation.

The mineralised veins that have been identified in the sequence strike either north-south or northwest-southeast and are steeply dipping. Mineralisation within the structures consists of quartz veins up to a metre wide with disseminated to massive auriferous sulphides, predominantly pyrite and galena with accessory arsenopyrite and chalcopyrite. Mineralisation may occur in the quartz veins, in clay gouge zones and in an envelope of sericitised schists, but is invariably indicated by a typical blue-grey or black colouration.

Gold values are closely correlated with sulphide content such that the tenor of mineralisation can be estimated visually in drill core and during open pit mining. Visible gold has not been reported in core and the low nugget effect is consistent with this and with the assumed presence of gold in very fine particle sizes, although no mineralogical studies have been carried out to confirm this.

The author (Galantas 2014) has reviewed this description by Howe 2012 and agrees that the description materially represents current knowledge, with the exceptions that a) the vertical extent of drilling has now proven the Kearney vein to persist to 337m below the surface and b) the author has seen early mineralogical studies carried out by Riofinex that demonstrated that gold was present on the surface of sulphide grains (notably pyrite) and within pyrite grains. Experimentation within the process plant has demonstrated that a small amount of visible gold can be liberated during processing by gravity means.

The ACA Howe 2012 description continues :-
Silver values are closely correlated with gold, averaging between 1.0 and 2.5 parts of silver to 1 part of gold. Silver values probably occur mainly in association with galena, but also alloyed with gold. No mineralogical studies have been carried out to confirm this or to determine if silver is present in a separate mineral species. The author (Galantas 2014) agrees with the general conclusion but not the statement that mineralogical studies have not been carried out. A recent paper by Birtel *et al.* (2011) reports on the study of electrum within the ore and during processing, using SEM analysis.

The ACA Howe 2012 description continues :- *Lead occurs as galena, and may return assays of several per cent. Lead and gold are closely correlated suggesting that they occur as part of the same* *mineralising event.* The author (Galantas 2014) agrees that there is good correlation but first hand knowledge of the deposit suggests two distinct mineralising events separated in time, during which displacement and re-working of mineralised material occurred.

The ACA Howe 2012 description continues :- *The vein swarm is transected and displaced dextrally by the Lack Shear Zone which strikes east-northeast and dips to the north in a zone 150 to 200 metres wide. Its latest movement clearly post-dates mineralisation. The veins are often dislocated by other shears and fractures and in plan this has resulted in a complex irregular lattice-work of mineralisation which does however, form a semi-continuous zone and across which any one particular channel sample may intersect anything from a quartz stringer to several veins or mineralised bodies. Detailed sampling shows a strong correlation between gold values above a 1 g/t Au cut-off and zones of quartz and gouge. These zones are visually very evident as they are characteristically blue-grey to black due to the associated fine-grained sulphide mineralisation.*

Vein structure is strongly developed in the more competent felsic schists and is generally narrower in the more ductile chloritic and graphitic schists. However, the latter only appear to be present in the extreme south of the deposit, in association with the Lack Shear, and there is no evidence from the drilling that these less favourable host rocks affect the down-dip potential of the structure elsewhere.

Post-glacial weathering of the deposit appears to have been minimal and limited to minor oxidation of pyrite in shallow parts of the more fractured and permeable sections of veins.

The author (Galantas 2014) considers the description materially accurate but notes that, during the Kearney open pit mining process, vertical and lateral continuity of pinch and swell vein structures were observed. Continuity of these structures were measurable in tens of metres, within an >800 metres long mineralised strike. For example, the photograph in Figure 8 shows part of an extracted ore zone, with horizontal continuity of an individual vein in excess of 200 metres, and vertical continuity to sub-outcrop, in excess of 20 metres. Adjacent veins remain to be worked beneath the overburden being removed at that time.

FIGURE 8: KEARNEY VEIN EXCAVATION IN FEBRUARY 2008

8 DEPOSIT TYPES

The following is a deposit type summary derived from ACA Howe's 2012 report:

The Cavanacaw deposit can be characterised as of Palaeozoic orogenic type. Orogenic gold deposits are typified by quartz-carbonate-sulphide dominant vein systems associated with deformed metamorphic terranes of all ages. Mineralisation displays strong structural controls at a variety of scales. Deposits are most commonly located on second- or third-order structures in the vicinity of large-scale compressional or transpressional structures formed at convergent margins (Groves et al. 2003).

The Cavanacaw gold deposit is one of several orogenic structurally controlled, mesothermal gold bearing quartz and quartz-sulphide vein systems located in the Caledonian basement rocks that underlie the area north of the Iapetus suture in the British Isles.

These deposits include Leadhills, Glenhead, and Clontibret in the southern Uplands terrane, Cregganbaun and Croagh Patrick in the north-western terrane, and Curraghinalt, Cavanacaw and Cononish in Grampian terrane. Although the rocks hosting each of these systems are very different, lithology was probably not an important control on gold mineralisation (Parnell et al., 2000). Rather, mineralisation was probably focused by fluid movement along shear zones within and between terranes in the latter stages of the Caledonian orogeny when strike-slip deformation was extensive. (Parnell et al., 2000 and Thompson et al., 1992). Late Caledonian sinistral strike-slip movement consolidated and separated the neo-Proterozoic (Dalradian) continental margin rocks in the north from exotic Highland Border and Midland Valley terranes to the south. The continuation of this zone in the Canadian Appalachians in Newfoundland (Baie Verte peninsula) and Nova Scotia (Meguma terrane) is also host to significant gold mineralisation (Kontak and Kerrich, 1995). Parnell et al (2000), in their paper entitled "Regional Fluid Flow and Gold Mineralisation in the Dalradian of the Sperrin Mountains, Northern Ireland" sought to develop a relative chronology of the complex vein systems in the gold prospects at Curraghinalt and Cavanacaw, characterize fluid chemistry both in the prospects and on a regional scale, constrain fluid and metal sulphur sources, identify structural controls on fluid migration, and document mineralogy and whole-rock geochemistry.

A remobilization event occurred during the Caledonian Orogeny which led to the quartz- sulphide brecciated deposit we see today. The poly metallic vein hosts Au-Ag-Pb- Zn and minor Cu. Galena addition occurred during re-mobilzation. A later reworking of minerals during the Variscian orogeny may have contributed by low temperature brines reworking gold at depth and overprinting of the metals, as well as reactivation of the shear zone.

9. REGIONAL AND CROSS BORDER EXPLORATION

Omagh Minerals currently holds eleven licences for exploration. These cover a total area of 767 km^2 and straddle Dalradian lithologies both north and south of the border (Figure 9). The age and structural geology of the rocks in these areas suggest that they are prospective for precious and base metal mineralisation. Recent work has centralised all of the pertinent historical data, for licences currently held, in a GIS database. Exploration activities have included outcrop mapping, sampling of float, bedrock, stream sediments and soils. The most relevant findings for each Licence zone, out with OM1-09, are summarised below.

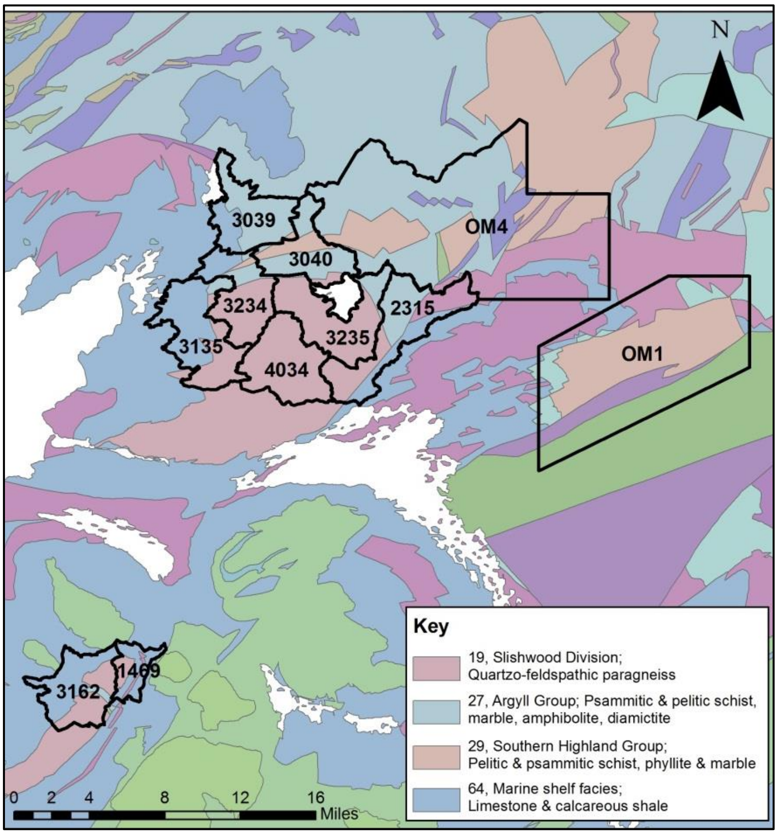


FIGURE 9: AN OUTLINE OF THE PROSPECTING LICENCE AREAS CURRENTLY HELD BY OMAGH MINERALS. BEDROCK GEOLOGY MAP 1:500,000 REPRODUCED FROM GSI DATASET.

9.1 OM4-11

A substantial licence area lies to the northwest of the mine site and borders the Republic of Ireland (ROI). The geology of this region, OM4-11, encompasses Dalradian units which belong to the Southern Highland and Argyll Groups. The main lithologies are psammites, pelites and semi-pelites which are similar in age and composition to the Mullaghcarn Fm of the Lack inlier (OM1-09). The geological evidence points to a complex history of deformation. The region is dissected by several NE-SW trending faults which are key prospecting targets, other faults cutting rocks of comparable age have acted as conduits for mineral rich fluids. Seven target areas were generated in 2011 in association with Aurum Exploration. Recent prospecting within the 248.4 km^2 licence has uncovered notable anomalies for Au, Ag and principal pathfinder elements. In one central area, cut by the prominent Pettigoe Fault, raised levels of Au, Ag, Pb, Zn and As have been found to occur throughout multiple sample types, including soils, float rock and small outcrops. These anomalies appear to intensify towards the south-western margin of OM4-11 where float rock samples returned assays of 2.7 g/t and 9.88 g/t in the vicinity of Glen Burn (Figure 10). In addition, a sample of schist containing quartz and minor pyrite assayed at 1.4 ppm Ag with raised Pb (945 ppm) and Zn (465 ppm) concentrations. The Glen Burn area is underlain by the youngest Dalradian unit, the Croaghgarrow Formation, and parts of the Mullyfa Formation.

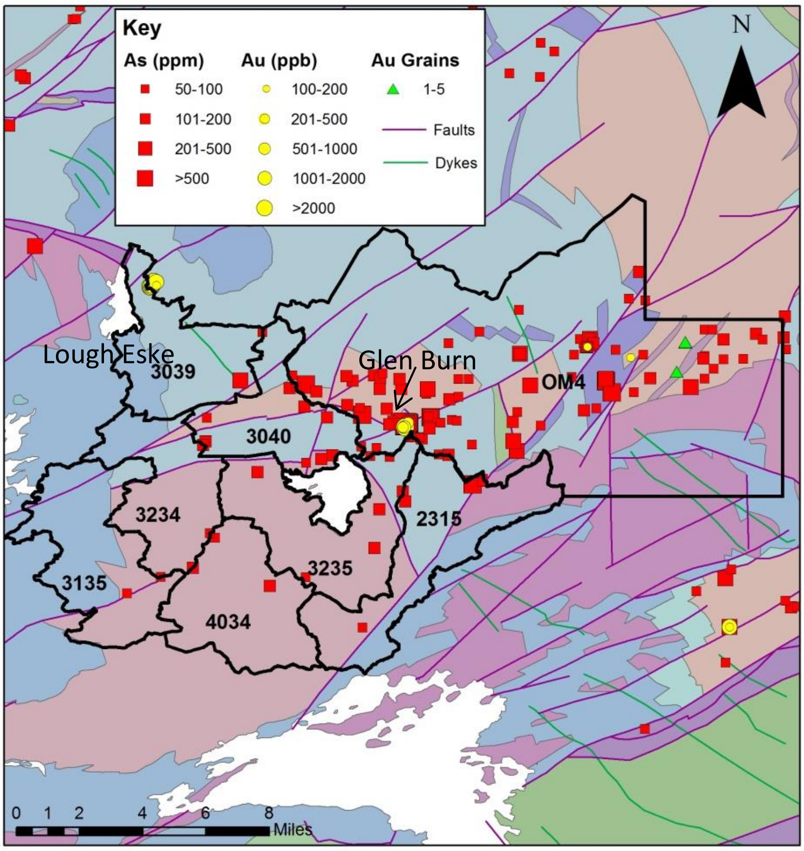


FIGURE 10: SUMMARY OF SIGNIFICANT GEOCHEMICAL ANOMALIES IN OM4-11 AND ROI LICENCE BLOCK 1. DATA DERIVED FROM HISTORIC AND RECENT EXPLORATION ACTIVITY. SAMPLES INCLUDE FLOAT ROCK, OUTCROP, SOILS AND STREAM SEDIMENTS.

9.2 ROI LICENCE BLOCK 1

In pursuit of the geochemical trend, and with an aim to intensify long-term exploration activities, Omagh Minerals invested in seven ROI licences immediately to the south west of OM4-11. These contiguous areas cover an expanse of 281 km^2 in county Donegal and host similar Dalradian age rocks and structures. Recently published results from the Tellus Border project demonstrate significant topsoil As anomalies and Au in stream sediments, within the licence block (Figure 11). Furthermore, historical work records bedrock Au in the vicinity of Lough Eske; along with gold concentrations 2000 – 5000 ppb in float samples. The Pettigoe Fault, which is a significant feature of OM4-11, continues through the ROI licences where it joins the ‘Lough Derg Slide’, a major curved thrust system that separates Moinian and Dalradian metasedimentary units. The intersection of these structures provides conditions favourable to the formation of mineral deposits. The Upper Dalradian rocks, which fringe the north and east of the Slide, contain observable gold particles in stream sediments (Moles & Schaffalitzky, 1992).

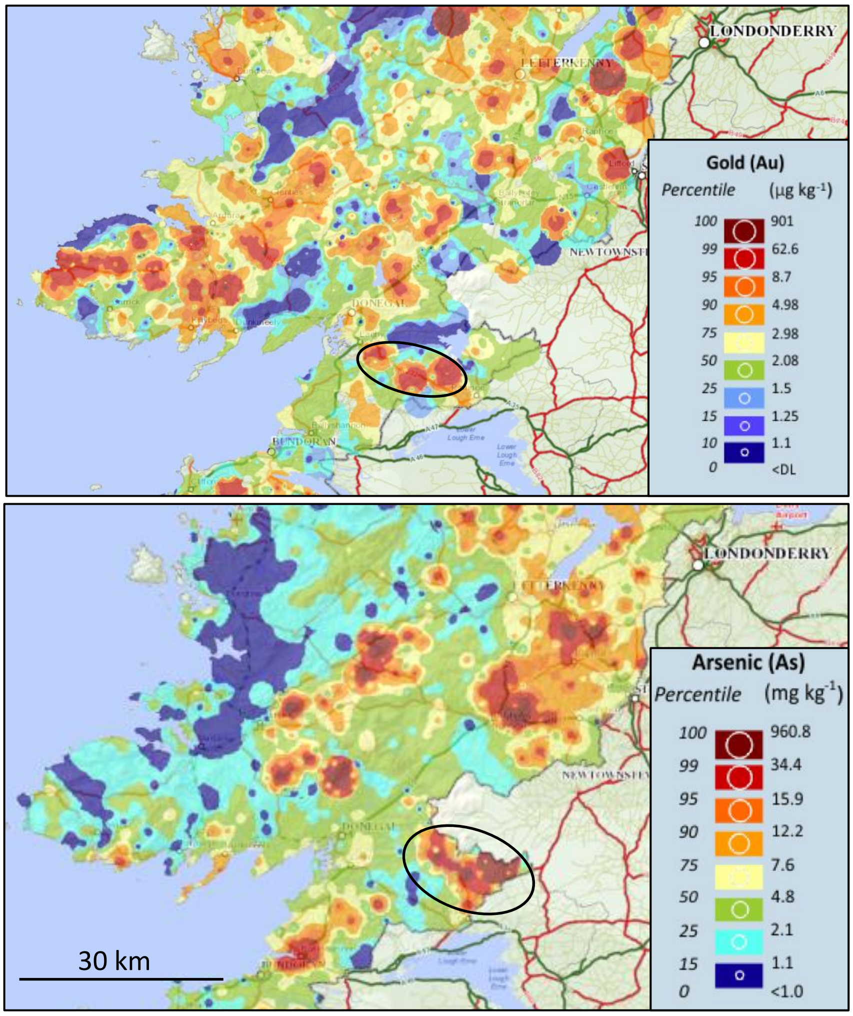


FIGURE 11: GOLD IN STREAM SEDIMENTS (TOP) AND ARSENIC LEVELS IN TOPSOILS (BOTTOM). DATA SOURCE: TELLUS BORDER VEIWER (http://www.tellusborder.eu)

9.3 ROI LICENCE BLOCK 2

Omagh Minerals proactively secured two new licences which are geographically separate from the main licence 'block 1', but which produced strong Ag, As and Sb signatures for topsoils collected and analysed during the Tellus Border project. These areas cover 47.75 km^2 to the west of Manorhamilton in Co Leitrim (Figure 9). A faulted block of predominantly Slishwood Division psammites (Grampian sub-group) is found in this region. The area is bound in the north and south by major regional faults, the Pettigoe Fault and the Fair Head – Clew Bay(FHCB) line, respectively. The FHCB line is correlated with the Omagh Thrust which runs directly south of the mine site in OM1-09. Therefore, there are strong parallels between the geological setting in the Manorhamilton and mine site areas. In addition, a small inlier of Meelick Schist sits within the new licence zone, it belongs to the Argyll Group and is contemporaneous with lithologies in OM4-11. Preliminary exploration was carried out as part of the Tellus Border research contract, awarded to Omagh Minerals by GSNI in April 2013. Fieldwork associated with this study identified stream sediment, panned concentrates and bedrock samples with raised Ag and Sb; some in combination with detectable Au and high As. These element associations point towards orogenic Au mineralisation (Coulter & Stinson, 2013).

9.4 SUMMARY

Exploration activities have focussed on Dalradian meta-sedimentary units. The size and orientation of faults is considered a key factor and sampling strategies have taken into account proximity to major and smaller scale structural features. Broad targets have been generated for the current licence areas and further prospecting will seek to trace the source of anomalies established thus far. The current licences will be held for the periods listed in Table 14; after which an application for renewal will be considered. The results highlighted in this chapter are derived from samples that were analysed in accordance with the quality assurance and quality control measures outlined in Chapter 12.

LICENCE CODE	Licence start	Period remaining
OM4-11	1/1/2011	32 months
3039	30/03/2012	48 months
3040	30/03/2012	48 months
3235	30/03/2012	48 months
2315	30/03/2012	48 months
4034	31/05/2013	62 months
3234	31/05/2013	62 months
3135	31/05/2013	62 months
3162	18/03/2014	71 months
1469	18/03/2014	71 months

TABLE 14: DURATION OF PROSPECTING LICENCE REMAINING FOR EACH AREA.

10. HISTORICAL EXPLORATION WITHIN THE MINE SITE

Galantas has explored the Lack Inlier on License OM1/09 since 1995. The history of exploration has been discussed in detail in Section 6. Table 15 below summarises channel sampling and core drilling local to the Cavanacaw Mine site.

TABLE 15: SUMMARY OF HISTORICAL AND RECENT EXPLORATION AT CAVANACAW MINE

number, from	number, to	Year	number	number of samples	Activity	total depth
TRENCHES						
line01	line 23	pre 1990	24	120	Rio channel	57
OMTRL288	OMTRL647	pre 1990	317	2872	Rio channel	3,615
T375	T522	2006	39	123	Galantas channels	285
OM-CH11/JA01	OM-CH11/JA39	2011	38	778	Galantas channels	78
OM-CH11/JS01	OM-CH11/JS38	2011	37	1464	Galantas channels	148
OM-CH11/KR01	OM-CH11/KR29	2011	28	428	Galantas channels	74
OM-CH11/KY01	OM-CH11/KY09	2011	9	230	Galantas channels	25
OM-CH-12/KY-10	OM-CH-12/KY-32	2012	23	763	Galantas channels	76.40
OM-CH-12/KR-01	OM-CH-12/KR-24	2012	24	588	Galantas channels	56.90
		Total	**539**			**4,415.30**
DIAMOND DRILLING						
OMBHL1	OMBHL167	pre 1990	153	1294	Rio ddh	13,963
OM-DD-06-01	OM-DD-06-14	2006	14	428	Galantas ddh	1,037
OM-DD-07-15	OM-DD-07-49	2007	34	1361	Galantas ddh	4,841
OM-DD-11-51	OM-DD-11-103	2011-2012	52	630	Galantas ddh	6,538
OM-DD-12-99B(KY)	OM-DD-13-148(JA)	2012-2013	46	775	Galantas ddh	8,911.5
		Total	**299**			**35,290.50**

TABLE 15: SUMMARY OF HISTORICAL AND RECENT DRILLING AND CHANNEL SAMPLING

Since 2008 exploration has concentrated largely on the Cavanacaw deposit and has mainly taken the form of diamond drilling (discussed in Section 11 of this report) and channel sampling, discussed in this Section. All channel samples during this period were collected from diamond sawed channels with dimensions 10cm wide and 10cm long by 5cm deep. During sampling, precautions were taken to ensure that clay gouge was not washed away.

10.1 CHANNEL SAMPLING, 2012

10.1.1 KEARNEY VEIN

The Kearney vein has been sampled by Riofinex and Galantas (OML) over the period 1987 to 2012 as discussed in Section 6 of this report. The last set of channel samples were cut at the north end of the Kearney open pit in August / September 2012.

10.1.2 JOSHUA VEIN

The Joshua vein was extensively channel sampled over a 225m section in a large trench in 2011 and reported in detail by Howe 2012. No further channel sampling took place on the Joshua vein post June 2012.

10.1.3 KERR VEIN

The Kerr pit, which is designed to be incorporated into a tailings storage area, exposes four steeply dipping quartz-sulphide veins, trending north-northwest (vein no.4) but the strike varies for each vein. The veins are generally less than one metre wide and extend over strike lengths of up to 25 metres before pinching out, although vein 4 increased to approximately 70m when excavated at depth.

During 2011 the Kerr pit was channelled along four distinct veins. Each vein pinches and swells along its length. The veins were sampled perpendicular to strike, typically in 2m lengths and extending into barren wall rock. A total of 28 channels were cut at 5m intervals and sampling was performed in 10cm lengths, resulting in the collection of 428 samples. The best of these, OM-CH-11/KR-22 gave a grade of 22.78 g/t Au over 40cm. The average grade was 2.5 g/t Au over 0.7m.

During 2012 the pit floor was dropped by 2m and the western most vein, vein 4, was shown to have developed the most (Figure 12). It showed greater continuity and an increase in width with the decrease in pit level than the three veins to the east. Vein 4 was channel sampled according to the procedure outlined above. This gave 8 channels, 2-3m wide, and showed the grade increasing substantially in the north, the most notable result being 23.37 g/t Au over 0.5m on channel OM-CH-12/KR-08. The average grade increased to 8.1 g/t Au over 0.9m at 144.3m RL.

On a further 5m drop in pit floor, 14 channels were sampled, demonstrating an increase in width and strike of vein 4 (Figure 12). Channels of two metres width in the north were increased to channels of five metres width in the south. This allowed for incorporation of the alteration zone. Each channel finished in barren wall rock. The grade and continuity of vein 4 was generally good, showing a small reduction with pinch and swell along its length. The average grade at RL 139.4m was 5.2 g/t Au over 0.9m. Primary or first stage massive sulphide mineralisation of pyrite and chalcopyrite, is found in South Kerr in a 50m channel dug to a depth of 2.5m.

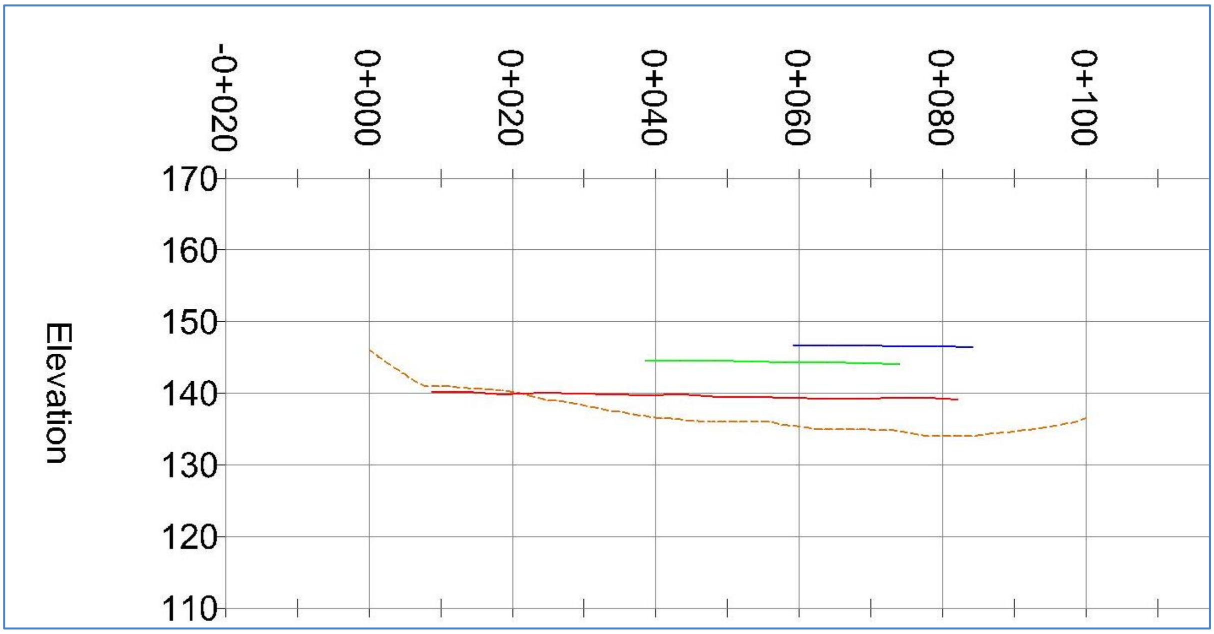


FIGURE 12: CHANNELLING SAMPLE ELEVATIONS IN KERR PIT

The average channel sample grades for the channel samples taken at three different elevations are as follows:

Upper (Blue) 2.5 g/t Au over 0.7 m (elevation 146.5 m)

Middle (Green) 8.1 g/t Au over 0.9 m (elevation 144.3 m)

Lower (Red) 5.2 g/t Au over 0.9 m (elevation 139.4 m)

The Kerr pit has now reached its design limits.

10.2 EXPLORATION POTENTIAL AND PRIORITISED PROJECT TARGETS IN OM1-09

As discussed above, the main focus of exploration has been on the Cavanacaw deposit.

Tables 16 and 17 below show the exploration potential and prioritised project targets from the OM1/09 Licence area held by Galantas.

Target name	No	Central Grid Ref.	Potential tonnes range (t)		Potential Grade Range (g/t Au)	
Resource extension targets						
			Low	High	Low	High
Kearney	31	H401/710	300,000	745,500	4.5	6.7
Joshua	32	H3970/7072	272,000	750,000	4.0	5.5
Elkins	35	H4061/7130	200,000	400,000	2.0	4.0
Kerr	33	H3995/7065	180,000	360,000	2.0	4.0
Gormley	34	H3974/6982	230,000	460,000	3.3	6.5
Sammy's	40	H4036/7138	30,000	60,000	2.1	4.2
Princes	37	H3935/7004	20,000	40,000	19	38
Garry's	38	H3936/6955	80,000	160,000	0.7	1.3
TOTAL			1,312,000	2,975,500		
Exploration targets						
Peter's	41	H3915/7137	4,000	13,000	4.5	9.0
63 Gram	52	H3910/7190	33,000	101,000	4.5	9.0
North of Sammy's Barn and East Cousins	28 & 51	H3980/7171 & H3980/7183	135,000	810,000	4.5	9.0
Cornavarrow Burn	4	H34977/69417	60,000	360,000	4.5	9.0
Corlea Burn	22	H388/726	60,000	360,000	4.5	9.0
Legphressy	26	H345/704	60,000	360,000	4.5	9.0
Cousins	50	H3925/7120	48,000	145,000	4.5	9.0
TOTAL			400,000	2,149,000		
TOTAL EXPLORATION POTENTIAL*			1,712,000	5,124,500		

TABLE 16: EXPLORATION POTENTIAL IN LICENCE OM1-09

*the potential quantity and grade disclosed in this table is conceptual in nature as there has been insufficient exploration to define mineral resources in these areas. It is uncertain if further exploration will result in the targets being delineated as a mineral resource. This exploration potential, expressed as ranges, is not a mineral resource does not have demonstrated economic viability.

The disclosed target potential quantity and grade ranges have been assessed based upon the reasonable extrapolation from defined resources and / or surface soil sampling, pionjar sampling, boulder sampling, favourable geophysical or favourable geochemical signatures, the results of which elevate these areas as highly prospective first order targets for further exploration.

Howe (2012) carried out a similar analysis of target potential. Most of the target estimates are similar, with the exception of Joshua and Kearney veins. The range of both of these targets has widened. In particular the Joshua target has increased significantly in the upper tonnage range bound due to an increase in known strike length and the lower and upper grade range bound has increased due to a higher known average grade and greater overall knowledge of grade variation.

Ref. No.	Name	No.	Central Grid Ref.	Score	Remarks including target type airborne geophysical anomalies of 2005, if any
			OM1 Licence Targets		
31-2005	Kearney	31	H401/710	10	Drilled with reserves (1994)and resources (1995, 2004, 2006-2013), IP anomalies over 300m strike at S end and on 5 lines over 400m at N end of mapped 1000m of IP extended strike, weak VTEM anomalies over only N half of strike. On freehold.
32-2005	Joshua	32	H3970/7072	10	Drilled with resources (1995, 2007-2013), IP anomaly with 200m strike of 600m total, Pionjar anomaly. Largely on freehold.
33-2005	Kerr	33	H3995/7065		Drilled with resources (1995), N extension of 500m indicated by IP, Pionjar anomalies over 300m. On freehold
34-2005	Gormley Main	34	H3974/6982	9	Drilled with resources (1995), coincides with minor public lane to Crocknageragh dead-end.
35-2004	Elkin’s	35	H4061/7130	9	Drilled with resources (1995), IP anomaly at S end of mapped vein trace over two lines and 50m extends S for 400m
36-2005	Gormley West 2	36	H3962/6974	9	Drilled with resources (1995)
37-2005	Princes	37	H3935/7004	9	Drilled with resources (1995)
38-2005	Garry’s	38	H3936/6955	9	Drilled with resources (1995)
39-2005	Kearney North	39	H4002/7202	8	Drilled with Inferred resources - low Au but high grade boulders locally and just downstream
40-2005	Sammy’s	40	H4036/7138	8	Drilled with Inferred resources of 2008 – low Au Pionjar gold anomaly on S strike, central two line IP anomaly.
41-2005	Peter’s	41	H3915/7137	7	Drilled no resources - low Au, one high grade boulder
42-2005	Brendan	42	H4059/7033	6	Drilled no resources - low Au
43-2005	Gormley West 1	43	H3972/6974	6	Drilled no resources - low Au
52-2005	63 Gram	52	H3910/7190	6	63 g/t Au and 3 other Pionjar and float Au anomalies and scattered IP anomalies in 150 x 150m area associated with west end of black schist sub-outcrop mapped over 800 x 30m trending ENE mapped by Pionjar, associated with the southern edge of a VTEM conductivity high 4.5 km ENE x 0.5 km wide just N of a 1.7 km parallel conductivity low about 50m wide.
4-2003	Cornavarrow Burn East Showing	4	H34977/69417	5	Stream sediment Au with samples exceeding 1,000 ppb, pans with 8-12 colours, anomalous float (1.5, 2.9 and 14.6 g/t Au). Outcrop with 0.13 to 1.15 g/t Au, anomalous Ag, Pb. N trend Landsat linear feature 60 m upstream. 11 km NE trend linear feature discordant to strike 100m to SE.
25-2005	Commings Bog	25	H410/720	3	Apparently non-cultural, 2 line, 100m NNE strike VTEM anomaly with <23 g/t Au nearby in soil (not coincident). May be due to massive sulphide related gold mineralisation below bog. A deep trench was dug on the anomaly and no mineralisation was found in bedrock material in the excavator bucket. Proper examination of the trench was not possible due to the unstable nature of peat side walls. Rock samples from the excavator bucket had been identified as subjected to alteration. Two drill holes were put down adjacent to the trench but no mineralisation was found and the anomaly remains largely unexplained. Consequently the target grade has been

					reduced from a 5 to a 3.
28-2005	North of Sammy's Barn.	28	H3980/7171	5	Possible northward continuation (with 2-300m strike) of Kearney main structure on three VTEM lines to W of Kearney North structure. Possible source of gold rich boulders.
44-2005	Discovery	44	H4041/7023	5	Named vein, not drilled
45-2005	Black	45	H3998/6980	5	Named vein, not drilled
46-2005	Sharkey	46	H3959/7009	5	Named vein, not drilled. Good float boulder.
22-2003	Corlea Burn	22	H388/726	4	Target has been followed up by Riofinex. See Pionjar gold in till anomalies. May have been surveyed with IP. Source of gold anomalous float samples may be local structures related to NE trending Landsat linear features but could also be dispersion from the Kearney-Joshua etc float gold cluster. Weak but potentially significant VTEM anomalies on 2 lines in area 200 x 100 elongated WNW? 3 lines of fuzzy response similar to North of Sammy's Barn, max 1km along flight line.
26-2005	Legphressy	26	H345/704	4	3 line VTEM with N trend Landsat linear break linkage to gold anomalies of 19-2003 – Unshinagh
49-2005	North Sharkey	49	H3925/7040	4	Pionjar Au anomaly and IP anomalies on six lines in area 200 x200m.
51-2005	East Cousins	51	H3980/7183	4	Four Pionjar Au anomalies and scattered IP anomalies on 7 lines in area 150m NE x 100m SE
1-2003	Aghadulla West Burn	1	H363/685	3	Stream sediment Au associated with N trending shears related to 4 mineral showings and to structures related to Landsat linear features.
2-2003	Aghadulla East Burn	2	H368/688	3	Stream sediment gold is probably locally derived from northerly trending shear structures related to the local, northerly trending Landsat linear features. Area of weak, subtle VTEM electromagnetic anomalies in the uppermost reaches of the Aghadulla East Burn.
3-2003	Aghadulla Main Burn below confluence	3	H3612/6805	3	Stream sediment gold is probably locally derived from northerly trending shear structures related to the local, northerly trending Landsat linear features.
11-2003	Upper Corradinna Bridge	11	H3755/7039	3	Followed up by Riofinex Pionjar sampling and possibly an IP survey. Gold may be derived from structures associated with local Landsat linear breaks. Prospecting results in stream bed were disappointing but bedrock source of local Pionjar gold anomaly may lie to NW covered by peat
12-2003	Upper Creevan Burn, western tributary	12	H3782/6991	3	Stream sediment and Pionjar Au may be derived from structures associated with four local Landsat linear features.
14-2003	Greenan Burn Upper	14	H310/690	3	Stream sediment gold near mineralised Aghaleague Fault structure with graphitic and calcite – dolomite veins containing fuchsite in western tributary, and three NNE Landsat linear breaks provide focus for float and outcrop prospecting. Access on land between the two burns may be problematical due to forestry established since 1981 fires.
15-2003	Viv Burn and Croneen Barr hill	15	H2862/6529	3	Gold colours, stream sediment Au, one sample >1,000 ppb, anomalous As and Pb. Landsat and airphoto linears.
19-2003	Unshinugh	19	H347/717	3	2003 target enhances 2005 VTEM anomaly Legphressy: Target 26-2005, as possible source of geochem anomaly
20-2003	Dressoge, upper Kilmore Burn	20	H371/725	3	Gold in stream sediments possibly derived from structures associated with Landsat linear feature, probably exposed in stream section immediately upstream of stream

					sediment gold anomaly
48-2005	West Sharkey	48	H3930/7010	3	Pionjar anomalies and IP anomalies on two lines.
50-2005	Cousins	50	H3925/7120	3	Scattered Pionjar Au anomalies and IP anomalies on seven lines in area 160 x 180m. N side of Cavanacaw magnetic low of Riofinex and Geotech 2005.

TABLE 17: EXPLORATION TARGET SUMMARY

11 DRILLING

11.1 OVERVIEW

Between March 2006 and June 2007, 49 diamond drill holes totalling 5,877 m were drilled over the project area. This phase of drilling and historical drilling by Riofinex is described in the 2008 report and summarised in Section 6 (History).

Between March 2011 and June 2012, a total of 52 diamond drill holes for 6,418 m were drilled over the project area, focussing on three high scoring resource augmentation targets; namely Kearney (10 holes), Kerr (4 holes) and Joshua (38 holes).

Between June 2012 and September 2013, 46 diamond drill holes were completed totalling 8,911.5 metres of triple tube recovered core. These focused chiefly on intersecting the Kearney vein at depth, with 8 holes totalling 2,419.9 m, and delineating Joshua vein along strike with 31 holes (5061.4 m). Further exploratory drilling was carried out on the Kerr veins (4 holes, 923 m) and three others targeting the western lagoon and IP anomalies (507 m). The results of most of these holes are detailed in Galantas (2013), however, three holes on the Joshua vein (H146-148) were completed after the 2013 cut off and are reported for the first time herein.

Collar locations for all phases of drilling are shown in Figures 13 and 14 and recent drill hole details are contained in Table 18 below:

TABLE 18: CAVANACAW DRILLING 2012-2014

Hole ID	Location	Easting	Northing	Elevation	Depth	Angle	Azimuth
OM-DD-12-99B	Kearney Vein	240404	371203	141.3	431	-45	270
OM-DD-12-104	Kearney Vein	240406	371040	133.2	413.3	-45	270
OM-DD-12-105	Joshua Vein	239672	371202	182.8	142	-45	270
OM-DD-12-106	Kearney Vein	240024	370710	162.9	330.6	-45	100
OM-DD-12-107	IP anomaly	239784	371012	172.7	176	-45	80
OM-DD-12-108	Joshua Vein	239672	371223	182.3	151.7	-45	270
OM-DD-12-109	IP anomaly	239672	371223	182.3	257	-45	110
OM-DD-12-110	Joshua Vein	239608	370764	165.6	189	-45	90
OM-DD-12-111	Joshua Vein	239673	371154	182.1	167.6	-65	270

OM-DD-12-112	Joshua Vein	239836	371172	171.4	443.3	-45	270
OM-DD-12-113	Joshua Vein	239581	370864	169	160	-45	90
OM-DD-12-114B	Joshua Vein	239610	370740	165.2	321	-65	90
OM-DD-12-115	Kearney Vein	239994	370710	164.2	332	-45	100
OM-DD-12-116	Joshua Vein	239900	370610	150.7	155	-45	270
OM-DD-12-117	Joshua Vein	239584	370893	174.9	135	-45	90
OM-DD-12-118	Joshua Vein	239600	370688	165.9	218	-45	90
OM-DD-12-119	Joshua Vein	239582	370893	174.8	494	-45	90
OM-DD-12-120	Joshua Vein	239575	370814	166.2	140	-45	90
OM-DD-12-121	Joshua Vein	239645	370690	165.5	152	-70	90
OM-DD-12-122	Joshua Vein	239590	370637	171.2	172.9	-45	90
OM-DD-12-123	Joshua Vein	239625	370715	165.2	109	-45	90
OM-DD-12-124	Kearney Vein	239954	370782	165.5	376	-45	90
OM-DD-12-125	Joshua Vein	239623	370715	165.3	121	-70	90
OM-DD-12-126	Joshua Vein	239630	370661	169.2	117	-45	90
OM-DD-12-127	Kerr Vein	239902	370584	150.2	163	-45	90
OM-DD-12-128	Kerr Vein	239899	370695	151.9	211	-45	90
OM-DD-12-129	Joshua Vein	239711	370915	171.7	97	-45	270
OM-DD-12-130	Kerr Vein	240074	370609	163	224	-45	285
OM-DD-12-131	Joshua Vein	239625	370605	170.5	120	-45	90
OM-DD-12-132	Joshua Vein	239700	370948	174	98.4	-51	270
OM-DD-12-133	Kerr Vein	240050	370590	161.4	325	-45	260
OM-DD-12-134	Joshua Vein	239714	370638	166.3	92	-50	270
OM-DD-12-135	Joshua Vein	239697	370949	174	158	-70	275
OM-DD-12-136	Joshua Vein	239719	370555	169.6	118.8	-50	275
OM-DD-12-137	Western lagoon	239869	370577	156.7	74.2	-45	275
OM-DD-12-138	Joshua Vein	239642	3711132	183.7	77.2	-45	270
OM-DD-12-139	Kearney Vein	240074	370611	162.6	139.5	-45	70
OM-DD-12-140	Joshua Vein	239720	370579	168.5	78	-45	275
OM-DD-12-141	Joshua Vein	239624	370820	164.1	95.1	-45	90

OM-DD-12-142	Kearney Vein	240074	370611	167.6	212.5	-45	90
OM-DD-12-143	Joshua Vein	239721	370579	168.4	118.6	-60	270
OM-DD-12-144	Joshua Vein	239719	370530	169.7	110	-45	275
OM-DD-12-145	Kearney Vein	240047	371201	155.1	185	-50	90
OM-DD-13-146	Joshua vein	239826	370638	157.0	237	-45	273
OM-DD-13-147	Joshua vein	239604	370740	165.2	171.8	-70	90
OM-DD-13-148	Joshua vein	239611	370741	165.2	101	-50	90
				TOTAL	**8,911.5**	**2012-2014**	
				TOTAL	5,819.4	2006-2007	
				TOTAL	7,001.1	2011-2012	
				TOTAL	21,732	2006-2014	

TABLE 18: CAVANACAW DRILLING 2012-2014

FIGURE 13: RIO AND EARLY GALANTAS DRILLING AT CAVANACAW

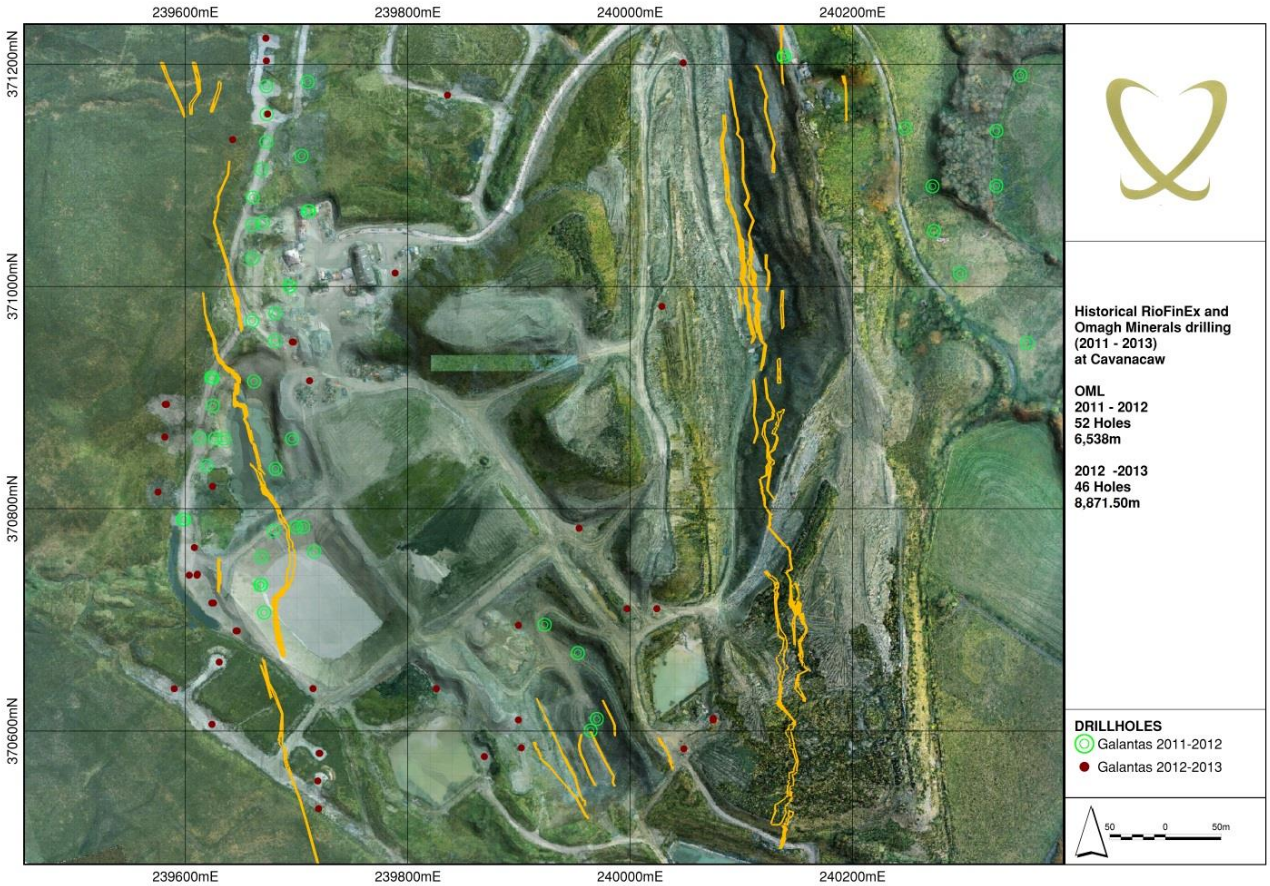


FIGURE 14: LATER GALANTAS DRILLING AT CAVANACAW

11.2 DRILLING METHODOLOGY

The drilling method outlined in the ACA Howe report of 2012 was in accordance with industry best practice and this has continued unchanged into the latest phase of drilling reported herein.

Drilling contractors utilised JKS Boyles BBS37 rigs with wireline equipment. Some holes were drilled using Galantas' own Atlas Copco CS14 rig. HQ triple tube coring was implemented (61.1mm) and hexagonal core barrels were brought in to maintain the desired drilling inclination in some of the later holes. The option of switching to NQ was kept and was required for the deeper sections of some holes.

Drilling and data recording activities were supervised by senior geological personnel to ensure industry best practices for drilling were maintained.

Up to six rigs were operational during the latest drilling programme. By January 2013 this had been reduced to one CS-14 Omagh Minerals owned rig operated by Omagh Minerals drilling personnel.

Drilling took place in 1.5m lengths, or runs, and extracted using a secondary winch system. The core was then very carefully pumped from the core barrel into a half tube and examined at the drill side for signs of mineralisation, the site was visited by a qualified geologist and a note made of the depth, width and mineralogy of the intersect.

The core was then transferred to core boxes where the hole number, start and end drilled depths, box number, measured intervals and any cavities encountered were marked on the boxes in indelible marker pen. The core was cleaned for the supervising geologist and taken to the core logging facility for geological logging. The core logging facility is well protected from the elements and is set up inside a converted shipping container with a perspex roof and power for light and heating. A custom built angled core rack can accommodate up to 24 boxes and allows access over the core for logging.

On completion of a drill hole the down hole survey was carried out using Flexit MultiSmart multi-shot surveying equipment to collect angle and azimuth data at, most commonly, 15m (sometimes 18m) intervals down-hole. This equipment was supplied by the drilling contractor. The surveying device is controlled via a StoreIT data pad, PC or palm-held unit. Data collected was validated and appended to the Micromine database for the project. The hole was cement grouted to below the rock head – till interface where ground water seepage is most likely to occur.

Core recovery data were collected by measuring the actual core lengths of each run and comparing this value with lengths written on core blocks by the driller. Any core loss was noted and the probable zone of core loss ascertained, often in consultation with the driller.

Once the core was measured up, geological logging was undertaken and pertinent geological and structural information collected, including lithology, alteration, structure, quartz vein characteristics and sulphide content. In addition, rock quality designation (RQD) data were collected. Geological information was recorded on detailed hand written log sheets and then manually entered into digital logs, which were merged into the Micromine database for the project.

Once logged the cores were photographed wet and dry and the photographs stored on file. Core boxes were then placed in racks ready for sampling.

Howe (2012) reported the monitoring of drilling practices and confirm that they conform to industry best practices. The report concluded that any geological samples taken were accurately measured from the collar and that the downhole surveys confidently arrange them in 3D space. These practices continue.

Two holes totalling 825m were drilled using the top marked Reflex Act II core orientation tool. This allowed for a mark to be accurately placed on the end of the core as it protruded from the core barrel. The mark was then continued along the entire length of the core making it possible to measure alpha and beta angles against it. This enabled the orientations of structures to be plotted with true dip and true dip directions. The structural boundaries and control can be then mapped in preparation for underground mining.

11.3 CORE RECOVERY

Core recovery reported in the ACA Howe report of 2012 was generally good ranging from 80 to 99%. Rio Finex data prior to this is not available although it is reported that conventional core barrels were used (ACA HOWE 2012) giving poor recovery. The Galantas report, 2013, highlighted the positive impact of the use of drilling muds from hole 95 onwards. The use of muds led to a 17% increase in recovery over the intersect. Two holes since then have less than 60% recovery over the intersect; holes 117 and 136, shown in Figure 15. Contrasts in competency between very hard quartz vein and extremely soft clay gouge/ crush from the adjacent shearing, explain losses in recovery. Clay particles are suspended and washed away by the circulating drilling fluids, even with the additive of muds. The effect of the addition of muds is to increase density and viscosity of the drilling fluids. However large differences in rock strength make total recovery across these intersects difficult to achieve. Gold grades may be affected by a loss in clay. Statistical analysis was carried out on the core recovery for the 2011 – 2014 drilling programme shown in Figure 16.

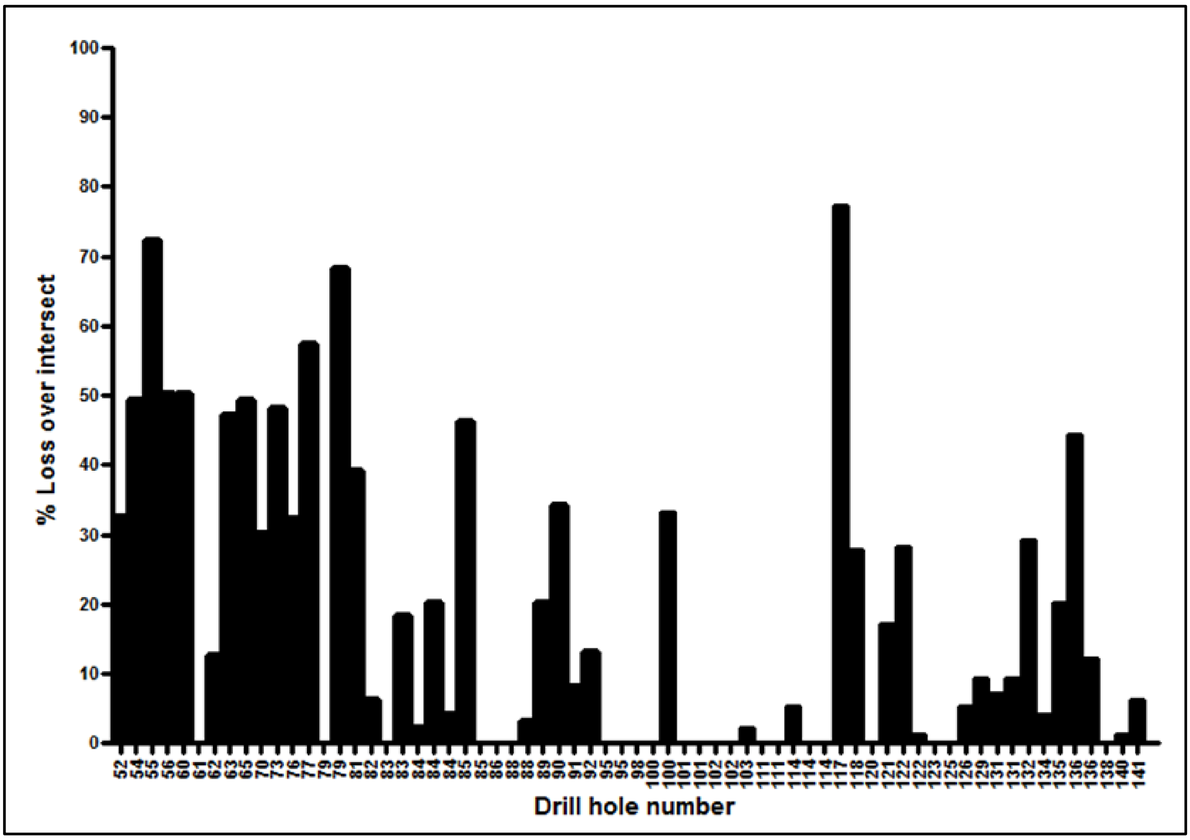


FIGURE 15: CORE LOSS % OVER DRILLED VEIN INTERSECTS

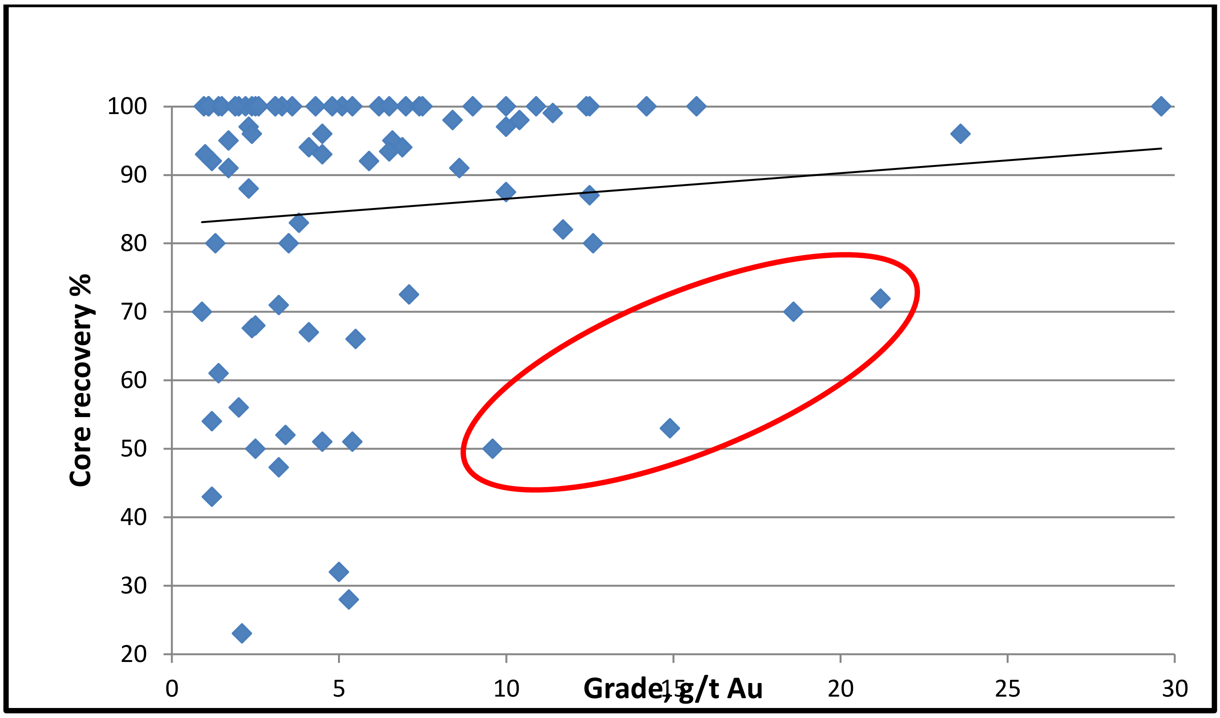


FIGURE 16: PLOT OF CORE RECOVERY VS. GRADE OVER THE VEIN INTERSECTS

Figure 16 shows four values highlighted in the ellipse, that may be identified as outliers. They are listed as: H56 at 10 g/t with 53% intersect recovery; H63 at 15 g/t and 51% recovery; H70 at 18 g/t and 70% recovery and H122 at 21 g/t and 72% recovery. Holes 70 and 122 have sufficient recovery for inclusion in the resource and there appears to be no direct relationship between recovery and grade. Figures 17 & 18 detail recoveries for individual drill cores extracted during the 2011-2014 Galantas drilling programme. The average recovery over this period is 85%. Figure 19 shows a cumulative frequency plot for the 2011-2014 drilling phase. This demonstrates that less than 7% of intersects are below 50% recovery.

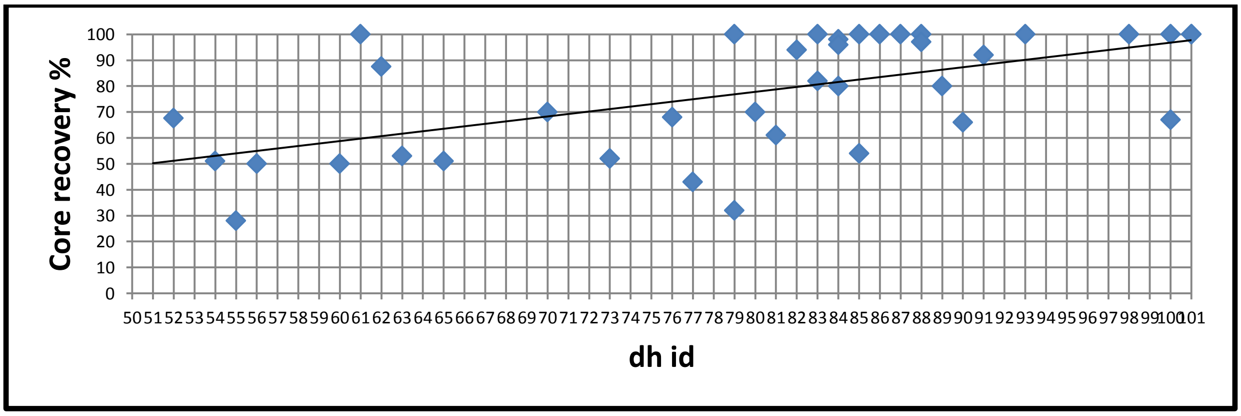


FIGURE 17: 2011 CORE RECOVERY AND DRILL HOLE ID

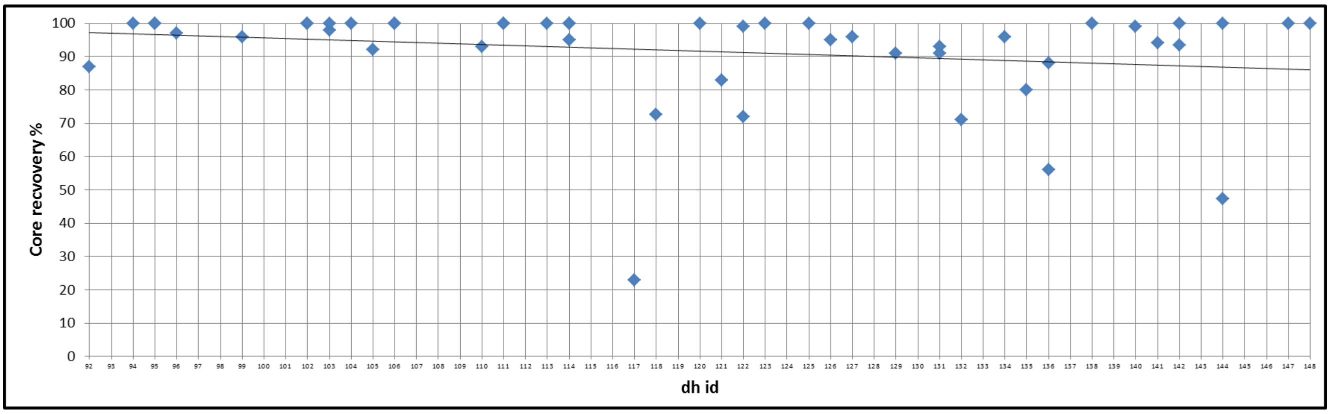


FIGURE 18: 2012-2014 CORE RECOVERY AND DRILL HOLE ID

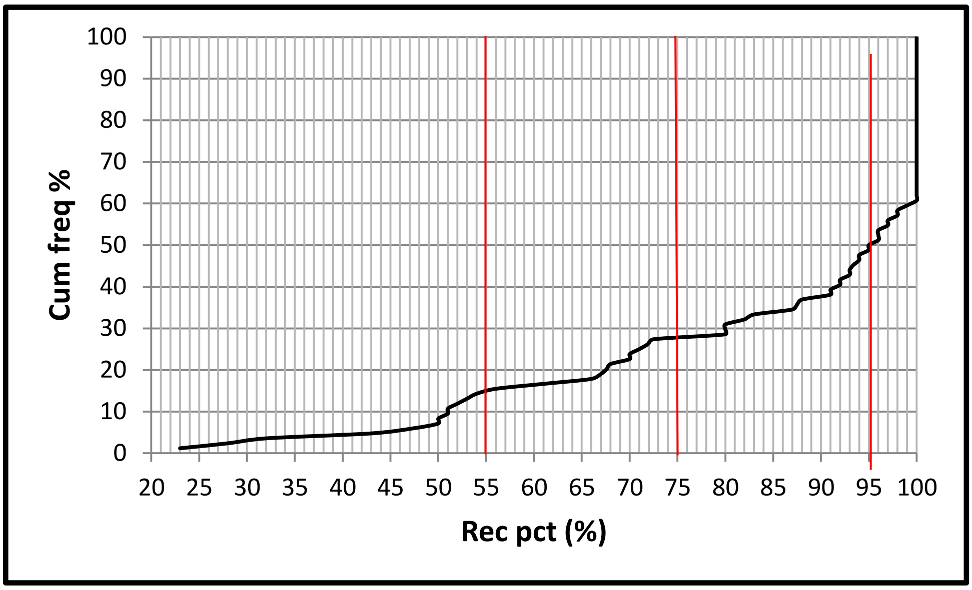


FIGURE 19: DRILL CORE RECOVERY CUMULATIVE FREQUENCY PLOT

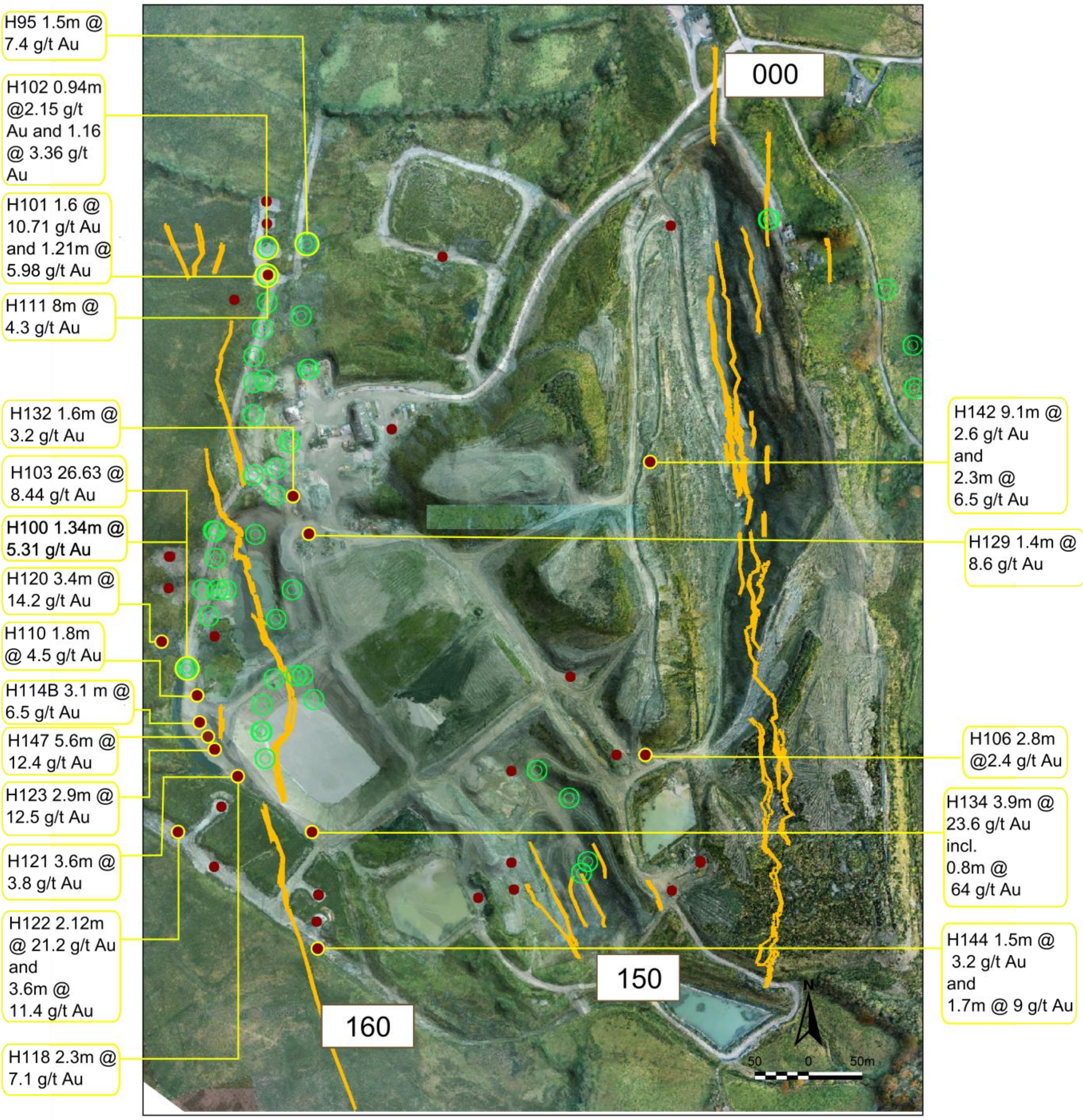
In conclusion, from 2011 to 2014, 51% of cores are shown to have intersect recovery of 95% or over; correlation between recovery and grade is very low showing that recovery is independent of grade. Therefore, Galantas deems the core results as adequate to include in resource estimation.

11.4 DRILLING RESULTS 2011-2014

Significant drill intersections exceeding one metre at 3 g/t Au are listed in Table 19 below and shown in Figure 20:

Hole ID	Location	from (m)	to (m)	Dh length (m)*	Au g/t
OM-DD-11-100	Joshua Vein	87.20	88.54	1.34	5.3
OM-DD-11-101A	Joshua Vein	86.70	88.30	1.60	10.7
And	Joshua Vein	100.00	101.21	1.21	5.98
OM-DD-11-102	Joshua Vein	79.81	80.75	0.94	2.15
And	Joshua Vein	113.66	114.82	1.16	3.36
OM-DD-11-103	Joshua Vein	166.07	192.70	26.63	8.4
OM-DD-12-106	Kearney Vein	162.5	165.3	2.8	2.4
OM-DD-12-110	Joshua Vein	75.7	77.5	1.8	4.5
OM-DD-12-111	Joshua Vein	119.2	127.2	8	4.3
OM-DD-12-114B	Joshua Vein	98.7	101.8	3.1	6.5
OM-DD-12-118	Joshua Vein	39.3	41.6	2.3	7.1
OM-DD-12-120	Joshua Vein	106.4	109.8	3.4	14.2
OM-DD-12-121	Joshua Vein	53.7	57.3	3.6	3.8
OM-DD-12-122	Joshua vein	133.7	135.82	2.12	21.2
And		142.2	145.83	3.6	11.4
OM-DD-12-123	Joshua Vein	60.8	63.7	2.9	12.5
OM-DD-12-129	Joshua	76.19	77.6	1.4	8.6
OM-DD-12-134	Joshua (south)	64.89	68.8	3.9	23.6
Including				0.8	64
OM-DD-11-95	Joshua Vein	122.4	123.9	1.5	7.4
OM-DD-12-132	Joshua (north)	75.9	77.54	1.6	3.2
OM-DD-12-142	Kearney	137.39	146.45	9.1	2.6
And		172.9	175.17	2.3	6.5
OM-DD-12-144	South Joshua	21.4	22.9	1.5	3.2
And		70.0	71.7	1.7	9
	Since Galantas (2013) cut-off				
OM-DD-13-147	Joshua vein	146	151.6	5.6	12.4

TABLE 19: CAVANACAW DRILL INTERSECTIONS 2012-2014



Kearney, Joshua and Kerr vein trace with recent hole intersects

FIGURE 20: SIGNIFICANT VEIN INTERSECTS ON KEARNEY, JOSHUA AND KERR

*Note that these are down-hole measurements and cited widths have not been converted to ‘true widths’ (corrected according to dip of hole) for the purpose of Table 19 and Figure 20.

11.4.1 JOSHUA DRILLING

Thirty one diamond drill holes, totalling 5,061.4m, were drilled in 2012-2014 on the Joshua vein. The locations for these holes were selected to increase the known depth and strike of the vein. Twenty eight of these were reported on in Galantas (2013), H146-H148 have been added to this current resource update.

Holes OM-DD-11-61, -62 and -58 were targeted to twin historical Riofinex holes OMBHL51, 73 and 84. Analytical results returned much higher grades than those in the corresponding Riofinex holes, which were drilled using a conventional core barrel, and suffered from correspondingly poor core recovery. Assay results from these Riofinex holes were therefore considered insufficiently reliable for resource estimation.

The best intersections identified during the 2012-2014 drilling programme are listed in Table 19. These include an exceptional value of 26.6 metres (4.5 m true width) at 8.44 g/t Au in OM-DD-11-103; and a very good intersect on south Joshua measuring 3.9 m (2.4 m true width) and grading at 23.6 g/t Au, including 0.8 m at 64 g/t Au, in OM-DD-12-134. A recent hole, targeting the Joshua vein, yielded a 5.6 m intersect (2.8 m true width) wide intersect grading 12.4 g/t Au. Figure 21 (below) shows a section at 370700N located on south Joshua, displaying three intersects whose grade is shown by circles and outlines the interpreted wireframe.

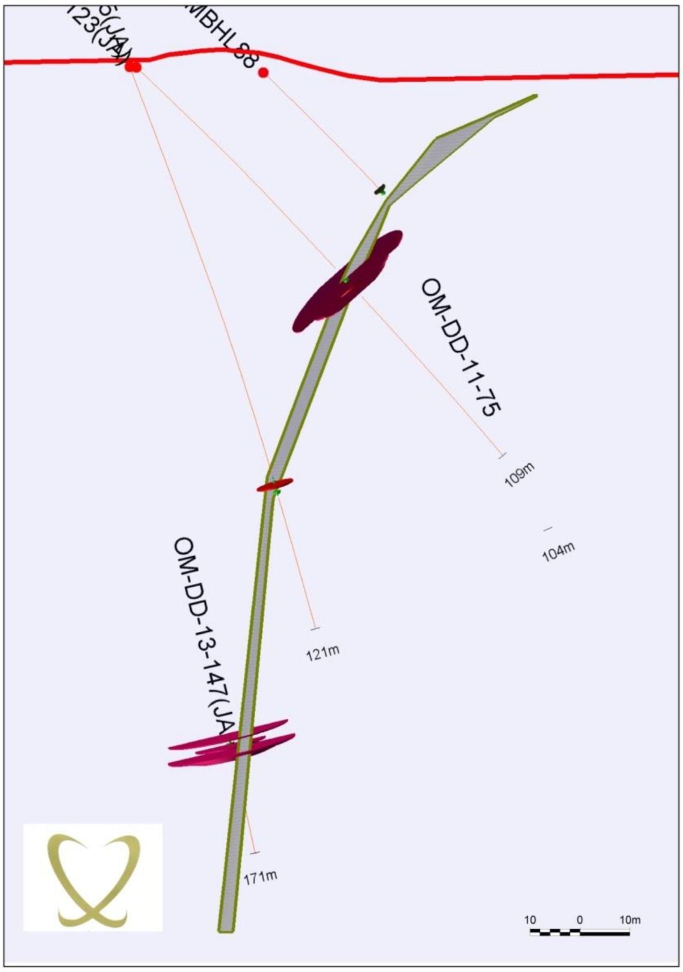


FIGURE 21: JOSHUA DRILL SECTION 3700700N

11.4.2 KEARNEY DRILLING

The objectives of the deep drilling programme on the Kearney vein remained unchanged from the previous drilling programme conducted during 2011 & 2012. As reported in the ACA Howe 2012 report and Galantas 2013, these were:

- *to upgrade the Inferred resources of 2012 for the Kearney Vein, above 0m elevation, to Indicated category (down to approximately150m below surface)*
- *to identify additional Inferred resources below 0m elevation and below the Inferred resources of 2008, down to the minus 160m elevation (down to 310m below surface).*

Drill hole OM-DD-12-142 yielded the best results during the latest exploration phase with an intersect of 9.1 m (4.9 m true width) grading 2.6 g/t Au, and 1.2 m (true width) at 6.5 g/t Au. This intersect is displayed in Figure 22 (below).

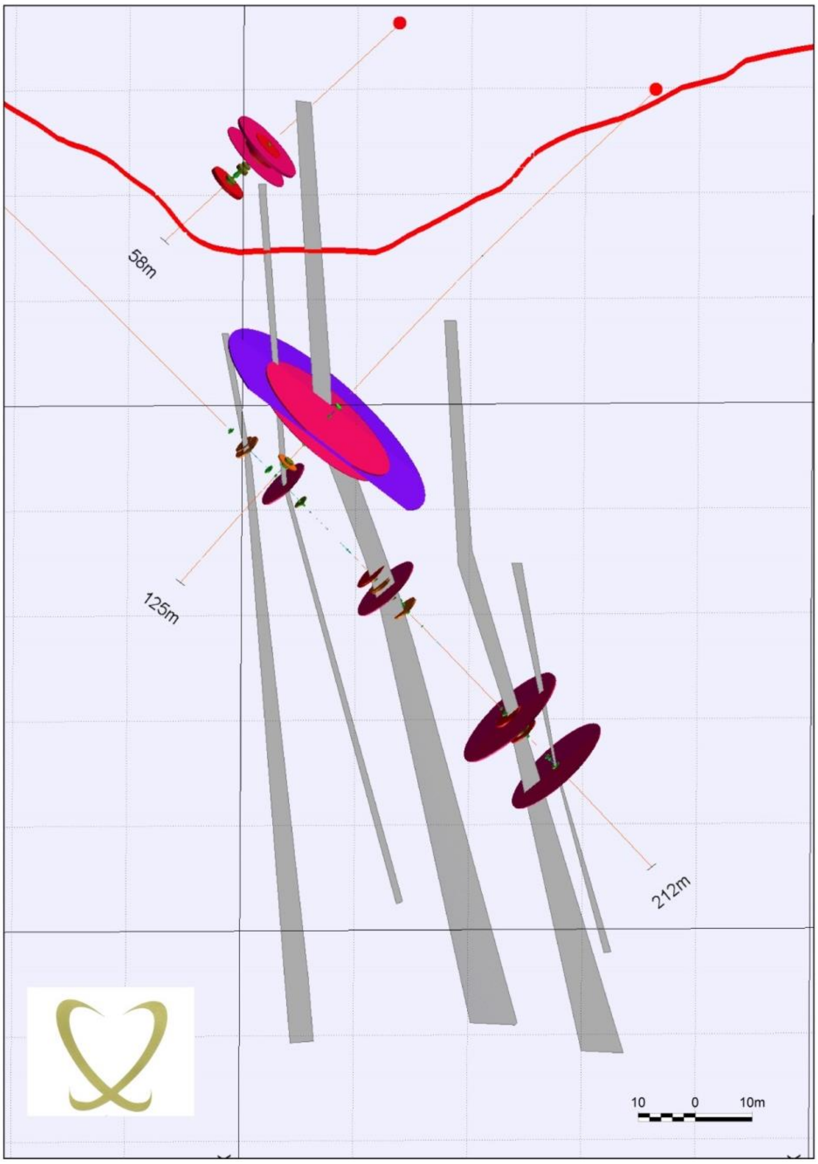


FIGURE 22: SECTION THROUGH THE KEARNEY VEIN LOOKING NORTH AT 370900 N

11.4.3 KERR DRILLING

Four holes were drilled on Kerr totalling 923 metres (Galantas, 2013). The aim of these holes was to improve understanding of the multi vein fan structure and to develop a potential resource.

The best of these intersected on drill hole OM-DD-12-127 at 4.6 g/t Au over 0.7 metres (true width).

12 SAMPLE PREPARATION, ANALYSES AND SECURITY

Geological due diligence requires that the quality of the assay data is controlled and analysed. In the mining industry this is known as quality assurance and quality control (QA/QC) and involves the minimizing of sampling error and the systematic monitoring of the samples accuracy and precision (i.e. repeatability). The more uneven the mix of metals (heterogeneity) in the sampled material, the more difficult it is to obtain a representative sample from which infer the nature and characteristics of the whole geological object.

The following was reported in the 2012 report by ACA Howe and is a fair and accurate description of the sampling procedure.

Exploration channel sampling activities were undertaken at the same time as drilling activities. All sampling and sample preparation was undertaken by Galantas personnel, with the exception of the Kearney pit channel sampling which was undertaken in 2006 by ACA Howe. Channel samples were collected and sealed in plastic bags along with a sample ticket displaying the sample number. The sample number for each sample was also written on the bag and entered in to the sample submission sheet. Channel samples weighing 0.5-2.0kg were collected at 10cm intervals from logged intervals of 5cm or 10cm wide sawn channels and sealed in plastic bags along with a sample ticket. Each sample number was then added to the sample submission sheet.

Channel samples were laid out in sequence order at the core shed and checked prior to being added to sample batches of drill core and sent to the laboratory for analysis together with batches of drill core. Channel locations were surveyed by site personnel during sampling and survey data were merged with the Micromine exploration database.

Drill core from the 2006-2007 and 2011-2012 programmes were selectively sampled with sample intervals based on mineralisation and lithology. Samples of wallrock were taken one metre either side of the vein to allow for appropriate dilution to achieve minimum mining width.

Once samples were chosen, sample intervals were written on the core prior to core cutting. The core was then orientated along the core axis and sawn in half with a circular diamond bench saw before both halves of core were replaced in the box in the original orientation. When cutting was complete, each interval was sampled with attention given to ensuring that an exact half of core was sampled for each interval and that no contamination had occurred.

Core samples were then placed in clear, sturdy sample bags and a sample number ticket inserted. The sample number was also marked directly on to the bag and a second ticket sealed within the bag opening to ensure correct identification at the laboratory. Once bagged, all samples were laid out on the floor in numerical order, checked, and bagged up in larger bags ready for dispatch. Core was *dispatched via courier or by Galantas personnel to OMAC laboratories, Galway, Republic of Ireland along with two inventories of submitted samples.*

Analysis of all samples generated from channels and drill core were undertaken by OMAC Laboratories (OMAC) of Loughrea, Co Galway, Ireland. OMAC joined the Alex Stewart group in 1999 and operated as its principal exploration laboratory until 2011 when Alex Stewart was taken over by ALS Minerals. OMAC is accredited to ISO 17025 by the Irish Accreditation Board (INAB). This standard relates to competency requirements for testing and calibration laboratories. INAB is a member of the International Accreditation Corporation (ILAC) and a signatory to the ILAC Mutual Recognition Arrangement whose signatories include Canada, Australia, South Africa and many countries within the EU.

OMAC participates in proficiency testing programmes and round robin programmes run in the mineral analysis sector twice a year, run by Geostats of Perth, Western Australia and CANMET, Canada. Geostats run a twice yearly round robin and in excess of 100 laboratories participate and their performance is circulated to sponsoring mining houses. The Proficiency Testing Programme for Mineral Analysis Laboratories (PTP-MAL) has been set up under the Canadian Certified Reference Materials Project (CCRMP) run by CANMET and OMAC has been involved with this programme since its inception and has received a maximum rating each year.

Samples are analysed for gold via Fire Assay and for a suite of 19 base metals (including lead, arsenic, copper and zinc) via ICP . Sample preparation for both analyses comprises drying of samples, jaw crushing to <2mm, riffle splitting of a 1kg sub-sample followed by homogenisation and pulverisation to 100µ according to ALS procedure codes P1 and P5. This sample preparation method is recommended by OMAC for gold bearing samples. All fractions during the sample preparation stage are retained for reference or QA/QC activities.

The new analytical facility at ALS (formerly OMAC) laboratory in Loughrea, County Galway was visited by Galantas personnel involved in the reporting of this document. At the laboratory, each stage of sample progression was examined and discussed with senior ALS personnel.

Each sample is barcoded upon entering the lab and tracked with the code as it is passed through the system to provide traceability. The preparation methods and protocols are based upon Gy's sampling theory. This is to reduce fundamental error of a representative sample. Homogeneity is achieved according to the interest of the sample.

Gold fire assay uses a 30g crushed, split and pulverized sub sample fused with a lead oxide/ carbonate/ borax/ silica/ flux at 1,100 °C using silver as a carrier. Fusion producing lead buttons less than 30 grams are rejected. Once de-slagging is achieved the buttons are cupellated at 950°C. Prills are parted and then dissolved in Aqua Regia. The reading is done by flame atomic absorption down to 0.01 ppm with a Varian Spectre AA 55 instrument. This procedure follows the ALS code Au-AA25.

Analysis of base metals following ALS procedure ME-ICPORE is now termed ICP-AES. A minimum sample weight of 5 g is subjected to acid digestion, a highly oxidising reaction with HNO3, KCLO4 and HBr with the final solution in dilute aqua regia and tested for 19 elements by ICP-AES. Elements tested are: Ag (to 5ppm), As, Bi, Ca, Cd, Co, Cu, Fe, Hg, Mg, Mn, Mo, Ni, p, Pb, S, Sb, Ti and Zn.

Based upon the inspection by OML personnel the author is satisfied that the procedures in sample preparation, fire assay, acid digestion, the machines used in final geochemical analysis and analytical procedures, conform to industry best practices and so are suitable for reporting under NI 43-101.

12.1 QUALITY ASSURANCE AND QUALITY CONTROL

Galantas has developed a robust and extensive QA/QC procedure covering sampling gathering, fundamental sampling errors, removal of heterogeneity, sample preparation and bar coding, analytical methods and finalised reporting. In house standard reference samples and blank samples are used in every batch of samples at a rate of 5%, or one in every 20 samples, and duplicates are included.

External controls, in place with ALS Minerals, monitor: sample size, sample preparation, analytical QC, method run numbers, QC data and sequence orders. An audit trail is available through the ALS webtrieve service and this provides a chain of custody of each sample as it passes through its lab.

12.1.1 OMAC INTERNAL LABORATORY QA/QC

Sample preparation QA/QC is carried out to assess the crushing and grinding. Sample preparation requires >70% of the crushed sample passes through a 2mm screen, 250g split pulverised to >85% sample passed through a 75-micron screen dry.

Blank samples are added at a rate of one in every ten samples at the start of the batch analysis. Standards are added at a random point and duplicates are added at the end of a batch. Batches are no more than 50 samples at any one time.

Final reporting is sent to Galantas in excel format along with an ALS digital certificate of analysis report which is then used to compile the final QC data.

12.1.1.1 OMAC STANDARD SAMPLES (2012-2014)

OMAC Certified Reference Material (CRM) totalled 58 as reported by OMAC within the latest drilling programme of 2012-2014. The results, shown in Table 20 and Figure 23 (below), demonstrate an excellent degree of precision on the values returned with the recommended CRM values. The OMAC standard assays display an average +0.88 percentage difference to the recommended values. None returned assays outside of the +/-10% CRM tolerance limits.

Geostats Pty CRM number	number of assays	mean Au, g/t	Standard deviation, Au g/t	CRM recommended value	% difference
Standard OxD87	1	0.41	0.00	0.417 +/- 0 .004 ppm	-1.68
Standard OxE101	2	0.60	0.00	0.607+/-.0005 ppm	-1.15
Standard OxF100	4	0.81	0.02	0.804+/-.006 ppm	0.75

Standard OxG84	2	0.92	0.01	0.922 +/- 0.010 ppm	-0.22
Standard OxK95	3	3.55	0.05	3.537 +/- 0.040 ppm	0.37
Standard OxP91	2	14.7	0.14	14.82+/- 0.10 ppm	-0.81
Standard G306-3	4	8.76	0.11	8.66 +/- 0.047 ppm	1.15
Standard G312-6	10	2.465	0.04	2.42 +/- 0.015 ppm	1.85
Standard G910-3	2	4.06	0.06	4.02 +/- .0.029 ppm	1.00
Standard G904-1	7	12.77	0.368	12.66 +/- 0.078 ppm	0.86
Standard MG-12	9	0.89	0.02	0.886 +/- 0.006 ppm	0.45
Standard OxL93	4	5.75	0.07	5.841 +/- 0.053 ppm	-1.56
Standard OxJ111	2	2.15	0	2.166 +/- 0.023	-0.74
Standard GLG904-4	1	0.20	0.00	204.08 +/- 6.07 ppb	-2
Standard GLG304-1	5	0.16	0.01	151.64 +/- 5.04 ppb	5.51

TABLE 20: OMAC LABORATORIES ANALYSIS OF CERTIFIED REFERENCE MATERIAL DRILLING PROGRAMME, 2012-2014

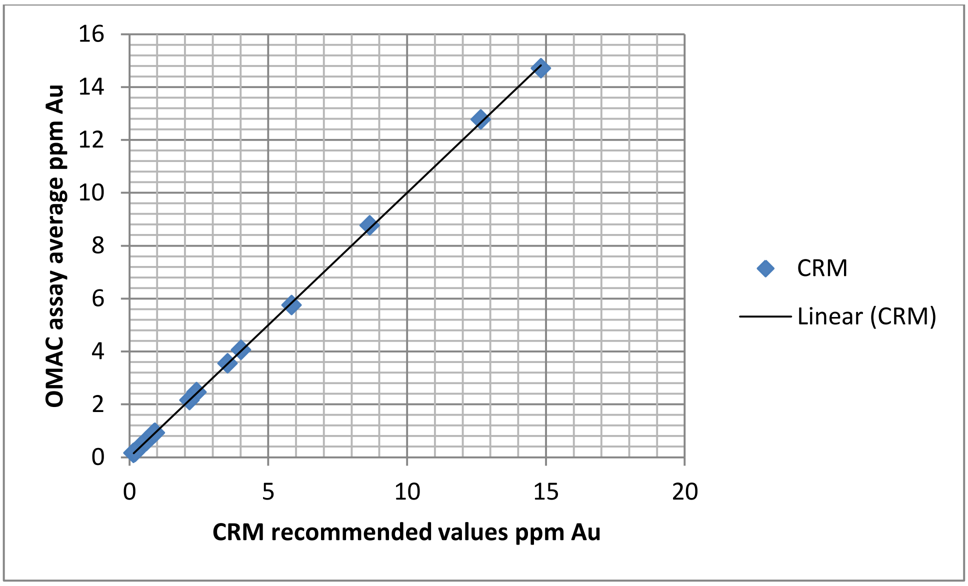


FIGURE 23: OMAC ASSAYS OF CRM V RECOMMENDED VALUE

12.1.1.2 OMAC BLANK DATA

ALS laboratories used a total of 89 blanks in its base and precious metal analysis as part of the internal QC procedure. All of these blanks recorded values of 0.01 ppm Au or less. The results indicate that there is no issue that would affect sample cross contamination in its sample preparation and analysis procedure.

12.1.1.3 REPEAT DATA

ALS used a total of 75 duplicates assays as part of their internal QA/QC procedure. Data from the repeats shows good correlation in the lower and middle grade ranges (Figure 24). As the order of magnitude increases, so does the variance. This is still within an acceptable limit and the overall correlation coefficient between repeat assays and original assays is 0.994.

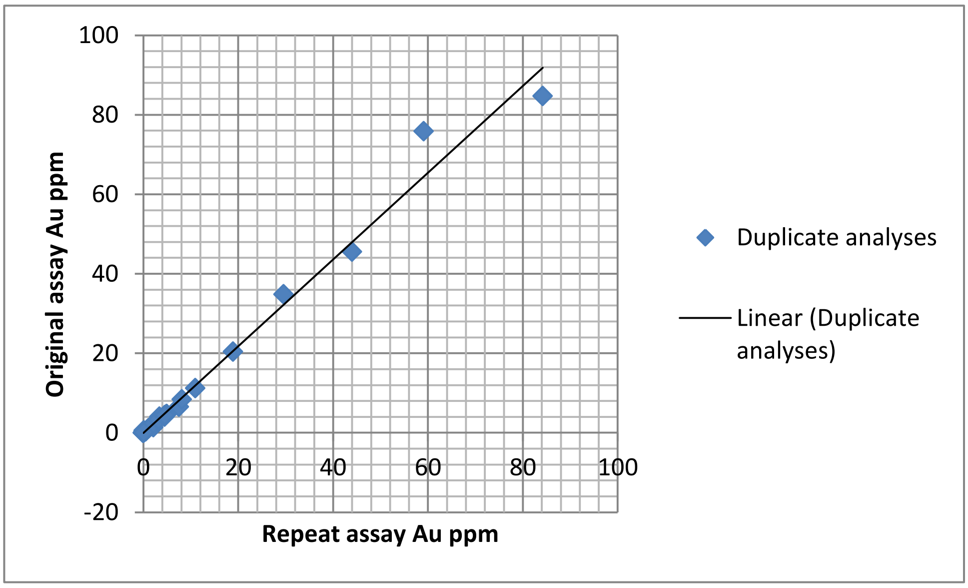


FIGURE 24: SCATTER PLOT – OMAC INTERNAL REPEAT ASSAYS

12.2 GALANTAS QA/QC

12.2.1 GALANTAS STANDARD SAMPLES, 2012-2014

As correctly detailed in ACA Howe’s 2012 report: *Galantas submitted four large bags of mineralised material from the Kearney pit to OMAC for the creation of standard reference material in 2011. The material was pulverised in its entirety according to OMAC’s procedure P5. The resulting pulp was then assayed eight times by fire assay with atomic absorption finish, according to OMAC’s then code Au4.*

Each sample (A, B, C and D) was then returned to Galantas to be used as sample reference material. The average assay values for each sample, and the value to which subsequent assays were compared, was 84.32 ppm, 8.20ppm, 5.51 ppm and 2.57 ppm Au for samples A,B,C and D respectively.

Galantas then inserted the resulting standard reference material at regular intervals into the sample stream every twenty samples during drilling and trenching.

For details of the standard samples in the 2011-2012 drilling phase please refer to the ACA Howe report of 2012. The results of this showed only reasonable repeatability in contrast with the standards from OMAC’s reporting which had a very good correlation. It was thought that transport caused gravitational settlement, although the reason for the discrepancy remains unknown.

Figures 25 to 28 show the reporting of a total of 58 standard samples inserted by Galantas as part of its internal QA/QC procedure.

Figure 25 shows standard reference material A has a mean of 80.17ppm Au with a standard deviation of 3.26 and a standard value of 84.32 ppm Au. A slight tendency to under report is shown with all but one reporting at or below the standard value.

Figure 26 shows standard reference material B which has a mean of 7.92 ppm Au and a standard deviation of 1.4 with a standard value of 8.2 ppm Au. The graph shows an even spread around the standard value line with 3 outliers on both sides of the standard value line.

Figure 27 shows standard reference material C which has a mean of 5.31 ppm Au, with a standard deviation of 0.71 and a standard value of 5.51. The plot shows a bias to below the standard value line with one outlier to the top of the graph.

Figure 28 shows standard reference material D which has a mean of 2.725, standard deviation of 0.27 and a standard value of 2.57 ppm Au. The values about the standard value are accurate but have a number of outliers to the high side of the standard value line displayed.

Collectively the graphs show a tendency to under report in the case of Galantas’ standard material. The cause of this is not known but could be a function of the style of mineralisation and heterogeneity in the material from which the standard values were derived.

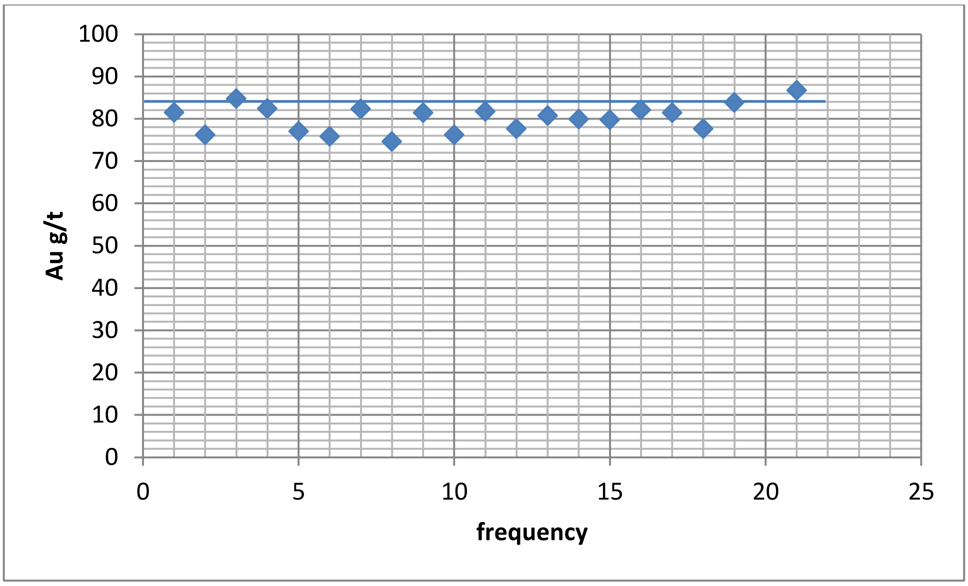


FIGURE 25: ASSAYS OF GALANTAS STANDARD A 84.32 PPM, 2012-2013

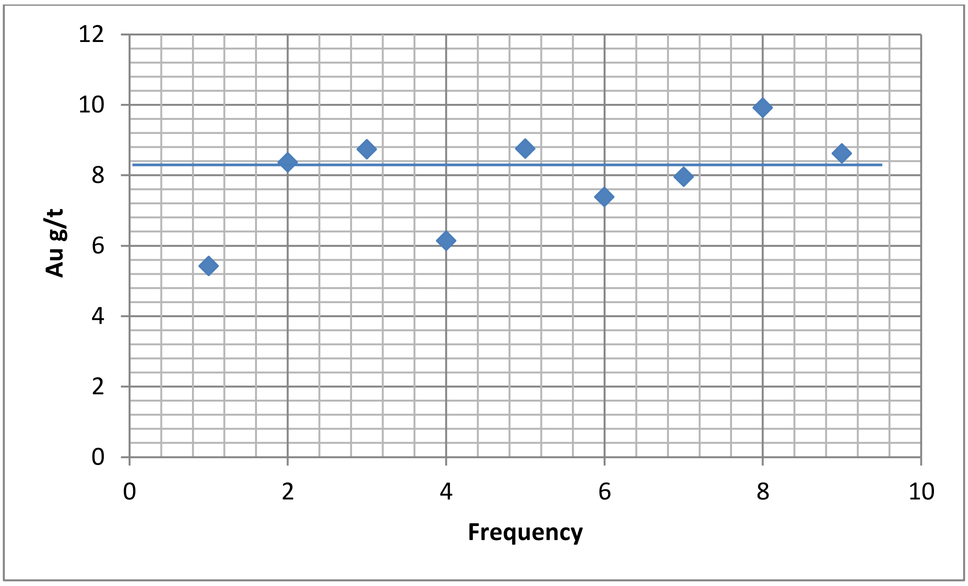


FIGURE 26: ASSAYS OF GALANTAS STANDARD B 8.20PPM, 2012-2013

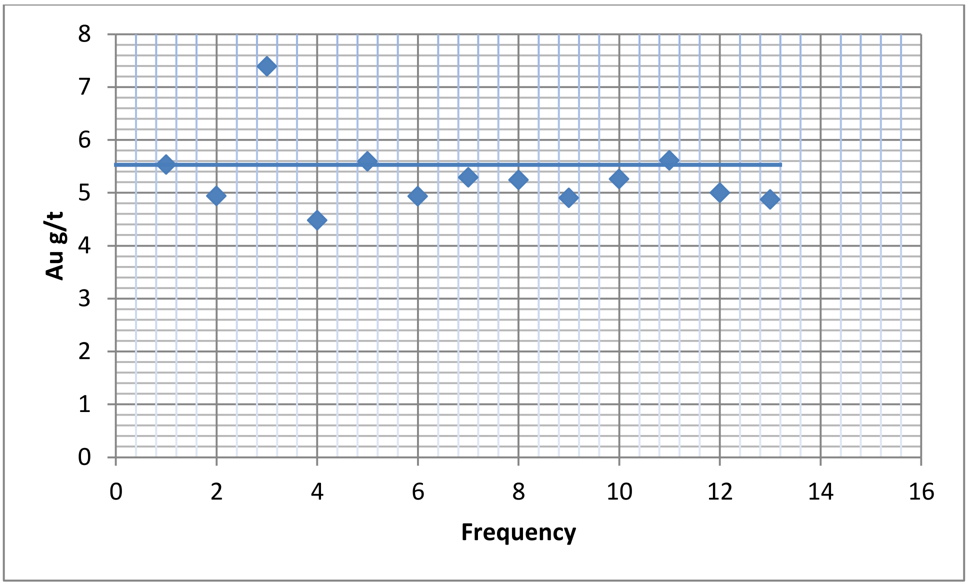


FIGURE 27: ASSAYS OF GALANTAS STANDARD C 5.51 PPM, 2012-2013

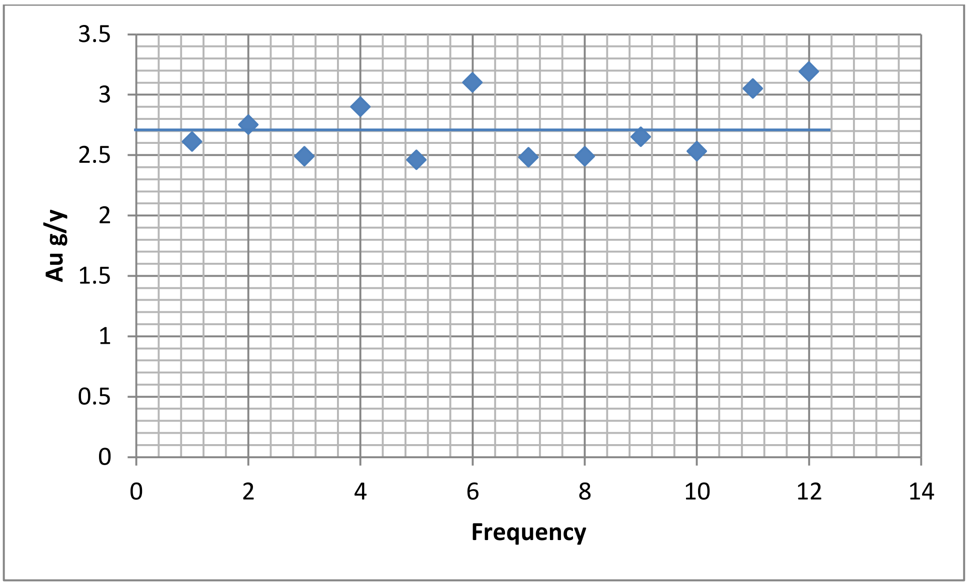


FIGURE 28: ASSAYS OF GALANTAS STANDARD D, 2.57 PPM 2012-2013

12.2.1.1 GALANTAS STANDARD SAMPLES, 2013-2014

In 2013, two new standards, derived from OML concentrate, were created by ALS using the method in Section 12.2.1. One of the new standards, standard 1, was incorporated twice into the most recent core dispatch. The mean and standard deviation values for standard 1 are 4.28 ppm and 0.18, respectively. Two sub-samples of standard 1, included in the H147 and H148 core batch, returned values of 4.4 ppm, only 2.7% above the mean (Figure 29). This is comparable with the results of older OML standard reference material (standards A-D). The data generated for this batch of core are deemed to be accurate.

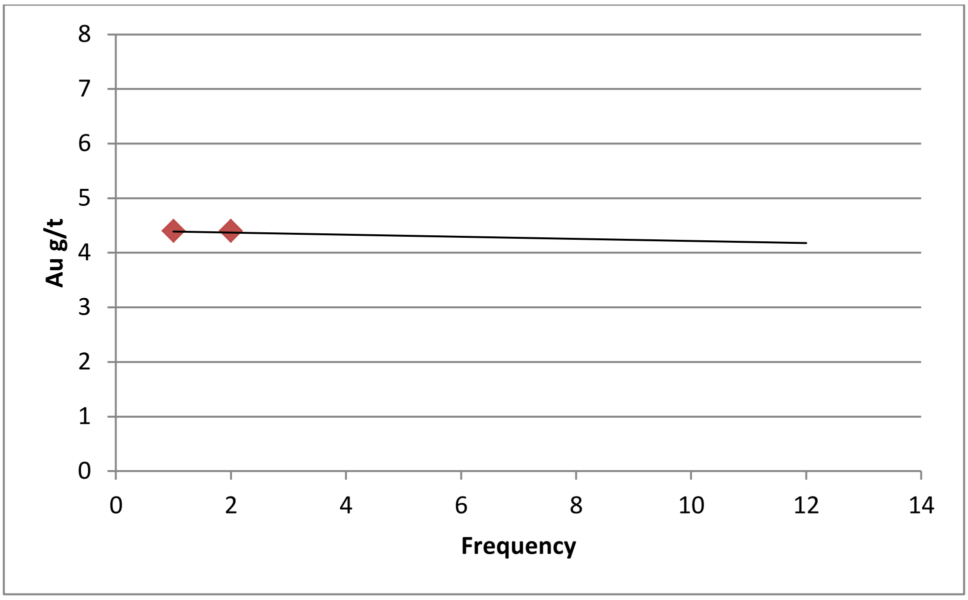


FIGURE 29: GALANTAS NEW STANDARD 1, 4.28 PPM, 2013-2014

12.2.2 PULP RE ASSAY SAMPLES

A total of 10 pulp samples were re-analysed by OMAC laboratory and results were compared to the original samples. This was to establish the accuracy, or repeatability, of the sampling procedure. The average difference as a percentage is 5.51% and the results are plotted in Figure 30 below. The average figure is affected by one outlier with a percentage difference of 20%, this is within an order of magnitude that is reasonable given the Au grade of the original is 8.66 g/t and the repeated sample gives 6.86 g/t.

Assay pairs show moderate repeatability and therefore good accuracy, without bias and within acceptable limits. The sample precision error described above may be a result of the heterogenic nature of the sampling material and the nugget effect present within the sample due to the style of mineralisation.

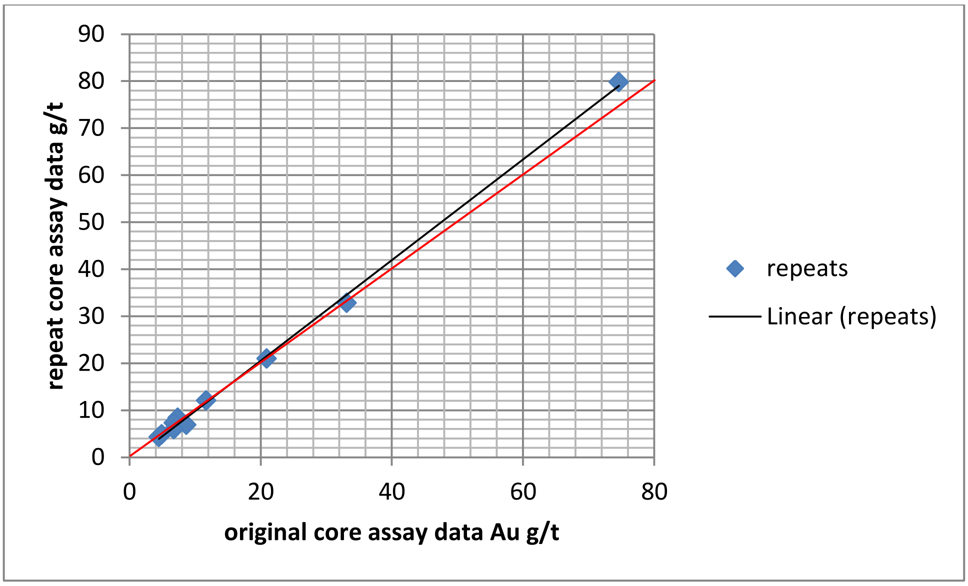


FIGURE 30: SCATTER PLOT OF RE-ASSAYED SAMPLES

Repeated samples requested by Galantas show a very good correlation, with a correlation coefficient of 0.999.

12.2.3 BLANK SAMPLES

In the 2012-2014 drilling programme a total of 58 internal blank samples were included in sample batches submitted to OMAC/ ALS. Out of these only 5 registered over 0.01 ppm Au, or 8.6%, of which the highest was 0.09 ppm Au. Blank material was derived from builders sand, un-mineralised drill core and un-mineralised rock from the rock stock pile.

12.2.4 GALANTAS DUPLICATES 2012-2014

Between 2012 and 2014, Galantas sent a total of 58 duplicate samples to ALS for analysis. The duplicate material was derived from quarter core. A comparison of duplicate and original sample Au values is plotted in Figure 31. Despite the heterogeneity of vein intersects, visible in the core samples, analysis of original and duplicates resulted correlation coefficient of 0.93.

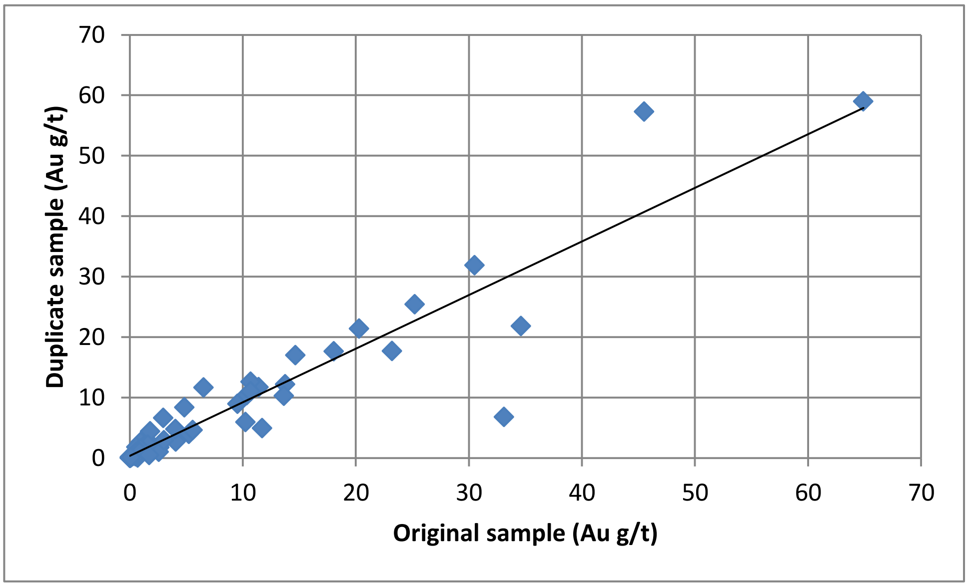


FIGURE 31: SCATTER PLOT OF GALANTAS ORIGINAL AND DUPLICATE SAMPLE ANALYSES

12.3 QA/QC CONCLUSIONS

Past reporting by ACA Howe has identified the robust QAQC procedures in place, developed by Galantas and ACA Howe. The quality of sample assays has been ensured through the monitoring of sample preparation, accuracy, precision and bias via the use of external controls (standards, blanks and repeats), and the review of internal laboratory controls (internal standards, blanks and repeats). The results of this work indicate that sample preparation techniques used at ALS laboratories are adequate to eliminate signs of cross contamination. Analytical machines used in the reading of element abundance display a high level of accuracy and precision, without bias. It is therefore concluded that the assay results from the drilling programme of 2012-2014 are representative of the samples collected.

However, some sampling bias may exist as shown in the Galantas standard results that indicate a tendency to under report, as shown in Figures 25 and 27.

13 DATA VERIFICATION

Galantas' own personnel and identified third parties gathered and compiled the information used in this report during the latest exploration phase.

In ACA Howe's 2012 report the following was documented:

The authors carried out checks during site visits and confirmed best practice logging and processing were being implemented, witnessed core cutting and sampling, verified channel sampling locations and reviewed internal reports.

It is the view of the author that industry best practice, as well as development, is continuing in the areas of exploration and geological understanding of the deposit.

During recent exploration, geological data, survey data, assay data, and bulk density data from drilling and channel sampling activities were merged into the current Micromine database for the project. Regular checks were performed on the database to ensure there were no errors in data entry. The receipt of raw data from the laboratory and third parties, usually in the form of excel spread sheets, allowed direct import in to Micromine, thus minimising any potential error arising from manual data entry.

The data supplied to Galantas by ALS and third parties appear reliable in the light of checks carried out by Galantas and the review of QA/QC practices.

In view of these checks, and the body of work previously submitted in the 2008 and 2012 reports, the author is of the opinion that the data cited in this report are reliable and adequate for use in the resource estimate.

14 MINERAL PROCESSING AND METALLURGICAL TESTING

Initial froth flotation test-work was carried out by Lakefield (1992). The plant that currently operates is based upon the results of that test-work. Further tests on tailored reagents have been carried out during operation of the plant, and improvements made. The plant is monitored for efficiency and environmental control by an on-site laboratory.

An up-rated version of the existing plant has been designed in order to process ore from the proposed underground mine, as discussed in Section 17 of this report. The author considers that no additional test-work is required.

15 MINERAL RESOURCE ESTIMATES

15.1 RESOURCE ESTIMATION OVERVIEW

Galantas has prepared an updated estimate of mineral resources for the main Kearney vein zone and Joshua vein, which have been the focus of historical and recent exploration. Other targets included IP anomalies within the mine site, McCoombs vein and Kerr vein, the latter of which is interpreted as a fan structure between two principal shear hosted vein structures. Historical targets include Sammy's vein, Elkins vein, Gormley vein, Garry's vein and Princes vein.

Resource estimation methodologies, results, validations and comparisons with previous estimates are discussed in this section of the report.

As part of this work, an updated project database was created, validated and used to visualise exploration and resource data during interpretation and modelling prior to estimation. Mineralised zones were interpreted and 3D wireframes created. Sample data were selected and statistical analysis performed on raw sample data to assess its validity for use in resource estimation. Following the generation of mineralised domains, raw sample data were composited in order to standardise sample support. Further statistical and geostatistical analyses were performed on composite data to assess grade characteristics and continuity.

Once the orientation and ranges of grade continuity were chosen, wireframe constrained block models were created and grade interpolation into each block model was undertaken using the inverse distance weighting algorithm. Upon completion of block estimation, the resulting block models were validated and density values written to the block model prior to reporting (PERC) */CIM compliant grade and tonnage estimates for the project.*

15.2 SOFTWARE

Updated resources for the Omagh Project were estimated using MICROMINE version 14.0.4.

15.3 DATABASE COMPILATION

The following is extracted from the 2012 ACA Howe report.

Prior to recent exploration activities, historical Riofinex drilling data for the project were captured in hard copy form and surface data captured and stored in DynoCADD software on site. In 2003, ACA

Howe compiled AutoCAD and GIS data for the project, including surface geochemical data, float sample data, stream sediment sample data, Landsat imagery interpretation data and limited surface mapping. This exploration data is discussed in detail in the ACA Howe 2003 and 2008 reports.

Prior to the partial re-estimation of resources over the Kearney vein in 2004, a MICROMINE resource development database was created which contained collar, survey, geology and assay data from historical Riofinex drilling data and pit channel sampling data for Kearney only. This database was validated by ACA Howe and became the master database into which was merged all other available Riofinex drilling data and recent drilling and channel sampling data collected by Galantas and ACA Howe. Geological data for a total of 28 historical reverse circulation Riofinex drill holes are missing from the drill logs, no reliable survey information for these holes is available and assay results from these holes cannot be verified. Therefore, these holes are absent from the database.

Between 2006 and 2007, ACA Howe received all available hard copy drill logs for historical Riofinex drilling on other veins, and manually entered geological, assay and survey data for each hole in to Excel spreadsheets prior to merging this data into the master database. Co-ordinate data for these holes were not included in the original drill hole logs and so were taken from ACA Howe GIS data held for the project, where co-ordinate data were originally digitised from the 1:2,500 maps depicting hole locations contained on the drill logs. Elevation data were taken from the OSNI digital topographic data obtained by ACA Howe in 2007.

The information above is considered a reasonable description of the dataset gathering procedure pre 2011.

During the drilling programme of 2011-2012 collar, assay, survey and geological data were recorded in the Micromine on-going drilling database. The information was then transferred to ACA Howe for the benefit of the 2012 report. ACA Howe employed their own validation checks on the data before incorporating it into a master project used for the compilation of the 2012 report.

The 2012-2014 drill programme saw all collar, survey, assay and geological data added to the excel master sheet of each dataset as raw data, before being processed into correct formats to allow transfer onto the Micromine master project database. This database was a merging of the validated master project from ACA Howe used in the reporting of the 2012 resource update and the Galantas 2012-2013 drill programme project.

After the cut-off date, the parent master project database was once again validated, using the validation tools within Micromine. This process eliminated incorrect values from the files and enhanced the consistency of the data. The resulting drill hole database contains all available historical drilling and sampling data for the entire project and is considered without apparent error and so robust for use in resource estimation.

Table 21 (below) summarises the data contained with the Omagh project database, used in resource estimation. Appendix 2 contains details of all the drill holes completed and Appendix 3 provides an overview of grade and vein width frequencies for Kearney, Joshua and Kerr.

TABLE 21: DATA USED IN RESOURCE ESTIMATION	
Data	**Number of Records**
Drill holes	302
Drill Hole Surveys	2,219
Drill Hole Assays (gold)	4,579
Drill Hole Geology	3,111
Channels	540
Channel Surveys	990
Channel Assays (gold)	7,371
Channel Geology	242

TABLE 21: DATA USED IN RESOURCE ESTIMATION

15.4 DATABASE VALIDATION

Once updating of the files was complete for the latest drilling programme, the entire drill hole database, which contains the channel data, was subject to validation checks. The validation function detects errors and inconsistencies in the collar, down hole survey, assay and geology files. The relationships between relevant fields and files was checked and any errors were reported. The following is a list of typical errors:

- FROM <previous TO
- FROM>=TO
- FROM or TO missing
- Collar missing or incorrect
- Record beyond total depth
- Hole excluded by collar filter
- Duplicate hole
- Non-consecutive surveys
- Duplicate collar entry
- Dips or Azimuths change by more than x
- Surveys beyond total hole depth
- Missing hole in interval file
- Compulsory field is blank

- Total depth missing
- Rate of Deviation
- Sample length intervals

  (Micromine training manual version 2010(12.0))

Errors encountered were rectified by referring to the original source data.

15.5 COLLAR LOCATIONS

Collar co-ordinate adjustments were carried out by Howe 2012, with regards the historical Rio drill holes, and is deemed thorough and robust. The corrections have been carried over in this report.

15.6 INTERPRETATION AND MODELLING

The following is taken from the Howe 2012 report:

Once the resource database was validated, all drill hole data and channel sampling data were viewed interactively in MICROMINE software 2D and 3D environments to aid in the interpretation of geology and mineralised zones in each area. Prior to conducting computerised interpretation of mineralised zones, the following were reviewed:

- *Regional geological setting*
- *Known geological controls on mineralisation*
- *General continuity of mineralisation*
- *Variability of assay grade within sampled veins*
- *Topographic and pit DTM Data*

The limits of mining within the Kearney pit have been recorded in four separate DTM surveys of the area. The southern end of the pit is contained in DTM _PRE-2010 SURVEY(SOUTH KEARNEY) file. This survey correlates to the DTM_OGL survey detailed in the ACA Howe report 2012 and has since been back filled. Another survey in 2010 (DTM COASTWAY_ SURVEY_2010) is also in the south end of the pit and extends the limit of the excavation in the area. The limit of maximum excavation in the main pit area was surveyed in May 2012 and contained in the file DTM_ALL_SURFS (MAY-2012). The later north extension and main pit back fill was surveyed in September 2012 which is recorded in DTM COASTWAY 2012. A total of four surveys were used to confine maximum pit extent in the north, south and maximum depth of the Kearney pit. A section at 370860N showing the pit level is shown in Figure 32.

This resource update has used all four surveys in the measurement of depletion by mining for the Kearney pit.

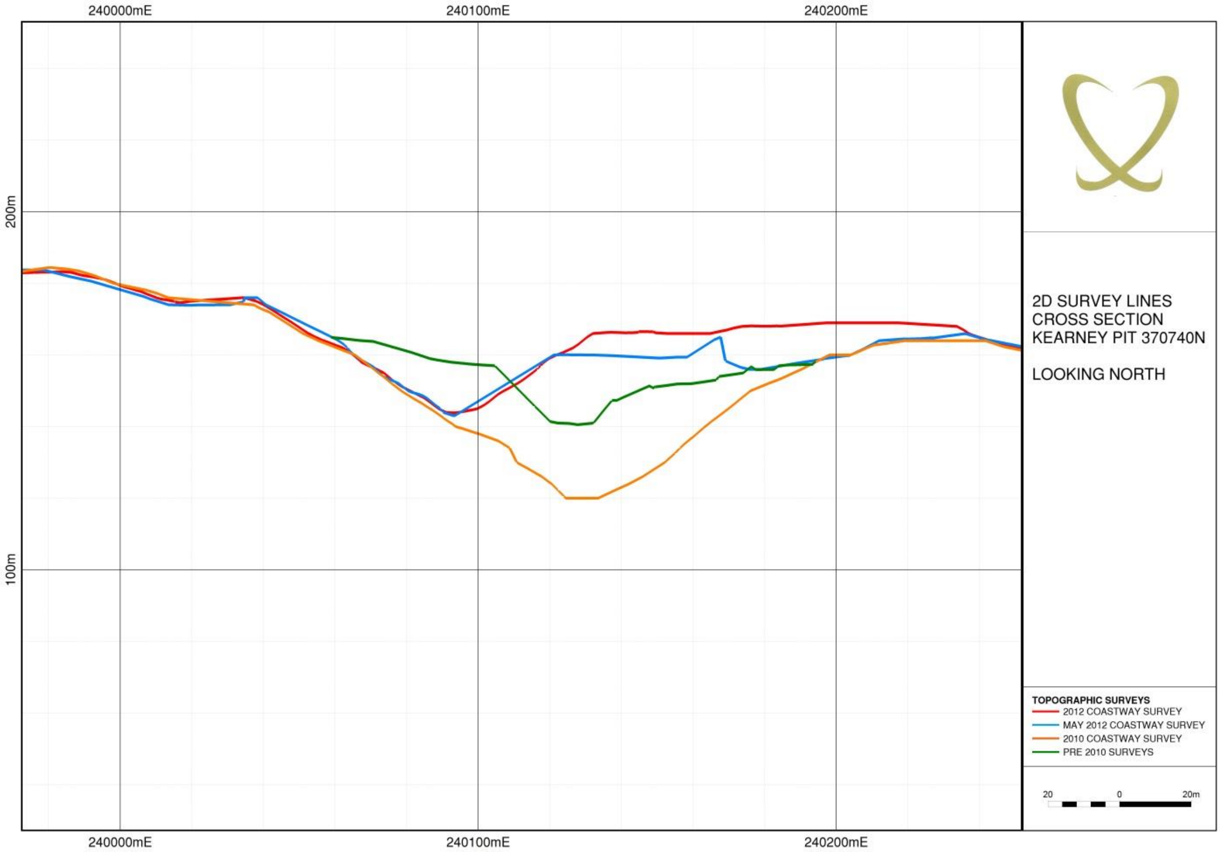


FIGURE 32: KEARNEY PIT PROFILES

The latest contracted survey (COASTWAY 2012), the Omagh Minerals in-house surveys completed in preparation of tailing cells on the Joshua vein, and the levels of the channelling sample programmes, have contributed toward determining the levels of the Joshua and Kerr excavations.

In 2007, ACA Howe acquired regional digital topographic data from the OSNI which covered the eastern half of licence OM 1/03 at 10m contours and merged this data with the local pit surveys, and created a surface DTM in MICROMINE which was used to constrain mineralised zones. In addition, outlying Riofinex holes were draped on to this surface to capture elevations for these holes.

15.6.1 INTERPRETATION OF MINERALISED ZONES

The process and strategies used in the ACA Howe report of 2012 were deemed of sound reporting standard and appropriate for the deposit type. Given the short time period that has elapsed between the ACA Howe 2012 report, the 2013 update and this report, the method of interpretation remains the same.

Mineralised zones at each of the named veins were interpreted in 2D and 3D in MICROMINE by generating vertical cross sections perpendicular to the interpreted strike of mineralisation. On screen, these cross sections depicted collar traces annotated with logged geological and sampled assay intervals (including down hole graphs of Ag, As, Pb and Zn data), historical and recent channel sampling data and slices through the topographic surface.

Only potentially mineralised core was sampled during recent drilling activities. Selective sampling of veins and clay gouge essentially defines the mineralised zones in each hole and these assisted with the sectional interpretation of the veins.

Section control files were developed for Kearney and Joshua to limit each section view from east to west. Strings were created in each section window, constraining zones of mineralisation exhibiting more than, or equal to, 2 g/t Au over a drilled length of more than, or equal to 0.9m. In short sections the width may be reduced to 0.75 m. The criteria used to define mineralised zones were generally adopted. However, in areas where gold assays greater than 2 g/t Au were returned from drill hole intervals less than 0.9m, these were included and bulked out to a true width of 0.9m by the inclusion of waste material, on condition that the weighted average gold grade of the drill hole interval was greater than 2 g/t. This strategy was employed to ensure the inclusion of all potentially mineable vein widths and care was taken not to over-dilute mineralised zones.

The following techniques were employed whilst interpreting mineralised zones:

- Cross sections spaced 40m apart in the section control file were displayed interactively, with a clipping window of 20m (distance constraint to the north or south of the plane of the cross section). This is in keeping with the average drill spacing.

- All interpreted strings were snapped (constrained) to corresponding drill hole intervals, and so constrained in the third dimension.

- Where any interpreted mineralised zone did not extend from one cross section to the next, because of vein truncation, it was projected half way to the next section and terminated.

- At Kearney and at each of the named veins, the interpretation was extended beyond the first and last cross sections by a distance of half the section spacing or 50m. At depth, the interpretation was extended by a distance of half the drill hole spacing in the z plane or 50m. The use of one or the other of these extent parameters is informed by the observed level of geological/grade continuity.

- Within the pit environs, mineralised zones were defined by using both pit floor channel sampling and drill hole sampling to define the sub-surface continuity. Continuous vein zones, as interpreted on cross sections, were extended above the surface of the pits to include channel sampling data, constrained to the surface 2 ppm Au string. Because of mining and bulk sampling activity, historical surface channel sampling positions lie above the Kearney and Joshua pit floors, the wireframe therefore extends above the current surface. It is valid to include these “suspended” channel samples on condition that the resulting block model is constrained to the current topographic surface prior to reporting the updated estimate of resources. In contrast to the method adopted by Howe 2012; Joshua and Kearney drill intersects were strung to every individual historic channel for the purposes of the current report.

15.7 WIREFRAMING

Once interpreted strings were created to define mineralised zones, the strings were used to generate three-dimensional solid wireframe domain models of each mineralised vein at Kearney and Joshua. At Kearney, the vein zone is shown to be continuous over the drilled strike length of more than 800 m, however, individual veins within this zone, which can be mapped from one section to another, are not as continuous. The continuity, orientation and geometry of any given mineralised vein may vary. Vein continuity varies from between 30 m to 450 m along strike, influenced by either structural disruption and offset, as observed in pit mapping, or gold grade criteria. Therefore, each continuous/semi-continuous mineralised vein, interpreted in three dimensions and defined by grade criteria, was considered an individual domain for estimation.

Some of the intersections north of the Kearney pit allow alternative interpretations of continuity and branching. Where these exist, wireframe interpretation has been guided by the pattern of branching in the pit, where veins can be seen to branch upwards and to the north.

Once each solid wireframe was created, it was visualised in 3D space and validated using MICROMINE solid object validation functions to ensure wireframe surface continuity and generation of solid model volume. Once validated, each wireframe was given a domain name so that the assay database could be coded, and each assay flagged by the domain it informs.

A total of 32 domains were modelled by wireframe, 21 on Kearney Vein and 11 on Joshua Vein. Details of each interpreted domain are contained in Table 22 below. Wireframe solids for all domains are shown in Figure 33.

TABLE 22: DOMAIN DETAILS						
Wireframe	Description	Volume	Strike Extent	Max depth extent	Strike	Dip
			metres		Degrees	
Elkins 1A	Elkins Main vein, South section	15,717	116	84	342	65
Elkins 2A	Hanging wall splay	3,198	135	24	350	70
Elkins 3A	Sub-parallel footwall vein	8,066	124	69	350	70
Elkins 4A	Sub-parallel footwall vein	3,113	66	33	350	70
Elkins 5A	Elkins Main vein, North section	9,773	90	62	350	70
Elkins 6A	Sub-parallel vein 75m in footwall	4,697	102	67	350	70
Joshua 1	Main vein, south extension	27273	284	144	162	87
Joshua 1A	Main vein splay	4390	34	152	160	86
Joshua 1B	HW lens	1123	29	46	186	86
Joshua 2	Main vein, central dip reversal	104665	169	234	168	75
Joshua 2A	HW lens	906	30	40	180	73
Joshua 3	Main vein	32627	165	161	162	85
Joshua 4	North HW vein	12672	156	95	176	65
Joshua 5	North Joshua vein/ splay	8084	63	135	193	79
Joshua 6	North vein/ splay	14992	86	153	174	79
Joshua 6A	North vein/ splay	775	20	51	180	73
Joshua 7	North vein/ splay	9522	57	146	159	79
Kearney South 2012	Main vein below trenches at S end of pit	2869	36	40	194	83
Kearney 1	Main vein below trenches at S end of pit	68127	235	190	179	84
Kearney 2	HW vein below trenches at S end of pit	16203	107	94	182	66
Kearney 3	Main vein below trenches central pit	47038	250	185	176	87
Kearney 4	Main vein central pit	95553	266	273	175	81
Kearney 5	FW vein central pit below trenches	23261	203	185	173	85
Kearney 6	FW vein, central pit	84728	207	309	175	86
Kearney 7	HW vein south pit	23825	97	152	175	68
Kearney 8	HW lens central Kearney	5843	19	100	176	85

Kearney 9	HW vein, central pit	87653	269	256	183	86
Kearney 10	HW lens, central pit	20260	50	143	180	78
Kearney 11	HW lens, central pit	2230	40	77	178	76
Kearney 12	FW vein central pit	50899	144	200	184	83
Kearney 13	HW vein, north end of pit	34550	106	367	179	76
Kearney 14	Sub-parallel HW lens at N end of pit	2424	41	90	180	72
Kearney 15	Sub-parallel HW lens at N of pit	15409	89	129	179	72
Kearney 16	sub-parallel HW lens, N of pit	5069	70	81	180	75
Kearney 17	sub-parallel HW lens, N of pit	5263	75	84	179	85
Kearney 19	HW vein below channels, north pit	1738	93	29	169	80
Kearney 20	FW lens, north pit	8947	75	99	172	85
Kearney 21	HW lens, N of pit	6292	75	82	180	83
Kerr 1	Main vein	11857	130	163	160	75
Kerr 1A	Kerr splay	1592	84	32	144	85
Kerr 2	Kerr splay	901	48	23	158	79
Kerr 3	Kerr splay	117254	54	25	151	80
Kerr 4	Kerr splay	400	69	7	160	83
Kerr 5	Kerr splay	4147	59	142	158	71
Gormleys 1A	Main Gornley's Vein	21,546	255	73	310	80
Gormleys 2A	Sub-parallel vein to southwest	9,250	172	82	310	80
Gormleys 3A	Sub-parallel vein to southwest	4,526	100	61	310	80
Garrys 1A	Main Garry's Vein	6,140	100	65	320	76
Sammy's 1A	Main Sammy's Vein	10,619	154	83.6	185	75
Sammy's 2A	Sub-parallel vein to west	7,545	94	115	185	75
Princes 1A	Main Princes Vein	3,492	77	67	310	78

TABLE 22: MINERALISED DOMAIN DETAILS

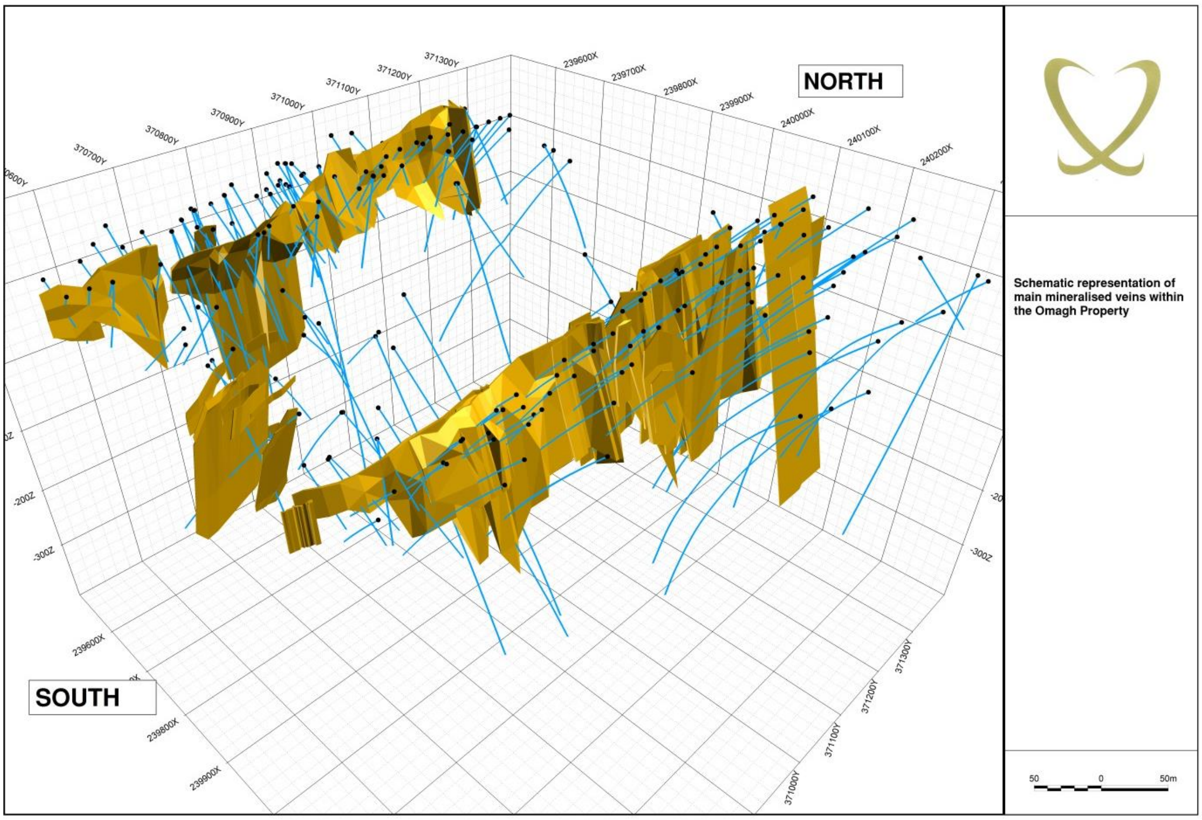


FIGURE 33: WIREFRAMES OF MAIN ZONES

15.7.1 SAMPLE DATA SELECTION, TOP-CUTTING AND COMPOSITING

15.7.1.1 SAMPLE DATA SELECTION

In the ACA Howe report of 2012 a thorough and detailed statistical analysis was undertaken to assess the sample characteristics of drill hole and channel data for the Kearney vein, the Joshua vein and other named veins. For the writing of this report a classical analysis was undertaken to describe the dataset. The mean, median, mode, variance and standard deviation were all determined, and appended to the statistical analysis already completed in previous reporting. The results of this are displayed in Table 23.

Analysis of the statistics in Table 23 shows that there is a difference in gold grades returned from channel samples and drill hole samples. Possible reasons for the apparent discrepancy include surface enrichment, differing sample intervals and inclusion of wall rock, core loss, and ore body shape.

As commented on in the Howe 2012 report, *surface enrichment is considered as unlikely since fresh sulphides occur at surface and oxidation is limited to very minor limonite development.*

Sample assay data derived from drill holes which targeted the Kearney and Joshua veins show higher gold grades in Galantas drill holes compared to nearby Riofinex holes. The loss of clay gouge as a source of Au from an adjacent depleted quartz vein was identified as an issue (Wolfenden & Cliff, 1996). Once the gold is washed away in the drilling fluids then the sample is a poor representation of the intersect. The use of triple tube core barrels and drilling muds employed by Galantas has mitigated the grade loss and lead to a greater confidence in the reporting of the resource.

TABLE 23: SAMPLE STATISTICS FOR DIFFERENT DATA TYPES

		HISTORICAL RIOFINEX SAMPLES		**GALANTAS SAMPLES, 2006-2007**					**GALANTAS SAMPLES, 2011-2014**					
	ALL SAMPLES	ddh	Kearney channels	Kearney ddh	Joshua ddh	other zones ddh	Kearney channels	Joshua channels	Kearney ddh	Joshua ddh	Other ddh	Kearney Channels	Joshua Channels	Other channels
Minimum	0	0	0	0.01	0.01	0.01	0.01	0.01	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	626	626	123.82	165.12	59	101.76	75.25	125.44	54.08	83.30	13.60	26.10	125.44	58.30
No of points	11950	1334	5105	1342	365	218	373	2179	500	994	53	995	2245	1018
Sum	34734.03	3613	15663	3537	1235	330	1636	11082	917.07	2872.94	43.97	628.30	11082.69	1740.44
Mean	2.91	2.71	3.07	2.64	3.39	1.51	4.39	5.09	1.83	3.043	0.83	0.63	4.94	1.71
Variance	104.72	372.98	70.18	77.34	57.10	55.45	81.02	115.24	38.12	75.67	6.25	4.09	112.64	31.50
Std dev	10.23	19.31	8.38	8.79	7.55	7.45	9.00	10.74	6.17	8.70	2.5	2.02	10.61	5.61

TABLE 23: SAMPLE STATISTICS

From Table 23 above, the relatively low Au grades from Kearney in the latest drilling programme could be explained by the sampling of a wider and more diffuse hydrothermal alteration zone that is absent on Joshua. Hole 142 is an example of this, with a down-hole intersect of the aureole in-excess of 100 m.

Surface channel samples, at Kearney and Joshua are clustered relative to sub-surface drill core samples (the average spacing along strike is 5m *for channel samples compared to 20-40m for drill core samples) and therefore care is required when considering these samples for use in resource estimation, since potential exists to extrapolate these clustered grades over unrealistic distances, and overstate this grade data in parts of the resource containing relatively sparse drilling data.*

The review of raw sample data from different sample types suggests that all channel and drilling data for the Kearney Vein Zone is suitable for use in resource estimation of Kearney and that Riofinex drilling data and recent drilling data over other named veins is suitable for use in resource estimation, with the exception of some twinned holes at Joshua vein.

The above assessment of the dataset from ACA Howe 2012 is still deemed accurate and a fair representation of the dataset and Galantas is in agreement with the conclusions of the 2012 ACA Howe report.

The verified assay file was flagged so that each sample assay was assigned a domain in accordance to where it was named in a wireframe. Classical statistical analysis was undertaken on assays within each domain to investigate the statistical characteristics of each domain and to provide useful information when considering top-cutting data prior to estimation.

15.7.1.2 TOP-CUTTING

Top-cutting is an important step in resource estimation, and particularly so for the estimation of resources at Cavanacaw since extreme grades (>100ppm Au) have been reported both from sampled drill core, and surface channel samples. Whilst extreme grades are real, they are not representative, and occur as outliers that have the potential to over-estimate domain grade if left un-capped. There are several domains that contain too few samples to reliably determine the top-cut grade on a domain by domain basis, and in order to not over-cut the sample data, a top-cut value was determined from all drilling data, and all channel data.

When considering an appropriate top-cut grade, the sample histograms for both drill and trench data were reviewed in 2008 in order to see the grade at which the histogram tail deteriorates, i.e. where grades become non-representative for each domain. In addition, sample data were sorted into descending order and several top-cut values applied in order to see what affect the top-cut value had on the coefficient of variation (CV), the measure of population variability, as well as the loss of metal from the sample population. A top-cut value was chosen that resulted in a CV of close to 1-1.2 (the desired CV for this type of vein hosted gold deposit), but that did not remove a significant amount of metal from the deposit so as to underestimate resources.

Top-cut analysis performed in 2008 suggested an appropriate top-cut to be 75ppm Au, for both trench data and drill data, which resulted in a CV of 1.4. On a domain by domain basis, any assays greater than 75ppm Au were replaced with 75ppm Au. This resulted in the application of a top-cut value to 8 out of 44 domains.

In view of the statistical similarities between 2008 and 2014 data, it was decided that retaining a top cut of 75 ppm Au for use in the 2014 estimate was an appropriate measure. Accordingly, all assay values exceeding 75 g/t Au were cut to 75 g/t Au.

15.7.1.3 COMPOSITING

Data compositing was undertaken on top cut sample assay data prior to geostatistical analysis and interpolation, in order to standardise the sample database and so generate sample points of equal support to be used in estimation. Historical and recent drill hole sampling was undertaken over drilled intervals of between 0.10m and 3.0m, averaging 0.5m. A composite length of 0.3m was chosen as compatible with a minimum mining width of 0.9m (3 composites). This compositing strategy is the same as that used for the 2008, 2012 and 2013 resource estimates.

Raw drill hole samples within each mineralised zone were flagged by domain in the sample database and composited to 0.3m intervals, starting at the drill hole collar and progressing downhole.

Compositing was stopped and restarted at domain boundaries and at the end of every hole. Though isolated and rare, un-sampled intervals within the domain model were inserted into the sample database and assigned a grade value of 0 prior to compositing. The minimum permitted composite length was 0.1m, defined in order to capture the final grade interval downhole, commonly at the edges of mineralised domains. In these instances, a final composite was created if the interval was greater than 0.10m. If the final interval was less than 0.10m, a weighted average was calculated from the final two composites. Channel sample data were composited in the same way.

15.7.2 GLOBAL REFERENCE ESTIMATE

Once the data had been subject to top cutting and was composited, the grade and tonnage of each wireframe was measured to establish the global reference estimate. This is used to establish a preliminary grade and tonnage and to cross check the later block model global estimates.

15.7.3 GEOSTATISTICS

The geostatistics detailed in this chapter have been carried out to interpret the assay data set in a systematic way, to provide qualitative and quantitative insights and link these in three dimensions to describe spatial continuity.

The geostatistical study completed in 2008 and maintained in the Howe 2012 report was deemed to be sufficient and thorough, no additional samples greatly increased the specific populations of domains used in the variogram modelling in the report. The author concludes that the validity of the study is therefore intact for the purposes of this report. An excerpt of the original study is included below.

The purpose of geostatistical analysis is to generate a series of semivariograms that describe the orientations and ranges of grade continuity and that can be used as the input weighting mechanism and search ellipse parameters for Kriging algorithms or to define the search ellipse parameters for Inverse Distance Weighting interpolation of the Kearney deposit and other named veins. At Kearney, geostatistical analysis was conducted on drilling data within those domains that contained enough sample points for potentially meaningful analysis, and exhibited the greatest vein continuity, in an attempt to define reliable search orientations and ranges that could be applied to these and other vein domains within the deposit. Variographic analysis was undertaken for domains K3, K4 and K6.

For each domain, variograms were calculated and modelled for the composited gold data and constrained by the domain. A range of omni variograms with variable lag distance was generated to estimate the possibility of generating good directional variograms and to determine the optimum lag distance to be employed. The optimum lag distance was determined to be between 25m and 40m, which reflects the average drilling grid dimensions over the Kearney deposit.

Downhole experimental variograms were modelled to assess the short range grade variability, and to determine the expected nugget effect. The nugget effect from all three domains was found to be very low, due in part to the decrease in grade variability following compositing.

The experimental semivariograms models for the directions of maximum grade continuity were attempted, and although the models for the first direction (main direction) appeared reliable and were based on a significant number of sample points, the sample points captured in models for the *second (dip) and third (across dip) directions were few, and well defined variograms, for use in Kriging, could not be modelled. In narrow, structurally controlled vein deposits such as those within the Omagh project, the third direction semi-variograms are notoriously difficult to model and therefore the third direction is defaulted as being perpendicular to the other main directions.*

The directions of maximum continuity within domains K3, K4 and K7 were found to be 250°, 355° and 160° respectively, with no reliable plunge component modelled. These directions correlate well to the observed strike orientation of these mineralised domains. The second directions (dip) were found to be -80° -75° and -60° respectively and approximate the dip angle of each mineralised domain. Third directions could not be modelled.

Specific ranges of continuity along each of the three modelled directions for each domain could not be modelled definitively and variograms were relatively poor, but ranges for the first and second directions in all three domains were between 40m and 80m, the former being considered reasonable given the local structural controls that can affect vein continuity over relatively short strike extents. Two-dimensional variography performed on pit channel sampling data in 2004 suggested grade continuity over a range of 20m-40m along strike.

Although the generated variograms parameters for domains K3, K4 and K7 are not sufficiently defined to be used as inputs to Kriging, the orientations of the first and second directions approximate the geometry of interpreted mineralised zones and so are considered valid inputs to define the search ellipse used in the interpolation process. In the absence of third direction parameters, the third direction in each domain was defaulted as being perpendicular to the other main directions, and approximates the across dip direction of each mineralised domain. The range in the third direction was input as being 1/3 the range of the other directions, to honour the narrow thickness of the vein zones in the across dip direction.

Search ellipse orientations for other domains of the Kearney vein zone were taken to be the strike, dip and across dip orientations of each domain, which is valid. The search ranges generated for domains K3, K4 and K6 were applied to all other domains as it is reasonable to assume similar grade continuity in other vein zones within the same system.

Following variographic analysis, the ranges in the first, second and third directions, used to determine the search radii employed during the interpolation process, were set at 60m, 60m and 20m respectively. Generated semi-variograms from the 2008 study are contained in Appendix V of the Howe (2008) report.

15.8 BLOCK MODELLING

The spatial extent of a blank block model was determined covering the parameters of the domain wireframes. A block size was then defined with consideration of drilling density and the variability of the ore body. The empty block model was constrained to the wireframe domain as best allowed by the dimension of the block size. Each block was then assigned the code of the wireframe it was constrained within.

The initial filling was undertaken by blocks with the dimensions 15m by 5m by 10m followed by sub-blocking down to 1.5m by 0.5m by 1m where appropriate. The parent block dimensions were chosen to honour the generally accepted rule that parent blocks should be not less than half the general *exploration grid, which at Kearney is generally 40m by 40m, and in some areas 20m by 20m. Therefore the y (strike) block dimension is 15m. The x block dimension of 5m honours the narrow thickness of the vein zones and the z dimension of 10m is in line with proposed mining bench height. Sub-blocking down to one tenth of parent block dimensions is required in order to maintain the resolution of the mineralised envelopes so as to accurately honour wireframe volume.*

Once blank block models had been created each was constrained by the DTM of the topographic surface and combined limits of the Kearney pit. Those blocks above the DTM were deleted and those remaining below were ready for interpolation. Block model characteristics for each model are contained in Table 24 below.

2012-2014 BLANK BLOCK MODEL DETAILS					
Model	**Dimension**	**Extent (Irish Transverse Mercator Grid**		**Parent Block Size (metres)**	**Minimum sub-blocks (metres)**
		Min	Max		
Kearney	Eastings	240062	240221	5	0.5
	Northings	370493	371360	15	1.5
	RL	-205	165	10	1
Joshua	Eastings	239576	239778	5	0.5
	Northings	370508	371203	15	1.5
	RL	-72	184	10	1

TABLE 24: 2012-2014 BLANK BLOCK MODEL DETAILS

15.9 GRADE INTERPOLATION

Gold grades were interpolated into the empty block model for each deposit using the Inverse Distance Weighting (IDW) interpolation, raised to third power. Each block model was populated on a domain-by-domain basis using composited, top-cut assay data. A closed interpolation approach was adopted, whereby only composite assay data situated within each domain, were used to interpolate the grade of blocks within that domain. This was done by filtering out all sample assays apart from those in the relevant domain. Variographic analysis in 2008 was not thought to be robust enough to define the input parameters required for a reliable kriged estimate of each domain at each deposit, however the observed nugget effect, derived from down hole variograms, is considered reliable and is found to be low for each of the domains investigated through variography (<10%). One of the main advantages that kriging has over IDW interpolation is that the nugget effect, or grade variability over very short distances, is factored in to the kriging algorithm whereas it is assumed to be zero when using IDW. The presence of a very low nugget effect therefore validates the use of IDW as a reliable interpolation method.

Grades were interpolated into each block using the inverse of the distance from the centre of the block being estimated, to the surrounding sample points used to estimate the block grade, as a mechanism to preferentially weight each sample point. The inverse of this sample to block distance is commonly raised to a power of 2 or 3 in structurally controlled vein gold deposits to ensure that samples closest to the block being estimated are given more weight, as vein hosted gold deposits typically exhibit a high degree of grade variability along the vein.

There is a requirement to ensure that grade variability within the Kearney and Joshua veins is kept local so as not to place undue weight on the clustered surface channel sample data, which represents a larger sample population than drilling sample data, but over relatively small and constrained portions of the veins. Accordingly, inverse distance cubed (IDW^3) interpolation was used at Kearney and Joshua veins in order to constrain grades more locally than would be the case if IDW^2 was used.

Interpolation of each deposit block model was undertaken on a domain by domain basis and for each domain, grade interpolation was run several times at successively larger search radii until all blocks received an interpolated grade. Concentric search ellipses were used in order to avoid grade smearing and to preserve local grade variation.The radii of the search ellipses were determined by the results of variographic analysis carried out in the 2008 report where consideration of appropriate ranges of continuity, applicable to this type of deposit, was given. The ranges were chosen to be 60m, 60m and 20m in the three main directions. For all domain interpolations, the first search radii were selected to be 1/3 of the range in all directions. The second, larger search radii were selected to be 2/3 the range. Successive search radii were selected to be equal to twice and three times the ranges in all directions until all domain blocks received an interpolated grade.

To increase the reliability of the estimates, when model blocks were interpolated using search radii not exceeding the full ranges, a restriction of at least three samples from at least two drill holes or channels was applied. When blocks were interpolated using search radii exceeding the range, the restriction was reduced to at least one sample from at least one drill hole or channel. Sample data over the Kearney and Joshua veins is 'clustered' in that it comprises a very large number of channel sample assays in comparison to a small number of drill hole assays. If a large number of channel assay values are picked up in the search ellipse, then these points will contribute unduly to the interpolated grade of the block in comparison to the less numerous drill hole points. In order to avoid this bias, declustering was undertaken using the MICROMINE sector method whereby the search ellipse, regardless of the radii employed, is divided into four sectors and a constraint used during interpolation, a maximum of four points per sector is allowed. Therefore, the maximum combined number of samples allowable for the interpolation is 16.

The interpolation strategy employed to estimate block grades at Kearney and other named veins is contained in Table 25 below.

TABLE 25: INTERPOLATION STRATEGY				
Interpolation Method	IDW3 (Kearney and Joshua) IDW2 (other veins)			
Interpolation Run Number	1	2	3	>3
Search Radii	1/3 Range in all directions	2/3 Range in all directions	Equal to the range in all directions	3 time Range in all directions
Search Radii, metres	20	40	60	80
Minimum Number of samples	3	3	3	1
Maximum Number of samples per sector	16	16	16	16
Minimum Number Holes/Channels	2	2	2	1

TABLE 25: INTERPOLATION STRATEGY

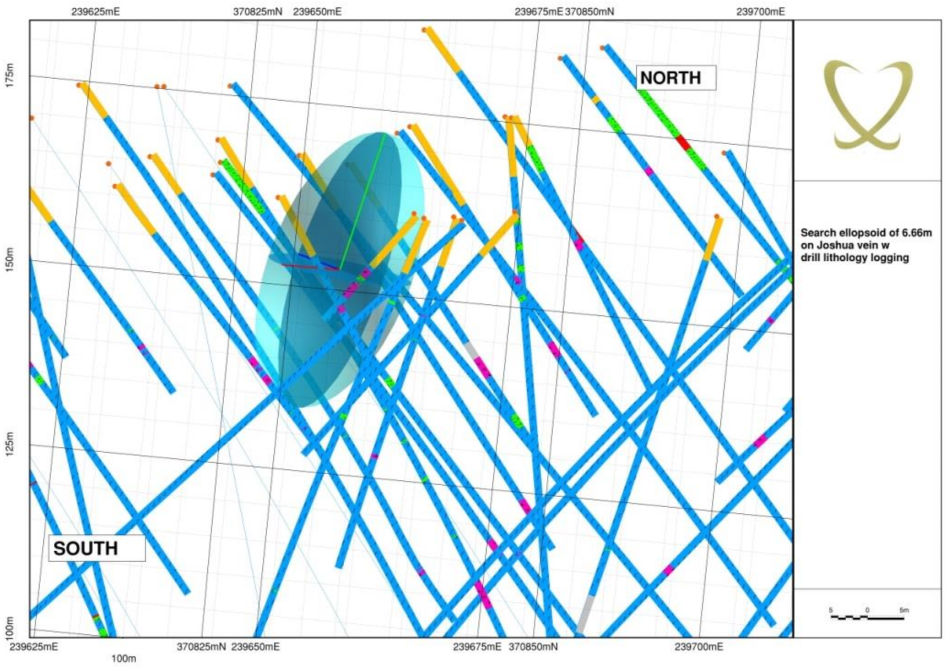


FIGURE 34: SEARCH ELLIPSOID ON JOSHUA VEIN

*The orientation of the search ellipsoids was adjusted for each domain so that the main axis of the ellipse was coincident with the long axis (strike) of the domain. The second axis was orientated perpendicular to the first axis and parallel to the dip, and the third axis was orientated perpendicular to axes 1 and 2, and to the plane of the vein (*Figure 34*).*

Figures 35 to 37 below are vertical longitudinal sections of Kearney and Joshua veins with block models showing gold grade distribution.

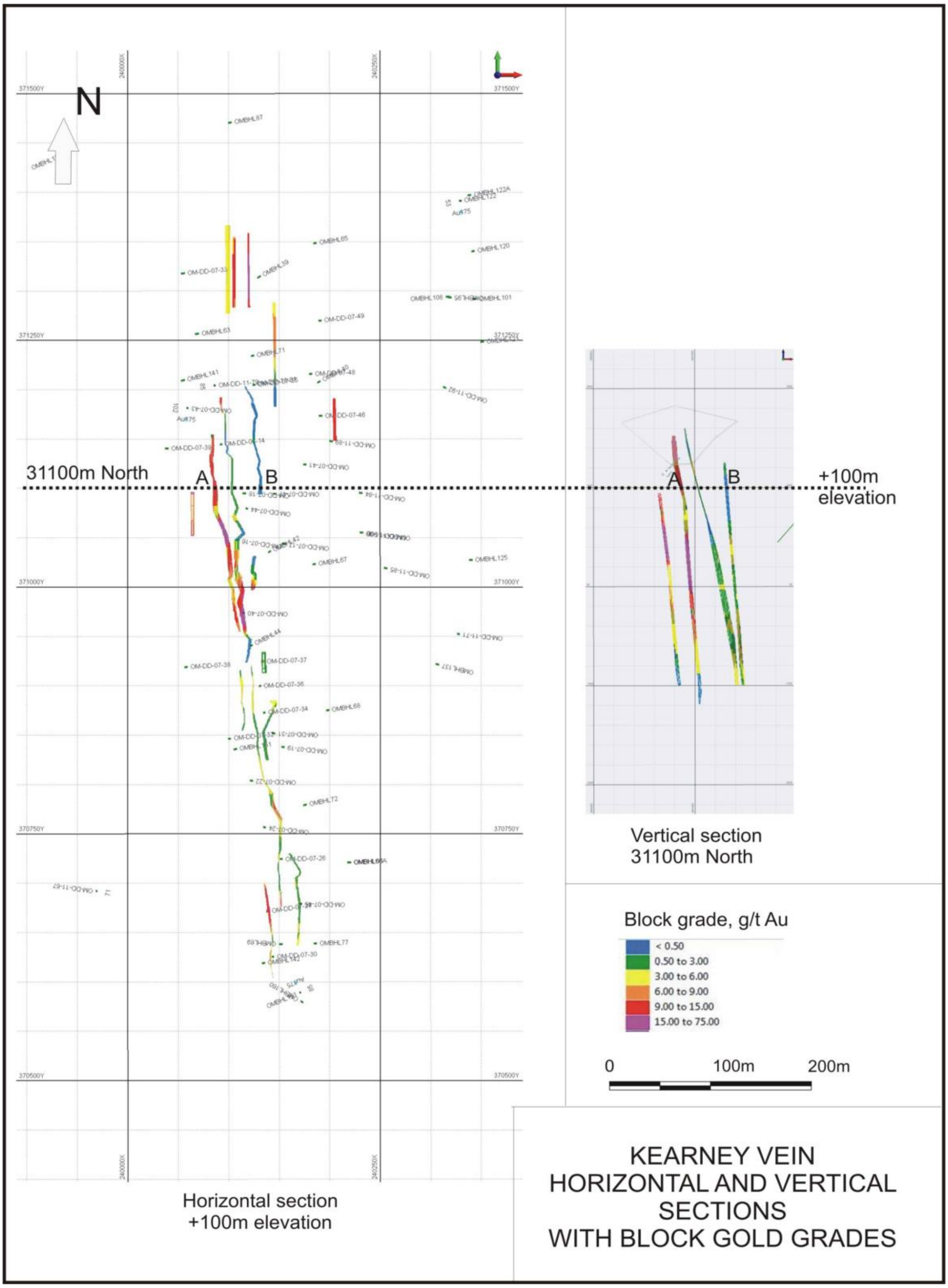


FIGURE 35: PLAN AND SECTION OF THE KEARNEY VEIN (ACA Howe 2012)

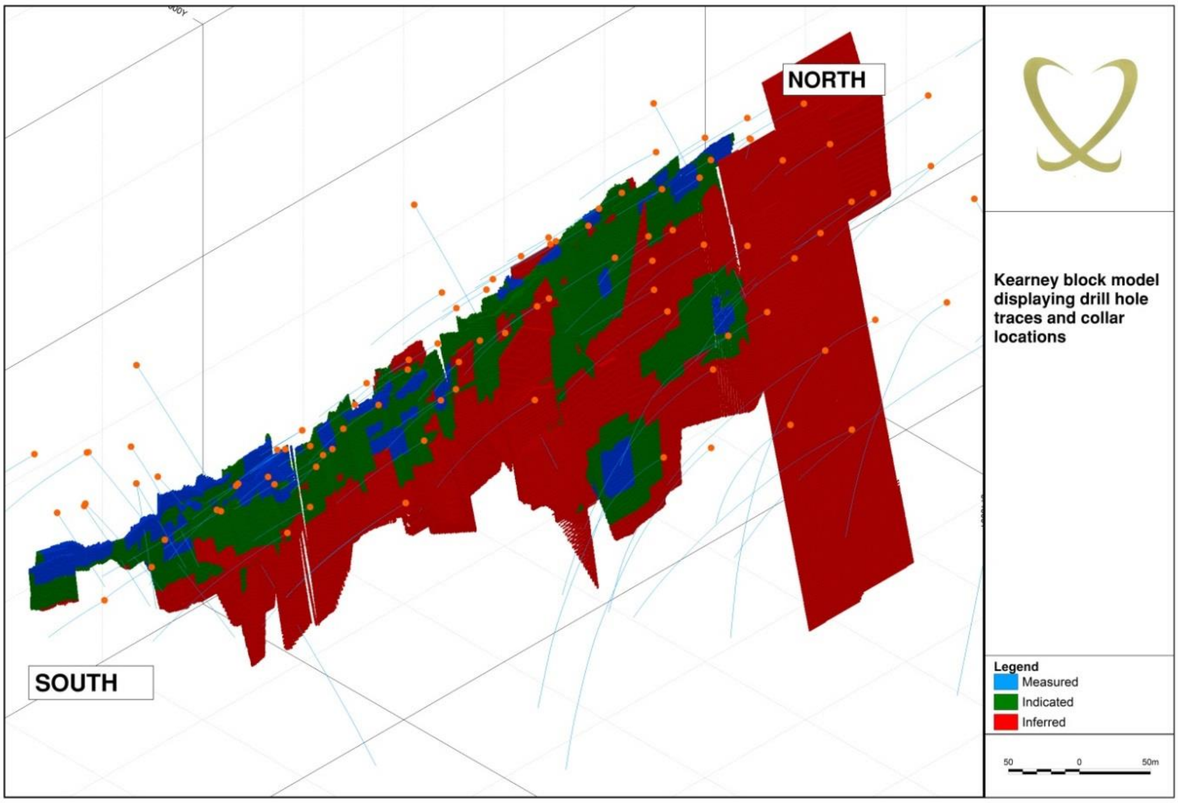


FIGURE 36: KEARNEY BLOCK MODEL

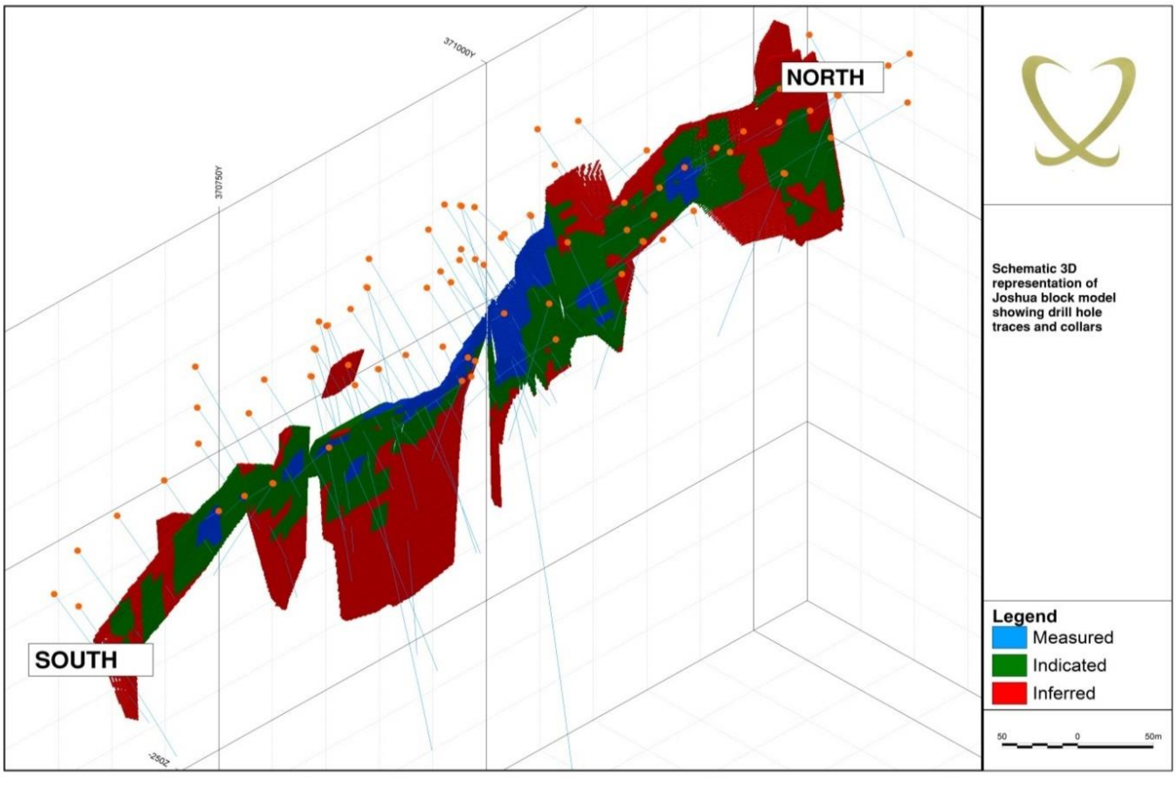


FIGURE 37: JOSHUA BLOCK MODEL

15.10 RESOURCE CLASSIFICATION

As reported in the ACA Howe 2012 report:

The CIM Definition Standards on Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Resource Definitions and adopted by the CIM council on December 11, 2005, provide standards for the classification of Mineral Resources and Mineral Reserve estimates into various categories. The category to which a resource or reserve estimate is assigned depends on the level of confidence in the geological information available on the mineral deposit, the quality and quantity of data available, the level of detail of the technical and economic information which has been generated about the deposit and the interpretation of that data and information. Under CIM Definition Standards:

- ***An "Inferred Mineral Resource" is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological or grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.***
- The CIM definition also cautions against using Inferred resources in higher level economic studies and where used in lower level studies , such as Scoping Studies, recommends specific cautionary language. This recommendation is reflected as a prohibition within section 2.3(1)b of NI 43-101.

Under the Pan-European Reporting Code (PERC), the definition of Inferred resources is very similar but permits qualified use of Inferred resources within economic studies and the definition is expanded as follows :-

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling.
Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.
An Inferred Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Where Inferred Mineral resources are extrapolated beyond any sampled region containing data points, the proportion extrapolated must be described and disclosed.
The 'Inferred' category is intended to cover situations where a mineral concentration or occurrence has been identified and limited measurements and sampling have been completed, but where the data are insufficient to allow the geological and/or grade or quality continuity to be confidently interpreted. Due to the uncertainty which may be attached to some Inferred Mineral Resources, it cannot be assumed, but normally would be expected, that a major part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.
Confidence in the estimate is usually not sufficient to allow the appropriate application of technical and economic parameters or to enable a reliable evaluation of economic viability. For this reason, there is no direct link from an Inferred Resource to any category of Mineral Reserves (see Figure 1). It is accepted that mine design and mine planning may include a proportion of Inferred Mineral Resources.
If this category is considered in mine design, planning and/or economic studies, the results of which are publicly reported, full disclosure and the effect on the results of the studies must be stated.

Inferred Mineral Resources may only be included in mine design, mine planning, and/or economic studies provided that there exists a mine plan and a statement of Mineral Reserves, which states that Inferred Mineral Resources have been used. Where a material amount of mining in the mine plan includes Inferred Mineral Resources, a comparison of the results with and without these Inferred Mineral Resources must be shown, and the rationale behind their inclusion must be explained.

Modifying factors and assumptions that were applied to the Indicated and Measured Mineral Resources to determine the Mineral Reserves must be equally applied to the Inferred Mineral Resources if these are included within a mine plan, but the Inferred resource must nevertheless be reported as such, and not as a reserve.

For the avoidance, of doubt, it is reiterated that caution should be exercised if this category is considered in technical and economic studies. At the discretion of the Competent Person, a Company may include all or part of its Inferred Mineral Resource for the purpose of internal planning, scoping or strategic studies. Any such reliance on Inferred Resources should be made clear in the report. In such circumstances, the results are not considered to be sufficiently reliable to ensure that all of the Inferred Mineral Resource will eventually become a Mineral Reserve. Any such reliance on Inferred Resources in a mine plan should be made clear in the report. Inferred Mineral Resources cannot be converted to Mineral Reserves, and must not be stated as part of the Mineral Reserve.

Under CIM definition :-

- ***An "Indicated Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.***

PERC provides a similar definition :-

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

The Indicated Mineral Resource has sufficient confidence to support generalised mine design, mine planning, and/or economic studies.

An Indicated Mineral Resource requires that the nature, quality, amount and distribution of data are such as to allow the Competent Person to confidently interpret the geological framework and to assume geological continuity of mineralisation, with sampling at a pattern and spacing appropriate to the geological characteristics and complexity of mineralisation.

Confidence in the estimate is sufficient to allow the application of technical and economic parameters, and to enable an evaluation of economic viability. 'Grade or quality' is to be interpreted broadly, to include all relevant chemical and mineralogical characteristics.

Under CIM definition :-

- ***A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.***

A similar definition is used in the PERC code :-

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Mineral Reserve or to a Probable Mineral Reserve.

The occurrence of mineral(s) of economic interest may be classified as a Measured Mineral Resource when the nature, quality, amount and distribution of data are such as to leave no reasonable doubt, in the opinion of the Competent Person determining the Mineral Resource, that the tonnage, mineralogy, and grade or quality can be estimated to within close limits, and that any variation from the estimate would be unlikely to significantly affect potential economic viability.

This category requires a high level of confidence in, and understanding of, the geology, mineralogy, mineability and amenability to processing of the mineral deposit.

Confidence in the estimate is sufficient to allow the appropriate application of technical and economic parameters and to enable an evaluation of economic viability with a high level of confidence.

Classification, or assigning a level of confidence to Mineral Resources has been undertaken in strict adherence to the Pan-European Reporting Code, and follows the procedures described in Micromine Training V 11.0 Module 22 - Resource Estimation (2011).

Classification of interpolated blocks is undertaken using the following criteria;

- *Interpolation criteria and estimate reliability based on sample density, search and interpolation parameters.*
- *Assessment of the reliability of geological, sample, survey and bulk density data.*
- *Assessment of geological/grade continuity over the various domains at each deposit.*
- *Drilling exploration grid.*

The interpolation strategy dictates the classification of blocks to some degree since the parameters of each interpolation run result in a greater level of confidence in assigned block grade during the first interpolation runs. Interpolation runs at search radii larger than the defined range, capturing fewer points from fewer holes or channels results in a lower level of confidence in block grade, even though the block estimates are reliably calculated from available sample points.

Blocks have been classified as "Measured" if the following criteria are fulfilled:

- *Blocks captured in the first interpolation run at distances equal to 1/3 of the range in all directions, and;*
- *A minimum of four drill holes or channels must be captured.*

These criteria determine that Measured blocks were generated for no more than 20 m vertical depth below the Joshua, Kearney and Kerr detailed channel sampling. At Kearney, all these resources have now been mined. At Joshua, Measured resources have been partially depleted, but remaining resources extend for up to 18 metres below the present surface.

Indicated resources have been classified at the Kearney, Elkins, Joshua and Kerr deposits, which have all been drill tested both historically and recently on a relatively dense exploration grid (variable between 20m by 20m to 60m by 60m).

Blocks have been classified as "Indicated" if the following criteria are fulfilled:

- *Blocks in any domain that have been captured in the first and second interpolation runs at distances up to 2/3 of the range in all directions, and have not been classified as "Measured".*
- *A minimum of two drill holes or channels must be captured.*

Inferred Resources have been classified at all deposits.

Blocks have been classified as "Inferred" if the following criteria are fulfilled:

- *Blocks in any domain at any deposit that have been captured in any run equal to, or exceeding the range in all directions, and have not been classified as either "Measured" or "Indicated" blocks.*

15.11 DENSITY

In January 2008, Galantas undertook a density determination study of different ore types within the Kearney pit, in order to calculate an average bulk density value for the deposit that could be used to update the tonnage estimate. Five ore types were identified at the Kearney deposit and other named veins; primary quartz/sulphide, secondary quartz/sulphide, high sulphide clay gouge, low sulphide *clay gouge and altered wall rock. These different ore types are present in varying proportions over the Kearney deposit as a result of multi-phase ore genesis, making density determination of individual veins difficult.*

Galantas collected 25 samples of each ore type (a total of 125 samples) and undertook density determinations at their on-site laboratory. The average density for each ore type is contained in Table 26 below:

TABLE 26: DENSITY VALUES	
Ore Type	**SG**
Primary quartz/sulphide ore	3.636
Secondary quartz/sulphide ore	2.743
High-sulphide clay gouge	2.814
Low-sulphide clay gouge	2.767
Altered wall rock	2.767

TABLE 26: DENSITY VALUES

Once average density values were determined for each ore type, coded geology within each mineralised wireframe was extracted from the geological database and a list compiled of geological codes and their frequency. Ore types were then assigned to each code based on the descriptions of logged material. A weighted average density value was then determined, based on the frequency of each ore type in logged mineralised zones. The density value applied to the tonnage estimate for the Kearney deposit is 2.984. Density determination was not undertaken by Riofinex during historical drilling at other named veins. Therefore, the average value determined for Kearney was applied to other named veins, with the exception of Elkin's. Given that there are observed similarities between veins of the Kearney vein zone and other named veins, Galantas considers it reasonable to apply this density value to other named veins.

Recent initial data regarding Joshua vein densities suggests that this approach is reasonable.

At Elkin's, logging of recent drill core has shown that primary quartz/sulphide ore is the dominant ore type within these mineralised veins, exhibiting well formed cubic pyrite +/- arsenopyrite, chalcopyrite and galena that form often massive accumulations within veins. Therefore, the density value applied to the tonnage estimate is 3.636.

15.12 RESOURCE TABLE

The 2014 updated resource estimate for the Kearney deposit and Joshua vein is summarised in Table 27, with resources classified in strict accordance with PERC Definition Standards.

The resources listed here in Table 27 have been derived directly from the relevant block models and are subject to a 2 g/t Au cut-off grade.

	MEASURED			INDICATED			INFERRED		
	tonnes	grade Au (g/t)	contained Au (Oz)	tonnes	grade Au (g/t)	contained Au (Oz)	tonnes	grade Au (g/t)	contained Au (Oz)
KEARNEY	76,936	7.48	18,490	383,220	6.66	82,055	909,277	6.61	193,330
JOSHUA	54,457	7.25	12,693	216,211	7.92	55,046	291,204	10.74	100,588
KERR	6,848	4.63	1,019	12,061	4.34	1,683	23,398	3.2	2,405
ELKINS				68,500	4.24	9,000	20,000	5.84	3,800
GORMLEYS							75,000	8.78	21,000
GARRY'S							0	0	0
PRINCES							10,000	38.11	13,000
SAMMY'S							27,000	6.07	5,000
KEARNEY NORTH							18,000	3.47	2,000
total	**138,241**	**7.25**	**32,202**	**679,992**	**6.78**	**147,784**	**1,373,879**	**7.71**	**341,123**

TABLE 27: GALANTAS 2014 RESOURCE ESTIMATE

The mineral estimate as prepared, is compliant with current standards and definitions required under NI 43-101 and is reportable as a mineral resource by Galantas Gold Corporation. Numbers are rounded and gold grades are capped at 75 g/t gold. A cut-off grade of 2 g/t gold and a minimum mining width of 0.9 m has been applied. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

To the best knowledge of Galantas, 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 in this report.

15.13 MODEL VALIDATIONS

Model validations are an essential part of the reporting method. The easiest type of validation is a block model global reference estimate and a comparison to the wireframe global reference estimate. The Kearney block model report volume is 606,891.75 cubic metres, compared to the wireframe report volume of 608,181.67 metres cubed. The difference is 1,289.92 metres cubed or less than 0.5%.

The Joshua block model volume is 214,686 metres cubed, whereas the wireframe is 217,027.87 metres cubed. The difference is 2,341.87 or 1%.

Kearney and Joshua veins display good correlation in volumes and tonnage and within allowable limits, the difference being the naïve estimate given by the wireframe volume calculation.

A local validation was then carried out by taking plan, section and long section model slices through the Kearney and Joshua block models. Screen shots of these model validations coloured by gold grade are shown in Figures 39, 40. Detailed inspection of the original composited drill hole gold grades and the final block model was undertaken, working through the section control files for Kearney and Joshua. This is a largely visual process examining whether the drill data honours the interpolated blocks. Drill hole traces, gold grades and block model grade data is displayed and compared to assess the nature of the distribution of high grade areas around key intersects, and vice versa.

Figures 38 and 39 show good correlation between composited grades and block limits, controlled by the direction of drill strings. Some smoothing may be evident on sections where grades are interpolated into blocks that were not sampled. This is unavoidable despite the estimation methods used. The blocks in general do show a close correlation to the drill intersects, although some grade smoothing does occur. It is the view of the author that the block grade values match the composited grades used to interpolate the blocks. Therefore, the estimated grades in the blocks reliably represent the original geological data.

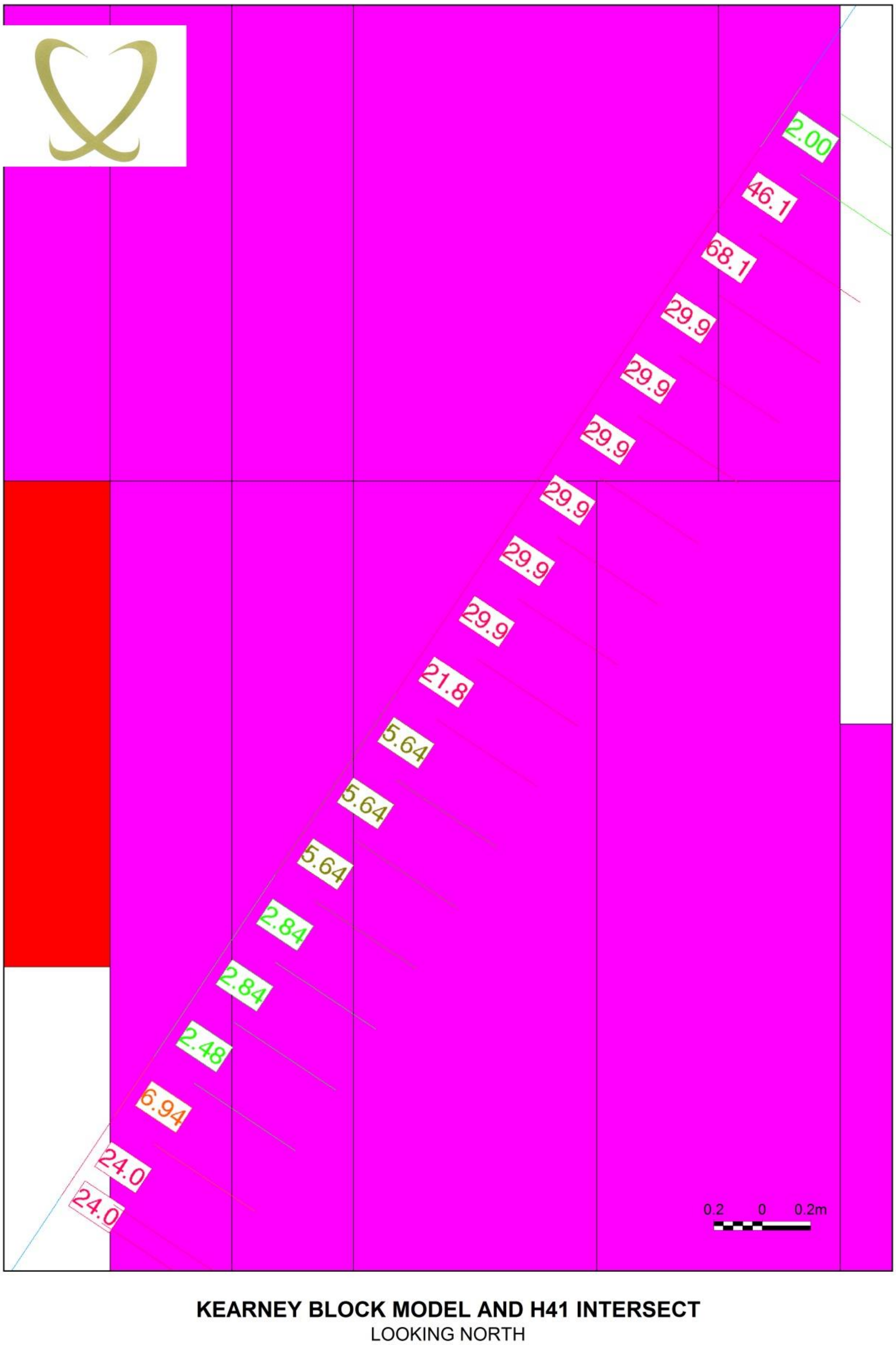


FIGURE 38: 2D VISUAL VALIDATION OF THE KEARNEY BLOCK MODEL AND DRILL HOLE DATA

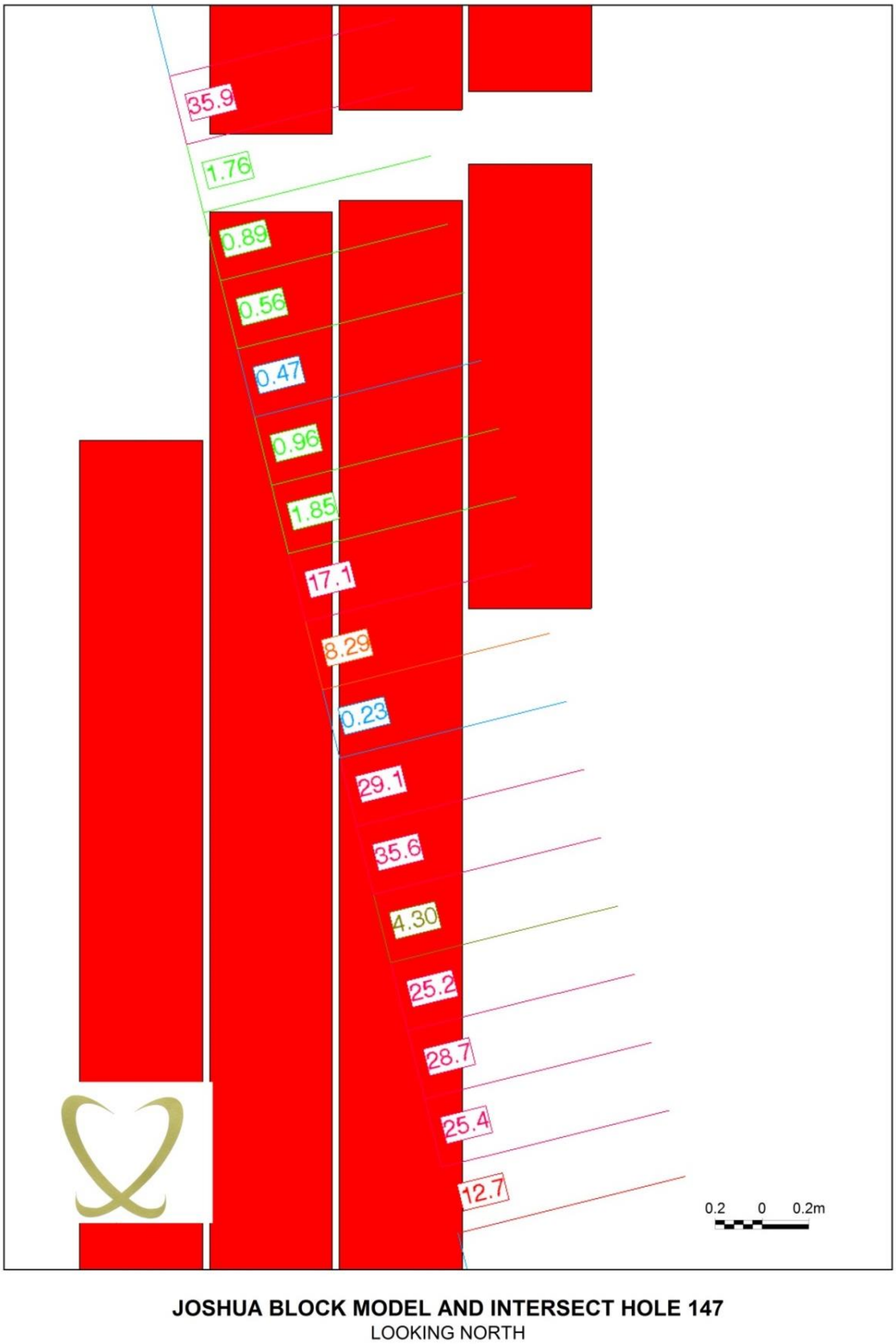


FIGURE 39: 2D VISUAL VALIDATION OF THE JOSHUA BLOCK MODEL AND DRILL HOLE DATA

15.14 COMPARISON WITH PREVIOUS RESOURCE ESTIMATES

The additional drilling since May 2013 and the process of re-stringing historical channels to drill core intersects, has led to an increase in the total resource and an increase in Measured and Indicated resources for the Kearney and Joshua veins.

Table 28 below shows resources according to year, zone, category and cut off. A comparison of the 2013 and 2014 estimates indicates a substantial increase (26.3%) in the Measured and Indicated resources.

- The increase in tonnage on Kearney was due to the deep infill drilling at depth.
- The Joshua vein tonnage and grade increased as a result of additional drilling.

A cut-off grade of 2.5 g/t Au applied in the 2012 report was reduced to 2.0 g/t based upon results of the economic study.

- **COMPARSION OF 2012 RESOURCE ESTIMATE, 2013 AND 2014 RESOURCE ESTIMATES**

ZONE	CATEGORY	2012 ESTIMATE CUT-OFF 2.5 g/t Au			2013 ESTIMATE CUT-OFF 2.5 g/t Au			2014 ESTIMATE CUT-OFF 2.0 g/t Au		
		TONNES	Grade (Au g/t)	Au ozs	TONNES	Grade (Au g/t)	Au ozs	TONNES	Grade (Au g/t)	Au ozs
KEARNEY	MEASURED	0	0	0	55,896	6.09	10,941	76,936	7.48	18,490
KEARNEY	INDICATED	270,900	7.94	69,000	327,542	6.56	69,057	383,220	6.66	82,055
KEARNEY	INFERRED	490,000	8.54	135,000	831,860	6.16	164,651	909,277	6.61	193,330
JOSHUA	MEASURED	13,000	6.48	2,800	59,002	4.92	9,331	54,457	7.25	12,693
JOSHUA	INDICATED	66,800	6.27	13,000	250,140	5.32	42,804	216,211	7.92	55,046
JOSHUA	INFERRED	173,000	8.48	47,000	395,886	6.45	82,148	291,204	10.74	100,588
ELKINS	INDICATED	68,500	4.24	9,000	68,500	4.24	9,000	68,500	4.24	9,000
ELKINS	INFERRED	20,000	5.84	3,800	20,000	5.84	3,800	20,000	5.84	3,800
KERR	MEASURED	2,250	6.75	500	2,250	6.75	500	6,848	4.63	1,019
KERR	INDICATED	5,400	5.03	900	5,400	5.03	900	12,061	4.34	1,683
KERR	INFERRED	26,000	4.58	4,000	26,000	4.58	4,000	23,398	3.2	2,405
GORMLEYS	INFERRED	75,000	8.78	21,000	75,000	8.78	21,000	75,000	8.78	21,000
GARRY'S	INFERRED	0	0	0	0	0	0	0	0	0
PRINCES	INFERRED	10,000	38.11	13,000	10,000	38.11	13,000	10,000	38.11	13,000
SAMMY'S	INFERRED	27,000	6.07	5,000	27,000	6.07	5,000	27,000	6.07	5,000
KEARNEY N.	INFERRED	18,000	3.47	2,000	18,000	3.47	2,000	18,000	3.47	2,000

TABLE 28: COMPARISON OF THE 2012, 2013 AND 2014 RESOURCE ESTIMATES

16 MINERAL RESERVE ESTIMATES

Mineral Reserve Estimates are not included in this report, in compliance with NI.43-101 requirements for independence.

Mineral Resource estimates used within the mining study are displayed in the table below (Table 29). This is further discussed *in 17.14 – Mining Study* and *17.17 – Inclusion of Inferred Resource.*

	Measured			Indicated			Inferred		
OREBODY	**Tonnes**	**Grade**	**Ounces**	**Tonnes**	**Grade**	**Ounces**	**Tonnes**	**Grade**	**Ounces**
KEARNEY	41,485	5.08	6,773	294,021	5.32	50,292	239,364	5.40	41,527
JOSHUA	51,581	4.89	8,112	232,753	5.27	39,450	86,044	6.91	19,109
	93,066	**4.97**	**14,886**	**526,774**	**5.30**	**89,742**	**325,408**	**5.80**	**60,635**

TABLE 29: MINING STUDY MEASURED, INDICATED & INFERRED RESOURCE ESTIMATES SCHEDULED

Minerals Resources that are not Mineral Reserves do not have demonstrated economic viability.

17 MINING ECONOMIC STUDY

17.1 MINING OVERVIEW

Mining operations will progress from previous open pit mining to future underground mining.

All current surface infrastructure will remain in place and will be upgraded where necessary to meet the operating parameters. New infrastructure is required to meet the specific needs of servicing an underground mine.

The economic veins of the deposit will be extracted using a combination of Sublevel Longhole Stoping and Shrinkage Stoping mining methods. Paste backfill will be utilized to maximize extraction whilst increasing stability and will reduce the surface footprint required for tailings disposal.

Initial underground mining will concentrate on the Kearney and Joshua veins that form the majority of the resource and reserves. Both areas will be accessed from a reinforced concrete adit that will be constructed in the existing Kearney pit, leading to the main underground decline.

The main decline will branch to provide access to the ore bodies. Cross Cut accesses will link the decline to the economic veins at 18m and 36m vertical intervals for Longhole and Shrinkage stopes respectively. Sill drives will be mined providing initial ore production whilst further delineating the veins.

The Joshua and Kearney ore bodies consist of multiple parallel veins of strike lengths of up to 850m. Average width is reported as 1.5m with a planned mining width of 0.9m, although this can be reduced to 0.75m over short distances. Sill drives will run in parallel and under strict geotechnical parameters.

Shrinkage stoping is carried out utilizing hand held air leg drills and broken ore will be scraped to the ore passes and loaded out of the draw points for hauling to the underground crusher. Sublevel Longhole stoping utilizes a production drill to drill between sills to extract the ore.

Ore is conveyed out of the mine from a central crushing location at the start of the main decline and will be deposited at the ROM ore storage facility next to the mill.

Development drilling and all mucking and hauling will be fully mechanized. Rubber tyred mechanized equipment will be used to maximize production and provide flexibility across the operation.

Stope drilling will be carried out using anti vibration hand held drills for Shrinkage stopes and a Longhole production drill for Sublevel Longhole stopes. Alimak raise climbers will be used for raising and to transport consumables and equipment into the Shrinkage stopes.

17.2 SITE LAYOUT

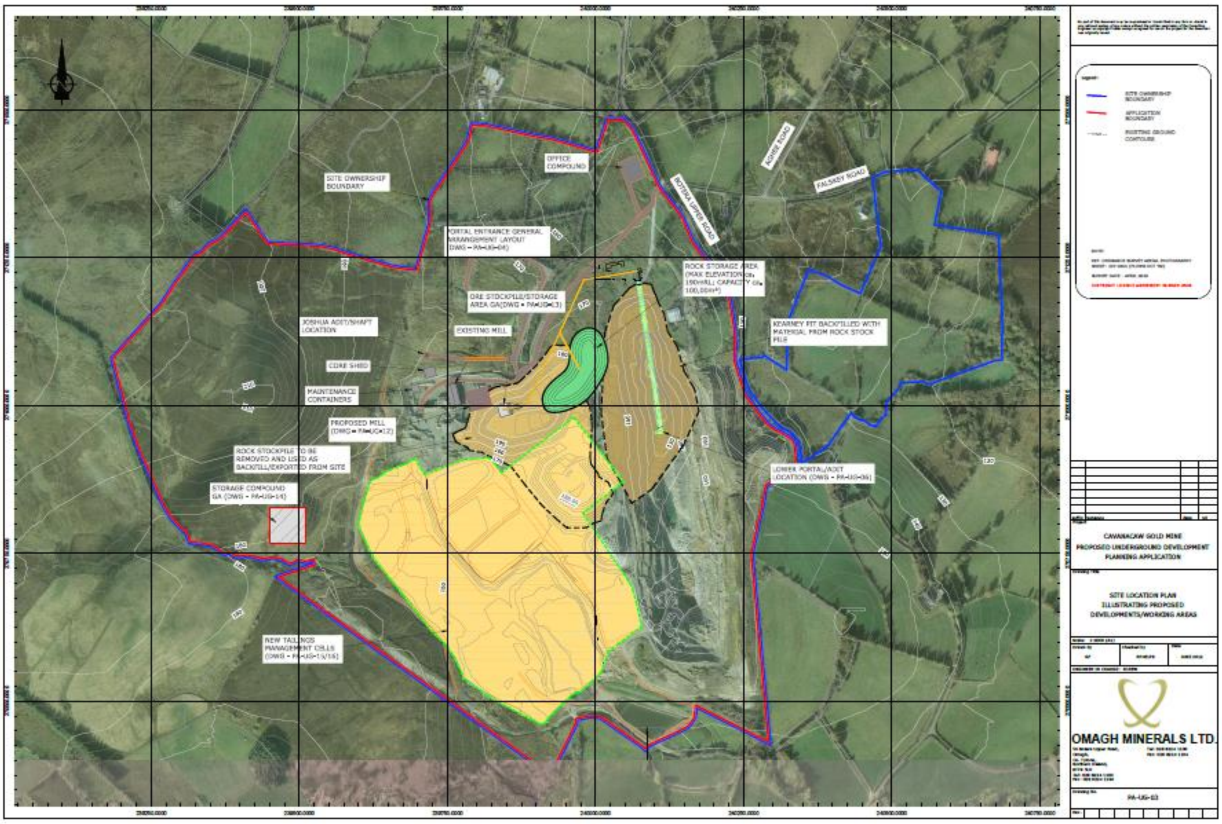


FIGURE 40: MINE SITE LAYOUT

17.3 MINING PARAMETERS

The following assumptions and parameters have been included in the study.

- Mine design, mining methods and mining equipment has been selected to produce 30,000 - 40,000 Au ounces per annum.
- Mine production should ramp to 1250 tpd at full production.
- Underground mining crews operate on three 8 hour shifts over 5 days per week.
- Mine designs and strategies adhere to UK Mining Legislation and Health and Safety Laws.
- Underground mine to be accessed via a reinforced concrete adit. This will be located in the existing Kearney pit and allows backfilling of the existing excavation.
- Installation of an underground crusher and conveyor system to haul ore and waste to the surface stockpiles.
- Development dimensions and profiles are as small as possible to minimise the amount of waste generated.
- Split face firing of ore development drives should be considered to maximise grade and reduce additional dilution.
- Capital works should be fit for purpose but minimised as much as possible to reduce pre-production expenditure.
- Early cash flow is required so mining schedule should reflect this by prioritising ore and ramping up to full production as quickly as possible.

17.4 ADIT INSTALLATION

A steel re-enforced concrete adit will be constructed in the existing Kearney pit. This will form the entrance from the surface to the underground mine. The Kearney pit will then be backfilled with development waste, stockpiled surface waste and retreated tails, burying the adit and allowing the surface to be rehabilitated.

17.4.1 KEARNEY PIT REHABILITATION

Rehabilitation work is required in the Kearney pit to stabilise the western wall prior to the commencement of the construction works for the adit, lower portal and underground development. The lower portal is a box structure that will be constructed in the same manner as the adit and will be the point at which the underground development enters virgin rock.

The Kearney pit requires approximately 150,000m^3 or 300,000t of waste fill to establish the floor for the ramp. This includes a 15% compaction factor. The ramp, which will form the floor of the adit will be graded to 1:-6 or 16.7%.

Once this floor has been established, ground support works will be required along the western wall. An area of 2000m^2 will be meshed and bolted to reduce the risk of rock fall before equipment and personnel commence work on the lower portal and adit.

Ground support consists of double twisted hexagonal PVC costed mesh pinned to the face of the wall using 1.8m resin grouted bolts. This provides additional support to the face as well as securing the surface support.

In unison with the ground support works on the western wall, a 3m high bund is erected on both the west and east sides of the pit floor to delineate the path of the adit. The containment bunds protect personnel, equipment and infrastructure from rock falls.

The bunds are constructed from waste rock and are located a minimum of 3 metres from pit walls and will be 3 metres high. The Kearney pit has shallow angle pit walls so it is unlikely that any potential rock fall will gather sufficient speed to breach a well-positioned and designed bund wall. Given that the pit wall profile in much of the pit is less than the 45° angle of repose for loose rock material, any spalling or rock fall from the walls should be slow moving and further retarded by the friction caused from the sub 45° profile.

The installation of rock bunds are a safety measure to collect any material that may be dislodged during operations in the pit. Having bunds in place will also centralize travel through the pit and maintains a minimum 10 metre roadway, through which the adit will eventually be constructed.

An overview of the Kearney pit rehabilitation plan can be seen in the figure below (Figure 41).

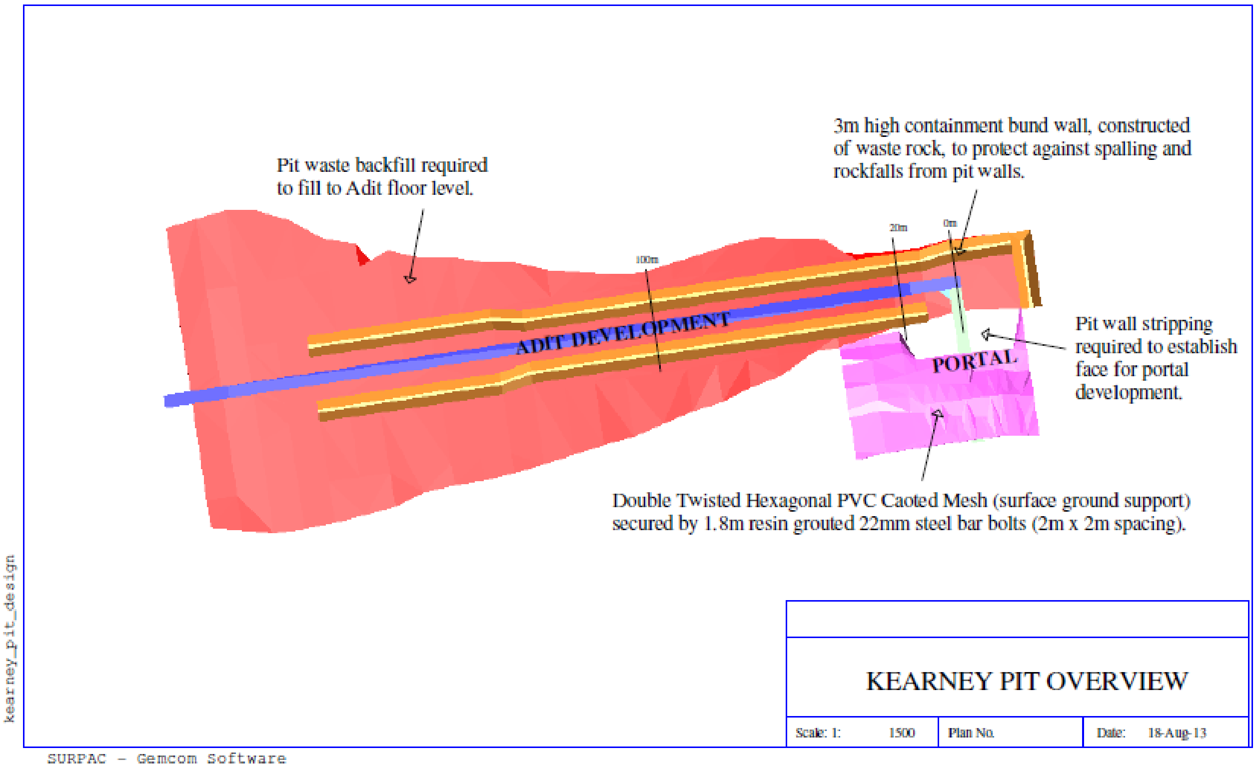


FIGURE 41: KEARNEY PIT REHABILITATION OVERVIEW

17.4.2 MINE ADIT

The concrete adit is constructed using prefabricated galvanised steel arches that will be secured to a 400mm reinforced concrete floor. The arches are spaced at 1.0m intervals and have a mesh and shotcrete covering between them. The adit dimensions are 4.0mW x 4.0mH and has been scaled down from the engineered design shown in the figure below (Figure 42)

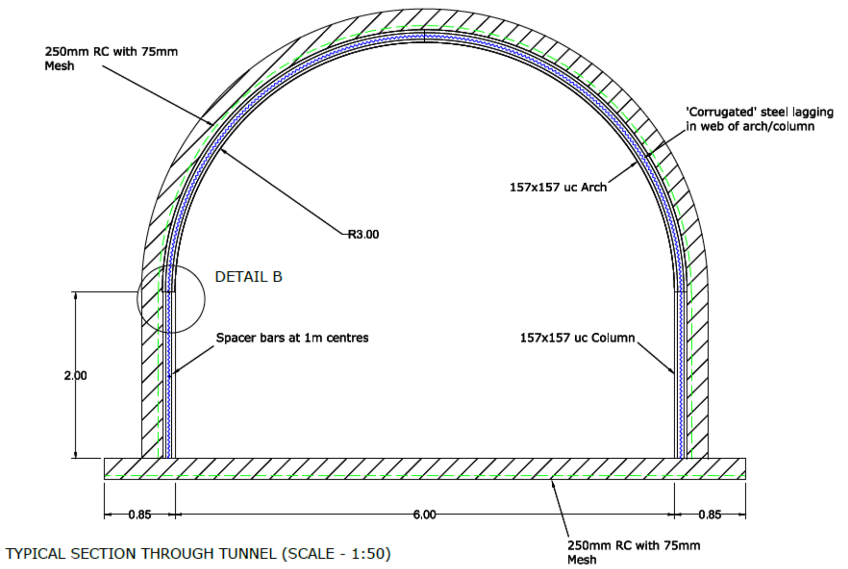


FIGURE 42: MINE ADIT CROSS SECTION

The total length of the adit at a gradient of 16.7% is 240m.

The walls and roof of the adit will be formed with 250mm fibrecrete and 3mm steel mesh with a minimum strength of to 40MPa or prefabricated arch segments that can be bolted and secured in place on the concrete foundation.

Once the adit is constructed, waste rock generated from the underground mine along with surface waste stockpiles and retreated tailings will be used to fill the Kearney pit back to the surface level. This will require approximately 650,000m^3 or 1.3 million tonnes of fill material.

17.5 GEOTECHNICAL PARAMETERS

Omagh Minerals Limited engaged SRK Consulting to develop a high level conceptual underground design criteria for development drives and stope spans for an underground mine.

Geological logs, downhole surveys and core photographs of exploration holes were provided which intersect the HW and FW of the ore bodies. Analysis of core photographs, information collected during previous open pit mining studies and prior knowledge of shear hosted gold deposits have been used to classify the rock mass.

Immediate mining units consist of metasedimentary rocks that range in strength from 10MPa to 40MPa based on field estimates. The variable strength is a result of shear zones characterised by low strength rocks. There are generally three joints sets that are in the main planar with infill ranging from clean joints to weak infill sheared areas.

17.5.1 ROCK MASS CHARACTERISATION

Rock Mass Characterisation has been calculated using,

1. Q Parameter Estimates based on the Norwegian Geotechnical Institutes Tunnelling Quality Index.
2. Bieniawski's Rock Mass Rating System (RMR) and Laubscher's Modified Rock Mass Rating (MRMR)

17.5.2 Q - QUALITY RATING

Underground mine design support estimation uses the Q – Quality Index. The factors considered are displayed below,

$$Q = \frac{RQD}{Jn}\, x \frac{Jr}{Ja}\, x\, \frac{Jw}{SRF}$$

The inputs are included below,

- Rock Quality Designation (RQD) is rated as *20-60 – poor*
- Joint Roughness (Jr) is rated at *1.5 – rough or irregular, planar*
- Joint Set Number (Jn) is rated as *12 – three joints sets plus random*
- Joint Alteration (Ja) is rated as *3 – silty or sandy clay coatings, small clay fraction*
- Joint Water Reduction Factor (Jw) is rated as *0.66 – medium inflow or pressure, occasional outwash of joint fillings*
- Stress Reduction Factor (SRF) is rated as *2.5 – Low stress, near surface*

These inputs were calculated for the orebody, FW and HW rock masses. A summary of the Q values is shown below (Tables 30 and 31).

Parameter	Hangingwall	Orebody	Footwall
RQD	28	41	58
Jn	12	12	12
Jr	1.5	1.5	1.5
Ja	3	3	3
Jw	0.66	0.66	0.66
SRF	2.5	2.5	2.5
Q Value	0.3	0.5	0.7
Rock Mass Description	Very Poor	Very Poor	Very Poor
Q' value	1.1	1.7	2.5

TABLE 30: Q ROCK QUALITY RATING

17.5.3 RMR AND MRMR – ROCK MASS RATING SYSTEM

	Rating Values		
	Hangingwall	**orebody**	**Footwall**
1 - Strength	4	4	4
2 - RQD	8	10	14
3 - Spacing	8	8	8
4 - Condition	15	15	15
J persistence	2	2	2
J aperture	3	3	3
J roughness	3	3	3
J infilling	3	2	3
J weathering	4	4	4
5 - Groundwater	10	10	10
RMR	45	47	51
Adjustments	-7	-7	-7
RMR	38	40	44
Description	Poor Rock	Fair Rock	Fair Rock
Class number	IV	III	III
RMR - MRMR adjustments			
	Hangingwall	**orebody**	**Footwall**
Weathering deterioration	98%	98%	98%
Shear zones	85%	85%	85%
Joint Orientation	90%	90%	90%
Blasting	98%	98%	98%

TABLE 31: RMR AND MRMR ROCK MASS RATING

17.5.4 MINE DESIGN PARAMETERS

A Q rating of Good (>10) is recommended for shrinkage stoping, with only occasional spot bolting required for localised support. The Q rating for the HW and FW should be at least fair (>4). Below this, Poor quality HW and FW rock masses can unravel during drawdown of broken ore. Additional stope support is limited to short bolts and light mesh, but can improve support once the broken ore, the overriding support factor (which has limited effectiveness) has been drawn down to create a void.

As the Q ratings for the HW, FW and Ore rock masses are below 4, additional stope support will be required within the HW and backs of developed stopes to maintain stable conditions. Unsupported spans will be restricted according to the local rock quality.

The maximum stable stope dimensions and support requirements have been determined using the Modified Stability Graph Assessment and Modified Rock Tunnelling Quality Q' Value.

17.5.5 MODIFIED STABILITY GRAPH ASSESSMENT

Each if the three domains were assessed individually. The Stability Number (N') was used to determine the Hydraulic Radius. The Hydraulic radius is a function of the Length (L) and Height (H) of the stope.

$$HR = \frac{Area\ of\ Stope}{Perimeter\ of\ Stope} = \frac{L\ H}{(2L + 2H)}$$

17.5.6 MODIFIED ROCK TUNNELLING QUALITY Q' VALUE

To calculate the Modified Stability factor (N') the following has been taken into consideration.

$$N' = Q' x\ A\ x\ B\ x\ C$$

1. Q' – Q values previously calculated for the three domains (Table – 6.1.1)
2. A – Rock Stress Factor (7 for stopes and 11 for HW and FW)
3. B – Joint Orientation factor (0.5 for stopes and 0.7 for HW and FW)
4. C – Gravity Adjustment Factor (2 for stopes and 7 for HW and FW)

The table below (Table 32) displays the factors and calculated N' values.

Parameter	Stope Wall (F/W)	Stope Back (Ore)	Stope Wall (H/W)
Q'	2.5	1.7	1.1
A	1.0	0.7	1.0
B	0.7	0.5	0.7
C	6	2	6
N'	10.5	1.19	4.62

TABLE 32: N VALUES FOR UNDERGROUND ROCK TYPES

17.5.7 STOPE SPANS

Hydraulic Radii have been calculated for a combination of stope dimensions. Tables were created to display the stability graph analysis for each of the three domains and were used to calculate the most suitable stope dimensions (Table 33).

Hangingwall

Hydraulic Radii for Various Stope Geometries											
Vertical	Stope Length (m)										
Stope Height	5	10	15	20	25	30	35	40	45	50	55
5	1.27	1.71	1.92	2.06	2.14	2.21	2.25	2.29	2.32	2.35	2.37
10	1.69	2.54	3.06	3.41	3.66	3.85	3.99	4.11	4.21	4.29	4.36
15	1.89	3.04	3.81	4.37	4.79	5.12	5.38	5.59	5.77	5.92	6.05
20	2.01	3.37	4.35	5.09	5.66	6.13	6.50	6.82	7.09	7.32	7.52
25	2.10	3.61	4.75	5.64	6.36	6.95	7.44	7.86	8.22	8.53	8.80
35	2.20	3.92	5.30	6.44	7.40	8.21	8.90	9.51	10.04	10.50	10.92
45	2.26	4.12	5.67	7.00	8.13	9.12	9.99	10.76	11.44	12.06	12.61
55	2.30	4.25	5.94	7.40	8.69	9.82	10.84	11.75	12.57	13.31	13.99

N'= 4.62

Key		Hydraulic Radii (m)
	Stable	4.2
	Unsupported Transitional	6.50
	Stable with Support	9.50
	Supported Transitional	11.50
	Unstable	11.50
	Ore body dip	75.00

Footwall

Hydraulic Radii for Various Stope Geometries											
Vertical	Stope Length (m)										
Stope Height	10	10	15	20	25	30	35	40	45	50	55
5	1.71	1.71	1.92	2.06	2.14	2.21	2.25	2.29	2.32	2.35	2.37
10	1.69	2.54	3.06	3.41	3.66	3.85	3.99	4.11	4.21	4.29	4.36
15	1.89	3.04	3.81	4.37	4.79	5.12	5.38	5.59	5.77	5.92	6.05
20	2.01	3.37	4.35	5.09	5.66	6.13	6.50	6.82	7.09	7.32	7.52
25	2.10	3.61	4.75	5.64	6.36	6.95	7.44	7.86	8.22	8.53	8.80
35	2.20	3.92	5.30	6.44	7.40	8.21	8.90	9.51	10.04	10.50	10.92
45	2.26	4.12	5.67	7.00	8.13	9.12	9.99	10.76	11.44	12.06	12.61
55	2.30	4.25	5.94	7.40	8.69	9.82	10.84	11.75	12.57	13.31	13.99

N'= 10.50

Key		Hydraulic Radii (m)
	Stable	6.3
	Unsupported Transitional	9.00
	Stable with Support	11.25
	Supported Transitional	13.10
	Unstable	13.10
	Ore body dip	75.00

Stope Back - Ore body

Hydraulic Radii for Various Stope Geometries											
Ore body Thickness	Stope Longitudinal Span (m)										
Stope Width	5	10	15	20	25	30	35	40	45	50	55
0.5	0.23	0.24	0.24	0.24	0.25	0.25	0.25	0.25	0.25	0.25	0.25
1	0.42	0.45	0.47	0.48	0.48	0.48	0.49	0.49	0.49	0.49	0.49
2	0.71	0.83	0.88	0.91	0.93	0.94	0.95	0.95	0.96	0.96	0.96
3	0.94	1.15	1.25	1.30	1.34	1.36	1.38	1.40	1.41	1.42	1.42
4	1.11	1.43	1.58	1.67	1.72	1.76	1.79	1.82	1.84	1.85	1.86
5	1.25	1.67	1.88	2.00	2.08	2.14	2.19	2.22	2.25	2.27	2.29
10	1.67	2.50	3.00	3.33	3.57	3.75	3.89	4.00	4.09	4.17	4.23
15	1.88	3.00	3.75	4.29	4.69	5.00	5.25	5.45	5.63	5.77	5.89

N'= 1.19

Key		Hydraulic Radii (m)
	Stable	2.3
	Unsupported Transitional	4.29
	Stable with Support	7.30
	Supported Transitional	9.20
	Unstable	9.20
	Ore body dip	0.00

TABLE 33: STOPE SPAN STABILITY TABLES

Stope dimensions selected are 30m strike length and up to 35m in height between levels. The maximum width is 3m for the purposes of this calculation. A stope with dimensions 30mL x 35mH x 3mW has the following classification by domain,

- Stope Back – Stable (0.94)
- HW – Stable with Support (8.21)
- FW – Transitionally Unsupported (8.21)

The calculated requirement is for HW support only. However to reduce risk and further improve safety, ground support will be installed in all three domains. This will further reduce the likelihood of ground failure.

17.5.8 DEVELOPMENT

The main access decline, along will all capital development and infrastructure will be located in the FW. From the Q Value analysis in 17.5.2 the FW domain has a higher Q rating but is still classed as very poor. Ground support in the three domains will be tailored to the conditions encountered.

All development will be reduced to the minimum practical dimensions for the mining method used. This is further discussed in Section 1*7.6.3 – Development Ground Support*.

17.6 UNDERGROUND MINE LATERAL DEVELOPMENT

Underground lateral development will be completed using single boom jumbo mounted drills and explosives.

17.6.1 LATERAL DEVELOPMENT ASSUMPTIONS

The assumptions used for Lateral Development calculations are listed in Table 34 and display the two most common development profiles which account for 75% of the planned total development.

	Drive Dimensions		Comments
	2.7mW x 3.8mW	3.2mW x 3.8mH	
Face Area (m^2)	10.26	12.16	Cross sectional area of development
Jumbo bore length	3.7	3.7	
Round Length (m)	3.5	3.5	Rounds should pull these distamces, does not include stripping and drag cuts.
Overbreak (m)	0.15	0.15	150mm overbreak around the excavation, equal to approximately 10%
Density	2.8	2.8	Ore and waste density the same
Tonnage per round	115.6	137.0	
1.8m Resin Bolts Per Round	4	2	Back Bolts (Alternating rows of 2 x 1.8m and 3 x 2.4m with 1.75m row spacing)
1.8m Split Set Bolts Per Round	4	4	Wall Bolts (Two per row with 1.75m row spacing)
2.4m Resin Bolts Per Round	0	3	Back Bolts (Alternating rows of 2 x 1.8m and 3 x 2.4m with 1.75m row spacing)
Resin Cartridges per Round	5	5	In Capital Development Only (One slow set and one fast set resin cartridge)
Mesh per Round	0	0	Sheets of mesh per cut
Shotcrete Surface Area (m^2)	25.0	25.0	Average Area to be shotcreted
Shotcrete Allowance (m^3)	3.7	3.7	Average Shotcrete thickness of 150mm
Intersection drilling 64mm (m)	36.6	36.6	Cable bolt holes drilled to 6.1m average
Rehab/Stope Cable bolt support (m)			Will allow 5% of ground support costs for Rehab
Holes per face (45mm)	38	42	Average holes per face per dimensions
Holes per face (102mm)	5	5	Reamer/Relief holes drilled
Perimeter Holes	15	15	Perimeter holes - 0.8m spacing around perimeter - Charged with Low density explosive (50%
Lifter Holes	5	5	Lifter holes per face - Charged with 700mm packaged explosive (normally wet holes)
Drilling metres (45mm)	140.6	155.4	Holes per face * drill steel length
Drilling metres (102mm)	18.5	18.5	Holes per face * drill steel length
Ground Support Drilling metres (33mm)	8.3	12.4	28mm Resin bolts optimum hole size is 33mm
Ground Support Drilling metres (38mm)	8.3	8.3	39mm Split Set bolts optimum hole size is 38mm
Total Ground Support Drilling metres	16.6	20.7	Ground support holes drilled 15% longer than required.
Hole Volume (m^3)	0.005	0.005	45mm diameter hole area * round length - (0.3m uncharged collar + 0.2m primer)
Main Holes ANFO per round (Kg) ($\rho = 0.95$)	87.0	106.4	Holes charge with ANFO = total holes drilled - (perimeter holes and lifters)
Perimeter Holes ANFO per round (Kg) ($\rho = 0.95$)	22.3	27.3	Perimeter holes only charged 1m from butt with Anfo
Total ANFO per round (Kg) ($\rho = 0.95$)	109.4	133.7	Perimeter holes only charged 1m from butt with Anfo
Packged Explosive Lifter (400mm)	43.8	43.8	Lifter holes (5 x 700mm cartridges per hole)
Packaged explosive primer (200mm)	33	37	1 primer per hole except lifters
Detonator	38.0	42.0	1 detonator per hole

TABLE 34: MINE DEVELOPMENT ASSUMPTIONS

17.6.2 DEVELOPMENT DIMENSIONS

Development dimensions have been carefully considered to reduce the amount of waste generated in capital and access development whilst minimising dilution in the ore drives. Table 35 shows the development dimensions for various types of drive that will be required.

	Minimum Dimensions (m)		Gradient		
Development Type	Width	Height	Ratio	%	Profile
Adit	4.0	4.0	6	16.7	Arch
Upper Decline	4.0	4.0	25	4.0	Arch
Main Decline	3.2	3.8	8	12.5	Arch
Footwall Drive	3.2	3.8	75	1.3	Arch
Ventilation Drive	3.2	3.8	50	2.0	Arch
Stockpile/Remuck	3.5	3.5	50	2.0	Square
Crosscut/Access	3.2	3.8	10	10.0	Arch
Ore Drive	2.7	3.8	75	1.3	Arch
Drawpoint	3.0	3.0	100	1.0	Square
Refuge	1.8	3.0	100	1.0	Square
Sump	2.5	3.0	6	16.7	Square

TABLE 35: MINE DEVELOPMENT DIMENSIONS

17.6.3 DEVELOPMENT GROUND SUPPORT

Development ground support will involve the use of the following recognised support techniques.

- Fibrecrete
- Galvanised Steel Friction Bolts (1.0m and 1.8m lengths)
- Galvanised Steel Resin Grouted Bolts (1.8m and 2.4m lengths)
- Galvanised Steel Welded Wire Mesh

The minimum standard for 'supported ground' in the underground mine combines surface support reinforced with bolts. Therefore the ground will be deemed 'unsupported' if it does not have either of the following:

- Galvanised Steel Welded Wire Mesh secured with either Friction or Resin Bolts, or
- Fibrecrete secured with either Friction or Resin Bolts.

The following table (Table 36) has been produced to reflect the anticipated ground support requirements for the mine based on similar ground conditions and dimensions experienced in other operating mines around the world.

	Min Dimensions (m)		Bolt Requirements Per 3.5m Round			Fibrecrete	
Development Type	**Width**	**Height**	**2.4m Resin**	**1.8m Resin**	**1.8m Friction**	**Depth (mm)**	**(m^3)**
Upper Decline	4.0	4.0	3	2	4	150	4.5
Main Decline	3.2	3.8	3	2	4	150	3.7
Footwall Drive	3.2	3.8	3	2	4	150	3.7
Ventilation Drive	3.2	3.8	3	2	4	150	3.7
Stockpile/Remuck	3.5	3.5	3	2	4	150	4.7
Crosscut/Access	3.2	3.8	3	2	4	150	3.7
Ore Drive	2.7	3.8	0	0	8	75	1.9
Drawpoint	3.0	3.0	0	0	8	75	1.8
Refuge	1.8	3.0	0	0	0	150	4.0
Sump	2.5	3.5	0	4	4	75	1.7

TABLE 36: DEVELOPMENT GROUND SUPPORT REQUIREMENTS

17.6.4 RESIN GROUTED BOLTS

Resin grouted bolts are installed by inserting a fast setting resin cartridge into a drill hole and driving a bolt through it. Once the resin mixes and sets to form an anchor at the toe of the hole the bolt can be tensioned to increase its load bearing capacity. When coupled with a slow setting resin the bolt can achieve much higher load bearing capacity.

The resin bolts selected will be installed with a fast set resin anchor at the toe of the hole which allows the bolt to be almost immediately tensioned. A slower setting resin will be used in the remainder of the hole which allows the bolt to be fully encapsulated and bond to the surface for greater support and as such work particularly well in weaker rock types.

Resin grouted steel bolts will be used in capital installations, nominally any development that has a lifespan of over 2 years, such as the main decline and storage areas. These are less susceptible to corrosion if fully encapsulated so are ideally suited for this purpose.

17.6.5 FRICTION BOLTS

Friction bolts, or split sets as they are known, provide robust and economic ground support options. They are installed by driving them into a slightly smaller diameter hole. The bolt then closes to provide a tight bond with the surface of the hole. The friction of the rock surface against the steel gives it its load bearing properties.

39mm diameter split set bolts will be used, driven into a 38mm hole as per manufacturer's guidelines. The bolts are expected to have a minimum load bearing capacity of up to 9.1 tonnes for a 1.8m bolt when installed correctly and fitted with a 150mm x 150mm plate.

These bolts will be galvanised to reduce corrosion and will mainly be used in openings that are expected to be used for less than 2 years. They will also be used for additional wall support in capital openings.

17.6.6 FIBRECRETE

Fibrecrete is a surface support installation that involves spraying a concrete mix onto the exposed rock surface. It has high strength characteristics, nominally +30 MPa at 28 days, along with excellent surface bond and durability. The additional of high strength, high modulus, synthetic plastic fibres further increases durability.

Typically the mix contains the following products:

Component	Mix	t/m^3
8-10mm Aggregate	23.5%	0.550
0-2mm Sand	47.0%	1.100
Cement (Cem1)	19.2%	0.450
Water	8.7%	0.203
Fibres	0.3%	0.006
Stabilizer	0.1%	0.003
Plastaciser	0.1%	0.003
Accelerator	1.1%	0.025
	100%	**2.3**

TABLE 37: TYPICAL FIBRECRETE MIX COMPONENTS

The mix results in an average density of approximately 2.3 t/m^3.

A mechanised spray applicator will be used to apply the shotcrete. Shotcrete supply will be from a surface batching plant and will be transported to development areas by transmixer.

17.6.7 WELDED MESH

Welded mesh will be used in addition to Fibrecrete in areas that need additional surface support. It will also be used for a contingency in times of fibrecrete supply disruption.

Areas of high heavy vehicle traffic, such as cross cut intersections and loading bays, may require additional support or rehabilitation.

17.6.8 GROUND SUPPORT MANAGEMENT PLAN

A comprehensive Ground Support Management Plan (GCMP) will be considered prior to commencement of mining operations. A GCMP is a detailed document, reviewed annually, which outlines the various types of ground support used at the mine. It should be compiled by a qualified Geotechnical Engineer and use empirical data to recommend the most effective types and patterns of ground support for any given conditions encountered in the mine.

The GCMP can then be used as a reference document for all employees and supervisors so there is a clear standard to be adhered too.

17.6.9 DEVELOPMENT DRILLING

Development drilling is completed using a single boom jumbo mounted drill rig, such as the Atlas Copco T1D Boomer. Faces are drilled using a combination of 45mm blast holes and 102mm relief or reamer holes.

The number of holes required per face range depending on the development dimensions but is limited to a maximum burden and spacing of 1.0m. Perimeter holes have a reduced burden of 0.8m.

17.6.10 DEVELOPMENT CHARGING

Development charging is completed using a charging platform. The platform is coupled to an interchangeable utility vehicle.

The charging platform consists of an operator basket, 350kg capacity ANFO kettle and pressure regulator assembly. It allows ANFO to be air driven into the blast holes resulting in a density of approximately 0.95kg/m^3. Exploration drilling results indicate dry ground conditions so emulsion and wet application explosives have been discounted at this stage.

Development powder factors range between 1.0 to 1.2 Kg/t.

ANFO charged holes are initiated using a high explosive emulsion cartridge coupled with a half second delay detonator. This allows sequential blasting and is expected to reduce peak particle velocity vibration in the surrounding rock types, thus reducing the impact on the immediate community and infrastructure surrounding the operation.

17.7 UNDERGROUND MINE VERTICAL DEVELOPMENT

Vertical development will be excavated by two main methods. A raise bore will be used for the large diameter shafts and raises required for ventilation whilst smaller diameter production Alimak raises; mined using hand held drills will be used to access stopes. Long hole raises will be used for Sublevel Longhole Stoping.

17.7.1 VENTILATION SHAFTS AND RAISES

The main ventilation shaft and raises will be 3.1m diameter. This will be the main exhaust for the mine ventilation system and will connect individual levels via shorter raises where required. Utilising a raise bore for the excavation of these shafts and raises will increase stability and longevity of these vital capital components.

17.7.2 EMERGENCY EGRESS AND LADDERWAY RAISES

Ladderways will be installed to provide a secondary means of egress. These will be located in raises throughout the mine. A typical ladderway system can be seen in Figure 43. Man-riding facilities will also be provided in the main emergency egress shaft, in accordance with UK Mines and Quarries Act requirements. This will be comprised of a raise climber type system. A number of manufacturers are available.

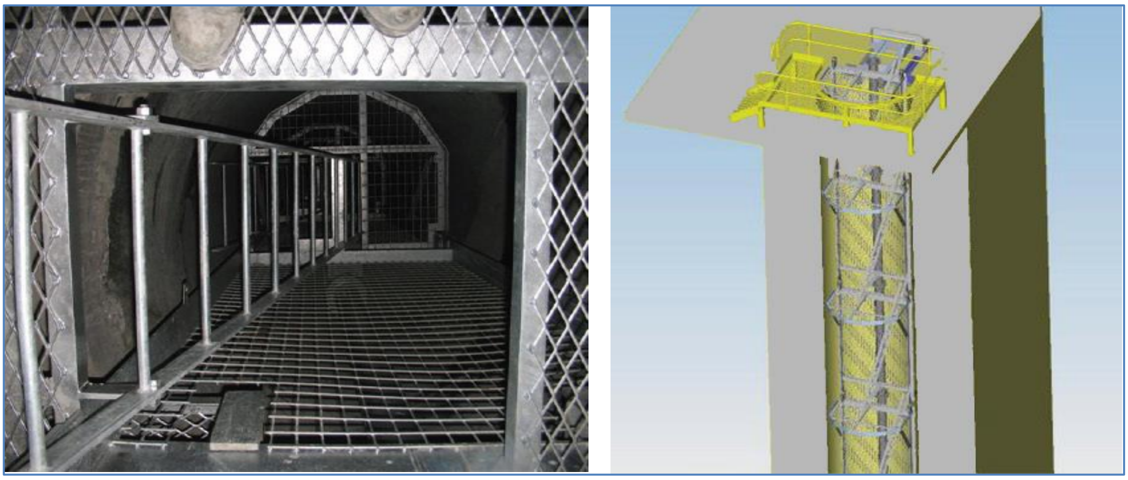


FIGURE 43: MINE EMERGENCY LADDERWAY SYSTEMS

The Ladderways will be installed by bolting and securing 6m prefabricated segments. These sections have a separate ladder and landing level with a trapdoor entry which provides a safe resting platform. Ladderway arrangements such as in Figure 43 (Left) will be used and will require installation in a raise bored at a maximum of 70^{o} to reduce the effort required to climb. This arrangement allows installation in smaller diameter raises.

The Ladderways shown above in Figure 43 (Right) allow for installation in vertical raises due to the ladder inclination of 60-70^{o} within the segment.

The Ladderways are installed quickly and provide instant egress. They are inexpensive to maintain and allow simultaneous evacuation for multiple personnel with minimal resistance and disruption to air flows.

17.7.3 MAN WAYS AND ORE PASSES

Manways will be hand mined using air leg drills. These will be located at either end of a stoping block and will be compartmentalised using timber to allow a ladder in one side and ore to pass in the other. These will be 1.0m-1.2m square profile and will be drilled and blasted.

17.7.4 STOPING RAISES

A central raise will be located in each of the Shrinkage Stoping blocks and will be used for access to the stoping round via an Alimak raise climber (Figure 44). These raises will be 1.5m square profile and will allow vertical transport of men, equipment and consumables used in the stoping cycle.

FIGURE 44: ALIMAK RAISE CLIMBER

Longhole raises will be drilled to open slots for Sublevel Longhole Stopes. The Sublevel Longhole Stope raises will be drilled using a production drill with 64-76mm blast holes and 102-152mm relief holes.

17.8 UNDERGROUND MOBILE EQUIPMENT

Underground mobile equipment has been selected to operate safely within the development dimensions specified in Section 17.6.2 whilst adhering to legislative requirements. Equipment has also been selected to meet the production profile of the mine.

The mobile equipment requirements, along with delivery schedule, are displayed in Table 38.

Mobile Equipment	YEAR 1				YEAR 2				YEAR 3				TOTAL
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	
Single Boom Jumbo Drill		1	1		1	1							4
Loader - 2m^3		1	1		1								3
Truck- 8.5m^3			1		1								2
Agitator Truck- 4.5m^3		1											1
Explosive Charging Platform		1											1
Fibrecrete Spraymec		1											1
Utility Loader		1			1								2
Light Vehicles	2				3								5

TABLE 38: MOBILE MINING EQUIPMENT REQUIREMENTS

During Q1 of Year 1, equipment is ordered with current lead times of up to 12 weeks. The equipment is required for commencement of mining operations in Q2, when surface works and rehabilitation of the Kearney pit is completed.

17.9 VENTILATION

Ventilation requirements will change throughout the LOM. The primary system to be installed is designed to satisfy the highest demand period during peak production of 225,000 tpa.

During initial development, fresh air from the surface will be ducted to the working face using an 110kW axial ventilation fan. Fumes will exhaust via the main decline and portal. This will provide sufficient volume and flow to allow the initial decline development to progress up to 2km using low loss ventilation duct.

Once the development reaches the 1105 RL, a 35m vent raise (1.5m diameter) will be raise bored to assist the exhaust network. This can be fitted with a ladder & raise climber to provide an alternate means of egress for the upper section of the mine.

Ventilation flows will be measured periodically to ensure that all working areas meet the legislative requirements. Dust exposure and air contaminants will be internally and independently measured to ensure compliance to prescribed HSE legislation.

17.9.1 PRIMARY VENTILATION

The primary ventilation network will involve the installation of the main 3.1m diameter ventilation shaft. This will be located to the east of the orebody and will provide a means of exhaust that will draw fresh air into the mine via the main decline directly to working levels. The exhaust raise system is extended via shorter internal raises as the mine develops deeper.

Ventilation requirements are primarily linked to exhaust emissions from the mobile equipment operating in the mine. The ventilation requirements are calculated in Table 39.

Mobile Equipment	Engine Rating (kW)	LOM Units	Total Engine Rating (kW)	Vent Flow Factor (m^3/s per kW)	Vent Flow Per Unit (m^3/s)	Total Required (m^3/s)
UG Loader - 4t	86	3	258	0.06	5.2	15.5
UG Haul Truck - 17t	168	2	336	0.06	10.1	20.2
Single Boom Jumbo	55	4	220	0.06	3.3	13.2
Utility Loader	63	2	126	0.06	3.8	7.6
UG Agitator - 4.5m^3	168	1	168	0.06	10.1	10.1
Shotcrete Aplicator	95	1	94.5	0.06	5.7	5.7
Utility 4WD	106	5	530	0.06	6.4	31.8
	741		***1733***			***104.0***

TABLE 39: VENTILATION REQUIREMENTS

Fan efficiency and friction loss are taken into consideration when selecting the Primary Fan capacity. The resulting total requirement at peak demand is 139 m^3/s.

17.9.2 SECONDARY VENTILATION

Secondary ventilation will provide fresh air to the main working areas. A combination of 45-55kW fans force air into the level to dilute contaminants caused by: diesel emissions, explosives fumes and dust. Low friction ventilation ducting will be used with diameters varying between 1067mm – 1400 mm depending on development dimensions.

Ventilation ducting will be provided to within 30m of the working face to ensure that personnel and equipment have suitable airflow for diluting contaminants and cooling purposes.

17.9.3 DIESEL EXHAUST EMISSIONS

Diesel Exhaust Emissions are to be monitored to ensure compliance to Health and Safety Legislation. Exhaust emissions will be periodically tested during routine maintenance servicing.

Monthly ventilation surveys will be carried out to measure exhaust emissions in secondary networks.

Exposure surveys will be carried out by an independent contractor on a quarterly basis to measure exposure levels to personnel over a shift.

17.9.4 DUST

Dust is generated during mining activity. Key areas of dust generation are drilling, blasting, mucking, crushing and conveying.

All drilling is wet to supress dust generation. Once blasted, development muck piles and drives are watered to supress dust during subsequent movement of the material. Sprays are installed at stope draw points whilst mucking and crushing and conveying will utilise fog sprays at transfer points.

Dust exposure is also to be independently monitored on a quarterly basis along with the diesel exhaust emissions.

17.9.5 EXPLOSIVE FUMES

Explosive fumes will be diluted through primary and secondary ventilation flows. Standard clearance times allow for fumes to clear and hand held monitors will be used during re-entry to the area to monitor air quality.

17.10 STOPING

A range of mining methods were considered during the feasibility process. The two most suitable methods to be considered for implementation are Shrinkage Stoping and Sub Level Longhole Stoping. Both methods of ore extraction are to be used depending on the geometry of the ore veins. The stopes once mined, will be back filled using development waste rock or a cemented paste comprising of tailings that have been stockpiled from past mining activity. Both stoping methods will be modernised to minimise traditional safety risks whilst maximising production.

Mine development will consist of a main decline mined in the FW of the orebody that will be used to access the ore body via short crosscuts into the veins. Sill levels will be spaced at 18m intervals which will allow 36m shrinkage stopes and Longhole stopes of up to 15m.

17.10.1 SUBLEVEL LONGHOLE STOPING (SLS)

Sublevel Longhole Stoping is the preferred mining method for ore extraction. SLS will be fully mechanised to optimise safe production of ore. The sublevel benches will be determined by ground conditions and could vary locally from 6.9-14.2m in length.

Due to limited geotechnical information on the various ore bodies and individual veins, the Feasibility study has reduced all SLS heights to 10.7m.

Bench heights can be varied by increasing or decreasing the development for the stope to include flat backing and floor benching. The development required to extract the veins in this fashion have been included in the Feasibility Study design.

Blasthole diameter, burden and spacing will be dependent on the vein widths. Table 40 displays a variety of blasting parameters that could be used for this type of SLS.

Vein Width (m)	Hole Diameter (mm)	Burden (m)	Powder Factor (Kg/t)
1.5	51	1.00	0.92
2	51	1.25	0.55
2.5	64	1.50	0.88
3	64	1.50	0.74

TABLE 40: BLASTING PARAMETERS FOR LONGHOLE STOPING

The patterns suggested for this type of mining are 'Dice 5' and 'Dice 7' patterns. Given the narrow nature of the individual veins it is suggested burden is varied to obtain a powder factor below 0.5 Kg/t, as long as this does not affect blast performance. Powder factors are based on pour loaded ANFO density of 0.8kg/m^3.

17.10.2 SHRINKAGE STOPING

The Shrinkage Stoping method will be used where vein widths narrow or are highly variable in width and dip. Veins between 0.75-1.5m wide will be mined using this stoping method, with up to 36m vertical distance between levels. This method of extraction will follow the traditional shrinkage stoping principle, but has been updated to implement a number of safer practices to overcome the shortcomings of traditional shrinkage stoping.

Development for the shrinkage stopes will consist of a sill drive, an extraction drive 6.0m below the sill (forming a sill pillar) and short draw points located along the extraction drive.

The stope will be accessed from the extraction drive via three raises which will be installed in the draw points. There will be three raises per stope, one at either end of the stope and one in the centre.

An Alimak raise climbing platform will be used to install the raises and will remain in the central raise for access into the stope when stoping commences, and for delivery of consumables and equipment to the bench. The raises at either end of the stope will be used as ore passes when mucking the swell material, they will provide ventilation and will be compartmentalised to allow a ladder to be installed for emergency egress. It is expected that one of raises will be salvaged and kept open for the next stope in the sequence.

Once vertical development has been completed, stoping will commence. Initially a back strip of the ore drive will be taken using a jumbo drill rig to provide the broken floor for the shrinkage stope. The swell from the initial blast will fall into the sill drive.

Anti-vibration handle stopers will be used to drill and install ground support once scaling of the backs has been completed and drill between the HW and FW contacts for charging.

The 38mm blast holes will be primed with a cartridge explosive and detonator and blow loaded with ANFO. Once fired the broken ore will be scraped to the ore passes to provide a level working floor. Ore will not be drawn from directly below the stope to ensure that there is no possibility of hang ups or voids which would compromise the safety of personnel and equipment working of the shrinkage floor.

The stoping cycle will continue until the upper limit of the stope. Once the stope has been fully mined, ore will be scraped to the ore passes and into the draw point. Ore will be loaded into trucks within the extraction drive and trucked to the underground crusher.

The stope will then be backfilled using a cemented paste delivered from a surface backfill plant that mixes tailings and cement and is pumped to the stope. Once filled, the backfill will be allowed to cure and then mining of the next stope in the sequence will continue.

17.11 BACKFILLING

Stopes will be backfilled using rough waste generated through development or a designed paste mixture. The paste mixture will consist of tailings, stockpiled in storage facilities from previous surface mining activity, and cement. Typically cement ratios will be between 6.5-13.0% depending on the strength and curing times required.

An instantaneous fill rate of 50m^3 /hr at 50% utilisation is required to meet the backfill capacity of 125,000m^3 pa. This is geared to maintain ore production of 225,000 tpa.

The backfill plant will operate for 16 hours per day but will only fill for 12 hours of this to allow time to commence pasting and to flush the system at the end of the shift. Therefore a maximum of 600m^3 of paste will be delivered to stopes per day.

17.11.1 PASTE FILL PROCESS

Tailings generated from previous processing will be harvested and stockpiled at the paste plant. Cement will be delivered to a silo and water will be pumped to a holding tank by the plant and a flushing tank by the hopper.

A loader will feed the tailings hopper and this will be transferred to the mixing tank by conveyor which can control the tonnage and moisture content.

A metered amount of cement and water will be added to the mixing tank to meet the required paste specification.

The paste, once mixed, will be transferred into a paste pump which will push the paste into the underground pipeline. A pump is required as the head is insufficient to move paste over the required strike length.

Paste will be produced at 50m^3/hr at 250 Pa yield stress at around 75% solids by weight.

17.11.2 PASTE RETICULATION

Paste will be pumped through a 250mm diameter schedule 40 steel pipeline located in the ventilation shaft or through dedicated cased boreholes. At each level, a schedule 80 transfer pipe will be located that can be used to divert the paste to the required level. Pressure sensors will be located along the pipeline to give real time information to the paste plant control room.

Once paste is delivered to the fill level, the pipeline can be reduced to 200mm diameter. The final 100m of the paste line into the stope can be reduced from steel to PN16 HDPE piping to simplify installation.

17.11.3 PASTE BACKFILL OF STOPES

Once a stope has been mined and has been cleared for backfilling, a paste barricade must be created. This is an engineered retaining structure that is constructed at the brow of the open stope and must retain the paste until it gains sufficient strength to be self-supporting.

17.11.4 BACKFILL BARRICADES

Paste barricades will be constructed using mesh, hessian lining and fibrecrete. Pinholes with rebar will be installed in the walls and brow and steel wire will be used to secure the mesh between them. A hessian lining will be used to help the fibrecrete bond to form a wall.

The diagram below (Figure 45) shows a typical paste wall, constructed of two layers of 3mm mesh and 16mm rebar.

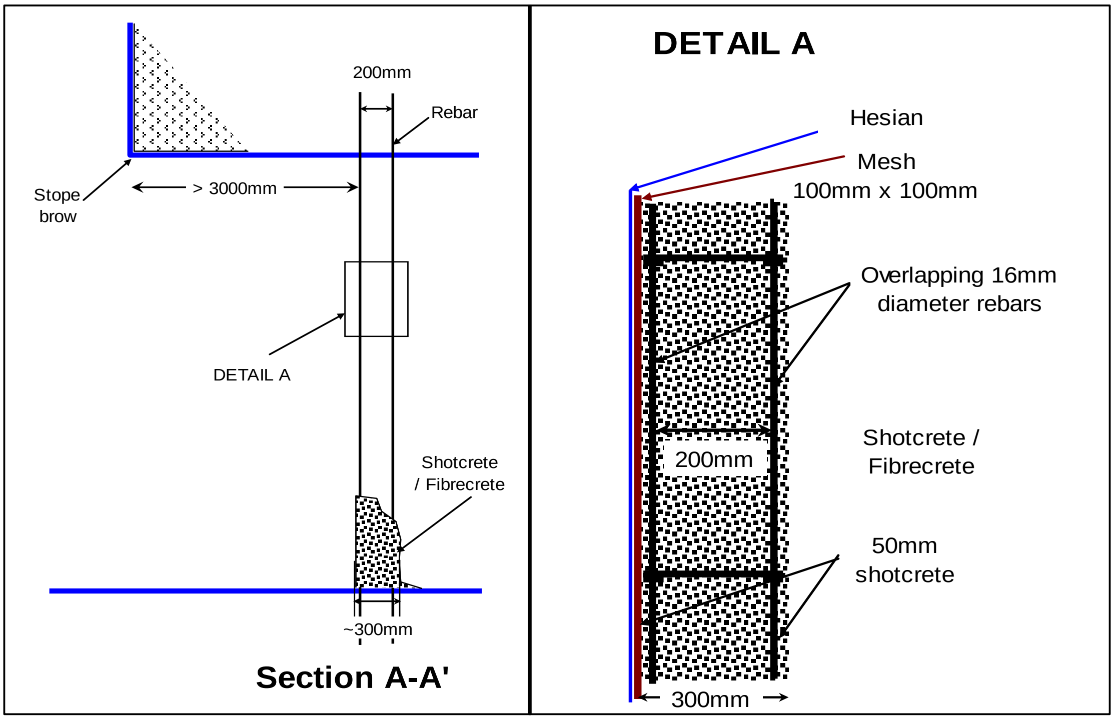


FIGURE 45: BACKFILL BULKHEAD WALL DESIGN

Once the initial pour has been completed to approximately 5m above the brow, the paste should be allowed to cure for 7 days to prevent liquefaction during subsequent pours. This will form a solid past plug which will support the pressure exerted by paste in subsequent pours.

Drainage pipes are recommended at the base of the wall to allow excess water to be drained.

17.11.5 CEMENT CONTENT

Detailed test work is required to calculate the required cement content for the paste fill and this will vary for various scenarios. Paste calculations have been based on the assumption that 6.5% cement will be required for paste in general stopes, raising to 13% if the paste is to be exposed as a sill pillar (Table 41).

Component	Density (t/m^3)	Fill	Fill (t/m^3)
Cement	1.44	6.5%	0.09
Tailings	2.00	53.5%	1.07
Water	1.00	40.0%	0.40

TABLE 41: PASTEFILL MIX COMPONENTS

17.12 UNDERGROUND INFRASTRUCTURE

17.12.1 MINE POWER

Mine power will be supplied from surface diesel generators or mains power.

Electricity will be fed by 11kV cables to the underground sub stations and distribution boards and reduced to 1kV for powering jumbo boxes, pumps and fans.

17.12.2 COMPRESSED AIR

A diesel compressor will be located on the surface and will feed the underground mine via 8'' pipeline. Compressed airlines will be reduced to a 4'' for the development drives.

17.12.3 MINE WATER

Mine water will be pumped underground from two holding tanks on the surface. These will be fed from the raw water storage pit.

17.12.4 UNDERGROUND CRUSHER

An underground jaw crusher will be installed in a crushing chamber located at the start of the main decline. Ore will be hauled by 17t articulated dump trucks that will dump onto an armoured face conveyor or apron feeder which will provide continuous feed to the crusher.

Crushing capacity of 50tph is required in line with mine production rates. A 75kW Crusher will be installed with >75% of material passing 50mm.

The crusher will direct feed onto the mine ore conveyors to the surface ROM storage shed.

17.12.5 ORE CONVEYORS

An ore conveyance system will be installed to convey ore from the underground crusher to the ROM storage shed.

The conveyors will be 600mm wide and powered by a variable speed drive motor unit ranging from 7.5kW – 30kW depending on belt length.

Each belt will be equipped with primary and secondary belt scrapers, galvanised dust covers, impact loading stations and safety monitoring devices including: belt tear/alignment, belt sequence, blocked chute and audible visual alarms. Dust suppression will be installed at all transfer points.

The total length of the conveyor from the crushing chamber to the ROM storage shed is 675m. Of this, 300m will be located underground. An additional 75m section will be required from a surface transfer for waste to be conveyed from the mine.

The route of the conveyor is depicted in Figure 46 (i).

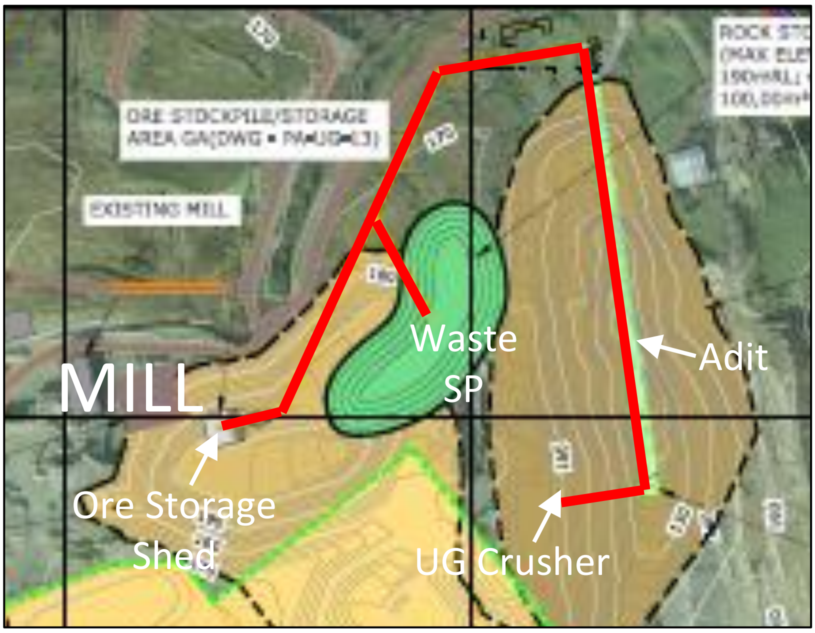


FIGURE 46 (i) : ORE CONVEYOR BELT ROUTE

Material will be transferred from the underground conveyor to the surface conveyor outside of the adit entry. Ore is then deposited into the ore storage shed next to the processing plant, whilst any waste that is conveyed will be diverted to the waste stockpile. The drawings below (Figure 46 (ii)) depict the transfer tower and ore storage shed.

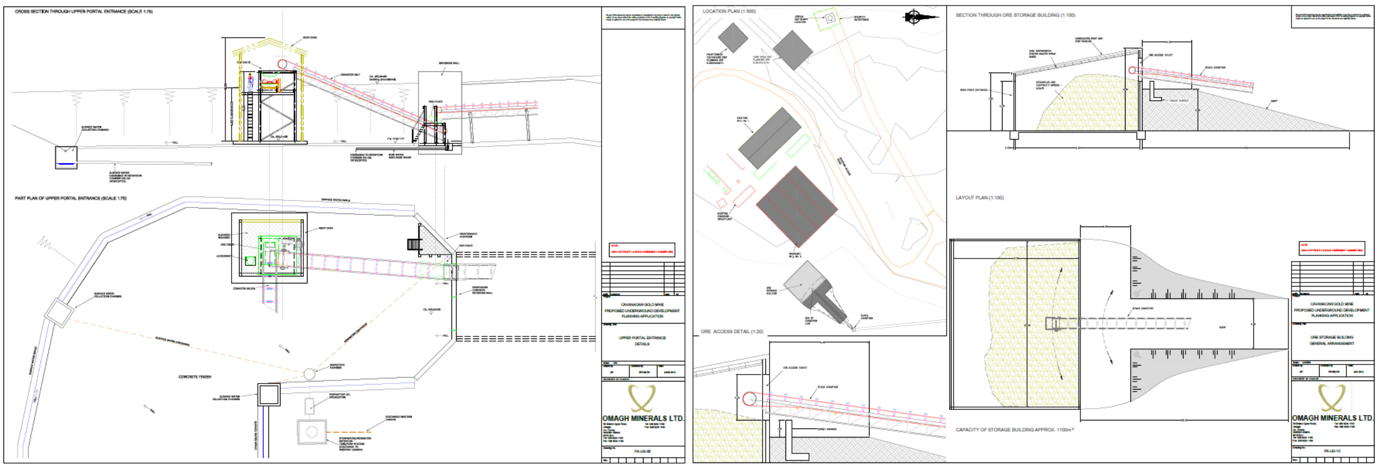


FIGURE 46 (ii): ORE CONVEYOR TRANSFER TOWERS AND ORE STORAGE SHED

17.12.6 SERVICE BAY

An underground service bay will be considered for daily servicing of mobile equipment. It will be used to carry out daily servicing such as refuelling, oil checks, greasing and minor repairs.

All engine, hydraulic and rock oils would be located in the service bay along with a grease injection gun.

A satellite station would be installed underground with a 2000 litre fuel pod that can be loaded in and out by a utility loader to be taken to the surface diesel bowser for filling. This would be used by the mobile equipment to negate the need to return to the surface once a shift for re-fuelling.

A 2000 litre capacity pod would have the capacity to fuel all the UG equipment required at peak production for three shifts. This can be further extended by ensuring trucks refuel at the surface rather than underground.

The service bay will be kitted with AFFF suppression equipment due to the storage of flammable liquids and will have an oil separator sump to prevent contamination of water supplies from oil spills.

17.13 LABOUR

Labour is to be sought from the local area with a minimum mix of 25% experienced underground miners to a maximum of 75% inexperienced backgrounds that can be trained to undertake the various roles at the mine.

The underground mine will operate three 8 hour shifts to ensure 24 hour operation Mon-Fri. The mine will carry out equipment servicing and maintenance on the weekend.

The processing facility will operate in line with the underground mine production.

	Y1	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**
Mining Personnel	31	64	64	66	67	53
Processing Personnel	14	22	22	22	22	18
Geology & Exploration Personnel	5	7	7	7	7	3
Administration Personnel	5	7	7	7	7	6
Total Personnel	**55**	**100**	**100**	**102**	**103**	**80**

TABLE 42 PERSONNEL REQUIREMENTS

17.14 MINING STUDY

The mining study has been estimated using the initial resource and utilising the most appropriate mining methods to design and cost a Life of Mine (LOM) schedule and budget. The mining parameters that have been considered are discussed below.

17.14.1 DILUTION

Dilution factors have been included in Mining Study calculations.

Reserve shapes may include both internal and external dilution.

Internal dilution is calculated within the reserve blocks from the Resource Model and is calculated as a composite tonnage and grade for the reserve shape.

External dilution is added to the composite tonnage and grade for the reserve shape. It is a percentage of the tonnes based on the method of mining added at zero grade. This increases the tonnage for the mining shape, reduces the overall grade but ounces contained remain constant.

Table 43 displays the estimated external dilution factors for the different mining methods.

	External Dilution Factor
Development Drives	10%
Longhole Stoping	20%
Shrinkage Stoping	20%

TABLE 43: DILUTION FACTORS APPLIED TO RESERVES

External dilution will be monitored and calculated as part of the stope reconciliation process. This is achieved by using a cavity monitoring system (CMS) to give a detailed 3D image of the mined volume. This is reconciled against the design volume and over break and under break is calculated. These dilution factors will be periodically reviewed and if necessary updated to reflect the actual value.

17.14.2 MINING RECOVERY

Mining recovery factors are included to take into account ore loss through mining activity. This generally affects stopes more than development as material can accumulate on the HW and FW of stopes or be left behind on the stope floor.

It is less likely to affect development as any ore accumulations on the floor would generally be recovered through the stoping process.

An estimate of Mining Recovery has been included in the Mining Study Estimate to account for these ore losses. Stopes have an estimated recovery factor of 95% whilst development has a slightly higher recovery estimated at 98%.

17.14.3 CUT OFF GRADE DETERMINATION

Mine Cut Off Grades have been calculated based on the mining schedule. This results in estimation of the Resource Cut Off Grade at a fixed gold price.

This can be further broken down by separating the fixed and variable components of mine expenditure to obtain an Incremental Cut Off Grade.

17.14.4 RESERVE CUT OFF GRADE (RCOG)

The RCOG reflects the minimum grade that can be mined based on economic factors. This can vary across a mine depending on mining method, depth and type of ore. The RCOG accounts for all costs incurred during mining and processing of ore and refining of concentrate. Therefore this is directly affected by operating costs and gold price.

17.14.5 INCREMENTAL CUT OFF GRADE (ICOG)

The ICOG is used to analyse ore blocks that fall below the RCOG but has to be mined to access existing reserves. Once broken, the mining cost reduces as a significant proportion of the cost is drilling and blasting. The incremental grade can ignore these costs and be measured against the costs incurred to haul, process and refine the already broken ore.

17.14.6 CUT OFF GRADES

The graph below (Figure 47) displays the COG in relation to the gold price.

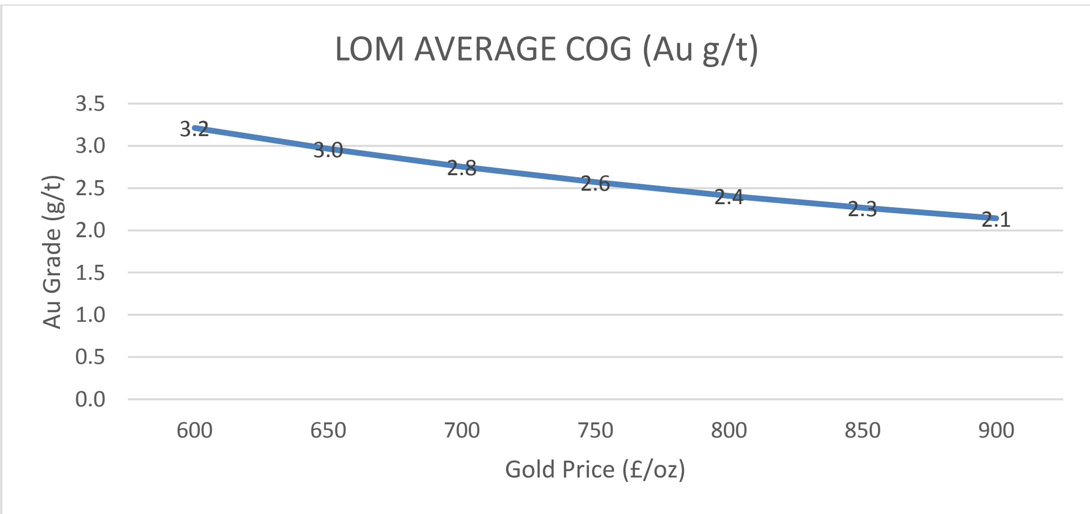


FIGURE 47: MINING COG VERSUS GOLD PRICE

Economic studies in the feasibility study have been calculated at a prevailing gold price of £700, £750 & £800/oz (Table 52). This is further discussed in Section 21.

17.14.7 MINING STUDY ESTIMATE

The Mine Study Resources have been created taking into account the above factors at a cut off grade of 2 g/t of gold. The LOM schedule has been designed with the inclusion of Inferred resource, which has been subject to the same parameters used to consider estimated Measured and Indicated resources.

The inclusion of Inferred resource is appropriate for the economic study as there are parts of the orebody when mined, will include Measured and Indicated Resources, along with Inferred resource that has to be mined as it is either included in development drives or is encapsulated within stoping blocks. The inclusion of Inferred material has been minimised so far as is considered practical within the mining plan and economic study. As such it equates to 36.7% of the total mine plan. That said, 64% of Inferred material occurs in the last 3 years of the mine life (Table 45). That time gap permits additional drilling from surface or underground, if required to confirm the additional data to re-assess the Inferred mineralisation. In some circumstances, routine face sampling of development drivages or raises can provide the additional sampling data. The percentage of material mined per year is shown below.

	Measured Resources			Indicated Resources			Inferred Resources			Mining Recovery
Year	**Tonnes**	**Grade**	**Ounces**	**Tonnes**	**Grade**	**Ounces**	**Tonnes**	**Grade**	**Ounces**	
1	2,390	2.84	218	42,662	5.70	7,819	6,658	7.41	1,586	98.0%
2	13,257	4.39	1,872	106,105	5.56	18,951	48,428	4.63	7,209	97.0%
3	21,018	5.05	3,410	121,053	5.18	20,155	81,996	4.87	12,843	96.2%
4	27,645	5.64	5,014	119,881	4.98	19,179	56,367	5.72	10,369	95.5%
5	27,887	4.77	4,273	93,568	5.17	15,546	73,991	6.61	15,722	95.2%
6	870	3.50	98	43,504	5.78	8,091	57,967	6.93	12,906	95.1%
Total	**93,066**	**4.97**	**14,886**	**526,774**	**5.30**	**89,742**	325,407	5.80	60,635	

TABLE 45 (i): ANNUAL MINE PLAN BY RESOURCE CLASS

	Measured	Indicated	Inferred
Year	**Ounces Mined (%)**	**Ounces Mined (%)**	**Ounces Mined (%)**
1	2.3%	81.3%	16.5%
2	6.7%	67.6%	25.7%
3	9.4%	55.4%	35.3%
4	14.5%	55.5%	30.0%
5	12.0%	43.7%	44.2%
6	0.5%	38.4%	61.2%
Total	**9.0%**	**54.3%**	**36.7%**

TABLE 45 (ii): ANNUAL MINE PLAN BY RESOURCE CLASS (PEA)

Total Measured and Indicated resources in the mining schedule are 104,627 oz's of gold which form 63.3% of the estimated 6 year mine life of 165,263 oz's gold.

17.15 MINE PRODUCTION

The Life of Mine (LOM) production plan is outlined in Table 49.

17.15.1 MINE LAYOUT

Mining will initially commence on only two of the known ore bodies. The Kearney and Joshua ore bodies will be mined concurrently over the forecast 6 year mine life. The main decline into the mine, from the adit will branch to service both areas (Figure 48).

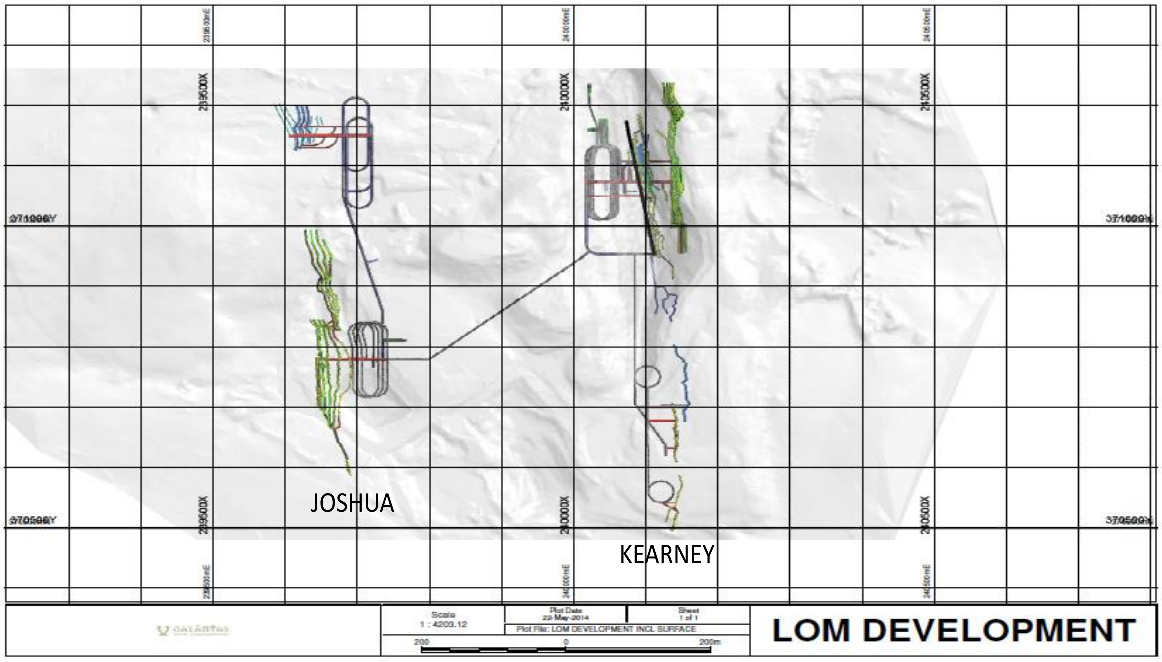


FIGURE 48: PLAN VIEW OF MINE DEVELOPMENT

17.15.2 KEARNEY OREBODY

Mining occurs predominantly in the northern end of the Kearney orebody where multiple veins run parallel to each other. Mining focuses on Measured and Indicated Reserves and as such further definition and exploration drilling will be completed to upgrade Inferred resources. Access to the mine is via the adit through the Kearney pit, highlighted in black in the figure below (Figure 49).

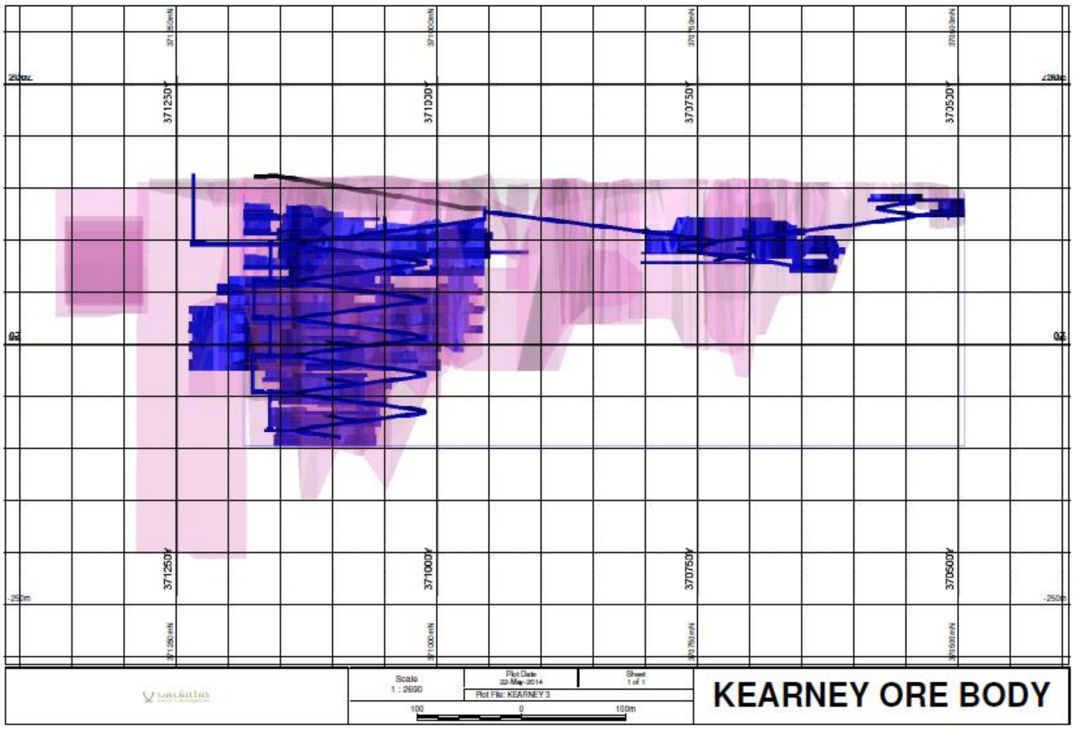


FIGURE 49: MINE PLAN FOR THE KEARNEY ORE BODY

The development required to access the Kearney scheduled resources is shown in Table 46 (below).

Kearney Development Metres	1	2	3	4	5	6	TOTAL
Decline	852	462	893	262	0	0	**2,469**
SP	75	45	75	45	0	0	**240**
Crosscut Access	351	369	431	61	0	0	**1,212**
Refuge Cuddy	85	46	89	26	0	0	**247**
Sump	22	12	23	7	0	0	**63**
Ventilation Drive	187	110	55	30	0	0	**382**
Sill Drive	914	904	889	533	0	56	**3,296**
Sill Drive Waste	50	62	110	69	0	39	**329**
Floor Benching Access	72	97	98	59	0	0	**326**
Floor Benching	700	908	960	281	0	112	**2,962**
Floor Benching Waste	16	74	43	4	0	24	**161**
Shrinkage Access	0	0	97	117	0	0	**214**
Shrinkage Sill	0	0	230	203	0	0	**433**
Shrinkage FWD	0	0	181	203	0	0	**384**
TOTAL (m)	**3,323**	**3,089**	**4,174**	**1,900**	**0**	**232**	**12,717**

TABLE 46: DEVELOPMENT TO ACCESS KEARNEY RESERVES

17.15.3 JOSHUA OREBODY

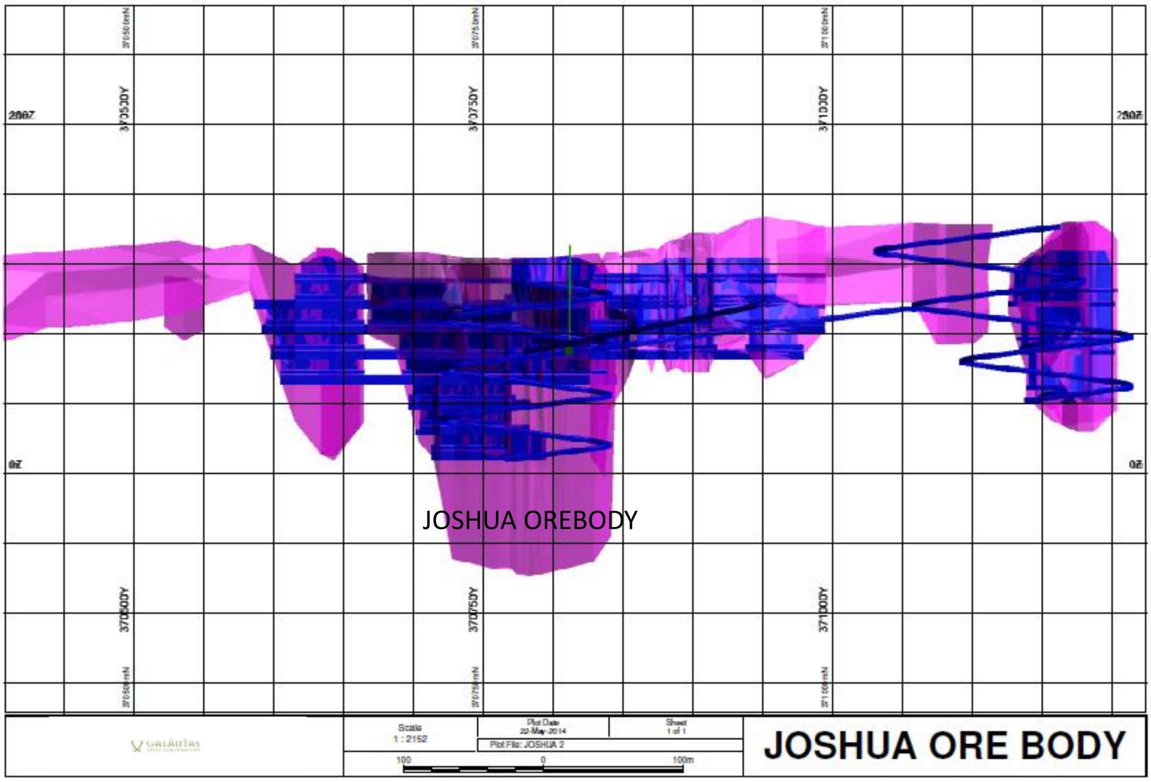


FIGURE 50: MINE PLAN FOR THE JOSHUA ORE BODY

The proposed mining method in the Joshua orebody is Longhole stoping in the wider central area of the known resource, expanding northwards to extract some shallow narrow vein material using shrinkage stoping. The incline to the North allows for a second adit to be mined to provide dual decline access to the mine (Figure 50). The northern incline will provide multiple diamond drilling platforms to further delineate veins and analyse the Inferred resource between the two mining fronts. Alternatively a second raise bored shaft may be utilised. The total development required to access the Joshua scheduled resources is shown in Table 47.

Joshua Development Metres	1	2	3	4	5	6	TOTAL
Decline	597	363	446	892	359	0	**2,657**
SP	15	60	45	75	45	0	**240**
Crosscut Access	83	179	203	0	0	0	**465**
Refuge Cuddy	60	36	45	89	36	0	**266**
Sump	15	9	11	23	9	0	**68**
Ventilation Drive	30	0	0	0	0	0	**30**
Sill Drive	44	1,553	562	0	0	0	**2,158**
Sill Drive Waste	0	103	35	0	0	0	**138**
Floor Benching Access	0	47	63	0	0	0	**110**
Floor Benching	0	850	754	45	0	0	**1,649**
Floor Benching Waste	0	74	14	0	0	0	**88**
Shrinkage Access	0	0	0	200	264	141	**605**
Shrinkage Sill	0	0	0	0	530	159	**689**
Shrinkage FWD	0	0	0	0	216	151	**366**
Flatback Access	0	0	0	0	46	46	**93**
Flatback	0	0	0	0	237	237	**475**
Flatback Waste	0	0	0	0	0	0	**0**
TOTAL (m)	**844**	**3,275**	**2,177**	**1,323**	**1,743**	**734**	**10,096**

TABLE 47: DEVELOPMENT TO ACCESS THE JOSHUA SCHEDULED RESOURCES

17.15.4 MINING RATES

The following mining rates have been determined to produce a mining schedule. These are based on operating performance of equipment, cycle durations and previous mine operating experience.

17.15.5 MINE OPERATING HOURS AND SHIFT DURATIONS

The mining operation will operate three 8 hour shifts, 5 days per week, 48 weeks per year.

During the course of an 8 hour shift the available working duration is significantly lower. Shift start up meetings, blasting re-entry, travelling to and from the work place, breaks and safety checks will all need to be included within the 8 hour period.

Therefore it is assumed that the average available working duration is only 5.3 hrs per shift.

17.15.5.1 EQUIPMENT AVAILABILITY

Equipment Availability of 81% has been used for this study. The calculation includes daily servicing and pre start equipment checks and an allowance of 1.5 hrs per shift has been allocated for this which is in line with industry standards.

Servicing will be completed on weekends by an external contractor as part of any leasing agreements so is not included within the availability calculations.

17.15.5.2 EQUIPMENT UTILISATION

Equipment Utilisation of 66% has been used for this study.

A standard calculation of utilisation of 49mins/hr has been used to allow for breakdowns and operator breaks, travel to and from equipment.

The 66% utilisation is calculated by taking 49mins/hr (81%) and applying it to the 6.5 hours the equipment is available each shift (taking into account the 81% equipment availability calculated in 17.15.1).

Each piece of equipment is expected to work 66% of the shift, or 5.3 hrs.

17.15.6 DAILY AND ANNUAL ORE PRODUCTION

Annual mine production is expected to peak at 225,000 tonnes per annum. To mine 35,000 to 40,000 equivalent gold ounces per annum the mined grade at full capacity must exceed 5.0 g/t. Table 48 (below) displays the annual and daily production rates require to meet the LOM plan.

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM AVERAGE
Annual Ore Production (t)	51,711	168,299	224,383	203,880	195,672	102,437	**157,730**
Monthly Ore Production (t)	4,309	14,025	18,699	16,990	16,306	8,536	**13,144**
Daily Ore Production (t)	198	644	858	780	748	392	**603**
Ore Mined Grade (g/t)	5.8	5.2	5.0	5.3	5.6	6.4	**5.4**
Mined Ounces	9,622	28,026	36,411	34,568	35,531	21,098	**27,542**

TABLE 48: MINE PRODUCTION (PEA)

17.16 MINING SCHEDULE

The mining schedule includes the resources enumerated in Table 45(i). Inferred resource that must be mined to reach existing measured and indicated resources has been included in the LOM plan and economic study.

17.16.1 MINING PHYSICALS

The LOM physicals (Table 49) generated by applying the mining parameters outlined in this document show:

- 946,381 tonnes of ore mined and processed at a head grade of 5.43 g/t Au.
- 148,729 ounces Gold recovered
- 23.3 km of mine development

	Production Profile By Year						
	1	2	3	4	5	6	TOTAL
Lateral Development (m)	3,379	4,312	4,601	2,837	1,675	601	**17,405**
Floor Stripping / Flatbacking (m)	788	2,051	1,930	389	284	420	**5,862**
Vertical Development 1.5m (m)	50	85	0	0	0	0	**135**
Vertical Development 3.1m (m)	65	93	62	0	0	0	**220**
Development Waste (t)	94,925	76,563	106,436	38,578	0	3,341	**319,843**
Development Ore (t)	51,711	127,447	105,222	36,399	22,059	5,246	**348,083**
Longhole Stope Ore (t)	0	40,852	119,161	167,482	121,310	48,405	**497,209**
Shrinkage Stope Ore (t)	0	0	0	0	52,303	48,785	**101,089**
Total Ore Mined (t)	51,711	168,299	224,383	203,880	195,672	102,437	**946,381**
Total Mined (t)	146,635	244,862	330,819	242,459	195,672	105,777	**1,266,224**
Average Grade Au (g/t)	5.79	5.18	5.05	5.27	5.65	6.41	**5.43**
Gold Mined (oz's)	9,622	28,026	36,411	34,568	35,531	21,098	**165,255**
Ore Processed (t)	51,711	168,299	224,383	203,880	195,672	102,437	**946,381**
Mill Feed Grade (g/t)	5.79	5.18	5.05	5.27	5.65	6.41	**5.43**
Mill Recovery (%)	90%	90%	90%	90%	90%	90%	**90%**
Gold Recoved (oz's)	8,659	25,223	32,770	31,111	31,978	18,988	**148,729**

TABLE 49: ANNUAL MINE PLAN PHYSICALS (PEA)

17.17 INCLUSION OF INFERRED RESOURCE

As previously stated, 36.7% of the LOM schedule above is Inferred resource. The Inferred material included in the economic study and LOM plan represents only 20.6% of the estimated Inferred resource for combined Kearney and Joshua orebodies.

The following cross section (Figure 51) has been included to show how the part of the Kearney orebody is scheduled to be mined and demonstrates how the associated Inferred resource is mined in relation to Measured and Indicated resources.

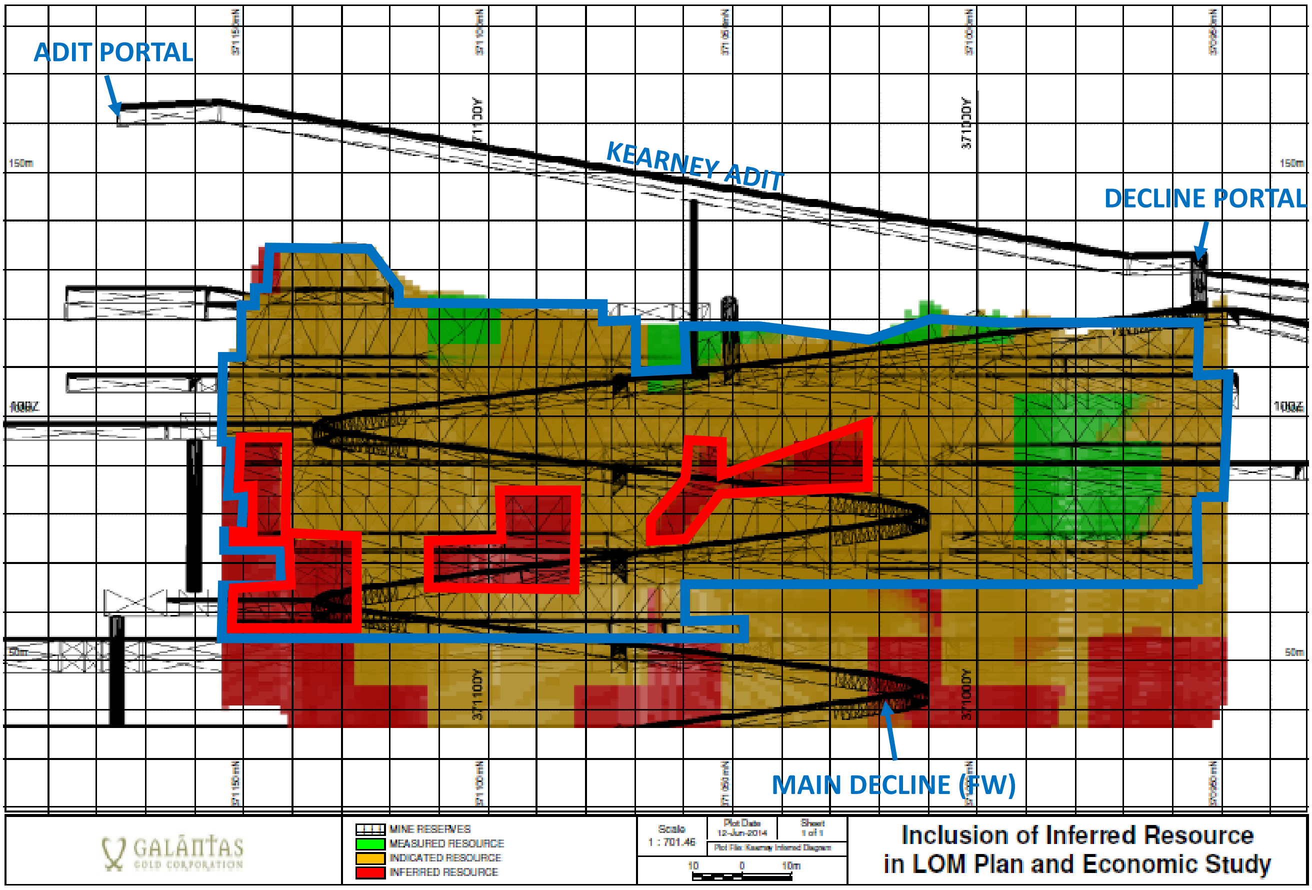


FIGURE 51: INCLUSION OF INFERRED RESOURCE

The Kearney ore body has 21 near parallel veins. One of these veins, shown as a north-south vertical cross-section in Figure 52, extends below the proposed mining section (outlined in blue) and demonstrates potential, with additional drilling, to extend the LOM plan.

Inside of the planned mining area for this vein, there are 3 distinct islands of Inferred resource that are accessible whilst mining the Measured and Indicated resources.

Inferred resource that has been included in the LOM plan, as above, is to be mined under two scenarios. The first is from development drives that are required to be mined to access measured and indicated resources and secondly as mineral (Inferred) that is encapsulated within a stope that also includes mineral classified as measured or indicated.

Mining is scheduled to be centralised on the Measured and Indicated areas. Inferred resource outside of these general zones is currently excluded from the mining plan until further exploration, by means of development, underground drilling and / or definition drilling has been completed to upgrade the resource.

The Joshua orebody has fewer veins than the Kearney orebody. Mining of the Joshua orebody, like Kearney, is also concentrated around the Measured and Indicated resources, with Inferred material at depth and towards the south excluded from the mining plan.

In compliance with NI 43-101 2.3.3(a) " ***the preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically***

to have 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."

17.17.1 COMPARISON OF LOM SCHEDULE INCLUDING AND EXCLUDING INFERRED RESOURCE

A schedule (which can be compared to Table 49) has been created including only Measured and Indicated resources, to analyse the impact on the economics of the mine excluding associated Inferred material (Table 50).

	Production Profile By Year Excluding Inferred Material					
	1	**2**	**3**	**4**	**5**	**TOTAL**
Lateral Development (m)	3,379	4,313	4,601	2,837	1,770	**16,900**
Floor Stripping / Flatbacking (m)	788	2,051	1,930	673	137	**5,579**
Vertical Development 1.5m (m)	50	85				**135**
Vertical Development 3.1m (m)	65	93	62			**220**
Development Waste (t)	94,931	89,532	134,960	65,516	28,478	**413,417**
Development Ore (t)	45,873	88,145	64,470	27,944	14,591	**241,023**
Longhole Stope Ore (t)	0	37,643	110,534	131,005	66,216	**345,399**
Shrinkage Stope Ore (t)	0	0	0	0	54,137	**54,137**
Total Ore Mined (t)	45,873	125,788	175,005	158,949	134,944	**640,558**
Total Mined (t)	140,804	215,319	309,964	224,465	163,422	**1,053,976**
Average Grade Au (g/t)	5.45	5.20	4.67	4.92	5.38	**5.04**
Gold Mined (oz's)	8,036	21,023	26,283	25,164	23,348	**103,855**
Ore Processed (t)	45,873	125,788	175,005	158,949	134,944	**640,558**
Mill Feed Grade (g/t)	5.45	5.20	4.67	4.92	5.38	**26**
Mill Recovery (%)	90%	90%	90%	90%	90%	**9000%**
Gold Recoved (oz's)	7,232	18,921	23,655	22,648	21,014	**93,470**

TABLE 50: ANNUAL MINE PLAN EXCLUDING INFERRED RESOURCE

The effect on the LOM plan of excluding associated inferred resources is:

- Reduction in mine life by one year to 5 years
- 37% Reduction in recovered gold to 93,470 ounces
- 9% Rise in COG to 2.81 g/t
- Reduction in Revenue by £34.2 Million (estimated using the mid case gold selling price of £750 per ounce).
- Increase the estimated cost of gold from £394 to £464 per ounce.

The Capital requirements for a mining plan that excluded related Inferred resources (even though they are materially developed) would be 33% less (£7.8m) than that which included associated Inferred resources, due to reduced scale of output. However, in this particular context, to base the capital requirements of the project on Measured and Indicated resources only, could under-capitalise the mine and underestimate productive capacity.

It should also be noted that there is additional Inferred material that is potentially accessible from the planned development should definition drilling upgrade the resource to Measured or Indicated status. This will potentially lead to an extension of mine life and reduce the operating cost, as the capital infrastructure and access development costs will already have been incurred and amortised.

Under CIM / Canadian National Instrument 43-101 definitions the results of the detailed mining study excluding inferred resources are expected to meet the criteria for definition as a Feasibility Study.

18 MINERAL PROCESSING

18.1 PROCESSING FACILITY

The processing facility is fed with ore from the underground mine. Ore is deposited by ore conveyors into a storage shed that feeds directly to the processing facility. Ore in the storage shed has been previously crushed by the underground jaw crusher will be approximately 50mm.

Ore size is further reduced by a cone crusher to below 10mm. The crushed ore is screened and any oversize (+10mm) gets returned to the cone crusher whilst the undersize (-10mm to +1mm) is conveyed to the ore bins. Fines below 1mm will be pumped to a thickening tank and will later enter the regrind ball mill.

Crushing will be completed during the day and will deposit enough ore into the ore bins to allow for overnight processing.

Ore is conveyed from two ore bins under a magnet to remove any foreign objects. This conveyor is fitted with a belt weightometer which calculates mill throughput which is used in the reconciliation process. Ore is then transferred to a feed bin which allows for constant flow of material into the primary ball mill.

The primary ball mill grinds the material to 75% passing 180um and 40% passing 75um (Figure 52). Additional water is supplied into the primary ball mill from the thickening tank.

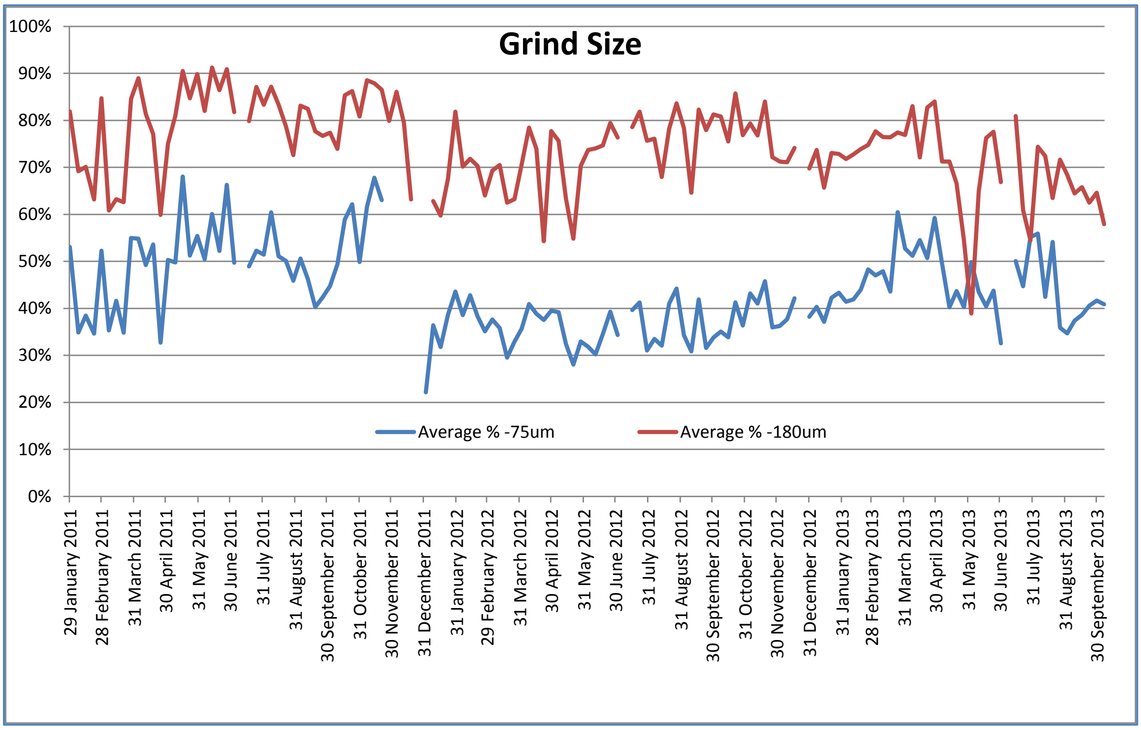


FIGURE 52: PREVIOUS GRIND SIZE OF MATERIAL FROM THE PRIMARY BALL MILL

The grind material is pumped to a cyclone where the overflow is pumped to the primary condition tank and the underflow feeds a gravity concentrator to remove free gold and coarser sulphide particles. The gravity concentrator sends concentrate to the holding tank whilst the overflow feeds into a screw classifier. The screw classifier is used to regulate water within the plant and the overflow and excess water from the screw classifier is sent to a thickener.

Underflow from the screw classifier is fed into the secondary ball mill for regrinding. The secondary ball mill and the thickening tank both discharge into a second cyclone that sends overflow to the first conditioning tank and underflow back into the secondary ball mill.

Copper Sulphate is added in the first conditioning tank to prepare the material for the float cells. The material is pumped into a second conditioning tank in which xanthates are added to further assist the froth flotation process.

The material is gravity fed into rougher banks below the second conditioning tank. Three inline banks, each containing 8 rougher cells, are used to float the sulphide material which is collected in the troughs and delivered to a two cleaner banks.

Remaining concentrate from the rougher banks is pumped through two scavenger banks, each containing 6 cells. Concentrate that is not collected in the scavenger banks passes into a final gravity separator before being lost to tails. The scavenged material is returned to the cleaner bank whilst the gravity recovered concentrate is pumped to the concentrate holding tank.

Concentrate from the cleaner banks, containing four cells, is pumped directly to the concentrate holding tank. A smaller scavenger tank is located at the end of the cleaner bank and is used to retrieve any remaining material and return it to the initial cleaner bank. Excess water is pumped back to the thickening tank.

Concentrate is to be filtered from the holding tank using two parallel filter presses which discharge the final concentrate into the bagging area below.

18.2 PROCESS FLOW SHEET

The process flow sheet is shown below and highlights the components of the processing cycle (Figure 53).

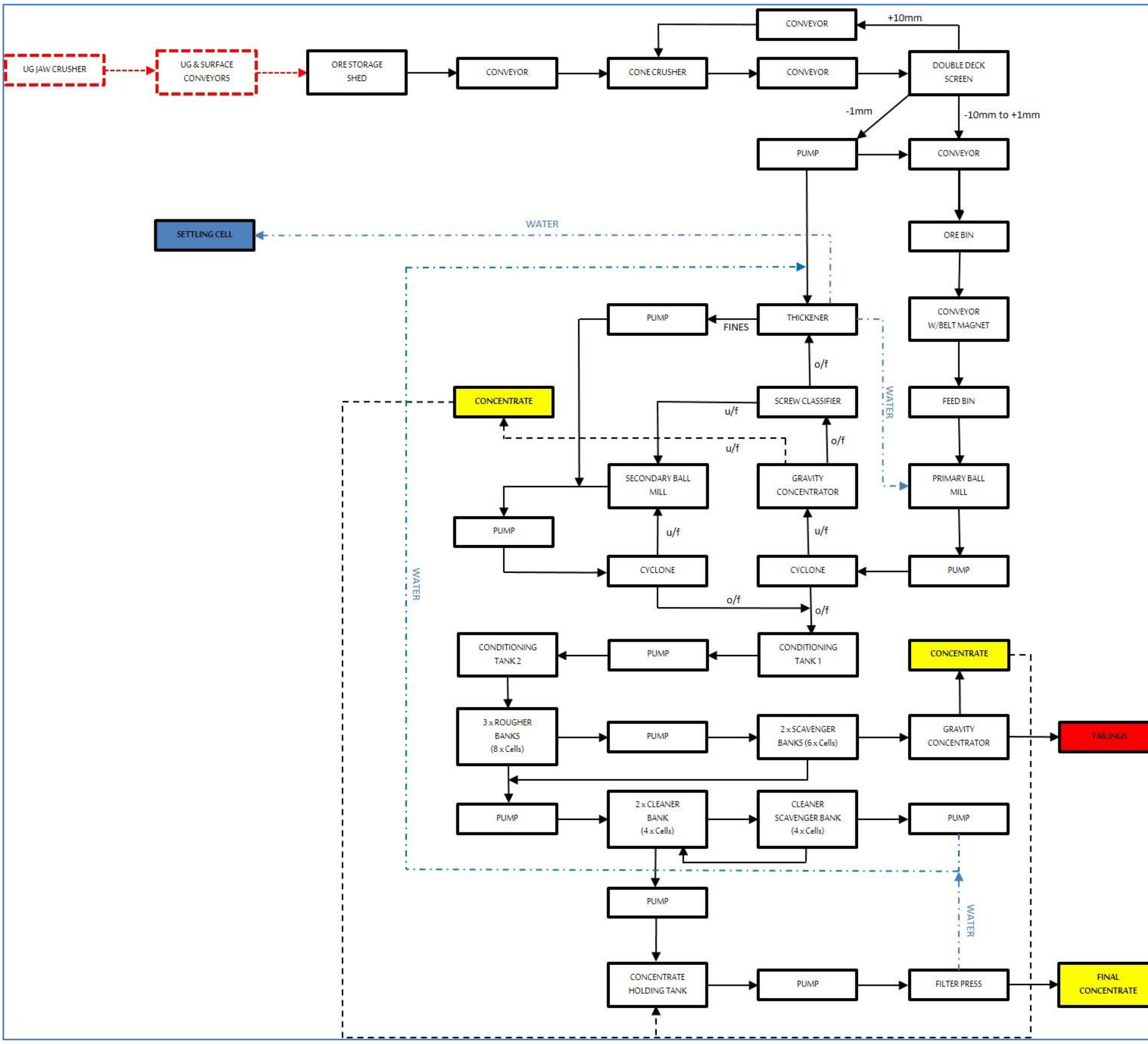


FIGURE 53: THE PROCESS FLOW SHEET

Concentrate is to be removed by the two gravity concentrators and the froth flotation circuit and all concentrate material is delivered to the concentrate holding tank. This is filtered and the resulting final concentrate is bagged for shipping to the smelter.

Tailings are sent to the paste cells with any excess water from the thickener gravity fed to the settling cell.

18.3 MILL RECOVERY

Mill recovery has been estimated at 90%. Past performance data from 2012-2013 (Figure 54) shows an average of 87 % recovery.

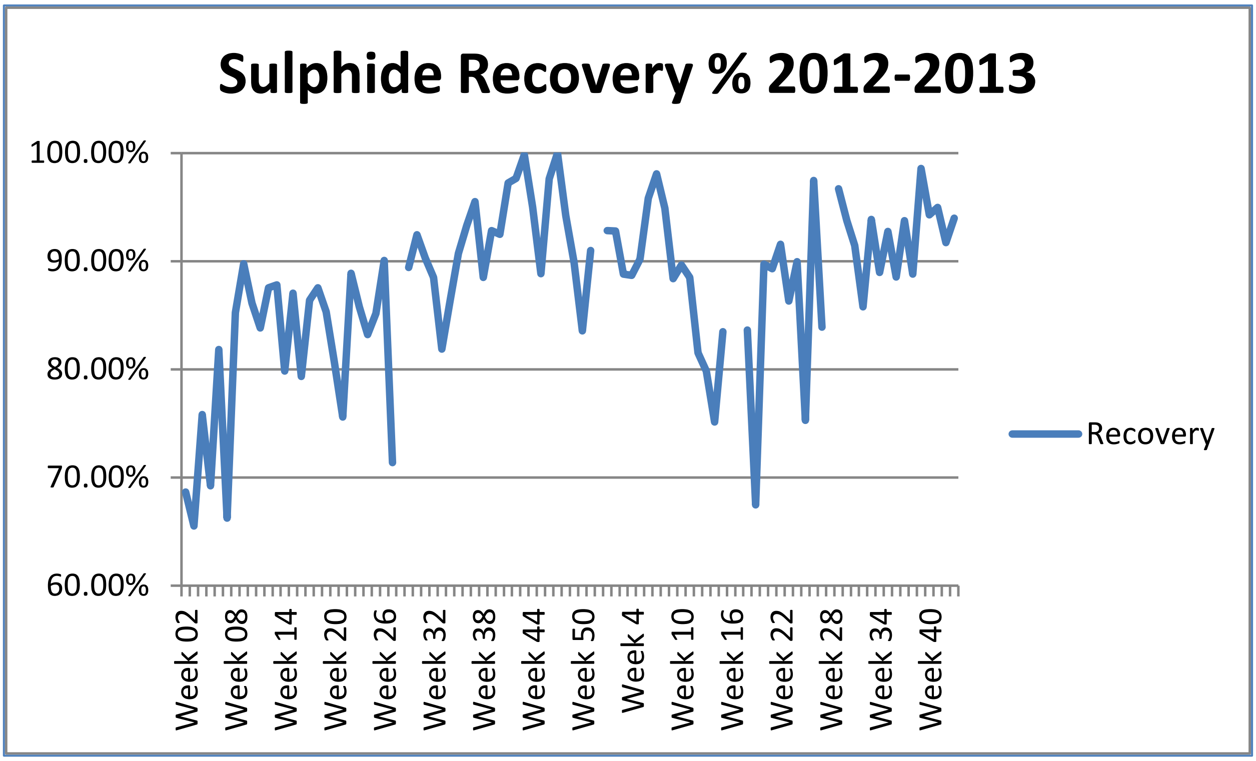


FIGURE 54: RECOVERY OF SULPHIDES IN THE MILL 2012-2013

Factors that have been taken into account when estimating the mill recovery are:

- The upgrade of the processing facility will streamline the process cycle
- The addition of the gravity concentrators to remove free gold and coarser sulphide particles

18.3.1 CONCENTRATE

Historically, the final concentrate for the NSR (Net Smelter Return) comprises of approximately:

- Gold – 100 g/t
- Silver – 250 g/t
- Lead – 9-10%
- Arsenic – 1-6% (Penalty Element)

Concentrate is bagged in IBC bags weighing between 1.0-1.3 tonnes.

Gold grade within the concentrate is expected to increase with the addition of the gravity concentrators. These will remove free gold particles that are not captured during the flotation process and were previously discharged to tails, or trapped within the mill.

18.3.2 REFINING

Concentrate is shipped from Belfast to a Canadian Smelter for refining. Bagged concentrate is loaded into shipping containers with average shipment weight of 24 tonnes.

Metal prices received are 95% of the average price of the previous month for gold, silver and lead. Refining charges and penalty charges for arsenic also apply.

Historical freight and refining charges average 15% of the gold price. This has been calculated by converting revenue into equivalent gold ounces so all revenue calculations are based on 85% of the forecast gold price to account for freight and refining.

18.3.3 TAILINGS

Tailings are discharged into the paste cells. Water from the paste cells settles and migrates through the cleaning ponds to the final polishing pond. Tailing will then be harvested for paste fill to reduce the surface imprint. The processing plant is supplied with re-cycled water from the polishing pond. Surplus water enters the catchment drainage via a V notch weir arrangement, where volume and water quality are routinely measured to ensure the standards regulated by the statutory authority (Northern Ireland Environment Agency) are met. The author has examined two, in-depth recent reports by the agency that attest to satisfaction of the Agency's requirements (see paragraph 20.2).

19 CONTRACTS & MARKET STUDIES

19.1 OFF-TAKE AGREEMENT

Flotation concentrates are sold to Xstrata Glencore (Formerly Xstrata Corporation) under a life of mine contract. The contract is well known to the author who considers the terms at the positive end of industry norms. The contract includes a smelter payment based upon tonnage sold and pays 95% of metal values for gold, silver and lead (the latter above certain cut-off percentage). It includes a fixed price shipping charge and advance payment provisions.

19.2 GOLD PRICE IN US DOLLARS AND UK STERLING

The Gold concentrate output from the Omagh Mine, which also contains silver and lead credits, is sold in US dollars (USD). Most of the value is accrued from the gold content. The following graph is compiled from data published by the Bank of England of average monthly gold price in US$ and UK £ (Sterling) per troy ounce (Figure 55). The graph is from December 2010 to April 2014.

Galantas currently has a policy of being un-hedged in regard to gold production.

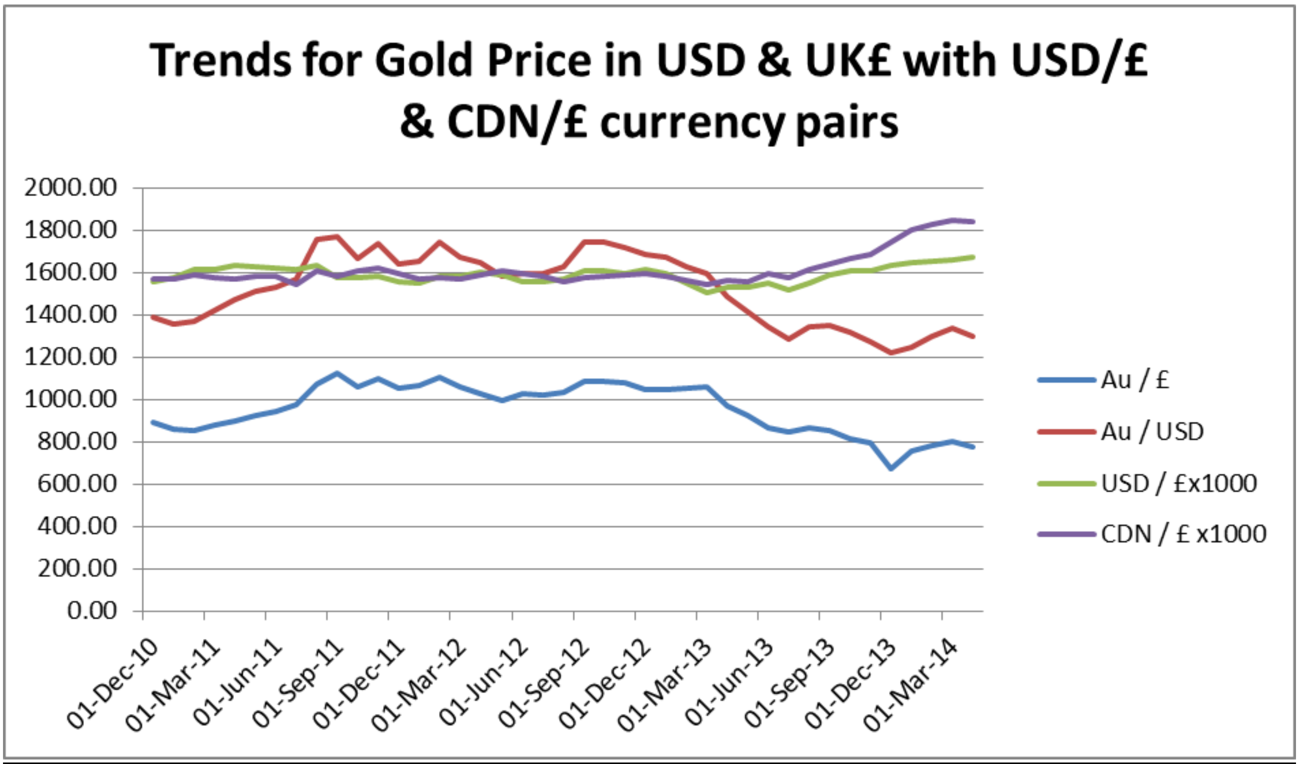


FIGURE 55: GOLD PRICE TRENDS IN USD AND UK £

19.2.1 THE US DOLLAR / UK £ STERLING AND CANADIAN DOLLAR / UK £ STERLING CURRENCY EXCHANGE RATES

The graph also includes Bank of England data that gives the monthly average spot exchange rate of US $ to UK£ Sterling and Canadian $ to UK£ Sterling. Sales revenues at the Omagh mine are designated in US Dollars and are converted to UK£, as Operating, Exploration and Capital costs are designated in UK£. Thus a stronger US$ / weaker UK£ is to the Galantas Gold Corporation's financial benefit.

The accounts of Galantas Gold Corporation (the Company) are expressed in Canadian Dollars. The majority of costs at the mine are incurred in UK£ Sterling and are converted to Canadian Dollars at the average rate for the relevant accounting period. When costs are expressed in Canadian Dollars terms within the Company's financials, there is an increase in costs when there is a fall in value or weakening of the Canadian Dollar against Sterling. A weakening of the Canadian dollar also increases the value of UK£ based net assets which are converted at period end rates, when expressed in Canadian dollars.

A historical currency policy of Galantas has been adopted of converting incoming payments into the currency required within a short period of receipt, thus avoiding the taking of a large currency position on either side of the market.

Whilst it is recognised that the USD/UK£ currency pair can impact overall profitability of the project and the CDN/UK£ currency pair can effect profit recognition, neither currency pair appear to suffer the same amount of volatility as the gold price. The volatility of the gold price can effect project profitability and for that reason three price scenarios have been examined for the project. To accommodate the volatility of the USD / UK£ currency pair, the project has been assessed in gold prices based in UK£. These prices are at a mid-price of £750 per ounce, an upper price of £800 per ounce and a lower price of £700 per ounce.

An average gold price of £800 per ounce (an increase of 6.7% or £50 per ounce on the mid-price) results in an 28% improvement in the financial outcome of the model.

An average gold price of £700 per ounce (a decrease of 6.7% or £50 per ounce on the mid-price) results in an 28% decline in the financial outcome of the model.

The lowest average quarterly price from between December 2011 and March 2014 was £761.24 per ounce, which was in the fourth quarter of 2013. Although Gold prices have hardened markedly since that time, a forward looking average quarterly price for gold of £750 within the short term would not seem unreasonable within the current context, with volatility on a month to month basis.

The main uses for gold are currently in Jewellery, Investment, Technology and Reserve Asset Management. According to the World Gold Council, *"After an exceptional 2013, gold demand made a robust start to 2014 - virtually unchanged year-on-year at 1,074.5 tonnes. Jewellery demand gained moderately, largely due to the environment of lower gold prices compared with Q1 2013 and seasonal factors in many markets. Divergence was seen within the investment space: net ETFs flows were zero, compared with 177t of outflows in Q1 2013, while bar and coin investment unsurprisingly fell far below the record Q1 levels of demand seen a year ago. Central banks continued to purchase gold for its diversification and risk management properties."*

One factor that may aid price stability is the report of an agreement (19th May 2014) between 21 European banks, including the European Central Bank to restrict sales of significant amounts of gold. The agreement is effective 27 September 2014.

The main factors affecting gold price historically appear to be fiscal and international instabilities. For the most part, future events are not easily forecast and the relative stability and purchasing power of gold forms part of its investment attraction. Current instabilities include the United States budget deficit / bond purchases, Euro zone deficits, bank instabilities and tensions within the Middle East and Ukraine. A low interest rate environment is seen by several commentators to be supportive of the price of gold and these commentators expect such conditions to persist globally with local variations outside of the US.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 PERMITTING

The underground mine, uprated processing plant and the export of a limited quantity of country rock from the underground mine will require planning permits to be issued through the Planning Service, Department of the Environment for Northern Ireland. OML submitted a detailed Environmental Impact Assessment with a planning application on 6th July 2012. Since that date neighbour and statutory consultations have taken place. Several statutory consultees have written with comments encouraging approval. Notable are positive comments by Roads Service and Omagh District Council. Consultations continue with statutory consultees and Galantas is confident any remaining issues can be satisfactorily addressed to create a positive economic benefit for the local community whilst preserving strong environmental control.

20.2 RECENT ENVIRONMENTAL STUDIES

Galantas notes two recent environmental studies on the operating mine site. The first of these studies prove conclusively that the country rock found at the mine is not acid forming and that some of the rocks are indicated to be potentially acid neutralising. The sampling was carried out by independent, environmental monitoring company Pentland Macdonald Ltd of Belfast. They undertook the collection of a representative set of 100 samples, with analysis taking place at the SGS Minerals Services Ltd laboratory in Cornwall. This extensive study is consistent with the results of earlier studies, which also showed no acid generation potential.

In a second report, a detailed Northern Ireland Environment Agency (NIEA) water study (June 2013) has declared Galantas subsidiary, Omagh Minerals Ltd, operator of the Omagh gold-mine, fully compliant with its water outlet requirements. The NIEA study, which is the second one of its type on the gold-mine property with similar results, backs up routine sampling data with more detailed continuously recorded information and also demonstrates that no acidic drainage from the mine takes place.

The mine is subject to a wide range of environmental monitoring by regulatory agencies. Routine sampling is carried out on behalf of Galantas by an independent sampling contractor and monitoring takes place of noise, dust, surface water and ground water. There are no material adverse environmental issues of which the author (having investigated the matter) is aware.

21 CAPITAL, OPERATING COSTS

Given that mining within the open pits has drawn to a close and stocked low grade material has ceased being routinely worked, there have been substantive reductions in the labour force from 50 persons to 5. Operating costs have been substantially reduced through redundancies and costs are expected to fall. No forward looking guidance has been offered on cost reduction.

Life Of Mine (LOM) Capital expenditure for the Underground Mine has been estimated as per the table below (Table 51). It should be noted that potential Leasable Equipment capital expenditure does not include costs associated with potential lease purchase arrangements which would spread the expenditure over 4 years.

LOM Capital Expenditure (PEA)	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM
Capital Excluding Leasable Equipment	£1,679,432	£4,149,604	£422,355	£390,534	£0	£0	**£6,641,926**
Capital Leasable Equipment	£1,273,469	£1,334,177	£0	£0	£0	£0	**£2,607,646**
Contingency 15%	£442,935	£822,567	£63,353	£58,580	£0	£0	**£1,387,436**
Working Capital	£1,000,000	£0	£0	£0	£0	£0	**£1,000,000**
GRAND TOTAL	**£4,395,836**	**£6,306,349**	**£485,708**	**£449,115**	**£0**	**£0**	**£11,637,007**

TABLE 51(i): LIFE OF MINE CAPITAL EXPENDITURE (including associated inferred resources)

In general, it should be noted that the estimated capital expenditure is reduced compared to similar projects. This is because it is an estimate of additional capital expenditure over and above that already incurred in land purchase, mill establishment, tailings disposal arrangements, surface equipment, etc. The capital value of these existing assets is not included.

The Operating Costs and Revenue arising has been estimated against three average gold prices (Table 52). A mid-price of £750 per ounce was selected as a mid-value, with a sensitivity of £50 per ounce applied, with a £800 per ounce upper bound and a £700 per ounce lower bound.

Gold Price £800/oz	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Operating Costs	£5,693,338	£10,430,904	£11,964,071	£11,261,136	£10,830,431	£8,459,001	**£58,638,882**
Capital Cost	£4,395,836	£6,306,349	£485,708	£449,115	£0	£0	**£11,637,007**
Revenue	£5,711,798	£16,637,139	£21,615,178	£20,520,831	£21,092,405	£12,524,609	**£98,101,960**
Net Cash Generated	(£4,377,377)	(£100,114)	£9,165,399	£8,810,580	£10,261,974	£4,065,608	**£27,826,071**
Gold Price £750/oz	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Total**
Operating Costs	£5,693,338	£10,430,904	£11,964,071	£11,261,136	£10,830,431	£8,459,001	**£58,638,882**
Capital Cost	£4,395,836	£6,306,349	£485,708	£449,115	£0	£0	**£11,637,007**
Revenue	£5,354,810	£15,597,318	£20,264,230	£19,238,279	£19,774,130	£11,741,821	**£91,970,588**
Net Cash Generated	(£4,734,364)	(£1,139,935)	£7,814,451	£7,528,028	£8,943,699	£3,282,820	**£21,694,699**
Gold Price £700/oz	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Total**
Operating Costs	£5,693,338	£10,430,904	£11,964,071	£11,261,136	£10,830,431	£8,459,001	**£58,638,882**
Capital Cost	£4,395,836	£6,306,349	£485,708	£449,115	£0	£0	**£11,637,007**
Revenue	£4,997,823	£14,557,497	£18,913,281	£17,955,727	£18,455,855	£10,959,033	**£85,839,215**
Net Cash Generated	(£5,091,351)	(£2,179,756)	£6,463,502	£6,245,477	£7,625,424	£2,500,032	**£15,563,326**

TABLE 52(i): LIFE OF MINE OPERATING COSTS AND REVENUE (Including associated inferred resources).

For comparison purposes, the capital requirements, operating costs and revenue are reduced if only Measured and Indicated Resources are considered (ie associated inferred are excluded from the mining program), in a second study (Feasibility) with estimates tabulated below.

LOM Capital Expenditure (Feasibility Study)	Year 1	Year 2	Year 3	Year 4	Year 5	LOM
Capital Excluding Leasable Equipment	£1,527,251	£2,314,762	£274,612	£331,159	£0	**£4,447,785**
Capital Leasable Equipment	£1,273,469	£1,102,041	£0	£0	£0	**£2,375,509**
Contingency 15%	£420,108	£512,520	£41,192	£49,674	£0	**£1,023,494**
Working Capital	£1,000,000	£0	£0	£0	-£1,000,000	**£0**
GRAND TOTAL	**£4,220,827**	**£3,929,323**	**£315,804**	**£380,833**	**-£1,000,000**	**£7,846,788**

TABLE 51(ii) LIFE OF MINE CAPITAL EXPENDITURE (excluding all inferred resources)

Gold Price £800/oz	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Operating Costs	£5,273,700	£9,236,587	£10,539,606	£9,645,334	£8,718,805	**£43,414,031**
Capital Cost	£4,220,827	£3,929,323	£315,804	£380,833	(£1,000,000)	**£7,846,788**
Revenue	£4,770,397	£12,480,208	£15,602,800	£14,938,608	£13,860,538	**£61,652,552**
Net Cash Generated	(£4,724,130)	(£685,702)	£4,747,389	£4,912,441	£6,141,734	**£10,391,732**
Gold Price £750/oz	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Total**
Operating Costs	£5,273,700	£9,236,587	£10,539,606	£9,645,334	£8,718,805	**£43,414,031**
Capital Cost	£4,220,827	£3,929,323	£315,804	£380,833	(£1,000,000)	**£7,846,788**
Revenue	£4,472,247	£11,700,195	£14,627,625	£14,004,945	£12,994,255	**£57,799,267**
Net Cash Generated	(£5,022,280)	(£1,465,715)	£3,772,214	£3,978,778	£5,275,450	**£6,538,448**
Gold Price £700/oz	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Total**
Operating Costs	£5,273,700	£9,236,587	£10,539,606	£9,645,334	£8,718,805	**£43,414,031**
Capital Cost	£4,220,827	£3,929,323	£315,804	£380,833	(£1,000,000)	**£7,846,788**
Revenue	£4,174,097	£10,920,182	£13,652,450	£13,071,282	£12,127,971	**£53,945,983**
Net Cash Generated	(£5,320,430)	(£2,245,728)	£2,797,039	£3,045,115	£4,409,166	**£2,685,163**

TABLE 52(ii): LIFE OF MINE OPERATING COSTS AND REVENUE (excluding all inferred resources).

22 ECONOMIC ANALYSIS

The Internal Rate of Return and Net Present Values at each of the gold prices examined are estimated in Table 53. They do not consider the cost of providing project finance (cost of capital) or the various mechanisms by which finance may be provided.

Average Gold Price	Cash Flow per year including the provision of capital expenditure							IRR	NPV	NPV
	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total		rate	
Gold £800/oz	-£4,377,377	-£100,114	£9,165,399	£8,810,580	£10,261,974	£4,065,608	**£27,826,071**	95%	8%	£19,159,021
Gold £750/oz	-£4,734,364	-£1,139,935	£7,814,451	£7,528,028	£8,943,699	£3,282,820	**£21,694,699**	72%	8%	£14,531,372
Gold £700/oz	-£5,091,351	-£2,179,756	£6,463,502	£6,245,477	£7,625,424	£2,500,032	**£15,563,326**	50%	8%	£9,903,723

TABLE 53(i): LIFE OF MINE NET CASH FLOW (including associated inferred resources)

The study scheduled approximately 36% of the combined resources identified on the Kearney and Joshua veins. The cash cost of production, excluding depreciation, is estimated in UK Sterling and converted, for comparable purposes, into US $ (at $1.68/£), as shown in Table 54.

Gold (Au) Cash Cost							
	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Average
Cost (£/oz)	£657	£414	£365	£362	£339	£445	£394
USD $1.68 / UK£ / oz	$1,105	$695	$613	$608	$569	$748	$662

TABLE 54(i): CASH COST OF PRODUCTION (including associated inferred resources)

The comparable financial estimates for the case where all inferred resources are excluded from the mining plan is as follows :-

Average Gold Price	Cash Flow per year including the provision of capital expenditure						IRR	NPV rate	NPV
	Year 1	Year 2	Year 3	Year 4	Year 5	Total			
Gold £800/oz	(£4,724,130)	(£685,702)	£4,747,389	£4,912,441	£6,141,734	**£10,391,732**	46%	8%	£6,597,309
Gold £750/oz	(£5,022,280)	(£1,465,715)	£3,772,214	£3,978,778	£5,275,450	**£6,538,448**	28%	8%	£3,602,535
Gold £700/oz	(£5,320,430)	(£2,245,728)	£2,797,039	£3,045,115	£4,409,166	**£2,685,163**	11%	8%	£607,762

TABLE 53(ii): LIFE OF MINE NET CASH FLOW (excluding all inferred resources)

Gold (Au) Cash Cost						
	Year 1	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Average**
Cost (£/oz)	£729	£488	£446	£426	£415	£464
USD $1.68 / UK£ / oz	$1,225	$820	$749	$715	$697	$780

TABLE 54(ii): CASH COST OF PRODUCTION (excluding all inferred resources)

The following table compares the Financial Data of the economic studies where associated inferred are included and where no inferred resources are included.

	Including Associated Inferred Resources			Excluding All Inferred Resources		
Gold Price	**IRR**	**NPV rate**	**NPV**	IRR	NPV rate	NPV
£800 / oz	**95%**	**8%**	**£19,159,021**	45%	8%	£6,597,309
£750 / oz	**72%**	**8%**	**£14,531,372**	28%	8%	£3,602,535
£700 / oz	**50%**	**8%**	**£9,903,723**	11%	8%	£607,762

TABLE 54(iii) COMPARISON BETWEEN ECONOMIC STUDIES

22.1 TAXATION

The profits generated by the project would, after allowance for accumulated losses capable of being carried forward, likely be subject to United Kingdom Corporation tax. As of March 2015, those rates of tax, assessed annually, are at 20% for the first £300,000, 21.25% on the subsequent £1,200,000 and 21% over £1,500,000. Taxation has been excluded from the economic analysis detailed above.

23 ADJACENT PROPERTIES

The following information is based upon Howe 2012 and has been amended.

The ground to the northeast of MR 1/08, comprising DG1/08, DG 2/08, DG 3/11 and DG4/11, is held by Dalradian Resources Inc.

The principal deposit lying within the Dalradian Resources licences is the Curraghinalt gold deposit located approximately 23 kilometres NE of Cavanacaw (see Figure 4). The Curraghinalt deposit contains mesothermal gold mineralisation, with gold disseminated in a swarm of quartz-sulphide veins hosted by the Dalradian-aged Mullagharn formation and is underlain by the Omagh thrust, a stratigraphic and structural setting similar to that of the Cavanacaw deposit. The Curraghinalt veins dip 60-70 degrees north, and strike west-northwest which is in contrast to the predominant northerly trend of the Cavanacaw veins. The phases of mineralisation appear to be different, with an increased amount of copper mineralisation present and a reduced amount of lead mineralisation.

An NI 43-101 compliant resource estimate for the Curraghinalt deposit was prepared by Micon International Limited, and filed on SEDAR on January 13th 2012, as a report entitled “An Updated Mineral Resource Estimate for the Curraghinalt Gold Deposit, Tyrone Project, Co Tyrone and County Londonderry, Northern Ireland”, in which CIM compliant Measured and Indicated Resources of 1.13 million tonnes at 13.00 g/t Au and Inferred Resources of 5.45 million tonnes at 12.74 g/t Au, effective at November 30th, 2011 were reported. An increased resource estimate was announced on April 16th 2014. Measured resources were estimated at 15,100 ounces, Indicated resources 989,000 ounces and inferred resources 2,487,700 ounces. The author (Galantas 2014) has not verified this information and it is not necessarily indicative of the mineralisation on the Galantas property.

A number of copper-gold occurrences are hosted by Ordovician volcanic rocks of the Tyrone Inlier, located to the southeast of Curraghinalt and covered by the Dalradian Resources licence DG2/08. These occurrences are less relevant to the Omagh Minerals ground since no rocks of this age or type are known to occur.

24 OTHER RELEVANT DATA AND INFORMATION

None

25 INTERPRETATION AND CONCLUSIONS

The completion of the latest drilling programme has allowed Galantas to provide a resource update and economic studies.

This revised resource estimate, as of April 2014, is summarised in Table 55.

	MEASURED			INDICATED			INFERRED		
	tonnes	grade Au (g/t)	contained Au (Oz)	tonnes	grade Au (g/t)	contained Au (Oz)	tonnes	grade Au (g/t)	contained Au (Oz)
KEARNEY	76,936	7.48	18,490	383,220	6.66	82,055	909,277	6.61	193,330
JOSHUA	54,457	7.25	12,693	216,211	7.92	55,046	291,204	10.74	100,588
KERR	6,848	4.63	1,019	12,061	4.34	1,683	23,398	3.2	2,405
ELKINS				68,500	4.24	9,000	20,000	5.84	3,800
GORMLEYS							75,000	8.78	21,000
GARRY'S							0	0	0
PRINCES							10,000	38.11	13,000
SAMMY'S							27,000	6.07	5,000
KEARNEY NORTH							18,000	3.47	2,000
total	**138,241**	**7.25**	**32,202**	**679,992**	**6.78**	**147,784**	**1,373,879**	**7.71**	**341,123**

TABLE 55: GALANTAS 2014 RESOURCE ESTIMATE

Note: (1) Rounded numbers, gold grades capped at 75g/t. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Galantas considers that there is good potential to define additional resources at Cavanacaw at depth and along strike on defined veins, as well as on additional structures that may be identified through ongoing exploration. The targets are identified in this report. The recent drilling success and economic modelling demonstrates the potential to build a profitable gold mining business.

Galantas licences OM 1/09 and 4/10 are underlain by similar formations to the Cavanacaw deposit. Numerous exploration targets identified on these licences attest to undeveloped exploration potential.

26. RECOMMENDATIONS

The study concludes that the development of an underground mine on Galantas' Omagh property can be achieved, subject to the constraints detailed and has potential to achieve the robust economics demanded by today's financial environment. The inclusion of some associated inferred resources within the mining study (PEA) increases the risk associated with geological uncertainties and attention is drawn to the following statement.

In compliance with NI 43-101 2.3.3(a) " ***the preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have 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***."

The inclusion of a comparable detailed mining study (Feasibility), which does not include the use of inferred resources, provides a measure of the risk attached when using inferred resources within the economic study.

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28. CERTIFICATE

a) Roland Phelps
Galantas Gold Corporation
36 Toronto Street
– Suite 1000,
Toronto,
Ontario Canada M5C 2C5

b) I, Roland Phelps, C.Eng., B.Sc. (Hons), MIMMM as co-author of this report entitled “Technical Report on the Omagh Gold Project”, prepared for Galantas Gold Corporation and dated 26th July 2014, make the following statements :

c) I was admitted to the degree of Bachelor of Science (Honours) in Mining Geology Combined, from the University Of Leeds, England on 22nd July 1976.
I was elected a Member of the Institution of Mining and Metallurgy on 15th May 1980.
I was elected a Member of the Institution Of Mining Engineers on 23rd January 1980.
I am a Chartered Engineer and Registrant of the Engineering Council (Registrant No. 316051) and a Member of the Institution of Materials, Mining and Metallurgy.
I have practiced as a geologist and mining engineer in Minerals Exploration, Resource Development and Mine Development for over 30 years.
I am a “Qualified Person” for the purposes of National Instrument 43-101.

d) I have visited the property routinely since 2000 and at least 20 days specifically during the period covered by this report.

e) I acknowledge the assistance of M. Mawson B.Sc (Hons), FGS., (Exploration Geologist, Omagh Minerals Ltd), Dr. S. Coulter B.Sc(Hons), M.Sc, PhD, FGS (Senior Geologist, Omagh Minerals Ltd), and G. Harris BEng (Hons), ACSM, GradIMMM (Mining Consultant) in the preparation of this report but solely act as “Qualified Person” for the whole of this technical report.

f) I am not independent of the Issuer.

g) I have had routine involvement with the property since 2000, in a technical and managerial capacity.

h) I have read, and consider the report compliant with PERC and with National Instrument 43-101 and Form 43-101 F1.

i) I consider that, at the effective date of the technical report, to the best of my knowledge and belief, the technical report contains all material scientific and technical information that is required to be disclosed to make the technical report not misleading.

j) 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 website accessible by the public.

Roland Phelps

Roland Phelps, C.Eng MIMMM, President & CEO, Galantas Gold Corporation

Dated: 26th July 2014

APPENDICES

APPENDIX 1: COMPARISON OF CIM AND PERC CODES

A simplified table comparing CIM and PERC codes, with the differences for each highlighted.

CIM Code Defined Term	Equivalent PERC Code Defined Term
Measured Mineral Resource	**Measured Mineral Resource**
Indicated Mineral Resource	**Indicated Mineral Resource**
Inferred Mineral Resource	**Inferred Mineral Resource**
Proven Reserve – by defined economic study which does not include the use of inferred resources within the mine plan. (NI.43-101 requires independence under certain circumstances.)	**Proven Reserve** – by defined economic study which permits the restricted inclusion of inferred resources within the mine plan where qualified
Probable Reserve – by defined economic study which does not include use of inferred resources within the mine plan. (NI.43-101 requires independence under certain circumstances.)	**Probable Reserve** – by defined economic study which permits the restricted inclusion of inferred resources within the mine plan where qualified

The above table is included as the requirement to satisfy NI 43-101 Part 7.1(2).

APPENDIX 2: DRILL DATA

The tables below detail the results for intersects drilled during the most recent programme (2011-2013) and provide location data for all holes drilled on the Cavanacaw property since 2006.

CAVANACAW DRILL INTERSECTIONS 2011-2013					
Hole ID	**Location**	**from (m)**	**To (m)**	**Dh length (m)**	**Au g/t**
OM-DD-11-52	Joshua Vein	24.60	27.40	2.80	2.97
OM-DD-11-54	Joshua Vein	38.20	41.80	3.60	2.20
OM-DD-11-55	Joshua Vein	63.10	65.60	2.50	3.19
OM-DD-11-56	Joshua Vein	35.00	38.40	3.40	9.61
OM-DD-11-60	Joshua Vein	49.25	50.75	1.50	2.48
OM-DD-11-61	Joshua Vein	80.30	82.65	2.35	7.91
OM-DD-11-62	Joshua Vein	73.40	75.10	1.70	7.10
OM-DD-11-63	Joshua Vein	11.20	16.50	5.30	14.90
OM-DD-11-65	Joshua Vein	18.00	20.30	2.30	3.65
OM-DD-11-70	Joshua Vein	35.00	36.60	1.60	11.64
OM-DD-11-73	Joshua Vein	31.50	32.80	1.30	3.99
OM-DD-11-82	Joshua Vein	41.50	42.78	1.28	1.08
OM-DD-11-83	Joshua Vein	42.37	43.61	1.24	5.21
OM-DD-11-88	Joshua Vein	70.13	71.58	1.45	7.07
And	Joshua Vein	79.70	80.96	1.26	3.43
OM-DD-11-91	Joshua Vein	92.90	94.15	1.25	5.26
OM-DD-11-100	Joshua Vein	87.20	88.54	1.34	5.31
OM-DD-11-101A	Joshua Vein	86.70	88.30	1.60	10.71
And	Joshua Vein	100.00	101.21	1.21	5.98
OM-DD-11-102	Joshua Vein	79.81	80.75	0.94	2.15
And	Joshua Vein	113.66	114.82	1.16	3.36
OM-DD-11-103	Joshua Vein	166.07	192.70	26.63	8.44
OM-DD-11-84	Kearney Vein	199.00	200.50	1.50	3.78
And	Kearney Vein	206.58	207.80	1.22	4.21

And	Kearney Vein	275.50	277.00	1.50	5.27
OM-DD-11-85	Kearney Vein	288.00	289.50	1.50	7.16
OM-DD-11-89	Kearney Vein	233.93	235.80	1.87	10.08
OM-DD-11-90	Kearney Vein	230.19	235.96	5.77	7.89
OM-DD-11-90B	Kearney Vein	230.35	237.24	6.89	11.17
And	Kearney Vein	241.00	246.20	5.20	4.88
OM-DD-11-92	Kearney Vein	335.89	337.02	1.13	12.51
OM-DD-12-106	Kearney Vein	162.5	165.3	2.8	2.4
OM-DD-12-110	Joshua Vein	75.7	77.5	1.8	4.5
OM-DD-12-111	Joshua Vein	119.2	127.2	8	4.3
OM-DD-12-114B	Joshua Vein	98.7	101.8	3.1	6.5
OM-DD-12-118	Joshua Vein	39.3	41.6	2.3	7.1
OM-DD-12-120	Joshua Vein	106.4	109.8	3.4	14.2
OM-DD-12-121	Joshua Vein	53.7	57.3	3.6	3.8
OM-DD-12-122	Joshua vein	133.7	135.82	2.12	21.2
And		142.2	145.83	3.63	11.4
OM-DD-12-123	Joshua Vein	60.8	63.7	2.9	12.5
OM-DD-12-129	Joshua	76.19	77.6	1.4	8.6
OM-DD-12-134	Joshua (south)	64.89	68.8	3.9	23.6
Including				0.8	64
OM-DD-11-95	Joshua Vein	122.4	123.9	1.5	7.4
OM-DD-12-132	Joshua (north)	75.9	77.54	1.64	3.2
OM-DD-12-142	Kearney	137.39	146.45	9.1	2.6
And		172.9	175.17	2.27	6.5
OM-DD-12-144	South Joshua	21.4	22.9	1.5	3.2
And		70.0	71.7	1.7	9
OM-DD-13-147	Joshua	146	151.59	5.6	12.4
OM-DD-148	Joshua	75.2	77.15	1.95	3.1

CAVANACAW DRILLING 2006-2014							
Hole ID	**Location**	**Easting**	**Northing**	**Elevation**	**Depth**	**Angle**	**Azimuth**
OM-DD-06-01	Commings Bog	240964.0	371979.0	120.0	59.2	-45.0	126.0
OM-DD-06-02	Kearney Vein	240128.0	371017.0	171.5	60.0	-45.0	270.0
OM-DD-06-03	Kearney Vein	240128.0	370989.0	169.0	58.2	-45.0	270.0
OM-DD-06-04	Kearney Vein	240153.0	370726.0	156.4	38.0	-46.0	247.5
OM-DD-06-05	Kearney Vein	240144.0	371114.0	177.6	97.0	-45.0	270.0
OM-DD-06-06	Kearney Vein	240129.0	371089.0	178.6	81.8	-45.0	280.0
OM-DD-06-07	Kerr Vein	239868.0	370607.0	178.6	98.5	-45.0	61.0
OM-DD-06-08	Elkin's Vein	240667.0	371191.0	117.8	69.0	-45.0	270.0
OM-DD-06-09	Elkin's Vein	240663.0	371248.0	118.9	79.5	-45.0	270.0
OM-DD-06-10	Elkin's Vein	240671.0	371135.0	116.3	60.5	-45.0	270.0
OM-DD-06-11	Elkin's Vein	240687.0	371114.0	115.2	65.5	-45.0	270.0
OM-DD-06-12	Elkin's Vein	240714.0	371109.0	114.4	85.0	-45.0	270.0
OM-DD-06-13	Elkin's Vein	240715.0	371109.0	114.5	63.0	-65.0	270.0
OM-DD-06-14	Kearney Vein	240150.0	371145.0	172.0	122.0	-46.2	280.0
OM-DD-07-15	Kearney Vein	240138.0	371046.0	172.0	90.0	-42.1	270.0
OM-DD-07-16	Kearney Vein	240171.0	371039.0	157.5	120.0	-44.3	276.0
OM-DD-07-17	Kearney Vein	240210.0	371039.0	155.0	167.3	-45.2	270.0
OM-DD-07-18	Kearney Vein	240177.0	371091.0	171.0	154.6	-45.0	270.0
OM-DD-07-19	Kearney Vein	240205.0	370832.0	162.0	135.0	-45.2	275.0
OM-DD-07-20	Elkin's Vein	240657.0	371126.0	116.2	48.0	-45.4	270.0
OM-DD-07-21	Elkin's Vein	240669.0	371163.0	116.4	50.0	-45.3	270.0
OM-DD-07-22	Kearney Vein	240175.0	370800.0	159.0	102.0	-45.0	270.0
OM-DD-07-23	Elkin's Vein	240665.0	371219.0	118.9	51.0	-45.2	270.0
OM-DD-07-24	Kearney Vein	240190.0	370750.0	171.0	130.0	-47.6	270.0
OM-DD-07-25	Elkin's Vein	240662.0	371274.0	119.0	117.0	-44.6	270.0
OM-DD-07-26	Kearney Vein	240204.0	370725.0	173.0	164.2	-48.9	270.0

OM-DD-07-27	Elkin's Vein	240688.0	371162.0	115.7	76.0	-45.1	270.0
OM-DD-07-28	Kearney Vein	240190.0	370675.0	167.0	101.4	-50.3	270.0
OM-DD-07-29	Elkin's Vein	240700.0	371219.0	117.7	111.0	-45.0	270.0
OM-DD-07-30	Kearney Vein	240188.0	370626.0	162.0	116.2	-50.3	270.0
OM-DD-07-31	Kearney Vein	240202.0	370850.0	161.0	143.4	-45.0	270.0
OM-DD-07-32	Kearney Vein	240155.0	370850.0	163.0	110.0	-45.3	267.0
OM-DD-07-33	Kearney Vein	240114.0	371322.0	161.0	120.0	-44.8	268.0
OM-DD-07-34	Kearney Vein	240180.0	370875.0	161.0	167.0	-49.2	271.0
Hole ID	**Location**	**Easting**	**Northing**	**Elevation**	**Depth**	**Angle**	**Azimuth**
OM-DD-07-36	Kearney Vein	240176.0	370900.0	165.0	140.0	-48.8	275.0
OM-DD-07-37	Kearney Vein	240176.0	370925.0	159.0	131.5	-50.0	270.0
OM-DD-07-38	Kearney Vein	240117.0	370921.0	169.0	104.0	-45.0	270.0
OM-DD-07-39	Kearney Vein	240111.0	371141.0	179.0	110.0	-45.0	270.0
OM-DD-07-40	Kearney Vein	240172.0	370973.0	167.0	125.4	-45.0	270.0
OM-DD-07-41	Kearney Vein	240221.0	371122.0	175.0	281.7	-50.1	270.0
OM-DD-07-42	Kearney Vein	240136.0	370946.0	167.0	68.0	-45.1	270.0
OM-DD-07-43	Kearney Vein	240125.0	371175.0	177.0	102.0	-45.3	274.0
OM-DD-07-44	Kearney Vein	240172.0	371072.0	169.0	125.0	-45.2	270.0
OM-DD-07-45	Kearney Vein	240229.0	370678.0	159.0	144.2	-44.7	270.0
OM-DD-07-46	Kearney Vein	240241.0	371174.0	153.0	329.0	-44.5	270.0
OM-DD-07-47	Kearney Vein	240206.0	371093.0	166.0	243.0	-44.0	270.0
OM-DD-07-48	Kearney Vein	240229.0	371217.0	165.0	258.5	-45.5	270.0
OM-DD-07-49	Kearney Vein	240250.0	371275.0	165.0	252.0	-45.0	270.0
OM-DD-11-51	Joshua Vein	239661.7	370914.5	173.4	59.2	-45.0	99.8
OM-DD-11-52	Joshua Vein	239624.4	370917.6	175.3	50.0	-45.0	107.5
OM-DD-11-53	Joshua Vein	239623.2	370917.9	175.3	100.4	-70.0	117.0
OM-DD-11-54	Joshua Vein	239624.7	370892.6	172.8	52.0	-45.0	95.5
OM-DD-11-55	Joshua Vein	239626.8	370863.0	167.6	71.0	-45.0	95.0

OM-DD-11-56	Joshua Vein	239634.9	370862.9	167.0	50.3	-45.0	93.7
OM-DD-11-57	Joshua Vein	239695.9	370862.9	165.0	135.0	-75.0	279.5
OM-DD-11-58	Joshua Vein	239681.0	370835.8	163.0	50.0	-45.0	279.5
OM-DD-11-59	Joshua Vein	239680.6	370950.9	175.0	50.5	-45.0	282.3
OM-DD-11-60	Joshua Vein	239680.9	370975.9	175.3	60.5	-45.0	277.2
OM-DD-11-61	Joshua Vein	239612.9	370863.7	168.6	134.7	-45.0	99.0
OM-DD-11-62	Joshua Vein	239618.8	370838.7	165.5	83.3	-45.0	90.0
OM-DD-11-63	Joshua Vein	239699.7	370783.2	161.7	21.8	-45.0	279.4
OM-DD-11-64	Joshua Vein	239706.0	370783.6	161.7	51.0	-75.0	270.0
OM-DD-11-65	Joshua Vein	239679.3	370780.2	162.0	30.0	-50.0	94.4
OM-DD-11-66	Kerr Veins	239952.9	370670.1	150.7	31.0	-45.0	95.4
OM-DD-11-67	Kerr Veins	239923.2	370695.6	154.2	71.1	-45.0	90.0
OM-DD-11-68	Kerr Veins	239970.1	370610.8	148.5	50.5	-45.0	68.2
OM-DD-11-69	Kerr Veins	239964.5	370600.3	148.5	38.0	-45.0	254.7
OM-DD-11-70	Joshua Vein	239668.4	370756.5	161.4	50.5	-45.0	90.0
OM-DD-11-71	Kearney Vein	240357.2	370948.9	138.6	395.0	-50.0	270.0
OM-DD-11-72	Joshua Vein	239716.0	370761.3	161.3	85.0	-70.0	270.0
OM-DD-11-73	Joshua Vein	239668.2	370731.5	161.9	40.0	-45.0	90.0
OM-DD-11-74	Joshua Vein	239670.9	370706.5	162.5	35.4	-45.0	88.7
OM-DD-11-75	Joshua Vein	239667.3	370731.4	161.6	103.7	-70.0	90.0
OM-DD-11-76	Joshua Vein	239660.1	370969.1	178.4	44.3	-45.0	270.0
OM-DD-11-77	Joshua Vein	239694.1	371000.0	173.1	73.0	-45.0	280.0
OM-DD-11-78	Joshua Vein	239693.6	371000.3	173.1	64.9	-45.0	318.0
OM-DD-11-79	Kearney Vein	240137.9	371206.7	166.9	85.0	-50.0	270.0
OM-DD-11-80	Joshua Vein	239660.1	371025.3	180.0	64.9	-50.0	270.0
OM-DD-11-81	Kearney Vein	240138.8	371206.8	166.8	143.4	-75.0	270.0
OM-DD-11-82	Joshua Vein	239660.3	371055.1	181.6	55.4	-45.0	270.0
OM-DD-11-83	Joshua Vein	239660.6	371079.9	183.0	65.7	-45.0	270.0
OM-DD-11-84	Kearney Vein	240271.9	371090.0	142.5	353.5	-45.0	270.0
OM-DD-11-85	Kearney Vein	240296.6	371011.4	144.7	372.0	-45.0	280.0

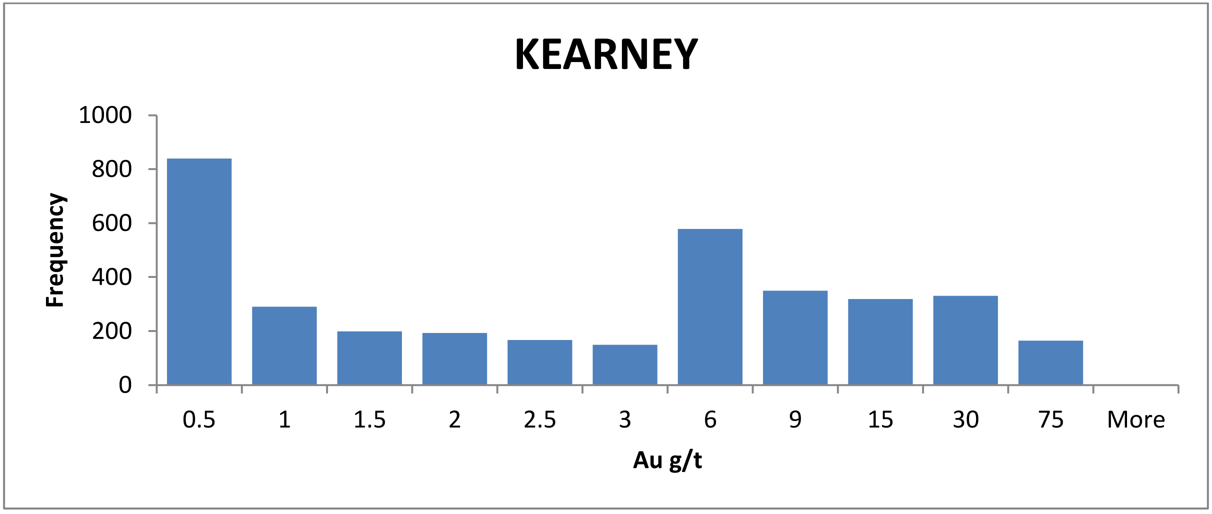
OM-DD-11-86	Joshua Vein	239668.7	371105.2	182.3	62.8	-45.0	261.4
OM-DD-11-87	Joshua Vein	239670.2	371057.8	181.6	82.9	-47.2	260.2
OM-DD-11-88	Joshua Vein	239672.8	371130.0	181.4	92.5	-45.0	277.5
OM-DD-11-89	Kearney Vein	240247.2	371142.0	150.6	263.0	-45.9	277.9
OM-DD-11-90	Kearney Vein	240273.3	371049.8	142.9	245.0	-44.1	277.8
OM-DD-11-90B	Kearney Vein	240273.3	371049.8	142.9	350.0	-44.1	277.8
OM-DD-11-91	Joshua Vein	239710.3	371067.8	179.1	115.0	-45.0	242.0
OM-DD-11-92	Kearney Vein	240351.0	371189.9	140.4	402.0	-44.3	288.0
OM-DD-11-93	Joshua Vein	239704.6	371117.5	181.0	113.0	-46.4	267.5
OM-DD-11-94	Kearney Vein	240351	3711090	137.9	449	-45	270
OM-DD-11-95	Joshua Vein	239710.0	371184.0	179.8	143.0	45.1	266.0
OM-DD-11-97	Kearney Vein	240329.5	371140.0	139.7	406.1	-42.5	274.8
OM-DD-11-98	Joshua Vein	239711.3	371067.8	179.1	187.8	-66.6	277.9
OM-DD-12-99B	Kearney Vein	240404	371203	141.3	431	-45	270
OM-DD-11-100	Joshua Vein	239598.9	370790.1	166.3	200.0	-48.8	91.8
OM-DD-11-101A	Joshua Vein	239673.0	371155.0	182.1	120.0	-45.9	281.8
OM-DD-11-102	Joshua Vein	239672.8	371180.0	183.1	134.0	-46.1	278.4
OM-DD-11-103	Joshua Vein	239598.0	370790.1	166.3	279.0	-71.0	93.4
OM-DD-12-104	Kearney Vein	240406	371040	133.2	413.3	-45	270
OM-DD-12-105	Joshua Vein	239672	371202	182.8	142	-45	270
OM-DD-12-106	Kearney Vein	240024	370710	162.9	330.6	-45	100
OM-DD-12-107	IP anomaly	239784	371012	172.7	176	-45	80
OM-DD-12-108	Joshua Vein	239672	371223	182.3	151.7	-45	270
OM-DD-12-109	IP anomaly	239672	371223	182.3	257	-45	110
OM-DD-12-110	Joshua Vein	239608	370764	165.6	189	-45	90
OM-DD-12-111	Joshua Vein	239673	371154	182.1	167.6	-65	270
OM-DD-12-112	Joshua Vein	239836	371172	171.4	443.3	-45	270
OM-DD-12-113	Joshua Vein	239581	370864	169	160	-45	90
OM-DD-12-114B	Joshua Vein	239610	370740	165.2	321	-65	90
OM-DD-12-115	Kearney Vein	239994	370710	164.2	332	-45	100

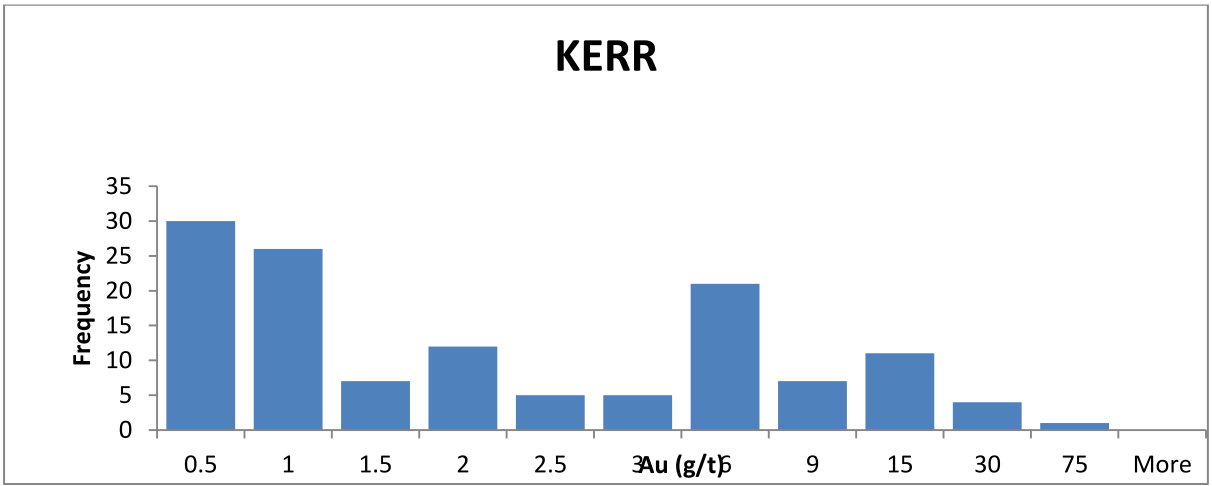
OM-DD-12-116	Joshua Vein	239900	370610	150.7	155	-45	270
OM-DD-12-117	Joshua Vein	239584	370893	174.9	135	-45	90
OM-DD-12-118	Joshua Vein	239600	370688	165.9	218	-45	90
OM-DD-12-119	Joshua Vein	239582	370893	174.8	494	-45	90
OM-DD-12-120	Joshua Vein	239575	370814	166.2	140	-45	90
OM-DD-12-121	Joshua Vein	239645	370690	165.5	152	-70	90
OM-DD-12-122	Joshua Vein	239590	370637	171.2	172.9	-45	90
OM-DD-12-123	Joshua Vein	239625	370715	165.2	109	-45	90
OM-DD-12-124	Kearney Vein	239954	370782	165.5	376	-45	90
OM-DD-12-125	Joshua Vein	239623	370715	165.3	121	-70	90
OM-DD-12-126	Joshua Vein	239630	370661	169.2	117	-45	90
OM-DD-12-127	Kerr Vein	239902	370584	150.2	163	-45	90
OM-DD-12-128	Kerr Vein	239899	370695	151.9	211	-45	90
OM-DD-12-129	Joshua Vein	239711	370915	171.7	97	-45	270
OM-DD-12-130	Kerr Vein	240074	370609	163	224	-45	285
OM-DD-12-131	Joshua Vein	239625	370605	170.5	120	-45	90
OM-DD-12-132	Joshua Vein	239700	370948	174	98.4	-51	270
OM-DD-12-133	Kerr Vein	240050	370590	161.4	325	-45	260
OM-DD-12-134	Joshua Vein	239714	370638	166.3	92	-50	270
OM-DD-12-135	Joshua Vein	239697	370949	174	158	-70	275
OM-DD-12-136	Joshua Vein	239719	370555	169.6	118.8	-50	275
OM-DD-12-137	Western lagoon	239869	370577	156.7	74.2	-45	275
OM-DD-12-138	Joshua Vein	239642	3711132	183.7	77.2	-45	270
OM-DD-12-139	Kearney Vein	240074	370611	162.6	139.5	-45	70
OM-DD-12-140	Joshua Vein	239720	370579	168.5	78	-45	275
OM-DD-12-141	Joshua Vein	239624	370820	164.1	95.1	-45	90
OM-DD-12-142	Kearney Vein	240074	370611	167.6	212.5	-45	90
OM-DD-12-143	Joshua Vein	239721	370579	168.4	118.6	-60	270
OM-DD-12-144	Joshua Vein	239719	370530	169.7	110	-45	275
OM-DD-12-145	Kearney Vein	240047	371201	155.1	185	-50	90

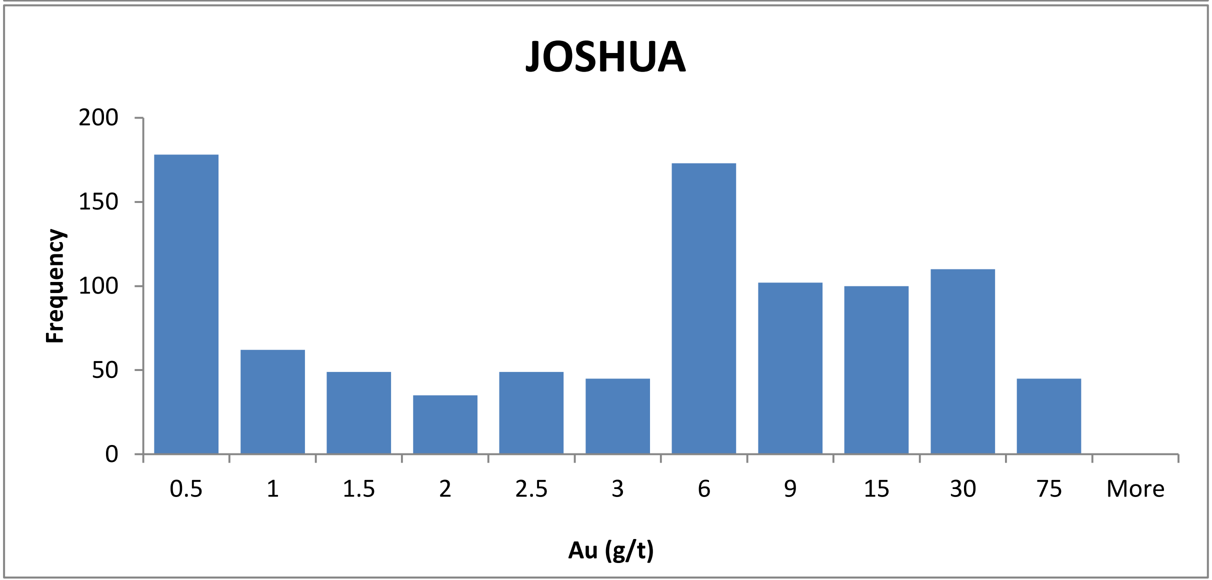
OM-DD-13-146	Joshua vein	239826	370638	157	237	-45	273
OM-DD-13-147	Joshua vein	239604	370740	165.2	171.8	-70	90
OM-DD-13-148	Joshua vein	239611	370741	165.2	101	-50	90
				TOTAL	5819.4	2006-2007	
				TOTAL	7001.1	2011-2012	
				TOTAL	8911.5	2012-2014	
				TOTAL	21732	2006-2014	

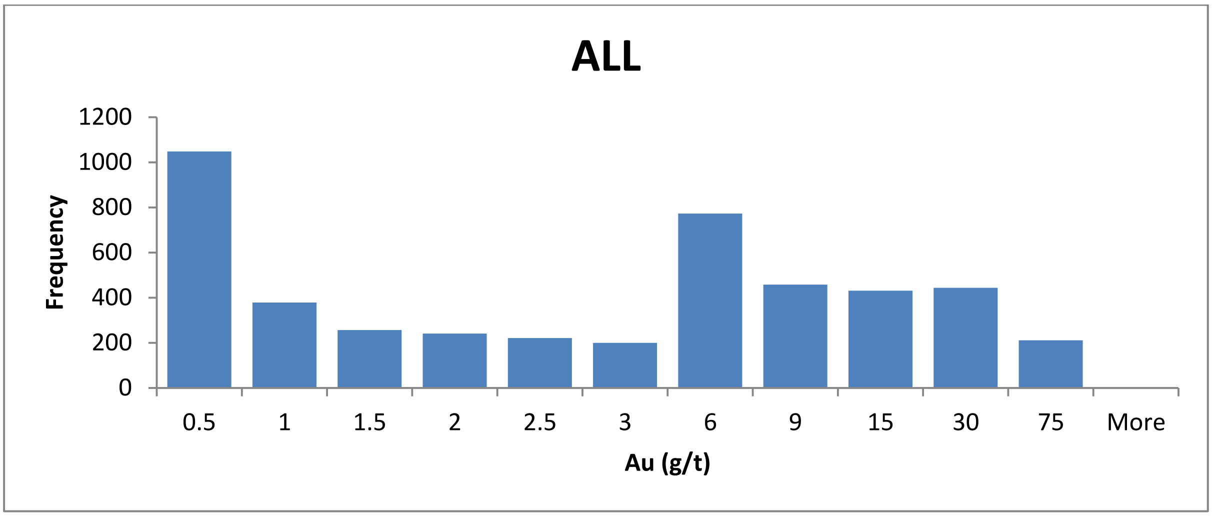
APPENDIX 3: VEIN WIDTH AND GRADE FREQUENCY

A series of histograms have been created from the complete channel and drill core sample assay database, to present a summary of vein width frequency and grade frequency for the three main systems: Kearney, Joshua and Kerr. The histograms are displayed below. Sample grade frequency:

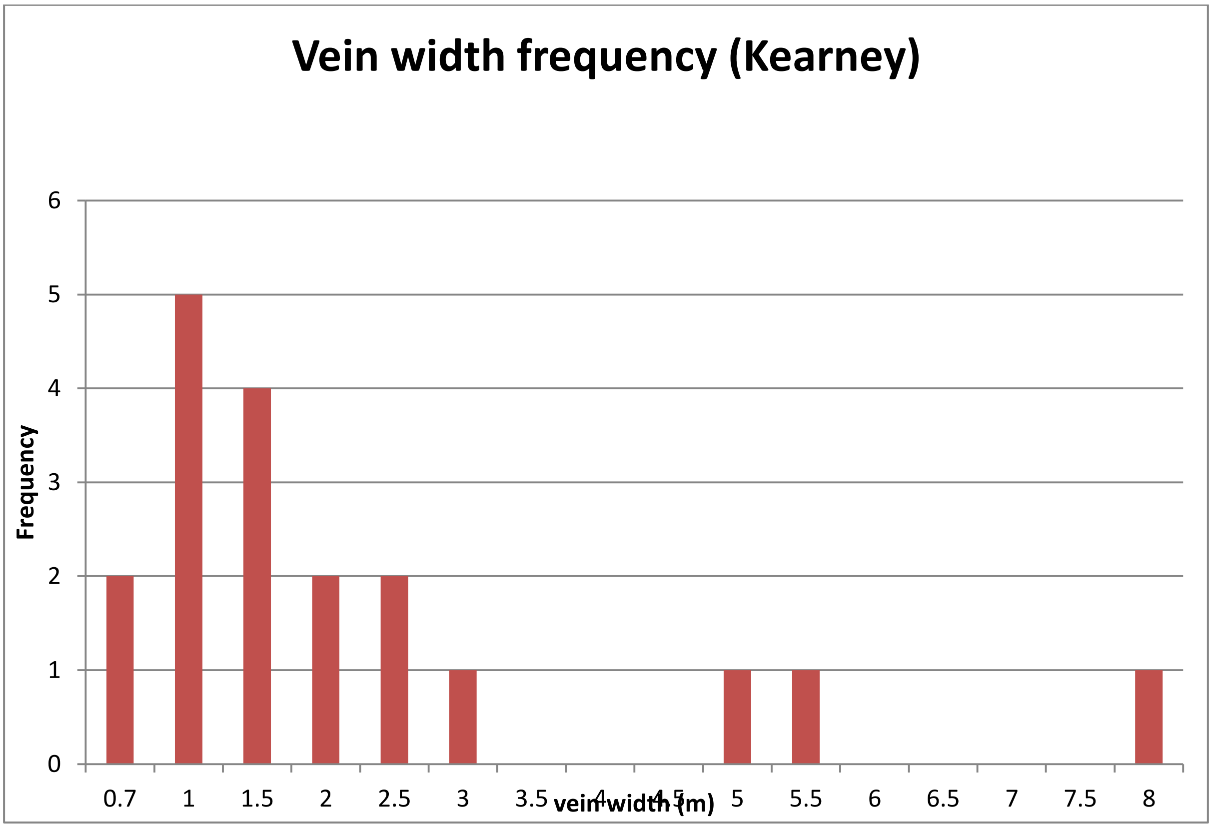


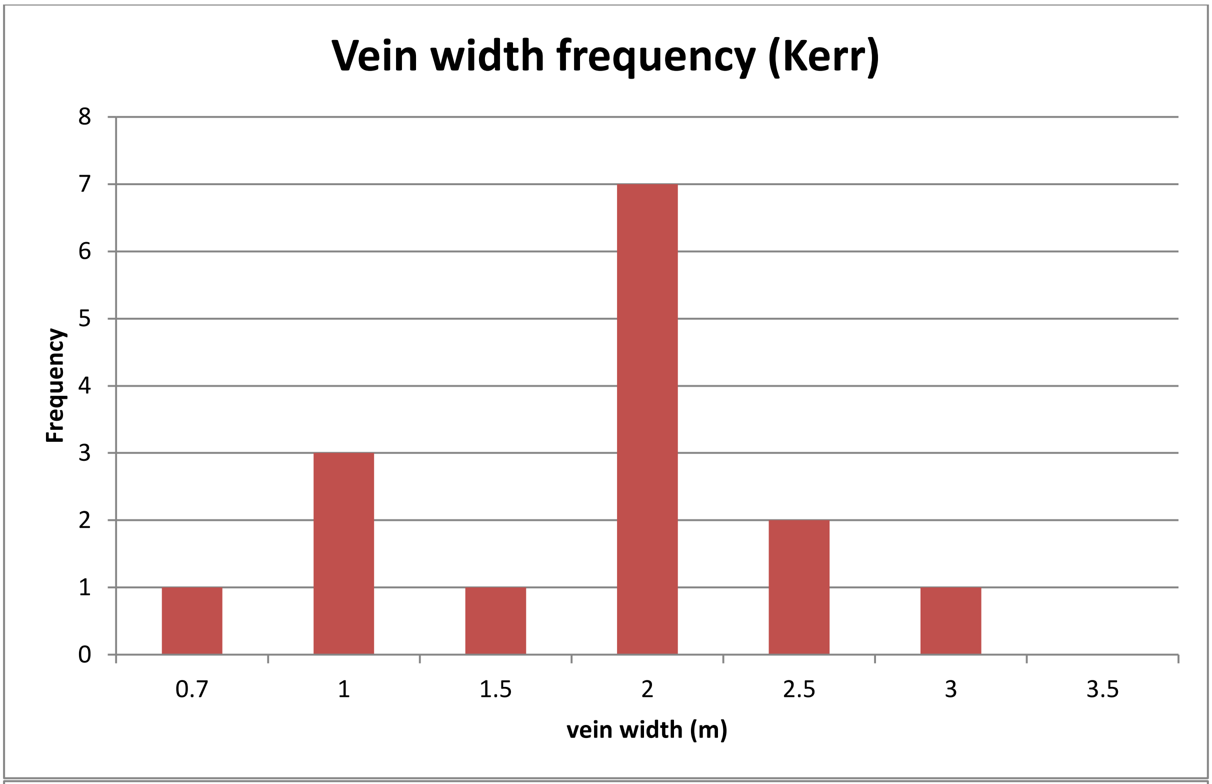


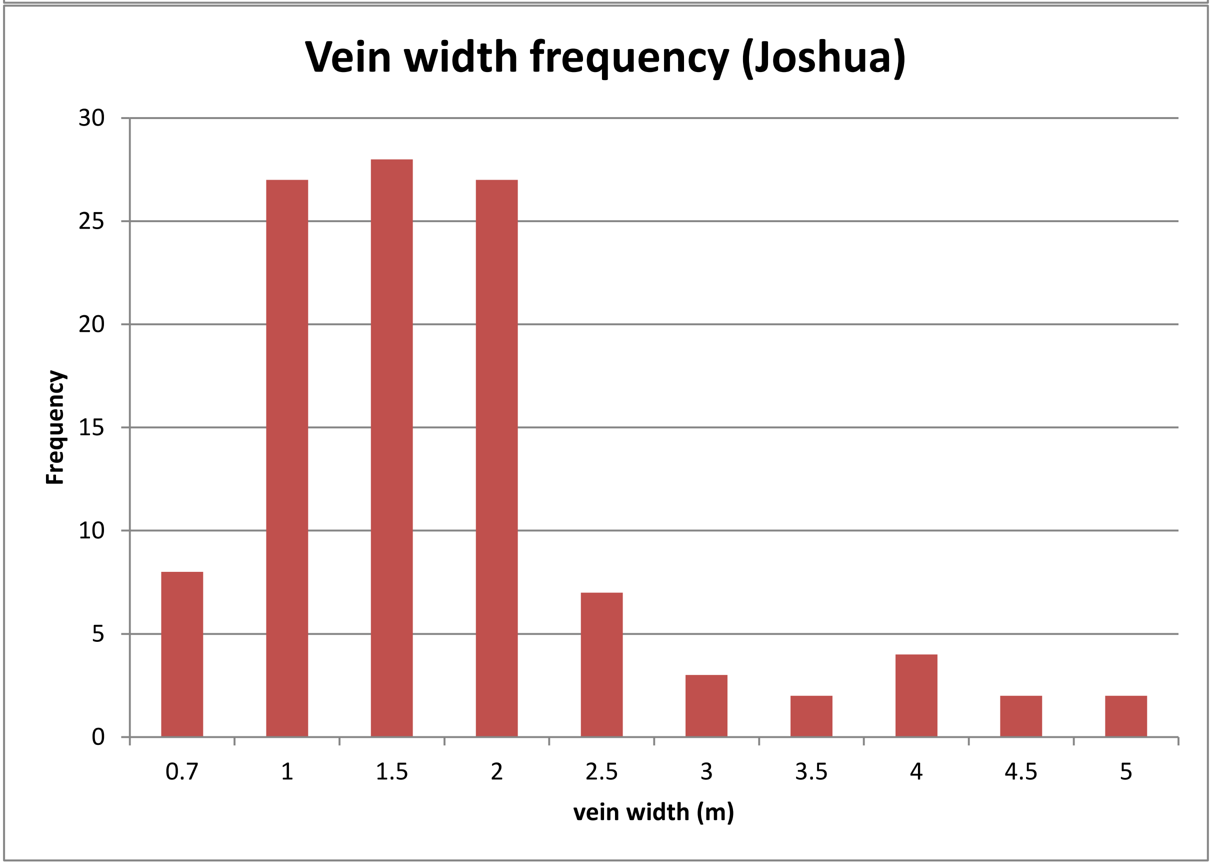




Vein width frequency:




Vein width frequency (Kerr)
Frequency
8
7
6
5
4
3
2
1
0
0.7
1
1.5
2
2.5
3
3.5
vein width (m)


Vein width frequency (Joshua)
Frequency
30
25
20
15
10
5
0
0.7
1
1.5
2
2.5
3
3.5
4
4.5
5
vein width (m)