A C A HOWE INTERNATIONAL LIMITED

TECHNICAL REPORT
ON THE
OMAGH GOLD PROJECT,
COUNTIES TYRONE AND FERMANAGH,
NORTHERN IRELAND

For

GALANTAS GOLD CORPORATION

by
ACA HOWE INTERNATIONAL LIMITED

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SUMMARY

Galantas Gold Corporation (Galantas) has retained ACA Howe International Limited (ACA Howe) to prepare an updated mineral resource estimate for the Cavanacaw gold deposit and an independent Technical Report, in accordance with the reporting standards and definitions required under Canadian National Instrument 43-101. This report presents a summary of the project geology and exploration programme, the results of the mineral resource estimate, and a preliminary economic assessment of a proposed underground mining operation based on the revised resource estimate.

This report has relied upon information on the same property that is contained in previous reports filed on SEDAR, on information provided by Galantas, and on published information.

The Cavanacaw mine is located 5 km west-southwest of Omagh, County Tyrone, Northern Ireland on land owned by Omagh Minerals Limited (Omagh Minerals), a subsidiary of Galantas. The mine is subject to a Mining Licence for gold and silver issued by the Crown Estate Commissioners. Galantas also holds exploration rights over 439 km² of land adjoining the mine.

The mine is easily accessible from Belfast and Omagh and is located on rough agricultural land.

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 1980s, Riofinex North Ltd (Riofinex) commenced exploration of an area of similar rocks located west-southwest 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 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 who commissioned metallurgical, mining and environmental studies.

In 1997, European Gold Resources Inc. (EGR) acquired Omagh Minerals 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. This 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 1.5 m section returning 1.15 g/t gold.

European Gold Resources Inc. was renamed 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 km² 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 resource estimate is 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 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 t 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 of the pit 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 orogen 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 gold 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 a metre wide with disseminated to massive auriferous sulphides, predominantly pyrite and galena with 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 licence OM 1/09 as discussed in section 6 of this report and shown in Figure 5. ACA Howe considers that visual prospecting is an effective additional means of investigating the known prospects in the short term, and could also be applied in a systematic way to investigation of the entire Dalradian area held under licence.

Exploration at Cavanacaw since 1995 has included 4,281 m of channel sampling and 26,380 m of diamond drilling, described in detail in previous reports. The 2011-2012 exploration programme comprised channel sampling and drilling at Joshua, Kearney and Kerr veins, directed towards evaluating the deeper resources that may be amenable to underground mining.

Diamond core drilling was mostly of HQ3 (61.1 mm) size and used triple tube core barrels to ensure good core recovery. Core handling, logging and sampling were carried out to best industry standards. Drillholes were surveyed at 6 m intervals by multi-shot equipment.

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 so serious as to invalidate the use of drill core samples for resource estimation.

Drill intersections included some exceptional values including 7.6 m at 8.44 g/t Au in hole 103 at Joshua vein and 3.5 m 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 10 cm intervals across the vein. Drill core sample intervals were determined by mineralisation and lithological type and were confined to the vein and immediate wallrock. 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 laboratory QA/QC procedures using blanks, standards and duplicates were monitored by Galantas and ACA Howe 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 are reliable and adequate for use in the resource estimate.

Initial froth flotation test work was carried out by Lakefield (1992) and the results of that test work were incorporated into the existing plant.

ACA Howe has prepared an updated estimate of mineral resources for the main Kearney Vein Zone and Joshua's Vein, which have been the main focus of historical and recent exploration, and for several other veins in the project area that have been drill tested by historical and recent drilling.

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. The database contains all channel sampling and drilling data generated up to the data cut-off on June 1st, 2012.

Mineralised zones were interpreted and 3D wireframes were created using Micromine software. Sample data was selected and statistical analysis was performed on raw sample data to assess the validity of this data for use in resource estimation. Following the generation of mineralised domains, raw sample data was composited in order to standardise sample support and further statistical and geostatistical analysis was 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 were written to the block model prior to reporting CIM compliant grade and tonnage estimates for the project. The total 2012 resource estimate for all veins at Cavanacaw is as follows:

CATEGORY	CUT-OFF 2.5 g/t Au		
	TONNES	Grade (Au g/t)	Au ozs
MEASURED	15,250	6.52	3,300
INDICATED	411,600	7.01	92,000
INFERRED	839,000	8.53	231,000

The block models which formed the basis for the resource estimate were validated by comparison of composite grade versus block grade both visually on cross section and in aggregate for each domain.

The following factors have generally caused decreases to the 2012 resource estimate relative to the 2008 resource estimate:

- The 2008 estimate is subject to depletion by mining from Kearney, Kerr and Joshua pits.
- A cut-off grade of 2.5g/t Au was applied to the 2012 estimate. No such cut-off was applied to the 2008 estimate.
- Further differences between the 2008 estimate and the 2012 estimate are caused by differences in the domain outlines caused by structural re-interpretations.

The reductions in resources have been largely offset by increases in resource tonnages and categories for Joshua's vein and some domains at Kearney's due to infill and resource augmentation drilling.

No mineral reserves have been estimated. The reader should understand that mineral resources are not mineral reserves and do not have demonstrated economic viability.

Open pit mining commenced in 2006. Selective mining is employed using a narrow bucket excavator to remove vein material over minimum widths down to 30 cm, and to depths of 1.5 m below bench level. The vein material is generally incompetent due to clay gouge content, brecciation and wallrock alteration which renders it amenable to mining by mechanical means without the need for explosives. By May 2012, mining was largely restricted to the northern end of the Kearney pit, mining in other parts of the pit 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.

Galantas has carried out an internal 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 prefabricated 'cut and cover' ramp installed within the backfilled open pit and a spiral ramp developed from the bottom of the prefabricated ramp at 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 new plant assumes a 50 tonnes per hour feed rate at an average diluted gold grade of 5.5 g/t.

A conveyor system is proposed to exit the mine and deliver ore and waste to separate piles. Ore will be drawn from the stockpile by front-end loader as required by the plant. A sub-station will be provided adjacent to the fan building for connection to mains power which will supply an estimated load of 3,000 kVA required for the mine and mill.

A gold price of $1,375/ounce has been used in the financial evaluation, which is conservative when compared to the average price of $1,538 for the past 24 months.

Galantas state that a full Environmental Impact Assessment has been prepared for the proposed underground mine and is shortly to be filed with the regulating authorities. Disposal of waste rock and tailings is key to the approval of a planning application. Approximately 50% of the tailings will be used for back-filling underground. The remaining tailings will be stored in permanent paste cells with berms constructed from the existing waste rock stockpile. It is also planned to backfill the Kearney pit with waste rock and tailings.

The host rocks of the Cavanacaw deposit are relatively impermeable and a pumping capacity of 200 gallons per minute is considered adequate to drain the Kearney pit and underground mine and to provide spare capacity in case of emergency or breakdown.

Acid drainage is not an issue at Cavanacaw, since sulphides are recovered in gold bearing concentrate, with both tailings and rock containing sufficient natural carbonate to create a buffer and maintain mildly alkaline water outflows.

The majority of restoration works will be completed within five years of the start of the project. Additional areas to be restored will be those paste cells still in development or use, the mill area, stockpile area and mine portal area. A new restoration plan will be included in the planning application for the underground mine, which restores the natural landform previously present in the area of the Kearney open pit and permits extra landscaped storage of tailings and rock together. The object of the restoration plan is to minimise the amount of post-closure restoration required by completing as much restoration as possible on an on-going basis.

Omagh Minerals owns the freehold land upon which the existing open-pit mine has been constructed and has recently purchased additional land to the west which overlies part of the Joshua vein discovery. Galantas management consider that Omagh Minerals owns sufficient land required for operation of an underground mine of the scale anticipated.

The underground mine, uprated processing plant and the export of a limited quantity of waste rock from the underground mine will require planning permits to be authorised by the Department of the Environment for Northern Ireland. Galantas management considers it has a very strong case for approval in view of the positive economic impact of the project on the local community and the creation of employment opportunities.

Galantas has estimated capital and operating costs at US$ 26.7 million and US$ 25.0 million respectively based on detailed data and analysis.

The mine design proposal by Galantas is for a mining rate for the recovery of 50,000 ounces per year of gold in concentrate. ACA Howe regard this target as challenging, and have performed sensitivity analyses, using the same labour cost, for 40,000 ounces per year and 30,000 ounces per year gold in concentrate. A gold price of $1375/Oz has been used in each case and a USD/GBP exchange rate of 1.60 applied. The study is based upon detailed data available from the internal cost study by Galantas and the quoted Mine Life Revenue is net of Capital and Operating costs.

The results of this Preliminary Economic Assessment are as follows:

Case Study Output/Yr	Mine Life Revenue Net	IRR	NPV @5% £	NPV @ 5% US$	Mine Life Yrs	Cash Cost Gold in concentrate US$ per ounce
50,000 ozs	£80.1 m	81%	62.6 m	100.2 m	5	500 $
40,000 ozs	£75.4 m	69%	57.3m	91.6 m	6	538 $
30,000 ozs	£71.3 m	54%	51.2 m	82.0 m	8	600 $

For the purposes of the preliminary economic assessment, the Mine Life stated in each case has been calculated using the sum of Measured, Indicated and Inferred resources on the Joshua and Kearney veins only from the ACA Howe study, with Inferred resource making up some 70% of the total. In conformity with NI 43-101, section 2.3, it should be noted that this 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 categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

1. INTRODUCTION

Galantas Gold Corporation (Galantas, the Company), has retained ACA Howe International Limited (ACA Howe) to prepare an updated mineral resource estimate for the Cavanacaw gold deposit and an independent Technical Report, in accordance with the reporting standards and definitions required under Canadian National Instrument 43-101 (NI 43-101). This report presents the results of the mineral resource estimate and a summary of the project's geology and exploration potential.

The updated mineral resource estimate is based on historical exploration and drilling and channel sampling campaigns at the Omagh Gold Project that commenced in August 2011 and are continuing through 2012. The project database has been updated to reflect new drilling and channel sampling data generated from this work until the resource cut-off date of June 1st, 2012.

Galantas has carried out an internal 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 report includes a preliminary economic assessment based on parameters from that study.

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

This report documents exploration activities and data collation undertaken at the Omagh Gold Project during the period from April 2011 to April 2012, comprising channel sampling within the Kearney Pit and Joshua pit environs and exploration and resource definition drilling undertaken over the Kearney Deposit, Elkins Vein, Kerr Vein and Joshua Vein. In addition, and in the light of new drilling data, a formal, CIM compliant re-estimation and update of resources for the project has been undertaken and is detailed in this report.

Since December 2005, ACA Howe has assisted with the planning and implementation of the 2006-2007 and 2011-2012 drilling campaigns and monitored progress, data collection and data collation during this period.

ACA Howe has compiled the information and data contained in this report using a combination of:

- first hand data collection and observations from a total of nine site visits to the project undertaken by ACA Howe personnel between July 2006 and March 2012 in order to collect data and review drilling activities;
- information and data received from Galantas and third party sources listed below in the section "References and Sources", which we have assumed to be correct but which we have not independently verified although we are not aware of any information in those documents that is incorrect.

To the best of our knowledge, having taken all reasonable care to ensure that such is the case, the information contained in this report is in accordance with the facts and makes no omission likely to affect the import of such information.

2. RELIANCE ON OTHER EXPERTS

This report has relied upon information on the same property that is contained in previous reports filed on SEDAR (ACA Howe International Limited, July 21st, 2003; ACA Howe International Limited, August 20th, 2004; Galantas Gold Corporation, December 5th, 2005; Galantas Gold Corporation, December 5th, 2005; and Galantas Gold Corporation, July 18th, 2008).

Information in these reports is supplemented with information from other sources, listed in Section 27, References, and other sources such as published topographic and geological maps, government sources, geographical atlas information, internet resources, Galantas personnel, local knowledge and first hand observations and interpretations by the Qualified Persons.

While exercising all reasonable diligence in checking, confirming and testing it, ACA Howe has relied upon the data presented by Galantas, and any previous operators of the project, in formulating its opinion.

The various agreements under which Galantas holds title to the mineral lands for this project have not been thoroughly investigated or confirmed by ACA Howe. The descriptions were provided by Galantas as received from DETI and CEC, the entities responsible for licensing mineral exploration in Northern Ireland (see Section 3.1).

The description of the property is presented here for general information purposes only, as required by NI 43-101. ACA Howe is not qualified to provide professional opinion on issues related to mining and exploration title or land tenure, royalties, permitting and legal and environmental matters. Accordingly, the authors have relied upon the representations of the issuer, Galantas Gold Corporation, for Section 4 of this report, and have not verified the information presented therein.

3. PROPERTY DESCRIPTION AND LOCATION

3.1. MINERAL LEGISLATION AND LICENSING

The following text is a summary of information on the website of the Department of Enterprise, Trade and Investment (DETI) of the Government of Northern Ireland, http://www.detini.gov.uk.

The Mineral Development Act (Northern Ireland) 1969 (the 1969 Act) vested most minerals in the previous equivalent of DETI and, together with subsequent subordinate legislation, enables it to grant prospecting licences and mining licences and leases and to collect royalties for base metals extraction. Provisions for mineral exploration are separate from those for mining. There is no automatic continuity between exploration and mine development work.

The legislation covers all minerals with the exceptions of:

- gold and silver which belong to the Crown Estate and were not vested in the DETI,
- the few mineral deposits (mainly salt) which were being worked at the time of the 1969 Act, and
- common substances including crushed rock, sand and gravel and brick clays.

The Crown Estate Commissioners (CEC) issue prospecting Option Agreements for precious metals. Companies wishing to explore for precious metals would normally hold both CEC options and DETI prospecting licences.

Prospecting licences, which are exclusive to the holder, covering up to 250 km^2, are normally valid for two years with the possibility of two extensions of the same duration for a total of 6 years. Agreed work programmes and annual or more frequent reporting is required. Licensees are required to give up to 4 weeks' notice of intention to enter land and must seek the agreement of landowners. Compensation is payable by the licensee to the landowner for any damage which may be caused during exploration. Planning permission is not required for the early stages of exploration although the Planning Service of the Department of the Environment must be informed of the planned work including the nature and scale, time and location of the company's activities and drill hole locations.

Under the provisions of the 1969 Act, a company which wishes to open a mine requires a DETI mining licence or mining lease. There are separate provisions in the 1969 Act dealing with mining licences and leases. Mining requires planning permission from the Department of the Environment.

DETI collects royalties for base metals. Precious metals royalties are negotiated with and payable to the Crown Estate.

Legal verification of ownership of property and exploration and mining rights is outside the scope of ACA Howe's expertise but these aspects are summarised for general information and have been legally reviewed elsewhere by a specialist.

3.2. LOCATION

The Cavanacaw mine is located 5 km west-southwest of Omagh, centred on latitude 54° 35′ 00″ north, and longitude 7° 22′ 50″ west. This is equivalent to IH 40046E and 70748N on the Irish National Grid system which is used for topographic maps and for exploration data.

3.3. OMAGH MINERAL LICENCES

Through its wholly owned subsidiary Omagh Minerals, Galantas holds various exploration and mining rights and permits including exclusive exploration rights for gold and silver and other minerals, over the licence area shown in Figures 1 and 2.

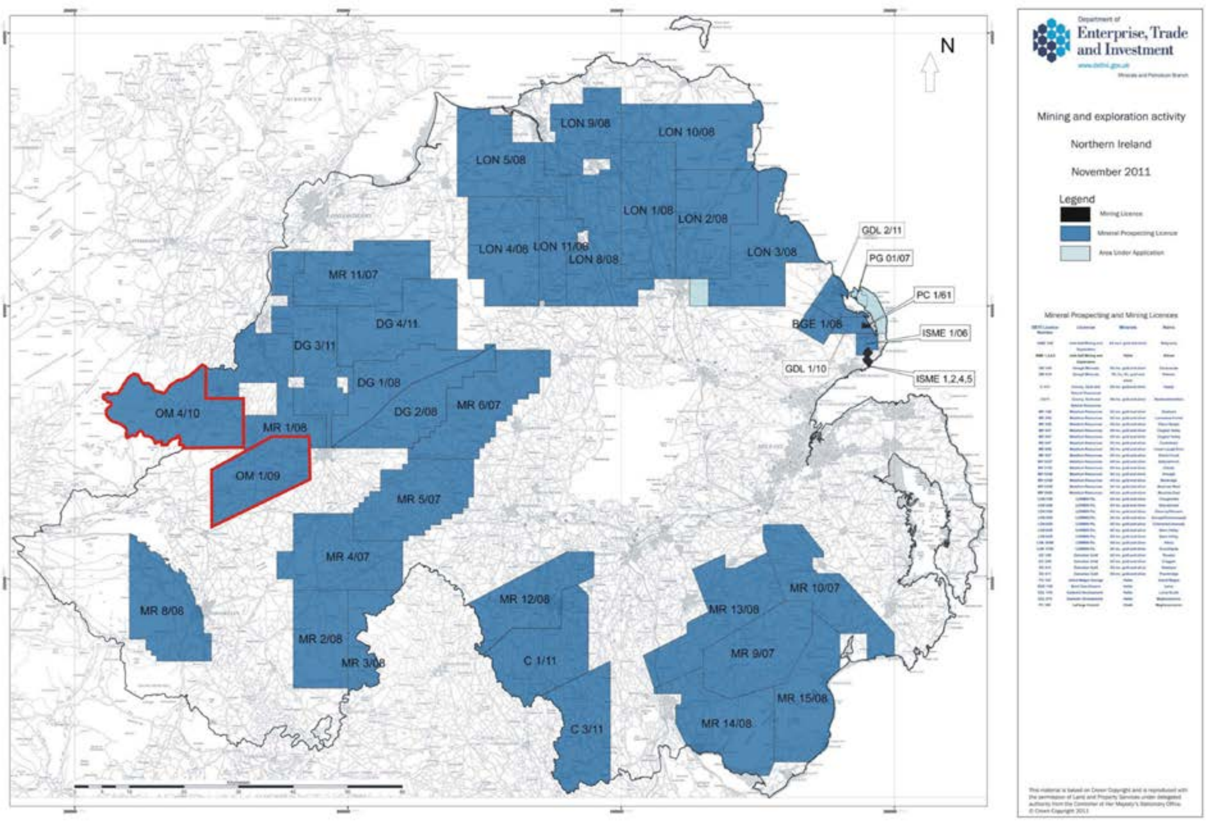


FIGURE 1. LOCATION OF PROSPECTING LICENCES OM1/09 AND OM4/10

On July 3rd, 2012, a representative of the Minerals division of OSNI confirmed to ACA Howe that the current prospecting licences OM 1/09 ($189km^2$) and OM 4/10 ($250km^2$), were both in good standing at that date (Figure 1). It was confirmed that a coincident Crown Estate licence exists for gold and silver.

On July 9th, 2012, a representative of the Crown Estate Mineral Agent confirmed to ACA Howe that Mines Royal Options Agreements for gold and silver, coincident with Prospecting Licences OM 1/09 and OM 4/10, were both in good standing at that date, and were due to expire on July 18th, 2013 and December 31st, 2012 respectively, and the Crown Estate had granted a Mines Royal mining lease to Omagh Minerals, dated April 24th, 2006 and expiring on June 22nd, 2015 for the area shown on Figure 1, covering most of the Omagh Minerals freehold, but excluding the Elkins veins.

Since the early 1990s, by various purchases, Omagh Minerals has built up the freehold title of the Omagh mine site in the townlands of Cavanacaw Upper, Botera Upper and Tattykeel, in Omagh, Co. Tyrone, which extends to about 67.6 hectares (167 acres) as shown in Figure 2.

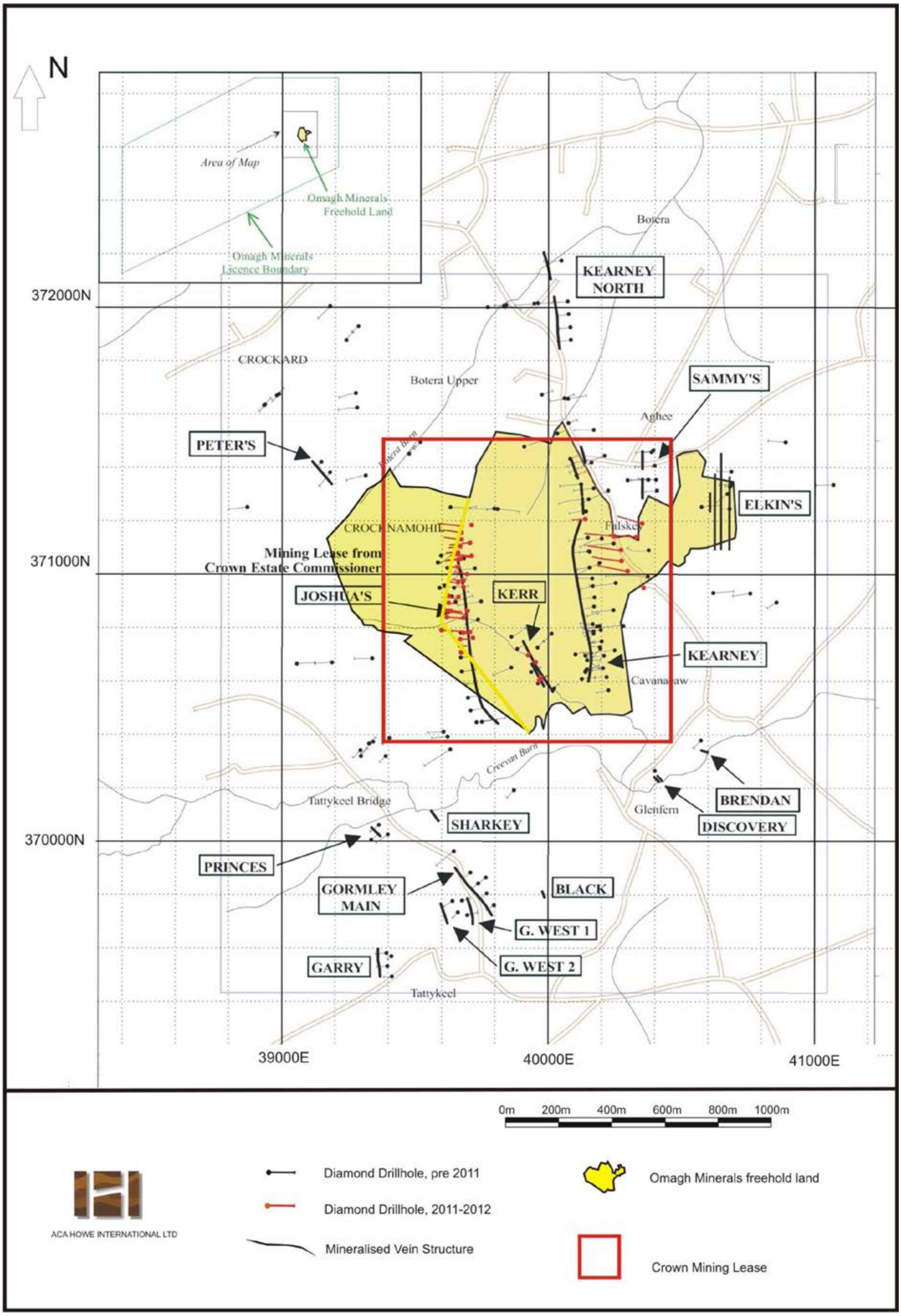


FIGURE 2. CAVANACAW PROJECT SHOWING FREEHOLD AND CROWN LEASE

3.4. PERMITS

Following a protracted public enquiry which convened intermittently for a total of 24 days, finally ending on January 19th, 1994, The Department of the Environment for Northern Ireland (DoE NI) granted planning permission for open pit mining of gold and silver and associated minerals in the Cavanacaw area (see Figure 3) on May 23rd, 1995, in relation to Application No. K/92/0713 by Omagh Minerals. Numerous conditions determined by the Public Enquiry were incorporated. On December 7th, 2005, a representative of the Omagh Divisional Planning Office of DoE NI confirmed to ACA Howe that the area of the land covered is about 62 hectares (153 acres). Since work had commenced within 5 years from the grant date, the permission remains valid.

Consent to Discharge of Effluent No. 2045/94, File No. TC 14/93, was issued to Omagh Minerals by the DoE NI on May 25th, 1995, incorporating numerous conditions determined by the public enquiry.

Under the conditions attached to the Crown Mining Lease and DETI Prospecting and Mining Licences (application number K/92/0713, granted May 23rd, 1995 and which remains valid) and the DoE NI consent to discharge effluent (file number TC 14/93), environmental liabilities exist and Omagh Minerals must satisfy environmental rehabilitation conditions including ripping and seeding of all waste dumps, restoration of open pit areas to include backfilling, tailings reclamation and seeding, disposing of waste material (an agreement was reached in 2008 with a haulage contractor to remove waste material from site for use in road construction, see Section 22) and minimal surface disturbance during exploration activities and payment of compensation to landowners in accordance with the conditions set out in the DETI Prospecting Licence.

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

The town of Omagh and the project area can be reached from Belfast by paved highway, more than half of which is motorway (see Figure 3). The road distance is approximately 95 km and requires less than 1.5 hours in good weather. Belfast is served by both a domestic and international airport with frequent daily flights to the rest of the UK and to Europe.

The Cavanacaw mine area is accessible over approximately 5 km of paved roads from Omagh. A number of farm roads also traverse the property in the main area of interest, though some of the unexplored upland sections are only accessible on foot. Two power lines of 33 Kv and 11 Kv traverse the eastern section of the licence, but power to the mill is presently generated by diesel.

The OM1/09 Prospecting Licence area lies on the south-western fringe of the Sperrin Mountains with elevations generally ranging from 140 to 160 m above sea level with rounded hills up to 330 m. Glacial tills up to 15 m thick support small livestock farms and low grade pasture on the lower slopes. Peat bogs on the upland sections support small scale manual peat cutting for domestic fuel. Renovation and new-build developments of individual houses on small plots are popular. There is some coniferous plantation forestry. A wind farm has been built on Tappaghan Mountain within the western part of the licence area in recent years.

The climate is temperate, characterised by cool summers and mild winters with about 1,500 mm of rainfall per annum. It is unlikely that mine production or exploration activities will be adversely affected by bad weather for any significant period. The area is not specially designated for scenic attraction or landscape value.

Omagh (population approximately 22,000) provides lodging and local labour, as do smaller local villages. Few experienced mining personnel are available locally, although the Irish Republic has a number of underground base metal mines with associated services and experienced personnel.

The principal economic activities in the immediate project area are sheep and cattle farming.

The freehold land area presently owned by Galantas (see Figure 2) covers sufficient area for provision of present and planned tailings storage areas, waste disposal areas, processing plant sites and other infrastructure.

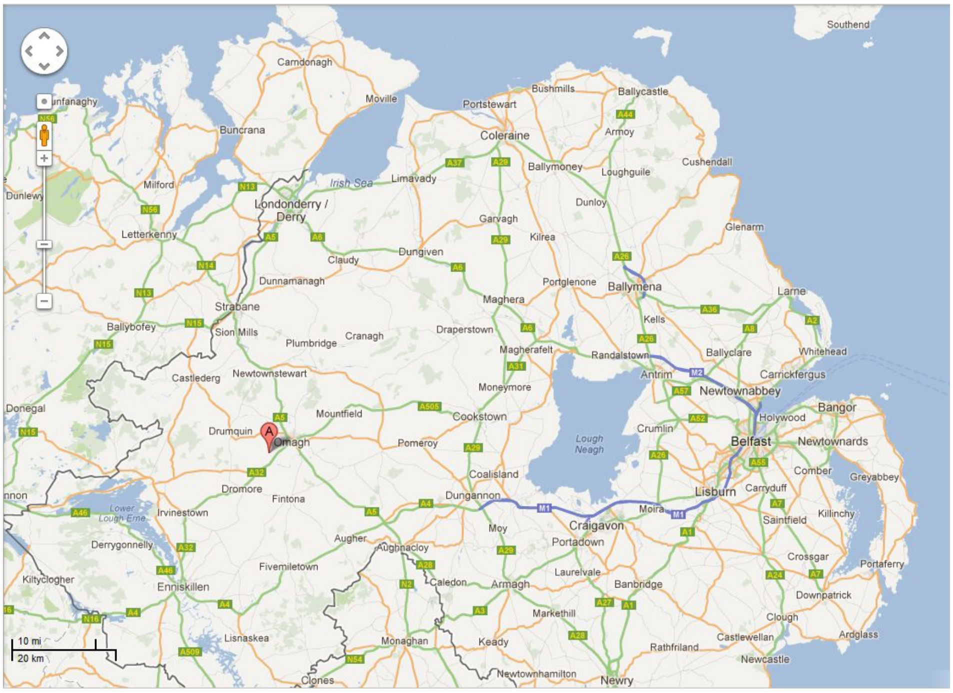


FIGURE 3. CAVANACAW MINE ('A'); LOCATION AND ACCESS

5. HISTORY

The occurrence of gold in the Sperrin Mountains in Northern Ireland has been known for several centuries but no mining operations have taken place prior to that at 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 metasedimentary rocks in the 1980s 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 et al., 1989).

5.1. PROJECT HISTORY

ACA Howe has documented the history of the Omagh project in the course of its involvement in the project, and these details of the project history are contained in previous reports (ACA Howe International Limited, 2003, 2005, 2008, all filed on SEDAR) to which the reader is referred. The following is derived largely from the 2008 report.

Following the discovery and exploration of vein gold at Curraghinalt in the Sperrin Mountains of Northern Ireland by Ennex International in the mid 1980s, Riofinex North Limited (Riofinex) commenced exploration of the geological inlier known as Lack, named after the village located 20 km west-southwest of Omagh town. Riofinex discovered the gold-bearing Kearney vein structure and the surrounding swarm of gold veins during the course of an exploration and resource delineation programme which included:-

- Geological mapping;
- Sampling of stream sediment, soil, loose boulders of mineralisation known as “float”, deep overburden using a petrol driven, hammer sampling tool known as a Pionjar,bedrock;
- Core drilling;
- Overburden stripping and intensive saw-cut channel sampling of exposed mineralisation in bedrock;
- Resource estimation;
- Evaluation of a mining project based on the Kearney structure;
- Environmental baseline studies.

Due to the early discovery of the Kearney vein swarm, the efforts of Riofinex were more concentrated in the relatively small surrounding area in the eastern part of the project area around Cavanacaw townland.

In 1990 the Riofinex project was transferred to Omagh Minerals who commissioned Kilborn Engineering Limited and Knight Piesold to study metallurgical recovery and the practicalities of mining with a focus on the Kearney deposit. Simultaneously, Wardell Armstrong carried out an environmental Impact Assessment (EIA) which was completed by late 1992. The Crown Estate Commissioners awarded a Lease of Rights to work gold and silver (Crown Estate mining lease) in 1993 with a 10 year initial term, over land including the Kearney vein, with effect 1 month after the award of Planning Permission.

Following a Public Enquiry in 1993 and 1994 which convened intermittently for a total of 24 days and ended on January 19th, 1994, conditional planning permission for a mining and processing operation was granted in 1995 over land owned by Omagh Minerals and Consent to Discharge Effluent was given. Planning permission remains valid since work commenced within 5 years of granting. All of the 40 planning conditions were fulfilled in 2001, enabling project development to commence.

In 1995, in line with the permission and consent conditions, Kilborn produced a revision of its earlier study, based on a larger forecast mill throughput.

In 1997, European Gold Resources Inc. (EGR) of Ontario acquired Omagh Minerals. Omagh Minerals excavated another open cut on the Kearney structure, commencing some 44 m north of the Riofinex excavation, and cutting some 5-6 m of vertical section through the deposit. The exposed bedrock surface was mapped and sampled in a similar way to the Riofinex open cut. ACA Howe carried out additional, extensive stream sediment geochemical surveys over the licence area and digitised the results and the Riofinex geochemical data. In 2000 and 2001, Omagh Minerals carried out selective mining trials at the southern end of the Kearney structure and produced high grade, sulphidic gold ore. In the following years Omagh Minerals produced gold bullion with full accreditation of the Irish source and produced and sold 18 carat gold jewellery under the Galantas brand name of a wholly owned subsidiary.

In 2003, EGR owned Cavanacaw Corporation of Ontario, which in turn owned all of the shares of two Northern Ireland Companies, Omagh Minerals Limited and Galantas Irish Gold Limited. EGR commissioned ACA Howe to prepare a technical report in compliance with Canadian National Instrument 43-101(ACA Howe International Limited, 2003, referred to herein as the 2003 report).

In 2003, EGR also commissioned ACA Howe to carry out compilation of exploration data and analysis of Landsat satellite imagery over the whole of the Lack inlier. These data were compiled in a geographical information system (GIS) using MapInfo software. Guided by interpretation of the GIS data, ACA Howe also carried out reconnaissance sampling, mapping, data compilation and interpretation to characterise and classify the outlying targets for further follow-up work, outside the small area, 2.8 by 2.3 km, which had been studied intensively by Riofinex and Omagh Minerals (ACA Howe International Limited, 2004A). The report is summarised in the 2008 NI43-101 report and concluded as follows:

> *Twenty four exploration targets were identified, widely distributed within the area of Dalradian rocks. Eleven targets are classified as being of high priority; seven of medium priority and six of low priority. All the high priority targets contain multiple sample points with direct assay evidence of significantly enhanced gold content in rock or soils, in areas with strong structural features, some with associated geophysical anomalies.*
>
> *Follow-up on these targets resulted in the discovery of gold mineralisation at Cornavarrow Burn East, where a 6.5 m shear zone containing disseminated pyrite and galena included a 1.5 metre section returning 1.15 g/t gold, 4.2g/t silver and 1366 ppm lead.*
>
> *Excellent prospects for new discoveries exist throughout the Prospecting Licence area including the area intensively explored and drilled by Riofinex and the outlying areas studied in this report. ACA Howe recommended a programme comprising geological, geochemical and geophysical prospecting followed by trenching, overburden stripping and diamond drilling at an estimated cost of US$241,725*

European Gold Resources Inc. was renamed Galantas Gold Corporation (Galantas) 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.

Airborne time-domain electromagnetic (VTEM) and magnetic surveys were flown by helicopter over most of the exploration licence including the Dalradian rocks of the Lack inlier in the summer of 2005 (Geotech Airborne Limited, August 2005). The main objective was to locate electromagnetic anomalies which may be due to conductive mineralised structures. The results identified numerous new geophysical targets and helped to prioritise the existing targets.

In December 2005 ACA Howe reviewed the project targets and prioritised them according to resource potential. They concluded that eight vein structures: Kearney, Joshua’s, Kerr, Gormley Main, Elkin’s, Gormley West 2, Princes and Garry had good potential for upgrading of the reserves and resources of the 1995 and 2004 studies. The results of this work are contained in Appendix I of the ACA Howe 2005 report.

Galantas continued their preparations for initial mining operations and, following development of a conceptual mining strategy at the Kearney pit, started to build the ore processing plant in November 2005 and commenced mining development in early 2006.

5.2. HISTORICAL MINERAL RESOURCE AND RESERVE ESTIMATES

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

Historical estimates of reserves and resources for the project and the data on which they are based are described in previous reports by ACA Howe International Limited (2003, 2004B, and 2008). 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 these are summarised below.

5.2.1. 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 ongoing 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 300 m 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.

5.2.2. 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 850 m strike length of a proposed open pit designed by Kilborn Engineering, and to a depth of 37 m (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 4.43 m 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 m.

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 shown in Table 1 below.

TABLE 1. 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**	

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.

5.2.3. CONFORMITY OF 1995 RESOURCES TO CIM CLASSIFICATION

Although justified under the reporting code of the time, extrapolation of surface channel data over the entire 850 m 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 14.10) 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 137 m based on sparse drill hole data at spacing of between 75 m and 200 m 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 137 m 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 (50 m by 50 m and 100 m by 100 m), and lack of demonstrated continuity, adhering to current CIM guidelines.

5.2.4. 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 m. 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 m 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 2 below.

TABLE 2. 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

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.

5.2.5. 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 m long section in the south end of the Kearney vein which had been stripped and sampled in the late 1980s 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 1 m 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 m 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 m by 6 m 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 2,870 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 (Table 3). 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 shown in Table 3 below.

TABLE 3. 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**	

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 after a significantly run of mine grade was found to be unachievable.

5.2.6. 2008 RESOURCE ESTIMATE

In 2008, 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 2008 NI 43-101 report, to which the reader is referred.

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 m (X), 0.5 m (Y), 0.5 m (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 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 11th, 2005.

TABLE 4. 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

Since 2008 this estimate has been subject to depletion by open pit mining and bulk sampling at Kearney, Kerr and Joshua veins and additional drilling and trenching have been carried out. The 2012 revised estimate discussed in Section 14 takes account of these factors.

5.3. 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 its economic limits as dictated by stripping ratio, by the property boundary and road to the east, and by rock stockpiles to the west. Mining methods and production are discussed in Section 16 of this report.

6. GEOLOGICAL SETTING AND MINERALISATION

The geological setting and the gold mineralisation are described from a combination of information identified in References and Sources (see Section 27) 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 4. The veins of the Kearney swarm are depicted in Figure 2.

6.1. 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).

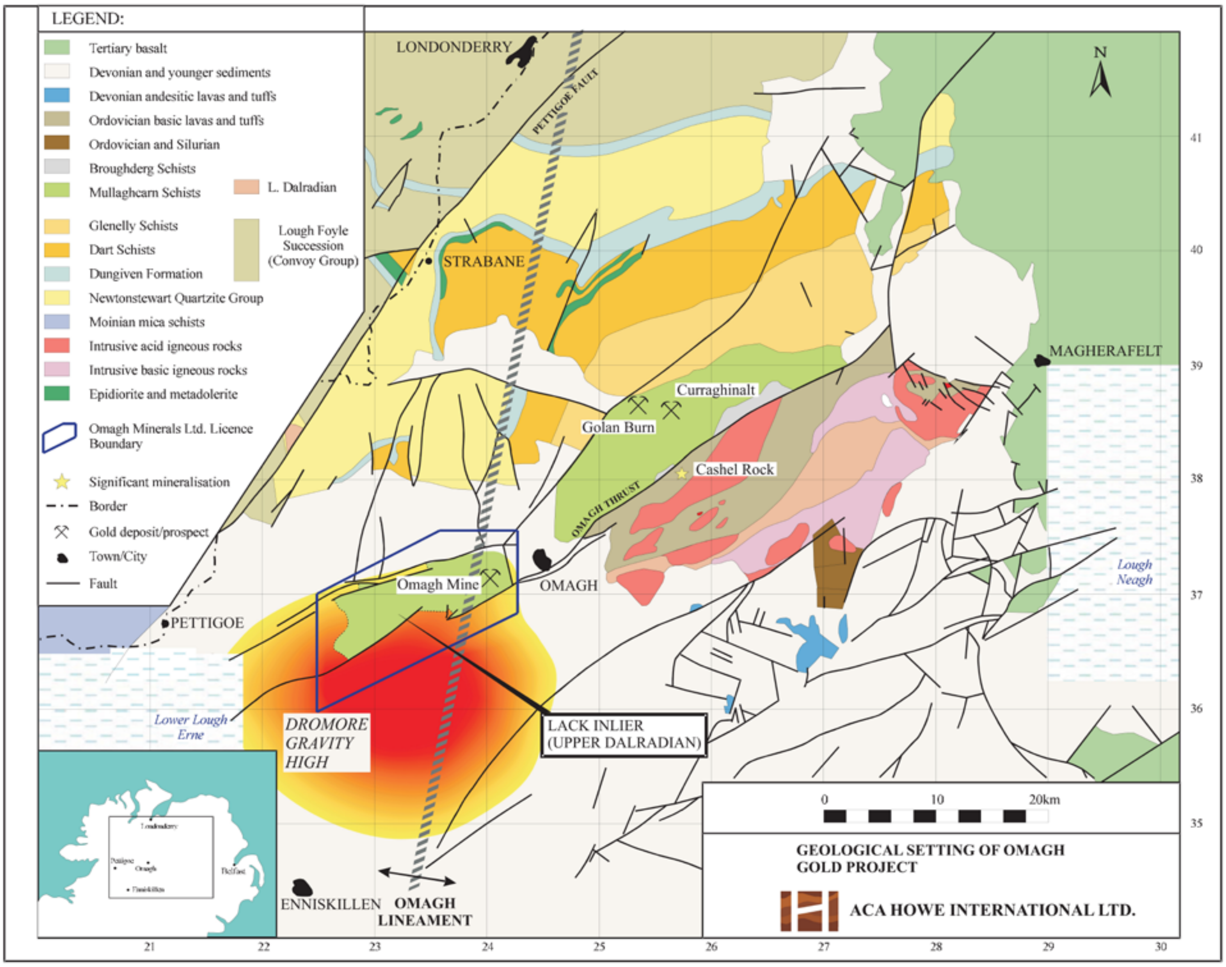


FIGURE 4. GEOLOGICAL SETTING OF OMAGH GOLD PROJECT

6.2. 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, psammite and pelite 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° in various strike directions but generally to the north-northwest. Minor fold axes are indicated, plunging at 6 to 40° towards west-southwest in the western half of the inlier. One plunge symbol in the north - central area plunges east-northeast at 20°.

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 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 6 km 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.

The Devonian and Carboniferous sedimentary rocks do not appear to hold gold potential based on the absence of geochemical anomalies and regional gold occurrences.

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.

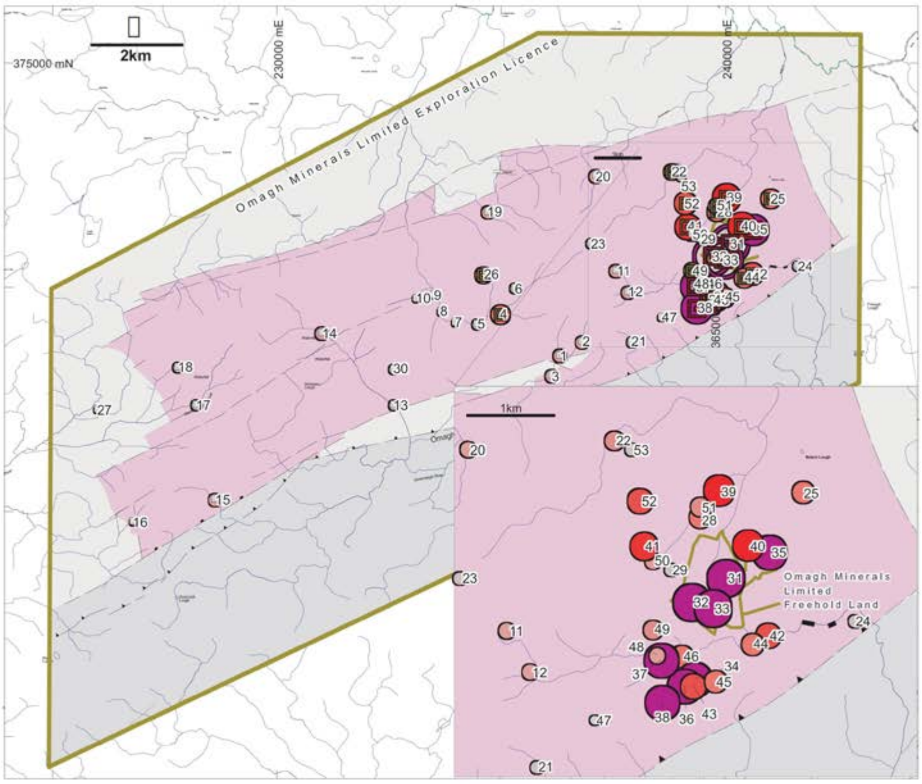
A major positive gravity anomaly known as the Dromore High is centred 10 km south of the centre of the Lack inlier (see Figure 4). A northern lobate “ridge” of this gravity anomaly trends east-northeastwards, coincident with the centre of the Dalradian inlier. Although the reason for the anomaly remains 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 (see Figure 2). 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 km^2 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 (see Figure 2). The largest of these is the Kearney vein with strike length of 850 m (1,000 m including an IP anomaly) and widths up to 6.6 m or more, dipping eastwards at 70°. The maximum vertical extent proved by drilling to date is 240 m.

The Cornavarrow Burn showings are named Cornavarrow Burn East Showing and Cornavarrow Burn West Showing. These are located some 5km 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 m 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° 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 m in horizontal width, possibly dipping west at 20°. 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 (see Figure 5).



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 5. LACK INLIER: GEOLOGY AND EXPLORATION TARGETS

6.3. MINERALISATION

Gold mineralisation is known in the Cavanacaw vein swarm (Figure 3) and in two showings in the Cornavarrow Burn some 5 km to the west. Numerous other mainly geochemical targets for undiscovered gold mineralisation exist throughout the licence area (see Figure 5) 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 (Figure 2).

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. Figure 3 shows the outlines of the veins currently known and the drill holes from which they were identified. 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 pelites, 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.

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.

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 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 m 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 pelites and is generally narrower in the more ductile chloritic and graphitic pelites. 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.

7. DEPOSIT TYPES

The Cavanacaw deposit can be characterised as of 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, characterise 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.

8. EXPLORATION

Galantas has carried out extensive exploration covering the area of its Lack licence OM 1/09 during the period 1995 to 2012. This work has identified numerous targets within the area of the Lack inlier. This exploration and the resulting targets are discussed in detail in the 2008 NI 43-101 report, and summarised in Section 5 (History) of this report.

Table 5 below summarises the year and amount of channel sampling and diamond drilling carried out at the Cavanacaw mine site in historical and recent times.

TABLE 5. SUMMARY OF HISTORICAL AND RECENT EXPLORATION AT CAVANACAW MINE SITE

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
		total	**492**			**4,281**
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
		total	**253**			**26,380**

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

8.1. CHANNEL SAMPLING, 2011-2012

8.1.1. KEARNEY VEIN

The Kearney vein was sampled at various times by Rio Tinto and Galantas over the period 1987 to 2008 as discussed in the 2008 report and in Section 6 of this report.

During 2011-2012 a limited programme of channel sampling was carried out over veins exposed in benches on the northern end of the Kearney pit, comprising nine channels and 230 samples. The assay results indicated sporadic high grades across narrow widths, with a maximum intersection of 11.5 g/t Au across 80cm in channel OM-CH-12/KY-09. Figure 6 is a photograph of the sampled veins after extraction of veins by selective mining.

FIGURE 6. KEARNEY PIT LOOKING SOUTH AFTER EXTRACTION OF VEINS

8.1.2. JOSHUA VEIN

A 225 m long section of the Joshua vein was exposed by stripping overburden in a large trench during 2011.

A total of 2,242 channel samples were collected at 10 cm intervals along 75 channels 1 to 5 m apart covering a 225 m section of Joshua vein between 370700N and 370920N. The nominal dimensions of each channel were 10 cm wide by 5 cm deep and 10cm long. Most channels were extended across the vein and 20 cm to 40 cm into wallrock.

Results indicated two richer sections with an average of 7.7 g/t Au over an average of 2.3 m width and 120 m strike length for the southern section of the Joshua structure (OMCH11-JS-01 to -25), and 10.9 g/t Au over an average of 1.4 m width and 38 m strike length for the northern section.

Figure 7 below depicts the location and assay results of this work.

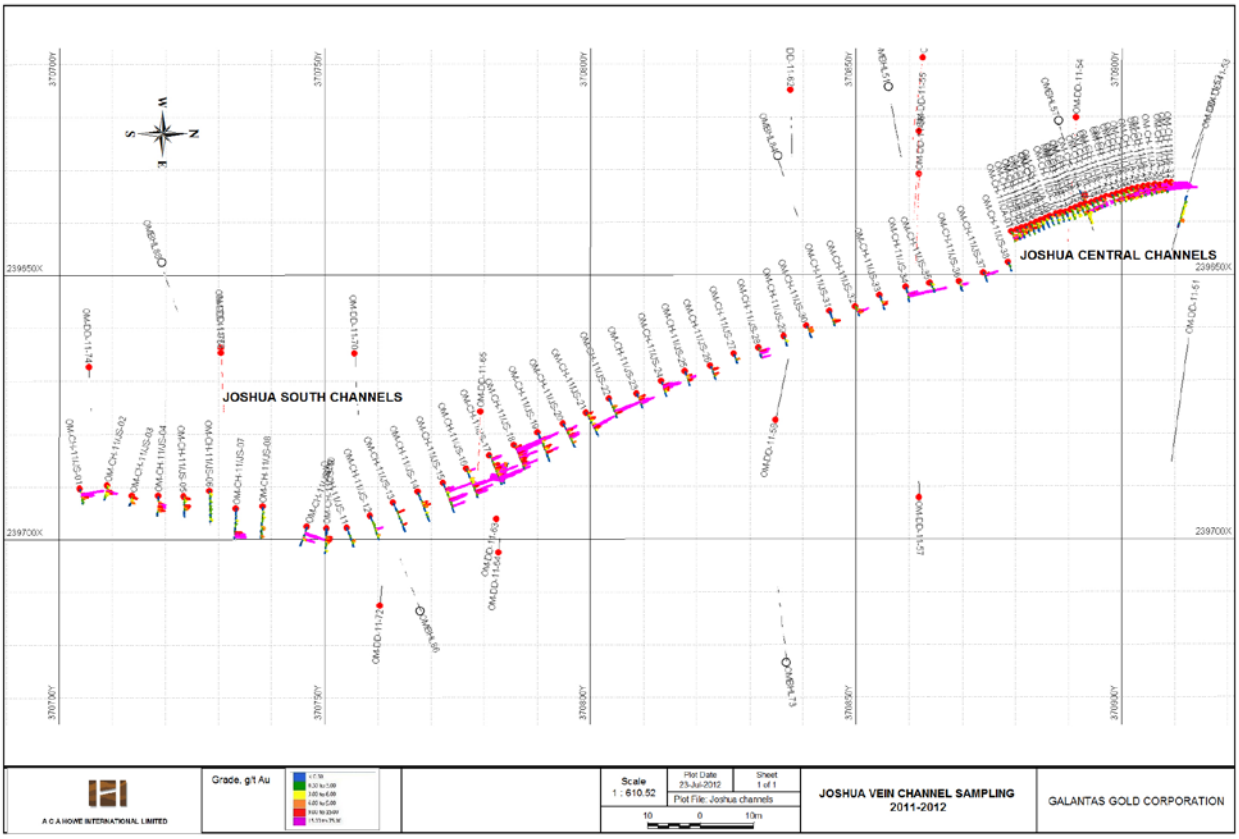


FIGURE 7. JOSHUA VEIN CHANNEL SAMPLING

8.1.3. KERR VEIN

The Kerr pit is approximately 5 m deep and extends over an area of about 75 by 75 m. It exposes four steeply dipping quartz-sulphide veins trending north-northeast. The veins are generally less than 1 m wide and extend over strike lengths of up to 25 m before pinching out.

A total of 28 channels were cut at 5 m intervals across the width of the veins and extending into barren wallrock. The channels were sampled at 10 cm intervals. The assay results for the 428 samples so collected indicated sporadic high grades across narrow widths, with a maximum intersection of 22.78 g/t Au across 40 cm in channel OM-CH-11/KR-22.

9. REGIONAL EXPLORATION

A large number of targets have been identified by past exploration on prospecting licence OM 1/09 as discussed in Section 6 of this report and shown in Figure 5. Exploration on prospecting licence OM 4/10 has mainly comprised reconnaissance stream sediment prospecting and some follow-up soil sampling.

ACA Howe notes that other prospects in the Dalradian including Curraghinalt, Cononish, Golan Burn and Glenlark were discovered by visual prospecting of float boulders in walls and streambeds. ACA Howe considers that this method is an effective additional means of investigating the already known prospects, and could also be applied in a systematic way to investigation of the entire Dalradian area held under licence.

10. DRILLING

10.1. OVERVIEW

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

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). Collar locations for all phases of drilling are shown in Figure 8 and drill hole details are contained in Table 6 below:

TABLE 6. CAVANACAW DRILLING 2006-2012

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
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-11-95	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-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-11-102	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
				TOTAL	5819.4	2006-2007	
				TOTAL	6552.1	2011-2012	
				TOTAL	12371.5	2006-2012	

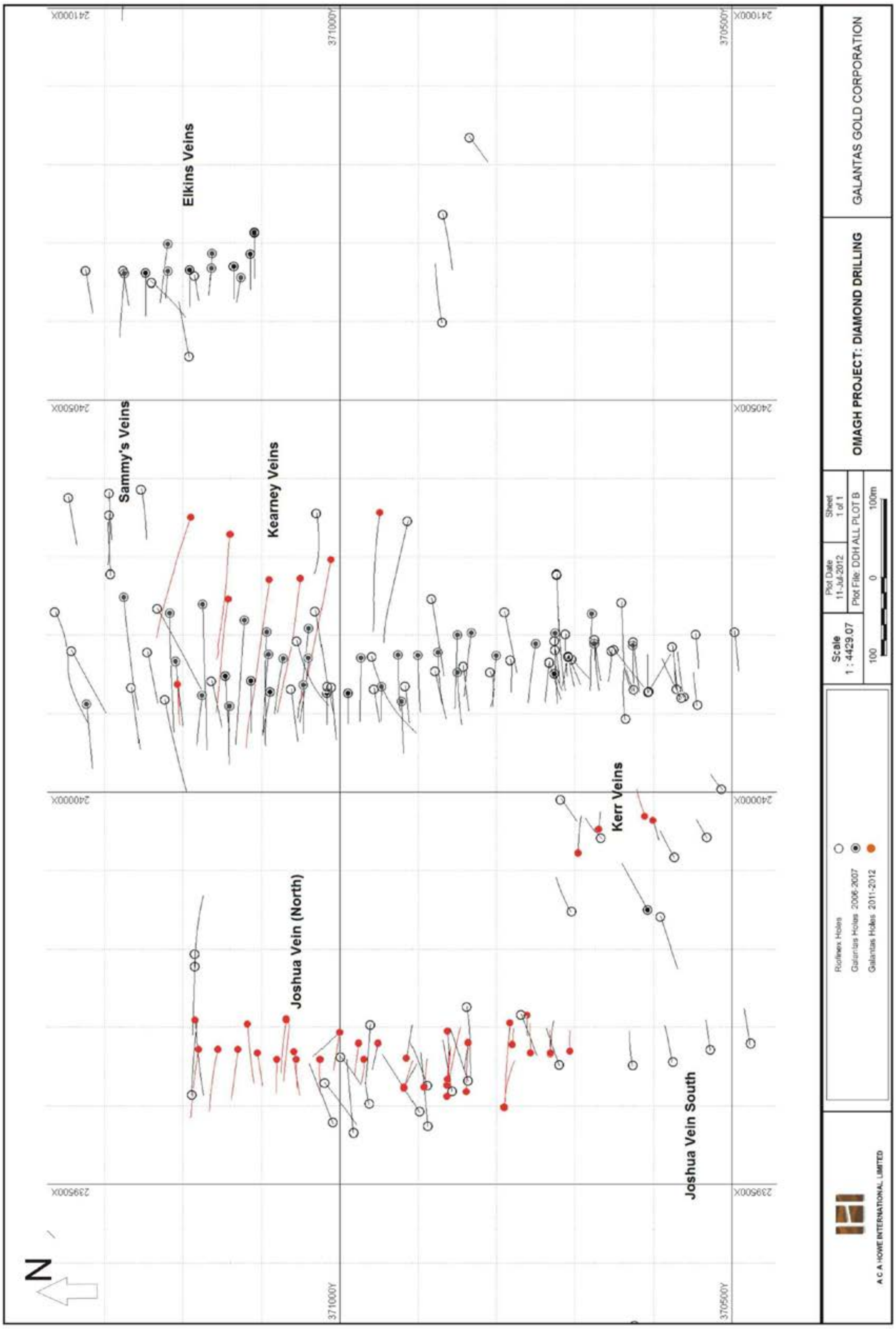


FIGURE 8. CAVANACAW DIAMOND DRILL LOCATIONS

10.2. DRILLING METHODOLOGY

Priority Drilling Limited was contracted to undertake the drilling and utilised Atlas Copco CS14 and JKS Boyles BBS37 rigs with wireline equipment. Some holes were drilled using Galantas' own Atlas Copco CS14 rig.

Core size was predominantly HQ3 (61.1 mm) with reduction to NQ3 (45 mm) in some deeper and more difficult holes. Triple tube core barrels were used to ensure good core recovery. All drilling activities were supervised by site personnel and data collection methodologies monitored by ACA Howe during the drilling campaign to ensure industry standard best practices for drilling were adhered to.

Each drill site was located by GPS and marked on the ground prior to the arrival of the rig. At any one time during the program there were up to six rigs present on site.

Drilling was undertaken in 3 m drilling runs and core transferred from the core barrel to wooden core boxes. The drilling contractors placed core blocks in the boxes to mark the end of each run, and the drilled depth was written on to the blocks. Once in the core boxes, the core was cleaned ready for inspection by the supervising geologist and transport to the core facility for processing.

Down hole surveying was completed by using Flexit MultiSmart multi-shot surveying equipment to collect angle and azimuth data at 6 m intervals down-hole, supplied by the drilling contractor. The surveying device is controlled via a StoreIT data pad, PC or palm-held unit and readings were collected at 6 m intervals down the hole upon completion of drilling. Data collected on site was transferred directly to ACA Howe unmodified, via e-mail, validated and appended to the Micromine database for the project.

At the rig, core boxes were clearly marked with the hole number, box number, measured intervals and start depth and transported from the drill site to the core processing facility using a tracked buggy to minimise core disruption during transport. The core processing facility, a converted barn situated in close proximity to the site offices, was well set up for processing core having custom built racks, protection from the elements and natural light.

The core was stored in racks prior to processing, and then laid out on an angled bench holding 25 boxes. 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 logging was recorded on detailed hand written log sheets and then manually entered into digital logs which were then forwarded to ACA Howe and merged into the Micromine database for the project.

Once logged, core boxes were photographed when both wet and dry and the photographs stored on file. Core boxes were ordered and placed in racks ready for sampling.

During the drilling program, ACA Howe was able to monitor drilling practices during frequent site visits and is able to verify that drilling practices implemented by Galantas during the program conform to standard industry best practice, and that survey control has ensured that logged and sampled geological zones can be confidently located in 3D space.

Galantas has recently commenced the orientation of core using a Reflex Act II instrument which is utilised to mark the end of the core with a vertical mark, such that an orientation line can be correctly run along the length of the drill core. Correctly oriented core can then be used to reference structural data collected during logging so that a more accurate picture of structural orientations and controls can be generated.

10.3. CORE RECOVERY

Core recovery during the 2006-2007 drilling was generally high, with average core recoveries for each hole ranging from 80% to 99%. Good core recovery continued throughout the 2011-2012 drilling. However these figures refer to the total core loss and not to the more critical core loss in mineralised intersections where core loss was sometimes significant, generally associated with broken vein material and with extreme contrasts in competence between soft clay gouge and hard quartz vein fragments. Average core recovery in the eighteen best intersections of 2011-2012 was around 60%, as shown in Figure 9 below.

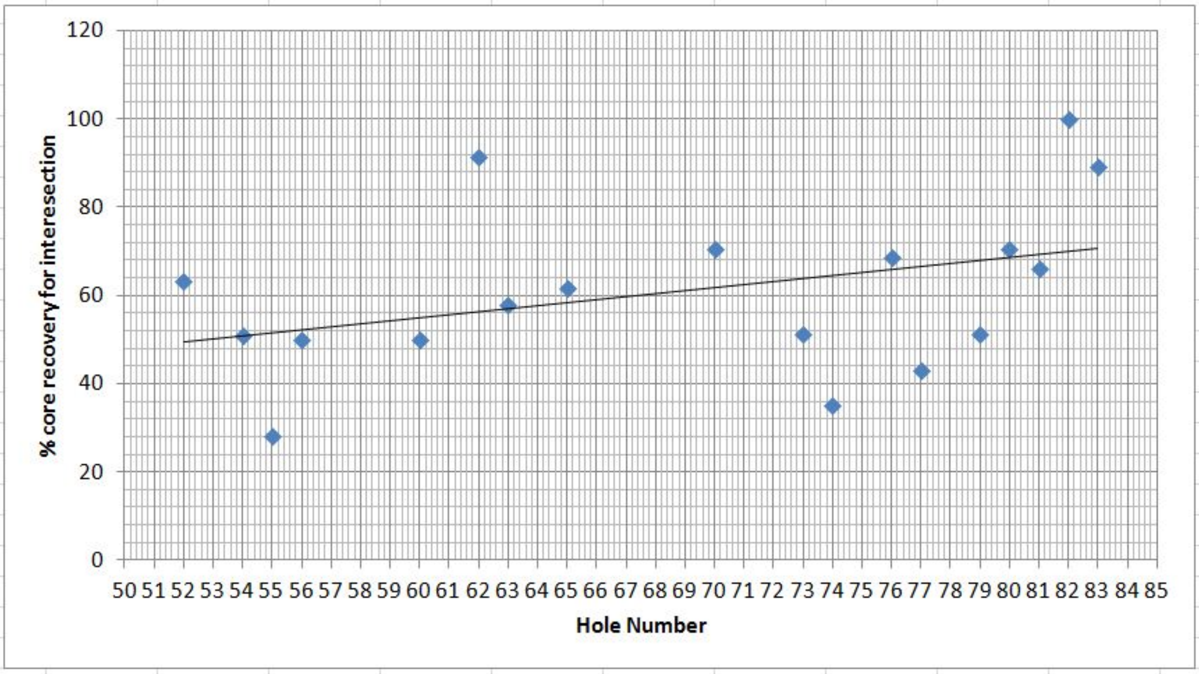


FIGURE 9. CORE RECOVERY 2011-2012 DRILL INTERSECTIONS

There is a possibility that the representativity of the resulting core samples is impaired by poor core recovery, especially if gold grades are unequally partitioned between competent and incompetent core. However there is no significant correlation between core recovery and gold grade (correlation coefficient = -0.057) as is shown by Figure 10 below. ACA Howe therefore concludes that poor core recovery is not so serious as to invalidate the use of Galantas drill core samples for resource estimation.

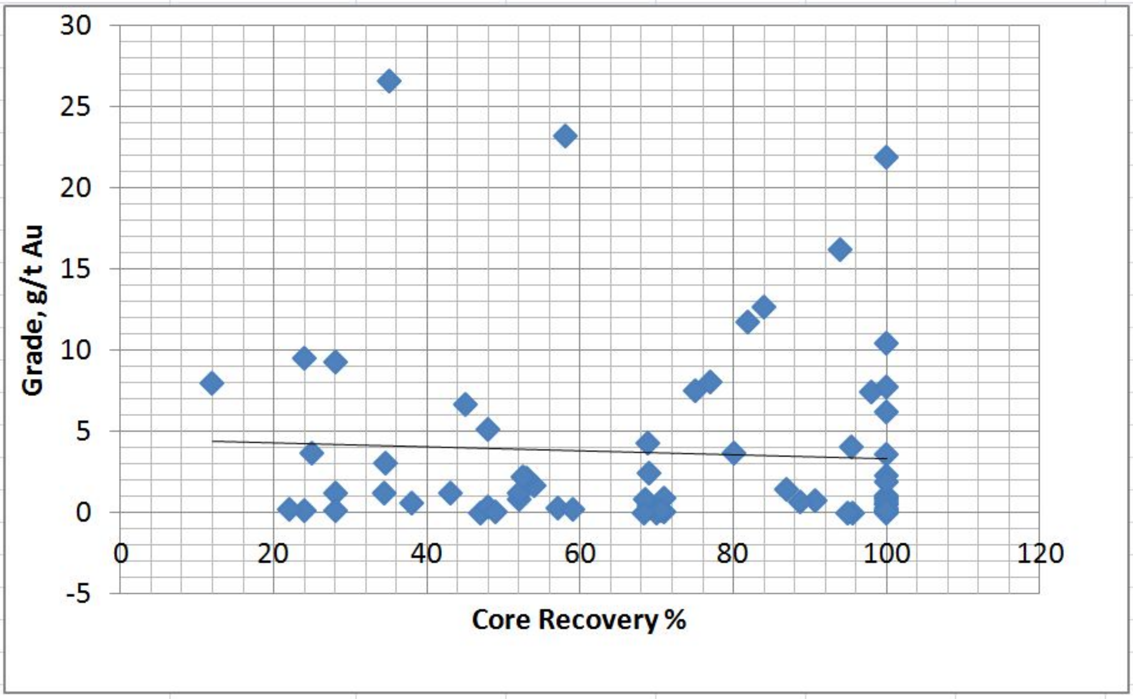


FIGURE 10.CORE RECOVERY V GRADE; 2011-2012 DRILL INTERSECTIONS

Records of core recovery for Rio Tinto's drilling are not available but it is understood that conventional core barrels were used and core recovery was correspondingly poor. In a number of twin holes drilled on the Joshua vein, good intersections in Galantas holes contrast markedly with poor intersections in nearby Rio Tinto holes. In these instances, only the Galantas intersections were used for resource estimation. Elsewhere on the Joshua vein, Rio Tinto intersections were used only for the estimation of Inferred resources.

10.4. DRILLING RESULTS 2011-2012

Significant drill intersections exceeding 1m at 3 g/t Au are listed in Table 7 below:

TABLE 7. CAVANACAW DRILL INTERSECTIONS 2011-2012

Hole ID	Location	from, m	to, m	width, 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

10.4.1. JOSHUA DRILLING

Thirty six diamond drill holes totalling 3,153 m were drilled at Joshua, targeted on proving the depth extent and grade of the mineralisation that is exposed and which was sampled by detailed channel sampling.

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.

Intersections for 2011-2012 drilling are listed in Table 7. These include an exceptional value of 26.6 m (7.6 m true width) at 8.44 g/t Au in OM-DD-11-103.

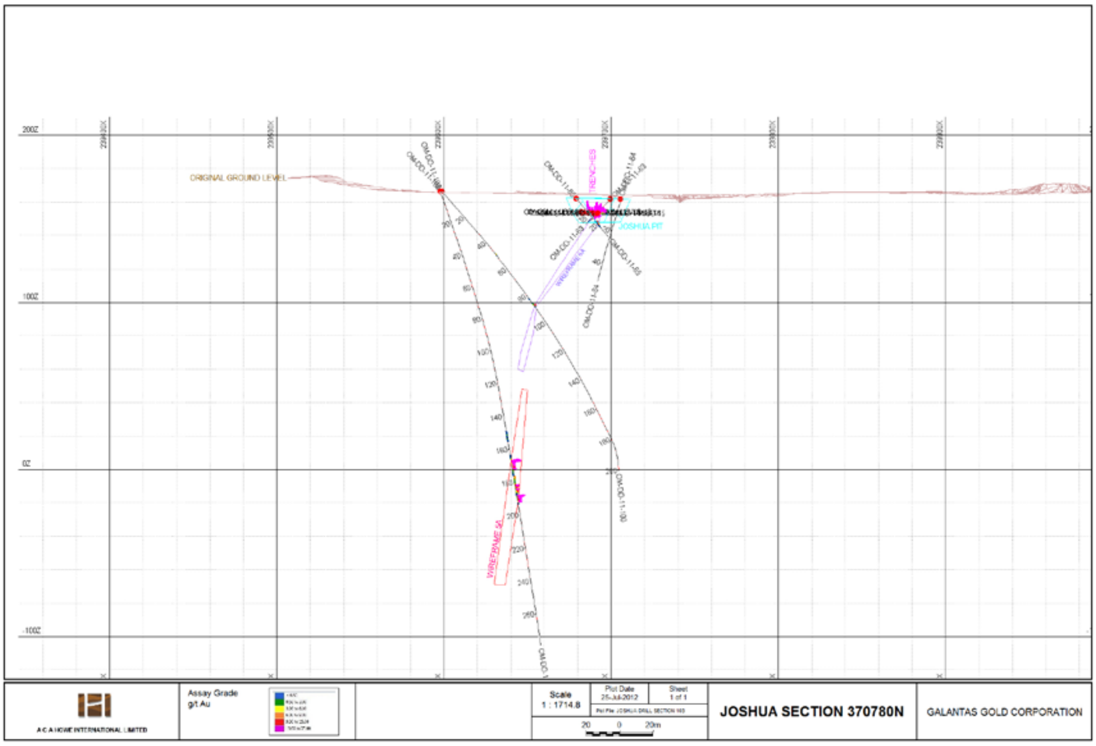


FIGURE 11. JOSHUA DRILL SECTION 370780N

10.4.2. KEARNEY DRILLING

Exploration and resource drilling at the Kearney Vein had two objectives:

- to upgrade the Inferred resources of 2008 for the Kearney Vein, above 0 m elevation, to Indicated category (down to approximately150 m below surface);
- to identify additional Inferred resources below 0 m elevation and below the Inferred resources of 2008, down to the minus 160 m elevation (down to 310 m below surface).

Intersections for 2011-2012 drilling are listed in Table 7. These include an exceptional value of 6.89 m (3.5 m true width) at 11.2 g/t Au in OM-DD-11-90B.

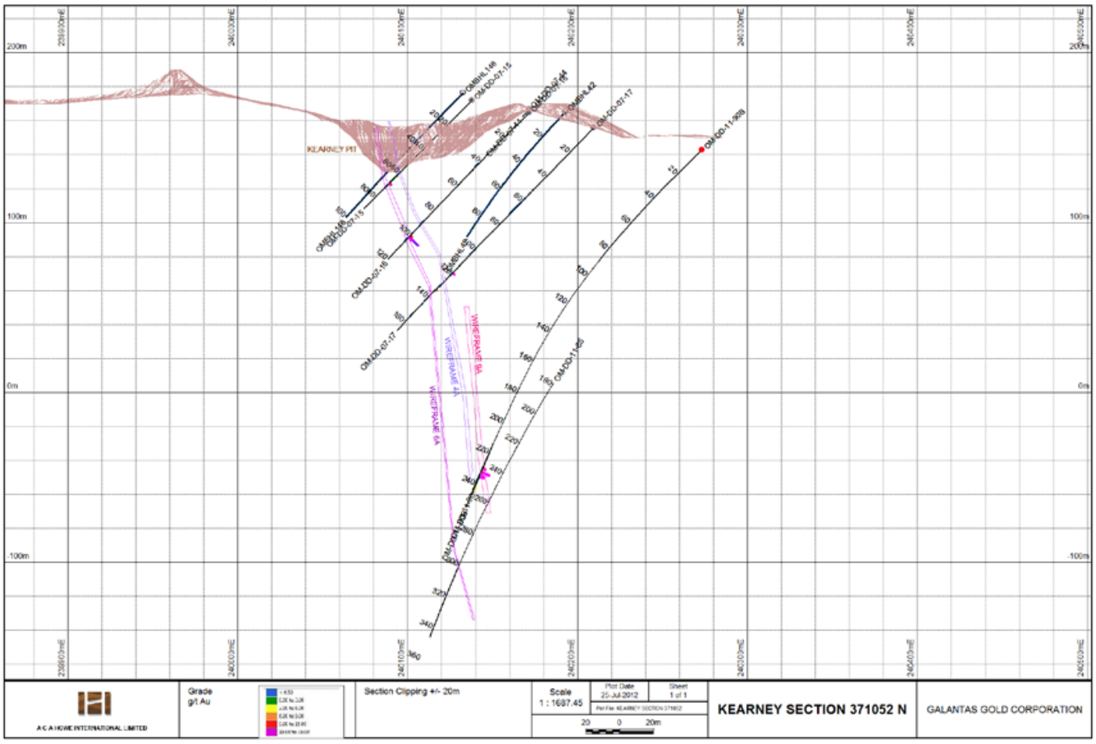


FIGURE 12. KEARNEY DRILL SECTION 371052N

10.4.3. KERR VEINS DRILLING, 2011

Holes OM-DD-11-68 and OM-DD-11-69 were collared immediately north of the known strike extent of the Kerr veins and targeted to intersect the veins at 10-30m depth. Neither hole intersected significant mineralisation, and it was concluded that the Kerr veins attenuate rapidly at depth and to the north.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

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 0.5-2.0 kg were collected at 10 cm intervals from logged intervals of 5 cm or 10 cm 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 programs was selectively sampled with sample intervals based on mineralisation and lithology. Samples of wallrock were taken 1 m 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 has 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 (OMAC) of Loughrea, County Galway, Republic of Ireland along with two inventories of submitted samples. One copy was signed by laboratory staff as proof of receipt and returned to Galantas, whilst the other copy was signed and retained by laboratory staff for reference. As a precaution, each sample submission sheet was e-mailed directly to the laboratory prior to delivery of samples.

Analysis of all samples generated from channels and drill core was undertaken by OMAC. 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 Program 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 program 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 1 kg 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.

Gold fire assay uses a 50g pulp sub-sample, fused with lead oxide/carbonate/borax/silica/flux at 1,100°C using silver as a carrier. Fusions producing lead buttons weighing less than 30 g are rejected. After de-slagging, buttons are cupellated at 950°c. Prills are parted in dilute nitric acid and finally dissolved in Aqua Regia. Reading is by flame atomic absorption to 0.01 ppm using a Varian SpectrAA-55 instrument. This procedure follows OMAC/ALS code Au4.

Base metal analysis by ICP (OMAC/ALS code ME-ICPORE) uses an oxidation digestion with the final solution in Aqua Regia. Results are reported for 19 elements, namely Ag, As, Bi, Ca, Cd, Co, Cu, Fe, Hg, Mg, Mn, Mo, Ni, P, Pb, S, Sb, Ti and Zn.

11.1. QUALITY ASSURANCE AND QUALITY CONTROL

OMAC operates an extensive quality assurance and quality control (QA/QC) programme which covers all stages of the analytical process from sample preparation, sample decomposition, instrumental finish and final reporting. External and in-house standard reference samples and blank samples are routinely included in each batch of samples received. Batches of no more than fifty samples at a time were processed and OMAC included one standard reference sample and one blank sample per batch, and a repeat assay for every tenth sample undertaken.

The QA/QC results for 2006-2007 drilling are discussed in the 2008 report which concluded that that the analytical techniques employed by OMAC are reliable in producing assay data that demonstrate a high level of accuracy and precision and that sample preparation practices employed at OMAC are satisfactory so as to minimise the risk of contamination. Therefore, assay results from drilling and sampling programs implemented during 2006-2007 may be regarded as representative of the samples collected. However, results from the submission of 2006-2007 field duplicates (1/4 core samples) indicated that some sampling error may have been introduced that may have affected the representativeness of the core samples collected for duplicate analysis.

11.1.1. OMAC INTERNAL LABORATORY QA/QC

As part of internal laboratory QA/QC, OMAC routinely conducted repeat assays on every tenth sample and inserted one sample reference standard and one blank sample in every batch of samples. During drilling activities, ACA Howe closely monitored the returned assay data from each batch of samples. OMAC's digital "Certificate of Analysis" report for each batch was e-mailed to ACA Howe unmodified, and from these reports an assay database was set up and a series of Excel spreadsheets were compiled to tabulate and plot internal OMAC repeat, standard and blanks assay data.

11.1.1.1. . OMAC STANDARD SAMPLES (2011-2012)

A total of 106 analyses of Certified Reference Material (CRM) were reported by OMAC during the course of the 2011-2012 drilling campaign. The results, shown in Table 8 below, indicate high accuracy and a very close correlation with the analytical values recommended by Geostats Pty, apart from one sample, G307-5.

The OMAC assays are on average 1.05% lower than the recommended values, suggesting a very slight bias toward under-reporting of gold values by OMAC during this period. The correlation coefficient between recommended values and OMAC assays was 0.99.

Assays of standard samples showed very good correlation with recommended values, with only two samples returning assays outside +/- 10% of the CRM recommended value.

TABLE 8. OMAC LABORATORIES ANALYSIS OF CERTIFIED REFERENCE MATERIAL

DRILLING AND TRENCHING PROGRAMME, 2011-2012

Geostats Pty CRM number	umber of assays	mean Au, g/t	standard deviation, Au g/t	CRM recommended value	% difference	remarks
Standard G904-6	10	0.35	0.01	0.36	-2.78	
Standard G305-3	6	0.70	0.01	0.72	-2.78	
Standard G910-10	1	0.98		0.97	1.03	
Standard G310-7	1	1.01		1.01	0.00	
Standard G308-6	4	1.25	0.03	1.28	-2.34	
Standard G910-1	9	1.39	0.07	1.42	-2.11	
Standard G907-7	6	1.51	0.01	1.54	-1.95	
Standard G302-7	6	2.08	0.05	2.13	-2.35	
Standard G308-3	4	2.45	0.03	2.50	-2.00	
Standard G999-4	6	3.02	0.04	3.02	0.00	
Standard G910-3	5	3.94	0.03	4.02	-1.99	
Standard G307-5	7	4.86	0.06	4.87	-0.21	Rogue value 8.08 excluded
Standard G910-5	6	5.29	0.06	5.23	1.15	
Standard G904-8	5	5.54	0.10	5.53	0.18	
Standard G907-8	3	6.79	0.02	6.78	0.15	
Standard G307-7	4	7.85	0.08	7.87	-0.25	
Standard G908-8	3	9.68	0.28	9.65	0.31	
Standard G904-1	5	12.49	0.29	12.66	-1.34	
Standard G910-4	7	16.88	0.21	16.94	-0.35	
Standard G306-4	3	21.76	0.28	21.57	0.88	
Standard G908-9	2	34.56	0.00	34.18	1.11	
Standard G310-10	3	48.88	0.63	48.64	0.49	
total	106		0.09		-1.05	Weighted for frequency

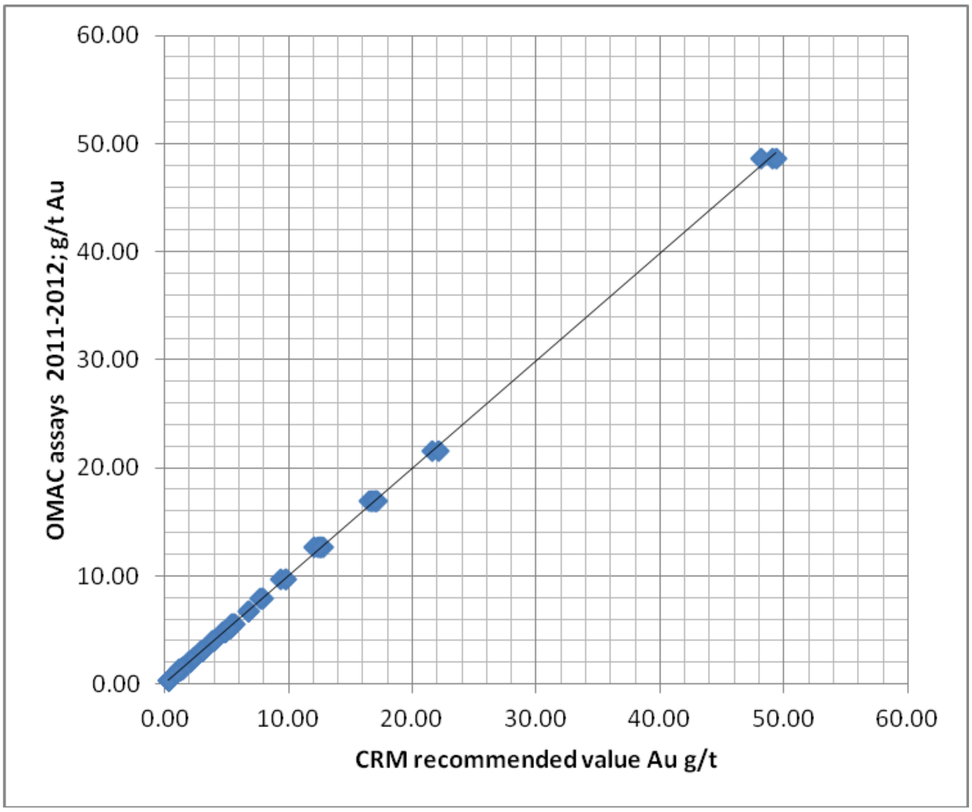


FIGURE 13.OMAC ASSAYS OF CRM V RECOMMENDED VALUE

11.1.1.2. OMAC BLANK DATA

OMAC inserted a total of 105 blank samples as part of its internal QA/QC procedure. All 105 assays reported gold values of less than 0.01 ppm. These results indicate that there is no significant problem of sample cross-contamination in the sample preparation and assay procedure.

11.1.1.3. REPEAT DATA

A total of 196 repeat assays were undertaken by OMAC as part of their internal QA/QC checks. Data from the repeat assays shows good correlation with the original assay, in all grade ranges. The scatter plots in Figure 14 below clearly indicate an acceptable level of correlation with no indication of bias. The correlation coefficient between originals and repeats was 0.996.

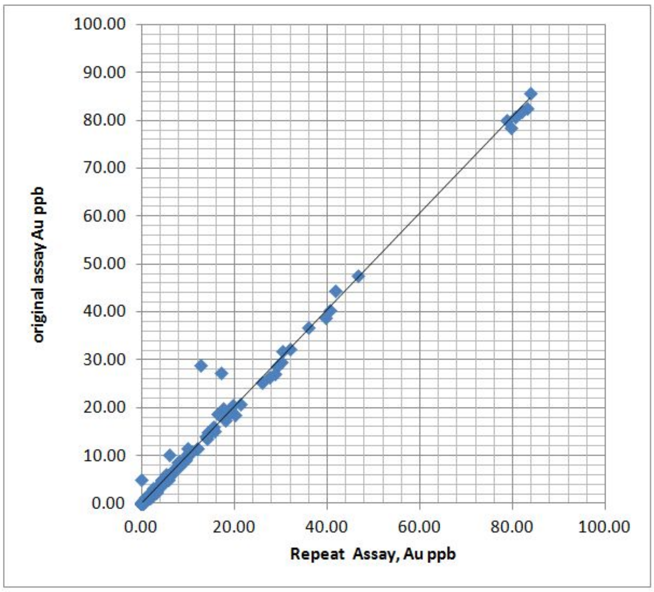


FIGURE 14. SCATTER PLOT – OMAC INTERNAL REPEAT ASSAYS

11.2. GALANTAS QA/QC

11.2.1. GALANTAS STANDARD SAMPLES, 2011-2012

In 2011, Galantas submitted four large bags of mineralised material from the Kearney pit to OMAC for the creation of standard reference material. 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 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-fifth sample during drilling and trenching.

In general, submitted standard samples showed only reasonable repeatability in all three grade ranges. The error plot for all samples shows that 76% of returned assays were within an acceptable +/- 10% of the expected value. Thirty two outliers are present (in a total of 139 samples) but do not demonstrate a systematic sample bias. This level of repeatability is in contrast to the very good correlation of OMAC assays with recommended CRM values discussed in Section 12.1.1. The reason for this discrepancy is not known, but it is possible that it is caused by inhomegeneity in the Galantas standards caused either by insufficient pulverisation or by gravitational segregation during transport.

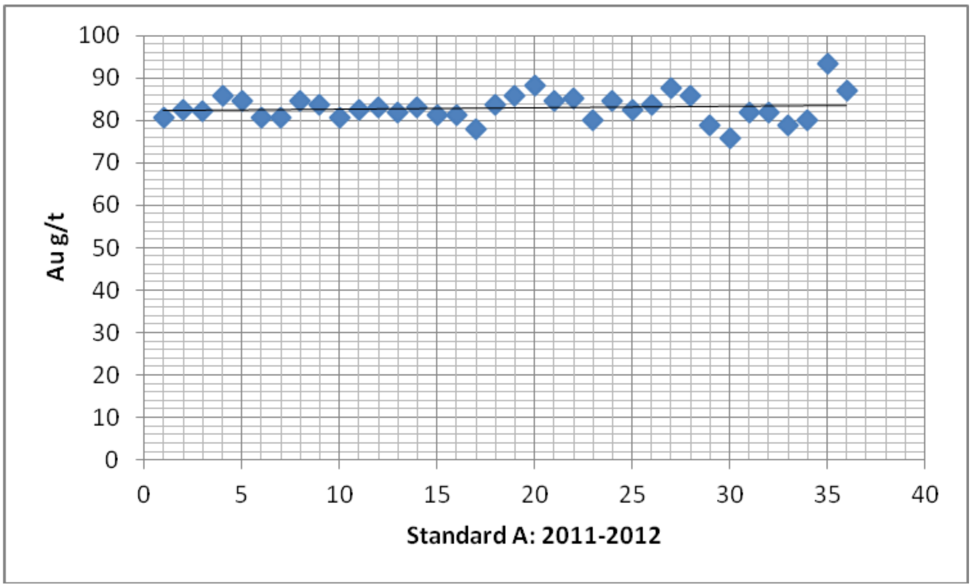


FIGURE 15.ASSAYS OF GALANTAS STANDARD A, 2011-2012

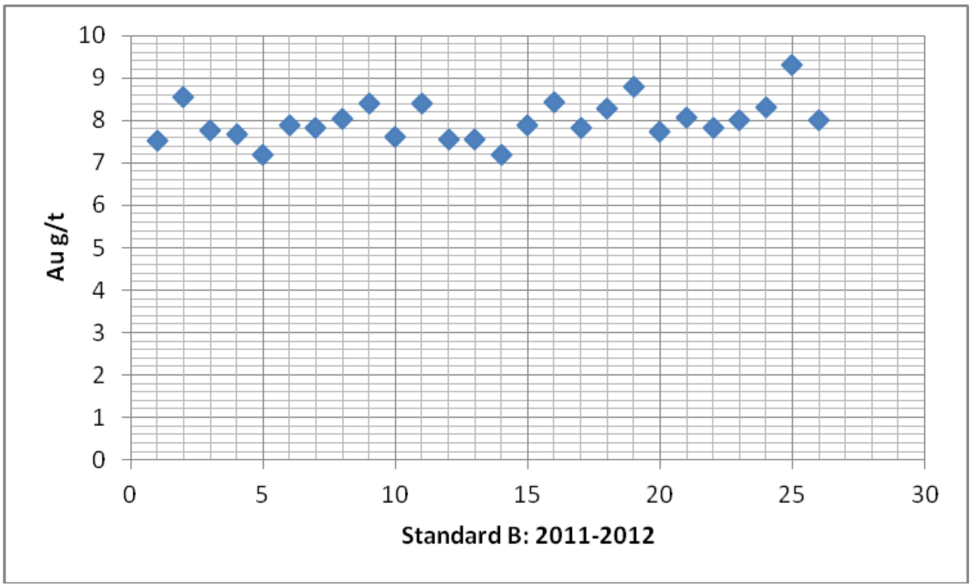


FIGURE 16.ASSAYS OF GALANTAS STANDARD B, 2011-2012

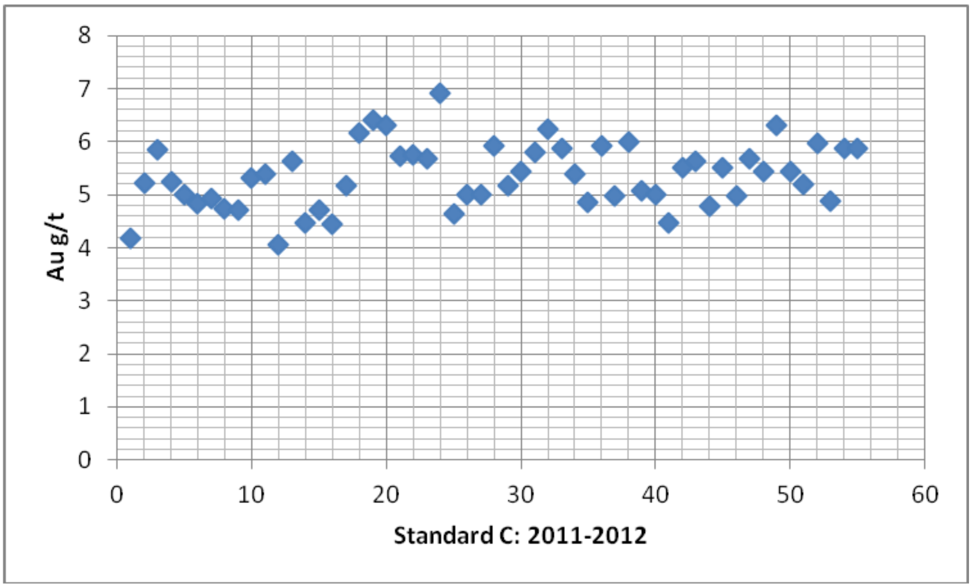


FIGURE 17.ASSAYS OF GALANTAS STANDARD C 2011-2012

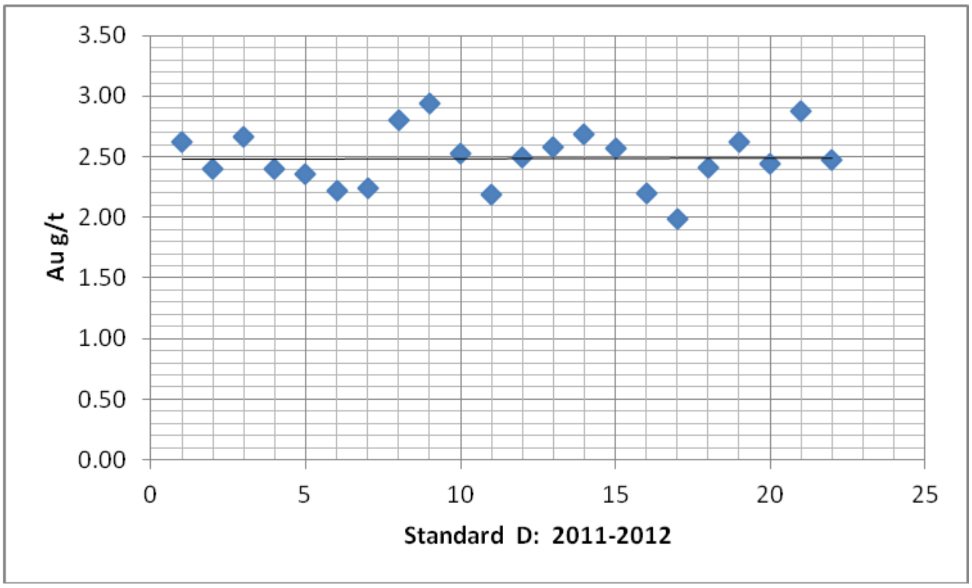


FIGURE 18.ASSAYS OF GALANTAS STANDARD C 2011-2012

11.2.2. QUARTER CORE SAMPLES

A total of ten quarter core samples were selected as field duplicates and cut, under ACA Howe's supervision, from a selection of core drilled between 2006 and 2012, representing a range of gold grades. The quarter core duplicate samples were prepared and assayed at OMAC according to the usual procedures.

Assay results indicated rather poor correlation, however assay pairs do show similar values to within an order of magnitude, as shown in Figure 19 below. Poor levels of precision highlight the grade variability that can exist in core samples due to the irregular distribution of mineralisation which tends to form clots and aggregates that are not always divided equally between the two core halves. This problem of representativity is exacerbated during re-sampling by the small sample size of the quarter core especially in samples of short length (some of the quarter core samples were 20 cm or less). 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 recent introduction of core orientation equipment may reduce any bias in core sampling by allowing a long axis line to be marked on the drill core to provide a reference for more consistent sampling of half core.

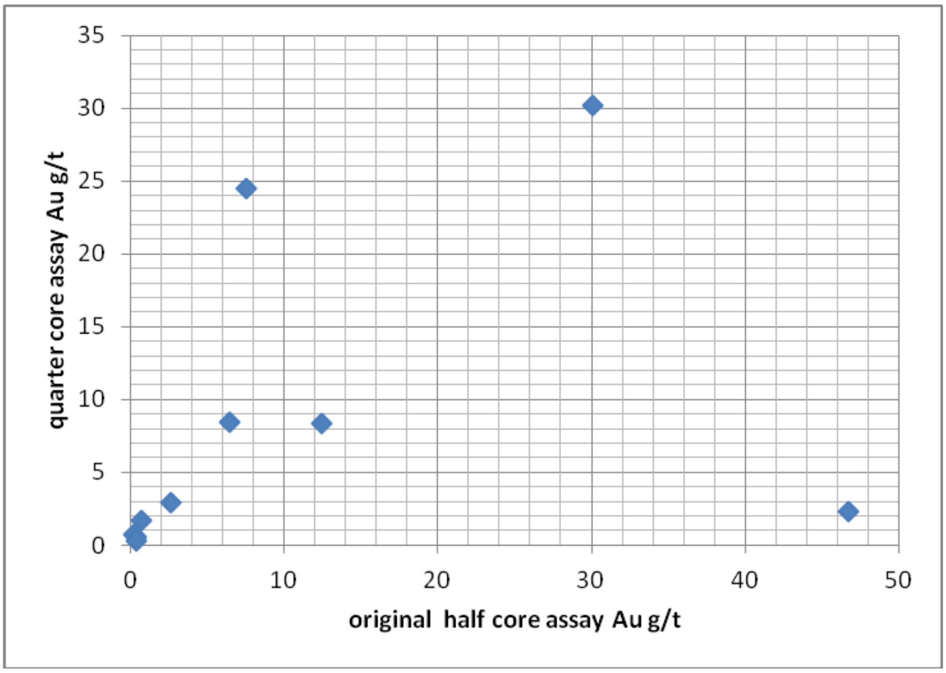


FIGURE 19.SCATTER PLOT OF CORE DUPLICATES

11.2.3. BLANK SAMPLES

A total of 160 internal blank samples were included in batches of samples submitted to OMAC by Galantas during the 2007-2008 and 2011-2012 drilling programmes. Every assay returned a value of less than the detection limit (<0.01ppm Au). This suggests that there are no contamination issues and that satisfactory quartz flushing practices are being implemented at OMAC.

11.3. QA/QC CONCLUSIONS

With the assistance of ACA Howe, Galantas has maintained a QA/QC programme that was designed to assess and monitor the quality of sample assay data by monitoring laboratory sample preparation, accuracy, and precision via the application of external controls (blanks, standards, duplicates and pulp re-assays), and by reviewing internal laboratory controls (internal and standard reference samples, repeat assays and blanks).

Results of this work indicate that the analytical techniques employed by OMAC are reliable in producing assay data with a high level of accuracy and precision and that sample preparation practices employed at OMAC are satisfactory so as to minimise the risk of contamination. Assay results from drilling and sampling programs implemented during 2006-2007 and 2011-2012 may therefore be regarded as representative of the samples collected.

However results from the submission of field duplicates (1/4 core samples) indicate that some sampling bias may have been introduced that may have affected the representativeness of the core samples collected for duplicate analysis.

The recent introduction of core orientation equipment may reduce any bias in core sampling by allowing a long axis line to be marked on the drill core to provide a reference for more consistent sampling of half core.

12. DATA VERIFICATION

ACA Howe has compiled the information collected during recent exploration activities using a combination of:

- first hand verification, data collection and observations during two site visits to the project, undertaken by the author between August 9th and 13th, 2011 between March 14th and 16th, 2012, and between May 16th and 18th, 2012 in order to collect data and review drilling activities,

- information and data received from Galantas and third party sources, listed below in the section "References and Sources", which we have assumed to be correct but which we have not independently verified although we are not aware of any information in those documents that is incorrect.

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.

During recent exploration, geological data, survey data, assay data, and bulk density data from drilling and channel sampling activities were forwarded to ACA Howe at regular intervals and 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 by ACA Howe, usually in the form of Excel spreadsheets, allowed direct import in to Micromine, thus minimising any potential error arising from manual data entry.

The data supplied to ACA Howe by Galantas and third parties appear reliable in the light of checks carried out by ACA Howe and the review of QA/QC practices. ACA Howe has not independently verified drilling assay data via independent check sampling but did supervise sampling of quarter core discussed in Section 10.

In view of these checks, ACA Howe is of the opinion that the data cited in this report are reliable and adequate for use in the resource estimate.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Initial froth flotation testwork was carried out by Lakefield (1992) and the results of that testwork incorporated into the existing plant.

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. ACA Howe considers that additional metallurgical testwork is not required since there is no indication that the mineralogy of the veins changes at depth.

14. MINERAL RESOURCE ESTIMATES

14.1. RESOURCE ESTIMATION OVERVIEW

ACA Howe has prepared an updated estimate of mineral resources for the main Kearney Vein Zone and Joshua's Vein, which have been the main focus of historical and recent exploration, and for several other veins in the project area that have been drill tested by historical and recent drilling. These comprise Elkin's Vein, Kerr Vein, Gormley Vein, Garry's Vein, Princes Vein and Sammy's Vein (referred to in this section as "the named veins").

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 the validity of this data for use in resource estimation. Following the generation of mineralised domains, raw sample data were composited in order to standardise sample support and further statistical and geostatistical analysis was 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 CIM compliant grade and tonnage estimates for the project.

14.2. SOFTWARE

Updated resources for the Omagh project were estimated using Micromine version 12.5.4 software.

14.3. DATABASE COMPILATION

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 (see Section 14.6, Interpretation and Modelling, below).

During the recent drilling and channel sampling campaigns of 2011 and 2012, all collar, assay, survey and geological data were appended to the master database and validated on an ongoing basis. Following data cut-off on June 1st, 2012, the master database was again validated, with validation functions run in Micromine to check for errors, so that the resulting database contains all available historical drilling and sampling data and all current exploration drilling and channel sampling data for the project, and is robust and suitable for use in resource estimation.

Table 9, below, summarises the data contained with the Omagh_07_12 database, used in resource estimation.

TABLE 9. DATA USED IN RESOURCE ESTIMATION

Data	Number of Records
Drill holes	255
Drill Hole Surveys	1054
Drill Hole Assays (gold)	3,582
Drill Hole Assays (multi –element ICPORE)	2,285
Drill Hole Geology	3,110
Channels	498
Channel Surveys	695
Channel Assays (gold)	6,520
Channel Assays (multi –element ICPORE)	3027
Channel Geology	242

14.4. DATABASE VALIDATION

Once the drill hole and channel database was updated with data from the recent drilling and sampling activities, a series of validation functions were run, designed to reveal the following errors:

- Duplicate drill holes or channels;
- One or more collar coordinates missing in the collar file;
- FROM or TO missing in the assay file;
- FROM $\geq$ TO in the assay file;
- Sample intervals non-contiguous;
- Overlapping sample intervals;
- First sample $\neq$ 0m in the assay file;
- First survey $\neq$ 0m in the survey file;
- Multiple surveys for the same depth;
- Azimuth not between 0 and 360 degrees in collar or survey file;
- Angle not between 0 and 90 degrees in collar or survey file;
- Azimuth or angle missing in survey file;
- Depth of hole less than depth of final sample;
- Down hole survey depth greater than drill hole depth.

Any errors encountered were rectified by referring to the original source data.

14.5. COLLAR LOCATIONS

The collar locations for Riofinex drillholes were recorded in a local grid and subsequently transformed to the regional Ordnance Survey of Ireland (OSI) coordinate system (Irish Transverse Mercator Grid), which has been used for all work carried out by Galantas. Part of the transformation involved adjusting Riofinex collar co-ordinates by +18 m which resulted in intersections in Riofinex holes drilled beneath the Kearney Pit correlating properly with overlying trenching results. However during modelling based on 2011 and 2012 data it became apparent that the 18 m adjustment had not been applied to the remaining Riofinex holes, as was evident from displacements of the intersections in Riofinex holes relative to Galantas trench and drill intersections. The adjustment was therefore applied to the collar positions of the remaining Riofinex holes in the database in order to correct this discrepancy.

The easting adjustment of historical Riofinex data in the resource database can be considered reliable such that the position of historical data in OSI coordinates recorded in the 2012 database is accurate.

14.6. INTERPRETATION AND MODELLING

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.

During the mining of the Kearney pit a number of topographic surveys were carried out which record the original surface topography,(DTM_OGL), the pit topography at 10/04/2008 (DTM_PIT_100408) and the final pit topography (DTM_COASTWAY). The DTM_PIT_100408 survey is said to represent the final depth of the southern end of the pit, but this cannot now be verified as this part of the pit has now been filled with waste. Subsequent mining was confined to the northern end of the pit where the final pit topography is represented by the (DTM_COASTWAY) survey. A limited amount of mining has since been confined to the northern pit slope in the area of channel samples.

ACA Howe has used the DTM_PIT_100408, the DTM_COASTWAY surveys and the channel sample surveys to derive a combined pit shell representing the limit of mining at June 2012 which has been excluded from the resource estimate (see Figure 20).

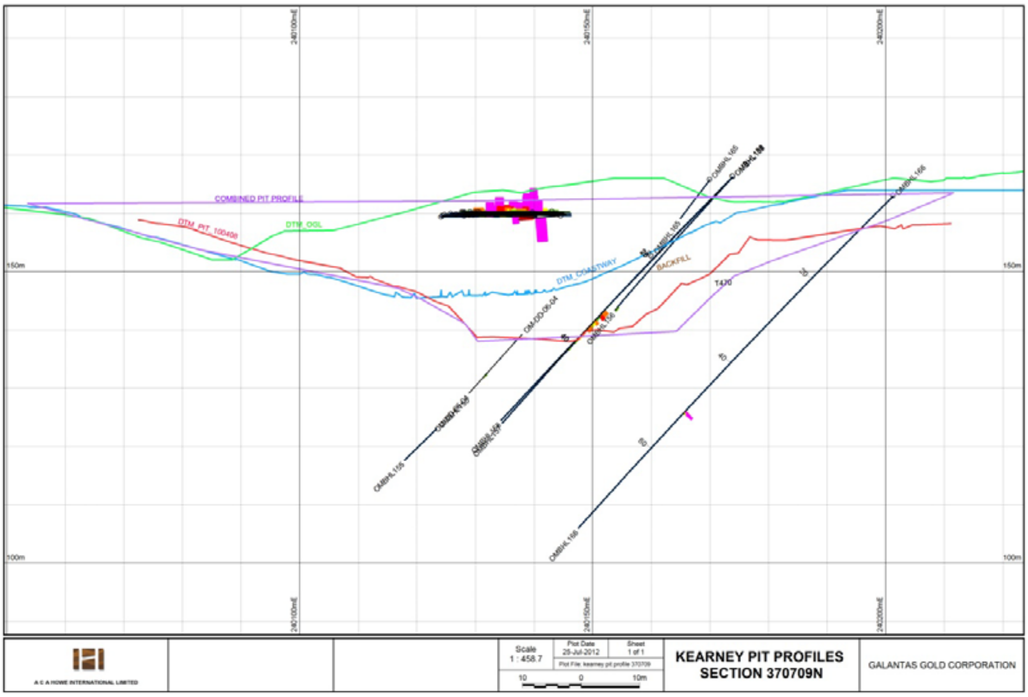


FIGURE 20. KEARNEY PIT PROFILES

Current topographic surveys representing the small exploration excavations at Joshua and Kerr veins were not available. However a partial survey of the excavation at Joshua was supplied by the mine surveyor and has been used to establish the lower limit of sampling. At Kerr, surveys of the trench locations were used to establish the lower limit of mining at that time.

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.

14.6.1. INTERPRETATION OF MINERALISED ZONES

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, and selective sampling of veins and clay gouge essentially defines the mineralised zones in each hole and assisted with the sectional interpretation of veins.

Strings were created, constraining zones of mineralisation exhibiting greater than 2.5 g/t Au over a drilled length of greater than 0.9 m. The criteria used to define mineralised zones were generally adopted. However in areas where gold assays greater than 2.5 g/t Au were returned from drill hole intervals less than 0.90 m, these were included and bulked out to a true width of 0.90 m by the inclusion of waste material, on condition that the weighted average gold grade of the drill hole interval was greater than 2.5 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 20 m apart were displayed interactively, with a clipping window of 10 m (distance constraint to the north or south of the plane of the cross section). At all other veins, where fewer drill holes are present on a semi-regular grid spacing of 50 m or 100 m, cross sections along hole fences were visualised, usually at spacings equal to the exploration grid;

- 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 50 m. At depth, the interpretation was extended by a distance of half the drill hole spacing in the z plane or 50 m. 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 and extended to include channel sampling data, constrained to the surface 3ppm 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.

Typical sections of the Kearney Vein Zone and other named veins are contained in Appendix II.

14.7. 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 other named veins. The vein zone at Kearney is shown to be continuous over the drilled strike length of over 800 m, however individual veins within this zone, which can be mapped from one section to another, are not 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 by 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 (see Figure 6).

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 46 domains were modelled by wireframe, including 22 on Kearney Vein, 6 on Joshua Vein and 6 on Elkins Vein.

Details of each interpreted domain are contained in the table below. Wireframe solids for all domains are shown in Figures 21 and 22.

TABLE 10.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 1A	Main vein, North section, beneath trenches	26,610	174	131	340	80
Joshua 2A	Main vein, South extension	10,614	151	68	345	83
Joshua 3A	North hanging wall vein	20,160	200	118	350	87
Joshua 4A	North footwall vein	23,679	275	132	345	82
Joshua 5A	Deep central vein or lens	25,663	100	121	155	67
Joshua 6A	Main vein, south section, beneath trenches	17,382	143	102	155	67
Kearney South 2012	Main vein below trenches at S end of pit	2,291	28	40	15	81
Kearney 1 2012	Main vein below trenches at S end of pit	17,914	206	109	15	81
Kearney 2 2012	HW vein below trenches at S end of pit	8,880	104	86	0	75
Kearney 5A	FW vein below trenches central pit	16,252	170	187	355	83
Kearney 3A	Main vein below trenches central pit	35,745	247	185	355	80
Kearney 7A	HW vein below trenches central pit	14,797	73	141	0	70
Kearney 25A	HW lens below trenches central pit	281	14	25	0	78
Kearney 8A	HW lens at N end of pit	5,594	22	96	0	84
Kearney 4A	Main vein at N end of pit	82,706	264	267	355	80
Kearney 6A	Sub-parallel FW vein at N end of pit	72,231	257	104	355	85
Kearney 12A	Sub-parallel FW vein at N end of pit	50,899	136	200	0	85
Kearney 20A	Sub-parallel FW vein at N end of pit	8,947	77	98	355	86
Kearney 24A	Sub-parallel FW vein at N end of pit	9,387	38	67	0	78
Kearney 19A	Sub-parallel HW vein at N end of pit, beneath channels	587	84	11	350	80
Kearney 9A	Sub-parallel HW vein at N end of pit	66,175	238	256	0	85
Kearney 13A	Sub-parallel HW vein at N end of pit	2,469	55	47	0	76
Kearney 14A	Isolated sub-parallel HW vein at N end of pit	2,135	29	86	0	71
Kearney 15A	narrow vein 50-150m N of pit	15,409	90	123	0	70
Kearney 16A	narrow vein 50-150m N of pit, hw	5,069	70	81	0	75
Kearney 17A	narrow vein 50-150m N of pit, hw	5,263	75	84	0	85
Kearney 21A	narrow vein 50-150m N of pit, hw	6,292	75	84	0	85
Kearney N1A	Main vein extension, 500-750m N of pit	14,477	245	77	353	76
Kerr 1A	Main Kerr Vein	8,120	128	68	155	70
Kerr 2A	Sub-parallel vein 5-20m southwest	6,661	125	60	150	72
Kerr 3A	Sub-parallel footwall vein exposed in pit	707	70	13	163	81
Kerr 4A	Sub-parallel footwall vein exposed in pit	802	55	9	160	85
Kerr 5A	Sub-parallel footwall vein exposed in pit	319	33	26	160	85
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

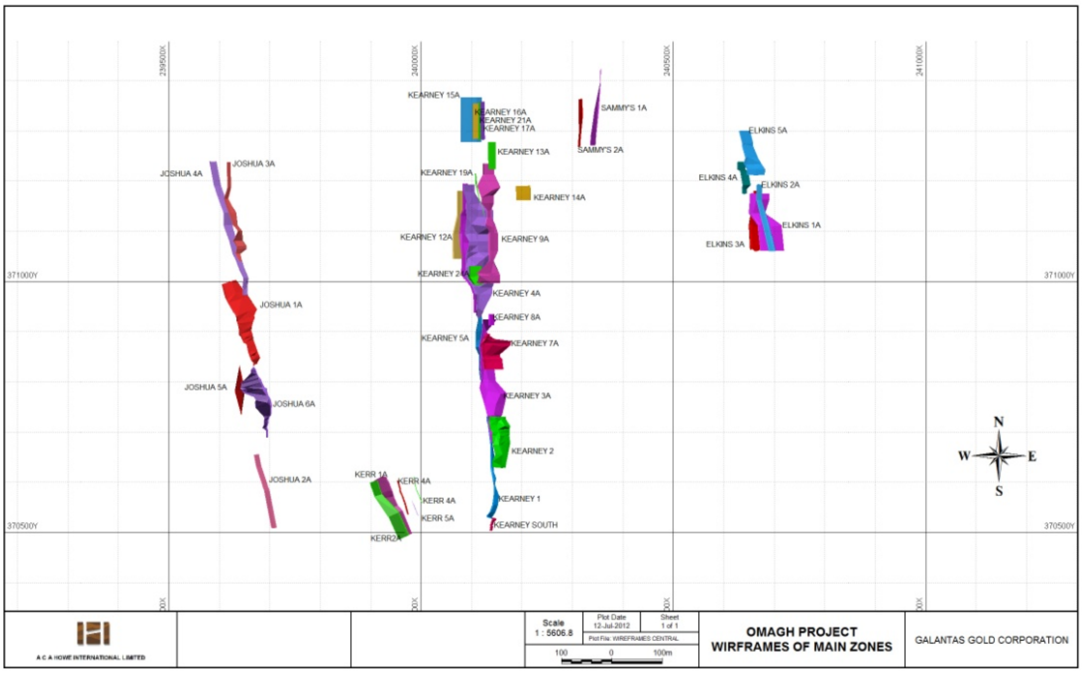


FIGURE 21.WIREFRAMES OF MAIN ZONES

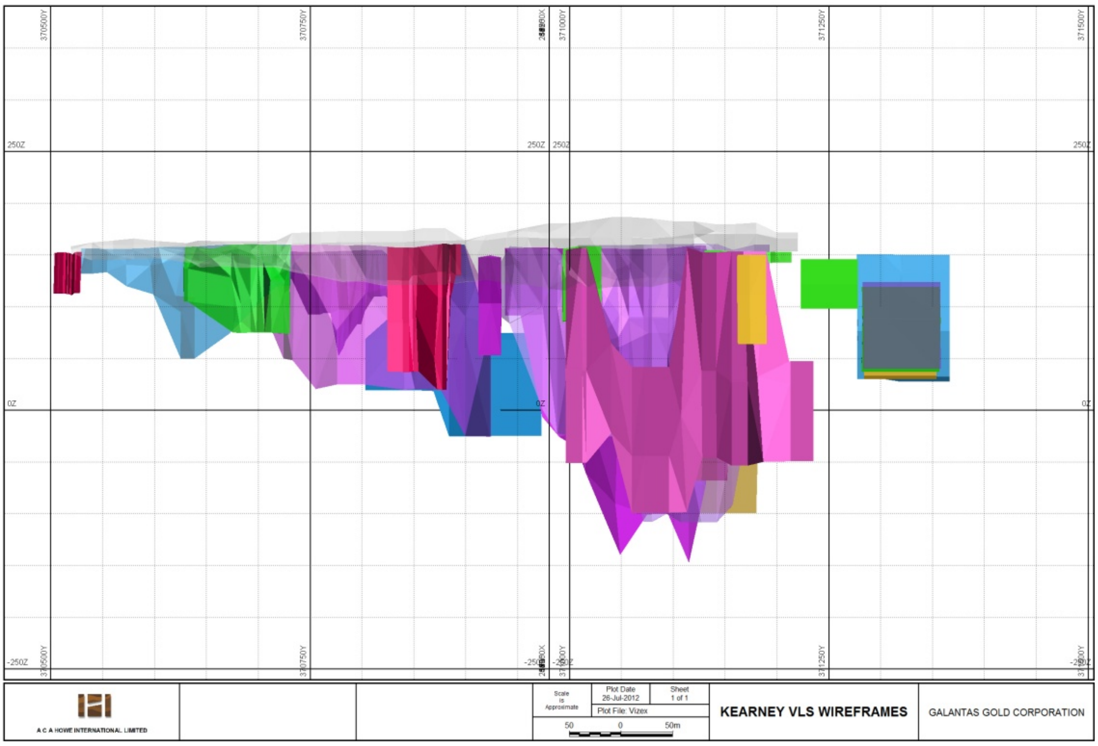


FIGURE 22. KEARNEY VERTICAL LONGITUDINAL SECTION WIREFRAMES

14.7.1. SAMPLE DATA SELECTION, TOP-CUTTING AND COMPOSITING

14.7.1.1. SAMPLE DATA SELECTION

Prior to selecting samples for use in resource estimation exploratory data analysis was undertaken on the raw sample database to assess the statistical characteristics of each sample type (historical Riofinex channel sampling, historical Riofinex drilling, recent Galantas/ACA Howe channel sampling and recent Galantas drilling) and to investigate the grade distribution of each.

Statistical data for each type of sampling is contained in Table 11 below:

TABLE 11.SAMPLE STATISTICS FOR DIFFERENT DATA TYPES

	ALL SAMPLES	HISTORICAL RIOFINEX SAMPLES		GALANTAS SAMPLES, 2006-2012				
		ddh	Kearney channels	Kearney ddh	Joshua ddh	other zones ddh	Kearney channels	Joshua channels
Minimum	0	0	0	0.01	0.01	0.01	0.01	0.01
Maximum	626	626	123.82	165.12	59	101.76	75.25	125.44
No of points	10564	1334	5105	1342	365	218	373	2179
Sum	37581	3613	15663	3537	1235	330	1636	11082
Mean	3.56	2.71	3.07	2.64	3.39	1.51	4.39	5.09
Variance	121.00	372.98	70.18	77.34	57.10	55.45	81.02	115.24
Std dev	11.00	19.31	8.38	8.79	7.55	7.45	9.00	10.74

The statistics from historical Riofinex drilling (mean global gold grade of 2.71ppm) compare closely with Galantas drilling (mean global gold grade of 2.64ppm) and it is documented (ACA Howe 2003 and 2005 reports) that low core recovery during historical Riofinex drilling was likely to cause underestimation of global grade. Statistical comparison of historical Riofinex channel sampling with Galantas/ACA Howe channel sampling shows a significantly higher global mean gold grade for recent Galantas/ACA Howe Kearney zone channel samples (4.39ppm) compared with historical Riofinex channel samples (3.07ppm). The difference is attributed to the fact that historical sampling, completed at 1m intervals, was not constrained to vein and/or alteration material, and was therefore diluted by the inclusion of waste wall rock. Galantas channel samples selectively sampled vein and alteration material and included minimal waste wall rock.

It is apparent that channel sampling of the Kearney Pit floor and Joshua Vein returned higher average gold grades than corresponding drill core samples from depth. Possible reasons for this apparent discrepancy include surface enrichment, differing sample intervals and inclusion of wallrock, core loss, and oreshoot shape.

Surface enrichment is considered as unlikely since fresh sulphides occur at surface and oxidation is limited to very minor limonite development.

Drill core sampling generally extended one metre into wallrock, whereas this was not the case for channel sampling, which would have resulted in relative dilution of the average grade of the drill core samples.

Comparison of twin hole assay data at Kearney and Joshua indicates generally higher gold grades in Galantas intersections compared to nearby Riofinex intersections, as discussed in Section 10. This is attributed to higher core loss in the Riofinex holes as compared to the Galantas holes in which core loss was controlled by the use of triple tube core barrels. Gold grades in clay gouge are reportedly frequently higher than those in adjacent quartz vein which may account for grade depletion if the clay gouge is preferentially lost from the sample.

Surface channel samples, at Kearney and Joshua are clustered relative to sub-surface drill core samples (the average spacing along strike is 1 m for channel samples compared to 20-40 m 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 treatment of clustered data for resource estimation is discussed in Section 16.7 below.

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.

Accordingly, the raw gold assay database was flagged so that each assay value was assigned to the domain it informed and 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. The raw mean gold grades for samples that fall within each of the domains are contained in Table 13. Domain histograms for the most densely populated domains are contained in Appendix III.

14.7.1.2. 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 overestimate 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 39 domains.

In view of the statistical similarities between 2008 and 2012 data, it was decided that retaining a top cut of 75 ppm Au for use in the 2012 estimate was an appropriate measure. Accordingly, all assay values exceeding 75 g/t Au were cut to 75 g/t Au. This involved a total of 23 assays, of which 10 were from Kearney_1_2012, 5 were from Kearney_South_2012, and 3 were from Princes_1A. Full details of top cut samples are contained in Appendix IV.

14.7.1.3. COMPOSITING

Data compositing was undertaken on raw sample 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 0.10 m and 3.0 m, averaging 0.5 m. A composite length of 0.3 m was chosen as compatible with a minimum mining width of 0.9 m (3 composites). This compositing strategy is the same as that used for the 2008 resource estimate.

Raw drill hole samples within each mineralised zone were flagged by domain in the sample database and composited to 0.3 m 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 0.1 m, 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 0.10 m. 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.

14.7.2. GEOSTATISTICS

A geostatistical study carried out in 2008 remains valid for the 2012 data set since the horizontal range determined from that study is dependent overwhelmingly on the close-spaced sampling carried out on the Kearney trenches. Accordingly, additional geostatistical study was not considered necessary for the revised 2012 resource estimate. An account of the 2008 geostatistical study is reproduced 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 25 m and 40 m, 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^o, 355^o and 160^o 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 40 m and 80 m, 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 60 m, 60 m and 20 m respectively.

Generated semi-variograms from the 2008 study are contained in Appendix V.

14.8. BLOCK MODELLING

Block modelling was undertaken in several stages. Firstly empty block models were generated covering the extents of all domain wireframes within each deposit. The block models were then constrained to the wireframe domains and each three-dimensional wireframe solid populated with blocks to create an empty resource block model for each deposit. Assigned blocks were then labelled with the code of the domain into which they were assigned.

The initial filling was undertaken by blocks with the dimensions 15 m by 5 m by 10 m followed by sub-blocking down to 1.5 m by 0.5 m by 1 m 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 40 m by 40 m, and in some areas 20 m by 20 m. Therefore the y (strike) block dimension is 15 m. The x block dimension of 5 m honours the narrow thickness of the vein zones and the z dimension of 10 m 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.

Block model characteristics for each model are contained in Table 12 below.

TABLE 12.BLANK BLOCK MODEL DETAILS

Model	Dimension	Extent (Irish Transverse Mercator Grid		Parent Block Size (metres)	Minimum sub-blocks (metres)	Number of Parent Blocks
		Min	Max			
Kearney South	Eastings	240040	240220	5	0.5	37
	Northings	370500	370995	15	1.5	34
	RL	-60	180	10	1	25
Kearney North	Eastings	240040	240220	5	0.5	37
	Northings	370900	371395	15	1.5	34
	RL	-220	180	10	1	41
Joshua	Eastings	239500	239800	5	0.5	61
	Northings	370500	371205	15	1.5	48
	RL	40	200	10	1	17
Elkin’s	Eastings	240540	240740	5	0.5	41
	Northings	371040	371320	15	1.5	20
	RL	20	130	10	1	12
Kerr	Eastings	239870	240060	5	0.5	39
	Northings	370460	370660	15	1.5	14
	RL	60	170	10	1	12
Minor Veins South	Eastings	239250	239860	5	0.5	103
	Northings	369400	370100	5	0.5	141
	RL	50	250	10	1	21
Minor Veins North	Eastings	239800	240500	5	0.5	141
	Northings	371200	372100	15	1.5	61
	RL	0	160	10	1	17

14.9. GRADE INTERPOLATION

Gold grades were interpolated into the empty block model for each deposit using the Inverse Distance Weighting (IDW) interpolation, raised to the second and third powers. 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. Variographic analysis in 2008 was not considered 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 at 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 a relatively small and constrained portion 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.

At the other named veins, IDW^2 interpolation method was used, given that only drilling data is used, on a relatively widely spaced exploration grid, so that samples farther away are given a larger weighting than if the third power was employed.

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 and consideration of appropriate ranges of continuity, applicable to this type of deposit. The ranges were chosen to be 60 m, 60 m and 20 m in the three main directions. For all domain interpolations, the first search radii were selected to be one-third of the range in all directions. The second, larger search radii were selected to be two-thirds of 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, whereby 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 13 below.

Once resource estimation was complete, all block models were constrained by the topographic surface and combined pit shell and all blocks inside these volumes were deleted prior to reporting.

TABLE 13.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	Greater than the 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

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 as listed in Table 10. 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.

Figures 23 to 25 below are vertical longitudinal sections of the Kearney and Joshua veins with block models showing gold grade distribution.

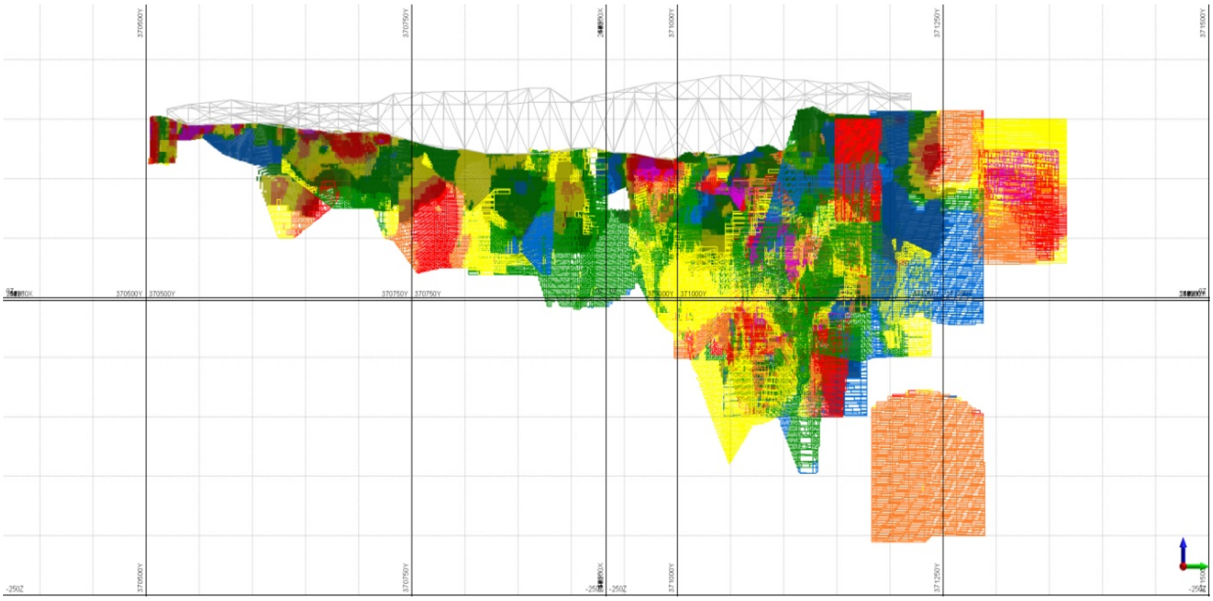


FIGURE 23. KEARNEY VEIN LONGITUDINAL SECTION BLOCK GRADES

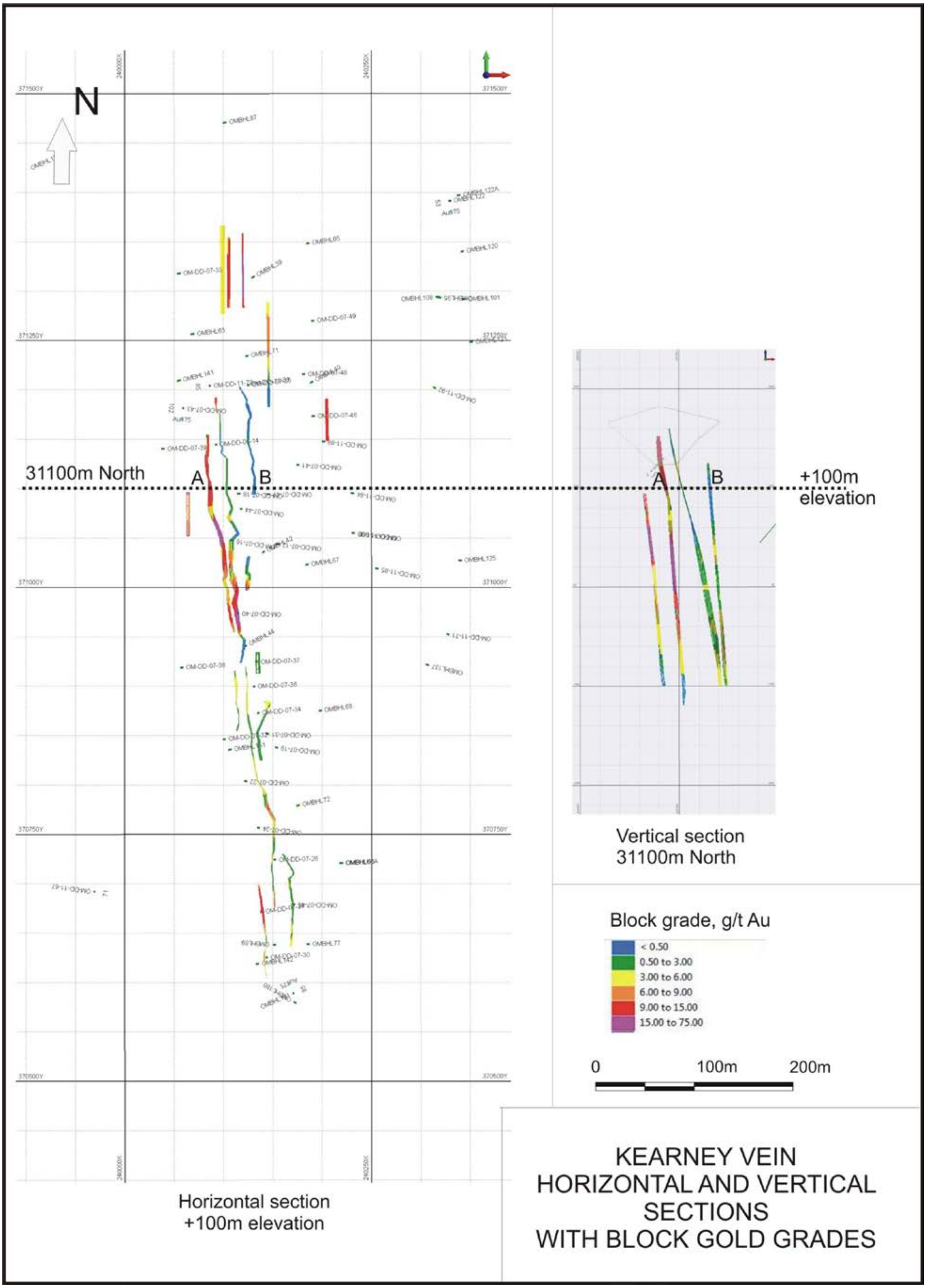


FIGURE 24. PLAN AND SECTION OF KEARNEY VEIN

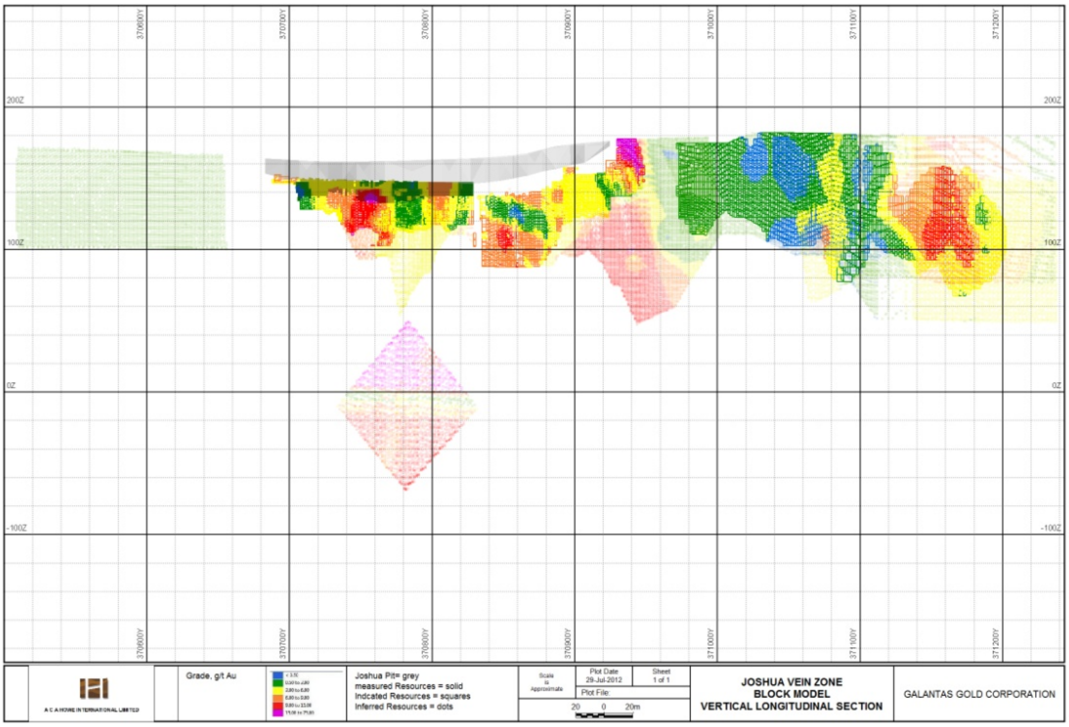


FIGURE 25. JOSHUA VEIN LONGITUDINAL SECTION SHOWING BLOCK GRADES

14.10. RESOURCE CLASSIFICATION

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 11th, 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.**

- **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.**

- **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.**

Classification, or assigning a level of confidence to Mineral Resources has been undertaken in strict adherence to the CIM Definition Standards on Mineral Resources and Mineral Reserves referred to above, 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, whereas 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 one-third 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 m 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 20 m by 20 m to 60 m by 60 m).

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 two-thirds 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.

14.11.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 14 below:

TABLE 14.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

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, ACA Howe considers it reasonable to apply this density value to other named veins.

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.

ACA Howe personnel have not independently verified density data used to calculated resource tonnages, but have reviewed the data and accompanying internal report (contained in Appendix VI) and assume that the data are correct. ACA Howe considers the applied density values to be reliable, having been determined from a total of 125 samples reflecting the density contrasts of different ore types, and honouring observed geological characteristics from core logging.

However, ACA Howe recommends that as part of future drilling campaigns, density measurements be taken of core from all named veins so that the density value applied to individual veins can be further refined.

14.12. RESOURCE TABLE

The 2012 updated resource estimate for the Kearney deposit and other named veins is summarised in the following table, with resources classified in strict 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 11th, 2005.

The resources listed here have been derived directly from the relevant block models and are subject to a 2.5 g/t Au cut-off grade.

TABLE 15.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

To the best knowledge of ACA Howe, 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.

The mineral estimate, prepared by Richard Parker, ACA Howe Senior Associate Geologist, is compliant with current standards and definitions required under NI 43-101 and is reportable as a mineral resource by Galantas Gold Corporation. However, the reader should understand that mineral resources are not mineral reserves and do not have demonstrated economic viability.

14.13. MODEL VALIDATIONS

Upon completion of resource estimations at Kearney and other main veins, model validations were run and a series of checks performed to validate each block model. Screenshots of each block model, coloured by gold grade are contained in Appendix VII.

Detailed visual inspection of the block models was undertaken following completion of each domain interpolation to ensure that all blocks received an interpolated grade. In addition, the proper assignment of domain codes to blocks was verified. Once modelling was complete, a series of sectional slices through each block model was undertaken, with drill hole traces, composite grade data and block grade data displayed and compared to assess whether block grades honour the general sense of composite grades, that is to say that high grade blocks are located around high grade composite sample grades, and vice versa. A degree of grade smoothing is evident in all block models, which is expected but the blocks grades do honour composite grades in all models.

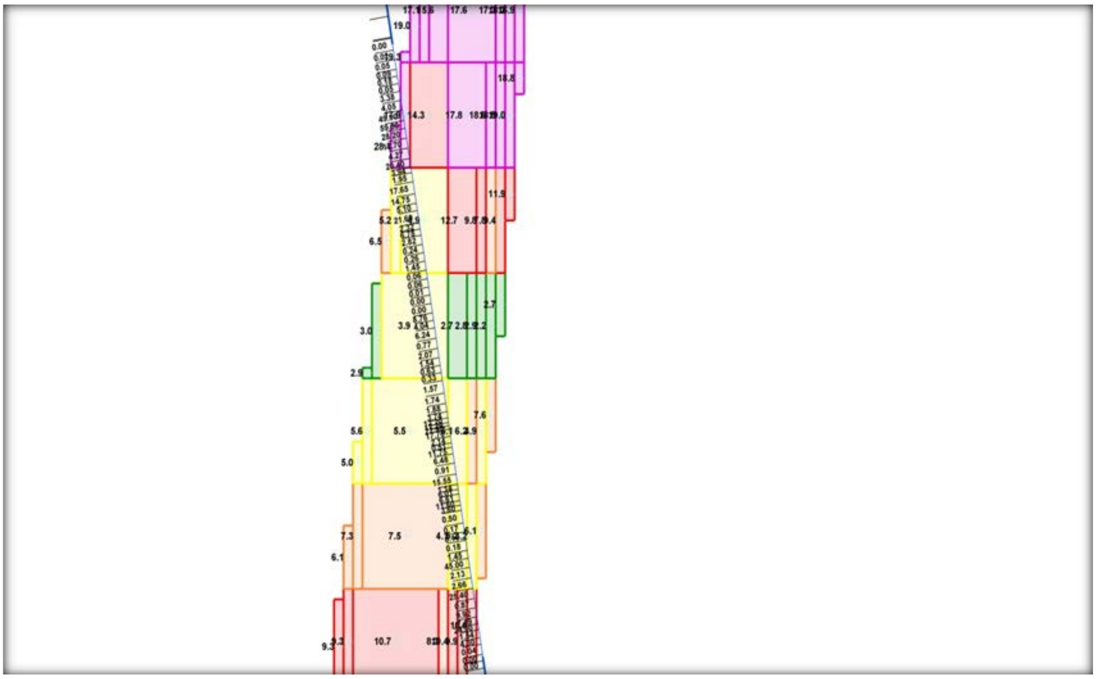


FIGURE 26.BLOCK GRADES V COMPOSITE GRADES, JOSHUA DOMAIN 5A

A useful measure of the reliability of resource estimates is the comparison of mean block grade and mean composite grade on a domain by domain basis. A good correlation between average block grade and average composite grade suggests that block grade data honours the composite data used in the interpolation process and therefore the estimated grades reliably reflect the input grades.

On the whole, mean block grades correlate well with input composite grades, within acceptable limits. A degree of grade smoothing is evident, which occurs in all estimations where grades are interpolated into blocks at unsampled locations, often at significant distances away from sampled points, regardless of estimation methodology.

A volume validation was also performed whereby the volume of each domain wireframe was compared to the volume of blocks within each domain, in order to assess the reliability of sub-blocking to honour the wireframe volume. Results of this validation showed that there was <1% difference in block model volume compared to wireframe volume, for all models.

14.14. COMPARISON WITH PREVIOUS RESOURCE ESTIMATES

Table 16 below lists resources according to year, zone, category and cut-off. In comparing the 2008 and 2012 estimates the following factors have generally caused decreases to the resource:

- The 2008 estimate is subject to depletion by mining and bulk sampling from Kearney, Kerr and Joshua pits, which was forecast in 2008 at 322,733 tonnes of ore;

- A cut-off grade of 2.5g/t Au was applied to the 2012 estimate. This has eliminated low grade resources in some veins (e.g. Garry's) and reduced the tonnage in other veins due to elimination of low grade blocks. No such cut-off was applied to the 2008 estimate. For comparison purposes Table 16 lists 2012 resources at 0 g/t cut-off, although these resources are not CIM compliant since they do not meet the test of having reasonable prospects of economic extraction;

- Further differences between the 2008 estimate and the 2012 estimate are caused by differences in the domain outlines caused by structural re-interpretations. In many cases such re-interpretation was necessary to accommodate adjustment of Riofinex collar locations discussed in Section 10, and entailed a simplification or straightening of domain outlines, causing a reduction in resource tonnage.

The reductions in resources have been largely offset by increases in resource tonnages and categories for Joshua's vein and some domains at Kearney's due to infill and resource augmentation drilling.

TABLE 16. COMPARSION OF 2008 RESOURCE ESTIMATE AND 2012 RESOURCE ESTIMATE

ZONE	CATEGORY	2008 ESTIMATE			2012 ESTIMATE					
		CUT-OFF 0 g/t Au			CUT-OFF 0 g/t Au			CUT-OFF 2.5 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	78,000	6.35	16,000	0	0	0	0	0	0
KEARNEY	INDICATED	350,000	6.74	76,000	421,000	5.55	75,000	270,900	7.94	69,000
KEARNEY	INFERRED	730,000	9.27	218,000	667,000	6.63	142,000	490,000	8.54	135,000
JOSHUA	MEASURED	0	0	0	16,500	5.52	2,900	13,000	6.48	2,800
JOSHUA	INDICATED	0	0	0	106,000	4.35	15,000	66,800	6.27	13,000
JOSHUA	INFERRED	160,000	3.96	20,400	237,000	6.57	50,000	173,000	8.48	47,000
ELKINS	INDICATED	113,000	3.30	12,000	135,000	2.80	12,000	68,500	4.24	9,000
ELKINS	INFERRED	29,000	3.82	3,600	26,000	4.67	3,900	20,000	5.84	3,800
KERR	MEASURED	0	0	0	6,500	3.38	700	2,250	6.75	500
KERR	INDICATED	0	0	0	10,000	3.61	1,100	5,400	5.03	900
KERR	INFERRED	60,000	4.03	7,800	33,000	4.09	4,300	26,000	4.58	4,000
GORMLEYS	INFERRED	115,000	6.57	24,300	105,000	6.70	22,600	75,000	8.78	21,000
GARRY'S	INFERRED	40,000	1.27	1,600	18,000	0.84	500	0	0.00	0
PRINCES	INFERRED	10,000	38.93	13,000	10,000	38.11	13,000	10,000	38.11	13,000
SAMMY'S	INFERRED	30,000	4.26	4,100	53,000	3.69	6,300	27,000	6.07	5,000
KEARNEY N.	INFERRED	55,000	1.97	3,500	43,000	2.47	3,400	18,000	3.47	2,000

15. MINERAL RESERVE ESTIMATES

No mineral reserves have been estimated.

The reader should understand that mineral resources reported in Section 14 of this report are not mineral reserves and do not have demonstrated economic viability

16. MINING METHODS

16.1. OPEN PIT MINING

16.1.1. KEARNEY PIT

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 its economic limits as dictated by stripping ratio, by the property boundary and the public road to the east, and by rock stockpiles to the west. The worked base of the pit at that time was at approximately 125 m above mean average sea level, or some 40 m below surface level.

Selective mining is employed using a narrow bucket excavator to remove vein material over minimum widths down to 30cm, and to depths of 1.5 m below the bench. The vein material is generally incompetent due to clay gouge content, brecciation and wallrock alteration which renders it amenable to mining by mechanical means without the need for explosives. Ripping of ore by a single tooth attachment, on a hydraulic excavator, is required and rarely a hydraulic hammer is utilised. Prior to an upgrade of some mill components in 2011, a grade control geologist defined components of pit production as high grade ore, low grade ore, low grade stock or clean country rock. The basis for discrimination was by visual examination, backed up by samples tested in the on-site laboratory. The testing process employed for the majority of samples was a sulphur testing process using an Eltra CS500, infra-red absorption unit. Periodic samples are also tested by fire-assay. Testing with use of the Eltra unit is much quicker than fire-assay and knowledge of the local gold / sulphur relationship is used to determine field estimates for gold. After the mill upgrade, the process plant cut-off grade was lowered from 3 g/t to around 1.0 g/t gold and therefore all mineralised material from the pits is now sent to the mill. Mill feed is also supplemented from low grade stock. The vein material in the open pit is easily distinguishable visually from the country rock which allows selection by an experienced excavator operator, usually without the need for close grade-control supervision. Ore is loaded into either 25 tonne or 40 tonne Volvo articulated dump trucks and transported to the mill.

Mining of the veins usually takes place in 1.5 m lifts, after which country rock is broken by either a Komatsu D65 Dozer, or an 80 tonne Hitachi EX800 tracked excavator, both with single tooth rippers, loaded into 40 tonne dump trucks and transported to rock stockpiles. Ripping of the country rock is usually carried out in a south to north direction, due to the preferential orientation of rock joints.

During the calendar year 2011, a total of 46,871 tonnes of ore (grading 4.46 g/t Au) was mined.

During the first quarter of 2012, due to limited production being available from open pits, which was blended with low grade stockpile material, overall grade to the process plant dropped to 3.54 g/t Au, whilst overall tonnage processed increased by 35 %.

TABLE 17.GALANTAS MINE PRODUCTION 2007- 2012						
Annual	2007	2008	2009	2010	2011	2012, to March 31st
Tonnes Milled				36,288	46,871	9,420
Average Grade g/t gold				4.74	4.46	3.54
Dry Tonnes Concentrate	996	1,708	1,969	1,500	2,265	268
Gold Grade Concentrate (g/t)	85.4	95.9	91.8	119	89	108
Gold Produced (oz)	2,734	5,264	5,817	5,696	6,479	933
Gold Produced (kg)	85	164	181	177	202	29
Silver Grade (g/t) (Concentrate)	212	221	234	338	234	261
Silver Produced (oz)	6,806	12,110	14,825	16,267	17,082	2,247
Silver Produced (kg)	211	376	461	506	531	70
Lead Produced (tonnes)	80	179	243	251	280	25
Gold Equivalent (oz)	3,161	5,905	6,469	6,402	7,308	1,006

16.1.2. KERR PIT

Similar mining methods are employed in the Kerr Pit, with a shallower pit base approximately 146 m above mean sea level. The pit is designed to form part of the tailings management system for the mine, with part forming one of the tailings paste storage cells and part forming a pre-settling area for water storage and re-cycling.

16.2. PROPOSED UNDERGROUND MINE

Galantas has carried out an internal 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 design parameters for the scoped underground mine are based around a production of 50,000 ounces per year at an in situ grade of 8.19 g/t gold being the grade of the total resource of the Kearney and Joshua veins (see Table 15). ACA Howe do consider this production rate somewhat challenging and, accordingly, other production scenarios are considered in the Preliminary Economic Assessment (PEA) below. Current drilling has targeted 10 years of production, with longer term plans targeting 15 years which is the approximate capacity of the tailings storage capacity proposed for planning permitting purposes. Allowing for mining dilution of 10% which is considered achievable for the mining method and a mill recovery of 85% proven by past performance, the annual ore to be mined is estimated at 245,000 tonnes per year with a maximum capacity of approximately 316,000 tonnes per year.

The mining method proposed by Galantas is ‘Shrinkage Stoping with Backfill’, or ‘Cut and Fill’ in areas not suited to shrinkage. Access to the underground will be via a prefabricated ‘cut and cover’ ramp installed within the backfilled open pit and a spiral ramp developed from the bottom of the prefabricated ramp at the base of the pit. Rubber-tyred diesel loaders, trucks and development jumbo rigs are envisaged with jackleg operations within the production stopes.

16.2.1. PERSONNEL

The current open pit operation provides employment for approximately 50 persons, drawn mostly from the local communities. The proposed operation is anticipated to provide employment for approximately 130 persons. Of these, over 120 employees are expected to be drawn from the local labour market, which contains individuals with appropriate skills. The current operation has a policy of training local labour to carry out specialist work, if experienced individuals are not immediately available within the local work-force. Current training procedures are suited to the application and /or the individual and vary from "on-the job" training, short (day) courses, day release courses over several years and experts brought in to train staff for specialist tasks. Training courses have been implemented using Omagh College and other training providers. It is Galantas' intention to continue this policy.

Galantas estimate that the wages cost of the proposed operation, including National Insurance and statutory pension provision will total approximately £4m per annum, and that over £1m per year will go to the government exchequer in the form of direct taxation and national insurance.

16.2.2. UNDERGROUND ACCESS

The proposed underground mine requires three accesses, two in stage 1 and a third in stage 2. These accesses provide emergency egress, access for transport of ore and waste, personnel, materials and ventilation. The stage 1 accesses will comprise a drift and a shaft. The stage 2 access comprises a drift or a shaft, with connections to the stage 1 access.

The stage 1 drift will be formed within the backfilled Kearney open pit. Concrete or shotcreted arches or steel arches with concrete cover will be set within compacted fill material, using a cut and cover method, allowing for a drift of minimum size 4 m width by 4 m height. A drift and lower portal are already the subject of a planning application but are also described within this document. The base of the drift will be situated close to the western wall of the Kearney pit, within a chamber constructed with reinforced concrete. The chamber will give access to a spiral decline developed within the area to the west (footwall zone) of the Kearney vein. This area is expected to contain the most structurally competent rock locally available. Lateral development, paralleling the strike of the veins will be driven at a gentle gradient to intersect a shaft. The shaft will provide both ventilation and a route of emergency egress. The shaft will either be blind drilled from the surface, or raise bored by a pilot hole drilled downwards and reamed to full width upwards. In either case, a rolled steel tube lining, welded in sections, will be grouted in to support the shaft walls, or a lining will be formed by spraying with fibre-reinforced shotcrete, as ground conditions determine. The shaft will be of a circular cross-section and approximately 2.5 m in diameter. Approximately half the cross-sectional area of the shaft will be occupied by galvanised steel grating platforms linked by fixed ladders, forming the second (emergency) egress. The siting and design of the shaft and furnishings is compliant with the Mines (Safety of Exit) Regulations 1988 (MSER'88) and the associated Approved Code of Practice relating thereto.

During the development period, until the shaft forming the second means of egress has been connected, the number of persons permitted to work underground at the development function shall not exceed 9 persons at any one time (other than up to three additional persons who are engaged temporarily in investigation, inspection or the taking of samples). This is in compliance with Regulation 7 of MSER'88.

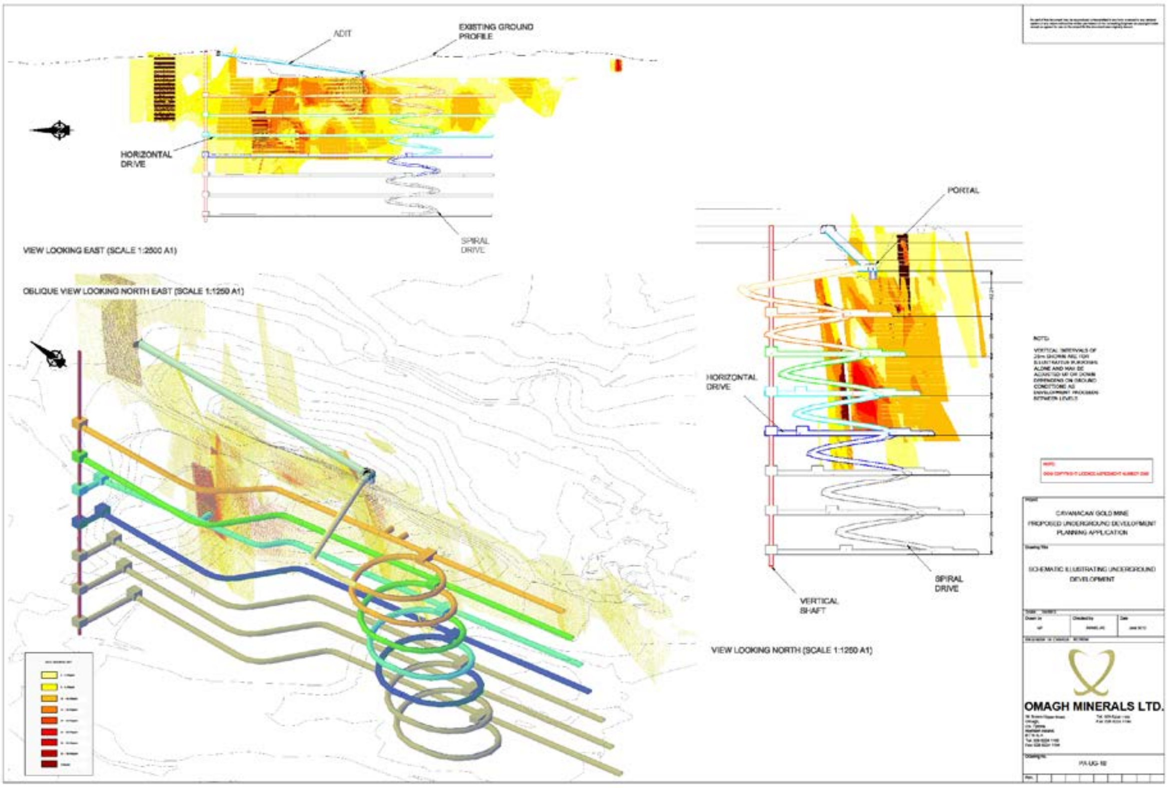


FIGURE 27. PLANNED UNDERGROUND DEVELOPMENT

Figure 27 shows the arrangement envisaged for the 'cut and cover' ramp within the backfilled open pit, the spiral ramp, and level development linking to the small diameter vertical shaft.

Consideration will be given to placement of the ventilation shaft in closer proximity to the ramp, developing a section of the raise as the ramp reaches each level. This would give assistance in ventilating during development of the ramp, and in the ventilation of the early development of the mine stoping infrastructure.

A second drift or shaft will be considered for construction during a second stage of mine development. It will be situated further to the west and will exit the mine at a position close to the processing plant. The function of the second drift is to provide emergency egress to an area of the mine remote from alternative egress and to provide ventilation and transportation access for veins in the western part of the mine freehold. A single linkage to the initial Kearney development will be made from a west to east direction and this development will enable access for Kerr, McCombs, Crystal and Joshua veins, lying to the west of the Kearney vein.

The underground plan allows for extraction to a depth of approximately 350 m below surface rock-head level. The plan is designed so as to allow mining to continue below 350 m as there is strong likelihood that additional gold resources will be identified at depths below 350 m. The underground plan allows for exploitation of mineral resources beneath the Omagh Minerals freeholds, though the company is also the holder of leases to minerals outside of its freeholds. For clarity, the planning application does not limit the underground depth of extraction below the Omagh Minerals freeholds, or limit the underground location of mining beneath that freehold land. This is because additional resources are expected to be available at depth and discovery of additional veins is anticipated during the excavation of the drive linking Kearney and Joshua detailed in the paragraph above. It is not known at this stage if those particular veins will prove economic to extract, less being known from a technical perspective.

16.2.3. UNDERGROUND MINING METHOD

The anticipated development calls for the concrete prefabricated ramp, to be installed within the waste filled open pit, to be at a gradient of 1 in 7, and for the spiral ramp to be developed at 1in 6 gradient and nominally 3.5 m wide by 3.8 m high.

Level spacing is envisaged to be between 30 m and 45 m. The height separation will be determined by local ground conditions. The ability to exit or vary the spiral development provides flexibility to the mine design to accommodate varying vertical height separations. Cross cuts in waste from the spiral ramp will be made to the vein.

A footwall drive parallel to the vein will be developed (as shown in Figure 27) and cross-cuts into the vein made. Stopes (sub-vertical working places), nominally 30 m long will be established with the formation of a 'reef drive' along the vein. Minimum stoping width will be nominally 1m, although narrower widths will be permitted up to 5 m in stope length. A central raise by either conventional raising by hand or by raise-borer will be made. Sill pillars will be left in place varying according to the specific engineering requirement of each level, although it is envisaged at this stage that these will be 5 m. Rib pillars are not envisaged as being necessary. Stopes will be extracted alternately, as 'primary stopes' which are then backfilled. Intermediate, 'secondary' stopes, will only be extracted following completion of all primaries on each level, thereby giving plenty of time for cemented backfill to cure and provide strong support. Cross cuts into the vein will form draw-points from which ore can be drawn from the stopes by load-haul-dump machine (LHD), loading into trucks on the footwall drive for transportation to the crusher. At each end of the stope, cross cuts to the stope will include provision for access to end raises constructed with timber within the broken stope ore, formed as the stope is mined. Mining of the vein will be by 'over hand shrinkage' stoping, or 'cut and fill' should shrinkage prove impractical. 'Shrinkage' requires the broken swell material, approximately 30%-40% of the blasted ore, to be drawn from the stope to establish a gap above the broken ore and into which to blast the next 'lift' of in situ, unbroken ore, to permit access for personnel and equipment for continued mining. When the mining of an individual stope is complete, the remaining 60-70% of the ore is drawn off. The stope is then backfilled with either waste development rock or tailings or a mixture of both and may be cemented as required. Where more than one vein stringer is workable and accessible by extension of the cross-cuts, a scheme of work will determine the detail of the stoping arrangement to ensure maximum stability. In all probability, extraction would commence with the hanging wall stringer, should included waste prevent economic extraction simultaneously.

A variant of the described mining method would be to create the initial in-vein 'reef drive' development at a few metres higher elevation than the associated lateral waste development with inclined box-holes linking the two. Chutes placed on the end of the box-holes would be used to regulate off-take of ore from the stope and discharge directly into haulage trucks. The chutes are hung between a pair of arched steel sets. After emptying the stope of ore, the box-hole can be filled with waste and grouted or concreted, at the same time as the stope is backfilled. The steel sets and chutes can then be potentially re-used. Air powered slushers could possibly be deployed in the safety of the protected raises to clean the stopes of ore, prior to back-filling.

A problem with shrinkage stoping in narrow vein mining is the possibility of 'hang-up' of broken ore, with the bridging of ore between the walls of the stope as 'drawing' off of the broken ore takes place. In the swell production stage this would present real danger to working personnel as they require to gain access on to the broken ore in order to drill and blast the next 'lift'; they would not be aware of any void that might have been created in the ore upon which they are working. In order to prevent personnel placing themselves in such dangerous situations, Galantas state they intend to pull swell utilising scrapers, scraping the ore into timbered chutes, constructed adjacent to the end access raises described above and advancing construction vertically within the shrinkage ore, with the access raises, as the stope is advanced.

An alternative stoping method may be deployed at the mine. The method, 'Cut and Fill' stoping, may be deployed in whole or in part. This is a method where the broken ore after each blast is drawn off using scrapers, pulling the ore into timbered chutes at the extremity of the stope; backfill material is then introduced into the stope via the central raise, distributed throughout the stope utilising the scraper, and on top of which a temporary floor is created (either timber or cement based). A fresh lift of ore is then mined upwards and the process repeated, cleaning down to the temporary floor. Broken ore is drawn from the base of the raise and loaded into trucks for transportation. The floor is either lifted in the case of a timber floor or retained in the case of a cement floor ready to accept the next lift of waste. The method has advantages where ore is not free flowing and is particularly suitable in areas that are structurally less competent.

For scoping purposes it has been assumed that each stope will be 30 m wide, 30 m high and have a 5 m thick sill pillar. A minimum mining width of 1 m is used although 0.7 m will be permitted over a short section of strike up to 5 m.

16.2.4. GROUND SUPPORT

Ground support of the spiral decline and lateral developments is generally to be by pattern rock-bolting, with a nominal pattern of 2off. x 1.8 m and 3off. x 2.4 m resin grouted rock-bolts with 150 mm x 150 mm domed steel plates, per 4 m length of spiral decline. 1.2 m split set rock bolts (or swellex) bolts additionally secure weld-mesh deployed in the crown area of the arch. Junctions, where spans are greater, will also require shotcrete support and fibre admixtures to the shotcrete may be deployed as necessary. Drives on individual levels are to be supported by the use of split sets (or resin bolts if required). Two 2.4 m split sets with plates will be installed per 1.5 m of drive.

Additional meshing and shotcreting will be deployed as required. Bolting takes into account the predominant joint and fracture sets. These have been studied in some detail within the open pit zone and within cores and have been reported upon by SRK Consultants of Cardiff. Further technical assessment work is continuing including oriented fracture studies on deep drill cores. An allowance has also been made for the partial use of steel sets if ground conditions determine this is required. It should be noted that bolt size, frequency and type is not definitive and will be varied in accordance with ground conditions.

Temporary in-stope support, to enhance operating safety for the stoping crew and to secure the hanging wall of the vein during the stope draw down procedure, will be installed. It is expected this will rely on split-set or swellex rock bolts with mesh or corrugated straps as required. The scoping study is costed on the basis of a 2 m x 2 m pattern of 0.9 m and 1.2 m split sets each with 150 mm x 150 mm x 4 mm steel plate. 2.4 m bolts would be used as ground conditions and vein width determined.

Permanent in-stope support will be acquired by a back-filling process. The materials to be used will be mainly clean tailings (which have safe low metal and sulphide content and are non-acid generating), with the addition of country rock from development as required. The addition of cement within back-fill aggregate has been allowed for.

Additional support will be required at draw-points. This will be in the form of pattern bolting, mesh and fibre shotcrete. Arch sets (temporary or permanent) may also be deployed.

Central stope raises will be supported by fibre shotcrete as required.

In general, the mining method, "Shrinkage Stoping with Backfill" provides good short-term support during the mining operation and excellent long-term support thereafter.

16.2.5. DRILLING

Drilling of shot-holes for blasting will be by two methods. A single boom drill jumbo will be utilised where access is available. Hand operated jackleg and stoper drills will be utilised within stopes, where a mechanised jumbo is not deployable. Pneumatically powered units are available with specially designed handles which reduce the exposure to hand arm vibration. The maximum shift exposure time for such units is approximately three hours and the stope crew may rotate through various functions in the stope to permit drilling by this method to be efficient. Electrically powered jackleg drills are also available, and will be tested. Water flushing will be utilised in both types of apparatus to minimise dust production. There is an option to utilise a long hole rig for box-holes etc and this will be examined and included in further study.

16.2.6. BLASTING

Blasting will be by a combination of Anfo (Ammonium Nitrate/Fuel Oil) with priming cartridges. Half second delay electric detonators and non-electric (Nonel) detonators will also be used where appropriate. Vibrations from blasting will be minimised and held within the existing planning consent levels (6 mm/s for inhabited dwellings and other buildings not owned by the operator).

Structural surveys of adjacent properties will be offered and initial monitoring carried out to ensure no damage results from the use of blasting underground. The mine management will be responsive in discussions with neighbours in terms of any disturbance and seek to minimise same.

16.2.7. LOADING AND TRANSPORTATION

The mine is to be worked by trackless mining methods, utilising diesel powered, rubber-tyred vehicles and electrically powered conveyors. The size and gradient of drifts and the spiral decline is determined by this requirement. In general, the cut and cover drift will be at a gradient of 1 in 6 and the spiral decline will have a gradient of 1 in 7. The spiral decline will be arch shaped and have a width of 3.5/4 m and a height of 3.8/4.5 m. In general the spiral decline dimensions are decided by the selection of a 20 tonne truck. Lateral driveages may be narrower because they do not have significant bends. Cross-cuts will generally be narrower with flared ends to accommodate the LHD scooptram.

Re-muck stations will be cut away from the spiral decline at mid-point between levels and roof height elevated to permit loading at these points. Re-muck stations are used as temporary storage locations during underground mine development and are a necessary part of the efficient operation of the development method. Re-muck stations may also be utilised as ramp safety run-off zones or other uses such as underground stores or service bays.

Truck loading points (or muck-bays) in lateral drives will also have a greater height profile and the points selected according to transport, ground conditions and potential box-hole placement.

The gradients of the spiral decline was determined as 1 in 7 as it permits a less intense gradient for haulage vehicles than the 1 in 6 gradient of the cut and cover adit, where a conveyor is intended to be the principal method of transporting mineral.

A crusher will be installed at the base of the conveyor in the concrete ramp, in order to reduce the size of ore/waste prior to conveying.

16.2.8. MINING SEQUENCE

The first lateral development will be made at a level 15 m below the base of working in the Kearney open pit. This is to form a Crown Pillar in safe ground below the open pit workings. Development and production has been planned to allow two levels a year to be worked, to allow for some redundancy in the availability of safe working places.

The total distance of drive to the first use of a full ventilation circuit and emergency egress is approximately 675 m. By prioritising this and achieving one round per shift and ensuring three shift working, it is estimated that this could be achieved in 20 weeks.

16.2.9. VENTILATION

The ventilation volume is determined by the amount of installed diesel horsepower. It is anticipated that up to 58 m^3/s flow will be required on the basis of 1001 diesel installed horsepower. In order to arrange that the direction of ventilation flow matches that of the conveyor situated in the cut and cover drift, a pair of main forcing fans will installed at the top of the vertical raise bored shaft. The fan will be installed within an embanked building which contains an air lock. Half the diameter of the shaft will be utilised as a ladderway with stagings (to M&Q HSE regulations). The provision of a small diameter lightweight emergency winder is being examined to speed emergency casualty evacuation through the remaining half of the shaft diameter, which will be clear of obstructions to airflow. The top of the shaft will contain an area for emergency winding if required. There will be a facility for reversal of ventilation should this become necessary in an emergency. Potential noise from the fan will be reduced by the use of surrounding embankments. Underground, fugitive dust production will be reduced at source by the use of sprays on broken ore piles at draw points and by wet drilling. Additional dust suppression sprays will be fitted at conveyor discharge points or tipping points as required, if the rock is insufficiently wetted. Auxiliary fans with ventilation ducting (both tubing and bagging) will be provided for use before the main ventilation system is developed and for the ventilation of blind developments after the main fans are operational.

16.2.10. PRINCIPAL EQUIPMENT

The main items of equipment required for the underground mining project are as follows :-

- Diesel Load-Haul-Dump units x2
- Electro-Diesel Single Boom Drill Jumbo x 2 (phased delivery)
- Jackleg / Stoper Drill sets x 4
- 20 t dump trucks x 2 (phased delivery)
- Rockbolting attachments
- Shotcreting unit
- Telehandler
- Primary crusher
- Conveyors
- Conveyor loader / (afc)
- Compressor
- 4 inch diam Victaulic steel / poly pipe
- Auxiliary fans and bagging
- Workshop equipment
- Main Pump and pipe-lines (polypropylene)
- Auxiliary pumps and line
- Gate-end electrical control boxes
- Diesel generators
- Transformers
- Fire suppression equipment
- Fuel transportation, storage and dispensing equipment
- Lubrication transportation, storage and dispensing equipment
- Explosive magazine
- Explosive Security arrangements
- Explosive transportation and use
- Utility vehicle for transportation of equipment and personnel
- Tractor & trailer unit

- Miners Ablutions trailer
- Miners changing room and dry
- Lamp room, lamps, breathers and charging rack
- Air quality testing equipment
- Underground communications system
- Back-fill and cementation equipment

The equipment list is not intended to be prescriptive or exclusive.

17. RECOVERY METHODS

17.1. INTRODUCTION

Initial froth flotation testwork was carried out by Lakefield (1992) and the results of that testwork incorporated into the existing plant. Concentration by froth flotation is carried out on ore extracted from open pits on the Kearney, Kerr and Joshua veins. The froth flotation process used is a low toxicity and environmentally safe process.

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 new plant assumes a 50 tonnes per hour feed rate at an average diluted gold grade of 5.5 g/t. Items envisaged requiring upgrade or replacement in the new plant include a cone crusher, ball mill, flotation cells, plus pumps, motors, hoppers, cyclones and electrical controls. Alterations to conveyors, feeder arrangements, concentrate handling arrangements and modification to the existing building and steelwork are also allowed for within cost estimates.

17.2. MINERAL PROCESSING

The existing plant comprises a three stage crushing system. A 24 inch x 16 inch Jaw Crusher feeds a 2 ft standard cone, which in turn feeds a 2 ft shorthead cone crusher, the latter operating in closed circuit. A clay screen is incorporated in the crushing circuit to reduce clogging. A 7.5 ft (diameter) by 10 ft Denver ball mill further reduces size and material is pumped to a cyclone. Cyclone oversize is fed to a secondary ball mill (approximately 1.5 m in diameter by 3 m), which also takes clay wash undersize removed in the crushing process. After conditioning, the material is processed to concentrate in a configuration of flotation tanks, as roughers, cleaners and scavengers. Water is extracted from the concentrate by a hydraulic filter press and the material placed in plastic lined bulk bags for shipment by standard 20ft steel container. The historical size range of milled concentrate has been between 40% and 60% passing 75 micron, and between 70% and 90% passing 180 micron. There has been a trend towards coarser grind as mill feed has decreased in grade and feed rate increased, but it is understood, with no apparent reduction in concentrate grade. The average gold grade of concentrate is approximately 100 g/t, but is occasionally below 80 g/t and above 110 g/t. Concentrate contains between 25% and 34% sulphur, routinely around 30% sulphur. Recovery is between 80% and 90% of gold feed through to concentrate. Shipping moisture varies between 7.5% and 9.5% water. The silver content of a typical shipment is approximately 220 g/t and lead content is variable, at between 6% and 14%. The silver - gold ratio is higher when higher lead contents are present due to silver occurring within the lead minerals (dominantly galena). A small penalty is payable for arsenic content, which typically is within concentrate at less than 6% but may run higher. Copper is present as a minor element, between 0.22% and 0.44% and is not payable within the contract.

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 new plant assumes a 50 tonnes per hour feed rate at an average diluted gold grade of 5.5 g/t. Items envisaged requiring upgrade or replacement in the new plant include a cone crusher, ball mill, flotation cells, plus

pumps, motors, hoppers, cyclones and electrical controls. Alterations to conveyors, feeder arrangements, concentrate handling arrangements and modification to the existing building and steelwork are allowed for within cost estimates.

Table 18 below indicates a summary of the requirements for the proposed plant.

Mill Application	Fresh feed (t)	Recirc Tonnage	Total Tonnage	Internal Diameter	External Diameter	Length	RPM	Power Req (kWh)
Closed circuit with cyclones	50	125	175	5.30m	5.49m	5.49m	12.4	2418.36
Open Primary feeding cyclone	50		50	3.93m	4.12m	3.96m	15.3	880.26
Closed Secondary feeding off Cyclone U/F	125		125	3.47m	3.66m	3.66m	16.3	631.53
Regrind Mill ClTl+ScCn		8	8	1.68m	1.83m	1.83m	25.5	61.86

Mill Ball Application	Maximum Size (mm)
Primary Mill closed with cyclone	93.90
Primary Mill open circuit	96.72
Secondary Mill fed with cyclone U/F	25.48
Regrind Mill ClTl+ScCn	11.89

Cyclones	Fresh Tonnage	Total Vol (m3/hr)	Max Cyc Vol (m3/hr)	Int. Dia. (mm)	Apex Dia (mm)	Vortex Finder	No of Cyc Req	Cyc No. Inc Mtce
Closed or Open Circuit Milling	175	437.5	38.96	250	57.15	76.2	11.23	14

Cell Application	Fresh feed (t)	Total Vol. (m3/hr)	Residence Time (hr)	Total Vol Req	Cell Size Req (m3)	Avail Cell Model	Cell Vol (m3)
Rougher (8 Cell)	50.00	142.86	0.67	95.24	11.90	DR500	14.2
Rougher (12 Cell)	50.00	142.86	0.67	95.24	7.94	DR300	8.5
Scavenger (8 Cell)	43.25	123.57	0.33	41.19	5.15	DR180	5.1
Scavenger (12 Cell)	43.25	123.57	0.33	41.19	3.43	DR100	2.8
Cleaner (2 Cell)	8.00	22.86	0.25	5.71	2.86	DR100	2.8

TABLE 18.SUMMARY OF THE REQUIREMENTS FOR THE PROPOSED PLANT

Detailed quotations have been received for all major replacement plant items and are included in the capital estimate.

18. PROJECT INFRASTUCTURE

18.1. GENERAL

Galantas state that surface facilities have been concentrated as closely as possible, bearing in mind safety considerations for minimum separation distances. The number of permanent buildings allowed for have been minimised to facilitate easy reclamation and minimise capital expenditure. Permanent buildings include the Fan House, which is to be built of concrete block-work with a rendered finish. The density of the materials proposed to be used assist in the absorption of noise and provide security to the shaft access. The remainder of buildings are of the portable type and include the miners changing room, ablutions block, offices and lamp room. The surface facilities are located some 100 m within the site boundary so as to minimise disturbance to neighbours, and the area is surrounded by landscaped, planted berms designed to provide a visual and sound shield.

A conveyor system is proposed to exit the mine and is designed to carry both wetted ore and waste rock at separate times. A pivoted stacking conveyor allows the operator to switch material stockpiles. The pivoting conveyor carries water sprays to prevent dust being raised. Material is drawn from either pile according to requirement. Ore is stocked within a simple steel cladded structure, which is equipped with water-sprays if required, to prevent the pile from drying out. Ore is drawn from the structure by front-end loader as required by the plant. Until the conveyor system is installed, low profile mine trucks, employed in mine development and haulage, will exit to surface and deliver to the same stockpiles areas described.

18.2. ELECTRICAL SUPPLY

A sub-station will be provided adjacent to the fan building for connection to mains power. The substation will serve the main fan(s), the conveyor arrangement, underground lighting, underground crusher, underground workshop, auxiliary fans, pumps, underground compressor and other items. Total electrical requirement for the mine is approximately 1,000 kVA. The mill will switch to the use of mains power (with diesel generator support) and expansion will increase mill demand from 1,000 kVA to 2,000 kVA. The existing generators will be supplemented with additional units and will be utilised as back up and for periods of high mains power cost. Total estimated load is estimated at 3,000 kVA.

19. MARKET STUDIES AND CONTRACTS

The market for gold, the primary constituent of the concentrate, has been rising steadily over a ten year period and has reached a plateau over the last quarter of 2011 and the first quarter of 2012 during which it has traded between £1100 and £1000 per ounce or between $1640 and $1750 on a monthly average basis. This is illustrated in Table 19 below.

TABLE 19. GOLD PRICE, DEC 2010 - APR 2012		
Date.	Gold Price	Gold Price
	Sterling £/ounce	US$/ounce
31-Dec-10	891.46	1390.79
31-Jan-11	859.27	1356.40
28-Feb-11	851.52	1372.30
31-Mar-11	881.19	1424.01
30-Apr-11	901.49	1473.81
31-May-11	926.06	1510.44
30-Jun-11	942.95	1528.66
31-Jul-11	974.50	1572.81
31-Aug-11	1073.17	1755.81
30-Sep-11	1121.46	1771.85
31-Oct-11	1057.05	1665.21
30-Nov-11	1100.30	1738.98
31-Dec-11	1055.00	1643.95
31-Jan-12	1067.76	1656.12
29-Feb-12	1103.55	1742.60
31-Mar-12	1057.94	1673.77
30-Apr-12	1030.29	1650.07

A gold price of $1375/ounce has been used in the financial evaluation, a price not seen since February 2011, indicated in Table 19 above.

Froth flotation concentrate is sold under a long term contract with Xstrata Corporation. This arrangement is expected to continue. The Xstrata contract forms the basis of revenue calculation. The concentrate produced is saleable worldwide and several smelters bid for the contract when it was first made available. It is Galantas' expectation that the existing Xstrata off-take contract may need to be confirmed at higher production levels but given the concentrate's saleability, no difficulty is anticipated in placing the concentrate at similar rates with alternate parties, should the Xstrata contract not continue.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1. ENVIRONMENTAL STUDIES

Galantas state that the proposed mine has been subject to a wide range of studies, which incorporate monitoring data arising over a 13 year period. The net result of the studies is that, provided the recommended ameliorating measures are put in place, the environmental and social impact will be restricted and kept to a minimum. Galantas inform that the full Environmental Impact Assessment has been prepared and is shortly to be filed with the regulating authorities. In September 2011, the Northern Ireland Environment Agency carried out a detailed compliance check on the waters leaving the mine drainage system and determined that the mine met all the required criteria.

20.2. TAILINGS AND WASTE ROCK DISPOSAL

Disposal of waste rock and tailings is key to approval of a planning application for an operating mine in the United Kingdom. The following outlines this process, as proposed by Galantas. A 15 year time span is considered, which although longer than current defined resources permit, is the time period that Galantas is planning for, with high expectation of resource expansion through exploration.

Tailings, comprising clean silt and sand, are the product of ore processing. The content is currently routinely monitored for environmental and operational purposes. Contained metal and other values are consistently substantially below environmental safety benchmarks. The tailings are non-acid generative.

The tailings are suitable for a variety of purposes. A coarser fraction is suitable for aggregate purposes. Tailings are suitable, when mixed with peat, to form an artificial soil. They are suitable for interstitial back-filling of open pits, when mixed with rock. They are suitable for back-filling into underground stopes, giving permanent support to those excavations. When not utilised for any of the aforementioned, disposal within surface paste cell arrangements is employed.

Galantas propose to use all of the above methods to dispose of tailings, with an emphasis on all except the aggregate use, which is held to a minor quantity. This is because only the aggregate use requires use of the public road network and Galantas has determined a policy, in response to consultation, to reduce this as far as possible.

Approximately 50% of the tailings produced are intended to be used for back-filling underground. In the back-fill process, tailings may be mixed with rock and or cement, dependent on the type and design of backfill required.

Approximately 336,000 m^3 of tailings can be safely stored in permanent paste cell arrangements on surface. The paste cells are designed to produce compartmentalised storage behind secure rock berms, lined with till as required. Cells are constructed to be free draining and are separated from large stored water volumes. The arrangement means that in the unlikely event of a berm failure, water is not present in sufficient volume to mobilise substantial outflow of sediment, thus providing maximum environmental protection. The individual heights of each paste cell allow post-filling landscaping to mimic existing land-forms and create a more natural appearance to the landscape than is possible using a traditional wet tailings arrangement. Paste cell storage requires larger quantities of rock than a single conventional wet tailings dam because of the number of berm impoundments to be formed and is thus more expensive to construct, although the construction cost may be spread out during the period of mining. The extra rock required for berm construction is available on site. Approximately 261,500 m^3 of rock and till is required to build the impoundments and 366,000 m^3 of till, peat and rock are required to landscape to the proposed final restoration surface. Up to 40% of the rock component will be required at the commencement of work to build the cells. It is proposed to use part of the waste rock stockpile for this purpose, created from the waste of the open pits. This will significantly reduce the amount of surplus country rock to be disposed of by road as part of the open pit restoration process.

ACA Howe inspected the current paste cell arrangements and was impressed with the construction and disposal system.

FIGURE 28. TAILINGS PASTE CELL BEING FILLED

Most restoration projects for large sites need additional quantities of top-soil or artificial soil to complete restoration landscaping. This is because soil is used for temporary landscaping purposes during the project on stockpiles, etc. Some soil becomes mixed with other materials as a result of the temporary uses, and a portion becomes unsuitable for re-use. Trials have been carried out and tailings mixed with peat have been found suitable for use as soil.

The Kearney open pit is intended for back-filling with rock. The pore volume of the coarse rock material, even well compacted, is potential storage space for tailings. Tailings may be mixed with rock-fill up to a ratio of 1:1 and produce a stable, compacted, mixed material. The Kearney open pit has an open volume of some 550,000 m³, producing a potential disposal of 642,000 tonnes of tailings. Since the tailings will, for the most part, be packed in interstitial volume (there being a 40% void), there is a net saving in disposal space estimated at only a loss of between 50,000 and 60,000 tonnes of rock disposal space. This disposal space has been replaced by the proposed re-creation of the pre-existing hill landform of approximately 750,000 m³ (1,350,000 tonnes) and the landform provides extra rock disposal capacity.

A fraction of the tailings produced is naturally coarser than the rest. It is estimated that, by use of a simple cyclone, a coarse fraction can be produced which contains the coarsest 25% of the material. This material has potential uses as aggregate filler in bitumen, concrete and plaster products, as well as being suitable for some other building purposes. Export of cyclone sands from the mine site is estimated potentially to average 13 vehicles per week, carrying 20 tonnes each of coarse sand, dependent on demand.

Table 20 below estimates tailings disposal within the disposal types proposed over a 15 year period.

TABLE 20.TAILINGS DISPOSAL ESTIMATES

Tailings Disposal Type	% of Tailings Produced	Tonnes Tailings Total	Tonnes Per Year	Number of Years Disposal
Disposal Demand		4,742,706	316,180	15
Backfill	50%	2,371,353		
Paste Cells		507,941		
Kearney Fill		641,750		
Artif. Soil to cells		105,700		
Cell Profiling		108,555		
Landscaping		778,973		
Export		180,000	12,000	

Rock production from underground development will be disposed of by utilising as back-fill in underground permanent support, by export from the mine site and by permanent elevation of the landscape of the mine-site. Approximately 100,000 tonnes of waste rock will be produced each year from main underground ramp and ore development. A further allowance of 60,000 tonnes per year has been provided against secondary development drives. The rock is clean of any contaminants and suitable for many construction purposes. Between 8,000 tonnes and 10,000 tonnes per year will be utilised as mine back-fill. Combined export of aggregate material (sand and rock) from the mine site will be limited to 40 loads per day, 200 loads per week. After disposal of sand aggregate, some 180+ loads per week may be allocated to the disposal of rock. It is anticipated that an average of 35 x 20 tonne, lorry loads per week will be required for rock disposal during the life of the underground mine, though this could increase if extra development is needed in a given 12 month period. A stockpile size of 100,000 tonnes has been allocated in site design to allow for peaks and troughs in mine supply and customer demand.

Galantas state the site construction requires some 2.1 m tonnes of rock and till for building and profiling paste cell retaining structures and restoring the landscape. As a consequence, there will be a reduced annual requirement for road traffic compared to the open pit closure plan currently permitted because some of the surplus rock currently required to be disposed of off-site can be integrated into the construction requirement.

Table 21 below estimates rock consumption and production for the underground mine.

TABLE 21. WASTE ROCK PRODUCTION AND DISPOSAL				
Rock / Till Produced	**Mine Life Cubic Metres**	**Mine Life Tonnes**	**Per year tonnes**	**Number of Years Disposal**
Ore & Ramp Development	820,585	1,477,054	98,470	15
Other Development	500,000	900,000	60,000	15
Stock	632,951	1,139,312		
Total for disposal	**1,953,537**	**3,516,366**	**234,424**	**15**
Backfill	75,000	135,000	9,000	15
Cell Construction	261,467	470,641		
Cell Till / Peat	77,000	138,600		
Cell Profiling	145,960	262,727		
Kearney Pit	425,000	765,000		
Landscaping	680,000	1,224,000		
Rock Export	291,667	525,000	35,000	15
Total disposal	**1,956,093**	**3,520,968**		

20.3. PUMPING

The strata in the mine area are relatively tight in terms of water permeability. The base of the open pit (at 40m below rock-head) is lower than the surrounding land. The majority of water entering the open pit does so along the rock-head / surface till interface and is derived from rainwater within the immediate catchment. There has been no significant spring or other water make exposed within the open pit.

Current water flows from the site average a daily rate of 111 gallons per minute (8.1 litres per second) with a peak maximum flood condition of approx 900 gallons per minute (68 litres per second). The average is calculated since flow monitoring began on 07/01/2008. Approximately 20% of the flow is generated from the open pit.

The pumping capacity to be installed allows for surface water drainage at the above rates to continue and be pumped from the open pit area (at its base), though this will be reduced with strategic drainage. Additional emergency pumping capacity of 200 gallons per minute will be situated underground, though additional water makes from “weeps” is not expected to exceed the range of 50-100 gallons per minute, with an average less than 50 gallons per minute. An installed capacity of 200 gallons per minute is allowed for to provide redundant capacity in case of emergency or breakdown.

Potential material water sources at depth have been investigated during the exploration drilling program. No holes have been encountered with material water make. Given the non-porous nature of the country rock, in the event that a material water make is encountered, it will be grouted using conventional cementitious materials.

20.4. WATER QUALITY

A settling system has been designed to maintain the existing standard of water treatment. The existing polishing pond arrangement is retained. The capacity of upstream settling has been expanded by the creation of a proposed larger settling pond.

No change is anticipated to the existing discharge consent, which incorporates the required environmental safeguards.

The final discharge arrangement, showing the monitoring station is shown in Figure 29.

FIGURE 29. FINAL WATER DISCHARGE FROM THE MINE SITE

20.5. ACID DRAINAGE

Acid drainage is not an issue at Cavanacaw, with both tailings and rock having been analysed to have sufficient natural carbonate content to buffer respective sulphide contents. Long term monitoring of the existing water outflows confirm the analyses to be mildly alkaline, which provides a helpful buffer to the local naturally acidic drainage, derived from a peat enriched catchment.

20.6. RESTORATION

The majority of restoration works on the site will be completed within 5 years of the start of the project. Outstanding areas to be restored will be those paste cells still in development or use, the mill area, stockpile area and mine portal area. Paste cell restoration will follow the completion of the cells concerned. The existing stockpile areas will be reduced considerably in scale due to the building of the paste cells and export of surplus rock and will be restored in line with the permitted closure plan. A new restoration plan will be included in the planning application for the underground mine, which restores the natural landform previously present in the area of the Kearney open pit and permits extra landscaped storage of tailings and rock together. The object of the restoration plan is to minimise the amount of post-closure restoration required by completing as much restoration as possible on an on-going basis.

20.7. PROJECT INFRASTRUCTURE

Road access to the mine is by tarred road. Permitting has recently been received for road improvements to the public road to facilitate export of rock from the mine.

Existing infrastructure includes a polishing pond, water outlet arrangements, offices, existing mill and workshops. Omagh Minerals owns the freehold land upon which the existing open-pit mine has been constructed and has recently purchased additional land which overlies part of the Joshua vein discovery. Galantas management consider that Omagh Minerals own all the land required for operation of an underground mine of the scale anticipated.

20.8. PLANNING / PERMITTING

Planning procedures in Northern Ireland examine the complexities of projects of this type in a very detailed manner, with extensive consultation with regulating bodies, political authorities and interested third parties. The strong and positive economic impact of the project, within a community that has seen the shrinking of employment opportunities, is an important driving force at local and regional level, so far as permitting procedures are concerned. Whilst delays in the permitting procedure are likely, Galantas management considers it has a very strong case for approval. An application has already been made for permitting for the cut and cover adit and lower portal box within the current open pit (intended to be back-filled).

The underground mine, uprated processing plant and the export of a limited quantity of waste rock from the underground mine (to be integrated into the local aggregates industry) will require planning permits to be authorised by the DoE NI.

An Environmental Impact Assessment is in the final stages of preparation, with the associated permitting submission.

21. CAPITAL AND OPERATING COSTS

The majority of mine costs are based in UK£ Sterling and the price of gold is expressed in US$. An exchange rate of $1.60 has been used for the purposes of the headline numbers. Capital costs include a 20% contingency. Operating costs include no contingency.

Operating costs and capital expenditure have been calculated in an extremely detailed and comprehensive manner. Capital expenditure is estimated at £16.7million (US$ 26.72 million), including a 20% contingency. Detailed quotations have been received from suppliers for all major items. A further US$3.3million 'working capital' has also been allowed for in the first year.

The operating costs covering one year of the operation are as stated in Table 22 below.

	£
Labour	4,057,654
Fuel and Energy	1,941,184
Explosives	625,686
Roof Support	2,012,880
Drill Steel & Bits	100,287
Pipe	50,000
Mine maintenance materials	418,832
Milling materials & Consumables	2,626,795
Raise Drilling	250,000
Raise accessories	855,196
Environmental monitoring & control	210,000
Cells & restoration	200,000
Royalties	1,687,500
Freight	242,912
Licences	45,000
Rates	45,000
Insurances	120,000
Telephones	15,500
Post travelling etc	12,000
Training & HSE	46,000
Public Relations.	8,500
Accountancy	8,200
Legals	50,000
Total (£Sterling). **Total (US$)**	£15,629,127 **$25,006,603**

TABLE 22.OPERATING COSTS PER ANNUM

22. ECONOMIC ANALYSIS

The economic analysis is in the form of a preliminary economic assessment under NI 43-101 (PEA). The preliminary economic assessment is preliminary in nature, includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

22.1. PRELIMINARY ECONOMIC ASSESSMENT

The mine is situated in Northern Ireland and costs are expended in UK£ Sterling. Gold concentrate, the source of revenue, is valued and paid for in US dollars. The US$ to UK£ exchange rate impacts gross revenue proportionally to the change in the exchange rate. The exchange rate from December 2010 to April 2012 has averaged 1.595 $/£. A lower US$/£ exchange rate reduces payment in UK£ Sterling and reduces profitability. An exchange rate of 1.60 USD$/UK£ was used for the purposes of calculation.

The Galantas evaluation covers three scenarios, all at a gold price of $1,375 per ounce, as stated below.

A. The targeted annual rate of 50,000 ounces gold within concentrate, but allowing one year for construction and mine development with no production, then the second year at a reduced rate of 30,000 ounces. This scenario gives a mine life of 5 years after the first construction year.

B. An annual rate of 40,000 ounces gold within concentrate, again allowing for one year construction and mine development with no production, then the second year at a reduced rate of 30,000 ounces. Costs were split into ‘fixed’ and ‘variable’ but with labour costs treated as fixed, i.e. the same as for the 50,000ounce case. This scenario gives a mine life of 6 years after the first construction year.

C. An annual rate of 30,000 ounces gold within concentrate, again allowing for one year construction and mine development with no production. Again, costs were split into ‘fixed’ and ‘variable’ but with labour costs treated as fixed, as for the 50,000 ounce case. This scenario gives a mine life of 8 years after the first construction year.

The average ore grade used in the study is that calculated for the combined resource of Joshua and Kearney veins, 8.19 g/t Au, with an allowance for mining dilution of 10%, considered appropriate for the mining method to be employed.

Net revenues are calculated without corporate taxes, amortisation, depreciation or the cost of financing. Case A, at 50,000 ozs/year, gives total Mine Life revenue of £80.1million after deduction of capital and operating cost, IRR of 81% and NPV at 5% discount rate of £62.6million (US$100.2million).

Case B, at 40,000 ozs/year, gives total Mine Life revenue of £75.4million after deduction of capital and operating costs, IRR of 69% and NPV at 5% discount rate of £57.3million (US$91.6million).

Case C, at 30,000 ozs/year, gives total Mine Life revenue of £71.3million after deduction of capital and operating costs, IRR of 54% and NPV at 5% discount rate of £51.2million (US$82.0million).

For the purposes of this preliminary economic assessment, the Mine Life stated in each case has been calculated using the sum of Measured, Indicated and Inferred resources on the Joshua and Kearney veins. In conformance with NI 43-101, section 2.3, it should be noted that this economic assessment is preliminary in nature, that it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

There is no pre-feasibility or feasibility study as defined by of NI 43-101 on the subject property.

23. ADJACENT PROPERTIES

The exploration licence immediately adjoining the Omagh Minerals licences to the east and northeast is numbered MR 1/08 and held by Metallum Resources (see Figure 1). No significant deposits or mineralisation are known on that licence area.

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 km 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.

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, County 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. ACA Howe 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

Galantas is conducting a 15,000m drilling programme, for which results for the first 6,418 m were received on June 1st, 2012. A revised resource estimate based on these and previous drilling and channel sampling results is summarised in the following table:

TABLE 23. 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

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

A revised resource estimate and report is scheduled on completion of the drilling programme.

ACA Howe considers that there is good potential to define additional resources at Cavanacaw at depth and along strike on the defined veins, as well as on additional structures that may be identified through ongoing exploration.

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.

The mine design proposal by Galantas is for a mining rate for the recovery of 50,000 ounces per year of gold in concentrate. ACA Howe regard this target as challenging, and have performed sensitivity analyses, using the same labour cost, for 40,000 ounces per year and 30,000 ounces per year gold in concentrate. A gold price of $1,375/Oz has been used in each case and a USD/GBP exchange rate of 1.60 has been applied. The study is based upon detailed data available from an internal cost study by Galantas and the quoted Mine Life Revenue is net of capital and operating costs. The capital cost of establishing the underground mine and processing plant is estimated at £16.7m including a 20% contingency.

The results of these studies appear in Table 24 below.

TABLE 24. RESULTS OF PRELIMINARY ECONOMIC ASSESSMENT

Case Study Output/Yr	Mine Life Revenue Net	IRR	NPV @5% £	NPV @ 5% US$	Mine Life Yrs	Cash Cost Gold in concentrate US$ per ounce
50,000 ozs	£80.1 m	81%	62.6 m	100.2 m	5	500 $
40,000 ozs	£75.4 m	69%	57.3m	91.6 m	6	538 $
30,000 ozs	£71.3 m	54%	51.2 m	82.0 m	8	600 $

For the purposes of this preliminary economic assessment, the Mine Life stated in each case has been calculated using the sum of Measured, Indicated and inferred resources on the Joshua and Kearney veins only from the ACA Howe study, Inferred resource making up some 70% of the total. In conformity with NI 43-101, Section 2.3, it should be noted that this 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 categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

26. RECOMMENDATIONS

On completion of the present diamond drilling programme a revised resource estimate should be prepared.

A pre-feasibility study should be completed based on underground mining of resources as per the revised estimate and expansion of the existing mill.

ACA Howe considers that visual prospecting is an effective additional means of investigating the already known regional prospects, and could also be applied in a systematic way to investigation of the entire Dalradian area held under licence.

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28. CERTIFICATES

CERTIFICATE OF QUALIFICATIONS

Richard Parker
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United Kingdom

I, Richard Thomas Grenville Parker Bsc., FGS., MIMMM, as co-author of this report entitled ***"Technical Report on the Omagh Gold Project, Counties Tyrone and Fermanagh, Northern Ireland" ('the Technical Report')*** prepared for Galantas Gold Corporation and dated August 10th, 2012, make the following statements:

- I received the degree of Bachelor of Science in Geology from the University of Newcastle upon Tyne, England, in 1968.

- I am a Chartered Engineer (323907, March 30th , 1983) registered with the Engineering Council (UK), a Professional Member of the Institute of Materials, Minerals and Mining (Member Number 46546) and a Fellow of the Geological Society of London (17806).

- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with two professional associations, as defined in NI 43-101 and past relevant experience I fulfil the requirements to be a "Qualified Person" as defined in NI 43-101.

- I visited the Property between August 9th and 13th, 2011 and between March 14th and 16th, 2012.

- I have practised as a geologist specialising in Mineral Exploration and Development for 42 years, and as a Chartered Engineer for 32 years. Examples of my relevant experience for the purpose of the Technical Report includes authorship of the following reports:

  Magellan Copper and Gold plc, St Anthony copper deposit: Resource audit &Technical Report,
  Anglo Asian Gold, Gedabek Au-Cu Mine, Azerbaijan: Resource Audit,
  Geology and Mineral Resources of the Bilbao Silver-Lead-Zinc Deposit, Mexico, for Xtierra Inc. NI 43-101Report,
  Technical Review of Hummingbird Resources Limited Gold and Iron Ore Exploration Projects in Liberia for Hummingbird Resources,, AIM Qualifying Report, November 2010,
  Geology and Revised Mineral Resources of the Bilbao Silver-Lead-Deposit, State of Zacatecas, Mexico" for Xtierra Inc. and dated 4th April 2011, NI 43-101Report,
  "A Technical Review of the Yanfolila Gold Concession, Mali, West Africa" for Compass Gold Corporation July 2011, NI 43-101Report,
  "A Technical Review of the Dandoko Gold Concession, Mali, West Africa" for Compass Gold Corporation July 2011, NI 43-101Report.

- I am responsible for items 1-16 and 24-28 of the Technical Report.
- I have not had any prior involvement with the property that is the subject of the Technical Report.
- At the effective date of the Technical Report to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

- I am independent of the issuer as set out in Section 1.5 of NI 43-101.
- I have had no prior involvement with the mineral properties in question
- I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101-F1.
- 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.

Dated August 10th, 2012 at Edinburgh, Scotland, UK.

"Signed"

Richard Parker, FGS, MIMMM, C.Eng,
Senior Associate Geologist, ACA Howe International Limited

CERTIFICATE OF QUALIFICATIONS

Nigel Pearson
Senior Associate Mining Engineer
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254 High Street
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United Kingdom

I, Nigel Bruce Pearson Bsc., C.Eng., FIMMM, as co-author of this report entitled ***"Technical Report on the Omagh Gold Project, Counties Tyrone and Fermanagh, Northern Ireland" ('the Technical Report')*** prepared for Galantas Gold Corporation and dated August 10th, 2012, make the following statements:

- I received the degree of Bachelor of Science in Mining Engineering from the University of Nottingham, England, in 1968.

- I am a Chartered Engineer (366505, May 14th, 1987) registered with the Engineering Council (UK), and a Fellow of the Institute of Materials, Minerals and Mining (Member Number 48121).

- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with professional associations, as defined in NI 43-101 and past relevant experience I fulfil the requirements to be a "Qualified Person" as defined in NI 43-101.

- I visited the Property between May 16th and 18th, 2012.

- I have practised as a Mining Engineer for 44 years and as a Chartered Engineer for 25 years. Examples of my relevant experience for the purpose of the Technical Report include the following:

  Senior Management roles on the Zambian Copperbelt,, at Ashanti Goldfields Corporation (Ghana), P.T.Freeport (Indonesia) and Richards Bay Minerals (South Africa).

  I have authored or co-authored many technical documents, including the following*:*

  Technical and financial evaluation of the Dugbe F Gold Project of Hummingbird Resources plc, November 2011.
  Review of the Meli Project of Ezana Mining, Ethiopia, a possible Heap Leach prospect, July 2010.
  Scoping study of the Dino polymetallic deposit, Peru, May 2010.
  Technical and Financial evaluation of the Sierra Miranda Mine, Chile, March 2008.

- I am responsible for items 17- 23 of the Technical Report.
- I have not had any prior involvement with the property that is the subject of the Technical Report.
- At the effective date of the Technical Report to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I am independent of the issuer as set out in Section 1.5 of NI 43-101.
- I have had no prior involvement with the mineral properties in question.
- I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101-F1.

- 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.

Dated August 10th, 2012 at Ingleton, Yorkshire, England ,UK.

"Signed"

Nigel B. Pearson, B.Sc., C.Eng., FIMMM
Senior Associate Mining Engineer, ACA Howe International Limited

APPENDIX I

EXPLORATION POTENTIAL AND PRIORITISED PROJECT TARGETS

(from the Howe 2005and 2008 reports)

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	400,000	600,000	4.5	9.0
Elkins	35	H4061/7130	200,000	400,000	2.0	4.0
Joshua's	32	H3970/7072	190,000	380,000	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			**1,330,000**	**2,460,000**		
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 East Showing	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*			**1,730,000**	**4,609,000**		

* 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.

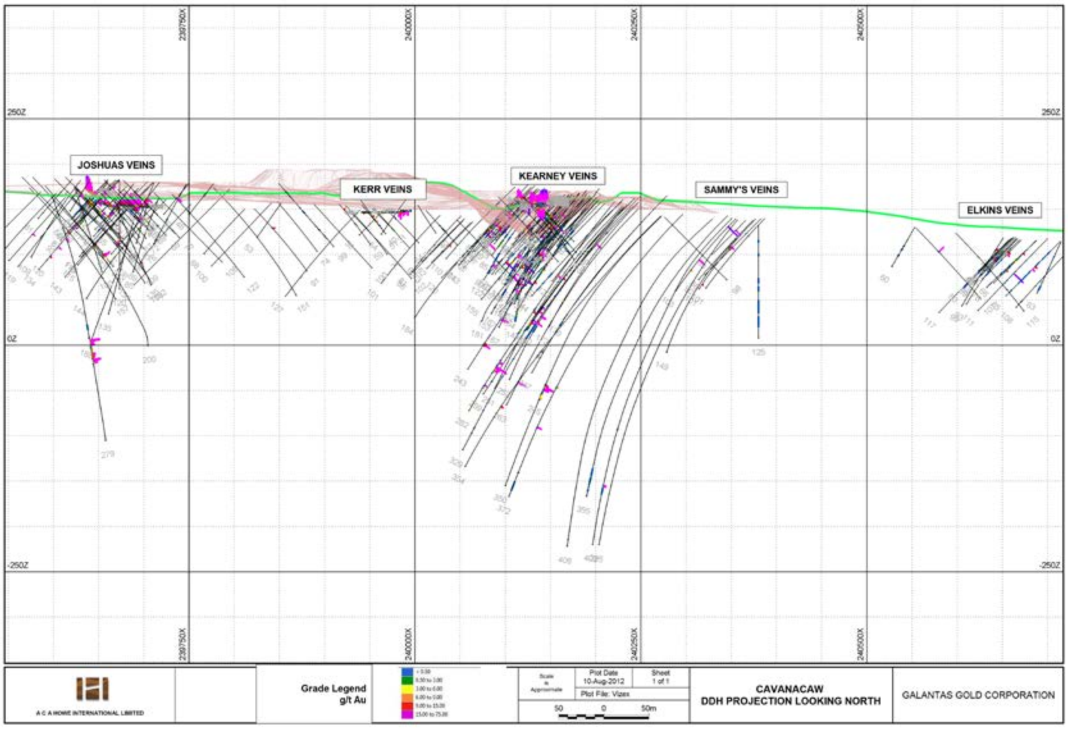
Ref. No.	Name	No.	Central grid reference	Score	Remarks including target-type airborne geophysical anomalies of 2005, if any
31-2005	Kearney	31	H401/710	10	Drilled with reserves (1995)and resources(1995 and 2004) , IP anomalies over 300m strike at S end and on 5 lines over 400m at N end of mapped 1000m of IP extended strike, weak VTEM anomalies over only N half of strike. On freehold.
32-2005	Joshua's	32	H3970/7072	10	Drilled with resources (1995), IP anomaly with 200m strike of 600m total, Pionjar anomaly. Largely on freehold
33-2005	Kerr	33	H3995/7065	10	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	38	H3936/6955	9	Drilled with resources (1995)
39-2005	Kearney North	39	H4002/7202	8	Drilled no resources - low Au but high grade boulders locally and just downstream
40-2005	Sammy's	40	H4036/7138	8	Drilled no resources - 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.
Apr-03	Cornavarrow Burn East Showing	4	H3498/6942	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 60m E upstream. 11 km NE trend linear feature discordant to strike 100m to SE.
25-2005	Commings Bog	25	H410/720	5	Apparently non-cultural, 2 line, 100m NNE strike VTEM anomaly with < 23 g/t Au in soil. May be due to massive sulphide related gold mineralisation below bog.
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 Riofinex and Omagh 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

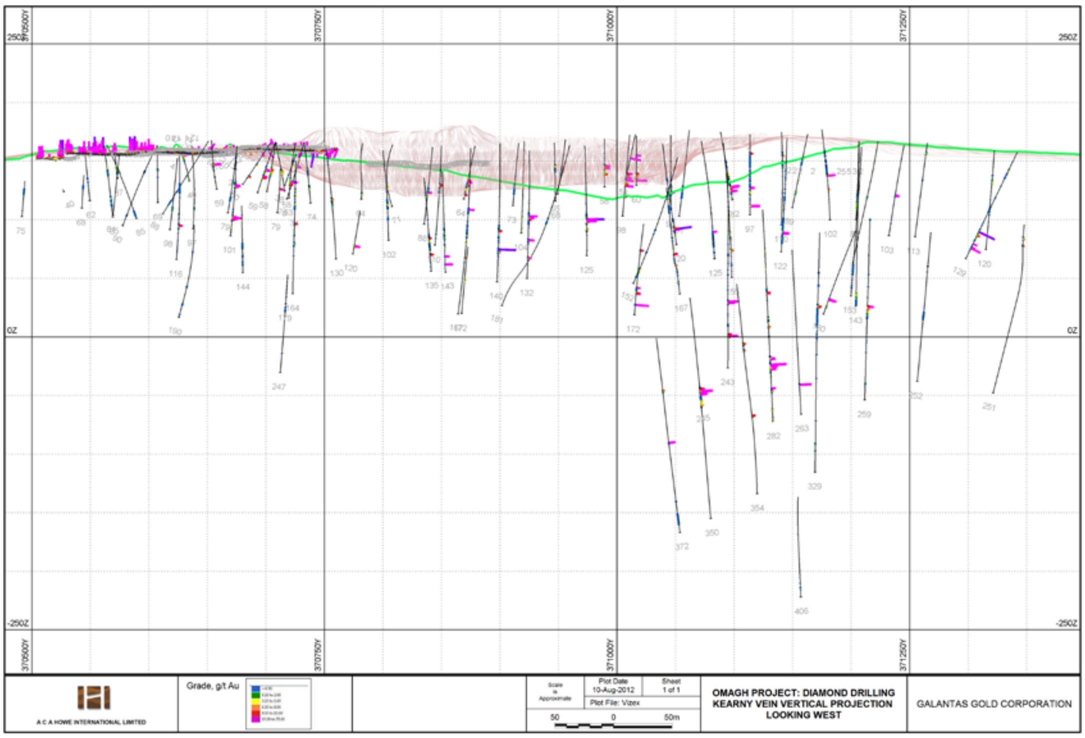
					local structures related to NE trending Landsat linear features but could also be dispersion from the Kearney-Joshua etc float gold cluster. Weak but potentially significant VTEM anomalies on 2 lines in area 200 x 100 elongated WNW? 3 lines with weak anomalies similar to North of Sammy's Barn, max 1 km along flight line.
26-2005	Legphressy	26	H346/703	4	3 line VTEM anomaly with N trend Landsat linear feature 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
Jan-03	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.
Feb-03	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.
Mar-03	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.
Nov-03	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 features. Prospecting results in stream bed were disappointing but bedrock source of local Pionjar gold anomaly may lie to NW covered by peat
Dec-03	BM 210.2 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 features provide focus for float and outcrop prospecting. Access on land between the two burns may be problematical due to forestry established since 1981fires.
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	Unshinagh	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.

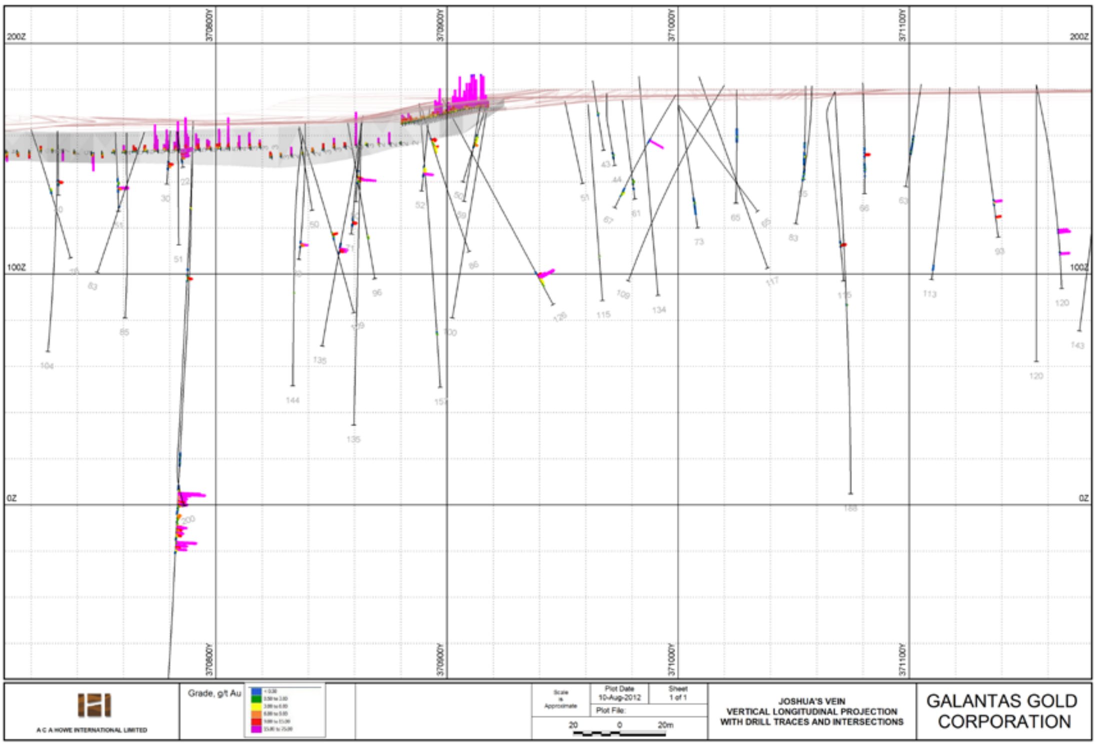
May-03	Cornavarrow Burn West Showing.	5	H3446/6920	2	Riofinex gold bearing showing relocated but resampling reported no gold or silver. Line of IP in Feb 1988. SE extension of showing possibly indicated by minor resistivity anomaly on IP line. Major IP anomaly due either to a NE dipping dolerite dyke (not apparently known to Riofinex) or to graphitic metasediments. Mapped Tertiary dyke type magnetic anomaly.
Jun-03	Cornavarrow Burn below The Small Point confluence	6	H353/700	2	Stream sediment gold is probably locally derived from structures related to the local, NE trending Landsat linear feature. Local magnetic high.
13-2003	Greenan Burn Lower	13	H326/674	2	2005 magnetic low. Associated with stream sediment Au, As and Pb anomalies and gold in float.
17-2003	Glenarn and Stranahone on Glendurragh River	17	H282/674	2	Source of weak anomalies may be structures related to two N trending Landsat linear features. Local magnetic high.
18-2003	Unnamed stream = Stranahone north tributary of the Glendurragh River	18	H2779/6824	2	Source of single point, weak stream sediment Au anomaly may be local structures related to N trending Landsat linear feature.
21-2003	Tattysallagh (Barrett's Glen)	21	H379/688	2	Source of gold anomalous stream sediment and float samples may be local structures related to N trending Landsat linear feature . Weak VTEM 4-500m NNE on two lines at H380692. Local magnetic low with mapped Tertiary dyke.
23-2003	Pollnalaght (AKA Pigeon Top)	23	H370/710	2	Local Pionjar gold anomalies and distant, radially distributed, gold anomalous stream sediments and float samples and the intersection of 5 Landsat linear features 700 metres northwest of the Pionjar anomaly, may be associated with intersecting fault brecciation and undiscovered gold mineralisation. W end of a VTEM conductivity high 4.5 km ENE x 0.5 km wide probably due to black schist seen at Target "63 gram".
24-2003	Lower Creevan Burn, Eastern end of the licence	24	H416/705	2	Panned gold colours, stream sediments exceeding 1,000 ppb, highly anomalous As and Pb, gold anomalous float. Pionjar Au at H413702 followed up by Riofinex. Check for Riofinex IP coverage.
29-2005	Cavanacaw Magnetic Low	29	H3946/7110	2	Possible demagnetisation anomaly or unmapped, conformable, Tertiary dyke fault intrusion within the Dalradian. See closely associated Target 50-2005 - Cousins
30-2005	Greenan Burn Lower (magnetic low)	30	H326/682	2	Local magnetic low possibly related to gold mineralisation in float, see Target 13-2003, Greenan Burn Lower
53-2005	North Crockard	53	H3900/7250	2	Single point Pionjar anomaly (Riofinex No. 56) and 3 IP lines with no anomalies or not surveyed
Jul-03	Cornavarrow Burn H33986923	7	H3398/6923	1	Train of 3 stream sediment Au anomalies. No focus or obvious vector to source. Could be derived from Cornavarrow Burn West Showing or similar in footwall zone of dolerite dyke.

Aug-03	Cornavarrow Burn H33656948	8	H3365/6948	1	Minor stream sediment gold anomaly with no obvious vector to source.
Sep-03	Cornavarrow Burn H33506985	9	H3350/6985	1	Minor stream sediment gold anomaly with no obvious vector to source.
Oct-03	Dooish Mountain East, east flowing tributary of Cornavarrow Burn	10	H3310/6977	1	Weak gold stream sediment anomaly may be derived from local mineralisation or boulder clay. N and E trend Landsat linear features within 100m.
16-2003	Carrickagreany east of Cloy	16	H268/648	1	Two isolated gold anomalous stream sediment samples in same stream system. Gold may be associated with faulted Dalradian amphibolites.
27-2005	VTEM anomaly 1.7km NW of Lack	27	H260/673	1	Single line VTEM anomaly over Chadian Claragh Sandstone Formation
47-2005	West Garry	47	H3857/6935	1	Pionjar and IP anomalies on 2 lines of 5 over 100x30m

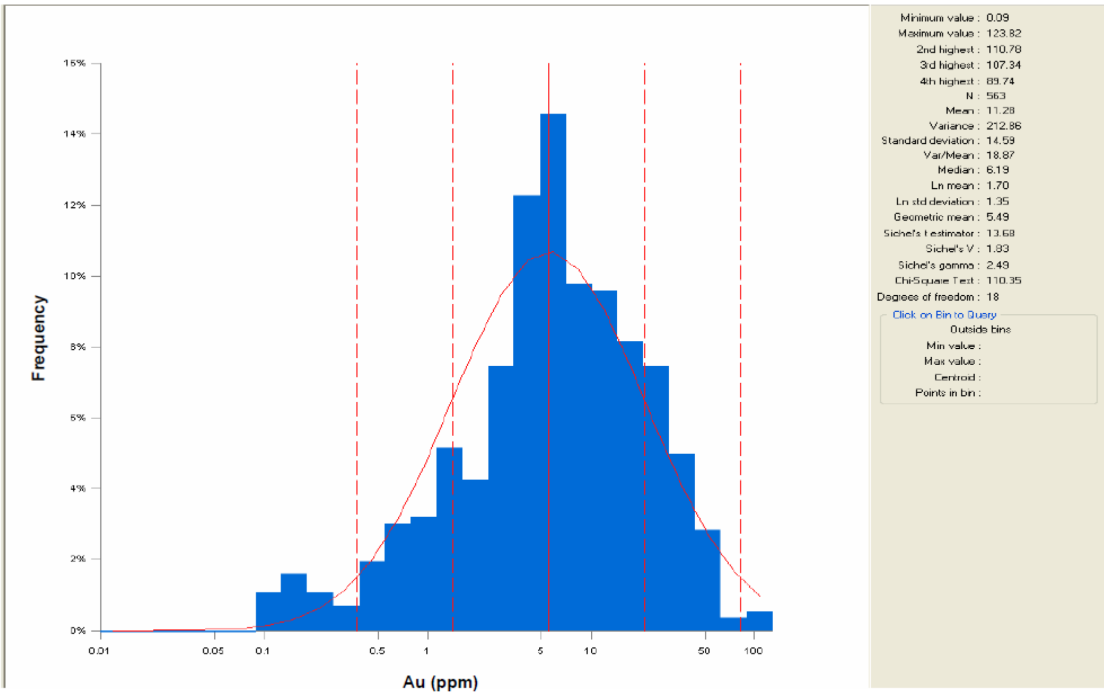
APPENDIX II
SECTIONS OF VEINS SHOWING DRILL TRACES AND INTERSECTIONS





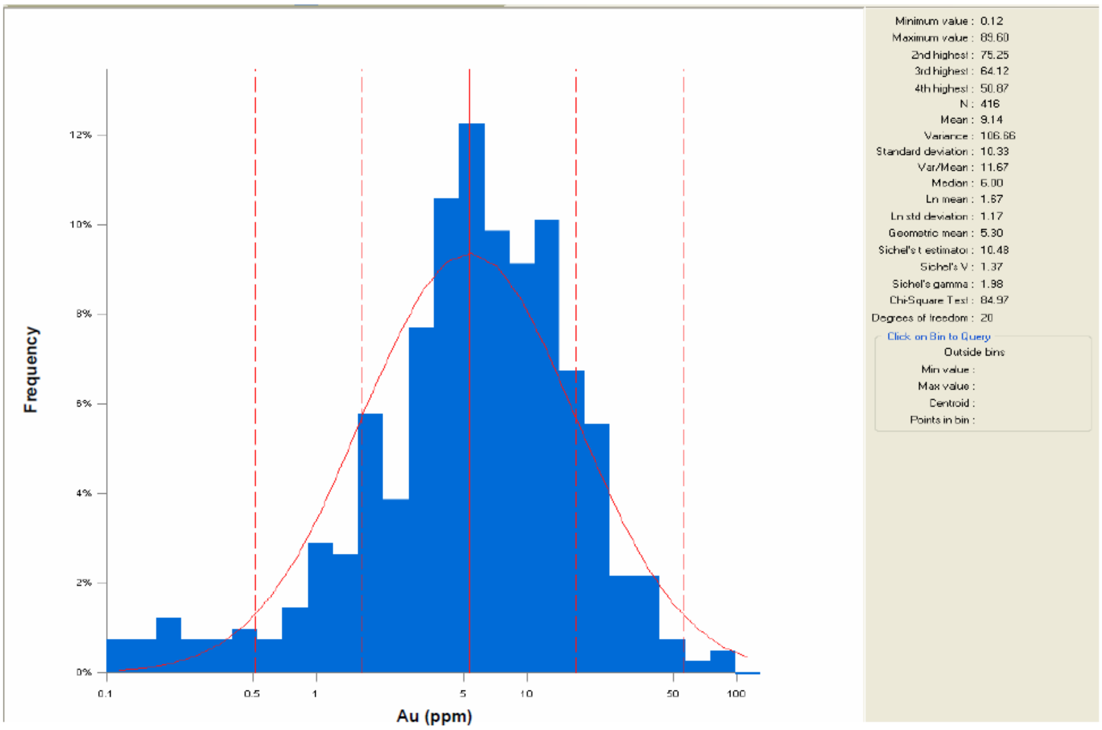

200Z
100Z
0Z
370800Y
370900Y
371000Y
371100Y
A C A HOWE INTERNATIONAL LIMITED
Grade, g/t Au
Scale is Approximate
Plot Date
10-Aug-2012
Sheet
1 of 1
Plot File:
20
0
20m
JOSHUA'S VEIN
VERTICAL LONGITUDINAL PROJECTION
WITH DRILL TRACES AND INTERSECTIONS
GALANTAS GOLD CORPORATION

APPENDIX III
DOMAIN HISTOGRAMS
(from the ACA Howe 2008 report)



A C A Howe International Limited

Figure : Kearney 1 Domain – Raw Assay Log Normal Histogram and Descriptive Statistics



A C A Howe International Limited

Figure : Kearney 3 Domain – Raw Assay Log Normal Histogram and Descriptive Statistics

APPENDIX IV
DETAILS OF TOP-CUT ASSAYS

DETAILS OF ASSAY VALUES TOP CUT TO 75 g/t/T Au						
Hole ID	**WIREFRAME**	**From (m)**	**To (m)**	**Interval**	**Au (ppm)**	**Au<75**
OMBHL59	PRINCES_1A	40.35	40.40	0.05	626.000	75.00
OMBHL59	PRINCES_1A	40.00	40.35	0.35	209.000	75.00
OM-DD-07-36	KEARNEY_5A_2012	108.35	108.45	0.10	165.120	75.00
OM-CH-11/JA-36	JOSHUA_1A	0.30	0.40	0.10	125.440	75.00
OMTRL312	KEARNEY_1_2012	2.00	3.00	1.00	123.820	75.00
OMTRL312	KEARNEY_1_2012	3.00	3.05	0.05	123.820	75.00
OM-CH-11/JA-32	JOSHUA_1A	0.50	0.60	0.10	122.120	75.00
OMTRL346	KEARNEY_1_2012	3.80	4.80	1.00	110.780	75.00
OMTRL346	KEARNEY_1_2012	4.80	5.00	0.20	110.780	75.00
OM-CH-11/JA-33	JOSHUA_1A	0.60	0.70	0.10	109.780	75.00
OMTRL319	KEARNEY_1_2012	2.45	3.35	0.90	107.340	75.00
Line23	KEARNEY SOUTH 2012	1.59	1.87	0.28	106.240	75.00
OM-DD-07-16	KEARNEY_6A	102.00	102.20	0.20	103.940	75.00
OM-DD-07-29	ELKINS_5A	53.95	54.10	0.15	101.760	75.00
OMBHL59	PRINCES_1A	40.40	40.80	0.40	100.000	75.00
OMBHL39	KEARNEY_17A	95.89	96.19	0.30	93.910	75.00
OMTRL342	KEARNEY_1_2012	5.35	6.35	1.00	89.740	75.00
OMTRL342	KEARNEY_1_2012	5.35	6.35	1.00	89.740	75.00
OM-DD-07-40	KEARNEY_4A	85.20	85.45	0.25	89.600	75.00
Line23	KEARNEY SOUTH 2012	1.87	2.63	0.76	87.680	75.00
Line21	KEARNEY SOUTH 2012	3.14	3.44	0.30	86.400	75.00
OMBHL95	SAMMYS_1A	12.35	12.48	0.13	83.770	75.00
T502		0.70	0.90	0.20	75.250	75.00

APPENDIX V
SEMI-VARIOGRAMS
(from the ACA Howe 2008 report)

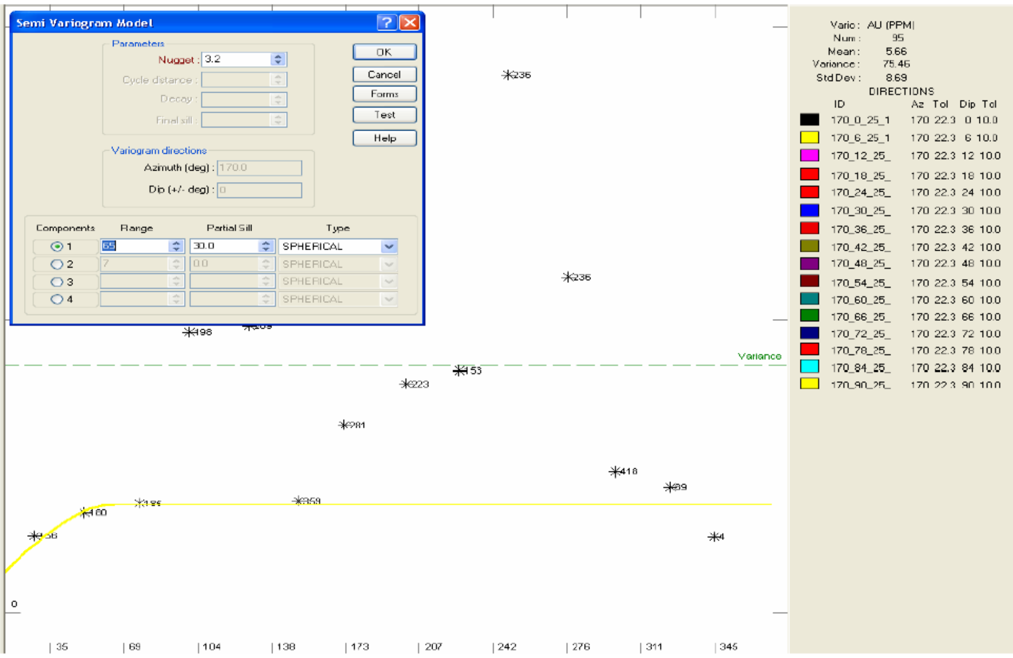


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Figure : Domain K3 Directional semi-variogram (1st Direction, azi 350, plunge = 0º) RANGE = 65m.

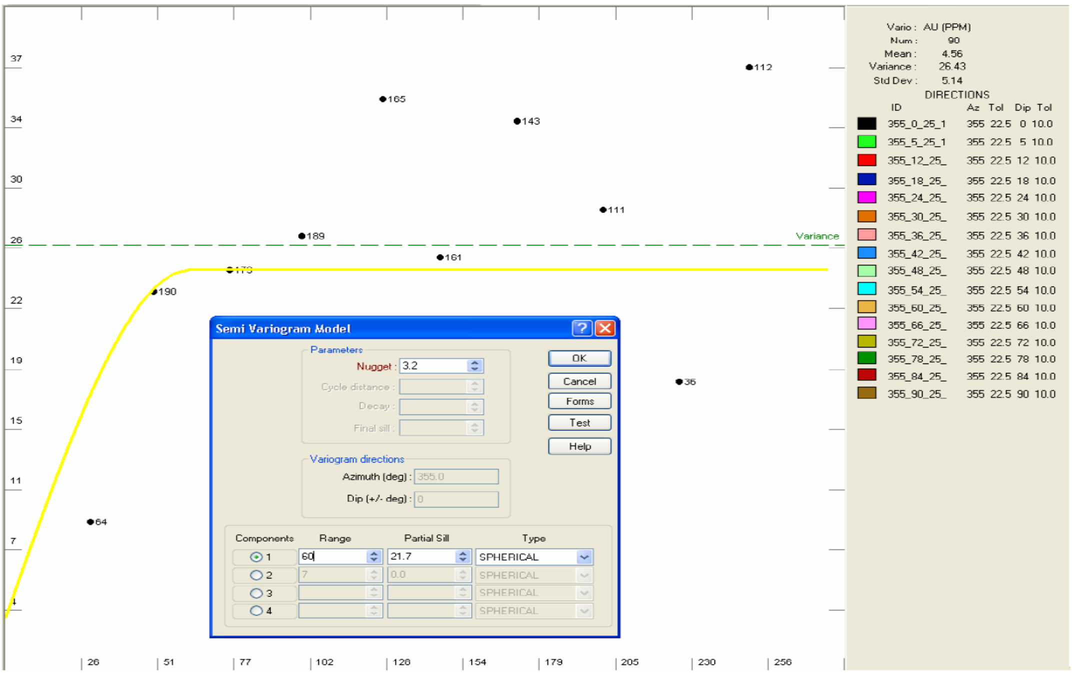

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Figure : Domain K4 Directional semi-variogram (1st Direction, azi 355, plunge = 0º) RANGE = 60m.

APPENDIX VI

MINERALISATION DENSITY DATA

(from McFarlane,J., and Eves, J., 2008)

2008 density determination

Rock Sample	Sample A Quartz & Sulphide Primary Stage	Sample B Quartz & Sulphide Second Stage	Sample C Sulphide Clay Gauge Dark	Sample D Sulphide Clay Gauge Light	Sample E Altered Host with Sulphide
Criteria	White quartz with well formed S mineralisation, difficient in Gn	Brecciated Qtz with fine S mineralisation often along fracture surfaces, Gn rich	S rich clay gouge dark grey to black in colour	Chloritic/micaceous clay gouge, difficient in sulphides often light green in colour	bleached +/or silicified psammite/schist at margins +/or between veins, contain thin stringer veins +/or disseminated sulphides
2008 SAMPLING	4.251	2.654	2.780	2.713	2.703
	3.374	2.743	2.774	2.731	2.692
	2.725	2.668	2.771	2.756	3.189
	3.357	2.760	2.804	2.765	2.672
	3.312	2.732	2.823	2.832	2.730
	4.522	2.843	2.894	2.768	2.692
	4.747	2.949	3.015	2.720	2.739
	3.592	2.736	2.784	2.751	2.764
	3.846	2.709	2.846	2.748	2.821
	2.977	2.681	2.837	2.808	2.732
	2.823	2.720	2.825	2.645	2.803
	4.193	2.768	2.832	2.902	3.009
	3.682	2.699	2.776	2.743	2.743
	3.634	2.694	2.758	2.772	2.716
	2.803	2.692	2.797	2.777	2.716
	4.105	2.778	2.770	2.744	2.735
	3.644	2.756	2.748	2.797	2.746
	2.882	2.856	2.833	2.771	2.728
	4.298	2.667	2.790	2.801	2.705
	4.243	2.726	2.778	2.751	2.735
	3.399	2.665	2.782	2.633	2.735
	3.680	2.678	2.810	2.811	2.748
	2.700	2.667	2.792	2.840	2.743
	4.388	2.705	2.784	2.753	2.888
	3.733	3.024	2.943	2.837	2.696
AVERAGE VALUE	**3.636**	**2.743**	**2.814**	**2.767**	**2.767**

collated types by flagging geology codes that fall within the Kearney wireframes

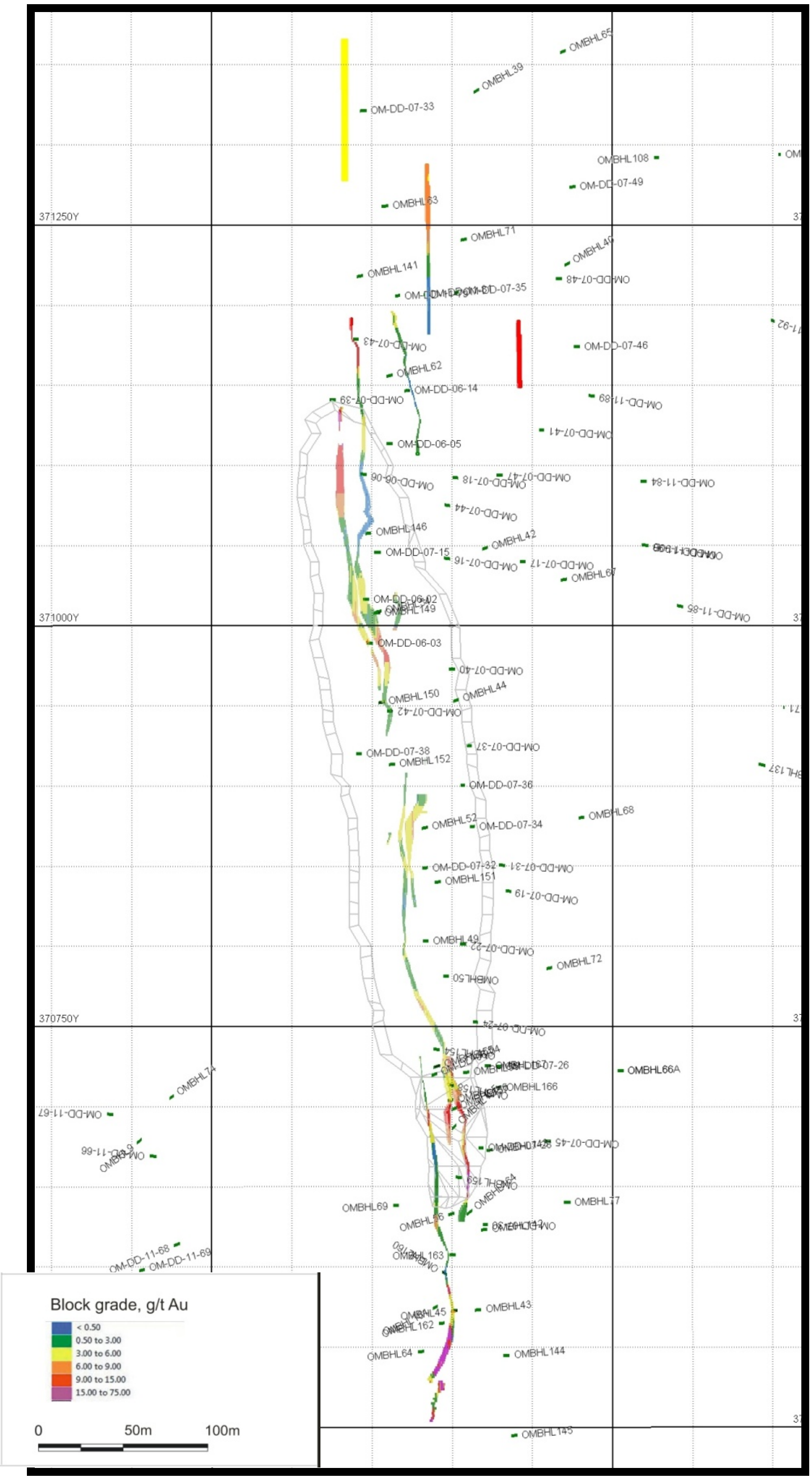
grouped codes	number of records	Ore Designation	Applied Density
Qtz	22	A/B	3.190
VN+PY	13	A/B	3.190
Qtzcarbsx	12	B	2.743
QtzCG	11	A/B+C	3.002
Qtzsx	20	A/B	3.190
FSc	4	E	2.767
VN+PY+GA+/-AP+CP	8	A/B	3.190
CG (DARK)	8	C	2.814
CG (LIGHT)	5	D	2.767
AltFScsx	6	E	2.767
sFPssx	14	E	2.767
QtzSc	2	E	2.767
AltFScQtzsx	10	E	2.767
CG+QV+PY	1	A/B+C	3.002

total number of samples	number of samples	weighted value	applied density	weighted density
136	22	0.162	3.190	0.516
136	13	0.096	3.190	0.305
136	12	0.088	2.743	0.242
136	11	0.081	3.002	0.243
136	20	0.147	3.190	0.459
136	4	0.029	2.767	0.081
136	8	0.059	3.190	0.188
136	8	0.059	2.814	0.166
136	5	0.037	2.767	0.102
136	6	0.044	2.767	0.122
136	14	0.103	2.767	0.285
136	2	0.015	2.767	0.041
136	10	0.074	2.767	0.203
136	1	0.007	3.002	0.022
			Weighted Average Density	**2.984**

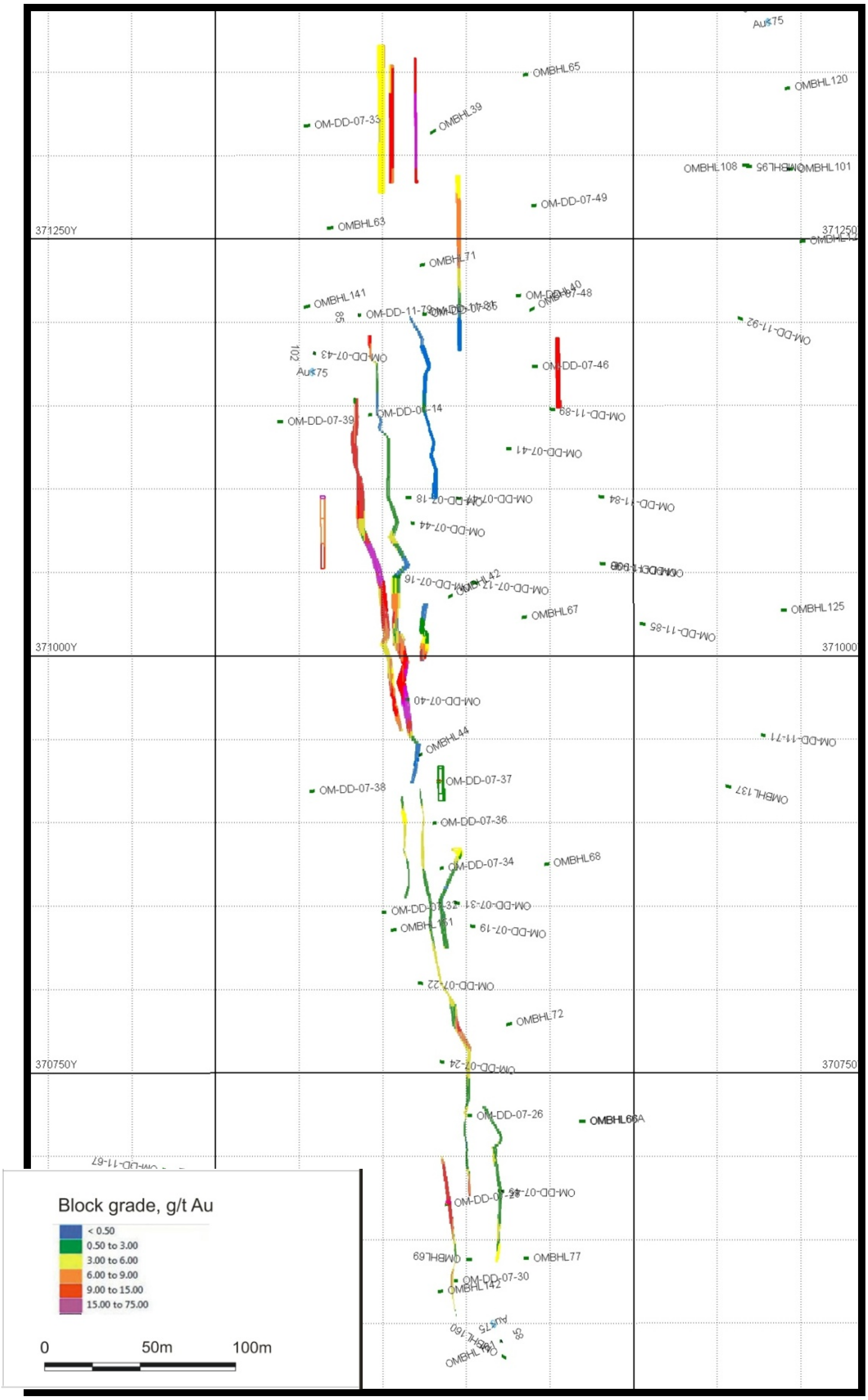
VALIDATION INFORMATION

Plastic Ball		
Diameter	39.04	mm
Radius	0.01952	m
Volume	3.1155E-05	m3
Mass	67.32521	g
Mass in kg	0.06732521	kg
Density by Diameter	2160.974626	kg/m3
Suspended	36.24	
Density by Lab	2165.827736	kg/m3
% Error	0.22%	

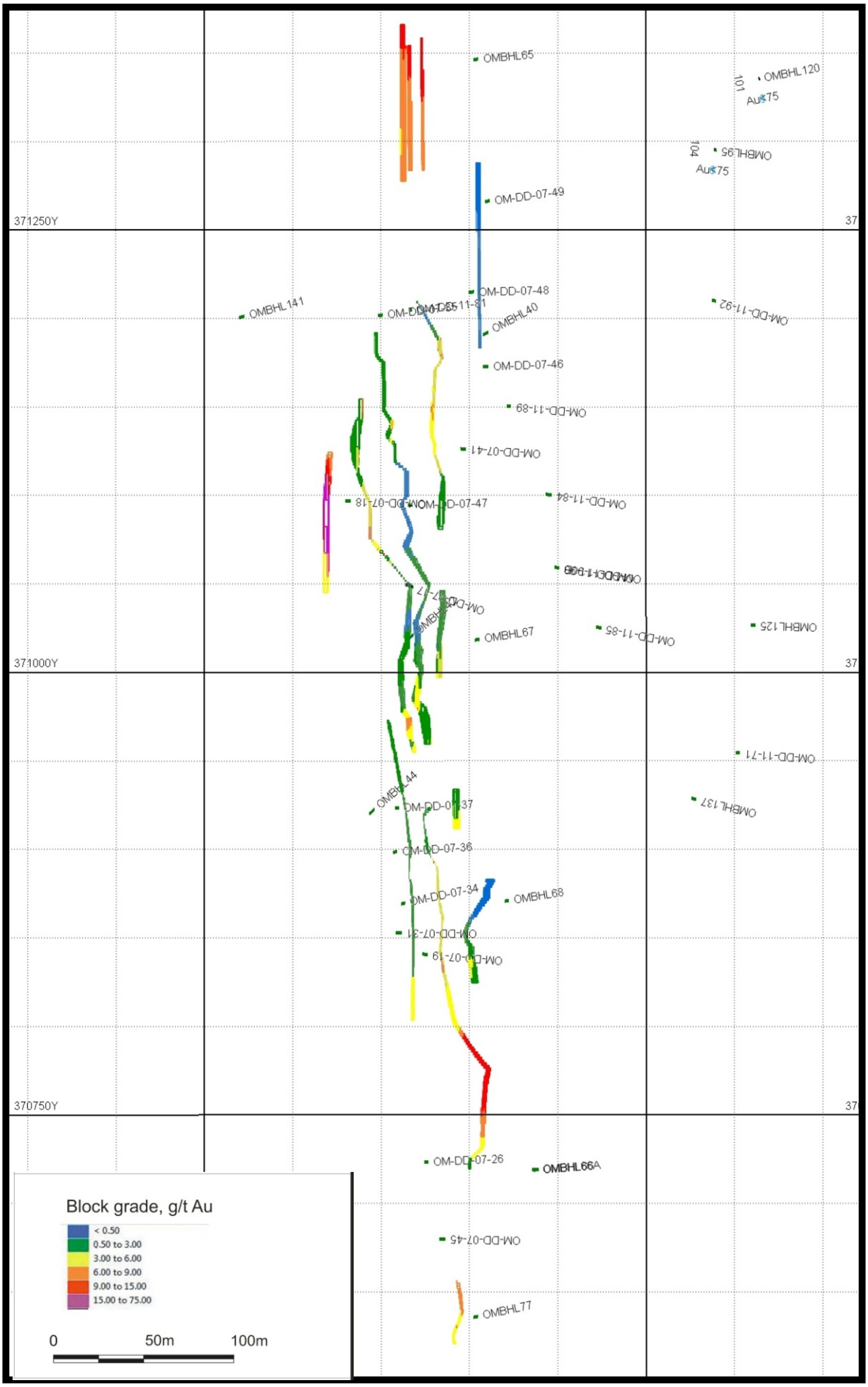
APPENDIX VII
HORIZONTAL SECTIONS OF BLOCK MODELS COLOURED BY GRADE
(all resource categories)



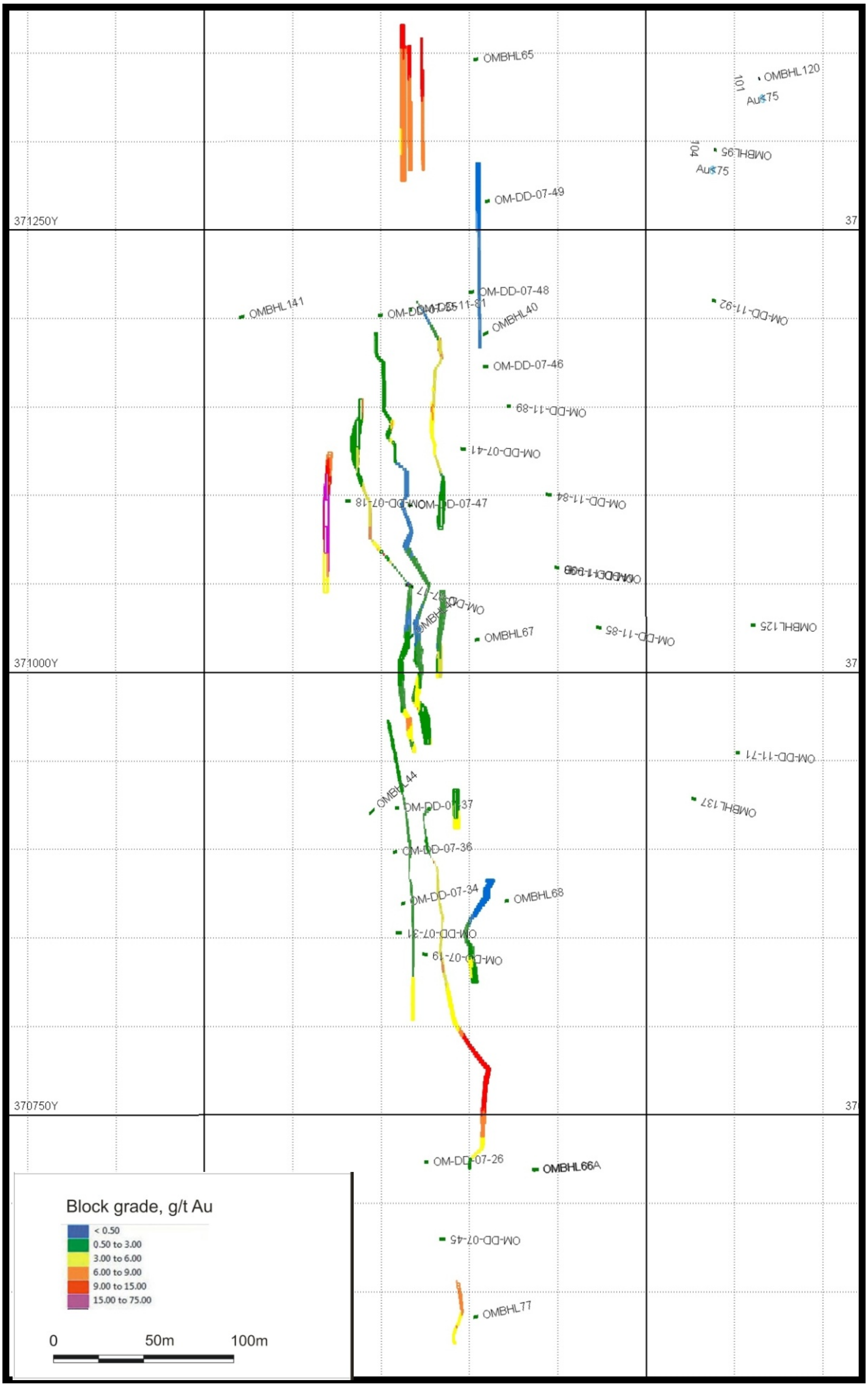
Kearney Vein +140m



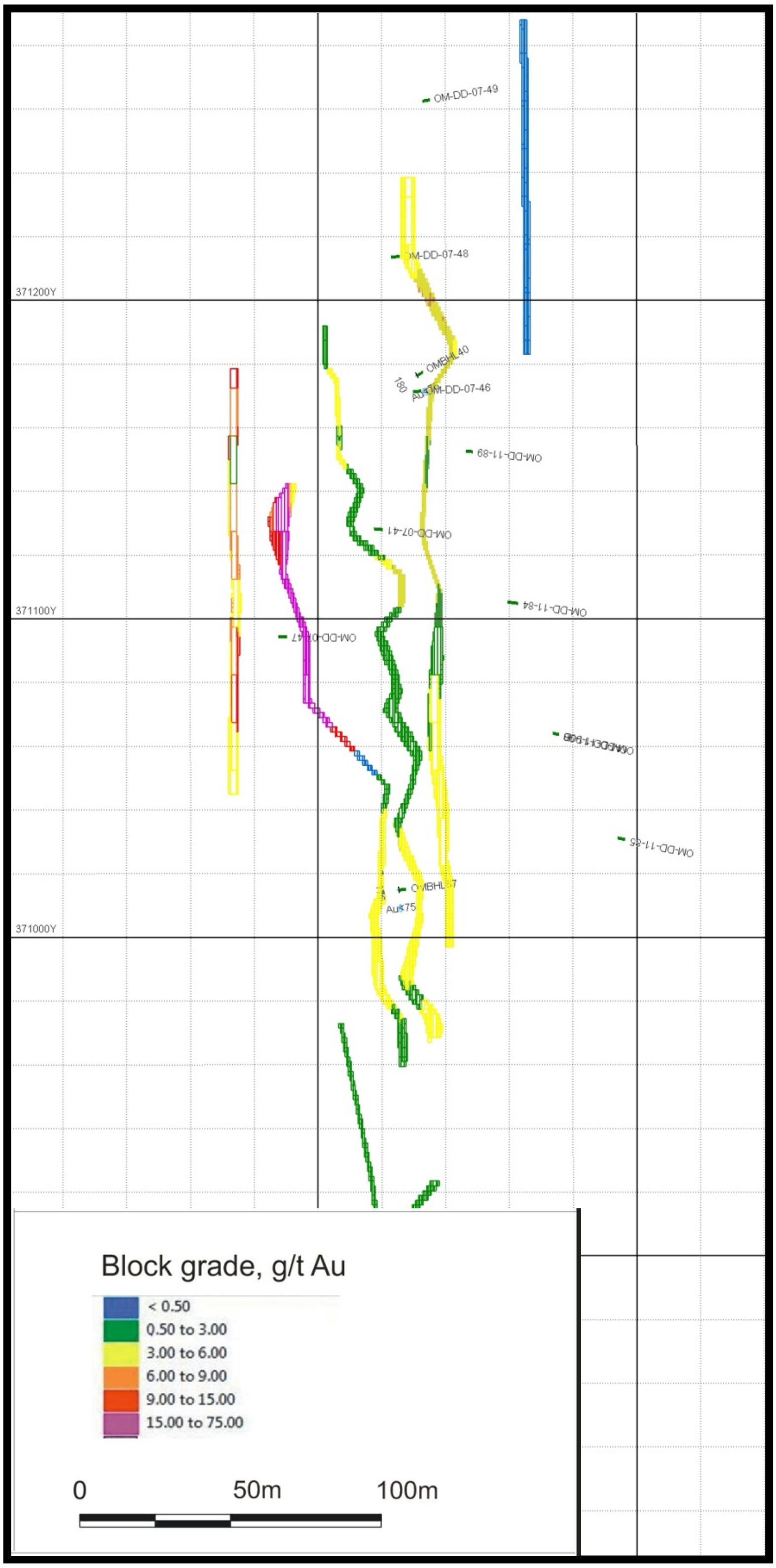
Kearney Vein +100m



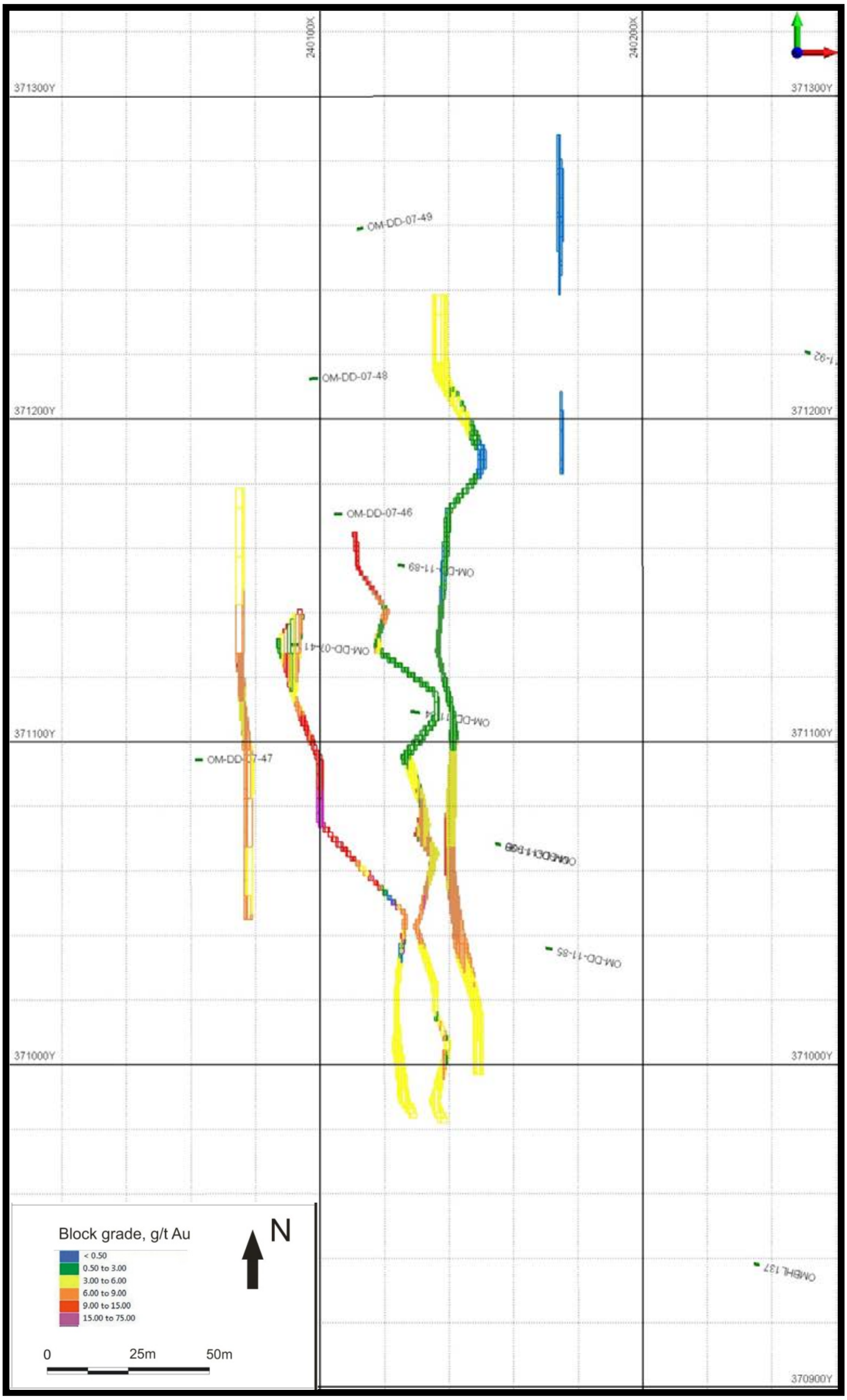
Kearney Vein +60m



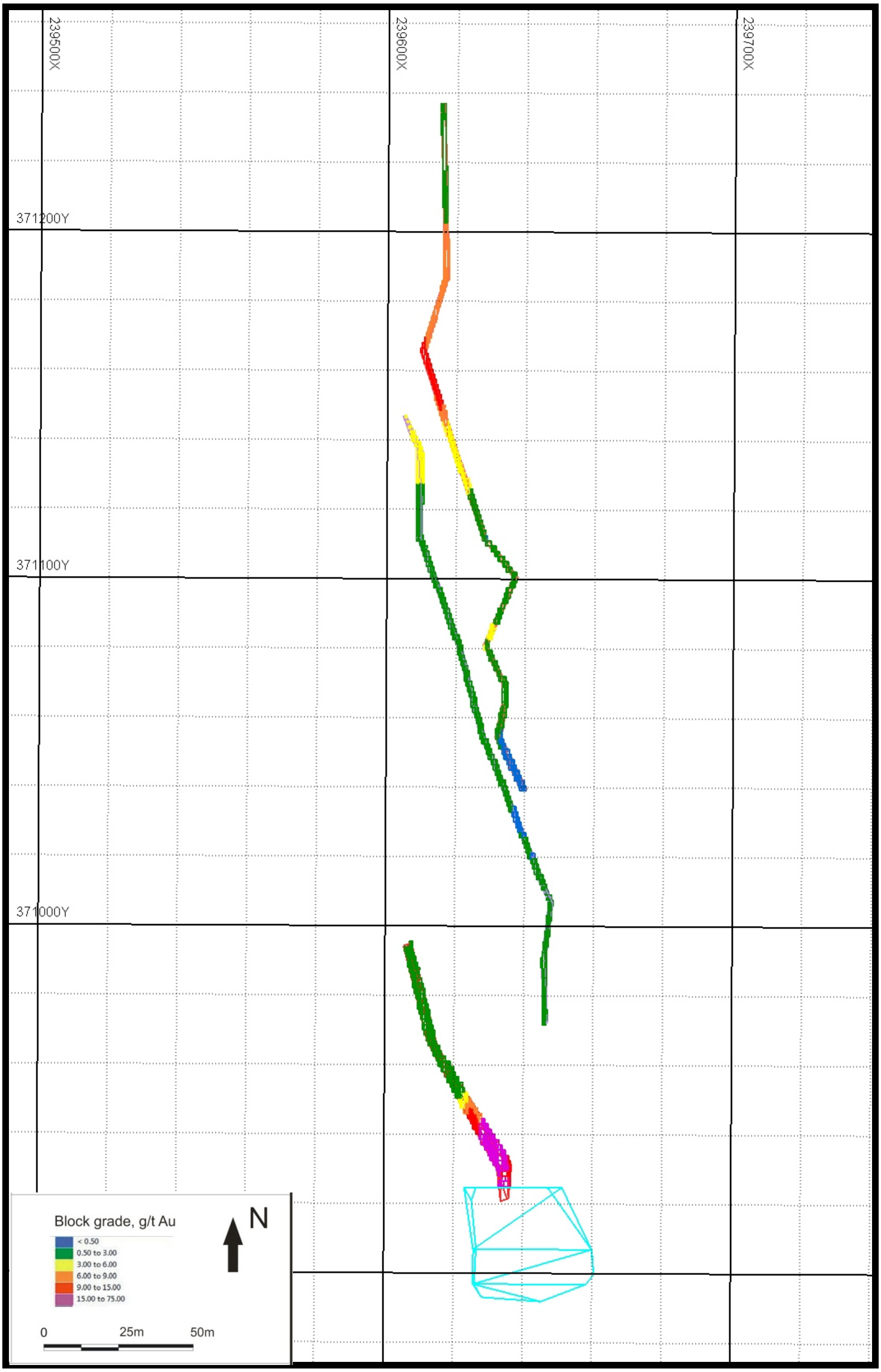
Kearney Vein +60m



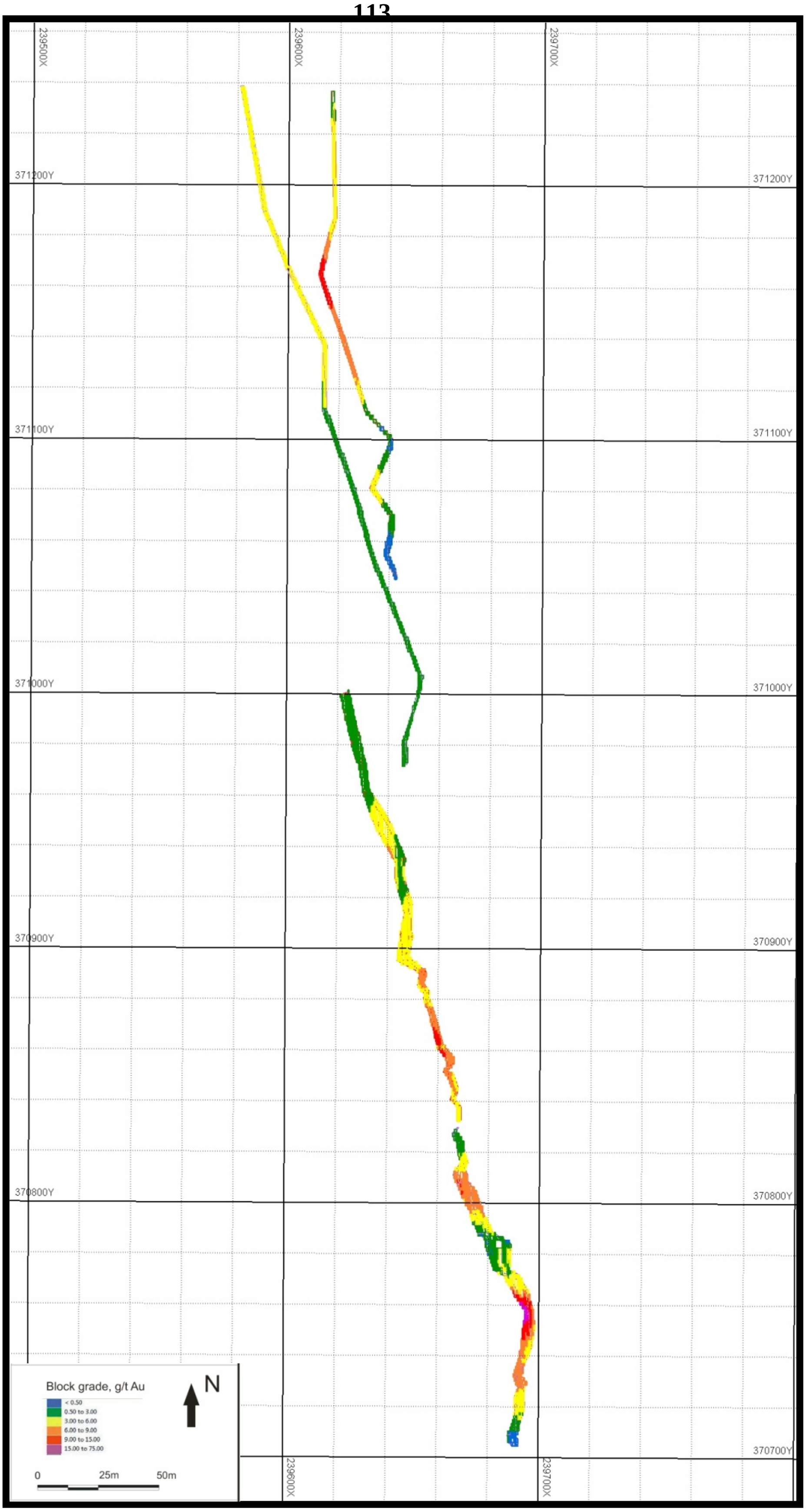
Kearney Vein +20m



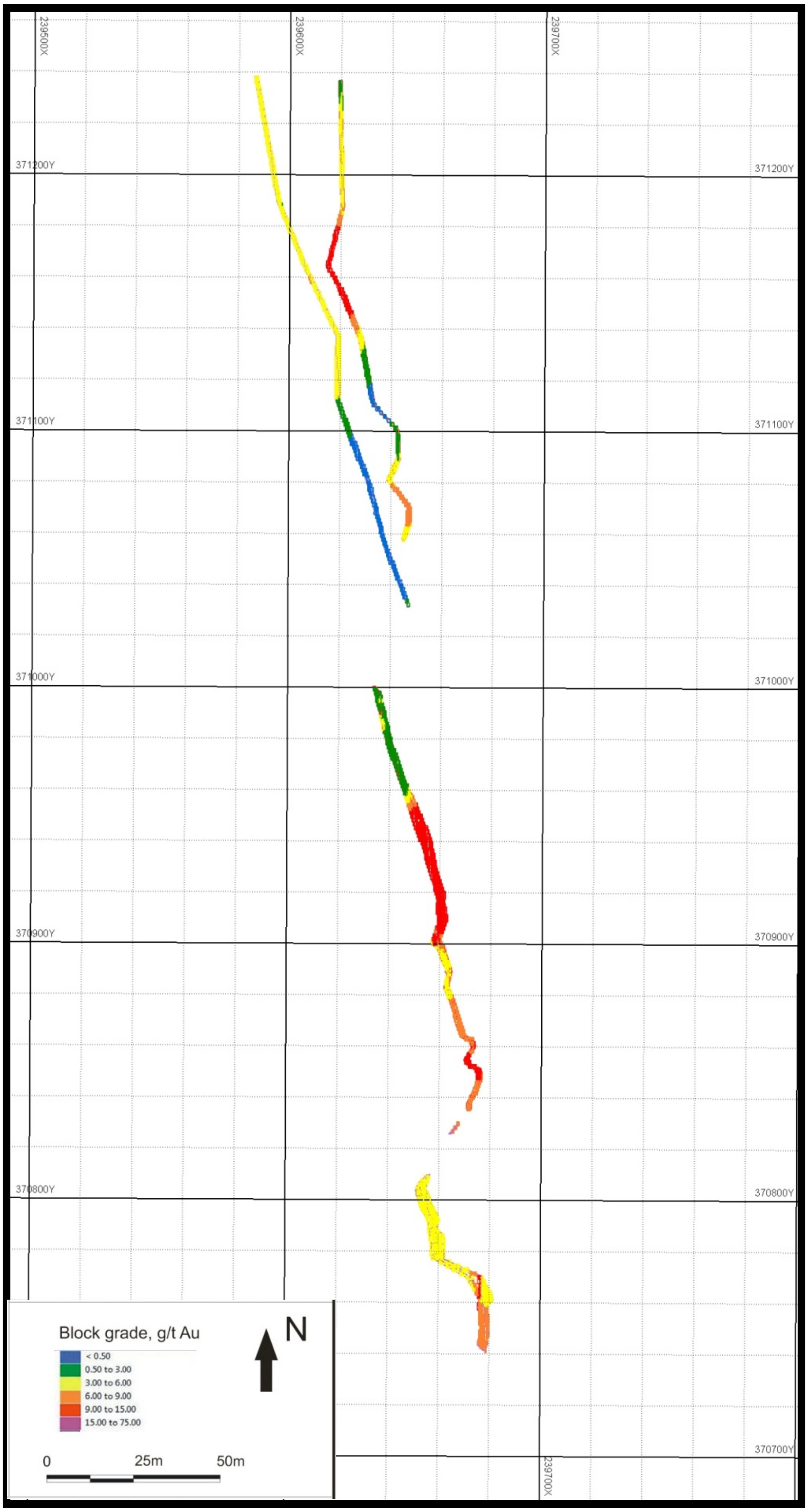
Kearney Vein -20m


239500X
239600X
239700X
371200Y
371100Y
371000Y
Block grade, g/t Au
< 0.50
0.50 to 3.00
3.00 to 6.00
6.00 to 9.00
9.00 to 15.00
15.00 to 75.00
N
0
25m
50m

Joshua +175m


239500X
239600X
239700X
371200Y
371100Y
371000Y
370900Y
370800Y
370700Y
Block grade, g/t Au
< 0.50
0.50 to 3.00
3.00 to 6.00
6.00 to 9.00
9.00 to 15.00
15.00 to 75.00
N
0
25m
50m

Joshua +140m


239500X
239600X
239700X
371200Y
371100Y
371000Y
370900Y
370800Y
370700Y
Block grade, g/t Au
< 0.50
0.50 to 3.00
3.00 to 6.00
6.00 to 9.00
9.00 to 15.00
15.00 to 75.00
N
0
25m
50m

Joshua +105m