National Instrument 43-101 Technical Report

KOPSA GOLD-COPPER DEPOSIT,

CENTRAL OSTROBOTHNIA, FINLAND.

<u>Prepared for:</u> **Belvedere Resources Ltd**
Suite 404, 999 Canada Place
Vancouver, British Columbia V6C 3E2

October 29th 2012

Authors:

David Pym, B.Sc. – Belvedere Resources Ltd
Toby Strauss, Ph.D (CGeol; EurGeol.) – Belvedere Resources Ltd
Markku Meriläinen, MSc – Outotec (Finland)
Pekka Lovén, MSc (MAusIMM (CP)) – Outotec (Finland) "QUALIFIED PERSON"

Table of Contents

Table of Figures

1

1 Summary

This Technical Report covers the Kopsa gold-copper deposit in Northern Ostrobothnia in western Finland. Belvedere Resources Ltd holds a 100% interest in the Kopsa project through its wholly owned subsidiary Belvedere Mining Oy, which also owns the nearby Hitura nickel mine. The Kopsa property is located at Latitude 63.77086°N, Longitude 25.23393°E, in the Haapajärvi community, western Finland.

The Kopsa Au-Cu deposit occurs within the Proterozoic aged late orogenic Kopsa Tonalite, a roughly rhomb shaped 1200 x 550 m intrusive, intruding a 1.92 Ga package of meta-greywacke and mica schists and intermediate pyroclastic volcanic rocks, interpreted as turbidite sequences. The whole sequence has been metamorphosed to upper-greenschist to amphibolite facies.

Ore minerals mainly occur as compact sulphide veins or as stringers and blebs in connection with quartz veining and silicification. Fine grained disseminated ore minerals occur outside the veins in the altered host rocks. In the higher grade areas of the deposit, the quartz veins and silicification form more of a stockwork, rather than discrete vein sets. The main sulphide minerals are arsenopyrite, chalcopyrite, pyrrhotite and minor pyrite. Gold is free milling (non refractory) with typical grains sizes of 10µm, and has a close association with bismuth.

The mineralisation has so far been defined over an area of approximately 700 m by 200 m with a maximum thickness of about 50 metres. The mineralisation strikes towards 105°, and dips roughly 20° to the SSW. In the north, the mineralisation comes to bedrock surface, part of which outcrops at the “Kopsa Outcrop”. The maximum depth for the model is about 125 metres vertical from surface. The average drill spacing is 17.8 m, with a maximum spacing of 47.9 m.

Kopsa has been variably classified throughout its history. Alteration and sulphide associations suggest fluids with at least some magmatic input, and the host intrusion also cuts the main regional d2 foliation associated with peak deformation suggesting mineralisation post-dates orogenesis. Belvedere geologists have tentatively classified the deposit as belonging to the intrusive related style of deposits.

The Qualified Person has determined that the existing data can support the following Mineral Resource Estimate (according to the JORC code) based on a cut-off grade of 0.4 g/t Au. The gold equivalence is based on a $1,200 /oz Au price and a $6,000 /tonne Cu price:

29th October 2012 Mineral Resource Estimate (0.4 g/t Au cutoff)						
Resource Category	**Tonnes**	**Au g/t**	**Cu ppm**	**As ppm**	**Au Ounces**	**Au_equiv. g/t**
Indicated	6,680,000	1.04	1526	4886	223,000	1.28
Inferred	1,800,000	0.76	1761	5191	43,000	1.03

The mineralisation was modelled as 10 separate high grade domains and one low grade domain. The mineral resource was calculated using block modelling to a maximum vertical depth of approximately 125 metres (Z=-15), made up of 10m x 10m x 5m parent blocks (with sub-blocks of 10m x 5m x 5m) constrained by the modelled wireframes. The block model was rotated to an azimuth of 105° to better fit the geometry of the body.

Block grades were estimated using a cigar shaped search ellipse and the inverse distance squared method. The major axis of the ellipse was horizontal towards 105° with a maximum range of 100m, and 25 m across strike. Estimates were based on 1m best-fit composites, which have been top-cut to 20g/t in the high grade domains and 7 g/t in the low grade domain. Blocks were estimated with a minimum of 5 and a maximum of 30 samples.

Bulk density of the mineralisation was based on actual specific gravity data collected during exploration. A total of 1674 measurements were taken from within the modelled mineralised zone, giving an average bulk density of 2.73 tonnes/m^3.

The combination of the shallow depth of mineralisation, (over 70% of the Indicated gold ounces being within 50 metres of the surface), and the proximity (<20km) to Belvedere's Hitura Nickel Mine and its existing infrastructure are both strong factors to be considered in the future development of Kopsa. These should be hugely beneficial to the economics of the project if developed.

It is recommended that on completion of the processing studies a full scoping study be completed to determine the optimum scale of the project, and the likely economics.

2 Introduction and Terms of Reference

This report was prepared at the request of Belvedere Resources Ltd, on the completion of the 2010 and 2011 diamond-drilling programme carried out on the Kopsa property. The objectives of the report were:

1. To assess the mineral resources at Kopsa and to classify them according to the “categories” compliant with NI 43-101.

2. To prepare the report as a National Instrument 43-101 Technical Report in accordance with Form 43-101F1. The instruction also stipulated that the report must be an “Independent Technical Report” as laid out under the definitions of “Independence” in section 1.4 of NI 43-101.

The sources of information for this report have been drawn largely from the results of the various drill programme (totalling 10,175.09 metres over 108 holes) carried out by Belvedere Resources since 2003. In addition, further historical drilling data (totalling 11,655.03 metres over 86 holes) by Glenmore Highlands Inc (a.k.a. Baltic Minerals) and Outokumpu has also been used. Copies of these files have either been supplied by Belvedere or are publicly available on various Finnish data repositories. References to the sources of information are provided throughout the report.

3 Reliance on Other Experts

This report was prepared as a National Instrument 43-101 Technical Report in accordance with Form 43-101F1 for Belvedere Resources Ltd.

The quality of information, conclusions and estimates contained herein is consistent with the level of accuracy as well as the circumstances and constraints under which the work was performed, data generated and provided by third party sources identified herein and, while it is believed that such information is reliable under the conditions and subjects to the limitations set forth herein.

This Technical Report has been prepared by the competent persons listed in the title page under the supervision of Pekka Lovén of Outotec Oy, acting as the “Qualified Person”.

Pekka Lovén, as Qualified Person, takes responsibility for the full contents of this report.

4 Property Description and Location

Belvedere Mining Oy holds a 100% interest in the Kopsa project. Belvedere Resources Limited holds a 100% interest in Belvedere Mining Oy, the latter being it's subsidiary.

The Kopsa property is located at Latitude 63.77086°N, Longitude 25.23393°E in the Haapajärvi community, Central Finland, 4 km NW from Haapajärvi, 120 km SW from Oulu and about 450 km NNE of Helsinki (Figure 1). There is a sealed road 2 km from the area and a gravel road to the area.

The property is covered by two Claims (Exploration Licenses), namely Kopsankangas, (Registry No. 7405/1) and Kopsankangas 2 (Registry No. 7686/1) with areas of 96.25 hectares and 96.78 hectares respectively (Figure 2). The validity period of the Kopsankangas claim (7405/1) has been extended to 7th May, 2010. The Kopsankangas 2 claim (7686/1) expired on 31st December, 2008. A three-year extension period for this claim was applied for on 11th November, 2008.

A Mining Lease application (K7405) was lodged on 11th March 2009. Consequently, both the claims (7405/1 and 7686/1) remain valid until the Mining Lease application has been processed. A 16 hectare extension to the Mining Lease Application was submitted on the 12th May, 2010.

A Claim Reservation (2010046) covering 663.6 hectares surrounding the mining lease application is valid until 10th May, 2011.

There are no known environmental liabilities on the Kopsa property. There are no royalties, back-in rights, payments or other encumbrances to which the property is subject. All the payments for damage compensations have been made up to date.

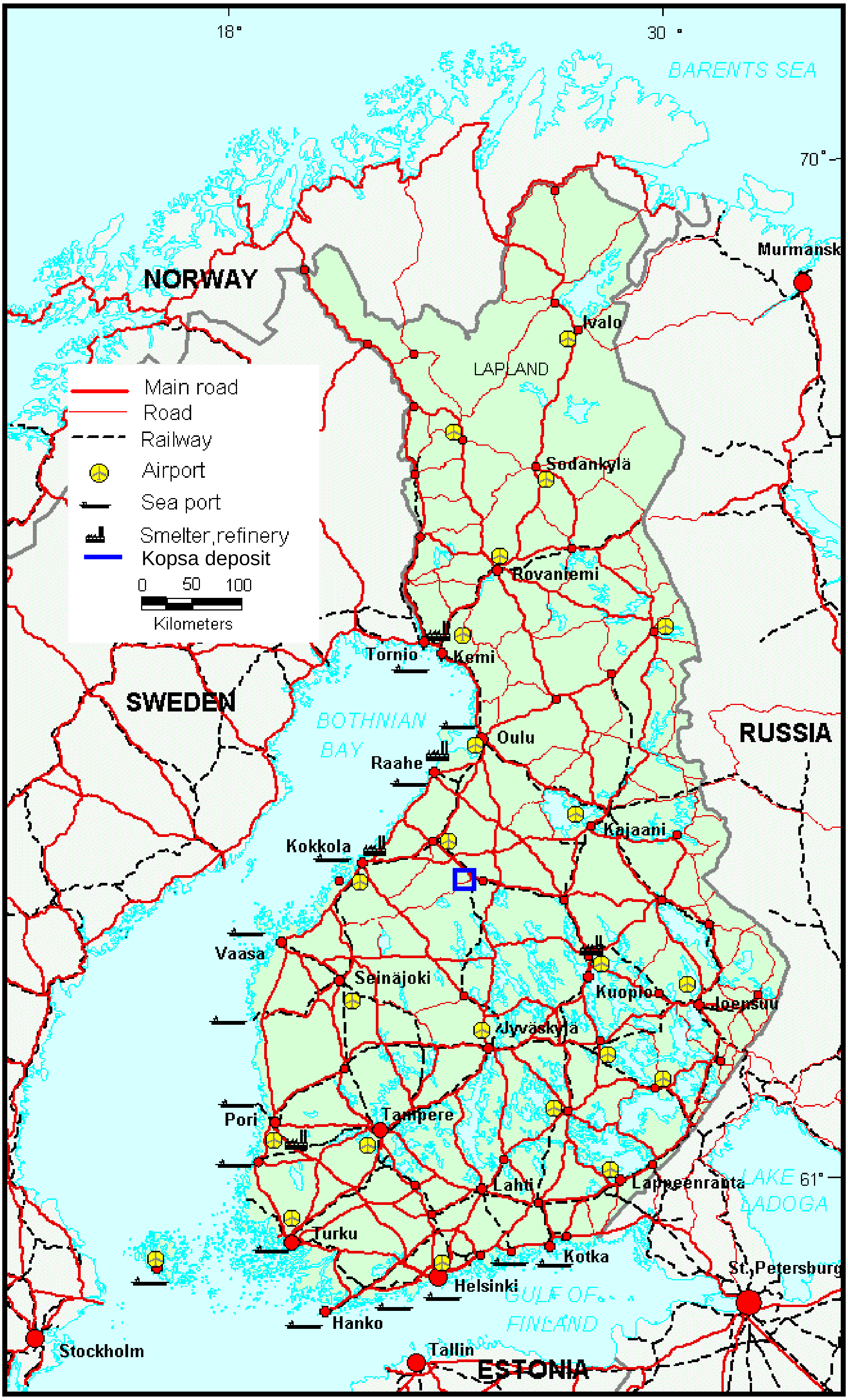

Figure 1 Infrastructure and location of Finland in relation to Belvedere's Kopsa Au-Cu deposit

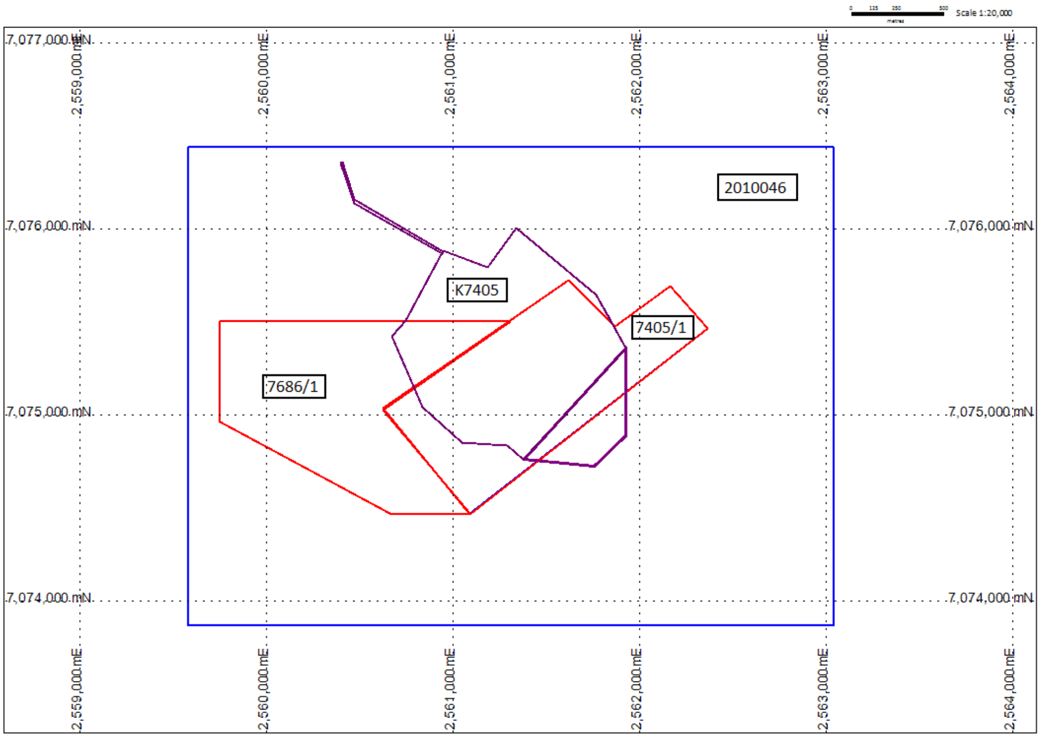

Figure 2 Land tenure of the Kopsa Property, showing Claims (red), Claim Reservations (blue) and Mining Lease Applications (purple)

The property is wholly owned by Belvedere and has no attached agreements or warrants with other parties. No expenditure requirements are attached to the property. The properties have not been legally surveyed, but boundaries are determined and finalised at time of application by the relevant Finnish government agency responsible for mining and exploration (currently the Finnish Safety and Chemical Agency (TUKES)). No environmental liabilities are extant apart from normal legal requirements for damage compensation to landholders resulting from any exploration works. No permits are required to perform exploration activities. Private individuals own the land and surface rights of the Kopsa claims areas.

The Qualified Person has examined the Claim Certificates, but has not reviewed the land ownership and has not independently verified the legal status or ownership of the Kopsa property, and is relying on the validity of mineral title claimed by Belvedere Resources Finland Oy.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Kopsa property is located 8 km by road from the town of Haapajärvi. The property can be accessed by 2 – 3 km of forestry dirt road, leading from the sealed Tiitonranta road off the Haapajärvi-Reisjärvi Highway no. 58 (FIG).

As is common with most of Finland the service infrastructure is excellent. The claim area is 8 km by road from the nearest railhead (at Haapajärvi) on the Nivala – Ylivieska railway line, which in turn is well connected to Oulu and/or Kokkola, two sea-port cities of Finland (Figure 3). The main railway line to Kokkola port (and Boliden's zinc smelter) runs through the town of Ylivieska. The nearest commercial airports are also at Kokkola (120 km by road to the west) and Oulu, (165 km by road to the north), with regular daily flights to Helsinki. Table 1 describes the relative location and importance of these localities.

Name of the locality	Distance	Significance
Oulu	165 km	Port, major town
Kokkola	120 km	Port, major town, smelter
Ylivieska	50 km	Major town
Hitura Nickel Mine	19 km	Belvedere's Hitura Ni mine
Haapajarvi	8 km	Community town, nearest railway service.

Table 1 Important localities and their significance around the Kopsa property

All the other infrastructural facilities are readily available in the area. A high voltage power line crosses the property to the south-east, about 1.0 – 1.5 km to the East of the main zone of mineralisation (Figure 4). Water is readily available from either the Kalajoki River (~2km) or Levälampi Lake (~1km). The Hitura nickel-cobalt mine and metallurgical plant (owned 100% by Belvedere) is about 13 km distance from the Kopsa deposit, provides a suitable site for processing the ore and for potential tailings storage areas, potential waste disposal areas, and possible heap leach pad areas. It is likely that the Kopsa ore will be hauled to the Hitura mine plant for treatment, the road distance is approximately 18 km. The nearest "village" is Tiitonranta about 2-3 km to the ESE of the deposit. Use of mining personnel would not be an issue, with the Hitura mine being close by and the local area having a long mining tradition.

Figure 3 Kopsa Au-Cu deposit in relation to regional infrastructure, and other Belvedere properties

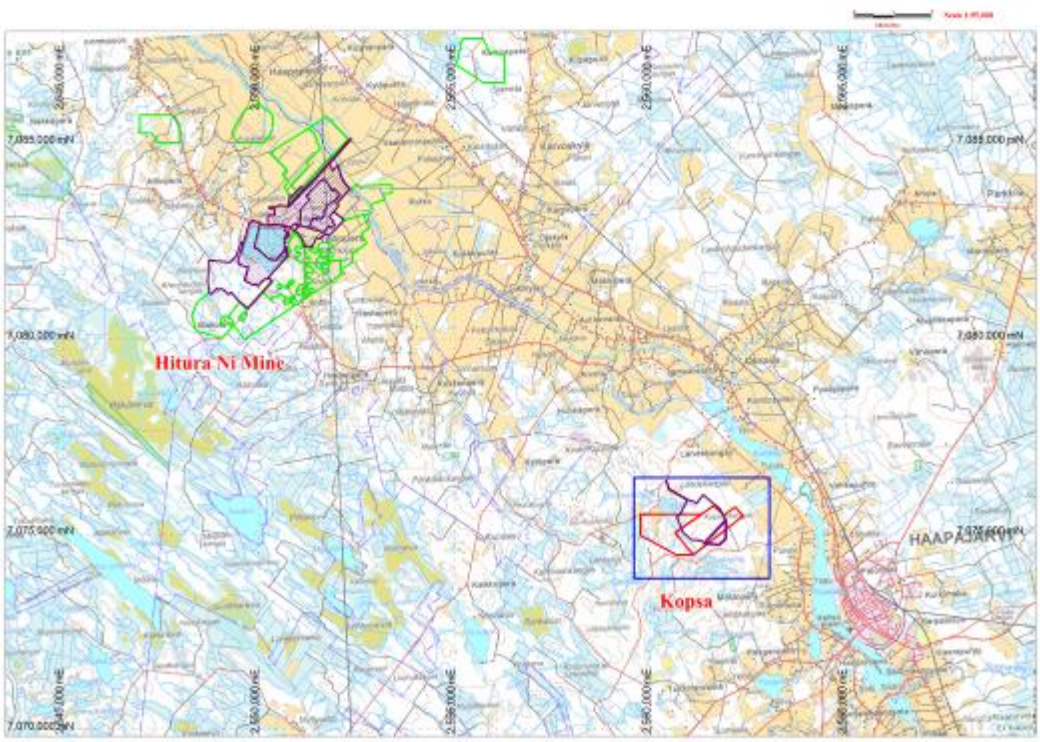

Figure 4 Kopsa deposit in relation to local infrastructure, and other Belvedere properties, including the Hitura Nickel Mine

The topography of the Kopsa property (Figure 5) is fairly plain and covered with birch, spruce and pine forest. The other occasional species of vegetation in the area is aspen. There is only one outcrop in the central part of the deposit. The elevations of the Kopsa property vary from 100m to 120 m above mean sea level. The only water body in the vicinity of the deposit is Levalämpi lake (200m x 250m) with an elevation of 106.5m MASL. The Kalajoki River flows about 2.5 km to the East of main Kopsa ore zone. The overburden thickness on the property varies from 0 m in the main Kopsa zone, where the outcrop is to about 25 m to the western part where small sand/ till hills are located. The climate of the area is temperate / sub-arctic, generally being covered by snow from November to April. Weather conditions follow the typical northern Fennoscandian climate, with a temperate summer and cold winter. The temperature is mostly between 10 and 25 °C during the summer months (June – August) and between 0 and –30°C (mean –8.9°C) during the winter months (December – February). The average annual rainfall is between 500 and 550 mm in the project area. The terrain is covered by snow for 5-6 months during the winter, during which time bogs, small rivers and lakes are frozen.

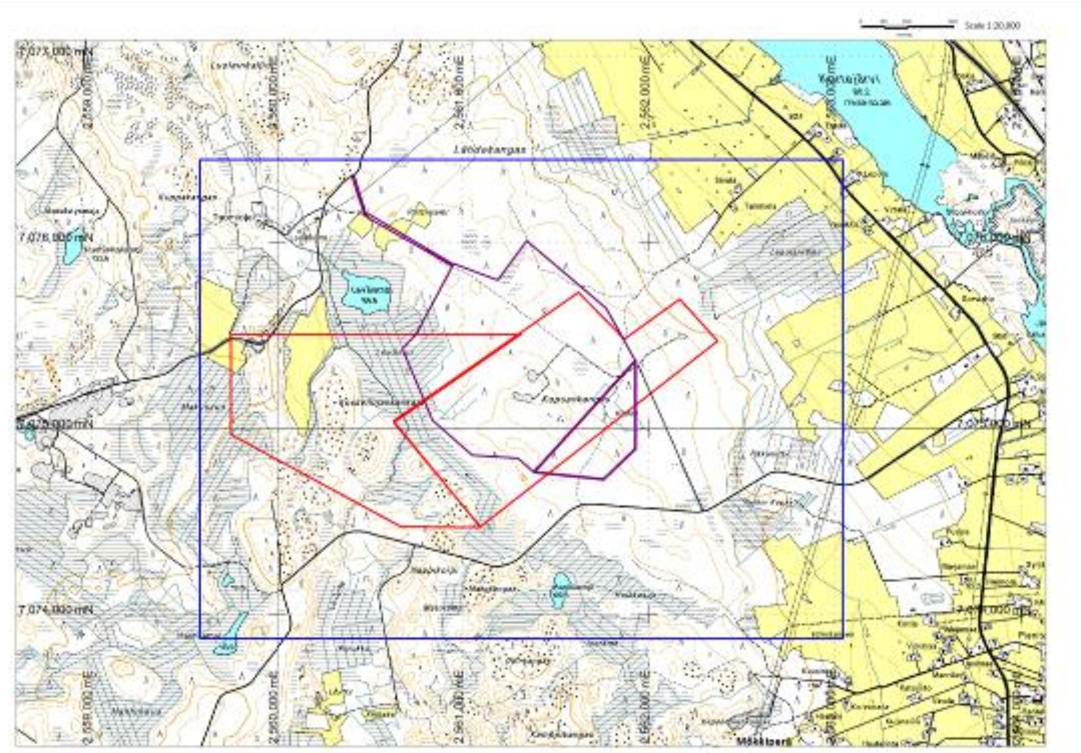

Figure 5 Topography of the Kopsa Property showing Claim Reservation (blue), Claims (red) and Mining Lease Applications (purple)

6 History

The Kopsa Au-Cu-Ag-As zones and associated mineralisation of the Sorola occurrence has been an exploration target since 1937, when boulders were traced to the Kopsa outcrop. The area has had numerous studies undertaken on it by various exploration companies.

6.1 Title

The previous claim holders of the Kopsa property are as below in Table 3.

Name of the Company	Period of exploration
North Finland Research Foundation	1943 – 1954
Geological Survey of Finland (GTK)	1939, 1961, 1983 – 1985
Outokumpu Oy	1940 – 1941, 1964 – 1966, 1971 – 1973, 1977 – 1978, 1980 – 1982
Baltic Minerals Finland / Glenmore Highlands Inc	1995 – 1999
Belvedere Resources Finland Oy (subsidiary of Belvedere Resources Ltd)	2002 - 2009
Finn Nickel Oy (subsidiary of Belvedere Resources Ltd)	2009-2010
Belvedere Mining Oy (subsidiary of Belvedere Resources Ltd)	2010 - present

Table 2 Historical holders of the Ängesneva Property

6.2 Exploration

The North Finland Research Foundation, (1943-54) and Geological Survey of Finland (1939, 1961, 1983-85) undertook surveys of glacial erratic boulders, bedrock mapping, diamond drilling, till geochemistry, IP surveys and magnetic surveys. In the 1980's the GTK exploration was focused chiefly around the Sorola satellite occurrence.

Outokumpu completed numerous works including till geochemistry, stratigraphic mapping, ground IP, a conductivity (Slingram) and magnetic survey, ore mineralogy study, diamond drilling in various phases (1940-41, 1964-66, 1971-73, 1977-78, 1980-82). Bedrock mapping, trenching, percussion and diamond drilling, VLF surveys, and a pilot flotation processing study were also completed.

Baltic Minerals/Glenmore Highlands completed (1995-1999) a diamond, percussion and RC-drilling program, trenching and ground magnetic survey, bedrock mapping in the surrounding areas, and a geochemical till survey.

Belvedere Resources (through its various subsidiaries) has undertaken IP surveys, ground magnetic surveys, structural studies, mineralogical studies, percussion drilling and several phases of diamond drilling, and continues to explore the deposit.

6.2.1 Drilling

Drilling has been conducted by numerous organisations over the years as detailed in Table 3.

Organisation	Period	Type	Holes	Metres	Comments
Outokumpu Oy	1939 - 44	Diamond	44	unknown	
Geological Survey (GTK)	1961	Diamond	3	unknown	
Outokumpu Oy	1970's	Diamond	50	unknown	
		Percussion	200	unknown	
Outokumpu Oy	1981 - 83	Diamond	29	unknown	
Geological Survey (GTK)	1984 - 85	Diamond	13	1,256.00	Irregular grid; holes 200-100 m apart
Glenmore Highlands/ Baltic Minerals	1995-97	Percussion	432	2,090.00	
		RC	32	2,115.00	
		Diamond	18	4,148.90	
Belvedere Resources Finland Oy	2002 -07	Percussion	48	510.00	
		Diamond	32	2,647.20	
Belvedere Mining Oy	2010	Diamond	31	3,194.98	
Belvedere Mining Oy	2011	Diamond	45	4,332.91	

Table 3 Summary of drilling programmes undertaken at Kopsa

The most recent drilling conducted on the Kopsa property prior to Belvedere's involvement was by Glenmore Highlands Inc through their Baltic Minerals Oy subsidiary. Glenmore conducted percussion, RC and diamond drilling during the period 1995 – 1999. The details of the diamond drilling programme are provided in Table 4 and Figure 6.

Assays were conducted on half core, with the remaining half core retained for reference. Samples were sent to the Geological Survey of Finland (GTK) laboratory in Kuopio. The GTK completed sample preparation, and assayed a milled split of the samples for Au and Cu. Au was assayed using the fire assay technique (GFAAS) on a 50 g sample, and the Cu was determined using the ICP-AES technique on a 5 g sample. The remains of the milled samples were air freighted to Activation Laboratories of Canada, where sample preparation was completed, and the samples were assayed by a combination of INAA (neutron activation) and ICP-AES, to give analyses for a suite of 47 elements plus gold.

A summary of the results from the Glenmore Highlands drilling is provided in Table 5.

Hole	Easting	Northing	Elevation	Depth	Azimuth	Dip	Year
KDD001	2561359	7075110	112.8	301.10	12	45	1997
KDD002	2561332	7074990	113.0	303.20	12	45	1997
KDD003	2561320	7074821	112.3	300.00	12	45	1997
KDD004	2561372	7075167	112.9	194.70	146.5	45	1997
KDD005	2561537	7075468	106.4	302.50	192	45	1997
KDD006	2561362	7075315	108.3	285.70	146.5	45	1997
KDD007	2561299	7075275	109.8	300.20	146.5	45	1997
KDD008	2561194	7075253	109.4	301.40	146.5	45	1997
KDD009	2561085	7075237	108.2	287.50	146.5	45	1997
KDD010	2561053	7075195	108.0	202.40	146.5	45	1997
KDD011	2561029	7074963	108.2	202.05	146.5	45	1997
KDD012	2561377	7075203	112.6	199.80	146.5	45	1997
KDD013	2561655	7075371	106.0	203.10	146.5	45	1997
KDD014	2561767	7075388	106.9	200.00	146.5	45	1997
KDD015	2561888	7075383	106.7	124.90	146.5	45	1997
KDD016	2561849	7075357	107.5	149.45	146.5	45	1997
KDD017	2561597	7075278	106.2	149.30	146.5	45	1997
KDD018	2561474	7075199	110.7	141.60	146.5	45	1997

Table 4 Details of the diamond drilling programme by Glenmore Highlands

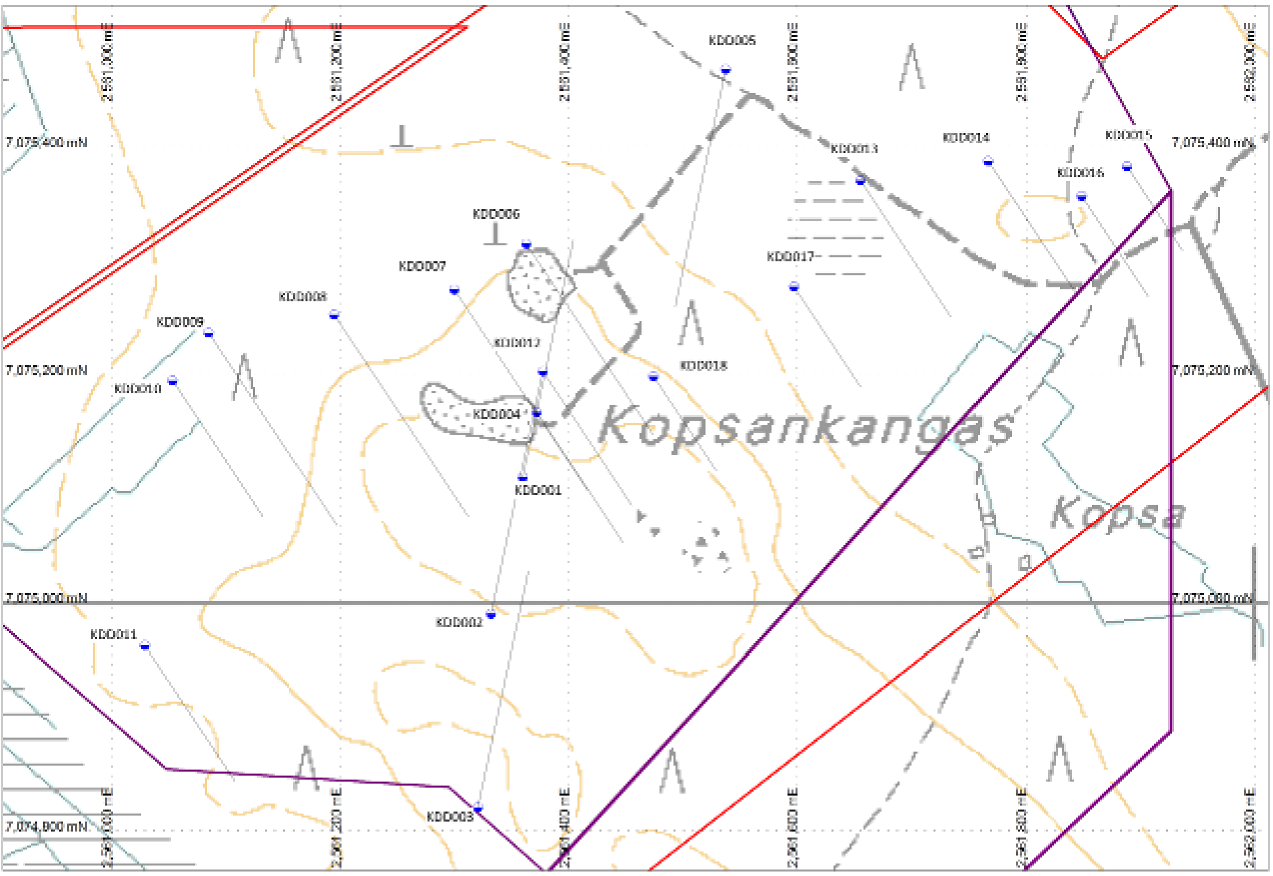

Figure 6 Location Map for the Glenmore Highlands diamond drilling

Hole	From	To	Interval	Au ppm	Cu ppm	As ppm	Grade*thickness (gm)
KDD001	18.20	69.80	51.60	2.45	2813		127
KDD002	158.80	165.35	6.55	1.36			9
KDD004	34.10	66.05	31.95	2.23	1225	5091	71
KDD004	80.55	165.90	85.35	1.52	1385	7688	129
KDD005	276.35	282.00	5.65	1.36			8
KDD007	107.90	119.60	11.70	1.59			19
KDD007	163.40	168.50	5.10	1.14			6
KDD007	193.55	201.85	8.30	0.86			7
KDD008	259.90	265.15	5.25	1.22			6
KDD009	76.20	83.00	6.80	1.15			8
KDD011	172.85	191.50	18.65	1.48			28
KDD012	4.30	74.40	70.10	0.97			68
KDD012	97.15	102.90	5.75	1.19			7
KDD015	89.25	102.35	13.10	2.89			38

Table 5 Highlights of Glenmore Highlands drilling results. Parameters for compositing were 0.5g/t Au cut-off, 7m at 0.0 g/t Au internal dilution. No top cut. Intervals shown are those with gradethickness greater than 2 gram.metres

6.2.2 Geophysical Surveys

Airborne Geophysical Surveys

The GTK carried out regional low altitude airborne measurements in the area employing magnetic, electromagnetic and radiometric methods. The prospect area and its surroundings have a good level of coverage. Low altitude surveys now extend to almost the whole of Finland.

Ground Geophysical Surveys

Numerous ground geophysical surveys have been carried out over the Kopsa Property over its long exploration history. These include gravity, magnetics, slingram (horizontal loop EM), and IP/resistivity. Belvedere has conducted its own ground geophysical surveys on the property, including magnetics and IP/resistivity.

6.3 Resources

A Historical Resource estimate for the Kopsa gold-copper deposit was discussed in a paper in Economic Geology in 1979 by Gaál and Isohanni. In this paper, a geological resource, estimated by Outokumpu Oy is stated as 24.6 million tonnes at 0.18% copper, 0.57 ppm gold, 4 ppm silver and 0.36% arsenic., with proven + probable reserves quoted at 1.1 million tonnes with the grade of 0.17% copper, 1.9 ppm gold, 4 ppm silver and 0.81% arsenic (Gaál & Isohanni, 1979). No further information on how this resource estimate was calculated is provided in the paper.

6.4 Production

There has been no historical production from the Kopsa deposit.

7 Geological Setting

7.1 Geology of Finland

The Fennoscandian Shield is the largest (> 1 million km^2) exposed area of Precambrian rocks in Europe, and similar to the famous Shield regions of Canada and Australia. The shield area constitutes large parts of Finland, north-westernmost Russia, Norway and Sweden. However, Precambrian rocks, covered by platform sediments, are known to continue to the south into the Baltic states of Estonia, Latvia and Lithuania, and southeast into Russia. In the west, the shield is bordered by the Caledonides and the reworking due to the Caledonian Orogeny is recorded in the western part of the Fennoscandian Shield. The bedrock can be subdivided into three broad domains that essentially comprise a Neoarchaean cratonic nucleus (Karelian, 3.2-2.7 Ga) flanked on both sides by Palaeoproterozoic mobile belts (see Figure 7). The Kola–Lapland domain, to the NE of the Karelian craton, records the amalgamation at around 1.9 Ga of several distinct crustal units of both Proterozoic and Archaean age, and is more characteristic of collisional tectonic processes. In contrast, the Svecofennian domain, to the SW of the Karelian craton, is entirely Palaeoproterozoic in age, and indicates relatively rapid formation and accretion of new crust between about 1.97–1.80 Ga. The geology of Finland and Fennoscandian Shield is well described by Lehtinen et al. (2005).

Archaean history of the Karelian Craton

The Karelian craton is characterised by narrow northerly trending greenstone belts surrounded by spatially more extensive granitoids and higher-grade gneiss domains. Although rocks up to 3.6 Ga are present throughout the craton, the earliest well-documented magmatic and metamorphic event seems to have taken place at around 2.84 Ga. The lower metamorphic grade greenstone sequences formed after this event, and were variably deformed and intruded by tonalitic to granitic magmas between 2.75–2.69 Ga. The Kuhmo and Suomussalmi greenstone belts are the most extensive well-preserved supracrustal units in the Archaean rocks of Finland, outcropping over a strike length of nearly 200 km, though seldom exceeding 10 km in width. They both contain abundant tholeiitic and komatiitic volcanic rocks, together with related intrusive and subvolcanic cumulates, and lesser felsic volcanic and volcanoclastic units.

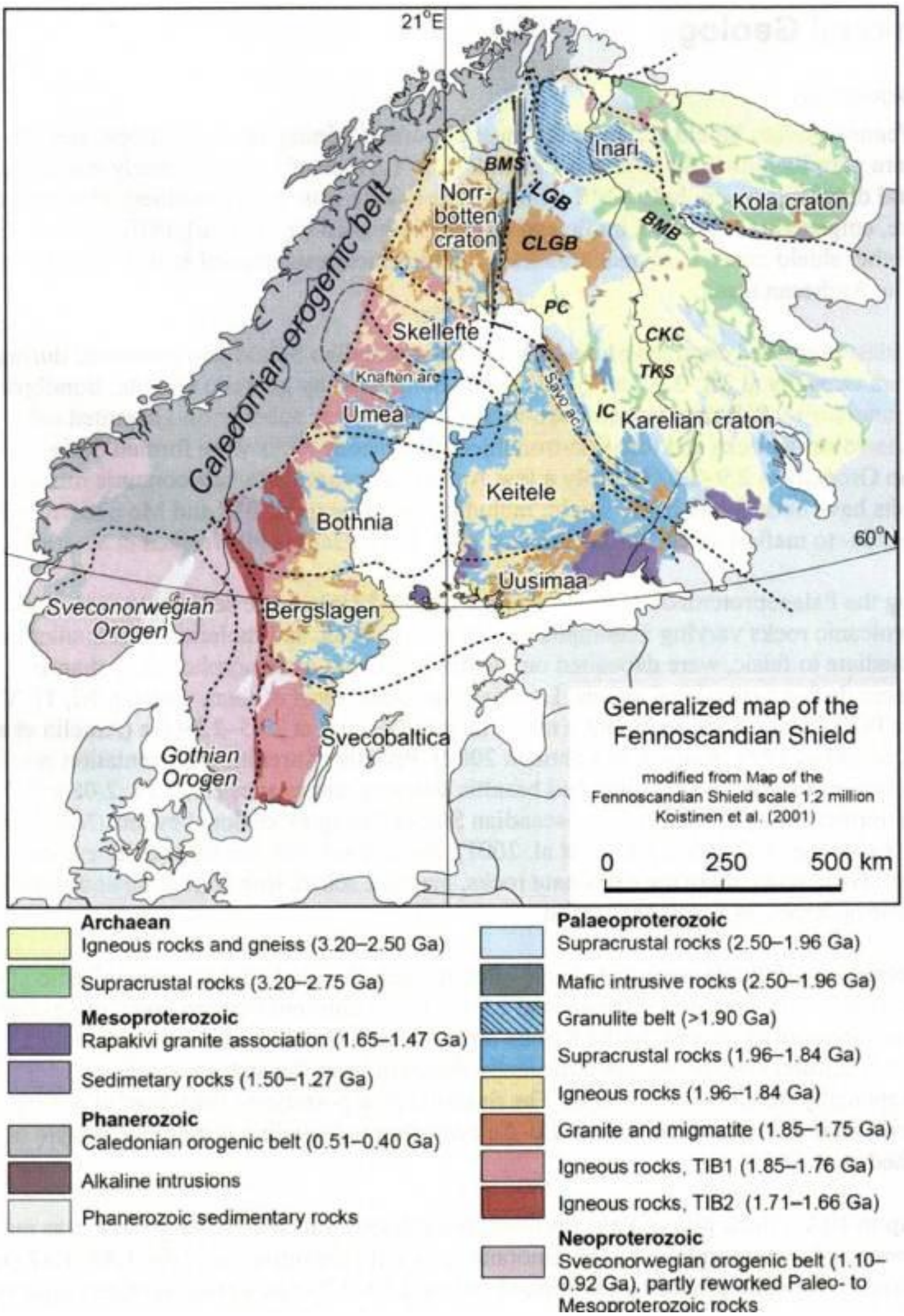

Figure 7 Simplified geological map of the Fennoscandian Shield with major tectonostratigraphic units (Ojala et al. 2007). Map adapted from Koistinen et al. (2001), tectonic interpretation after Lahtinen et al. (2005). LGB = Lapland Greenstone Belt, CLGC = Central Lapland Granitoid Complex, BMB = Belomorian Mobile Belt, CKC = Central Karelian Complex, IC = Iisalmi Complex, PC = Pudasjärvi Complex, TKS = Tipasjärvi-Kuhmo-Suomussalmi greenstone complex.

The Hattu schist belt, near the SW margin of the craton, in easternmost Finland, represents a rather different kind of supracrustal sequence (compared to Kuhmo and Suomussalmi) that records rapid crustal growth and deformation between 2.75–2.72 Ga. Felsic volcanoclastic sediments in this belt, and lithofacies, as well as geochemistry of granitoids and some basalts are consistent with a collisional arc setting.

Only a smattering of nickel-copper sulphides and VMS-type deposits has been discovered thus far in the Archaean greenstone belts of the Fennoscandian Shield, whereas known orogenic gold deposits are more abundant and are part of an important global mineralising event at 2.7 Ga. The Archaean Siilinjärvi carbonatite, intruded in an anorogenic setting at 2.6 Ga, hosts a major apatite mine in central Finland.

Early Palaeoproterozoic Rifting of the Karelian Craton

The northern part of the Karelian craton, particularly in Finnish Lapland, records a prolonged and episodic history of sedimentation, rifting and magmatism throughout the Early Palaeoproterozoic. The Lapland greenstone belt is the largest mafic-dominated province preserved in the entire shield. Sequences of bimodal komatiitic and felsic volcanics dated at around 2.5 Ga unconformably overlie the Archaean basement and represent the onset of rifting. Continued rifting of the Archaean crust resulted in the widespread emplacement of gabbro-norite layered intrusions between 2.45–2.39 Ga. These intrusions host the world class Kemi chromite mine, and also contain widespread PGE-Ni-Cu enrichment, which now are under extensive feasibility studies.

Terrigenous clastic sediments discordantly overlie these layered intrusions with further episodes of mafic magmatism recorded as sporadic lavas and sills dated at around 2.2 Ga, 2.10 Ga, and 2.05 Ga. The latest stage includes the Keivitsa Ni-Cu-PGE deposit and coincided with rifting and subsidence of the Karelian craton margin, recorded by coarse clastic turbidites, carbonates, iron formations and finer-grained graphitic schists, the latter hosting the extensive, low grade Talvivaara nickel deposits. Mining started at Talvivaara in 2008, and by ore tonnage, it is now the largest mine in Finland.

Rifting culminated in extensive mafic and ultramafic volcanism and plutonism (2.4-2.0 Ga) in central Finnish Lapland and northwest Russia, resulting nickel-copper deposits from komatiite-type to ferropicritic (Pechenga) in composition and having numerous orogenic gold deposits (e.g. Suurikuusikko / Kittila).

Fragments of oceanic crust (1.97 Ga) were subsequently thrust back onto the Karelian craton as the Jormua and Outokumpu ophiolites, the latter being best known for its Cu-Co-Zn deposits.

Early Proterozoic Svecofennian Domain

The most voluminous crustal growth in the Fennoscandian Shield occurred in the Palaeoproterozoic age at about 1.92-1.80 Ga. This was controlled by the amalgamation of several micro-continents and island arcs (Savo arc, Keitele and Uusimaa in Figure 7) with the Archaean nuclei. This period of complex evolution, involving multiple orogenies, together constituting the “Svecofennian Orogeny”, can be divided into a micro-continent accretion stage, a stage of large-scale extension of the accreted crust, and a continent-continent collision stage.

Northeast-vergent emplacement of the Outokumpu ophiolite onto the Karelian craton foreland is inferred to record the initial collision with Palaeoproterozoic micro continents and oceanic island arc(s), generating primitive tonalites from a low-K tholeiitic source. Continued volcanism within the arc(s) at 1.92–1.90 Ga led to the formation of volcanic-hosted massive sulphide deposits, including the Pyhäsalmi Zn-Cu mine. Reversal of subduction polarity following collision, or a further arc-arc collision is invoked to explain the most extensive phase of volcanism, magmatism and deformation in southern and western Finland between 1.89–1.86 Ga.

Ultramafic intrusions within reduced sedimentary sequences provided an important setting for nickel mineralisation, including the Vammala and Kotalahti nickel belts. The gold potential of this region is also being increasingly recognized, with the currently operating Orivesi mine possibly representing a metamorphosed high-sulphidation epithermal deposit, whereas other, vein-hosted gold occurrences go into the orogenic category, and are closely associated with major shear zones in the region.

Deep seismic studies in combination with geochemical and isotopic data indicate that extensional collapse and widespread intracrustal melting took place in the period 1.84–1.80 Ga. This is presently interpreted as a thermal and gravitational response to tectonic thickening of the lithosphere, although it is currently uncertain whether or not a mafic underplate was required as an additional heat source. A distinctly separate thermal input from the mantle is however invoked to account for later extension and rapakivi magmatism at 1.6 Ga.

7.2 Mining in Finland

Finland has a long history of mining dating back to 1540 when iron ore mining commenced in the southern part of the country. Since then some 240 metal mines have been exploited and the total tonnage of ore extracted is around 250 Mt.

The Finnish metallic mining industry reached its peak in 1970's with 17 active metal mines. Currently there are 12 active metallic mines operating in Finland: Pyhäsalmi Cu-Zn- mine, Outokumpu's Kemi chromite mine, Dragon Mining's Orivesi and Jokisivu Au- mines, Kittila Au- mine, Talvivaara Ni- mine, the Pahtavaara Au-mine in Lapland, Pampalo Au- mine, Raahe Laiva Au- mine and Hitura Ni- mine. Kylylahti Cu-Co mine and FQM's Kevitsa Ni-Cu-PGM mine are commenced production in 2012.

Despite the reduction in metallic mine production in Finland, the total mining volume in Finland has increased continuously since 1995. The main reason for this is the steady growth of industrial mineral mining and dimension stone quarrying. In total 39 mines and quarries covered by the Finnish Mining Act were in production in 2004 and the mines produced a total of 19.4 Mt of ore.

7.3 Gold in Finland

The FINGOLD database was first released in 1999 (Eilu 1999) and since that time the GTK has maintained and updated the database. Now the database has over 200 drilling-indicated gold occurrences which have at least one drill hole with a grade of 1 g/t Au over 1 m or 0.5 g/t Au over 5 m. Gold deposits in Finland (Figure 8 and Table 6) are principally described in Eilu (2007) and also in Ojala & Iljina (2008).

Archaean greenstone belts (Karelian cratonic nucleus)
In the Archaean domain of Finland, orogenic gold has been recognized in all greenstone belts. The largest number and the best-known examples are from the Ilomantsi greenstone belt in eastern Finland. The existence of gold deposits at Ilomantsi and in the Kuhmo and Suomussalmi greenstone belts has been known since the 1980s, whereas the first signs of gold mineralisation in the Oijärvi greenstone belt, in the westernmost part of the Finnish Archaean, were only discovered in 1996.

Nearly all occurrences in the Finnish Archaean are typical for the orogenic gold category (*sensu* Groves 1993): structurally controlled, gold-only, low-sulphur deposits hosted by the locally most competent lithological units, enriched in As, Au, Bi, CO_2, K, S, Te, and W, and characterised by carbonatisation, sericitisation and biotitisation. Mineralisation most probably took place during the D3 to D4 stages of the Archaean orogenesis at ca. 2.70−2.65 Ga.

Kylmäkangas, in the Oijärvi greenstone belt, forms an exception to the common style: it is an Ag-Au-Cu-Pb-Zn occurrence hosted by intensely silicified felsic metavolcanic rocks unrelated to quartz veining. The style of alteration, host rock type, siting of gold, and the metal association suggest that Kylmäkangas might be metamorphosed epithermal, not an orogenic, occurrence.

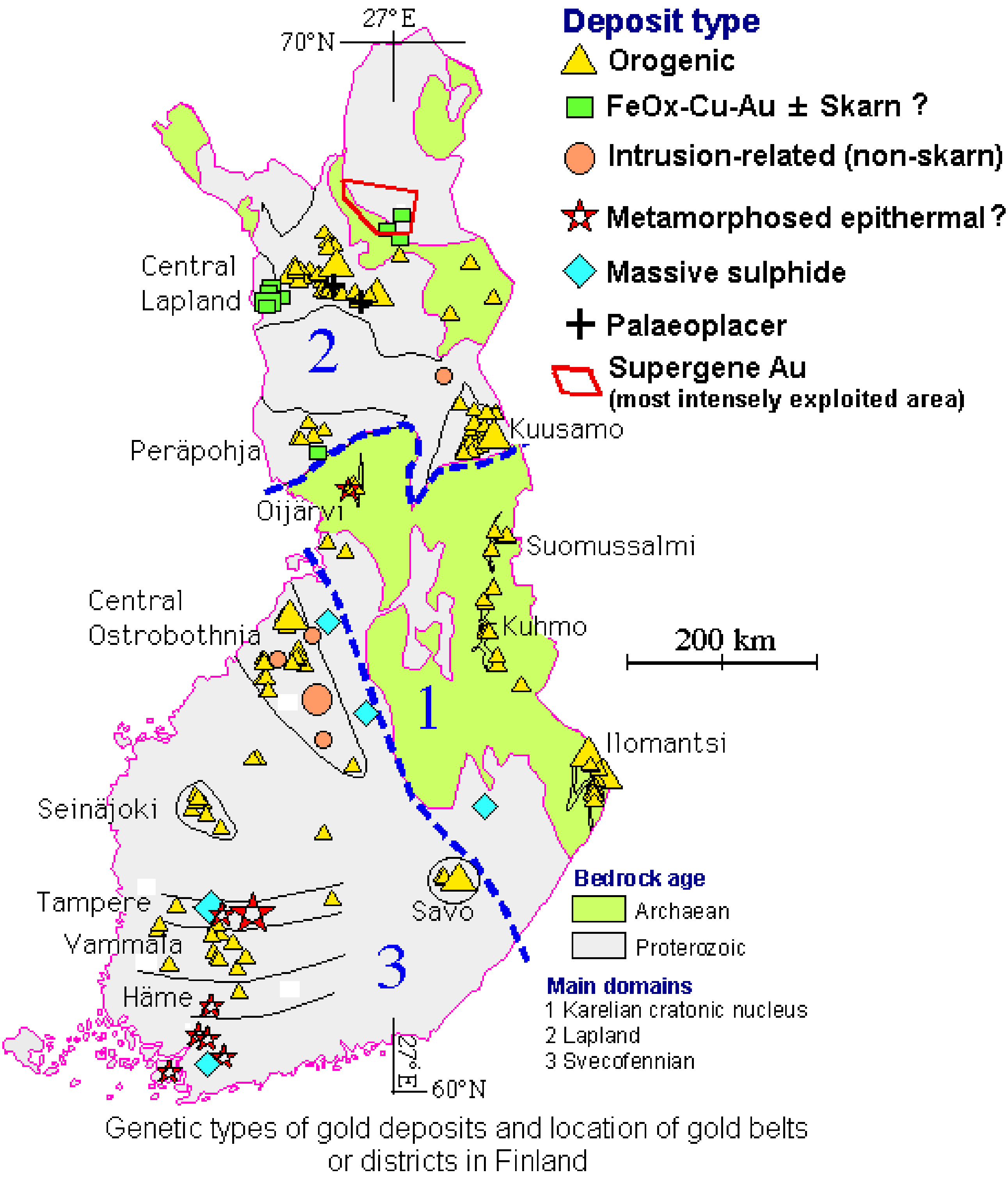

Figure 8 Genetic types of gold deposits in Finland. An Internet link to the map is:
http://en.gtk.fi/ExplorationFinland/Commodities/Gold/finland_deposit_type.html

Deposit type	Age group	Schist belt or other geological sub area	Examples
Orogenic (= mesothermal)	2700 Ma	Archean: Ilomantsi, Kuhmo, Suomussalmi, Oijärvi;	Kuikkapuro, Pampalo Jokisivu, Laivakangas,
	1900–1850 Ma	Proterozoic: Central Lapland, Kuusamo*, Peräpohja, Ostrobothnia, Savo, Tampere, Vammala	Osikonmäki, Pahtavaara, Saattopora, Suurikuusikko
Metamorphosed epithermal*	2700 Ma 1900 Ma	Oijärvi Tampere	Kylmäkangas Kutemajärvi
Skarn or Fe oxide-Cu-Au	(1880–) 1800 Ma	Central Lapland, Peräpohja	Hannukainen, Kuervitikko, Vähäjoki
Intrusion-related (non-skarn)	1900–1800 Ma	Central Ostrobothnia	Jouhineva*, Hirsi*
Massive sulphide	1920–1870 Ma	Raahe–Ladoga Zone	Haveri*, Outokumpu, Pyhäsalmi
Palaeoplacer	1900–1800 Ma	Central Lapland	Kaarestunturi, Outapää
Placer	Tertiary–Quaternary	Northern Lapland	Ivalojoki, Lemmenjoki

Table 6 Genetic types of gold mineralisation in Finland.

Note that there is some controversy regarding certain deposits or presence of deposit types. Such cases are marked by an asterisk (*). Massive sulphide deposits are included, as they have played a significant part in the production of gold in Finland, although gold has only been a by-product of these deposits, except for Haveri.
http://en.gtk.fi/ExplorationFinland/Commodities/Gold/genetic_types_of_gold.html

Palaeoproterozoic greenstone and schist belts of Lapland

At present, 78 drilling-indicated gold occurrences have been discovered in the Palaeoproterozoic greenstone belts (orogenic belts) of northern Finland. Genetic deposit types detected in the region include, at least, the orogenic, iron oxide-copper-gold (IOCG) and Paleoplacer types. The orogenic type can be further divided into the gold-only and the atypical-metal-association subtypes.

Most of the features of gold occurrences in northern Finland are similar to those detected in Palaeoproterozoic greenstone belts globally. In all epigenetic occurrences in northern Finland, structure is the regionally most significant control for mineralisation. Locally, the two most significant controls are structure and rock type. Fluid compositions suggest variable, mixed, origins for volatiles and metals with no obvious indications of a local source. The orogenic gold-only type is characterised by carbonatisation with sericitisation or biotitisation; PT conditions at 300–450°C and 1–3 kbar with pyrite, pyrrhotite and arsenopyrite being the main ore minerals; consistent enrichment of Ag, Au, As, CO_2, K, Rb, S, Sb, and Te.

Orogenic gold occurrences with atypical metal association are similar to the gold-only type, except having significant chalcopyrite ± cobaltite, gersdorffite and/or uraninite contents and enrichment in Cu and in some cases in Co, LREE, Ni and/or U and intense albitisation predating the gold-related alteration.

The iron oxide-copper-gold (IOCG) occurrences are characterised by regional albitisation ± scapolitisation, multi-stage local alteration, formation T at 400–600°C, main ore minerals of magnetite, pyrite, pyrrhotite, chalcopyrite ± cobaltite and consistent enrichment in Ag, Au, Bi, Cu, Fe, S, and Te.

Timing of gold mineralisation in northern Finland is not well constrained. Most of the orogenic gold mineralisation apparently took place during the continental collision epoch of the evolution of the Fennoscandian shield, at 1.85–1.79 Ga, although some orogenic mineralisation may be related to the earlier compressional stage, the microcontinent accretion, at 1.91–1.87 Ga. For the IOCG type of mineralisation, both of the extensional epochs of the Palaeoproterozoic orogenic evolution seem to be possible: the occurrences could have been formed during the continental extension at 1.88–1.85 Ga, or orogenic collapse and stabilisation at 1.80–1.77 Ga, or both. For the IOCG deposits in the Kolari area, the ca. 1.80 Ga timing appears to be the most probable.

Palaeoproterozoic schist belts of Svecofennian domain

The Svecofennian domain contains the most variable styles of gold mineralisation in Finland. At least orogenic, granitoid-related non-skarn, porphyry, epithermal and VMS-styles of mineralisation have been suggested.

Orogenic gold mineralisation has been detected in all schist belts and it is the dominant style in nearly all areas, whereas the other genetic types show much more restricted presence. Most of the orogenic gold deposits are typical gold-only occurrences. Several occurrences in Southern Ostrobothnia differ from all the others with a prominent Sb content, and some occurrences in the Raahe–Haapajärvi and Southern Savo areas have high Cu concentrations. High Ag, Co, Cu or Zn contents have resulted in suggestions for orogenic gold mineralisation locally overprinting pre-metamorphic, VMS, SEDEX, porphyry or epithermal base metal mineralisation. Granitoid-related non-skarn Au-Cu and porphyry Au-Cu occurrences seem to be restricted to the Raahe–Haapajärvi area and the Central Finland granitoid complex. There, the deposits are, at least spatially, related to I-type calc-alkaline granitoid intrusions.

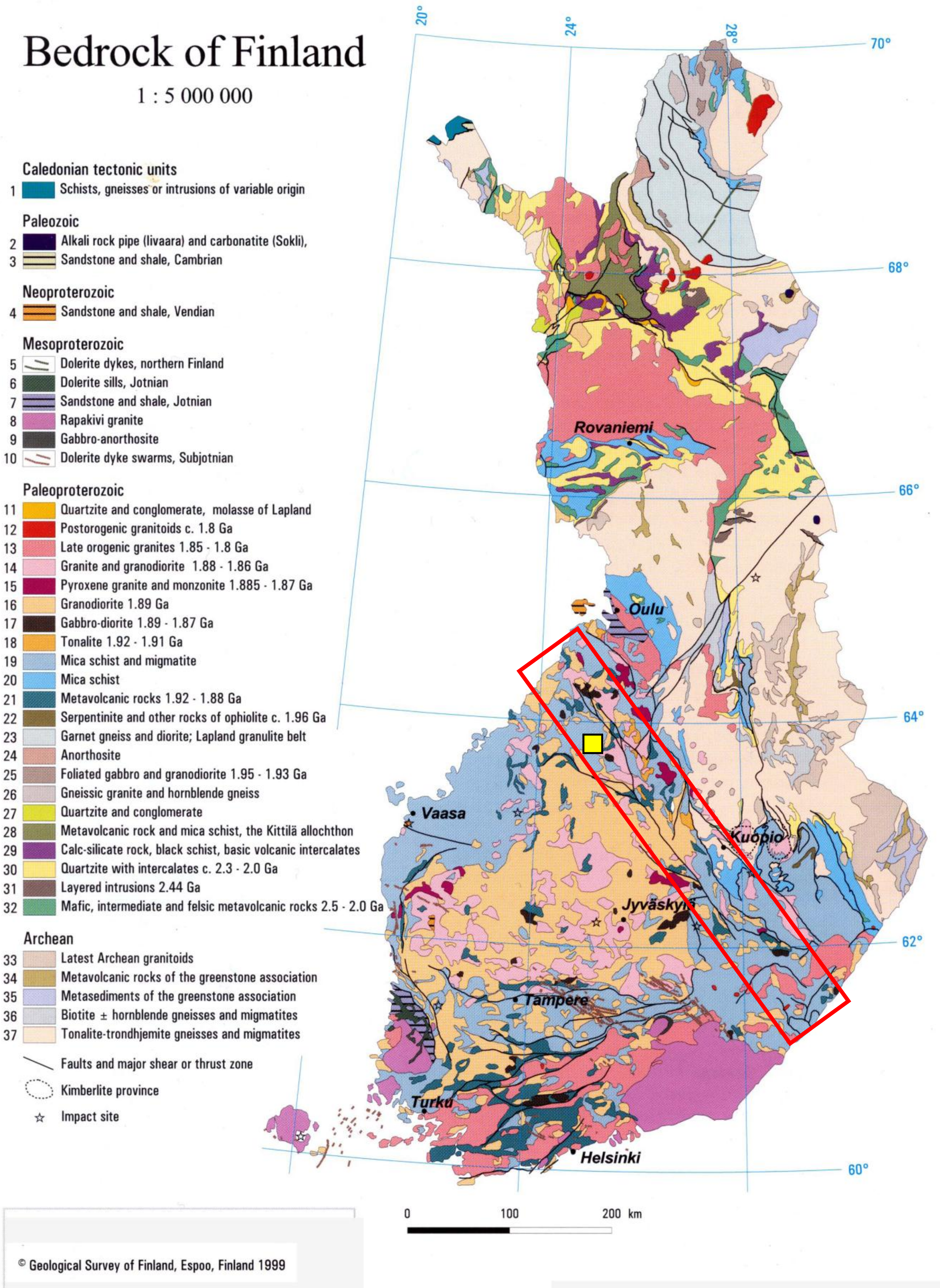

Figure 9 Geological setting of the Kopsa Property (yellow square) on the bedrock map of Finland. The Raahe-Ladoga suture zone is marked by the red area.

Epithermal and gold-rich VMS deposits have been detected in the Raahe–Haapajärvi area, and Tampere, Häme and Uusimaa belts. Especially in the Uusimaa belt, the epithermal- and VMS-style occurrences seem to be closely related, and with the few data there exists, it is difficult to say into which genetic class an occurrence would go. Also there are occurrences, like Satulinmäki in the westernmost part of the Häme belt, where there are features indicating to orogenic, and other features suggesting metamorphosed epithermal style of mineralisation. Only for the Kutemajärvi (Orivesi) and Iilijärvi deposits do almost all reported features indicate metamorphosed epithermal gold mineralisation without any significant later introduction of gold.

There are very few radiometric age data for gold mineralisation in the Svecofennian domain, and the timing must be constrained from indirect indications. The VMS and epithermal gold occurrences were probably formed during the early accretional, volcanic-arc stages of the Svecofennian orogeny, at ca. 1.92−1.89 Ga. Orogenic and intrusion-related occurrences may have had formed during the main collisional and compressional stages of the region, at 1.90–1.87 Ga or 1.85–1.79 Ga, or during both times.

7.4 Geology of the Kopsa Deposit

The Kopsa Property belongs to the so-called Raahe-Ladoga zone (e.g. Korsman 1988, Ekdahl 1993), which runs parallel to the Archaean craton margin (Figure 9) and represents the product of complex Palaeoproterozoic subduction and collision processes (Gaál 1986 and 1990). The Raahe-Ladoga deformation zone is divided into different shear zones especially in the north western part of the zone and is the most important sulphide ore zone in Finland according to the amount of deposits and occurrences.

Central Ostrobothnia (Figure 10) consists of moderately to strongly metamorphosed, in places also intensively sheared Palaeoproterozoic rocks (Kousa et al. 2000). The supracrustals are mostly migmatised mica gneisses intercalated with minor quartz-feldspar gneisses, graphite and mica schists and amphibolites of volcanic origin and locally with some dolomite and skarn. Volcanic rocks (mainly felsic and mafic) have only limited extension, but host numerous massive sulphide deposits (Pyhäsalmi, Vihanti).

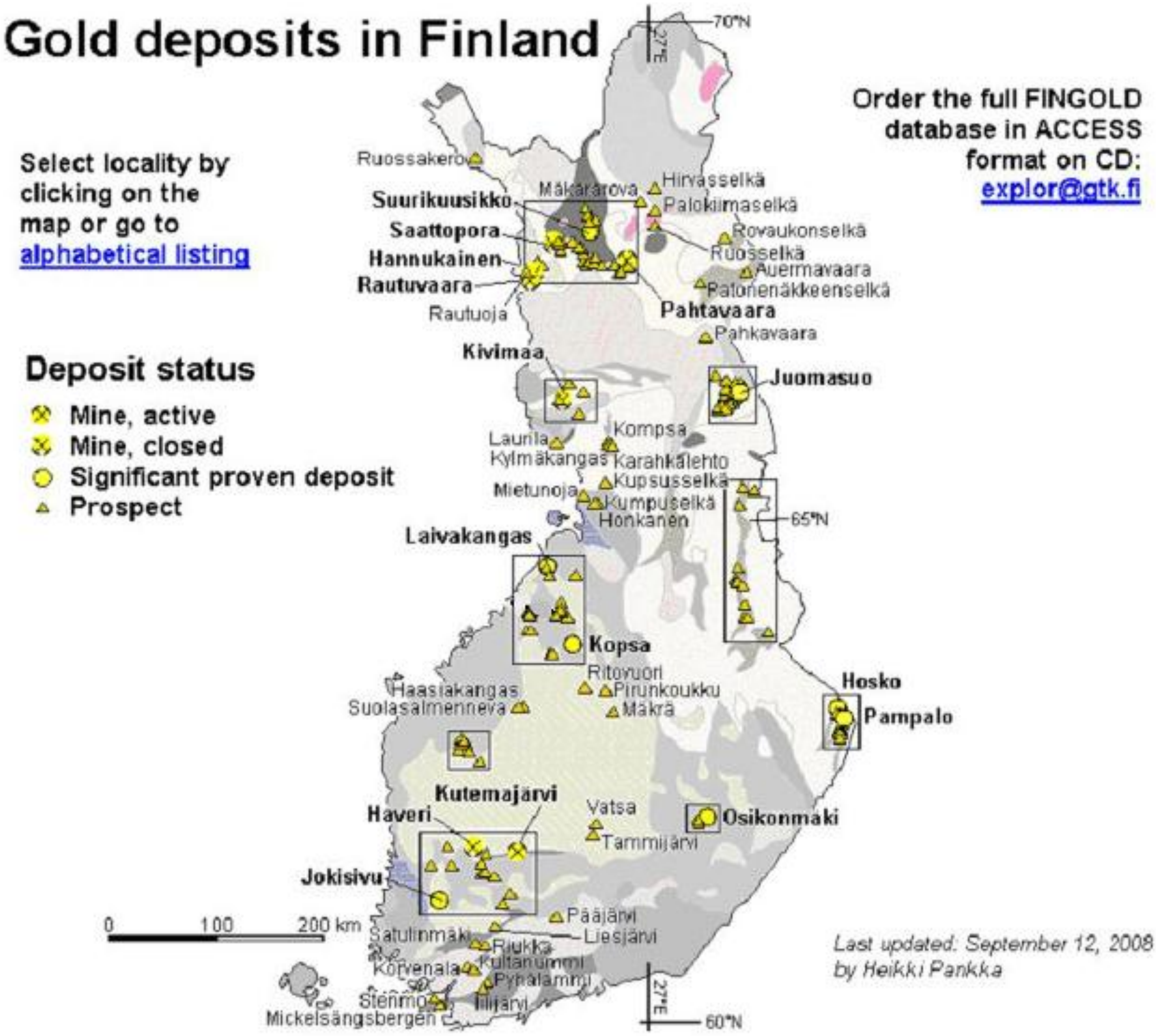

Figure 10 Gold deposits map of Finland showing the location of the Kopsa Property in Northern Ostro-Bothnia. Note the status of projects on the map is out of date.

7.4.1 Geological Setting

The Kopsa Au-Cu deposit occurs within the Proterozoic aged late orogenic Kopsa Tonalite, a roughly rhomb shaped 1200 x 550 m intrusive, intruding a 1.92 Ga package of meta-greywacke and mica schists and intermediate pyroclastic volcanic rocks, interpreted as turbidite sequences. The whole sequence has been metamorphosed to upper-greenschist to amphibolite facies. A voluminous suite of intrusives ranging from Gabbroic – Granodiorite in composition which are aged from 1.88 -1.87 Ga intrude the turbidite sequences and form the principal host to gold mineralisation in the area. It is not known if the Kopsa intrusive is a small isolated stock or is an apophosis to the regionally extensive Haapajarvi granodiorite complex to the south.

7.4.2 Lithology and Petrography

Quaternary geology

The overburden of the Kopsa Property was deposited during and immediately after the end of the last glaciation. Only one outcrop occurs on the property, the rest being buried beneath a shallow layer of till, soil and peat. The average thickness of the overburden is 2 – 4 meters.

Bedrock geology

The bedrock of the Kopsa area (Figure 11) comprises two main rock types: tonalite intrusion and turbidite metasediments. Quartz veining is also an important constituent. In addition a plagioclase porphyry is intersected in some drill cores within the intrusion. The lithological characteristics of the deposit have been studied from boreholes as well from outcrops, pits and trenches.

Metasediments: Metasediments were observed in the wall rocks of the Kopsa tonalite stock at Sorola. The metasediments display a rhythmic alternation of metapsammites and metapelites (with altered cordierite porphyroblasts), indicating graded bedding. This suggests that the sediments were initially turbidites. The folded bedding surface (S0/S1) is cut by axial planar foliation (S2). The resulting fold axis has a steep plunge. At Sorola, mineralisation is accompanied by potassium alteration.

Kopsa Tonalite: The Kopsa tonalite is predominantly medium-grained and grey in colour. In the Kopsa outcrop the tonalite is usually altered (bleached due to silicification and potassium-feldspar alteration), but apparently had a fresher appearance in the (now filled) Kopsa pit. The tonalite is mostly isotropic and non-foliated, although 5-10% of the exposed surface shows slight foliation in 2 – 4 metre thick bands. In addition to this medium-grained grey tonalite there are also a few xenoliths of dark grey, fine-grained and isotropic tonalite in the Kopsa outcrop. The xenoliths are good deformation markers since their mosaic pattern indicates thorough fracturing of the mineralised rock.

Plagioclase Porphyry: The porphyry unit has been intersected rarely in drillholes within the Kopsa Tonalite, but can be of significant thickness (> 10m intersections in drill core). It consists of a dark fine-grained matrix with plagioclase crystals up to 8 mm in size. Its relationship to the intrusive is unclear as the contacts have not been observed in outcrop. Contacts can be gradational or sharp in drill core and it may represent a minor late phase within the intrusive cycle or a cross-cutting dyke. There are similar porphyry units documented throughout the region which commonly occupy marginal positions of larger intrusive complexes or occur as sills/dykes within the meta-volcanic/ turbidite sequences. They can be regionally extensive occurring over 10's of kilometres of strike. Mineralisation has not been observed in this unit, though to date it has been intersected only outside of the main mineralised body.

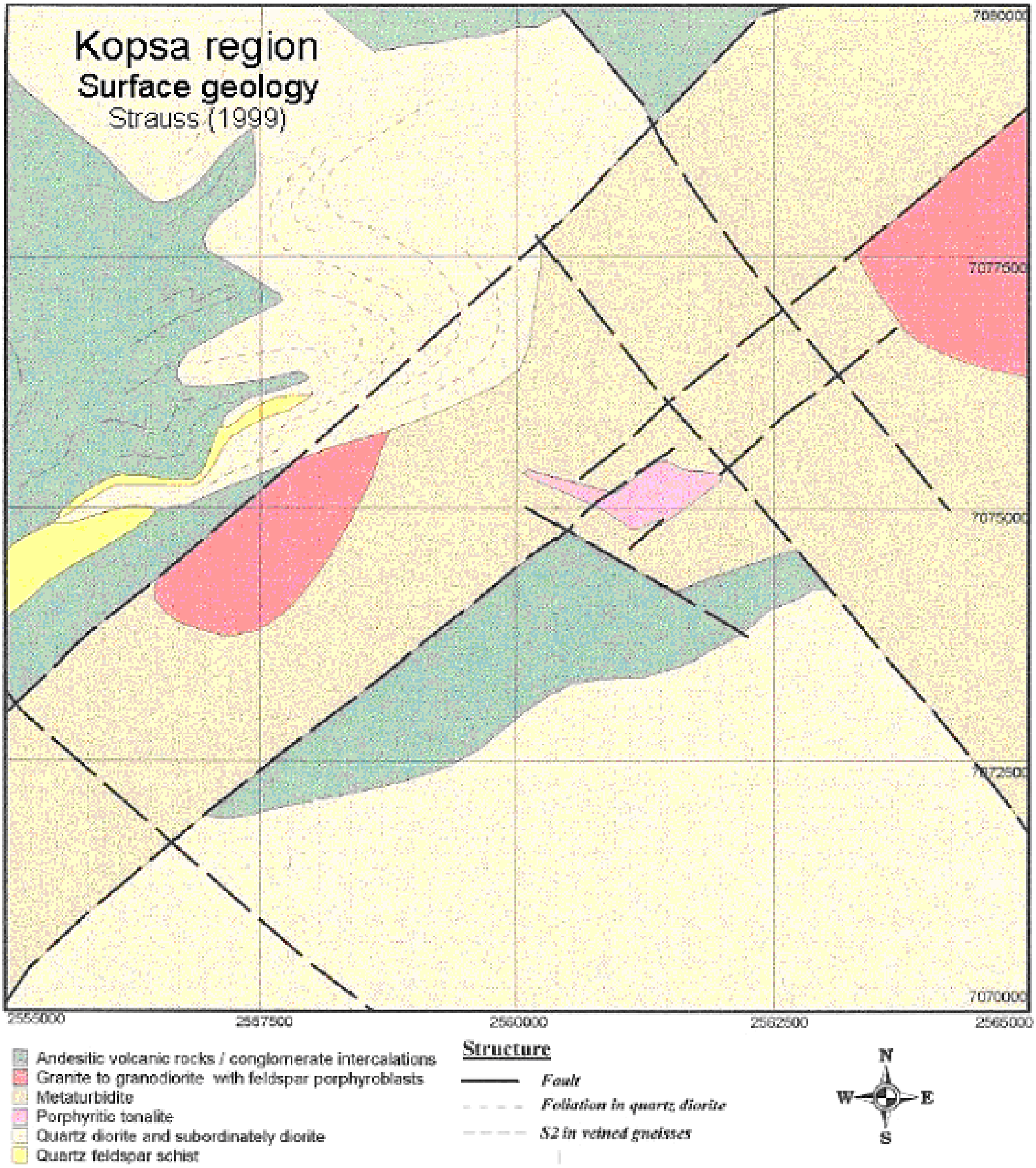

Figure 11 Bedrock geology of the Kopsa Region. The Kopsa intrusion is shown in pink.

Quartz Veins: The all important quartz veins are typically clear grey in colour. Their thickness varies from 1 mm or less up to 0.5 m. The veins have a strong structural control and are associated with fractures and shear zones of all generations. Veins of the earlier generation tend to be white and have been deformed and do not appear to carry gold mineralisation. The later clear grey Quartz veins usually contain arsenopyrite, chalcopyrite and pyrrhotite (rarely pyrite) +- dark green amphibole as breccia fillings or within fractures within the quartz veins. The sulphide fracture networks both cross-cut and are in turn cross-cut by the quartz veins suggesting multiple episodes of mineralisation. Quartz veins in outcrop and in drill core pinch, swell and step along pre-existing fracture networks and generally show poor continuity laterally and at depth. In the central higher grade part of the deposit they form a continuous stockwork, but in the outer portions of the mineralised zone occur as more isolated veins, which can be traced in outcrop over distances of greater than 10m.

7.4.3 Alteration

The tonalite is altered into granodioritic composition by K metasomatism and simultaneous depletion of Ca and Na. This is often visually observed as a whitening of the plagionclase crystals in the tonalite and at its strongest produces a "blizzard" texture characteristic of the central part of the ore zone along with the addition of hornblende and secondary biotite. Pervasive silicification and quartz stockworking are always associated with these zones. In the mica schist, K alteration is displayed by formation of biotite and muscovite. The shear zones are characterised by intense silicification.

7.4.4 Structure

In August, 1997 Dr. G. Gaál, assisted by geologists working with Glenmore Highlands, mapped the Kopsa outcrop and the Kopsa "open pit", an area that was excavated to bedrock (and subsequently refilled). A 2m x 2m grid was painted onto the washed bedrock (oriented NW-SE), enabling very detailed structural and geological mapping at a scale of 1:100. With this method very precise observations could be made on rock composition and geometry with special reference to quartz veins, silicification, sulphide impregnation, shear fractures and foliation. This study indicated that there are four principle shear directions. They are from older to younger:

1 Shear parallel to foliation 056°

2a Shear with apparent sinistral displacement, average strike 020°, dip subvertical in the Kopsa outcrop, but (almost) absent in the Kopsa pit.

2b Shear with apparent dextral displacement, average strike 300°, dip subvertical in Kopsa outcrop and 45°NE in Kopsa pit. The resulting vector points to oblique slip with upthrust to the south.

3 Youngest shear with E-W strike and unknown dip. It bends the 2a shear planes in a sinistral sense.

Silicification and sulphide impregnation is controlled by the intersection of the 2a (020°) and 2b (300°) shears, i.e. dominant mineralization direction is at 340-360° with important high grade vein mineralization along the 2a and 2b shears with a lesser control in the 3rd (090°) fracture direction. The 2b shear direction is the most developed, both in the Kopsa outcrop and the pit. These are consistent with dextral strike slip movements in the direction of 290°.

A detailed analysis of the measured structures from oriented drillcore is provided in Section 10.2. In 2005, Peter Sorjonen-Ward of the Geological Survey of Finland (GTK) attempted to bring together the data collected from drilling, detailed outcrop mapping, and more regional information into a regional framework, that could be used to explain the observed structures in relation to the distribution of the mineralisation (Section 7.4.5).

7.4.5 Geological Interpretation

The Raahe-Ladoga suture zone (Figure 9) has experienced a range of metamorphic (subducting plate during convergence and/or thickening of the crust during collision, late thermal event) and magmatic (different phases of granitoid intrusion) processes that have contributed to generation and migration of gold-bearing fluids (Kontoniemi 1998). These fluids were particularly focussed into obliquely oriented dilational sites, and the role of relatively competent rock units (granitoids, plagioclase porphyry and coarse, quartz-rich sediments) was important in channelling fluids to higher crustal levels.

The Kopsa gold project is situated close to one of the main structures of the Raahe-Ladoga suture zone. Mineralisation is hosted within a tonalite intrusive body, emplaced within a sequence of variably metamorphosed sediments, volcanics and volcaniclastics. The host tonalite is interpreted to have been emplaced into a syn-deformational dilational jog, which has been subject to continued deformation leading to the partition of the tonalite into different structural domains. The structural domains are defined by zones of pervasively microfractured massive rock separated by higher strain, ENE-ESE trending arcuate shear zones. The primary domains may be sporadically disrupted and segmented by arcuate NW-SE dextral faults and shear zones. The measured drill data and the measurements in outcrop are consistent with a 3d fault network generated under sub-horizontal NNW-SSE compression. In such a stress field the south dipping ENE-ESE trending shear zones, behave by partitioning the deformation into zones of high and low strain as depicted schematically in Figure 13 (a and c) and Figure 14. Sorjonen-Ward also indicated that it is possible to get some degree of overlap between damage zones of two vein regimes.

The report by Sorjonen-Ward goes into much greater details on vein and shear fracture orientations, and how they may be modified under continued stress within various domains. However, much of this is somewhat speculative, and quite complicated. This model as presented here provides a plausible explanation, supported by mapping and drill data, for why the steeply dipping NNW trending mineralised veins are constrained to shallow south dipping sheets of mineralisation.

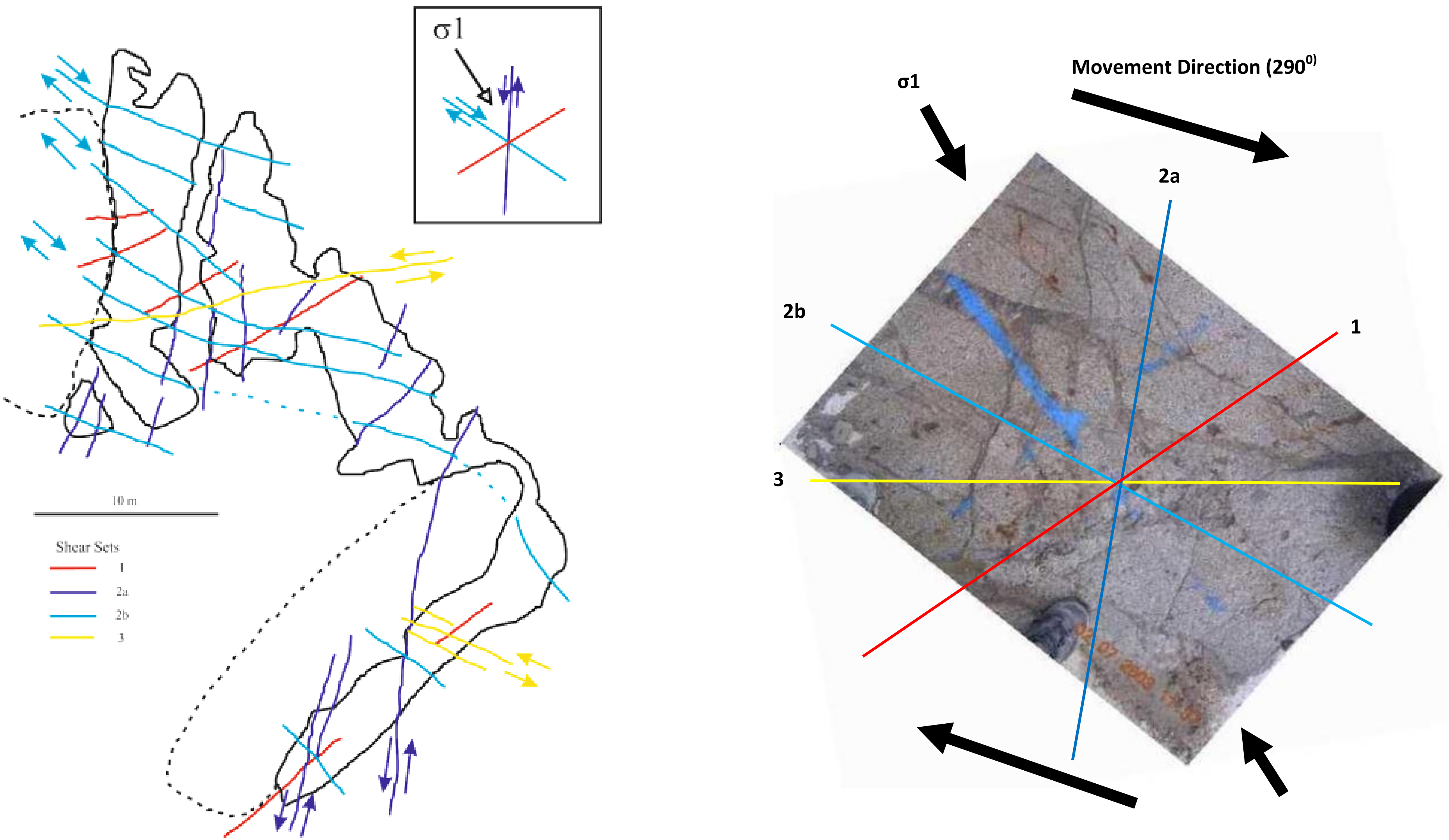

Figure 12 (a) Sketch of outcrop structures (Baltic Minerals Report, 1997); (b) Photo of veining in outcrop demonstrating directions

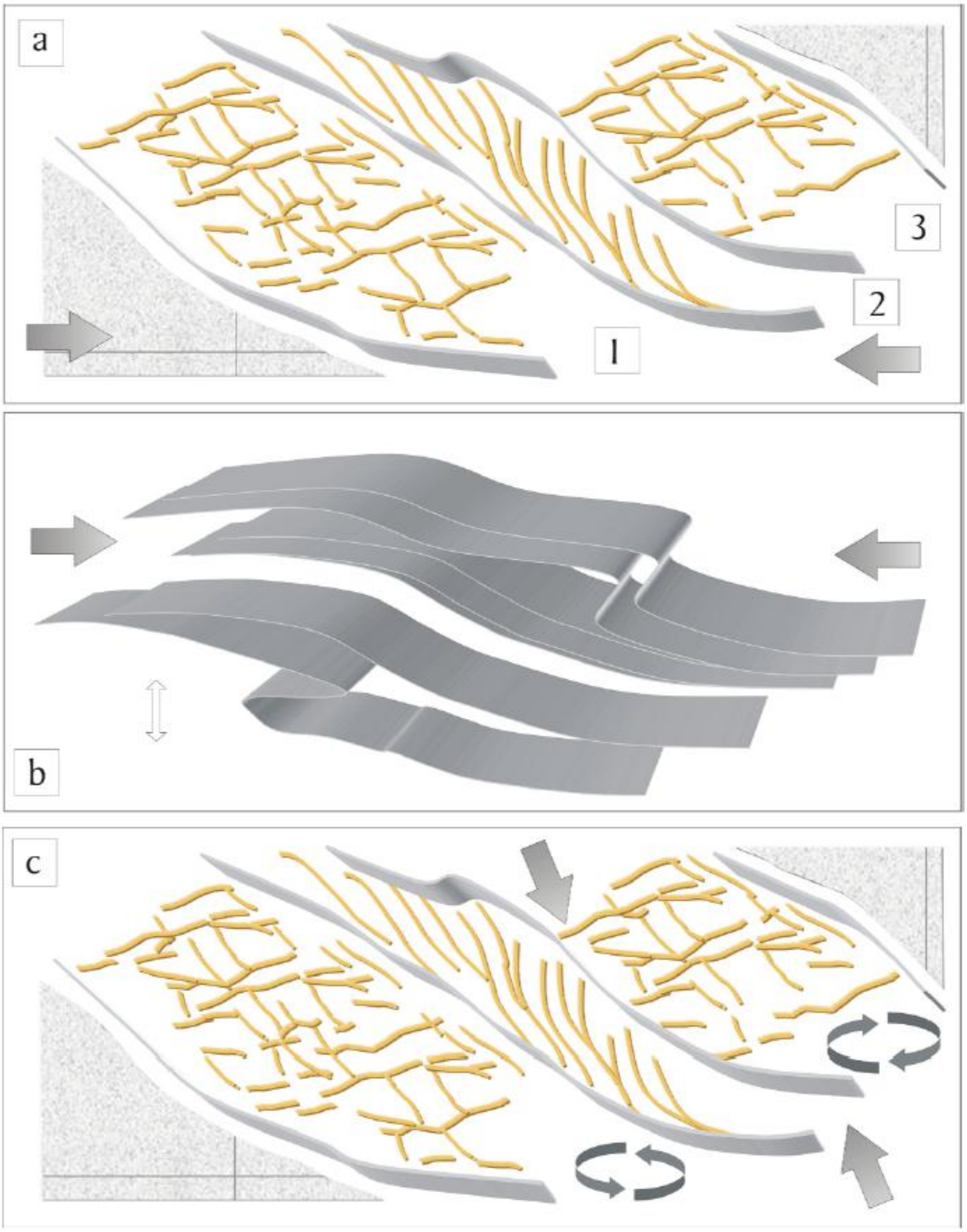

Figure 13 Schematic vertical sections of deformation partitioning. In Scenario a, heterogeneous strain accumulation leads to two deformation styles, bounded by moderately dipping reverse sense shear zones. Domains 1 and 3 shorten by failure along conjugate brittle-ductile fracture network, while in Domain 2, more intense foliation and shear zone formation will promote anisotropy and preferential failure subparallel to shear zones. Hence we may expect different vein orientation domains. Scenario b represents effect of anisotropic layering on promoting vertical dilation and subhorizontal vein systems. Scenario C represents deformation partitioning into discrete higher strain shear zones and massive domains with conjugate fracture systems. Antoclockwise or clockwise rotation, and hence alternating extensional and contractional shear kinematics depends on relative displacement rates of interacting shear zones. (Sorjonen-Ward, 2005)

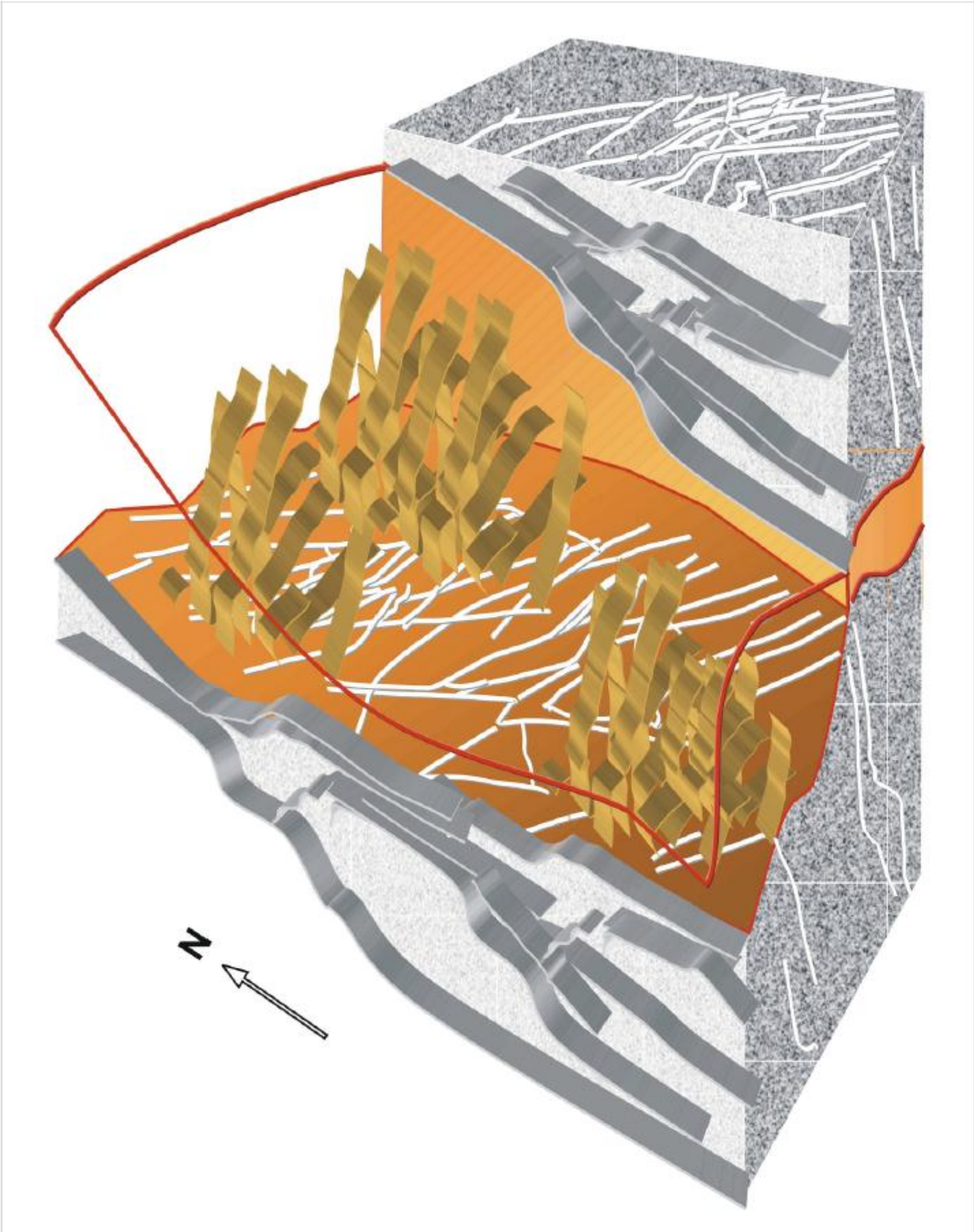

Figure 14 Schematic view of the two vein regimes, showing a panel of massive tonalite (brown) bounded by S-dipping, NW-vergent sheared tonalite domains (gray speckle), attempting to illustrate how gently dipping sheets of tonalite may have steeply dipping vein arrays. Sheared domain is associated with E-ESE trending transtensional shear veins (white vein traces depicted on floor of massive tonalite panel), whereas steep N-NNW trending conjugate veins (brown) occur within massive panel. (Sorjonen-Ward, 2005).

The structural data and interpretations have identified that the main mineralisation direction is indeed hosted in veins and along fractures which are steeply dipping and trending in a NNW-SSE orientation. However, although there is a very definite measured preferred orientation to the mineralisation within the Main Zone (NNW), it is composed of a fracture network of steep N-NNW striking vein sets which may individually have limited continuity. This is evident from the study of the outcrop and seen in drill core, where veins pinch and swell, are displaced or terminated or even refracted by other veins and fractures. This is further supported by the fact that despite the clear preferred orientation of fracture and vein networks, the overall body of mineralisation is clearly seen, hosted within an EW striking, shallow southerly dipping enveloping surface (Main Zone). Thus continuity in the principle structural direction is rangebound by the inferred EW striking, south dipping bounding structures, and continuity of mineralisation on a macro scale is best defined within this enveloping surface, rather than in the measured structural directions (see variogram maps in Section 17.6).

8 Deposit Type

Kopsa has been classified variably as a Proterozoic porphyry copper, subsequently overprinted by an orogenic gold system, to an orogenic gold deposit to more recently an intrusive related type of deposit. The evidence for a porphyry origin is weak and is largely based on the potassic alteration and the copper gold association. Mineralisation however appears to have occurred just above the brittle ductile boundary at crustal levels > 10 km. Contacts with the country-rock are passive and appear in thermal equilibrium, further supporting a deep level of formation below that associated with porphyry coppers. Likewise the model for orogenic gold is not conclusive and is based largely on a quartz vein gold association. Alteration and sulphide associations suggest fluids with at least some magmatic input, and the host intrusion also cuts the main regional d2 foliation associated with peak deformation suggesting mineralisation post-dates orogenesis.

Belvedere geologists have tentatively classified the deposit as belonging to the intrusive related style of deposits, based primarily on mineralogy and mode of occurrence. Based on mineralogy, the host intrusion is at best weakly oxidised and probably reduced, the base metal enrichment is high relative to gold grade factors typical of intrusive related gold systems. The bismuth, antimony, tellurium, molybdenum association and alteration suggest high temperature fluids with at least some magmatic component and there is a regional association of gold mineralisation with the margins of large intrusive complexes. Until further studies are done, particularly on the composition and age of ore forming fluids relative to the intrusive host there can be no conclusive classification of the deposit.

9 Mineralisation

The strike length of the mineralisation has so far been defined to approximately 700 metres with a down dip extent of approximately 200 metres. The body has a maximum thickness of about 50 metres. The mineralisation has a strike direction of 105°, and dips roughly 20° to the SSW. In the north, the mineralisation comes to bedrock surface, part of which outcrops at the “Kopsa Outcrop”. The maximum depth for the model is about 125 metres vertical from surface.

Drilling to date has still not closed off mineralisation either along strike or down dip in the Main Zone. The IP and magnetic ground geophysical signatures closely support the observed mineralisation in drill core, and indicate significant potential in other areas of the intrusion within similar interpreted structural locations. Other domains which have been identified to date require further follow up and may have different macro orientations to what is seen in the Main Zone. The North Zone for instance, based on the limited drill data, appears to have a more S Easterly strike and a steeper overall dip then the main zone. New intersections in the interpreted South zones have too sparse a data to draw any conclusions at this point.

9.1 Mineralogy and Paragenesis

A number of mineralogical studies have been completed at Kopsa over recent years. These include studies by Prof. John Moore of Rhodes University in South Africa (2004) and Olavi Kontoniemi of the GTK (2009).

The mineralisation at Kopsa is hosted entirely by the Kopsa intrusion (tonalite – quartz diorite – granodiorite). Ore minerals mainly occur as compact sulphide veins or as stringers and blebs in connection with quartz veining and silicification. Fine grained disseminated ore minerals occur outside the veins in the altered host rocks. In the higher grade areas of the deposit the quartz veins and silicification form more of a stockwork, rather than discrete vein sets.

9.1.1 Veins

Most samples studied were dominated by quartz-rich veins: either as individual thick (>1 cm) veins or multiple thin vein stockworks. The veins were dominated by relatively coarse-grained interlocking quartz grains that show irregular, but generally not sutured, contacts. Some quartz is highly strained, but others not so. The quartz grains are commonly cut by fluid inclusion trails that become particularly intense close to large arsenopyrite grains. Arsenopyrite, where present, typically occurs in the center of the vein as a relatively massive vein-fill. Chalcopyrite and pyrrhotite commonly occur together as disseminations and fracture fill within the host tonalite and in quartz veins. In the main gold zone they are subordinate to arsenopyrite, but dominate arsenopyrite within the quartz veins and host rock peripheral to the main gold zones.

The other ubiquitous component of the quartz veins, aside from sulphides, is actinolitic amphibole, generally present as clusters of small, pale green laths. It is, however, only a minor component. Occasional grains of dusty microperthite are also present.

The general lack, however, of significant chlorite, muscovite and/or carbonate, together with the presence of amphibole and K-feldspar, indicates that the veins formed at relatively high temperatures (>450^{o} C, equivalent to the amphibolite facies).

9.1.2 Sulphide Mineralogy

The major sulphides present, in order of abundance, are arsenopyrite, chalcopyrite, and pyrrhotite with occasional löllingite and pyrite. Minor sulphides and oxides include stannite, bornite, ilmenite and rutile, and occasionally hydrothermal graphite.

Arsenopyrite forms massive vein-fill in the central parts of veins, and is relatively inclusion-free (indicating open-space filling), with minor chalcopyrite and rare gold and native bismuth grains, typically enclosed near grain boundaries. Where arsenopyrite occurs with löllingite, the arsenic phase typically has a lot of gold and/or Bi- and Bi-Te mineral icliusions. Microprobe analysis indicates relatively As-rich (48%), Fe-poor (33%) compositions for arsenopyrite.

Chalcopyrite and closely associated lesser pyrrhotite are more widely dispersed in the peripheral parts of the veins and in the adjacent wall-rocks. In places fine grained chalcopyrite has been observed in fractures cutting arsenopyrite, indicating a temporal relationship. Sphalerite was observed associated with chaclopyrite in samples by both Moore and Kontoniemi. Exsolution lamellae of cubanite have been observed in chalcopyrite, and chalcopyrite alteration products like digenite and covellite have been reported by Kontoniemi.

The minor amount of pyrite compared to pyrrhotite; the presence of hornblende; together with the lack of any observed low-T minerals such as tetrahedrite-tennantite, stibnite, etc., again support a relatively high-T mineralization event. Löllingite was absent from the samples studied by Moore (2004), but was recognised in the samples examined by Kontoniemi (2009).

The assemblage arsenopyrite-löllingite-pyrrhotite, and the high As% in arsenopyrite generally indicates high temperatures and fS_2.

9.1.3 Native Elements

There is a close association between native bismuth, as the dominant mineral, and gold/electrum in the suites of samples studied by both Moore (2004) and Kontoniemi (2009). Typically these are very fine-grained (virtually all gold observed was <10µm, with a maximum of 30µm) and they occur interstitially between quartz grains or along fractures and cleavages in amphiboles (preferred locality). Where there are large arsenopyrite grains, the bismuth-gold concentrations are typically in close proximity to the arsenopyrite grain boundaries in adjacent gangue minerals. The grains are extremely fine and blebby, resembling fluid inclusion trails.

Native bismuth, rarer bismuthinite and gold/electrum were all positively identified by electron microprobe analysis. Accurate analyses, however, were difficult due to the grain size. Gold fineness ranged from about 810 to 730 (Moore, 2004), and 610 to 530 (Kontoniemi, 2009) which straddles the gold-electrum composition boundary.

No tellurides were identified in the samples studied by Moore (2005). Kontoniemi, however, identified two tellurides with native bismuth and gold, as well as some Ag, Sn and Sb bearing sulphides.

There are very few bismuth and gold inclusions in arsenopyrite, generally in the outer margins. There are also some bismuth-gold disseminations associated with chalcopyrite. Most, however, are discrete clusters of fine blebs in gangue. All samples contained bismuth and gold, and the samples with most arsenopyrite contained most gold. Bismuth-gold associations are noted from numerous gold deposits, typically gold skarns and Archaean mesothermal deposits.

The presence of fluid-inclusion-like trails of bismuth-gold is indicative of the segregation of a Bi-dominant liquid phase, possibly exsolved from the adjacent arsenopyrite. Bi-Au mixtures have a low solidus (around 250^{o} C) and would behave as a liquid when other phases (arsenopyrite in particular) have crystallized. With cooling, the arsenopyrite would exsolve a Bi-Au-rich liquid that would later crystallize as 'maldonite', breaking down to native bismuth plus gold at low temperatures.

9.2 Geochemistry

The assay database contains 6,169 individual Au assays from all phases of drilling. A statistical summary of the Au, As and Cu assays are produced in Table 7. The statistics for arsenic are not entirely valid as for some of the assay methods arsenic grades > 1% As (10,000 ppm) are outside of the detection limits and the assays have been reported as 10,001 ppm As.

The average gold grade for all drilling within the modelled mineralised zone (see section 17), which is roughly based on a 0.4 g/t cut-off grade was 1.00 g/t Au. The average for Belvedere drilling was also 1.00 g/t Au, whereas for the Glenmore and Outokumpu diamond holes it was slightly higher, and for the Glenmore reverse circulation (RC) drilling it was significantly lower.

Statistically, there is very poor correlation between gold and copper (Figure 15), with much better correlations with arsenic (Figure 16) and logged quartz percentages (Figure 17). This not unexpectedly reflects the association of arsenopyrite in quartz veins being associated with gold mineralisation. Gold correlates best with bismuth (Figure 18), having a correlation coefficient of 0.97. Despite it's poor correlation with gold, copper correlates very well with silver (Figure 19) having a correlation coefficient of 0.87.

The assay data support a Au-As-Bi association and a separate Cu-Ag association.

Au g/t						
	Samples	**Min**	**Max**	**Mean**	**Std Dev**	**Median**
All Drilling	6169	0.0005	62.80	1.00	2.43	0.40
Belvedere	4404	0.0005	48.40	1.00	2.47	0.39
GMH Diamond	591	0.0005	35.00	1.11	2.39	0.40
Outokumpu	601	0.0005	62.80	1.17	2.92	0.60
GMH RC	573	0.0005	20.00	0.65	1.32	0.30
Cu ppm						
	Samples	**Min**	**Max**	**Mean**	**Std Dev**	**Median**
All Drilling	5992	41	22,800	1,506	1,441	1,100
Belvedere	4404	43	22,800	1,485	1,441	1,080
GMH Diamond	591	110	15,600	1,537	1,401	1,170
Outokumpu	424	48	17,700	1,906	1,717	1,400
GMH RC	573	41	9,786	1,337	1,181	993
As ppm						
	Samples	**Min**	**Max**	**Mean**	**Std Dev**	**Median**
All Drilling	5715	1	200,000	5,235	8,893	2,600
Belvedere	4404	1	200,000	5,034	8,859	2,370
GMH Diamond	324	14	79,000	5,766	11,891	1,400
Outokumpu	501	1,200	98,800	8,336	9,431	4,900
GMH RC	486	31	31,000	3,506	4,474	1,900

Table 7 Summary statistics for Au, Cu and As in the modelled envelope of mineralisation

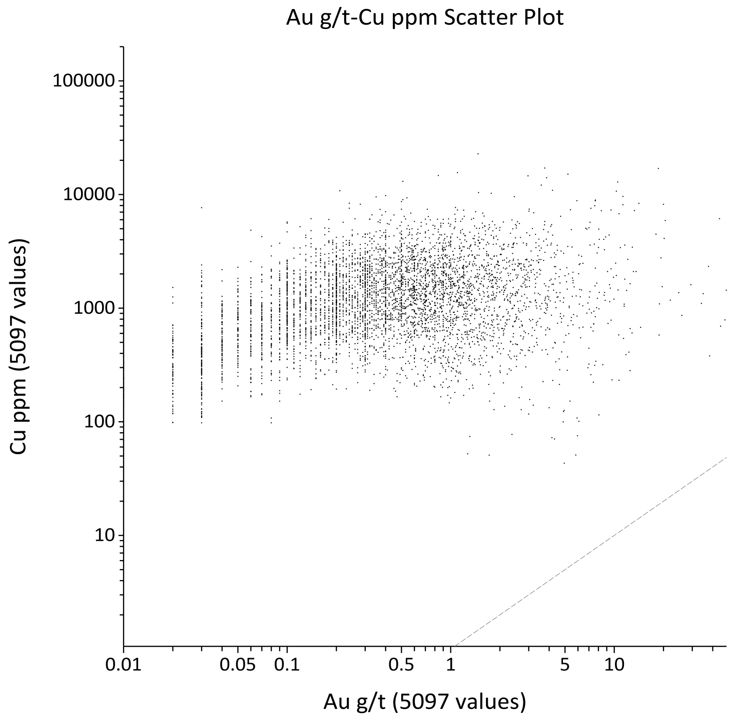

Figure 15 Scatterplot of Au versus Cu within the modelled mineralised shell

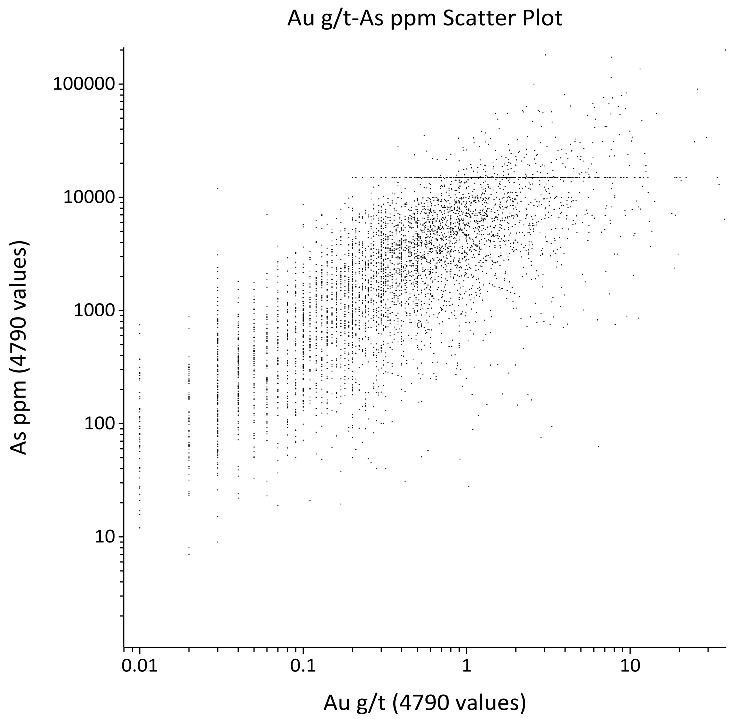

Figure 16 Scatterplot of Au versus As within the modelled mineralised shell

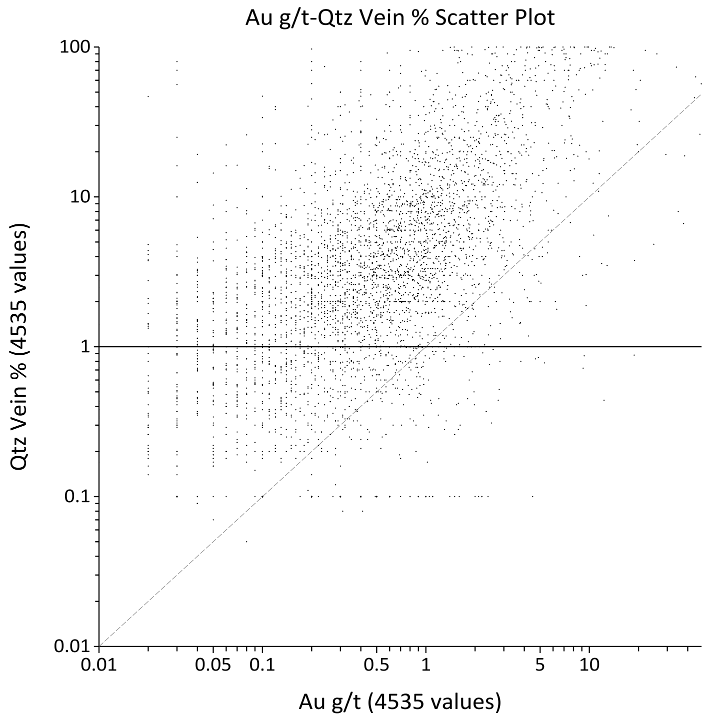

Figure 17 Scatterplot of Au versus quartz vein percentage within the modelled mineralised shell

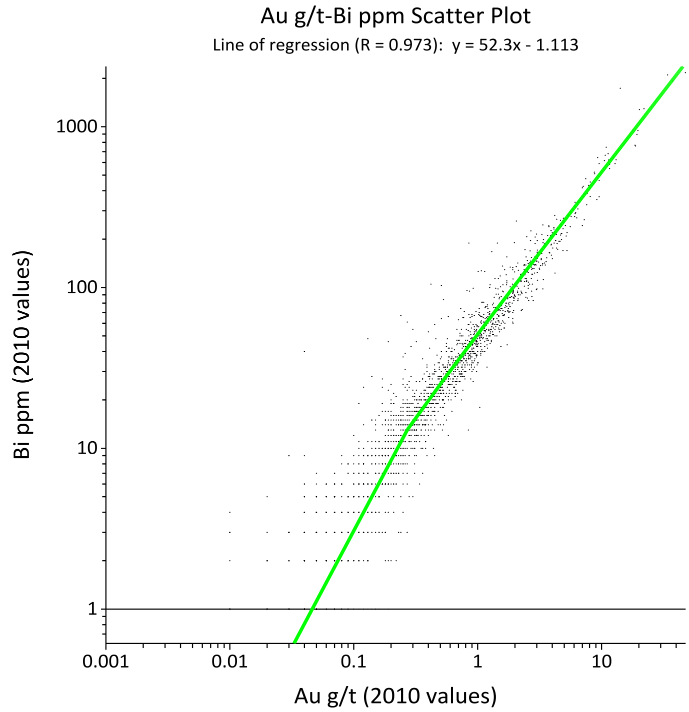

Figure 18 Scatterplot of Au versus Bi from within the modelled mineralised shell

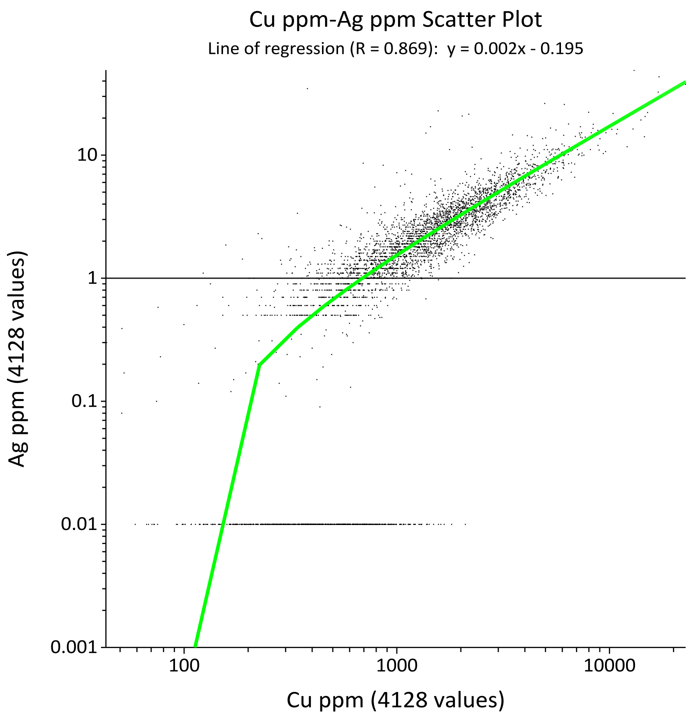

Figure 19 Scatterplot of Cu versus Ag from within the modelled mineralised shell

10 Exploration

Belvedere Resources has conducted numerous exploration programmes at Kopsa since it first acquired the property. These include ground geophysical surveys, geochemical surveys, structural studies, percussion drilling, and six phases of diamond drilling.

10.1 Geophysical Studies

Over the years numerous geophysical surveys have been carried out, some more useful and better quality than others. The most recent survey was an IP/Resistivity survey carried out by JVX of Canada in 2003/2004. This only covered the western half of the intrusion and extended out into the "panhandle" to the west. The results of this survey were merged with an earlier Glenmore Highlands survey carried out in 1997. The composite map with land tenure and the extents of the current modelled mineralisation is shown in Figure 20.

From this it is clear that the IP is a useful guide for the mineralisation, and provides further exploration targets both to the north and south of the Main Zone mineralisation..

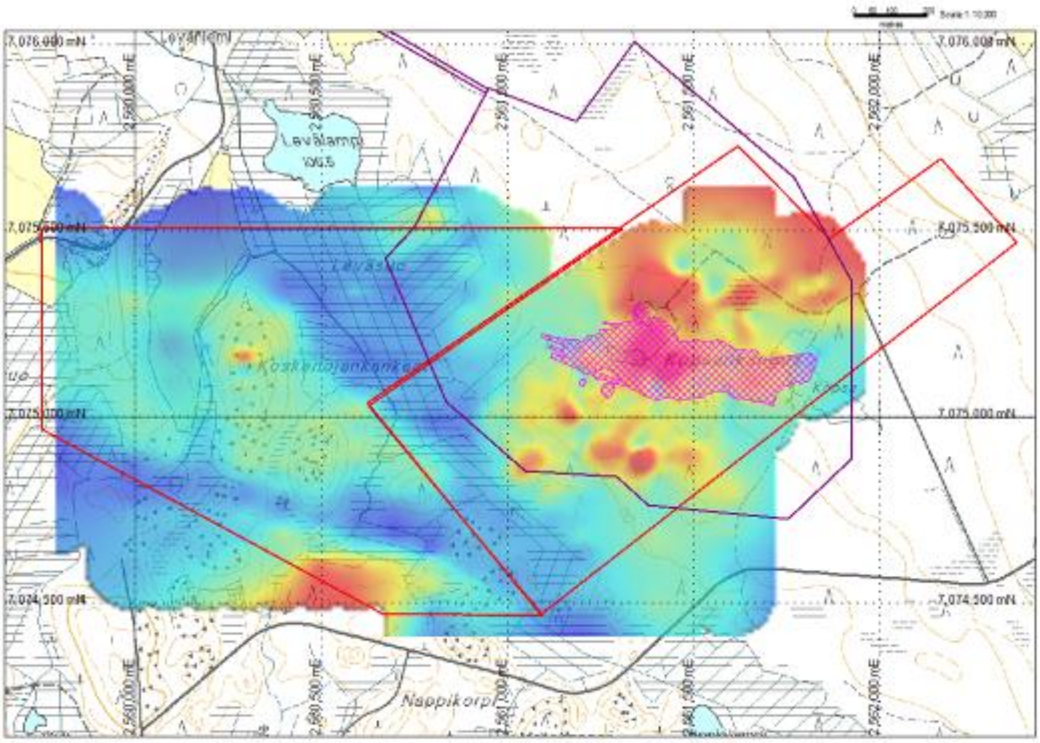

Figure 20 Compilation map of IP surveys with modelled mineralisation

10.2 Structural Study

A macroscopic study of the structural features in the limited outcrops of the Kopsa deposit has been carried out by Peter Sorjonen-Ward of the GTK in 2005. The macroscopic structural features are discussed in the Regional Geology section of this report (Section 7.4).

The details of the methodology for taking structural measurements are described in Section 12.3. This present section deals with the results of the structural measurements taken from drillholes and the establishment of the common trends. In total 9,952 structural measurements were recorded from 19 drillholes. The main structural features recorded are shear zones and vein sets (both with and without mineralisation).

All of the following structural analyses and stereonets and rose diagrams are based on measurements with a confidence level of 1 or 2 (9,370 measurements). All of the stereonets are Equal Angle, Lower Hemispere Projections. Data is plotted as contours of the poles to the plane of the structural features. The sets are displayed with the average plane, and the corresponding dip and dip direction of that plane. In the following discussions, plane orientations are discussed with reference to strike and dip.

When the full dataset of all the veins and fractures are plotted (Figure 21), it is apparent that there is a strong NNW trend in the strike, with virtually no veins and fractures in the EW direction.

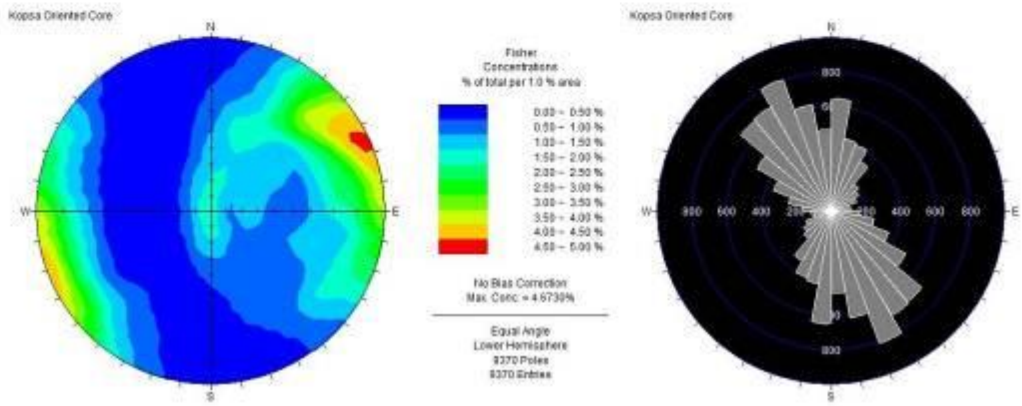

Figure 21 Stereonet and Rose diagram of all measured veins and fractures

When the veins are split according to the sample grade, it is possible to discern two populations from the rose diagrams. The closeness in the trends, and the high volume of data points makes it difficult to see these as separate populations in the stereonets for the <0.5 g/t Au and >1 g/t Au datasets (not shown). However, in the higher grade dataset (>3 g/t Au) these become clearer to see (Figure 22). The main vein population has an average trend of 332° dipping 86°W; the secondary population has an average trend of 013° dipping 85°W.

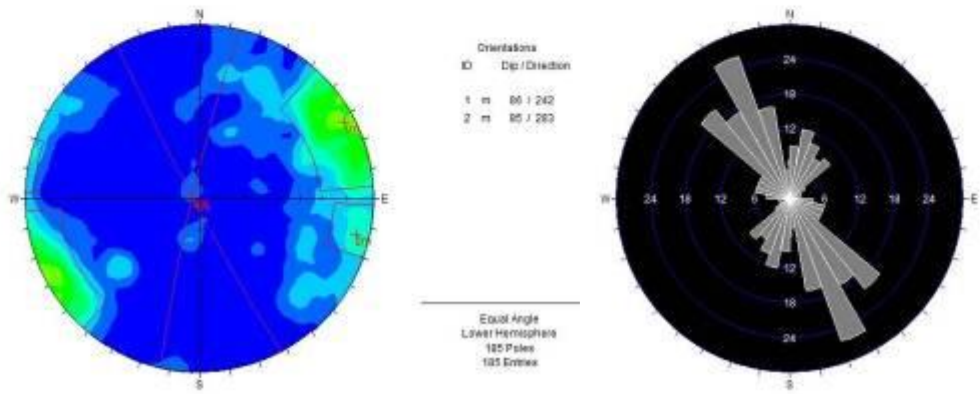

Figure 22 Stereonet and Rose diagram of all veins in samples with a grade > 3 g/t Au

The concept of Structural domains within the ore zone has been introduced in the report by Peter Sorjonen-Ward, and was discussed earlier in Section 7.4.5 on the geological interpretation. Oriented drillhole data exists for a number of the postulated structural domains, shown in Figure 23.

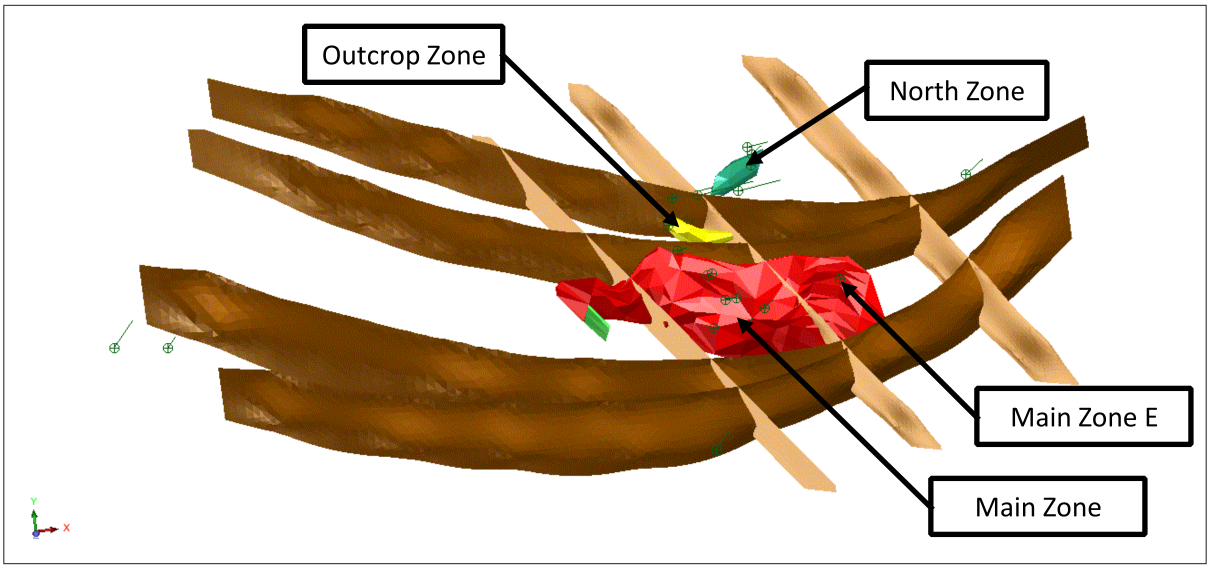

Figure 23 Interpretation of structural domains at Kopsa

The structural data has been split into the appropriate subsets, and the data is presented below in Figure 24Figure 27.

The data for all the veins and fractures within the Outcrop Zone (Figure 24), show two dominant directions: the main direction is averaging 309° dipping 61°S, with a secondary direction averaging 026° dipping 77°W. These compare very well with the 2b (300°) and 2a (020°) shear directions defined from the structural mapping of the Kopsa outcrop. The main difference between the drillhole data and the observations from the outcrop, is that the fractures and veins (related mainly to the 2b population) in an E-W direction seen prominently in outcrop are not represented in the drillhole data. This may be a function of the limited holes drilled into the outcrop (only BELKOPDD003 and BELKOPDD004) both of which were drilled in an EW direction.

The data for all the veins and fractures from the Main Zone (Figure 25) have an average strike of 347° dipping subvertically (88°W). This is similar to the measurements from the one hole drilled into the eastern extensions of the Main Zone (Figure 27) which has an average trend of 340° dipping subvertically (80°W).

The data from the North zone (Figure 26) shows two structural populations: a main direction of subvertically dipping structures trending 332° /83°W, and a flat lying population trending 334° and dipping 11°WSW.

The veins and fractures from the Main Zone, the eastern extension of the Main Zone, and the North Zone, are all fairly similar. However, it is clear that they display a slightly different trend to those structures seen in the outcrop, and supported by drilling below the outcrop. This is considered strong evidence in support of the structural domaining theory.

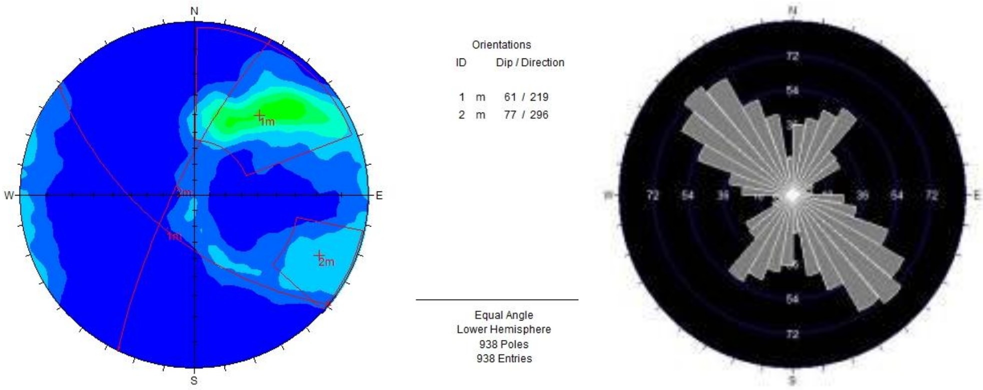

Figure 24 Stereonet and Rose diagram of all veins and fractures within the Outcrop Zone structural domain

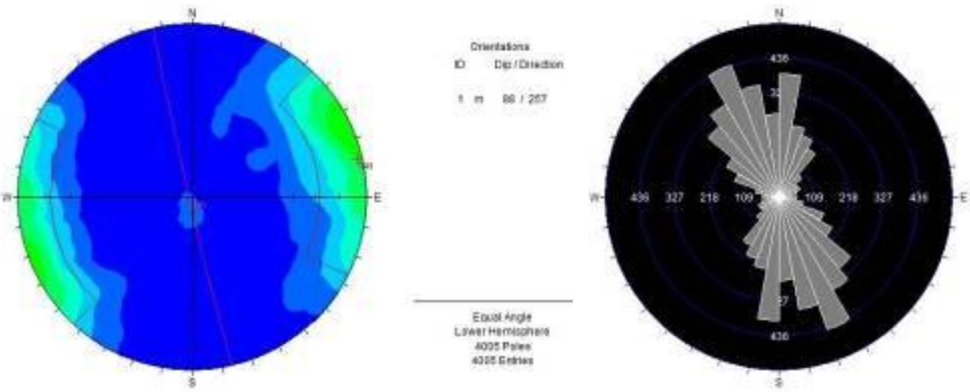

Figure 25 Stereonet and Rose diagram of all veins and fractures within the Main Zone structural domain

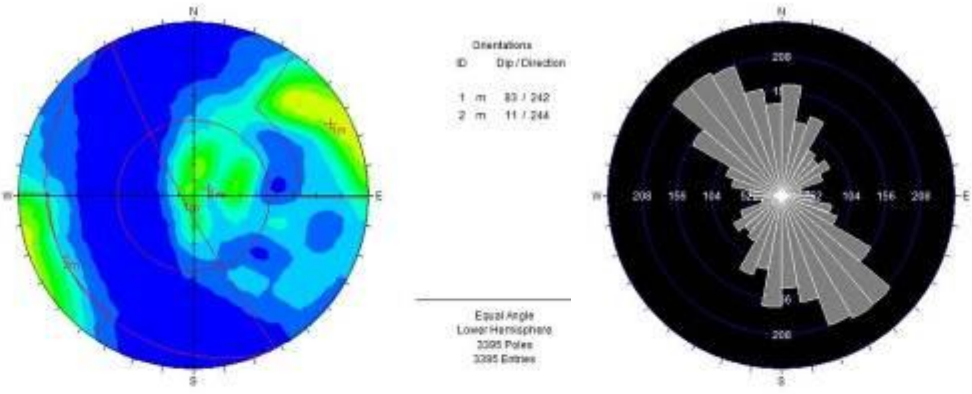

Figure 26 Stereonet and Rose diagram of all veins and fractures within the North Zone structural domain

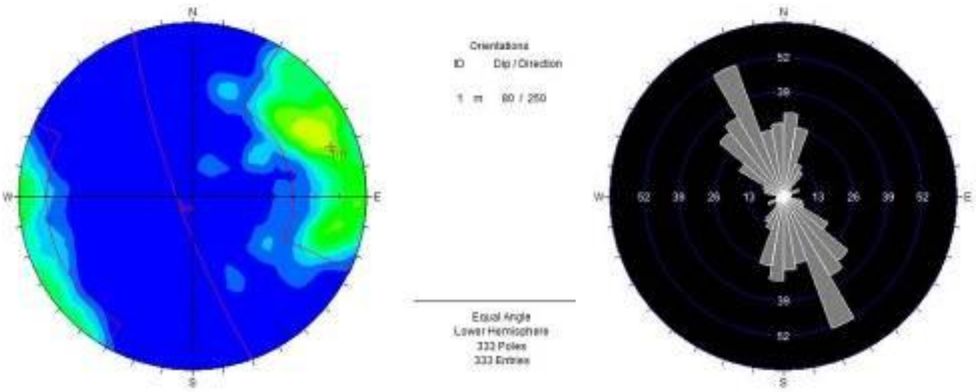

Figure 27 Stereonet and Rose diagram of all veins and fractures within the eastern extension of the Main Zone structural domain

Having determined the structural domains, it is now possible to analyse the veins from samples of higher grade constrained to within each of these domains. From the plots in Figure 28Figure 30, it is clear that the North Zone and the Main Zone have similar trends for the mineralised veins. The Outcrop Zone has two populations of mineralised trends, which broadly correspond to the 2a and 2b directions mapped in outcrop. These form an intersection that plunges 54 towards 179. This may be considered as the "primary" principal direction for the Outcrop Zone. The principal structural directions in relation to gold mineralisation for each domain are summarised in Table 8.

	Structure1		Structure2		Intersection	
Domain	Strike	Dip	Strike	Dip	Azimuth	Plunge
Main Zone	342°	81°W				
Outcrop Zone	357°	89°W	302°	59°S	179°	54°
North Zone	336°	85°W				

Table 8 Summary of principal structural directions associated with gold mineralisation in the various structural domains

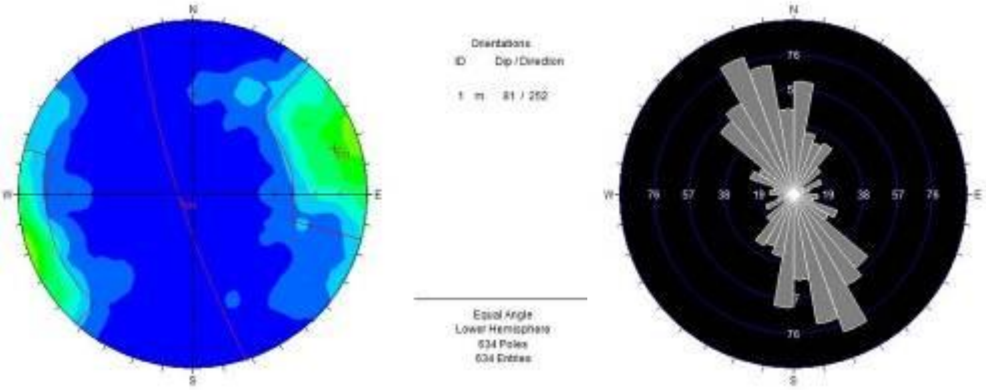

Figure 28 Stereonet and Rose diagram of all veins from samples with values over 1g/t Au from the Main Zone structural domain, including the eastern extension.

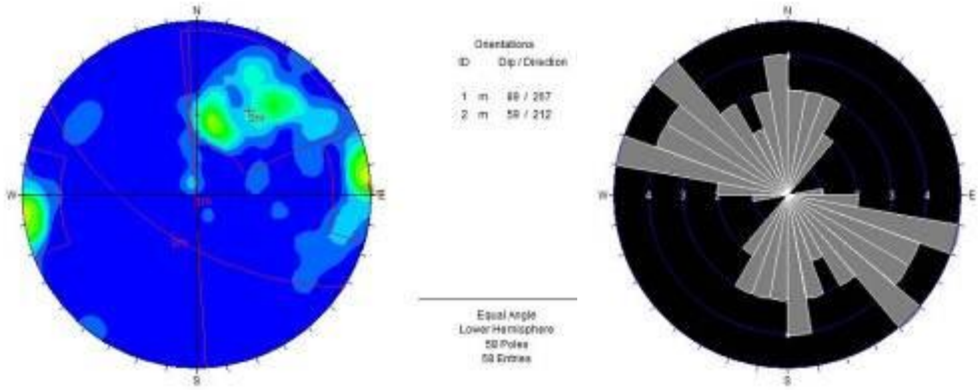

Figure 29 Stereonet and Rose diagram of all veins from samples with values over 1g/t Au from the Outcrop Zone structural domain

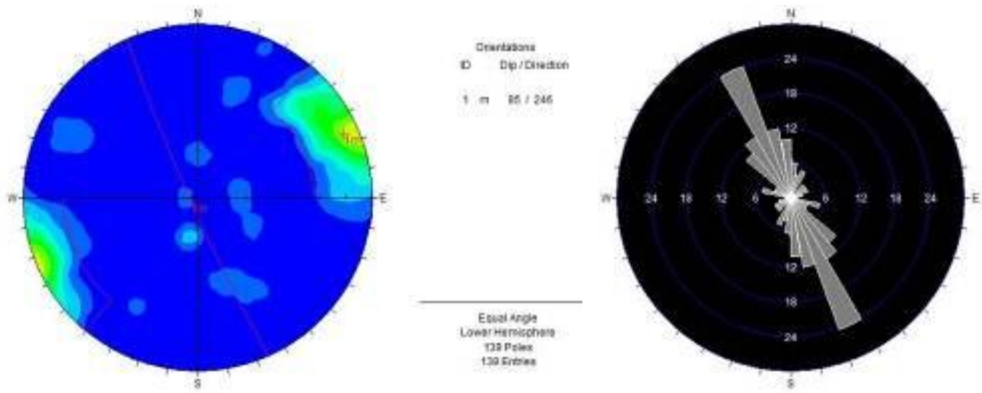

Figure 30 Stereonet and Rose diagram of all veins from samples with values over 1g/t Au from the North Zone structural domain

10.3 Review of Previous Drilling

When the data from the various periods of drilling at Kopsa are examined, it is clear that there are significant differences in the population distributions for Au. From the box-and-whisker plot (Figure 31) and the Q-Q (Figure 32) plot it is clear that the reverse circulation has substantially understated the tenor of gold mineralisation within the modelled wireframe of the mineralisation.

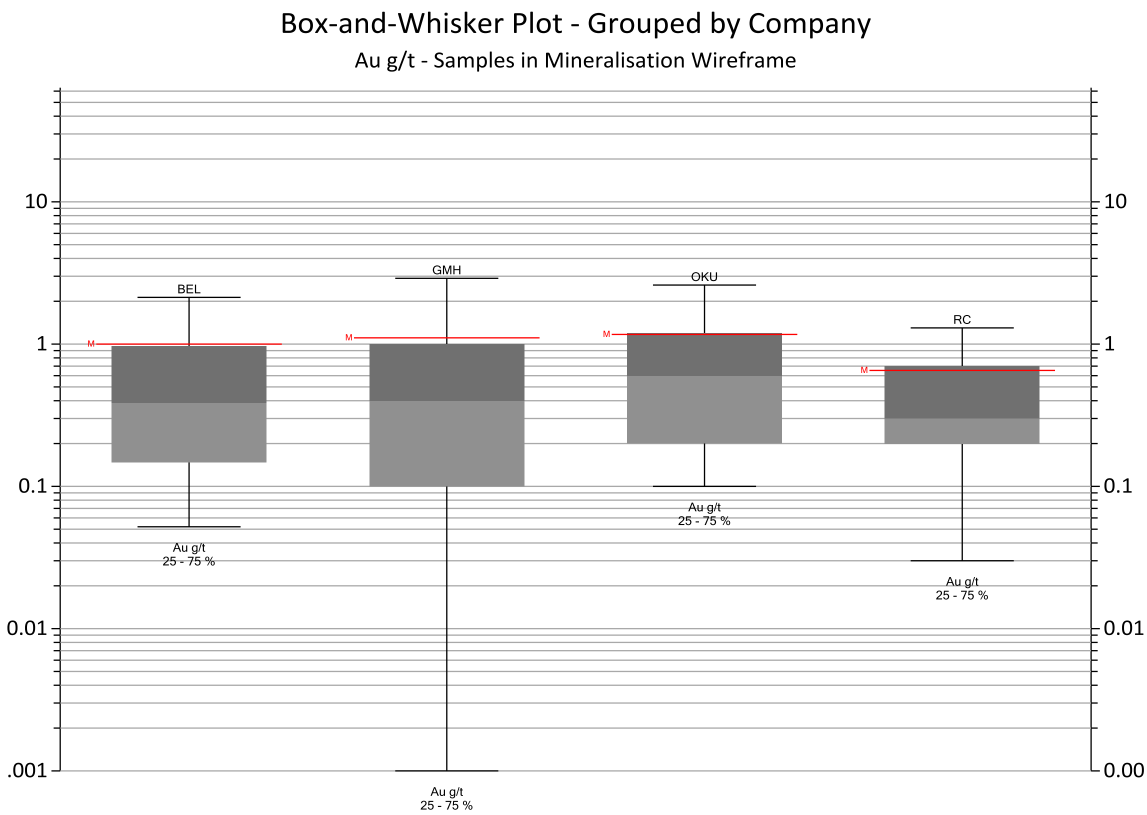

Figure 31 Box and Whisker Plot of the Au populations for Belvedere, Glenmore Highlands and Outokumpu diamond drilling as well as the Glenmore RC drilling from within the modelled wireframe. The boxes are the 2nd and 3rd quartiles, and the whiskers are at the 10% and 90% percentile. The mean for each population is shown in red.

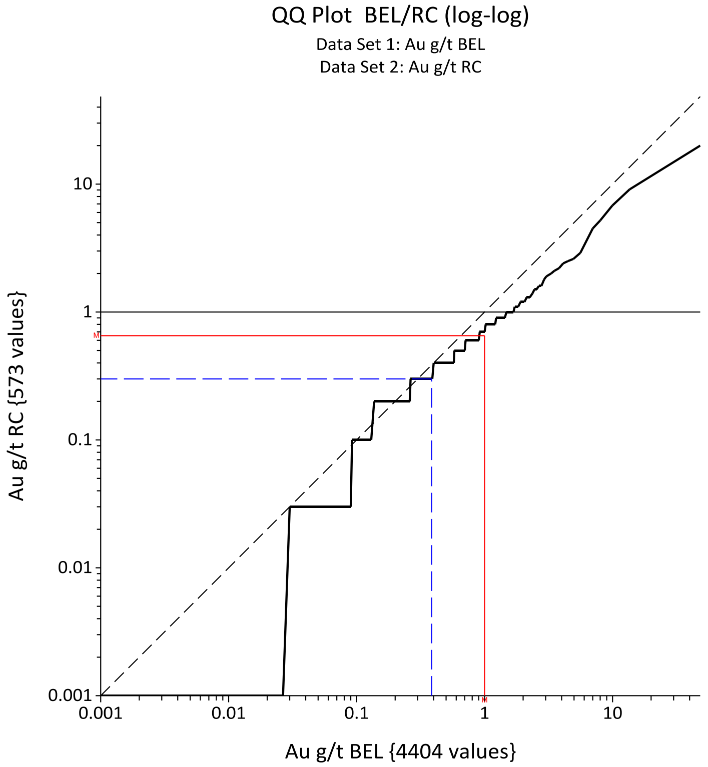

Figure 32 QQ plot of Au assays from Belvedere's drilling compared to the GMH RC drilling

Figure 33 shows the Q-Q plot comparing the Belvedere Au assays to the Outokumpu assays. This shows a reasonably close match, although up to about 3 g/t Au the Outokumpu assays tend to be marginally higher than Belvedere assays. Notwithstanding this, the similarity between the two populations would be sufficient to allow the Outokumpu holes to be included in the estimation process without biasing the estimate. However, uncertainties of the exact collar locations of many of the Outokumpu holes, combined with a lack of 3d surveys means there is a fairly low level of confidence in the actual sample positions. In addition, the Outokumpu data have no record of the assay techniques or any QA/QC procedures. For these reasons, the Outokumpu holes are not being included in the actual estimation, although they can be used for validation of the mineral resource estimate and for guiding exploration.

The Q-Q plot comparing the Belvedere Au assays and the Glenmore diamond drilling assays is shown in Figure 34. Like the Outokumpu assays, this shows a reasonable close match, with the GMH assays being of a marginally higher tenor than the Belvedere. As the collar locations have been located in the field, and good records exist of the sampling and assay procedures followed, this data is considered good enough to be included in the estimation process.

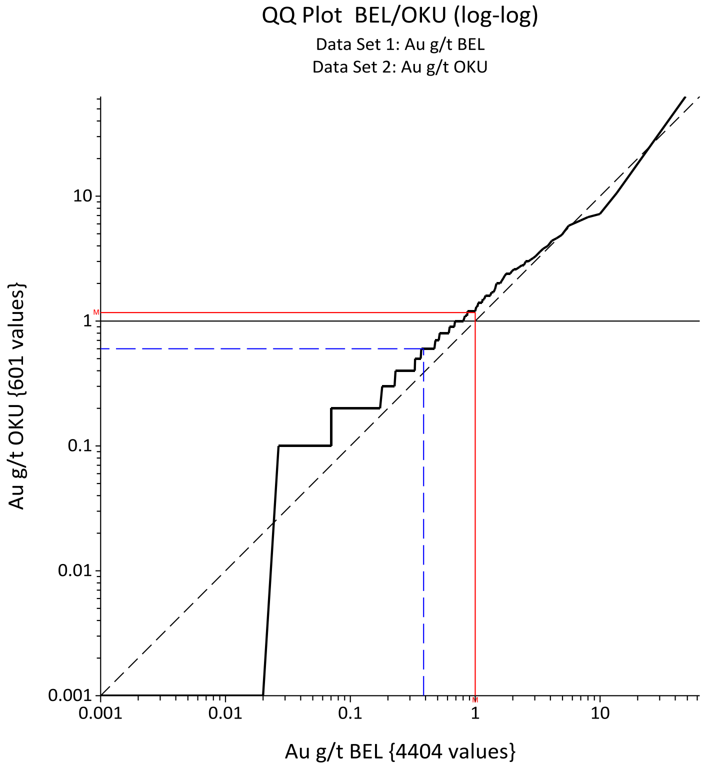

Figure 33 QQ plot of Au assays from Belvedere's drilling compared to the Outokumpu drilling

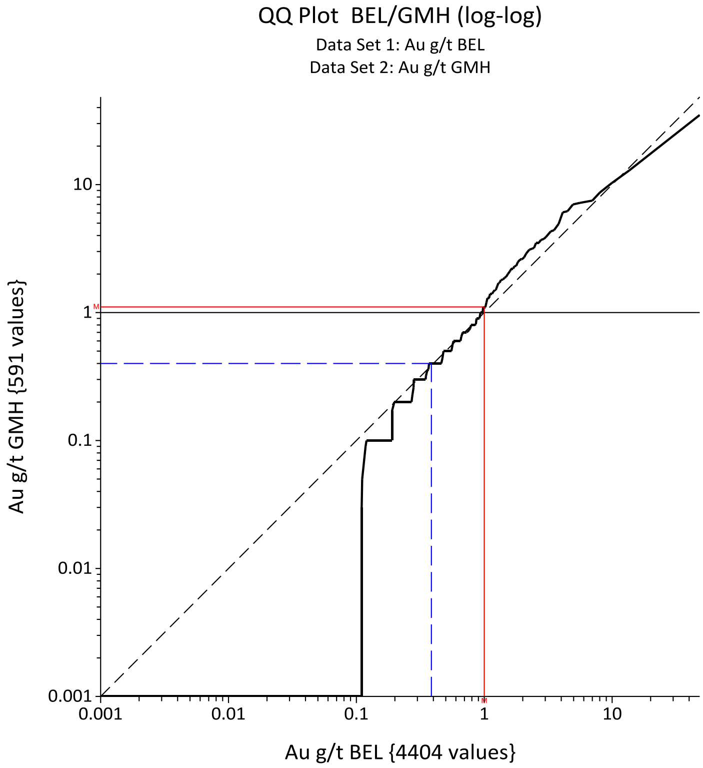

Figure 34 QQ plot of Au assays from Belvedere's drilling compared to the Glenmore Highlands diamond drilling

11 Drilling

Belvedere Resources has carried out six phases of drilling between January 2003 and December 2011. A total of 108 holes were drilled for a total of 10,175.09 down-hole metres (Table 9).

Phase	Year	Holes Drilled	Metres
Phase 1	2003	2	212.85
Phase 2	2003	9	1,045.70
Phase 3	2004	9	497.15
Phase 4	2006 - 2007	12	891.50
Phase 5	2010	31	3,194.98
Phase 6	2011	45	4,332.91
		108	10,175.09

Table 9 Summary of the Belvedere Resources diamond drilling programme

The aim of the drilling programmes has been to expand the known areas of mineralisation, to verify and infill the historical drilling, and to define the mineralisation within the main zone on a density sufficient for an Indicated Resource estimate.

11.1 Phase 1 Drilling – 2003

Belvedere's Phase 1 drilling programme at Kopsa was completed between 16th January, 2003 and 27th January, 2003. A total of 2 diamond drill holes (BELKOPDD 001 - 002) were drilled for a total of 212.85 down-hole meters (Table 10 and Figure 35).

Hole	Easting	Northing	Elevation	Depth	Azimuth	Dip	Year	Core mm	3D Survey
BELKOPDD001	2561450	7075139	112.94	108.65	270	45	2003	61.7	None
BELKOPDD002	2561410	7075191	113.14	104.20	270	45	2003	61.7	None

Table 10 Summary of the Belvedere Resources Phase 1 diamond drilling at Kopsa

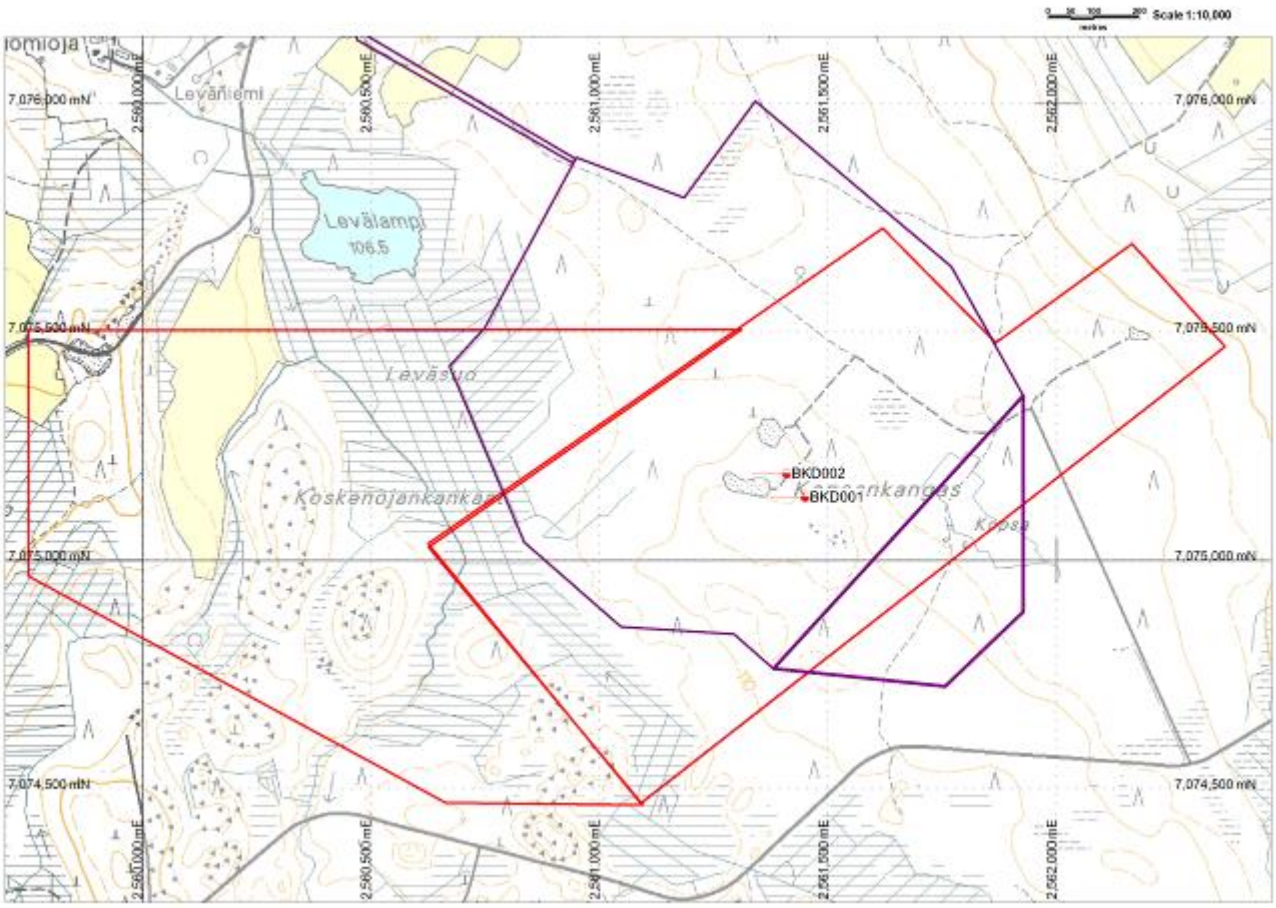

Figure 35 Location map for the Belvedere Resources Phase 1 drilling

11.1.1 Objectives

The aim of the first phase of drilling at Kopsa was to gain an understanding of the principle structural controls of the mineralisation, so as to optimise drilling.

11.1.2 Results

The results from Belvedere's Phase 1 drilling programme are summarised below in Table 11

Hole	From	To	Interval	Au ppm	Cu ppm	As ppm	Grade*thickness (gm)
BELKOPDD001	27.90	62.80	34.90	2.57	1305	7220	90
BELKOPDD001	76.00	81.55	5.55	0.63	947	3546	3
BELKOPDD001	91.60	100.65	9.05	1.26	1344	4266	11
BELKOPDD002	13.10	81.80	68.70	1.17	1971	4347	81

Table 11 Highlights of Belvedere Phase 1 drilling results. Parameters for compositing were 0.5g/t Au cut-off, 7m at 0.0 g/t Au internal dilution. No top cut. Intervals shown are those with gradethickness greater than 2 gram.metres

11.2 Phase 2 Drilling – 2003

Belvedere's Phase 2 drilling programme at Kopsa was completed between 10th June, 2003 and 10th July, 2003. A total of 9 diamond drill holes (BELKOPDD 003 - 011) were drilled for a total of 1,045.70 down-hole meters (Table 12 and Figure 36)

Hole	Easting	Northing	Elevation	Depth	Azimuth	Dip	Year	Core mm	3D Survey
BELKOPDD003	2561350	7075242	111.21	110.50	90	45	2003	61.7	None
BELKOPDD004	2561339	7075291	109.83	99.80	90	45	2003	57.5	None
BELKOPDD005	2561352	7075341	108.37	180.70	90	45	2003	57.5	None
BELKOPDD006	2561400	7075340	107.86	147.40	90	45	2003	57.5	None
BELKOPDD007	2561402	7075089	113.66	140.70	90	45	2003	57.5	None
BELKOPDD008	2561430	7075139	113.38	102.20	90	45	2003	57.5	None
BELKOPDD009	2561400	7075190	113.12	100.30	90	45	2003	57.5	None
BELKOPDD010	2561504	7075421	106.04	52.40	90	45	2003	57.5	None
BELKOPDD011	2561476	7075340	106.68	111.70	90	45	2003	57.5	None

Table 12 Summary of the Belvedere Resources Phase 2 diamond drilling at Kopsa

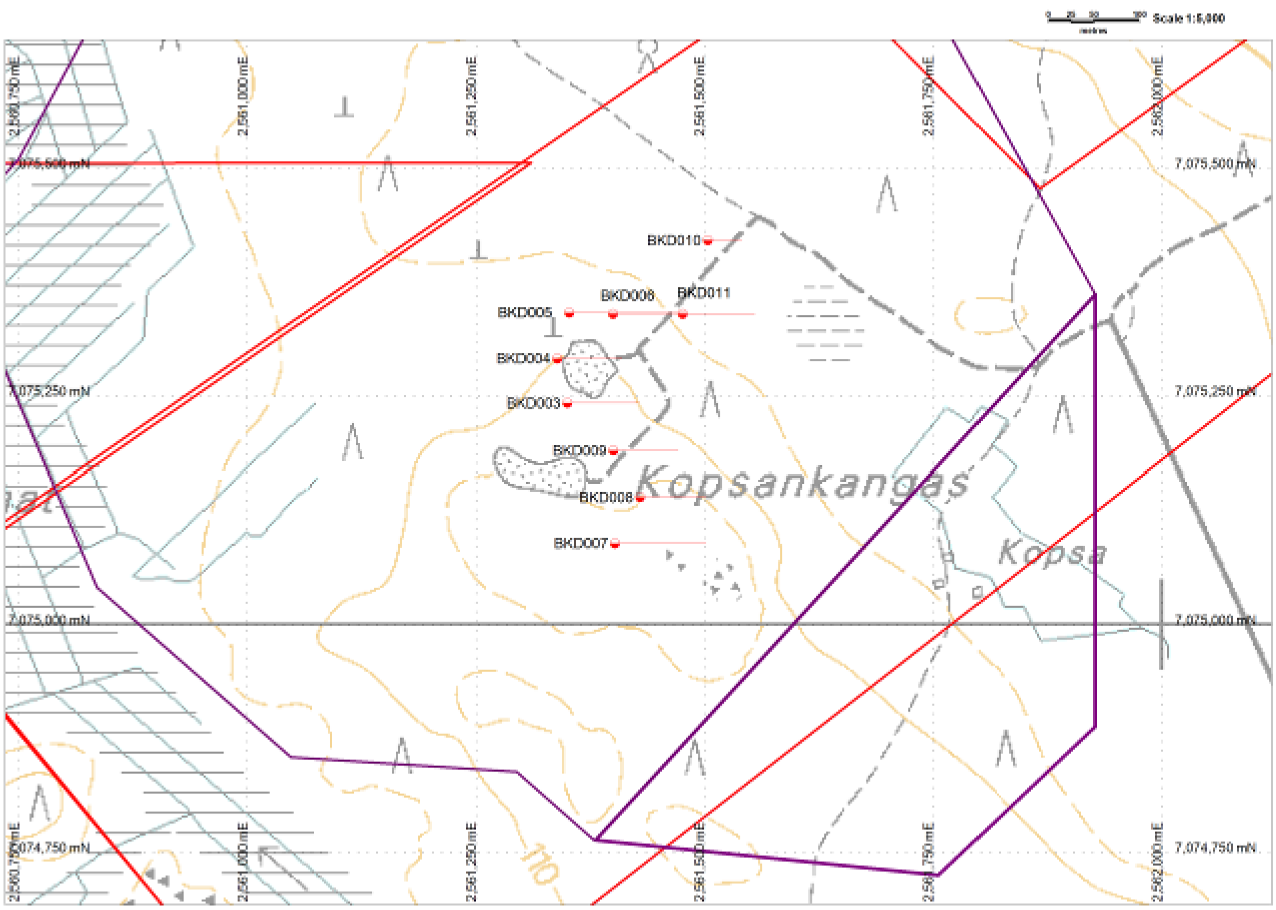

Figure 36 Location map for the Belvedere Resources Phase 2 drilling

11.2.1 Objectives

The aim of the second phase of drilling was to extend the higher grade gold copper mineralisation outlined in historical drilling, 300 metres northwards along the co-incident geophysical induced polarisation ("IP") anomaly to the northern margin of the intrusion. Two other strong IP anomalies within the Kopsa intrusion, previously un-drilled, were to be tested by drilling for near surface gold copper mineralization.

Holes BELKOPDD 003, 004, 005, 006, 011 and 010 were drilled on 50 metre sections north from the central zone with 005, 006 & 011 being drilled on the same section in the northern IP anomaly. Holes 007, 008 & 009 were aimed at extending the central zone mineralisation south (Hole 007) and eastwards (Hole 008 & 009). The two other strong IP anomalies within the Kopsa intrusion to the east and south of the central zone were not drill tested during this campaign.

11.2.2 Results

The results from Belvedere's Phase 2 drilling programme are summarised below in Table 13

Hole	From	To	Interval	Au ppm	Cu ppm	As ppm	Grade*thickness (gm)
BELKOPDD003	59.00	65.10	6.10	1.57	1675	7495	10
BELKOPDD004	11.80	18.55	6.75	0.62	617	5096	4
BELKOPDD005	96.25	105.30	9.05	1.49	658	6472	13
BELKOPDD005	151.40	157.40	6.00	0.58	1517	7690	3
BELKOPDD006	86.40	105.00	18.60	1.57	1124	21070	29
BELKOPDD007	8.20	17.30	9.10	1.42	597	6171	13
BELKOPDD007	23.80	30.10	6.30	0.73	1246	3855	5
BELKOPDD007	82.10	138.40	56.30	0.75	1452	5313	42
BELKOPDD008	30.40	81.80	51.40	1.05	1399	5996	54
BELKOPDD009	6.40	53.50	47.10	0.98	1163	6630	46
BELKOPDD009	66.30	75.40	9.10	1.17	1964	11783	11

Table 13 Highlights of Belvedere Phase 2 drilling results. Parameters for compositing were 0.5g/t Au cut-off, 7m at 0.0 g/t Au internal dilution. No top cut. Intervals shown are those with gradethickness greater than 2 gram.metres

11.3 Phase 3 Drilling – 2004

Belvedere's Phase 3 drilling programme at Kopsa was completed between 27th September, 2004 and 25th October, 2004. A total of 9 diamond drill holes (BELKOPDD 012-BELKOPDD 020) were drilled for a total of 497.15 down-hole meters (Table 14 and Figure 37).

Hole	Easting	Northing	Elevation	Depth	Azimuth	Dip	Year	Core mm	3D Survey
BELKOPDD012	2560277	7075178	114.84	78.50	45	45	2004	57.5	None
BELKOPDD013	2560379	7075167	112.71	30.50	45	45	2004	57.5	None
BELKOPDD014	2561380	7074859	112.92	54.20	45	45	2004	57.5	None
BELKOPDD015	2561460	7075087	113.92	88.00	60	45	2004	57.5	None
BELKOPDD016	2561500	7075115	112.31	79.00	60	45	2004	57.5	None
BELKOPDD017	2561648	7075153	106.94	52.00	60	45	2004	57.5	None
BELKOPDD018	2561510	7075406	105.46	28.75	60	45	2004	57.5	None
BELKOPDD019	2561506	7075386	105.88	31.80	60	45	2004	57.5	None
BELKOPDD020	2561905	7075323	106.88	54.40	60	45	2004	57.5	None

Table 14 Summary of the Belvedere Resources Phase 3 diamond drilling at Kopsa

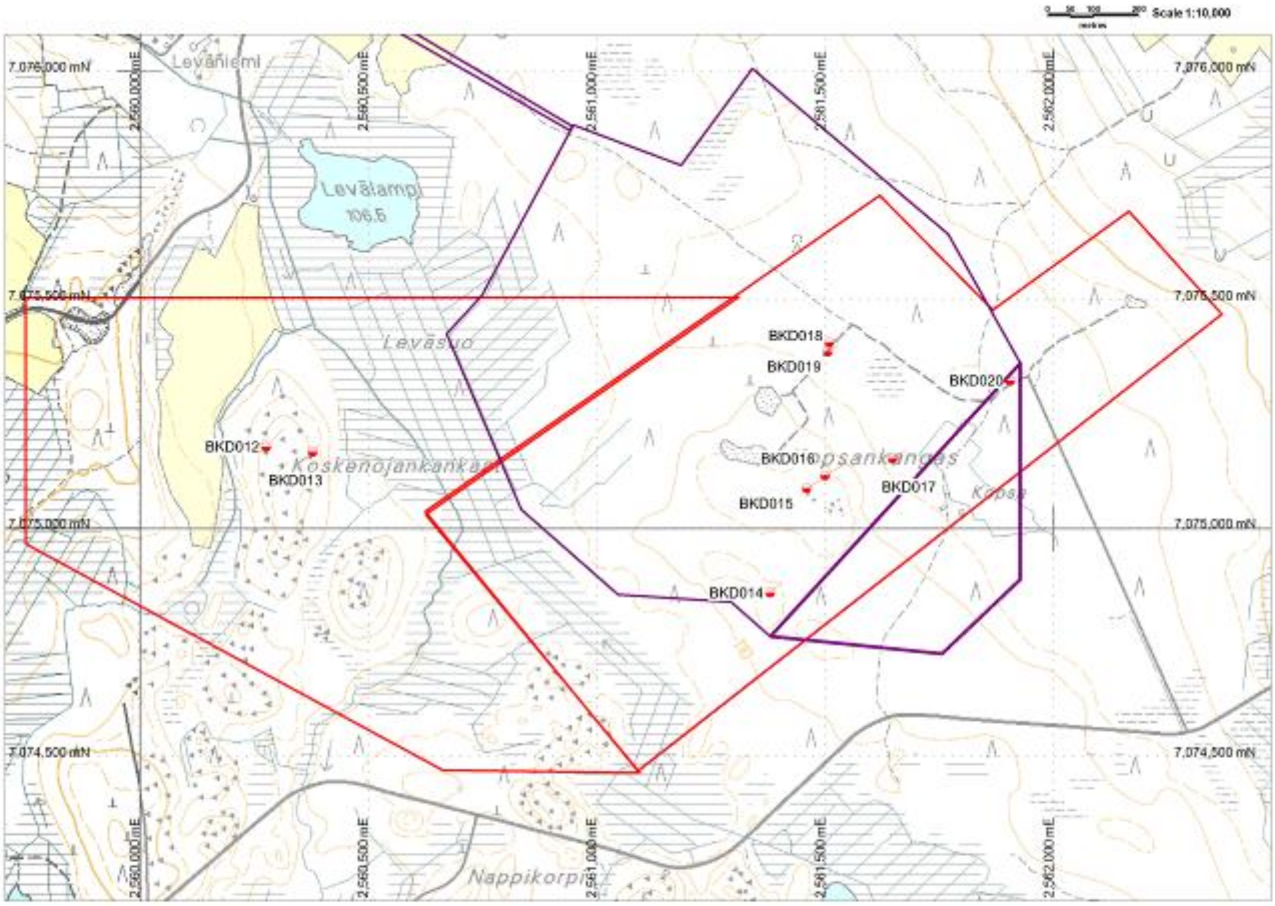

Figure 37 Location map for the Belvedere Resources Phase 3 drilling

11.3.1 Objectives

The purpose of the third phase of drilling was to extend the main zone mineralisation 150 metres to the east and to evaluate regional geochemical and geophysical target areas.

11.3.2 Results

The results from Belvedere's Phase 3 drilling programme are summarised below in Table 15

Hole	From	To	Interval	Au ppm	Cu ppm	As ppm	Grade*thickness (gm)
BELKOPDD015	23.75	85.80	62.05	1.29	1002	6096	80
BELKOPDD016	35.90	73.05	37.15	1.02	668	5965	38
BELKOPDD017	13.40	40.78	27.38	0.60	2088	5312	17

Table 15 Highlights of Belvedere Phase 3 drilling results. Parameters for compositing were 0.5g/t Au cut-off, 7m at 0.0 g/t Au internal dilution. No top cut. Intervals shown are those with gradethickness greater than 2 gram.metres

11.4 Phase 4 Drilling – 2006/2007

Belvedere's Phase 4 drilling programme at Kopsa was completed between 27th November, 2006 and 15th January, 2007. A total of 12 diamond drill holes (BELKOPDD 021-BELKOPDD 032) were drilled for a total of 891.50 down-hole meters (Table 16 and Figure 38).

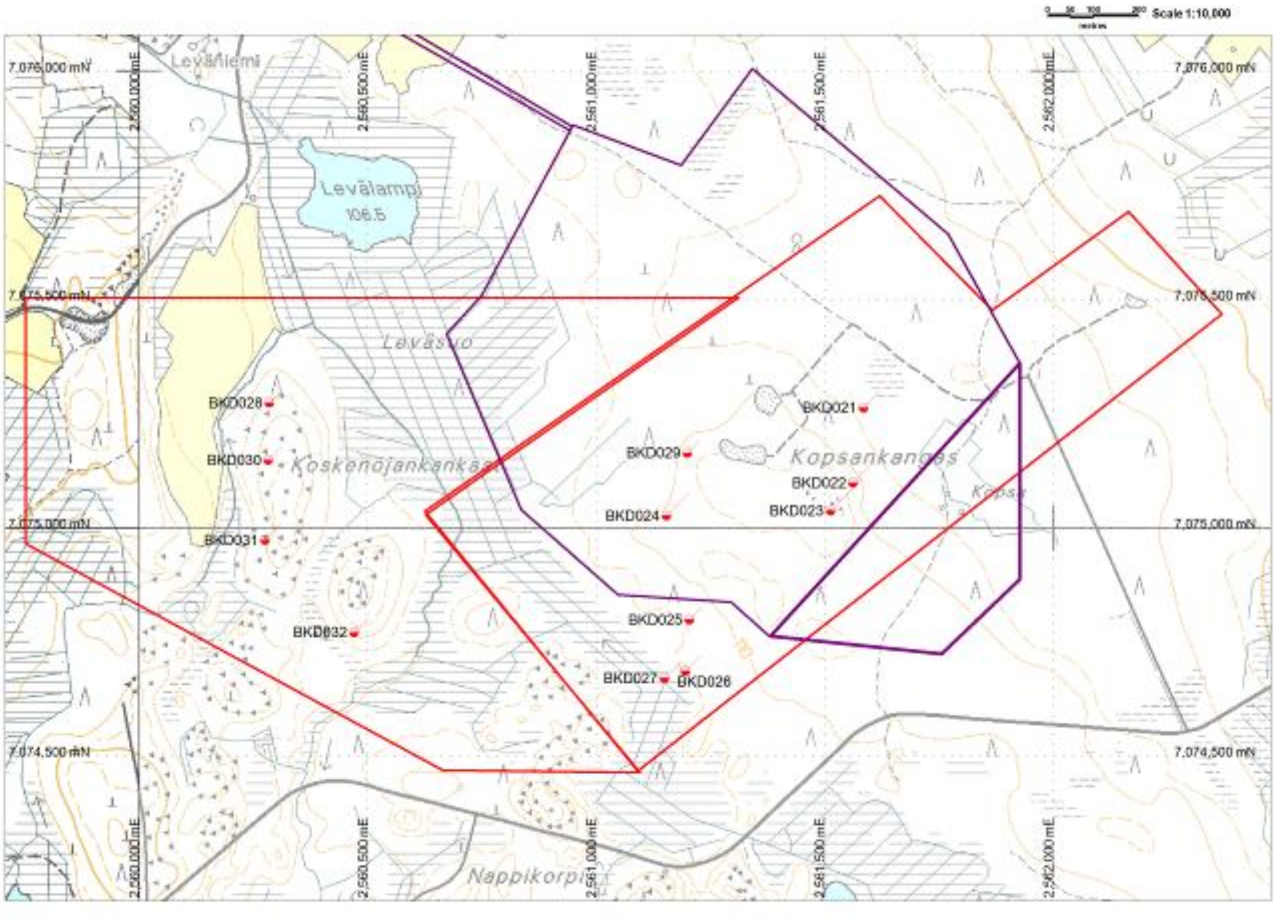

Figure 38 Location map for the Belvedere Resources Phase 4 drilling

Hole	Easting	Northing	Elevation	Depth	Azimuth	Dip	Year	Core mm	3D Survey
BELKOPDD021	2561585	7075264	106.36	69.35	45	45	2006	50.5	EMS
BELKOPDD022	2561562	7075099	110.27	91.40	45	45	2006	50.5	EMS
BELKOPDD023	2561513	7075038	113.73	139.40	45	45	2006	50.5	EMS
BELKOPDD024	2561153	7075027	109.64	100.50	45	45	2006	50.5	EMS
BELKOPDD025	2561203	7074800	109.11	52.75	45	45	2006	50.5	EMS
BELKOPDD026	2561195	7074687	107.75	68.00	45	45	2006	50.5	EMS
BELKOPDD027	2561150	7074672	106.80	71.95	45	45	2006	50.5	EMS
BELKOPDD028	2560285	7075275	110.95	51.95	45	45	2006	50.5	EMS
BELKOPDD029	2561200	7075165	112.46	56.55	45	45	2006	50.5	EMS
BELKOPDD030	2560282	7075149	115.16	55.90	45	45	2007	50.5	EMS
BELKOPDD031	2560275	7074974	114.47	70.00	45	45	2007	50.5	EMS
BELKOPDD032	2560470	7074772	113.64	64.05	45	45	2007	50.5	EMS

Table 16 Summary of the Belvedere Resources Phase 4 diamond drilling at Kopsa

11.4.1 Objectives

This drill program at Kopsa was designed to continue the success of the previous drilling campaign in 2004 in extending the “Main zone” mineralisation along strike and down-dip. Other drill targets at Kopsa included infill drilling along the “North zone”. The potential of this zone became apparent after integration of recently discovered historical data from the 1960’s with 3d modelling of the dataset. In addition the “South zone” was planned to be tested which had combined geophysical and geochemical anomalies, with similar characteristics to the “Main zone”. This anomaly covers a 500×200 metre area and had never been drill tested. The program was also designed to test regional geochemical and geophysical anomalies both within the main part of the Kopsa intrusion, and in structurally removed parts of the intrusion. BELKOPDD021 targeted the North zone, and BELKOPDD022 – 023 targeted extensions of the Kopsa main zone.

11.4.2 Results

The results from Belvedere’s Phase 4 drilling programme are summarised below in Table 17.

Hole	From	To	Interval	Au ppm	Cu ppm	As ppm	Grade*thickness (gm)
BELKOPDD021	45.75	55.19	9.44	0.83	4142	2189	8
BELKOPDD022	30.92	44.36	13.44	0.80	1180	4000	11
BELKOPDD022	56.00	86.31	30.31	1.49	1589	4202	45
BELKOPDD023	62.72	84.60	21.88	0.68	803	3530	15
BELKOPDD023	100.55	113.08	12.53	1.33	2139	5043	17
BELKOPDD029	20.75	35.27	14.52	1.32	2510	2849	19

Table 17 Highlights of Belvedere Phase 4 drilling results. Parameters for compositing were 0.5g/t Au cut-off, 7m at 0.0 g/t Au internal dilution. No top cut. Intervals shown are those with gradethickness greater than 2 gram.metres

11.5 Phase 5 Drilling – 2010

Belvedere's Phase 5 drilling programme at Kopsa was completed between March 3, 2010 and June 14, 2010. A total of 31 diamond drill holes (BELKOPDD 033-BELKOPDD 063) were drilled for a total of 3,194.70 down-hole meters (Table 18 and Figure 39).

Hole	Easting	Northing	Elevation	Depth	Azimuth	Dip	Year	Core mm	3D Survey
BELKOPDD033	2561184	7075183	112.19	43.40	20	45	2010	39.0	Deviflex
BELKOPDD034	2561163	7075127	111.22	87.74	20	45	2010	39.0	Deviflex
BELKOPDD035	2561070	7075012	109.25	68.10	20	45	2010	39.0	Deviflex
BELKOPDD036	2561213	7075117	112.86	118.45	20	45	2010	39.0	Deviflex
BELKOPDD037	2561188	7075056	110.12	145.60	20	45	2010	39.0	Deviflex
BELKOPDD038	2561300	7075207	111.53	85.14	20	45	2010	39.0	Deviflex
BELKOPDD039	2561281	7075158	113.14	100.30	20	45	2010	39.0	Deviflex
BELKOPDD040	2561267	7075121	113.85	100.70	20	60	2010	39.0	Deviflex
BELKOPDD041	2561368	7075249	111.15	76.70	20	45	2010	39.0	Deviflex
BELKOPDD042	2561350	7075200	112.40	115.40	20	45	2010	39.0	Deviflex
BELKOPDD043	2561326	7075136	113.18	133.60	20	45	2010	39.0	Deviflex
BELKOPDD044	2561458	7075226	109.63	50.20	20	45	2010	39.0	Deviflex
BELKOPDD045	2561445	7075164	113.06	81.00	20	45	2010	39.0	Deviflex
BELKOPDD046	2561389	7075016	113.52	181.01	20	45	2010	39.0	Deviflex
BELKOPDD047	2561309	7075089	113.85	172.10	20	45	2010	39.0	Gyrosmart
BELKOPDD048	2561484	7075130	112.59	91.70	20	45	2010	39.0	Gyrosmart
BELKOPDD049	2561443	7075014	113.19	180.10	20	45	2010	39.0	Gyrosmart
BELKOPDD050	2561507	7075195	109.06	83.50	20	45	2010	39.0	Deviflex
BELKOPDD051	2561559	7075189	107.56	86.80	20	45	2010	39.0	Deviflex
BELKOPDD052	2561543	7075295	106.44	54.50	20	45	2010	39.0	Deviflex
BELKOPDD053	2561600	7075159	107.72	58.74	20	45	2010	39.0	Gyrosmart
BELKOPDD054	2561627	7075084	108.69	94.60	20	45	2010	39.0	Gyrosmart
BELKOPDD055	2561603	7075027	110.20	112.00	20	45	2010	39.0	Deviflex
BELKOPDD056	2561661	7075026	108.70	119.00	20	45	2010	39.0	Deviflex
BELKOPDD057	2561494	7075002	113.04	160.80	20	45	2010	39.0	Gyrosmart
BELKOPDD058	2561602	7074720	112.25	119.00	20	45	2010	39.0	Deviflex
BELKOPDD059	2561743	7075252	105.92	57.60	20	45	2010	39.0	Deviflex
BELKOPDD060	2561410	7075093	113.35	100.30	20	45	2010	39.0	Deviflex
BELKOPDD061	2561356	7075146	112.13	95.00	20	45	2010	39.0	Deviflex
BELKOPDD062	2561307	7075144	112.80	100.10	20	45	2010	39.0	Deviflex
BELKOPDD063	2561253	7075152	113.21	121.80	20	45	2010	39.0	Deviflex

Table 18 Summary of the Belvedere Resources Phase 5 diamond drilling at Kopsa

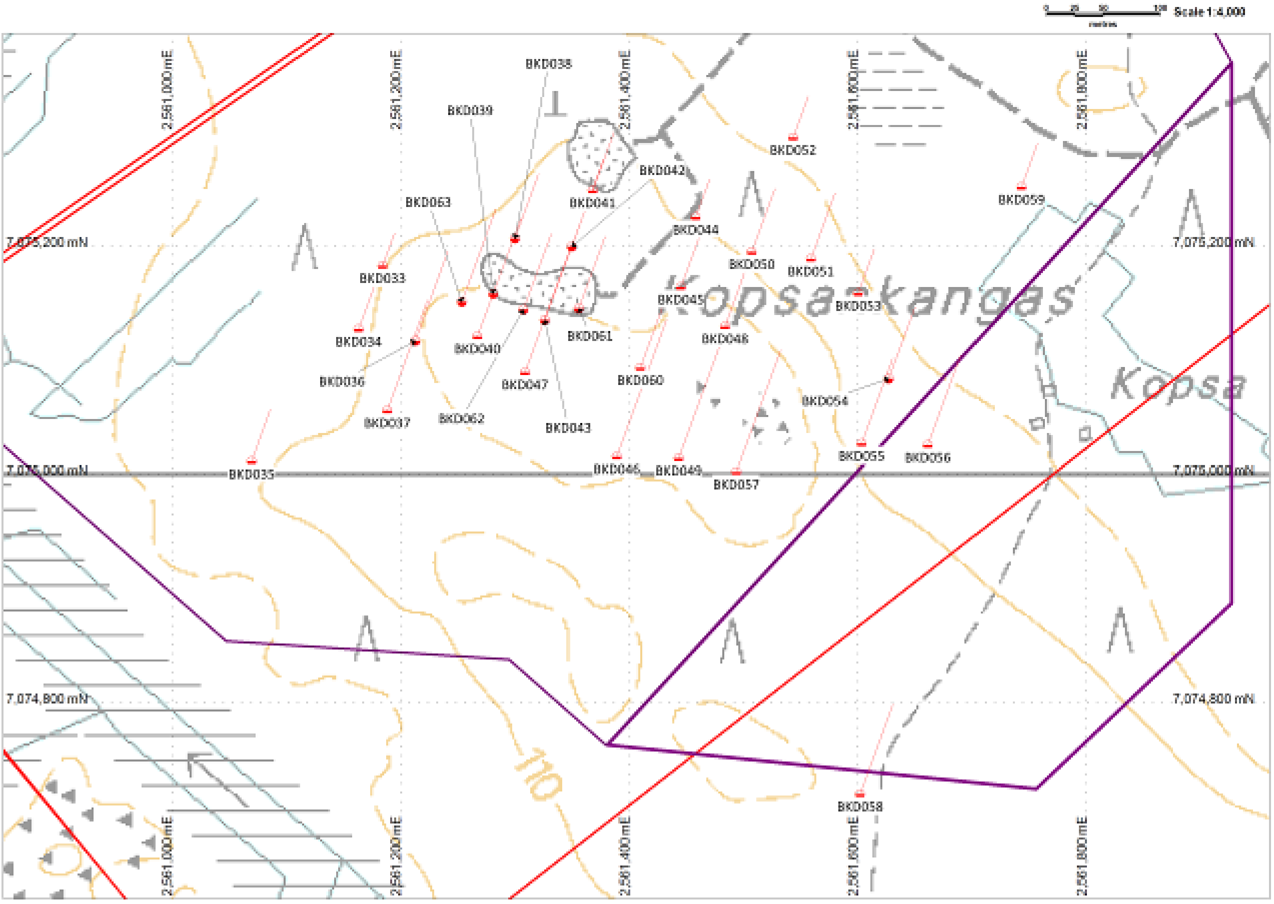

Figure 39 Location map for the Belvedere Resources Phase 5 drilling

11.5.1 Objectives

The main purpose for the Phase 5 drilling programme was to infill between existing drill holes in the Main Zone to improve confidence in the geological model and to some extent expand the resource on strike and at depth.

11.5.2 Results

The results from Belvedere's Phase 5 drilling programme are summarised below in Table 19.

Hole	From	To	Interval	Au ppm	Cu ppm	As ppm	Grade*thickness (gm)
BELKOPDD033	11.10	26.97	15.87	0.87	2671	4903	14
BELKOPDD034	41.53	64.78	23.25	1.41	2118	9788	33
BELKOPDD035	41.27	49.45	8.18	1.15	2314	3078	9
BELKOPDD037	52.48	59.64	7.16	2.28	762	12540	16
BELKOPDD037	97.10	115.27	18.17	1.32	2176	4557	24
BELKOPDD037	118.84	124.25	5.41	0.61	623	5664	3
BELKOPDD037	132.00	142.59	10.59	0.58	2486	3292	6
BELKOPDD038	15.41	24.03	8.62	0.87	1122	4249	8
BELKOPDD038	25.32	41.57	16.25	0.51	1522	2257	8
BELKOPDD039	25.28	77.00	51.72	0.98	2250	4013	51
BELKOPDD040	56.47	99.06	42.59	0.67	1383	4647	29
BELKOPDD041	14.80	32.01	17.21	0.81	1932	4383	14
BELKOPDD042	33.76	47.13	13.37	1.64	1059	11003	22
BELKOPDD043	12.90	100.53	87.63	3.31	2624	9660	290
BELKOPDD045	8.92	33.97	25.05	1.64	1110	10932	41
BELKOPDD046	37.52	44.60	7.08	1.92	715	5047	14
BELKOPDD046	104.85	126.83	21.98	0.80	774	2554	18
BELKOPDD047	33.77	46.92	13.15	0.83	651	1891	11
BELKOPDD048	5.77	52.81	47.04	1.02	1069	8868	48
BELKOPDD048	71.68	82.88	11.20	6.85	1119	59226	77
BELKOPDD049	60.56	66.75	6.19	1.55	1396	21695	10
BELKOPDD049	82.20	106.48	24.28	1.17	1821	6807	28
BELKOPDD049	122.46	132.03	9.57	0.87	1946	4010	8
BELKOPDD052	33.50	40.21	6.71	1.56	2289	33319	10
BELKOPDD053	16.27	25.85	9.58	1.89	1223	13342	18
BELKOPDD054	53.93	66.39	12.46	0.78	2845	9954	10
BELKOPDD055	48.35	59.66	11.31	1.41	1794	6387	16
BELKOPDD055	80.66	92.92	12.26	0.78	2132	10039	10
BELKOPDD056	55.50	68.98	13.48	1.16	2645	7426	16
BELKOPDD057	104.48	129.06	24.58	1.02	928	8312	25
BELKOPDD057	150.50	157.15	6.65	1.02	1537	5155	7
BELKOPDD060	30.07	74.22	44.15	2.72	1386	11302	120
BELKOPDD061	29.84	59.61	29.77	0.78	2353	2714	23
BELKOPDD061	73.48	79.68	6.20	0.89	2070	3859	6
BELKOPDD062	26.03	69.02	42.99	1.31	2511	5725	56
BELKOPDD063	40.42	79.80	39.38	1.07	1614	5722	42

Table 19 Highlights of Belvedere Phase 5 drilling results. Parameters for compositing were 0.5g/t Au cut-off, 7m at 0.0 g/t Au internal dilution. No top cut. Intervals shown are those with gradethickness greater than 2 gram.metres

11.6 Phase 6 Drilling – 2011

Belvedere's Phase 6 drilling programme at Kopsa was completed between 1st August, 2011 and 4th December, 2011. A total of 45 diamond drill holes (BELKOPDD 064-BELKOPDD 106) were drilled for a total of 4,332.91 down-hole meters (Table 18 and Figure 39).

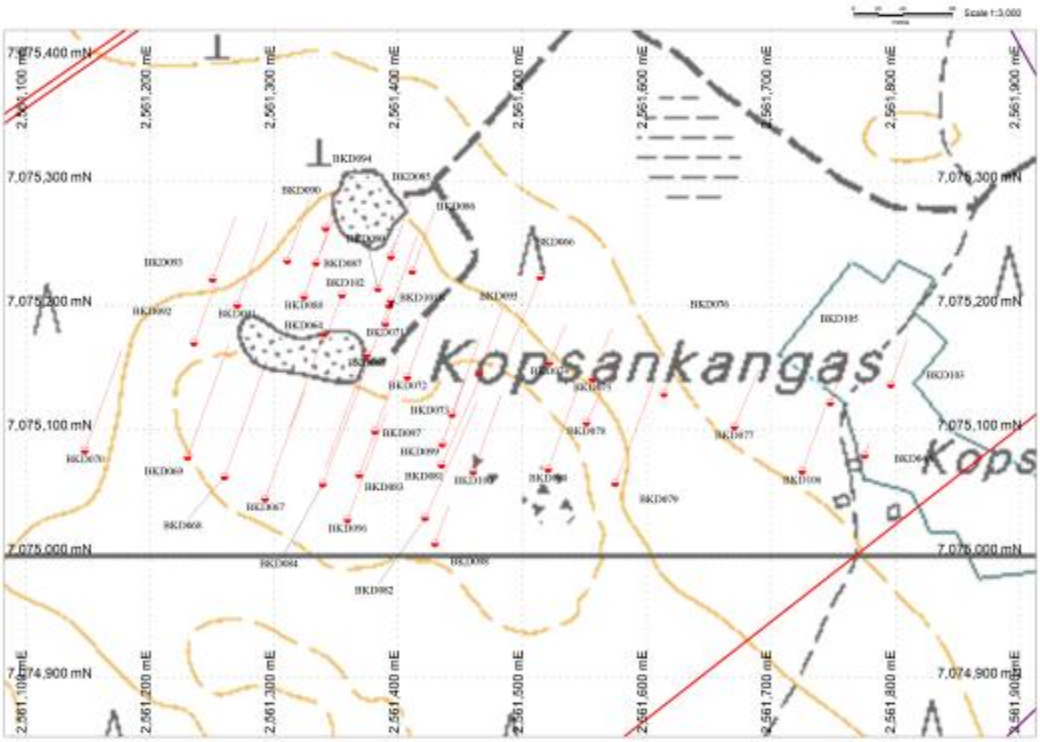

Figure 40 Location map for the Belvedere Resources Phase 6 drilling

11.6.1 Objectives

The main purpose for the Phase 6 drilling programme was to infill between existing drill holes in the Main Zone to improve confidence in the geological model and to some extent expand the resource on strike and at depth.

11.6.2 Results

The results from Belvedere's Phase 6 drilling programme are summarised below in Table 19.

Hole	Easting	Northing	Elevation	Depth	Azimuth	Dip	Year	Core mm	3d Survey
BELKOPDD064	2561340	7075177	112.49	125.50	200	50	2011	42	Reflex Gyro
BELKOPDD065	2561375	7075161	112.40	149.05	200	50	2011	42	Reflex Gyro
BELKOPDD066	2561514	7075223	107.63	104.75	203	50	2011	42	Reflex Gyro
BELKOPDD067	2561293	7075044	113.74	143.69	20	50	2011	42	Reflex Gyro
BELKOPDD068	2561260	7075062	113.20	155.39	20	50	2011	42	Reflex Gyro
BELKOPDD069	2561230	7075078	113.20	149.40	20	50	2011	42	Reflex Gyro
BELKOPDD070	2561147	7075083	109.29	134.55	20	50	2011	42	Reflex Gyro
BELKOPDD071	2561389	7075186	112.95	107.65	20	45	2011	42	Reflex Gyro
BELKOPDD072	2561407	7075143	112.55	86.70	20	50	2011	42	Reflex Gyro
BELKOPDD073	2561443	7075112	112.97	102.00	20	50	2011	42	Reflex Gyro
BELKOPDD074	2561522	7075154	110.30	65.95	23	60	2011	42	Reflex Gyro
BELKOPDD075	2561556	7075141	109.20	68.30	21	50	2011	42	None
BELKOPDD076	2561614	7075129	107.87	70.80	22	50	2011	42	None
BELKOPDD077	2561671	7075102	107.35	76.73	23	50	2011	42	None
BELKOPDD078	2561551	7075106	110.59	86.05	25	50	2011	42	Reflex Gyro
BELKOPDD079	2561574	7075057	110.61	143.47	21	50	2011	42	Reflex Gyro
BELKOPDD080	2561521	7075068	113.32	117.02	20	50	2011	42	Reflex Gyro
BELKOPDD081	2561435	7075072	113.61	128.96	24	50	2011	42	Reflex Gyro
BELKOPDD082	2561422	7075029	114.11	159.05	24	50	2011	42	None
BELKOPDD083	2561368	7075064	113.65	119.90	20	50	2011	42	Reflex Gyro
BELKOPDD084	2561339	7075056	113.17	131.60	21	50	2011	42	Reflex Gyro
BELKOPDD085	2561394	7075240	111.08	74.70	20	50	2011	42	Reflex Gyro
BELKOPDD086	2561411	7075228	111.70	81.00	20	50	2011	42	Reflex Gyro
BELKOPDD087	2561334	7075235	111.00	68.65	22	50	2011	42	Reflex Gyro
BELKOPDD088	2561324	7075207	111.88	95.30	20	50	2011	42	Reflex Gyro
BELKOPDD089	2561383	7075214	112.58	62.70	20	50	2011	42	Reflex Gyro
BELKOPDD090	2561310	7075236	111.25	83.60	20	50	2011	42	None
BELKOPDD091	2561270	7075201	112.07	110.50	20	50	2011	42	Reflex Gyro
BELKOPDD092	2561235	7075170	113.30	80.95	20	50	2011	42	Reflex Gyro
BELKOPDD093	2561250	7075221	111.15	83.75	20	50	2011	42	Reflex Gyro
BELKOPDD094	2561341	7075263	110.95	59.65	20	45	2011	42	Reflex Gyro
BELKOPDD095	2561465	7075145	113.05	89.62	20	50	2011	42	Reflex Gyro
BELKOPDD096	2561359	7075027	112.65	143.45	20	50	2011	42	Reflex Gyro
BELKOPDD097	2561381	7075099	113.09	86.40	20	50	2011	42	None
BELKOPDD098	2561429	7075008	112.82	50.50	20	50	2011	42	Reflex Gyro
BELKOPDD099	2561435	7075089	112.65	86.40	20	50	2011	42	Reflex Gyro
BELKOPDD100	2561460	7075066	114.03	101.65	20	50	2011	42	None
BELKOPDD101	2561394	7075204	112.00	82.58	200	45	2011	42	None
BELKOPDD101B	2561392	7075200	112.99	101.35	200	45	2011	42	Reflex Gyro
BELKOPDD102	2561354	7075209	112.23	77.40	200	60	2011	42	Reflex Gyro
BELKOPDD103	2561796	7075136	105.68	55.45	20	45	2011	42	None
BELKOPDD104	2561775	7075079	106.27	83.40	20	45	2011	42	None
BELKOPDD104A	2561775	7075079	106.27	13.50	20	45	2011	42	None
BELKOPDD105	2561748	7075122	106.20	59.60	20	45	2011	42	None
BELKOPDD106	2561725	7075067	106.97	74.30	20	45	2011	42	None

Table 20 Summary of the Belvedere Resources Phase 6 diamond drilling at Kopsa

Hole	From	To	Interval	Au ppm	Cu ppm	As ppm	Grade*thickness (gm)
BELKOPDD064	15.15	50.91	35.76	1.28	1741	3297	46
BELKOPDD064	58.00	67.37	9.37	1.67	1595	3453	16
BELKOPDD065	32.04	52.36	20.32	4.61	3618	5494	94
BELKOPDD067	107.52	113.36	5.84	1.08	1031	5094	6
BELKOPDD068	24.67	34.17	9.50	1.22	2267	2246	12
BELKOPDD069	48.97	58.32	9.35	1.47	2794	6007	14
BELKOPDD069	88.05	95.73	7.68	0.90	855	2625	7
BELKOPDD069	128.31	136.78	8.47	0.73	1688	6149	6
BELKOPDD071	4.86	25.83	20.97	0.91	1931	4262	19
BELKOPDD072	24.40	47.71	23.31	2.51	2837	7241	59
BELKOPDD073	39.71	56.87	17.16	1.05	1674	7239	18
BELKOPDD074	8.77	27.36	18.59	1.31	1060	12055	24
BELKOPDD074	40.07	60.05	19.98	1.94	335	3276	39
BELKOPDD075	23.25	41.30	18.05	1.00	1013	9550	18
BELKOPDD078	47.39	69.46	22.07	1.18	798	8603	26
BELKOPDD079	38.13	81.00	42.87	0.79	1719	6520	34
BELKOPDD079	86.12	134.19	48.07	0.52	1551	3873	25
BELKOPDD080	49.22	92.90	43.68	1.03	890	4695	45
BELKOPDD081	24.76	30.71	5.95	1.15	987	8450	7
BELKOPDD081	44.15	91.46	47.31	0.73	1255	4676	34
BELKOPDD082	8.62	21.36	12.74	2.79	801	6660	36
BELKOPDD082	86.62	108.30	21.68	1.23	2520	3727	27
BELKOPDD083	56.90	79.54	22.64	1.85	964	4062	42
BELKOPDD084	58.68	81.95	23.27	0.97	1458	4033	22
BELKOPDD084	84.82	96.92	12.10	0.59	1387	2623	7
BELKOPDD085	18.39	34.41	16.02	1.32	970	4186	21
BELKOPDD086	59.83	65.30	5.47	0.82	1210	7024	5
BELKOPDD088	39.44	47.60	8.16	1.30	2556	5806	11
BELKOPDD089	7.21	12.61	5.40	1.37	2783	6569	7
BELKOPDD090	12.73	22.40	9.67	1.79	1827	4127	17
BELKOPDD091	30.91	45.11	14.20	1.62	2219	5667	23
BELKOPDD091	71.61	80.62	9.01	0.77	1156	6495	7
BELKOPDD092	12.65	59.62	46.97	1.01	1920	3102	48
BELKOPDD095	5.25	51.76	46.51	1.24	1131	7265	58
BELKOPDD096	79.09	84.21	5.12	0.83	589	2311	4
BELKOPDD096	97.31	121.60	24.29	0.88	1732	3708	21
BELKOPDD097	29.90	58.87	28.97	1.16	2139	4006	34
BELKOPDD097	70.70	83.20	12.50	1.04	1091	6058	13
BELKOPDD099	17.30	81.42	64.12	1.01	1192	6771	65
BELKOPDD100	39.74	97.63	57.89	1.82	2351	7298	106
BELKOPDD101	12.27	74.11	61.84	1.84	1813	6365	114
BELKOPDD101B	12.12	73.01	60.89	2.00	1977	6454	122
BELKOPDD102	18.98	73.49	54.51	2.49	1844	4495	135
BELKOPDD106	24.85	38.27	13.42	1.28	3440	8610	17

Table 21 Highlights of Belvedere Phase 6 drilling results. Parameters for compositing were 0.5g/t Au cut-off, 7m at 0.0 g/t Au internal dilution. No top cut. Intervals shown are those with gradethickness greater than 2 gram.metres

11.7 Technical Specifications

11.7.1 Collar Locations

Holes were located using a handheld GPS, and the planned azimuth set up by using a number of foresights and backsights placed using a combination of GPS and a handheld compass. Final collar locations were surveyed by differential GPS at the end of the drilling programme.

11.7.2 Drilling Equipment

Phase 1 drilling was undertaken by Suomen Malmi Oy (SMOY) using a Diamec 260 rig on rubber tracks. Drilling was done using T76 diameter core barrels, producing a hole diameter of 76.3 mm and a core diameter of 61.7 mm.

Phase 2 and Phase 3 drilling were undertaken by OY Kati AB (Kati) using a Diamec 260 rig on rubber tracks. Except for the first hole (BELKOPDD 003) drilling was done using wire line WL76 diameter core barrels, roughly equivalent to HQ producing a hole diameter of 76.30 mm and core diameter of 57.5 mm. BELKOPDD 003 was drilled using a conventional T76 diameter core barrel, producing a hole diameter of 76.3 mm and a core diameter of 61.7 mm.

Phase 4 drilling was undertaken by Suomen Malmi Oy (SMOY) using a Diamec 260 rig on rubber tracks. Drilling was done using wire line BGM diameter core barrels, producing a hole diameter of 56 mm and core diameter of 42 mm.

Phase 5 drilling was undertaken by OY Kati AB (Kati) using a Diamec 262 rig on rubber tracks. Drilling was done using wire line WL56 diameter core barrel, roughly equivalent to BQ producing a hole diameter of 56.80 mm and core diameter of 39.00 mm.

Phase 6 drilling was undertaken by Suomen Malmi Oy (SMOY) using an Onram 1000 rig on rubber tracks. Drilling was done using wire line BGM sized core barrels, producing a hole diameter of 67.10 mm and core diameter of 50.5 mm.

In all cases, the collar casing was left in the holes to enable deepening of the holes at later time or for further surveys (e.g. geophysics) to be carried out.

11.7.3 Downhole Deviation Surveys

In the 4th phase of drilling the dip and azimuth deviation was measured by SMOY, using a Reflex EMS (Electronic Multi Shot) survey tool. This tool utilises variations in the magnetic field. If local magnetic effects become significant, the data produced can be incorrect.

In the 5th phase of drilling, the dip and azimuth variations of all holes were measured using non-magnetic survey tools. Two different instruments were used for the surveys Devico's DeviFlex multishot survey tool and Reflex Gyrosmart (Table 18).

In the 6th phase of drilling, the dip and azimuth variations of all holes (where possible) were measured using the Reflex Gyrosmart non-magnetic survey tool (Table 20).

11.7.4 Core Orientation

During the first four phases of drilling, Belvedere drilled oriented core in order to be able to record detailed structural measurements of various features within the deposit. The purpose of this was to gain a better understanding of the controlling structures, and to confirm that drilling was in the most suitable orientation. In Phase 1, core was oriented using a conventional spear mark orientation system. For Phases 2 and 3 core was oriented using the BallMark core orientation system. For Phase 4 core was oriented using the Ezy-Mark core orientation system.

No core orientation was done during Phases 5 or 6 at Kopsa.

11.7.5 Core Loss

In most places core recovery was one hundred percent. Where core loss occurred it has been recorded into the Assay database. Out of the 8396 samples assayed from Belvedere's drilling, only 536 (6.4%) are recorded as having core loss. The average core loss is 0.16 meters relating to samples with an average interval of 1.12 meters.

Out of the 536 samples with recorded core loss 139 have Au assay values greater than 0.5 g/t, 70 have assay values greater than 1.0 g/t and only 23 have assays greater than 3.0 g/t.

As the amount of core loss in the mineralised samples is relatively minor, no dilution of the sample values has been applied to the intervals.

12 Sampling Method and Approach

The entire drill core was logged and processed by Belvedere Resources' geologists. The logging and sampling procedures as undertaken during the total tenure of Belvedere drilling has been summarised below.

12.1 Core Handling Procedures

At the drilling site, the drillers placed the core into wooden core boxes, with wooden blocks marking drill runs. If any core loss occurs in a drilling run, the extent of this is also marked at the end of each run. Following a brief examination of the core on site by a Belvedere geologist, the core was transported to the Company's secured core processing facilities in Pyhäsalmi, in central Finland.

On arrival, the core is photographed in the core boxes (both wet and dry), and the metre depths marked onto the core (and box) with a wax pencil. The core is then logged by geologists and the lithological, mineralogical, structural, geophysical and rock mechanical (SG, RQD) properties recorded as required. Samples are marked out for assaying (as described below), and cut using a diamond saw. One half was sent for assay, and the other half was retained for verification purposes. All of the remaining core is stored in the marked core boxes, on pallets, at the Company's premises in Pyhäsalmi.

12.2 Sampling Methodology

The main criteria for selecting core for sampling were the presence and intensity of mineralisation, quartz veins, and shear zones. The mineralised portions of the drill cores were sampled regularly and continuously. Only a few check samples were collected from outside the zones containing significant visible mineralisation, veins or shear zones. Within the identified mineralised zones, the sample length was typically about 1 metre (mean = 1.05 m). The maximum and minimum sample lengths were 3.75m and 0.10m respectively.

No apparently high-grade mineralised intersection was sampled in conjunction with low-grade mineralisation and sampling across lithological contacts was avoided.

The poorly-mineralised/macroscopically barren zones between two mineralised zones were sampled if the length of the zone did not exceed 10 m. In case of thicker, apparently barren zones above and below the mineralised zone, about 5 m in each zone was sampled.

The samples were clearly marked with wax pencils to indicate the beginning and ending of the samples. The sample boundaries and sample numbers were marked both on the remaining cut core as well as on the core boxes.

After the sampling was completed, macroscopic quantitative mineralogical assessment of the samples was carried out. This provides an estimate of the volume percent of the ore minerals and vein and free quartz in the sample. The assessment is carried out for all the samples. The method is based on the visual estimation of the total width occupied by a particular mineral of the sample across the diameter of the core in a sample interval. The total width is then divided by the sample length and multiplied by 100 to get the volume percentage of the mineral in the sample.

12.3 Structural Measurements

Oriented core was taken from the first 3 phases of drilling. In the first phase of drilling BELKOPDD001 and BELKOPDD002, core orientation was recorded using a "pencil and spear".

A reference line was marked onto the reconstructed core along the bottom of the core axis. In the second (BELKOPDD003 - 011) and third (BELKOPDD012 - 020) phases of drilling core orientation was recorded using the Ballmark system. The reference line for Phase 2 was along the bottom of the core, but for Phase 3 was marked on the *top* of the core.

In recording the data, the geologist also recorded the level of confidence with all structures relating to a particular drill run. Due to various reasons (poor mark on the core, broken core, fracture perpendicular to the core at the point of orientation mark, poor skill of the driller etc.) the confidence of the orientation mark varied. The level of confidence was determined on the basis of continuity of the reference line from one drill run to the next. If the reference line on a drill run was within ± 15° to both the preceeding drill run and the subsequent drill run, or supported the continuation of a high confidence drill run, then the confidence level was assigned a value of "1". If the core reference line was only supported by only one other (subsequent or preceeding) drill run, the confidence was assigned a value of "2". If there was no support from either of the adjacent drill runs the confidence was assigned a value of "3". A summary of the quantity and quality of structural measurements from oriented core is provided in Table 22.

Hole	Azimuth	Dip	Confidence 1	Confidence 2	Confidence 3	Total
BELKOPDD001	270	45	368	61	34	463
BELKOPDD002	270	45	132	226	51	409
BELKOPDD003	90	45	187	214	40	441
BELKOPDD004	90	45	369	168	4	541
BELKOPDD005	90	45	765	339	38	1142
BELKOPDD006	90	45	413	544	108	1065
BELKOPDD007	90	45	827	380	183	1390
BELKOPDD008	90	45	454	181	2	637
BELKOPDD009	90	45	460	80	0	540
BELKOPDD010	90	45	142	108	0	250
BELKOPDD011	90	45	464	353	74	891
BELKOPDD012	45	45	57	0	9	66
BELKOPDD014	45	45	69	181	0	250
BELKOPDD015	60	45	198	103	0	301
BELKOPDD016	60	45	459	92	0	551
BELKOPDD017	60	45	304	30	17	351
BELKOPDD018	60	45	121	0	20	141
BELKOPDD019	60	45	155	0	1	156
BELKOPDD020	60	45	143	223	1	367
			6087	3283	582	9952

Table 22 Summary of structural readings taken from oriented core

The oriented core was reconstructed on the logging table, and the reference line drawn on the core. Structures were recorded with the alpha angle (measured with respect to core axis) and beta angle (measured with respect to the core reference line). The data was entered into the software "Dips" along with the borehole azimuths and dips and converted into real space dip and dip direction.

In recording structures, the geologist recorded the type of feature (e.g. vein, fracture, foliation, contact) the average thickness of the feature, any infill minerals identifiable. The structural database has also been merged with the assay database, so that Au, Cu and As values have been assigned to *all* structures falling with a particular sample interval. Using the "Dips" software the database can be easily analysed with the use of queries, and the results displayed separately.

The structural analyses have been discussed in Section 10.2.

12.4 Other Measurements

12.4.1 Specific Gravity

Measurements of specific gravity (SG) of representative samples were carried out for selected boreholes. SG was measured using 100-200 mm of intact drill core. The weight of the sample varied from 100 to 800 grams, depending on the size and composition of the sample. The measurement procedure used was the following:

- The weight of the sample was measured in air (Wa) with a scale capable of reading to an accuracy of 1 gram;
- The weight of the same sample was measured with the same scale by immersing the sample completely in water (Wb), by hanging it with a relatively weightless thread from the scale.

Attention was paid that the scale read zero grams before any measurement was taken. The specific gravity (SG) was calculated using the following formula:

$$SG = Wa/(Wa-Wb)$$

A total of 3,511 density determinations were done at Kopsa (Figure 41), giving an average density of 2.74 g/cm^3 with a standard deviation of 0.06 g/cm^3

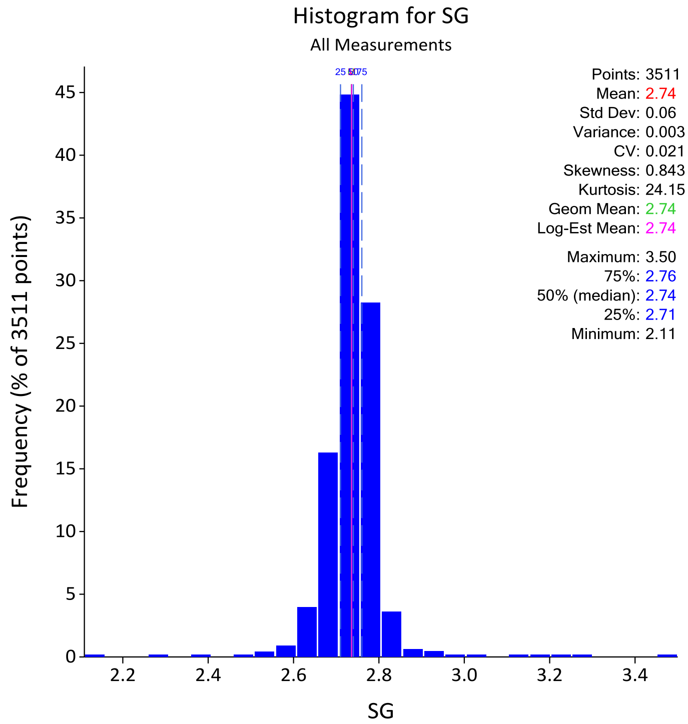

Figure 41 Histogram of all specific gravity measurements from Kopsa

Of these, 1674 samples were assayed from within the modelled mineralised zones, giving an average density of 2.73 g/cm^3 and a standard deviation of 0.06 g/cm^3. A histogram of the specific gravity measurements from within the mineralised shell is provided in Figure 42 .

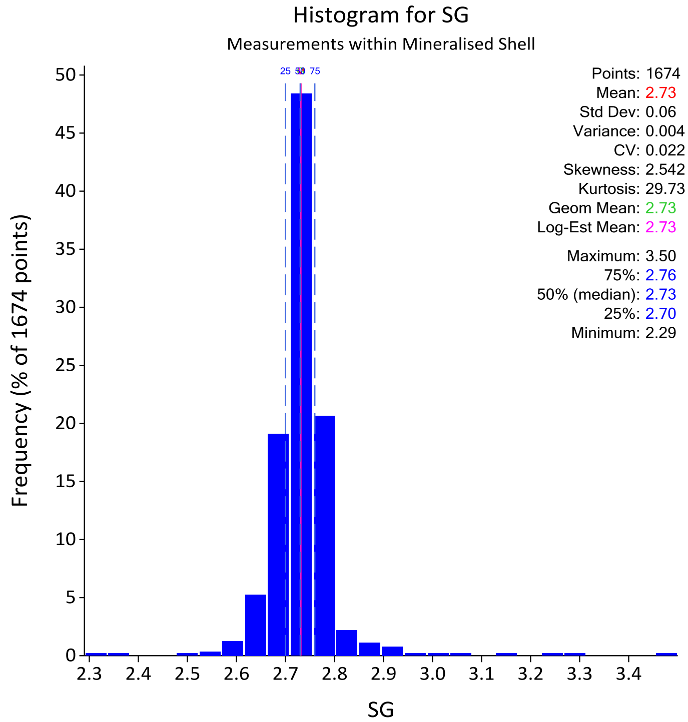

Figure 42 Histogram of specific gravity data within the modelled mineralised shell

12.4.2 Rock Quality Designation

This quantitative index was used to differentiate broken and low-quality rock zones from intact rocks. RQD is calculated by measuring only pieces of core that are greater than 100 mm in length for a given interval and dividing by the total length of the interval. RQD is measured in the following way:

- A convenient interval was chosen, usually drill runs separated by core markers;
- The lengths of pieces of intact core were determined using a measuring tape. Any pieces of core greater than 100 mm in length for a given interval were added together and divided by the total length of the interval;
- The resultant figure was then multiplied by one hundred to give the RQD as a percentage.

13 Sample Preparation, Analyses and Security

13.1 Sample Preparation and Analysis

The drill core was split into two parts using a diamond saw and dried in ambient temperature (heated, if necessary, in rare cases) before sampling. Core was cut wherever possible, along the true vertical plane of the core. One half of the split samples were collected in a strong polyethylene sample bag marked with the sample number in permanent ink. A paper tag with the sample number was also put inside the sample bag. The sample bags were then packed in sacks (about ten samples per sack) for transport to the laboratory. The sacks were dispatched using a local transport Company to the relevant internationally accredited laboratory for assaying.

The assays for the first and second phases of drilling were carried out at the Geological Survey of Finland (GTK) Laboratory (subsequently became Labtium Oy) in Kuopio, central Finland. Here the samples were dried and crushed using a crusher with Mn-steel jaws (method code 30 / 31). A sub-sample is split (method code 35) and pulverised in a tempered carbide steel grinding vessel (method code 40) or a hardened steel bowl (method code 50) to provide a 0.8 – 1.5 kg sub-sample. The rest of the crushed reject is bagged and labelled and returned to Belvedere for storage. After pulverising the pulp is split into further sub-samples for the different assay techniques. One of the sub-samples is analysed for Au, using method 705P. A 50g subsample, is concentrated by a lead fire assay, and then analysed for Au with ICP-AES, and a detection range of 0.01 – 100 ppm Au. Another subsample is leached at 90 °C in Aqua Regia (method 510) and then analysed for 14 elements by ICP-AES (method 510P). This method is not a “total digestion” method, and hence the (non-gold) trace element data needs to be recognised accordingly. The Phase 1 samples were also analysed for silver, bismuth and tellurium by graphite furnace atomic absorption spectrometry (GFAAS) using method code 511U. During Phase 2 the analysis of P was replaced by Ag in method 510P.

GTK: Detection Limits for Methods 705P, 510P and 511U					
Au	2 ppb	**Cr**	5 ppm	**P**	100 ppm
Ag	1 ppm	**Cu**	3 ppm	**Pb**	10 ppm
As	30 ppm	**Fe**	100 ppm	**S**	100 ppm
Bi	200 ppb	**Mn**	1 ppm	**Sb**	20 ppm
Cd	1 ppm	**Mo**	5 ppm	**Te**	10 ppb
Co	1 ppm	**Ni**	3 ppm	**Zn**	1 ppm

Table 23 Elements and detection ranges for GTK assaying methods

The assays from the third and fourth phases of drilling were carried out at the ALS Chemex laboratories in Öjebyn, Sweden. Here, the samples were prepared using PREP-22 method which comprises "Log sample in tracking system, weigh, dry, coarse crush the entire sample and pulverize entire sample to better than 85% passing 75 micron." This is applicable to samples up to 3kg. The gold analysis was carried out using the method Au-AA25 gold fire assay, which envisages analysing ore grade Au (0.01-100ppm) by fire assay (30g nominal sample weight) with AAS finish. In addition, the trace elements were analysed with ME-ICP61 method, in which thirty-three elements were analysed by HF-HNO_3-$HClO_4$ acid digestion, HCl leach, and ICP-AES. The method quantitatively dissolves nearly all elements for the majority of geological materials. Only the most resistive minerals, such as zircons, are only partially dissolved.

ALS Chemex: Analytes and Ranges for Au-AA25 and ME-ICP61							
Au	0.01 - 100 ppm	**Co**	1 - 10,000 ppm	**Mo**	1 - 10,000 ppm	**Th**	20 - 10,000 ppm
Ag	0.5 - 100 ppm	**Cr**	1 - 10,000 ppm	**Na**	0.01 - 10 %	**Ti**	0.01 - 10 %
Al	0.01 - 50 %	**Cu**	1 - 10,000 ppm	**Ni**	1 - 10,000 ppm	**Tl**	10 - 10,000 ppm
As	5 - 10,000 ppm	**Fe**	0.01 - 50 %	**P**	10 - 10,000 ppm	**U**	10 - 10,000 ppm
Ba	10 - 10,000 ppm	**Ga**	10 - 10,000 ppm	**Pb**	2 - 10,000 ppm	**V**	1 - 10,000 ppm
Be	0.5 - 1,000 ppm	**K**	0.01 - 10 %	**S**	0.01 - 10 %	**W**	10 - 10,000 ppm
Bi	2 - 10,000 ppm	**La**	10 - 10,000 ppm	**Sb**	5 - 10,000 ppm	**Zn**	2 - 10,000 ppm
Ca	0.01 - 50 %	**Mg**	0.01 - 50 %	**Sc**	1 - 10,000 ppm		
Cd	0.5 - 1,000 ppm	**Mn**	5 - 100,000 ppm	**Sr**	1 - 10,000 ppm		

Table 24 Elements and detection ranges for ALS Chemex assaying methods

The assays from the fifth phase of drilling were carried out at the Labtium laboratory at Rovaniemi in Northern Finland. Here the samples were prepared and assayed as follows. The split drill core (max. weight 10 kg) was dried at 70 °C (method code 10). The samples were then prepared along the robotised sample preparation line (ROBO1), which includes: jaw crushing of the rock samples to >70% < 2mm (method 32) with compressed air cleaning of the jaws between samples. The crushed sample is then split in a rotary splitter (method 34) to provide a 0.8 – 1.5 kg sub-sample. The sub-sample is then pulverised with LM2 pulverising mill (method 52). The rest of the crushed reject is bagged and labelled and returned to Belvedere for storage. The pulverising puck and the bowl are cleaned with glass bead blasting after every sample to overcome cross contamination. After pulverising the pulp is split into further sub-samples for assaying and archiving (method 38)

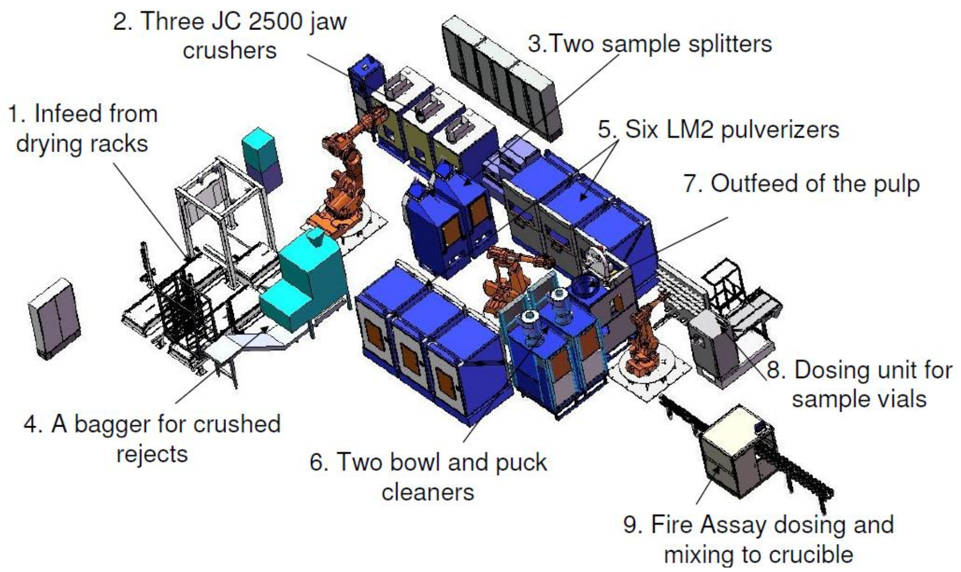

Figure 43 Schematic of Labtium robotised sample preparation unit

One of the sub-samples is analysed for Au, using method 704P. A 25g subsample, is concentrated by a lead fire assay, and then analysed for Au with ICP-AES, and a detection range of 0.01 – 100 ppm Au. Another subsample is leached at 90 °C in Aqua Regia (method 511) and then analysed for 28 elements by ICP-AES (method 511P). This method is not a “total digestion” method, and hence the (non-gold) trace element data needs to be recognised accordingly.

Labtium: Detection Limits for Methods 704P and 511P					
Au	10 ppb	**Cr**	1 ppm	**Pb**	10 ppm
Ag	1 ppm	**Cu**	1 ppm	**S**	20 ppm
Al	20 ppm	**Fe**	50 ppm	**Sb**	20 ppm
As	10 ppm	**K**	200 ppm	**Sc**	0.5 ppm
B	5 ppm	**Mg**	50 ppm	**Sr**	0.5 ppm
Be	0.5 ppm	**Mn**	1 ppm	**Ti**	1 ppm
Ba	1 ppm	**Mo**	2 ppm	**V**	1 ppm
Ca	50 ppm	**Na**	50 ppm	**Y**	0.5 ppm
Cd	1 ppm	**Ni**	3 ppm	**Zn**	1 ppm
Co	1 ppm	**P**	50 ppm		

Table 25 Elements and detection limits for Labtium assaying methods

The samples from the Phase 6 drilling programme in 2011 were carried out by ALS Chemex, to the same specification as in Phases 3 and 4. The only difference was that the samples were prepared at the ALS Chemex lab in Outokumpu, Finland rather than in Sweden.

13.2 Data Handling

Assay data was received from the laboratory as digital files. These were incorporated by Belvedere staff into the project database, and checked against the original sample information. Sample standards and blanks are at this time checked (see below). Any discrepancies with the expected data are reported to the laboratory for verification, and if required, reassaying.

13.3 QA/QC Procedures

The following QA/QC procedures were utilised to ensure the integrity and validity of the assay data.

During all handling of the core and samples under Belvedere's control (including drilling contractors and Company staff) no personnel were permitted to wear gold jewellery to minimise contamination. In addition the diamond saw was cleaned at the end of every shift by cutting a piece of brick or barren rock and water, to reduce the potential for cross contamination.

13.3.1 Blanks, Standards and Duplicates Procedures

The quality and accuracy of the assay data being received from the laboratories is routinely monitored by the use of blanks, standards and duplicates being inserted and assayed in the assay runs.

Since 2004, Belvedere inserts blank samples at the beginning, and in some cases also at the end, of every batch sent to the laboratory for assay. This serves to check that there is no contamination of Belvedere's samples from other (perhaps higher grade) samples being assayed at the Laboratory. Likewise the sample at the end of the batch (where included) indicates whether gold from Belvedere's samples is remaining in the preparation of the samples and thus not being assayed.

The use of standards provides a good idea on the accuracy and precision of the analyses. Belvedere inserts a reference sample (standard) with a known gold concentration every 20th sample. The actual standard used varies, but typically is one that contains a similar concentration of gold, as is expected from Belvedere's samples. The standards used by Belvedere were obtained from Activation Laboratories Ltd., Ontario, Canada. The values of these standards are provided (Table 26) with an acceptable range of values as defined by Activation Laboratories, or with a margin of +/- 10% of the defined values.

Standards	**Nominal Value Au ppm**	**Accepted Range Or ± 10%**
ST 10	0.82	± 0.08
ST 10A	9.78	± 0.53
ST 11	3.40	± 0.34
ST P7A	0.77	± 0.06
ST 3B	3.47	± 0.26
ST 12	9.98	± 1.00
ST P8	0.78	± 0.08
ST 3F	3.10	± 0.31
ST 10C	9.71	± 0.97
ST 10D	9.50	± 0.56
ST 3H	3.04	± 0.23

Table 26 Standard Au values used for Kopsa drill core assays

In addition, to the insertion of standards, it is typical for the laboratories to conduct a certain number of duplicate analyses of samples. This provides a further check on the precision of the analytical results.

Accredited laboratories are required to run their own QA/QC procedures including the use of a certain number of their own blanks, standards and duplicate analyses. This data is usually made available when the finalised assay results have been completed. Table 27 shows the total number of assays, blanks, standards and duplicates utilised for the QA/QC of gold assays for all three phases of Belvedere drilling.

		Laboratory			**Belvedere**			**Total**				
	Assays	**Blank**	**Stds**	**Dupl**	**Blank**	**Stds**	**Dupl**	**Blank**	**Stds**	**Dupl**	**All**	
Phase 1	**211**	10	12	11			9	10	12	20	**42**	**20%**
Phase 2	**813**	39	50	27				39	50	27	**116**	**14%**
Phase 3	**353**	17	32	14	1	2		18	34	14	**66**	**19%**
Phase 4	**564**	21	42	20	7	30		28	72	20	**120**	**21%**
Phase 5	**2723**	116	116	94	26	123		142	239	94	**475**	**17%**
Phase 6	**3732**	48	91	50	14	81		62	172	50	**284**	**8%**
Total	**8396**	251	343	216	48	236	9	299	579	225	**1103**	**13%**

Table 27 Summary of QA/QC procedures utilised for Belvedere drilling

The results of the standard assays submitted by Belvedere for all phases of drilling are in Figure 44 - Figure 50. These also show the acceptable ranges for analysis as detailed in Table 26. The data shows that, with only a few exceptions, the quality of the standard assays has been good. Figure 51 - Figure 54 shows the results of the duplicate analyses sent for assay. The dashed lines in the duplicate analyses plots show the ± 5% range of variation from the original.

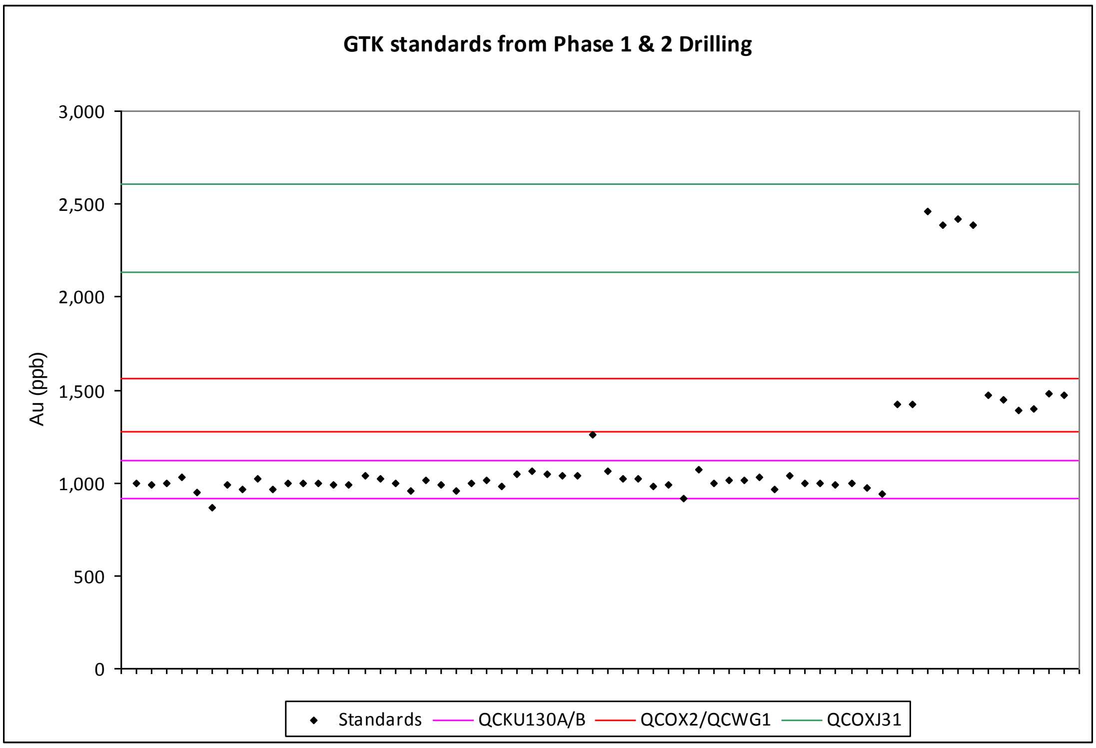

Figure 44 Standard assays and ±10% ranges from Belvedere Phase 1 and 2 Drilling

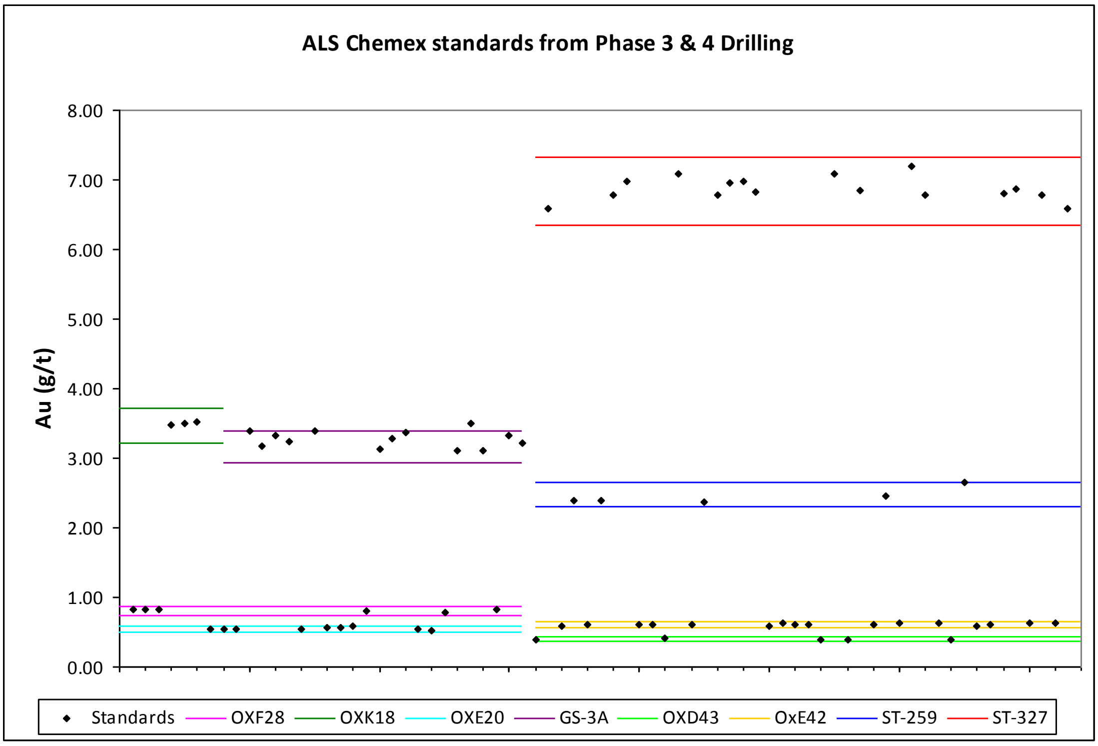

Figure 45 ALS Chemex standard assays and ranges from Phase 3 and 4 Drilling

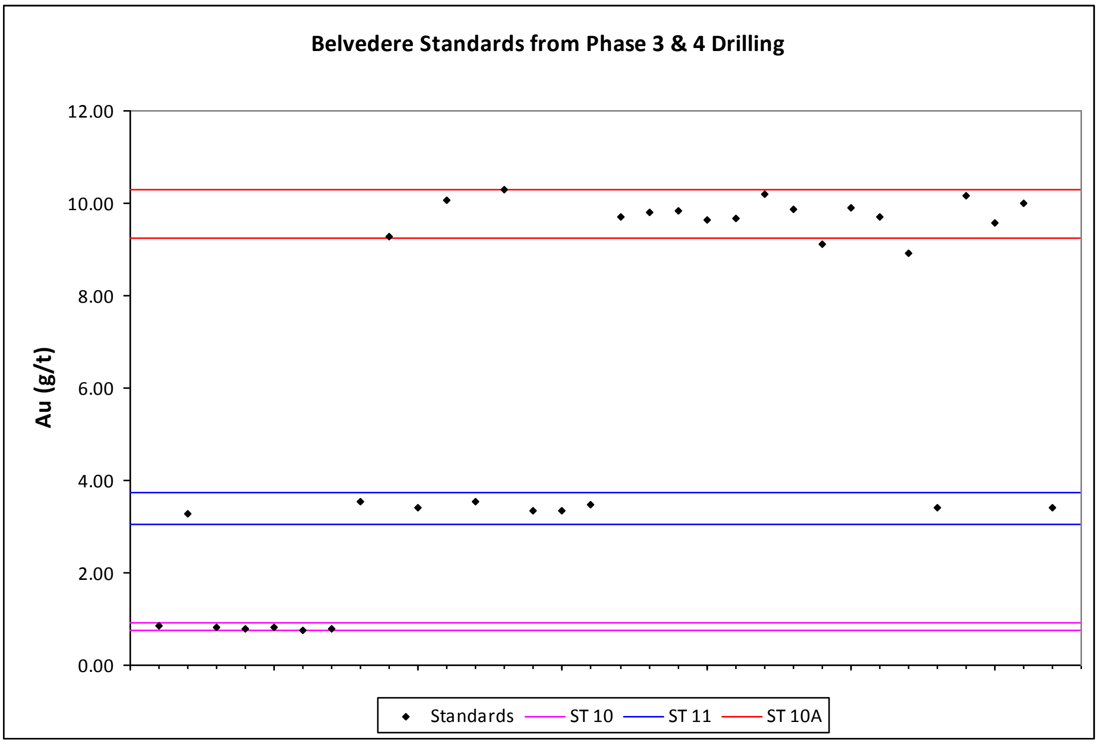

Figure 46 Belvedere standard assays and ranges from Phase 3 and 4 Drilling

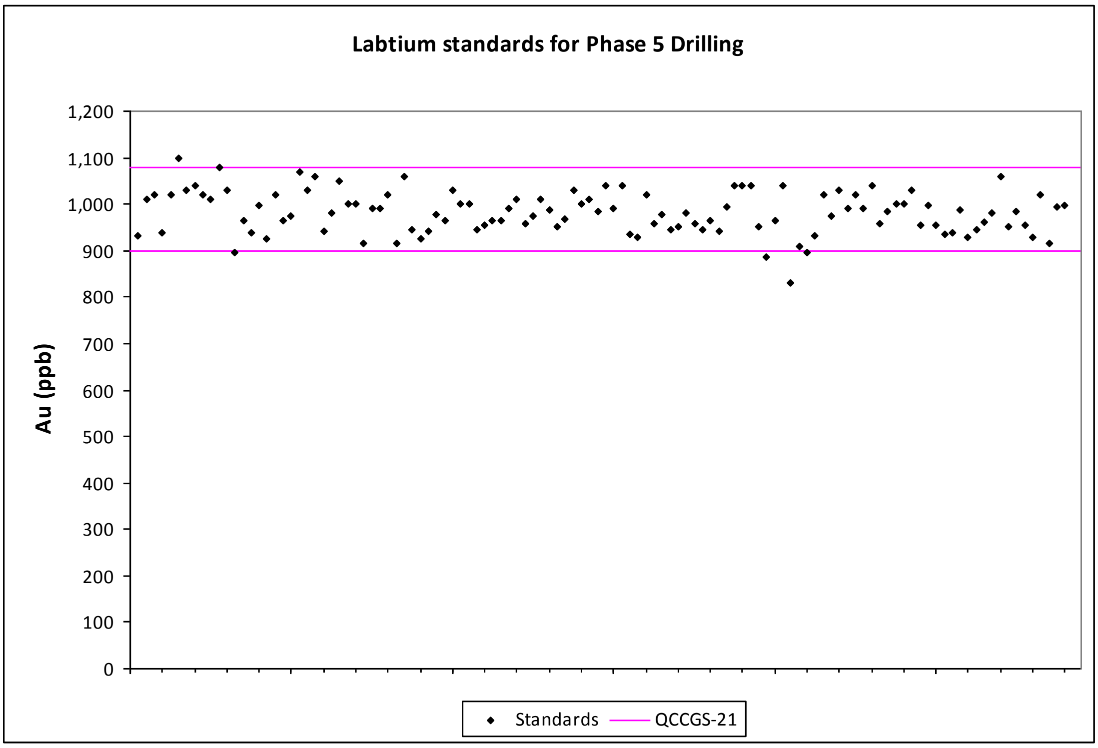

Figure 47 Labtium standard assays and ranges from Phase 5 Drilling

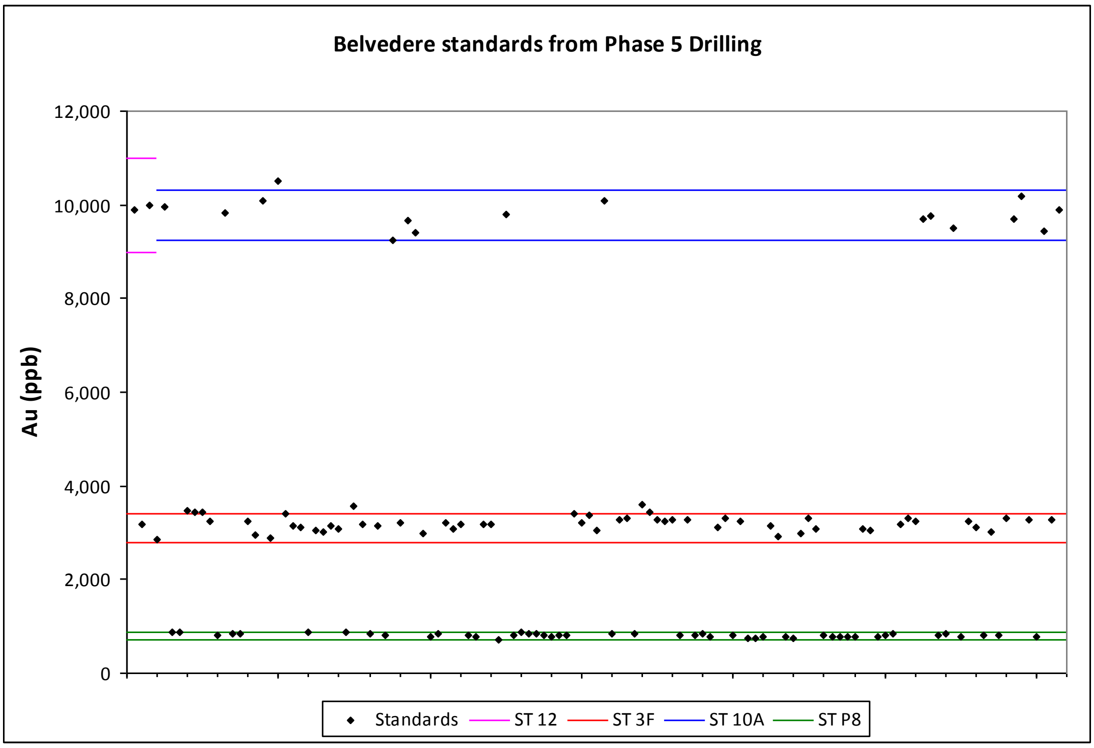

Figure 48 Belvedere standard assays and ranges from Phase 5 drilling

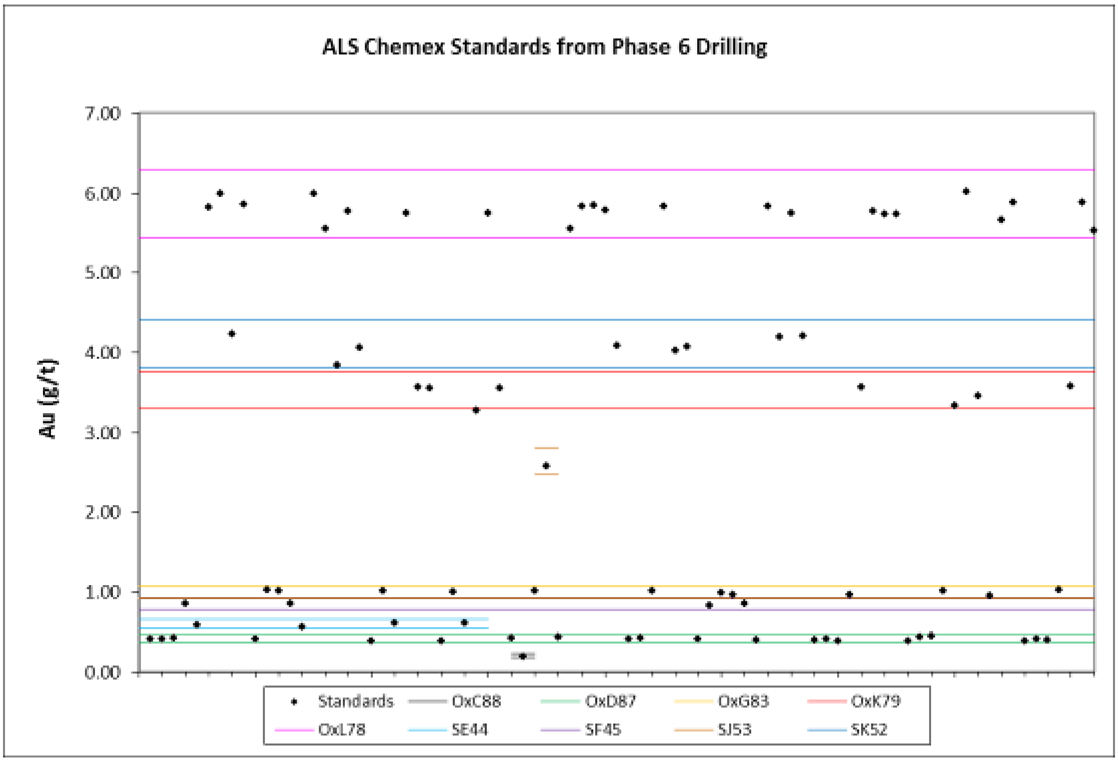

Figure 49 ALS Chemex standard assays and ranges from Phase 6 drilling

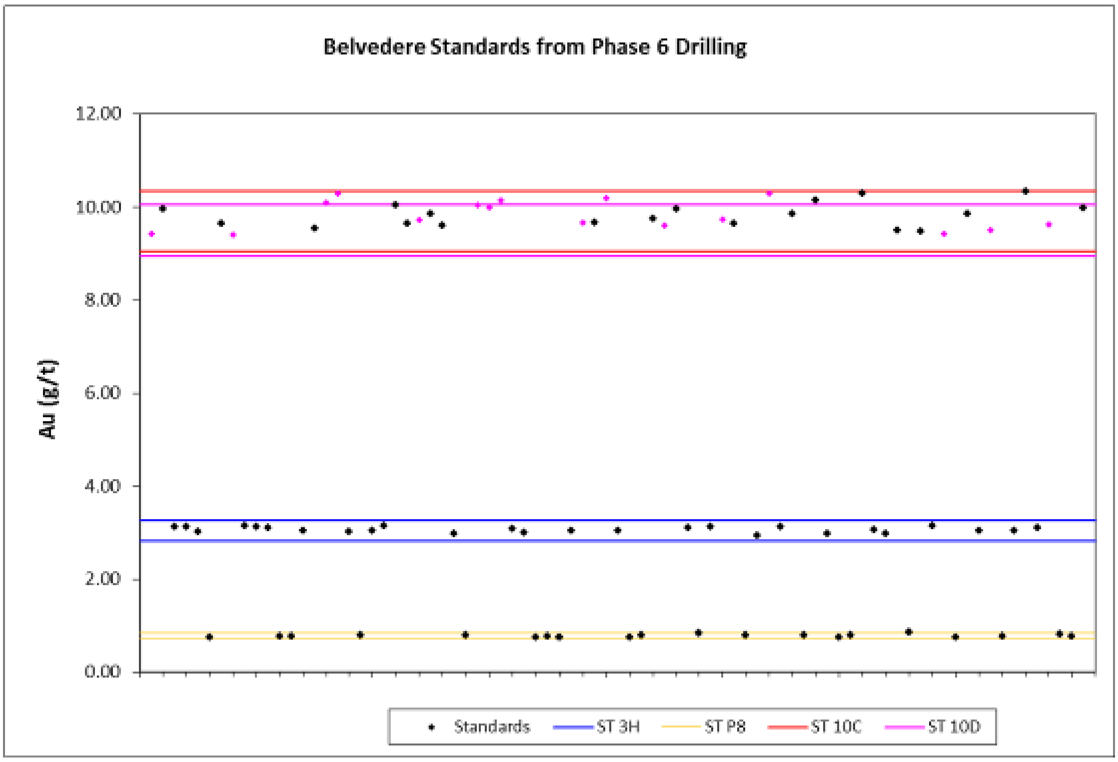

Figure 50 Belvedere standard assays and ranges from Phase 6 Drilling

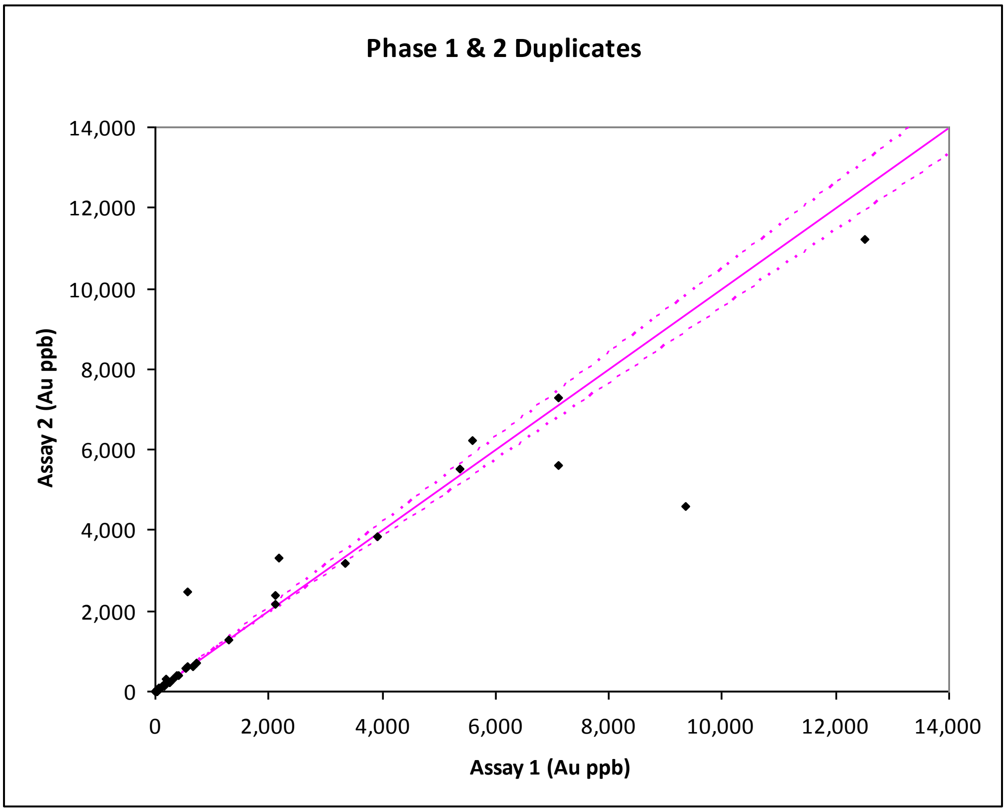

Figure 51 Duplicate sample assays from Belvedere Phase 1 & 2 Drilling. Dashed lines show ±5% range

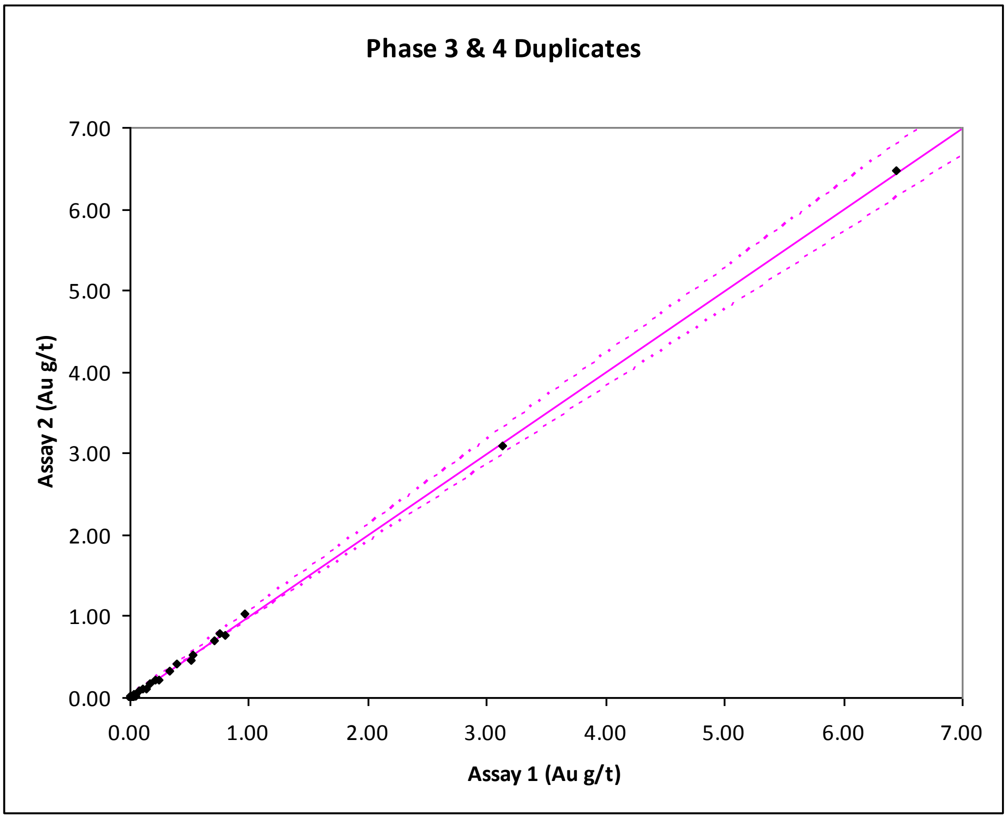

Figure 52 Duplicate sample assays from Belvedere Phase 3 & 4 Drilling. Dashed lines show ±5% range

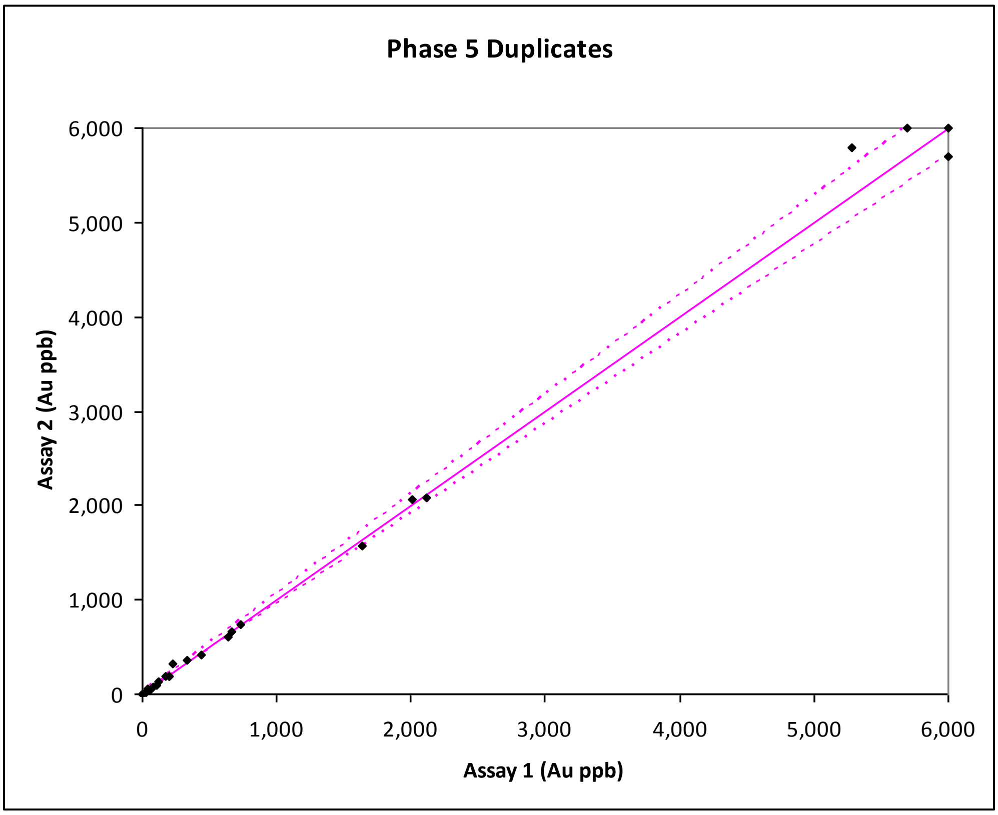

Figure 53 Duplicate sample assays from Belvedere Phase 5 Drilling. Dashed lines show ±5% range

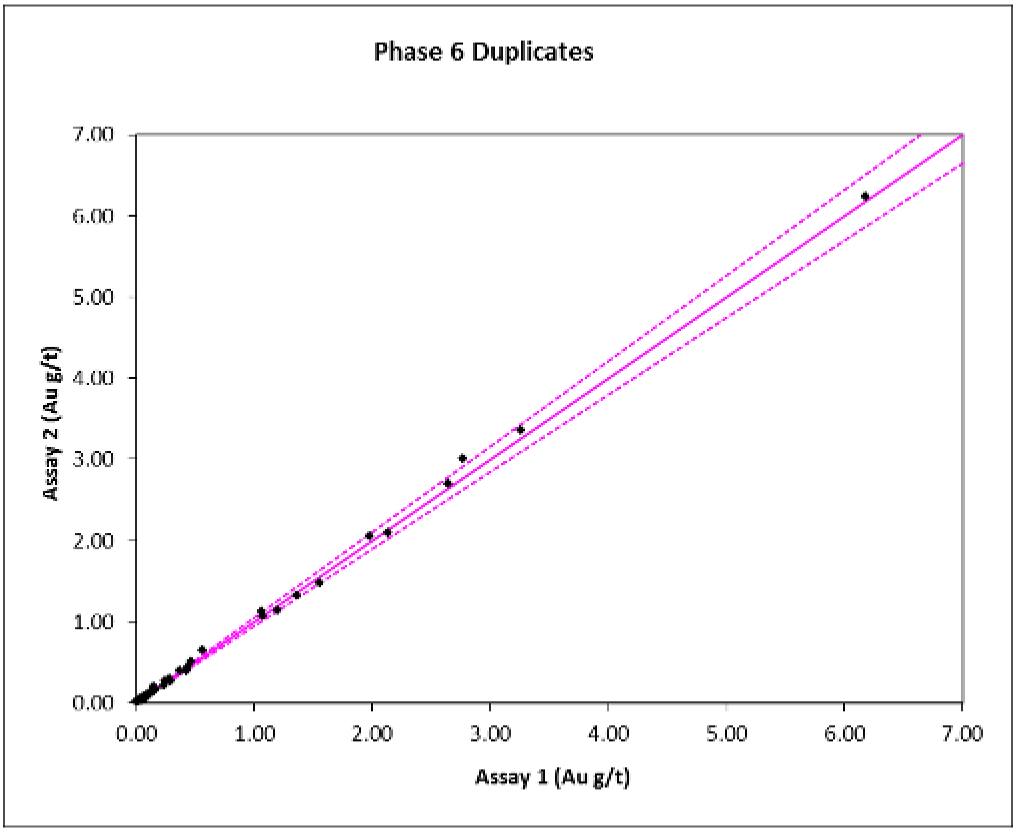

Figure 54 Duplicate sample assays from Belvedere Phase 6 Drilling. Dashed lines show ±5% range

No employee, officer, director or associate of the issuer of this report conducted any aspect of the sample preparation beyond that described. In the opinion of the Qualified Person, the sample preparation, security and analytical procedures were adequately undertaken.

14 Data Verification

Geological, geotechnical and analytical information for the Kopsa deposit consists primarily of the data generated by Belvedere Resources during it's various exploration programmes. Additional information is also available from the Glenmore Highlands period of exploration, as well as limited information from earlier Outokumpu and GTK periods of exploration.

The Qualified Person has relied wholly on information and data provided by Belvedere Resources to construct the geological model for the Kopsa deposit. The Qualified Person did not conduct fieldwork, other than a visit to inspect the property on 27th May, 2011, and a visit to Belvedere's core storage facility, to inspect the drill core on 2nd February, 2011. Whilst examining the drillcore, the Qualified Person verified the quality of logging and checked sample intervals and assay results in relation to the drillcore. The Qualified Person did not independently drill any holes, log core or independently sample drill core or obtained commercial assays of check samples.

The QA/QC procedures carried out during the Belvedere period of exploration are well documented. The assays were carried out in accredited laboratories, and certificates from the laboratories, stating the correctness of the assays are available, and have been provided to the Qualified Person. The assays were received electronically from the laboratory, and imported directly into drill hole database spreadsheets in Excel.

No details are available for the exact QA/QC procedures employed for the pre-Belvedere phases of exploration. However, the work was undertaken by professional personnel (competent persons), who undertook sampling of drill core in accordance with good and established industry practice. The assays were mainly carried out by the GTK laboratory in Finland. The GTK laboratory (now Labtium Oy) was awarded an international accreditation by the Centre of Metrology and Accreditation (FINAS) on 2.11.1994. Prior to this date, the laboratory was run in accordance with good and established industry practice. Consequently, the Qualified Person considers the data to be reliable for resource estimation purposes.

The Qualified Person has no reason to believe that any of the data or documentation provided is misleading in any way. However, no warranties regarding the source data provided by others can be given. It is the Qualified Person's opinion that the QA/QC procedures were adequate, and the exploration companies (Belvedere Resources, Glenmore Highlands, Outokumpu and the Geological Survey of Finland (GTK)) are believed to have carried out the work to industry standards and the data is thus believed to be reliable.

The Qualified Person has validated the data only for obvious errors in original source records by plotting the geological and analytical data in plan and cross sections, and modelled it in three dimensions to ensure that the digital output fits with topographic, lithologic, mineralisation, and analytical and other constraints of the deposit. No significant errors were found during this validation procedure. The assay and geological databases are estimated to be suitable to support resource estimates.

15 Adjacent Properties

The developed properties and the existing mines in the same region as the Kopsa Property are Belvedere Resources Hitura Ni-Cu mine; the Laiva gold mine owned by Nordic Mines, and Pyhäsalmi Zn-Cu-S owned by Inmet Mining Corporation (Figure 3, Figure 4, and Figure 10).

16 Mineral Processing and Metallurgical Testing

A number of metallurgical studies have been completed on Kopsa "ore" since 2005 to determine appropriate grind sizes and potential gold recoveries using several techniques.

16.1 2005 Study – McClelland Laboratories Inc

In 2005 Belvedere sent a batch of samples from to McClelland Laboratories Inc, in Nevada, USA for metallurgical studies. The objective of the study was to determine the amenability to whole ore milling / cyanidation treatment, feed size sensitivity, and ore variability. The tests were completed and a report received in December, 2006.

16.1.1 Method

A total of 61 whole drill core interval samples taken from the Glenmore Highlands drillholes KDD-1 and KDD-12 were selected for testing. One-half splits taken from all of the intervals were combined to produce a Master Composite.

Mechanically agitated cyanidation tests were conducted on the three composites, at varied feed sizes to determine gold recovery, recovery rate, reagent requirements, and sensitivity to feed size. The Master composite was evaluated at 80%-212µm, 150µm, 106µm and 75µm feed sizes. The two drill hole composites were evaluated at 80%-212µm, 75µm and 45µm feed sizes. Feeds were stage ground using a laboratory stainless steel ball mill.

Milled ore charges (2 kg ea.) were placed into open, baffled leaching vessels and settled in grinding water to achieve 40 weight percent solids. Natural pulp pHs were measured. High calcium hydrated lime was added to adjust the pH of the pulps to 11.0 before adding the cyanide. Sodium cyanide, equivalent to 1.0 gNaCN/L of solution, was added to the alkaline pulps.

Leaching was conducted by mechanically agitating the pulps in the leaching vessels for 72 hours. Slurry samples were removed by vacuum aspiration after 2, 6, 12, 24 and 48 hours to obtain pregnant solution samples for gold and silver analysis by A.A. methods. Solids removed during sampling were returned immediately to the appropriate leaching vessel. Pregnant solution volumes were measured and sampled. Cyanide concentration and pH were determined for each pregnant solution. Make-up water, equivalent to that withdrawn and lost to evaporation, was added to the pulps. Cyanide concentrations were restored to initial levels. Lime was added, when necessary, to maintain the leaching pH at between 10.8 and 11.2. Agitation was not interrupted during sampling procedures.

After 72 hours, the pulps were filtered to separate liquids and solids. Final pregnant solution volumes were measured and sampled for gold and silver analysis. Final pH and cyanide concentrations were determined. Leached residues were washed, dried, weighed, and assayed in triplicate to determine residual gold content.

16.1.2 Results

All three composites were amenable to whole ore milling/cyanidation treatment at the feed sizes evaluated (Table 28). Gold recovery increased with decreasing grind size for all three composites (Figure 55). The indicated optimum grind size with respect to gold recovery was 80%-45µm. Reagent requirements for the 45µm feeds were notably finer than for the coarser feeds.

Average gold recovery achieved from the master composite in 72 hours of leaching improved from 73% at an 80%-212µm feed size to 79% at an 80%-75µm feed size. Gold recovery from the higher grade KDD-1 composite improved from 77.3% at a 212µm feed size to 90.2% at a 45µm feed size. Gold recovery from the lower grade composite KDD-12 improved from 70.7% at the 212µm feed size to 85.0% at the 45µm feed size.

			Cyanidation		Head Grade		Reagent Requirements	
	Feed Size	Au Recovery	Extracted	Tail	Calculated	Assay	NaCN Cons.	Lime Added
Composite	P_{80}(µm)	%	Au g/t	Au g/t	Au g/t	Au g/t	kg/mt ore	kg/mt ore
Master	212	75.9	1.23	0.39	1.62	1.62	1.38	1.4
Master	212	69.8	1.18	0.51	1.69	1.62	0.60	0.9
Master	150	72.5	1.11	0.42	1.53	1.62	1.32	1.1
Master	106	75.8	1.16	0.37	1.53	1.62	1.37	1.2
Master	75	79.1	1.21	0.32	1.53	1.62	1.61	1.2
Master	75	79.5	1.16	0.30	1.46	1.62	1.27	1.1
KDD-1	212	77.3	2.15	0.63	2.78	2.63	1.13	3.0
KDD-1	75	86.7	2.34	0.36	2.70	2.59	1.59	2.0
KDD-1	45	90.2	2.48	0.27	2.75	2.68	2.48	4.0
KDD-12	212	70.7	0.94	0.39	1.33	1.21	1.81	2.2
KDD-12	75	82.1	1.01	0.22	1.23	1.21	1.64	2.5
KDD-12	45	85.0	1.02	0.18	1.20	1.22	2.10	4.9

Table 28 Summary metallurgical results from mechanically agitated cyanidation tests

Gold recovery rates were rapid, and not particularly sensitive to grind size. Gold extraction was substantially complete in 12 hours of leaching.

Cyanide consumption generally was high, and tended to increase with decreasing feed size. Cyanide consumption for the 75µm feeds ranged from 1.27 to 1.64 kg NaCN/mt ore. Lime requirements were low to moderate.

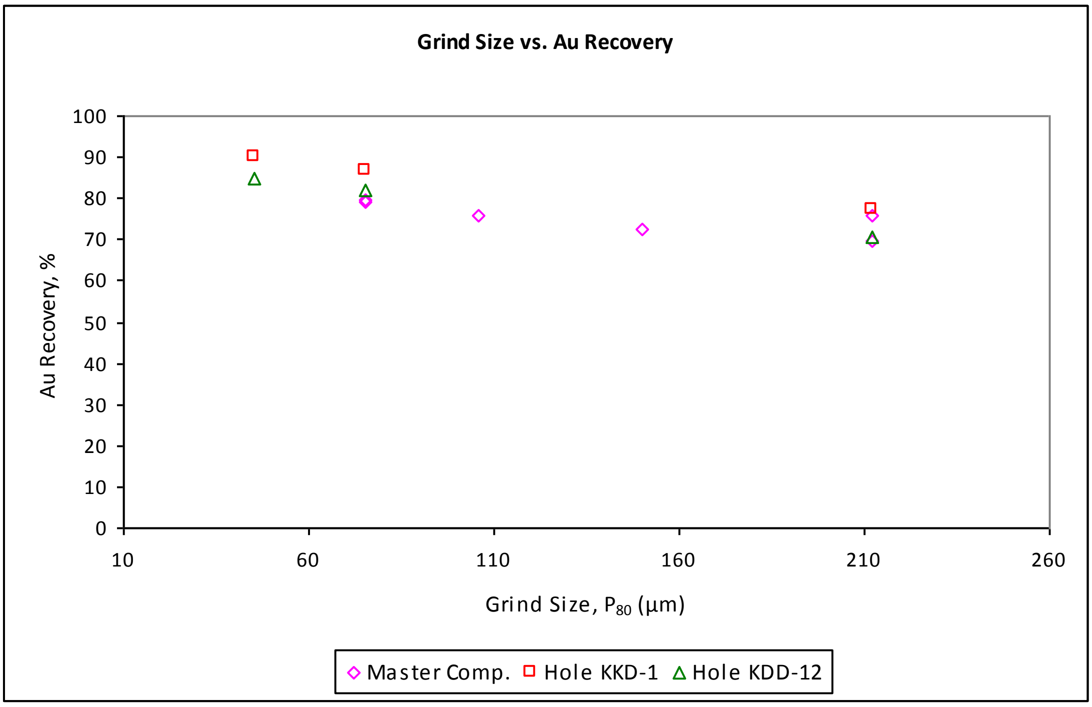

Figure 55 Summary of grind size versus gold recovery for different composites from mechanically agitated cyanidation tests

16.2 2011 – SGS Minerals Services

In 2011 Belvedere sent a batch of samples to SGS Minerals Services in Cornwall, UK. The objectives were to determine whether flotation can be employed as a means of increasing gold recoveries, and to further determine whether the "tail" of such a process could be clean of arsenic. Other objectives included whether gravity concentration could be used to generate a gold concentrate, and also whether a saleable copper concentrate could be generated, either directly from the ore, or from the bulk sulphide concentrate,

The samples used for the study consisted of 66.5 kg of material taken from Belvedere holes BELKOPDD001, BELKOPDD002 and BELKOPDD008 as well as 67 kg of already composited material from the above holes as well as BELKOPDD009. In total 133.15 kg of material was received by SGS. The samples were blended and crushed to 100% passing 1.7mm. 40 kg was split for testing with the remining material stored in cold storage to maintain the integrity of the sulphides.

The blended sample from the Kopsa gold deposit contained 1.8g/t gold, 0.16% copper, and 3.5g/t silver. Of the other analytes assayed, the highest potential contaminants were 9.39% Fe, 0.859% As and 0.87% S. The iron and arsenic could possibly dilute a bulk copper rich sulphide concentrate. The main gangue minerals were silica based, with silica assaying 78.4% SiO_2.

16.2.1 Implications for Gravity Separation

Over 97% percent of the gold is contained in the -106µm size fraction derived from the Screened Metallic's protocol. This suggests that there is little or no gold available for gravity beneficiation using traditional gravity methods. The gold will only concentrate as a result of the concentration of the sulphide minerals.

16.2.2 Bulk Sulphide Flotation

The initial bulk sulphide flotation (FT1 – FT3) was conducted at 4 different grind sizes, which highlighted that the finer the grind, the better the recovery of gold and copper. Grind size tests confirmed the results of previous studies that the optimal grind size (out of the 106µm, 75µm and 45µm sizes tested) was 80% passing the -45µm, generating recoveries of >90% for both Au and Cu.

Bulk sulphide flotation tests (FT4 – FT8) on the -45µm grind size, using various different reagent regimes succeeded in producing a bulk sulphide with high gold (Figure 56) and copper (Figure 56) recoveries. The highest rougher bulk flotation gold recovery achieved over 15 minutes was achieved in FT5 with 93.5%, with a weight pull to concentrate of 26.5%. The gold concentrate grade peaked at 21.9g/t Au. However the highest copper recovery of 97% was achieved in FT7, with a weight pull to concentrate of 29%. The copper grade peaked at 1.8% Cu.

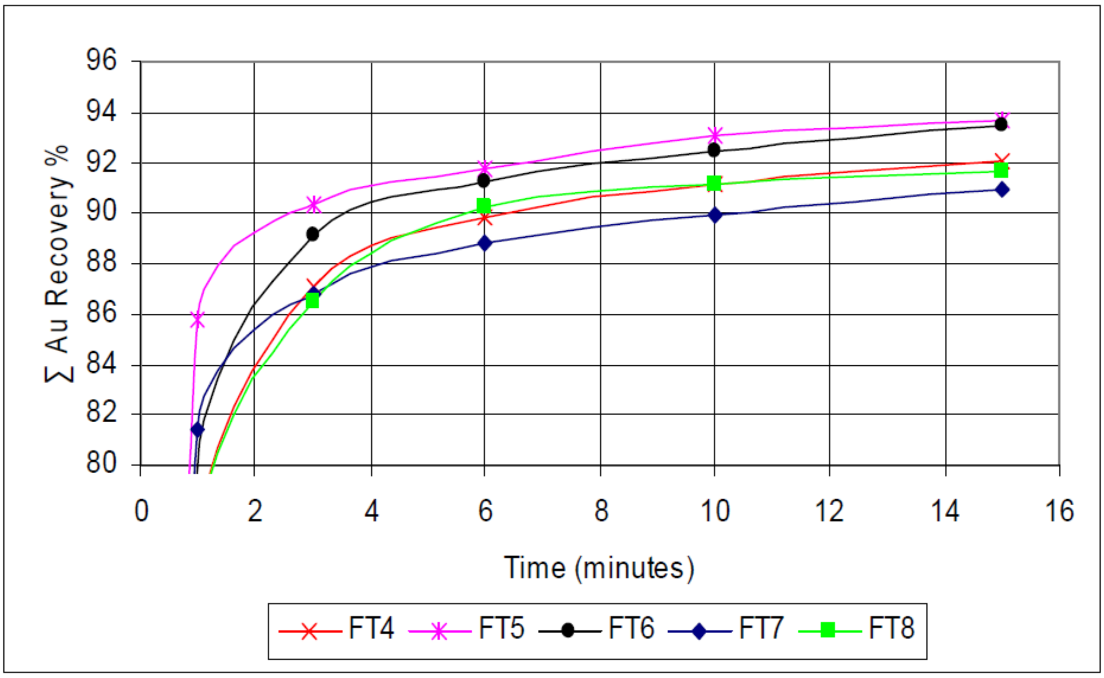

Figure 56 Bulk Sulphide Au Recovery (SGS)

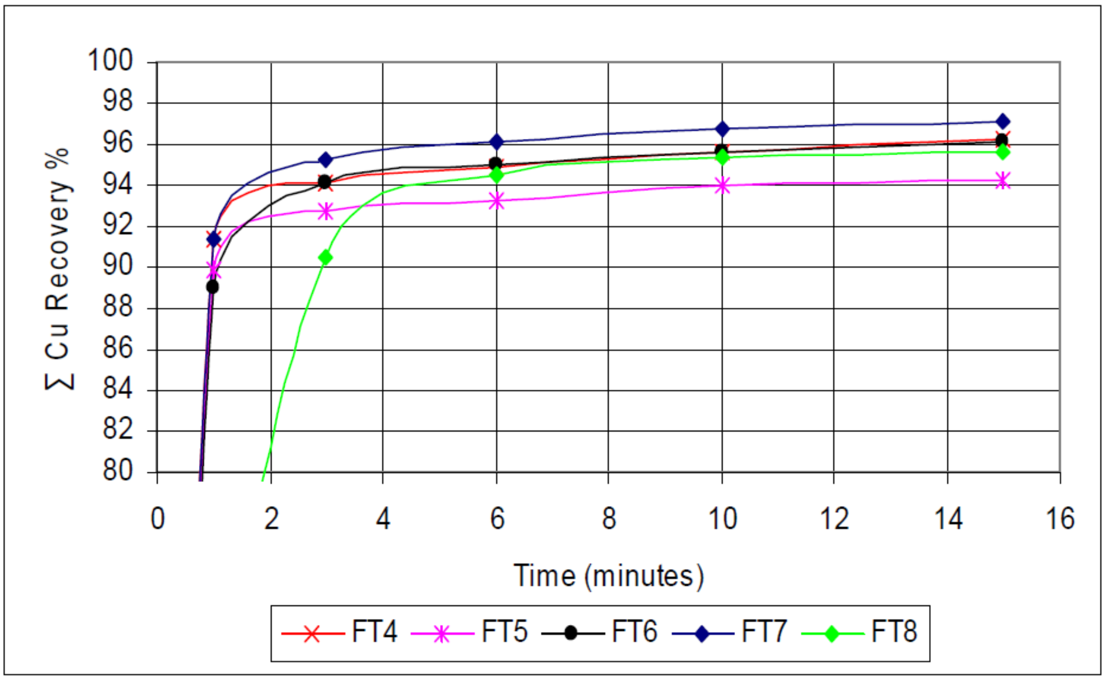

Figure 57 Bulk Sulphide Cu Recovery (SGS)

Flotation test 8 (FT8 was the first test to have arsenic assays conducted on the products. There was excellent recovery of arsenic to flotation concentrate, as after 10 minutes of flotation, 99.9% As recovery had been achieved, with the arsenic grade in concentrate being upgraded to 2.5% after 15 minutes. These high arsenic levels could render a low grade bulk concentrate unsaleable, or uneconomic due to the potential for high arsenic penalties. These high levels would also interfere with attempts to upgrade the copper in the copper concentrate as the cleaner tests showed.

The upside of the high arsenic recoveries through flotation was that it results in a very clean tailings, assaying less than 0.001% As. This is very significant from an environmental perspective.

16.2.3 Selective flotation to produce a saleable copper concentrate

Selective flotation was also trialled (FT9 – FT13). Flotation tests FT9-FT10 did not produce a selective copper concentrate, as 97% of the copper was recovered along with 75% of the gold. Low copper grades and mass pull to concentrate similar to the bulk flotation tests would imply that the arsenic was also reporting to concentrate at similar levels of over 90%.
However, selective flotation tests FT11-FT12 produced copper recoveries of 90-96% with much lower gold recoveries of 50%. The peak copper concentrate grades were 1.9-3.0% Cu at recoveries of 90% copper, containing 32% of the gold. The mass pull to concentrate was 10% lower than the bulk sulphide flotation, which was indicative of the fact the arsenic recovery was reduced from over 90% to less than 12%. Flash flotation of the copper using a selective collector would be a method by which a potential saleable copper concentrate grade could be made, albeit with more than three cleaner stages.

There is some potential for further investigation of selective copper flotation, leaving both the gold and arsenic in the tailings. The question as to whether the gold in the tailings could be leached would also have to be answered.

Bulk rougher flotation followed by differential cleaning to maximise the gold and copper recoveries followed by attempts to produce a clean copper concentrate was attempted in FT15 – FT20. However once floated, the gold and copper were linked with copper recoveries ranging from 67-90% and gold recoveries ranging from 65-75% coupled with copper cleaner concentrates of 2-6% Cu and 25-75g/t Au.

Analysing for arsenic showed that the arsenic followed the copper and gold into the bulk flotation cleaner concentrate. Arsenic recoveries were universally high. The results from the 6 tests suggest that the arsenic will follow the gold and copper once activated in the bulk rougher flotation circuit, and is not amenable to depression in the cleaner circuit. Without the Fe grade results it is difficult to say, but it could well be that a proportion of the arsenic is present in the form of löllingite ($AsFe_2$) which is an arsenide which behaves similarly to sulphides during flotation. This means separating the arsenic from the gold and copper has proven difficult, and would affect the economic value of any bulk concentrate, on the assumption that it could be sold with such a high arsenic content. The upside was the very low arsenic content in the flotation tailings which assayed less than 0.001% As.

In summary, selective flotation of copper produced slightly lower copper recoveries, but generated higher grades. The gold content was significantly reduced to below 50%, and 30% in a flash float scenario, with the arsenic recoveries being reduced to 4-12%. This would make it much more likely that a saleable copper concentrate could be made albeit with further stages of copper cleaning, using a copper specific flotation reagent at starvation dosage.

16.2.4 Cyanidation test of a bulk sulphide concentrate

A cyanidation test was completed on a bulk sample of cleaner 1 concentrate generated to produce enough weight for this test. This produced a recovery of 49% gold with a high copper content, with high reagent consumption rates. SGS recommended that the potential leachability of gold from selective copper flotation tailings would be worth investigating.

16.3 2011 – Optical Sorting of Ore (Comex)

In parallel to the more traditional beneficiation tests being carried out, it was decided also to test Kopsa ore for its suitability in making a coarse pre-concentrate, with the intention of reducing the volume of material that may require transporting to the Hitura mine site.

A small sample of Kopsa ore (<10 kg) was tested using a Comex OSX optical sorting system. The objective of this test was to determine whether it was possible to produce a concentrate based on the amount of quartz vein material in the rock using optical sorting technology.

The feed material for the study had particle sizes of 50 to 150mm.

The results of the initial testwork were very encouraging, in that there was about 79% quartz recovery, with a roughly 50% reduction in material. However, this was based on non-optimal conditions due to the small sample size.

Most of the particles containing quartz vein material that were discarded as waste, were categorised as waste due to the quartz vein material being on the underside of the particle and thus invisible to the detector. Further studies with a larger sample size, and using a new system whereby the sample is analysed from all sides is warranted. Also further studies whereby arsenopyrite is also detected alongside quartz should also be investigated.

16.4 2012 – Mini Pilot Study (GTK)

In December 2011, 190 kg of drill core were delivered to the GTK for processing. The objective was to produce a bulk sulphide concentrate for further studies. It was also important to ensure that the arsenic content of the tailings was low.

The samples were crushed and homogenised. The head grade of the homogenised samples used for processing was 1.20 g/t Au, 0.182% Cu, 0.694% As and 3.65 g/t Ag.

The minipilot was run on the 26th and 28th January 2012 in order to produce a bulk concentrate with high recoveries of gold, silver and copper. The circuit consisted (Figure 58) of a closed ball mill grinding circuit with a screw classifier, rougher flotation and scavenger flotation. The concentrates and tailings were collected in plastic containers.

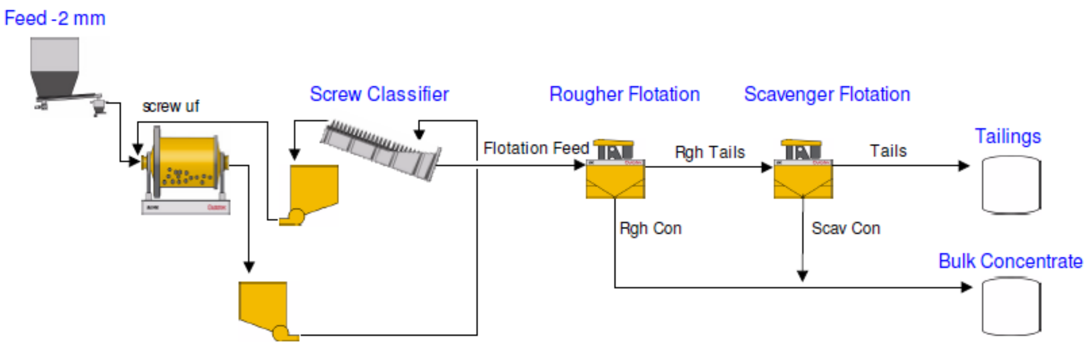

Figure 58 Kopsa Minipilot Flowsheet

The cell volumes were 50 litres for both the rougher flotation and scavenger flotation. A scavenger flotation was used as the rougher stage alone was not achieving the expected recoveries of the valuables with a low arsenic content in the rougher tailings. The used scale up factor between bench scale tests and minipilot was 3. Thus the retention time in the rougher flotation was three times longer than in the laboratory scale, i.e. 60 minutes in the rougher cell and another 60 minutes in the scavenger cell. The copper sulfate activator was fed into the mill and other reagents to the slurry stream prior to feeding into the rougher cell. For practical reasons, the rougher tailings were collected and the scavenger flotation was carried out on the following day. The flotation reagents were then fed also into the scavenger cell, which doubled the reagent consumption. The reagent consumption is shown in Table 29.

	Consumption g/t		
	Rougher	**Scavenger**	**Total**
$CuSO_4$	200	200	400
PAX	150	150	300
AERO407	150	150	300
MIBC	80	80	160

Table 29 Reagent consumption in Kopsa Minipilot

All flotation products: concentrates and tailings were collected and samples were taken for chemical assays. The rougher and scavenger concentrates were combined as bulk concentrate. The bulk concentrate was dewatered after settling and stored under water for further testing. The collected flotation products were not dried at any point of testing.

The material balance is shown in Table 30. The rougher flotation volume was initially 50 litres, but because arsenic in the rougher tailing tended to remain too high, the cell volume was doubled by adding a scavenger flotation stage, which yielded good results. The concentrate mass recovery in the rougher stage was only 14.5 % but was raised to 20.6 % after scavenger flotation. The gold grade in the bulk concentrate was 4.4 g/t with the recovery of 83.0 %. The copper grade was correspondingly 0.81 % with the recovery of 95.7 %. The assayed arsenic content of tailings was less than 0.01 %. The calculated as well as the assayed flotation feed during the run showed lower arsenic content than in the laboratory, 0.53 % As as an average.

Product	kg/h	wt%	Au g/t	RAu %	Ag g/t	RAg %	Cu %	RCu %	As %	RAs %	Fe %	RFe %	S %	RS %
Flotation Feed	15.1	100	1.08	100	3.61	100	0.18	100	0.51	100	2.77	100	0.76	100
Rougher Concentrate	2.2	14.5	5.31	71.2	15.8	63.6	1.02	84.4	2.96	84.5	7.87	41.2	4.32	82.8
Rougher Tails	12.9	85.5	0.36	28.8	1.54	36.4	0.03	15.6	0.09	15.5	1.91	58.8	0.15	17.2
Scavenger Concentrate	0.9	6.1	2.11	11.8	7.35	12.3	0.32	11.2	1.17	13.9	4.48	9.78	1.61	12.9
Bulk Concentrate	**3.1**	**20.6**	**4.37**	**83.0**	**13.3**	**75.9**	**0.81**	**95.7**	**2.43**	**98.4**	**6.87**	**51.0**	**3.52**	**95.7**
Tails	**12.0**	**79.4**	**0.23**	**17.0**	**1.10**	**24.1**	**0.01**	**4.34**	**0.01**	**1.57**	**1.71**	**49.0**	**0.04**	**4.26**

Table 30 Minipilot Mass Balance Calculation

Roughly 20 kg of bulk concentrate was produced in the minipilot test with assays as shown in Table 31.

	Au g/t	Ag g/t	Cu %	Fe %	As %	S %
Bulk Concentrate	4.39	13.35	0.87	7.01	2.53	3.70
Tailings	0.223	0.998	0.01	1.73	<0.01	0.04

Table 31 Main Elements of Flotation Process

Despite previous studies and bench tests indicating that a grind size of at least 80% passing -45µm, the minipilot only achieved a grind size of 75% passing -45µm. The GTK identified this coarser grind size as likely to be the main cause for the lower recoveries achieved in the mini pilot compared to bench scale tests.

The reagent consumption in the minipilot was considerably higher than in the bench scale tests. The reason for this was that the scavenger flotation was run separately after roughing. The chemical consumptions are likely to be lower and closer to the laboratory consumptions if a different flow sheet was used.

16.5 Discussion

A number of studies and approaches have been undertaken to investigate potential economically suitable processing routes.

The studies have all indicated that the gold is fine grained (typically less than 30 µm) and remains locked until ground to a -45µm grind size. However, there is no indication to suggest that the gold is locked in the lattice of minerals such as arsenopyrite. In other word the ore is non-refractory. The fine grain size of the gold effectively rules out direct gravity separation of the gold.

The direct cyanide leaching of the milled ore (2005 McClelland Labs) indicated that although gold recoveries of 85% - 90% could be achieved, the cyanide consumption was excessively high, and the copper would not be recovered.

Tests by SGS in 2011 were aimed to investigate making a flotation concentrate with a clean tail for environmental reasons. These tests successfully demonstrated that a bulk concentrate could be produced which recovered 93.5% of the Au and 97% Cu, and 99.9% of the As. However, the high As content of this bulk concentrate would likely make the product unsaleable. Cyanide leaching on a bulk concentrate only produced a 49% recovery of gold.

Further tests by SGS were aimed to investigate producing a saleable copper concentrate. This was achievable at slightly lower copper recoveries (90 – 96%), but considerably lower gold (50%) and arsenic recoveries (<12%), with the remaining gold and arsenic being lost to the tails. Interestingly, the tests showed that in producing a copper concentrate, the grind size should be further investigated, as there was little improvement in copper recoveries at grind sizes less than 106µm.

The minipilot test by the GTK showed that the bench scale tests could be successfully upscaled into producing a bulk concentrate with a clean tailings.

There are two main process routes to be further investigated, the first is based on generating a pre-concentrate that either can be sold/toll treated by other nearby mining operations, or simply used as a means to reduce the volume of ore that would need trucking to the Hitura Mine Site. A study is currently being planned using a much larger sample for optical sorting. Other parameters for the optical sorting will also include arsenopyrite along with the vein quartz content.

The second process route being investigated is in generating the end products. This will essentially be looking at producing a clean saleable copper (± gold) concentrate and a separate gold – arsenic concentrate that can then be leached to recover the gold. Whether this is based on first making a bulk concentrate or not will also be investigated.

17 Mineral Resource and Reserve Estimates

A mineral resource estimate for the Kopsa deposit was constructed using geological and assay information from all Belvedere drill holes and the Glenmore Highlands diamond drilling. The focus in this section of the report is on the methodology for estimating the gold resource. Raw assay data were composited and analysed to determine their basic statistical and geostatistical properties. This information has been used in testing modelling algorithms which have been compared and checked for validity. The final resource has been categorised into indicated resources, compliant with the JORC Code.

The mineral resource presented in this section of the report was estimated by Mr. Pekka Lovén of Outotec Oy, following the guidelines of the JORC Code. Mr. Lovén is a Qualified Person as defined by in the National Instrument 43-101 on the basis of training and experience in the exploration, mining and estimation of mineral resources of gold and base metal deposits as well as being a member of MAusIMM (CP), a recognized foreign professional association.

17.1 Database for deposit model

The drill hole database, managed by Belvedere Resources, was delivered to the Qualified Person in digital format for use with Surpac modelling software. The used database contains all the information up to the end of the latest drilling and resampling campaign, which was completed in December 2011.

The entire database (which includes the old Outokumpu holes, the GTK holes, the Glenmore diamond drilling and reverse circulation holes, as well as all of the Belvedere holes) has been used for the purpose of constructing domains and wireframes. However, for a number of reasons, only the Belvedere holes and the Glenmore diamond drill holes have been utilised for the actual mineral resource estimation.

The more restricted database used for estimation contains information on 126 drill holes with a total length of 14.323.99 metres and 12,381 assays (average assay interval 1.01 m). The assay table contains the assays of 36 elements, although due to the numerous phases of drilling, not all sample intervals have assay measurements for all elements. The lithology table contains 2,347 recorded intervals. The database includes 3,512 density (specific gravity) measurements, made from the Belvedere phase of drilling (Section 12.4.1). Geotechnical rock quality determinations (3,967) have been done for all the Belvedere drill holes using the RQD method (Section 12.4.2).

The Qualified Person has not validated the entire database for accuracy, but has compared randomly selected data entries in the database against the certified assay results provided by the laboratories. The Qualified Person has also ascertained that the database does not contain any duplicate records or overlapping sample intervals. Furthermore, collar elevations in the database appear to be within an acceptable margin of error when compared to topography. The Qualified Person is thus of the opinion that the database is suitable for the purpose of estimating a mineral resource.

17.2 Geological Model

The gold mineralisation at Kopsa is associated with quartz and arsenopyrite veining. These typically occur with a high density in the “stockwork” portions of the deposit and gradually decrease in density with a corresponding fall off in grade further from the “stockwork” zones. Copper is also likely to be an economic by-product should mining proceed, and so copper contents have also been considered, especially at the eastern end of the deposit.

The outer envelope of mineralisation is therefore based on a combination of high quartz vein percentages, high copper grades, and the 0.3 g/t Au contour. This cut-off is considered to be a reasonable starting-point should Kopsa be developed into an open pit mine. The wireframe was constructed from polygons digitised onto 20 profiles across the deposit, which were typically 20 – 40 metres apart.

Within this broad outer envelope of mineralisation, and in certain cases extending a short distance beyond, a number of higher grade bodies have been domained based on a combination of quartz vein content and gold grades. In total, 10 bodies (domain_a1 ….. domain_a10) have been digitised and collectively flagged as “domain_a”.

Those areas within the outer envelope but not within the higher grade “domain_a”, represents the low grade domains. This has been flagged as “domain_b”.

The final wireframe solids were analysed for errors such as overlaps or gaps and corrected. The final wireframe was cut against the bedrock surface.

It has approximate dimensions of 700 metres by 200 metres, with a maximum thickness of about 50 metres. The mineralisation has a strike direction of 105°, and dips roughly 20° to the SSW. In the north, the mineralisation comes to bedrock surface, part of which outcrops at the "Kopsa Outcrop". The maximum depth for the model is about 125 metres vertical from surface.

The distribution of the domains and their relative sizes are depicted in Figure 59 and Figure 60 and summarised in Table 32.

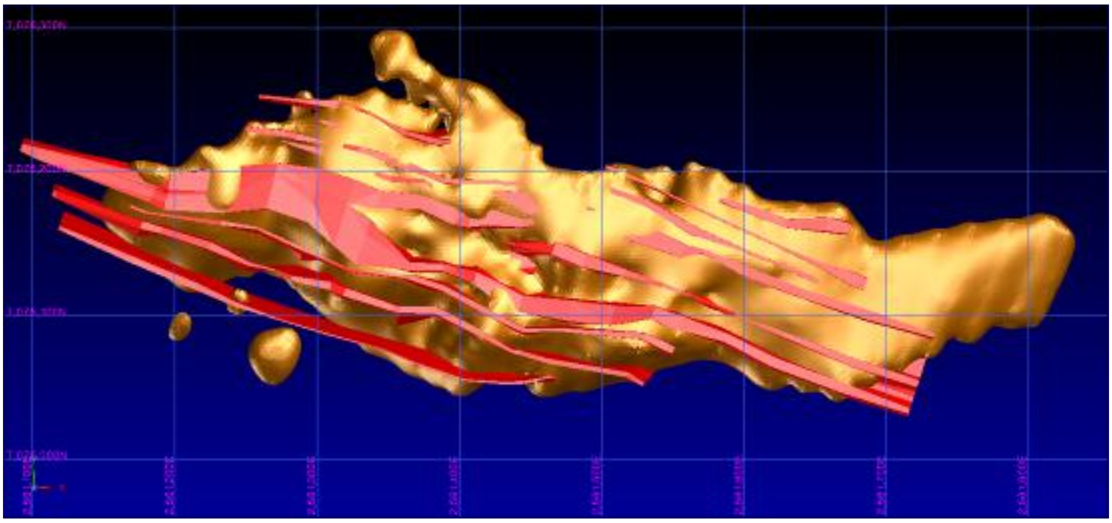

Figure 59 Plan view of mineralised domains at Kopsa: domain_a in red and domain_b in brown

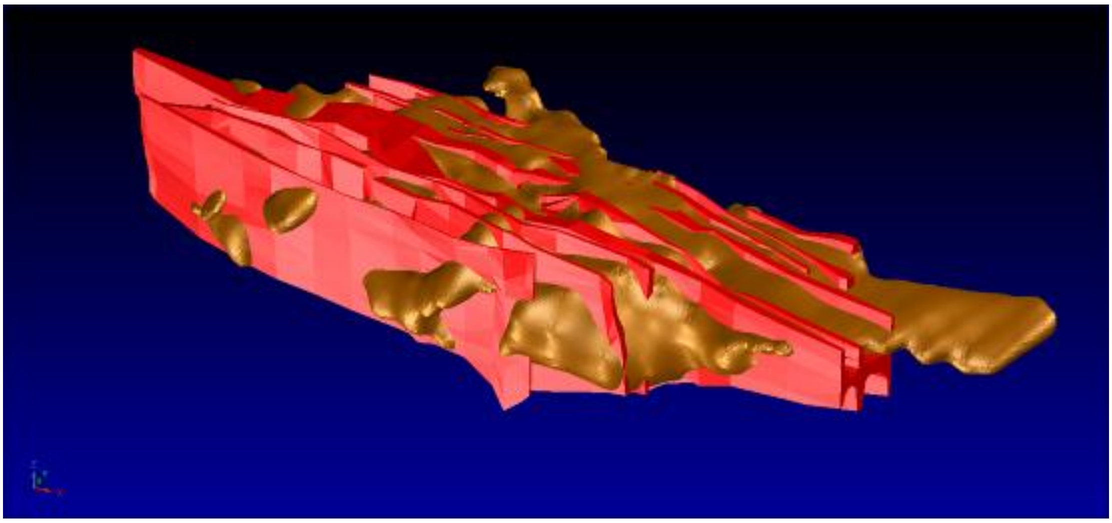

Figure 60 View looking NW of mineralised domains at Kopsa: domain_a in red and domain_b in brown

Domain	Volume m^3
a1	1,698,748
a2	270,836
a3	196,923
a4	59,378
a5	33,034
a6	142,405
a7	33,513
a8	121,022
a9	7,308
a10	31,204
Domain A	2,594,371
Domain B	2,132,215
Total Domain A + B	4,726,586

Table 32 Summary of domain sizes at Kopsa

17.3 Raw Sample Assays

The database was coded for the mineralised domain as explained in section 17.2. Samples were extracted from the database for the domain. Basic statistical studies, as well as grade estimation, were carried out using only the samples from the Belvedere drilling and the Glenmore Highlands drilling within the domained zones (Domains A & B).

Basic statistics were calculated for gold, copper and arsenic. The results are presented in Table 33. The table show classical statistical parameters for assays. A log histogram of the raw Au assays from Domain A is also provided in Figure 61. The statistics are quite typical for a skewed distribution and what is expected for a gold deposit of this type.

Domain	Attribute	Samples	Maximum	Mean	Median	Std Dev
Domain A	Au g/t	3,353	48.4	1.30	0.57	2.89
	Cu ppm	3,353	17,100	1,565	1,155	1,485
	As ppm	3,164	200,000	6,211	3,200	10,542
Domain B	Au g/t	1,637	20.2	0.47	0.20	1.02
	Cu ppm	1,637	22,800	1,374	1,010	1,330
	As ppm	1,573	79,000	3,029	1,190	4,944

Table 33 Basic statistics for raw assays within Domains A and B (only holes used for estimation)

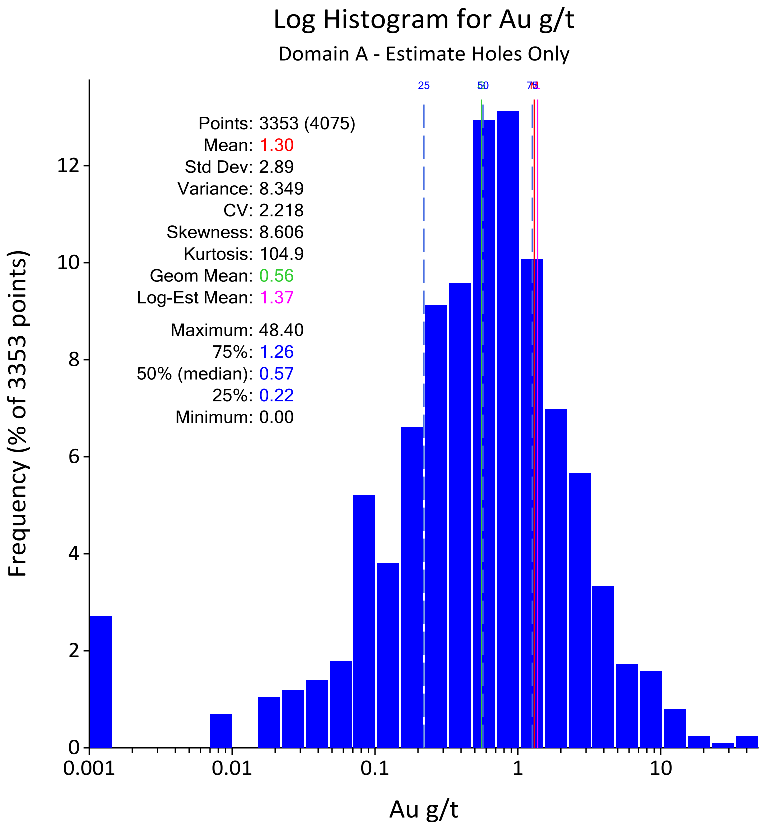

Figure 61 Log histogram of raw (uncomposited) Au assays within Domain A

17.4 Top-cutting

The distribution of gold values requires that high values which are real outliers to the population are top-cut to avoid introducing a bias into the estimation.

A mean and variance plot (Figure 62), which examines the impact on the mean and coefficient of variance with decreasing top cut, indicates that in Domain A there is a significant inflection at about 20 g/t Au. In other words, decreasing the topcut beyond 20 g/t Au would significantly decrease the mean and CV of the population, so that not only the outliers have been cut, but also a legitimate part of a skewed population. Correspondingly, in Domain B there is a significant inflection at about 7 g/t Au.

On this basis, the top-cut was selected to 20 g/t Au for Domain A and 7 g/t Au for Domain B. These topcuts have been applied to the raw samples prior to compositing the data.

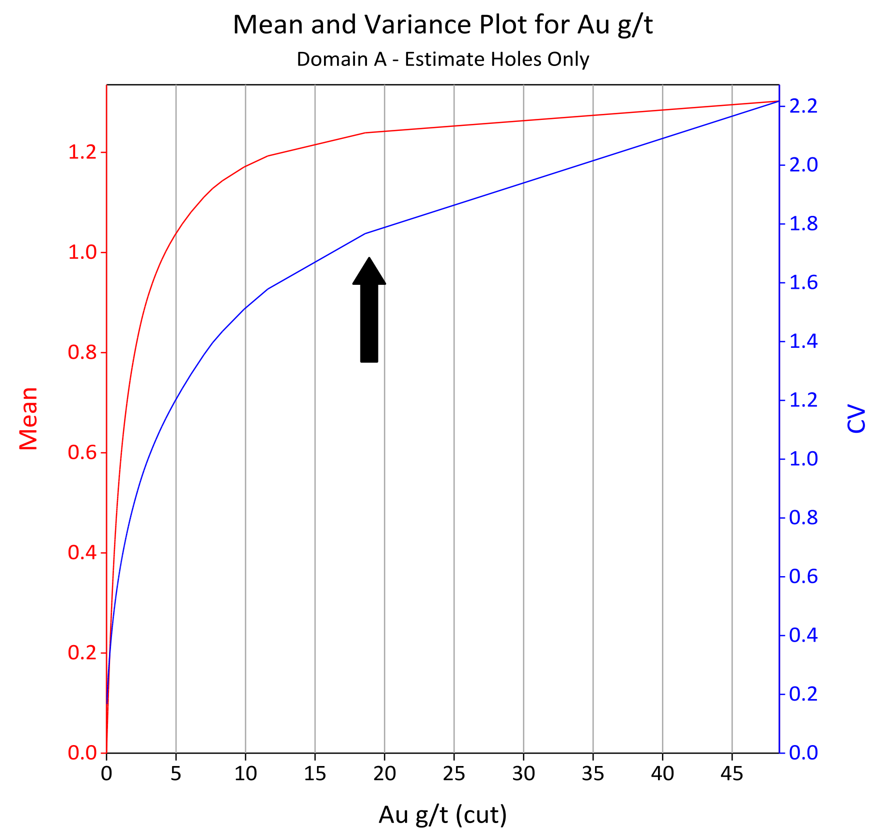

Figure 62 Mean and variance plot for Au in Domain A using raw samples from estimate holes only

Figure 63 Mean and variance plot for Au in Domain B using raw samples from estimate holes only

17.5 Compositing

Inside Domains A and B, the average sample of sample being used for estimation (Belvedere and GMH diamond holes) is 1.00 m (Figure 64). Consequently, it was decided to composite all samples to 1.00 m length.

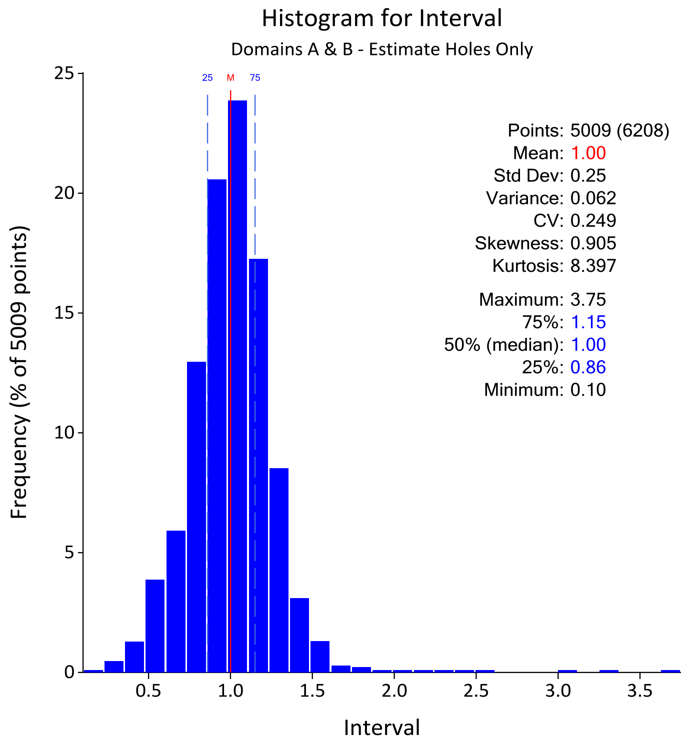

Figure 64 Histogram of raw sample intervals in Domains A & B

This was done by using a best-fit method to minimise the number of residual samples. The compositing was done separately for Domains A1, A2 … A10 and Domain B. Samples below the detection limit and absent samples were given a nominal grade of zero, and included in the compositing. Despite using the best-fit approach, the thickness and geometry of the modelled domains has meant that it was impossible to avoid residuals entirely, and composite samples with an interval of 0.50 m were excluded from the estimation process. In Domain A, 3 separate composites with a combined thickness of 1.25m were excluded. In Domain B, 34 separate composites with a combined thickness of 7.72m were excluded. The results of basic statistics after compositing are shown in Table 34.

Domain	Attribute	Samples	Maximum	Mean	Median	Std Dev
Domain A	Au g/t	3,220	20.0	1.17	0.63	1.79
	Cu ppm	3,220	16,309	1,543	1,184	1,324
	As ppm	3,220	121,829	5,492	3,364	7,526
Domain B	Au g/t	1,581	6.49	0.40	0.23	0.59
	Cu ppm	1,581	11,340	1,332	1,009	1,136
	As ppm	1,581	47,207	2,728	1,378	3,835

Table 34 Basic statistics for composites within Domains A and B (only holes used for estimation)

Histogram plots of the topcut Au grades after compositing are shown in Figure 65 and Figure 66 for Domains A and B respectively.

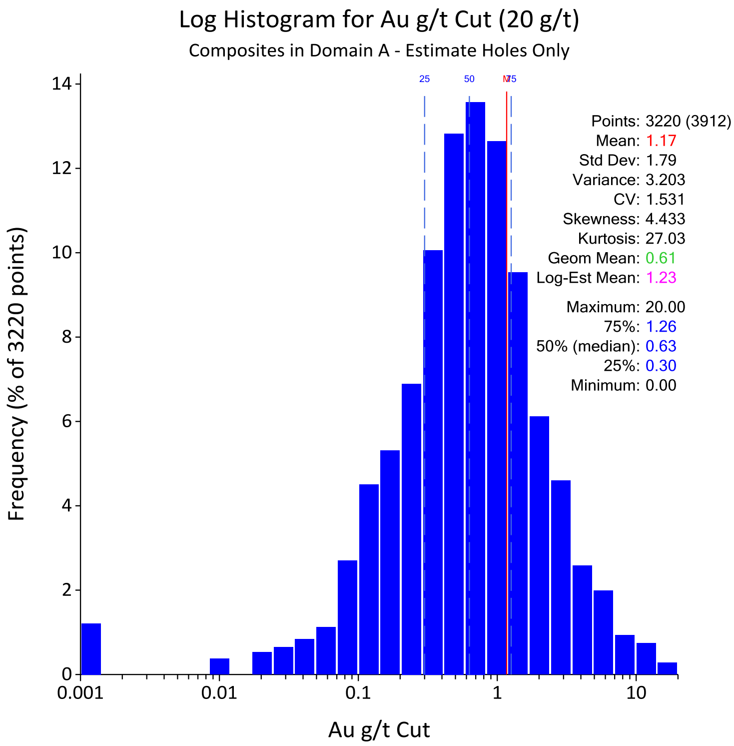

Figure 65 Log Histogram showing the distribution of Au (cut to 20 g/t) in composites for Domain A

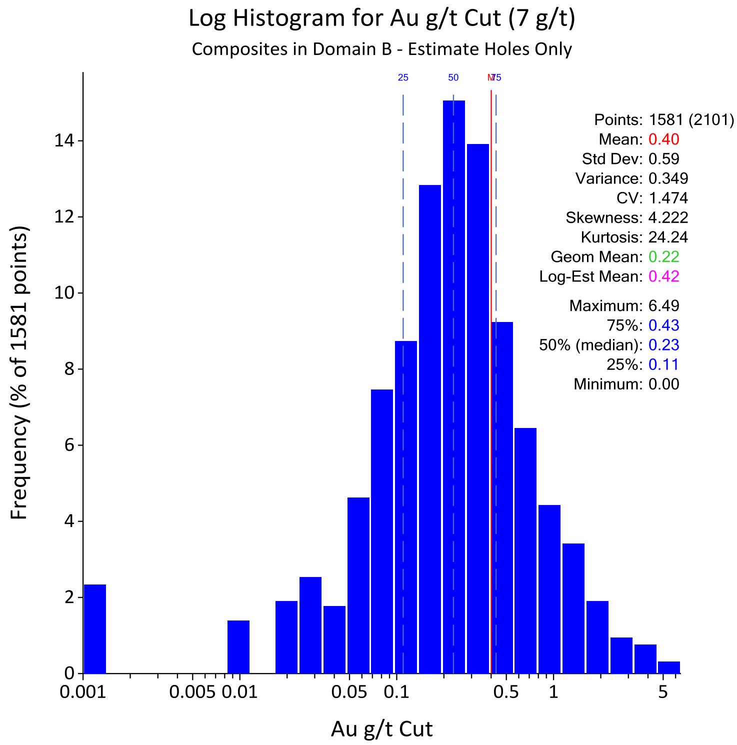

Figure 66 Log Histogram showing the distribution of Au (cut to 7 g/t) in composites for Domain B

17.6 Geostatistics

Prior to Domaining, an attempt was made to model the geostatistics within the initial broad mineralised shell, with the intention of determining model parameters for ordinary kriging. When looking at variogram maps of the composited data, reasonably good continuity could be seen in the horizontal plane, trending towards 105° (Figure 67).

However, it soon became apparent that despite reasonably good continuity in the horizontal plane, that there would be difficulties with the Across Strike continuity (Figure 68). This essentially showed that there are two separate and superimposed trends to the mineralisation. One trend is dipping shallowly to the south and in effect is parallel to the digitised mineralisation envelope. The other trend is dipping to the north at about 45°.

In terms of determining parameters for kriging, if one direction was selected, it would ignore continuity in the other direction. Geologically, these two trends are superimposed and cannot be separated by either domaining or by indicator kriging. Consequently, the best approach was to do the estimation using inverse distance weighting. The ellipsoid orientation was determined based on the parameters in the variogram maps, so that the major axis of the ellipsoid trended horizontally bearing towards 105°. As the range for the major axis was about 100m, and the ranges for both the across strike directions are about 25m, the ellipsoid has a ratio of 4:1:1.

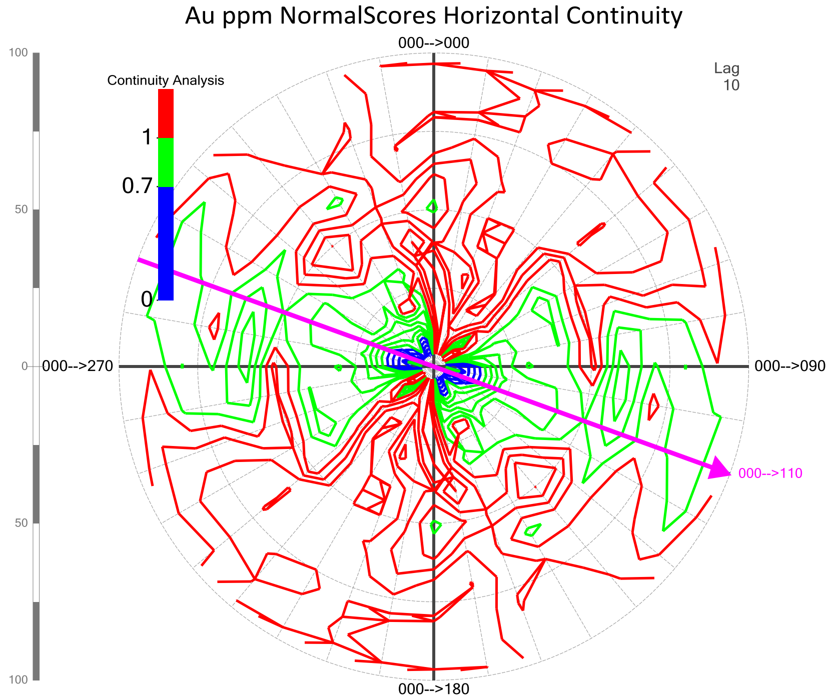

Figure 67 Variogram map in the horizontal plane showing horizontal continuity of composited Au grades inside the broad mineralisation envelope

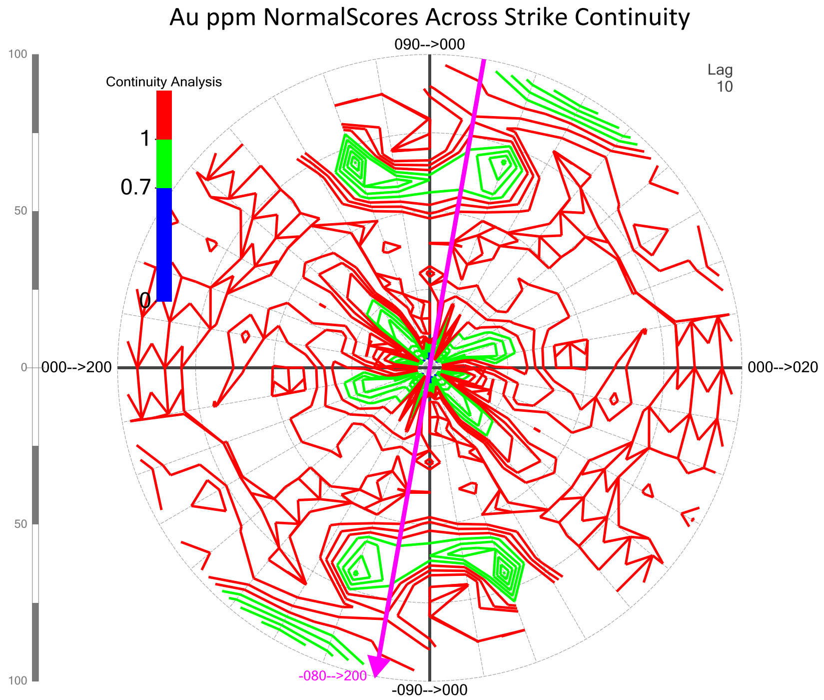

Figure 68 Variogram map in the across-strike plane showing two directions of continuity of composited Au grades inside the broad mineralisation envelope

17.7 Block Model

Block sizes within a block model are typically decided on the basis of sample spacing and mining parameters. A guideline for block size is an optimum size of ½ the drillhole spacing, and a minimum of ¼ the drillhole spacing. The maximum drillhole spacing at Kopsa is 47.9 m with an average spacing of 17.8 m. Should Kopsa be mined as an open pit, it is likely that a bench height of 5m would be used.

The Kopsa block model utilised regular shaped blocks measuring (X) 10 m by (Y) 10 m by (Z) 5 m in height. This block size is considered the most appropriate shape considering the morphology of the mineralisation and the distribution of sample information. To better conform to the mineralisation contacts sub-blocking was used. The block model is rotated to an azimuth of 105° to better fit the geometry of the deposit. Block grades were estimated for parent cells and distributed to their sub-blocks. Block model grade interpolation for all estimated elements was performed using Inverse Distance Weighting (IDW) with the following parameters:

Parameter	**X**	**Y**	**Z**
Origin (Min Coordinates)	2561000	7074900	-40
Max Coordinates	2561800	7075500	120
Block Size	10	10	5
Min Sub-Block size	10	5	5
	Bearing	**Dip**	**Plunge**
Rotation	105	0	0

Table 35 Block Model Parameters

17.7.1 Estimation Parameters and Search Distances

The search ellipsoid for the IDW estimation is essentially a horizontal, cigar shaped ellipsoid trending towards 105°. The angels of rotation and anisotropy factors of the anisotropy ellipsoid in Surpac ZXY LRL convention are as follows:

ANGLES OF ROTATION – Surpac ZXY LRL	
First Axis	105.00
Second Axis	0.00
Third Axis	90.00
ANISOTROPY FACTORS	
Semi-major ratio	4.00
Minor ratio	4.00

Table 36 Modelled parameters of the anisotropy ellipsoid

The estimation procedure itself used Inverse Distance Weighting (IDW) to estimate the values of the blocks. The value for each block has been determined by the average value of multiple estimations within the block (descretisation). For this estimate descretisation has been set to 3 x 3 x3 (i.e 27 points).

The estimation was undertaken separately for each domain: once for each high grade domain (domain a1, a2 …. a10) using the composites with a 20 g/t Au top cut and once for domain b using the composites with a 7 g/t Au top cut. Copper and arsenic were also populated in same estimation runs using the same search parameters as gold. All blocks were populated in the first estimation pass. The number of composites used for estimation along with other parameters utilised is tabulated in Table 37.

Block Model Estimation Parameters – Inverse Distance Weighting				
Interpolation pass	Maximum Search Radius (m) on major axis	Maximum vertical search distance	Minimum Number of Composites	Maximum Number of Composites
First	100	999	5	30

Table 37 Block model estimation parameters

17.7.2 Tonnage Calculations

The tonnages for each block have been calculated by multiplying the volume of the block by the specific gravity of the block. The specific gravity of all the blocks has been assigned as 2.73 g/cm^3, as discussed in Section 12.4.1.

17.8 Block Model Validation

Validation against the raw input data is essential to ensure reproduction of drillhole grades is realistic and representative in the model. Both statistical and spatial aspects of validation are important on a global and local scale.

17.8.1 Statistics

A useful validation method is to compare the global mean of the block model values to the global mean of the composites used for the estimation. The basic statistics of the Au, Cu and As grades estimated for the block model are shown in Table 38 compared to the de-clustered composite samples for each domain. It is clear from the table that the block model has slightly under-represented the gold grades and to a lesser extent the arsenic grades in both domains. Copper in the block model shows good agreement with the composite samples.

		Composite Samples Declustered			Block Model			
Domain	**Attribute**	**Max Grade**	**Mean Grade**	**Std Dev**	**Max Grade**	**Mean Grade**	**Std Dev**	**Comp Mean /BM Mean**
Domain A	Au g/t	20.0	1.15	1.73	6.94	1.03	0.64	112%
	Cu ppm	16,309	1,516	1,276	5,012	1,526	688	99%
	As ppm	121,829	5,477	7,406	37,670	5,147	2,979	106%
Domain B	Au g/t	6.49	0.41	0.61	3.79	0.36	0.19	114%
	Cu ppm	11,340	1,329	1,095	4,614	1,337	612	99%
	As ppm	47,207	2,771	3,985	19,466	2,589	1,287	107%

Table 38 Comparison of block model and de-clustered samples for Domains A and B for Au g/t, Cu ppm and As ppm

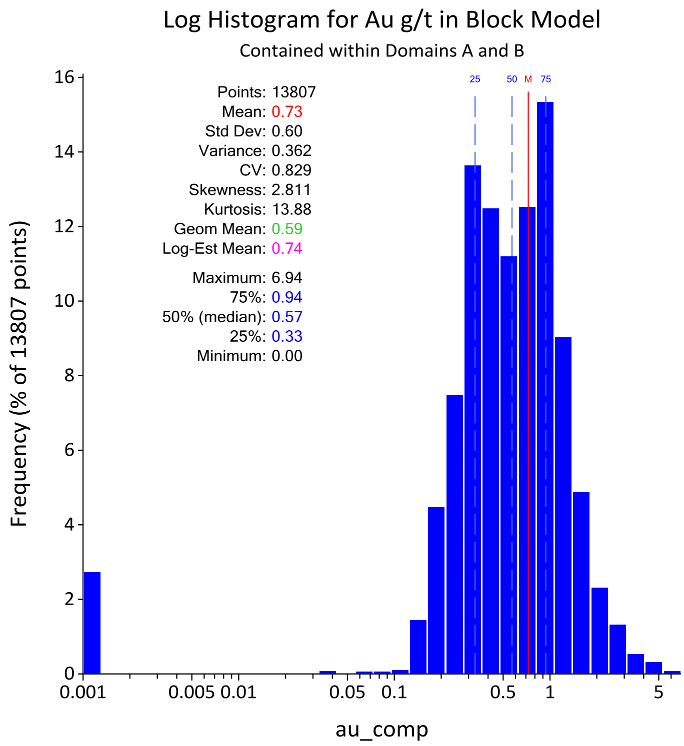

Figure 69 Log Histogram showing the distribution of gold values in the block model

17.8.2 Swath Plots

Swath plots comparing local mean grades in broad "swaths" of the block model and corresponding compoistes were generated for both Domain A (
Figure 70) and Domain B (Figure 71) along Easting 2561375, with a window of ±25m. The northings were split into 25 m steps. The block model shows good correlation on a local level with the input samples.

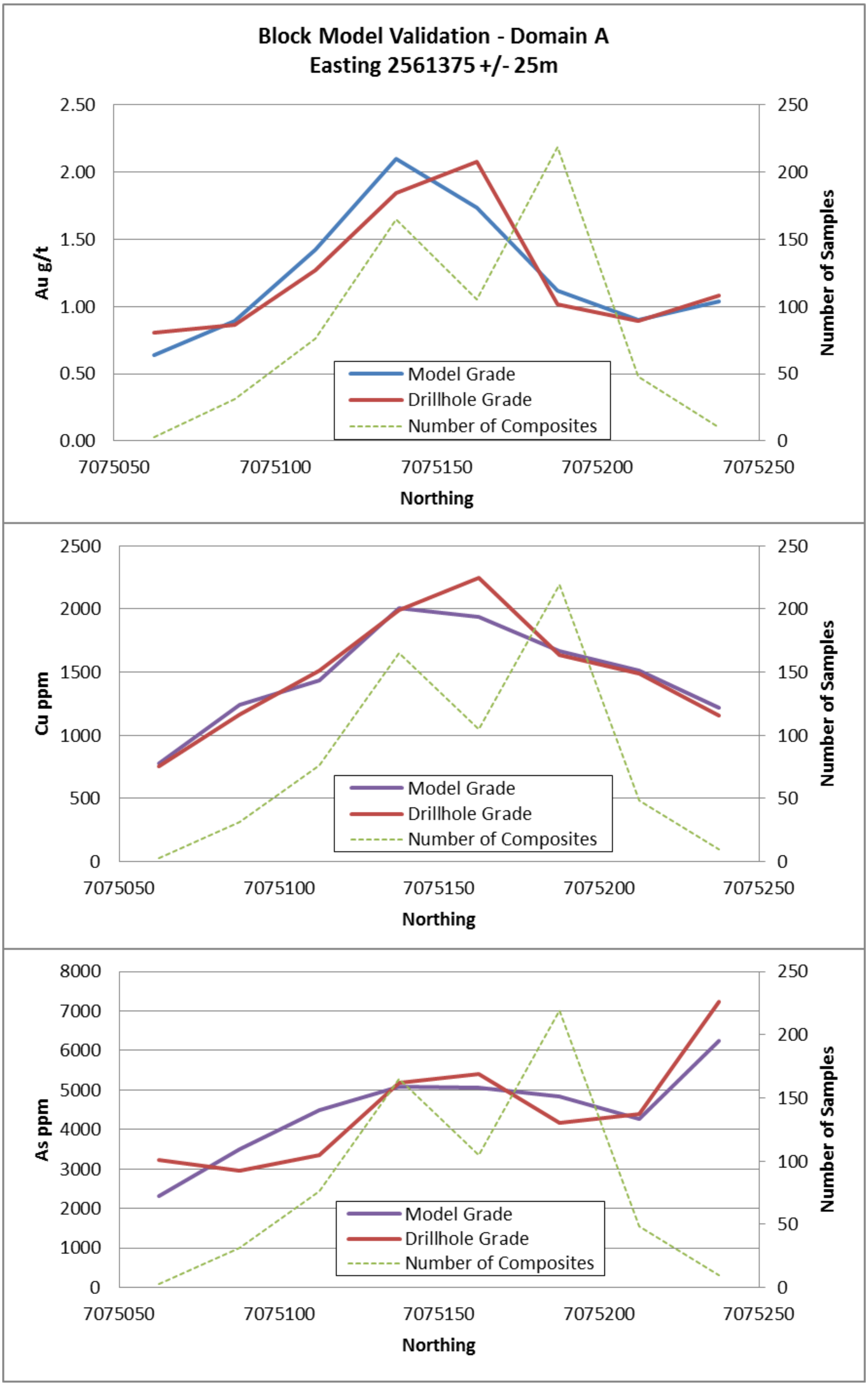

Figure 70 Swath Plots for Au, Cu and As in Domain A along line 2561375E ±25m

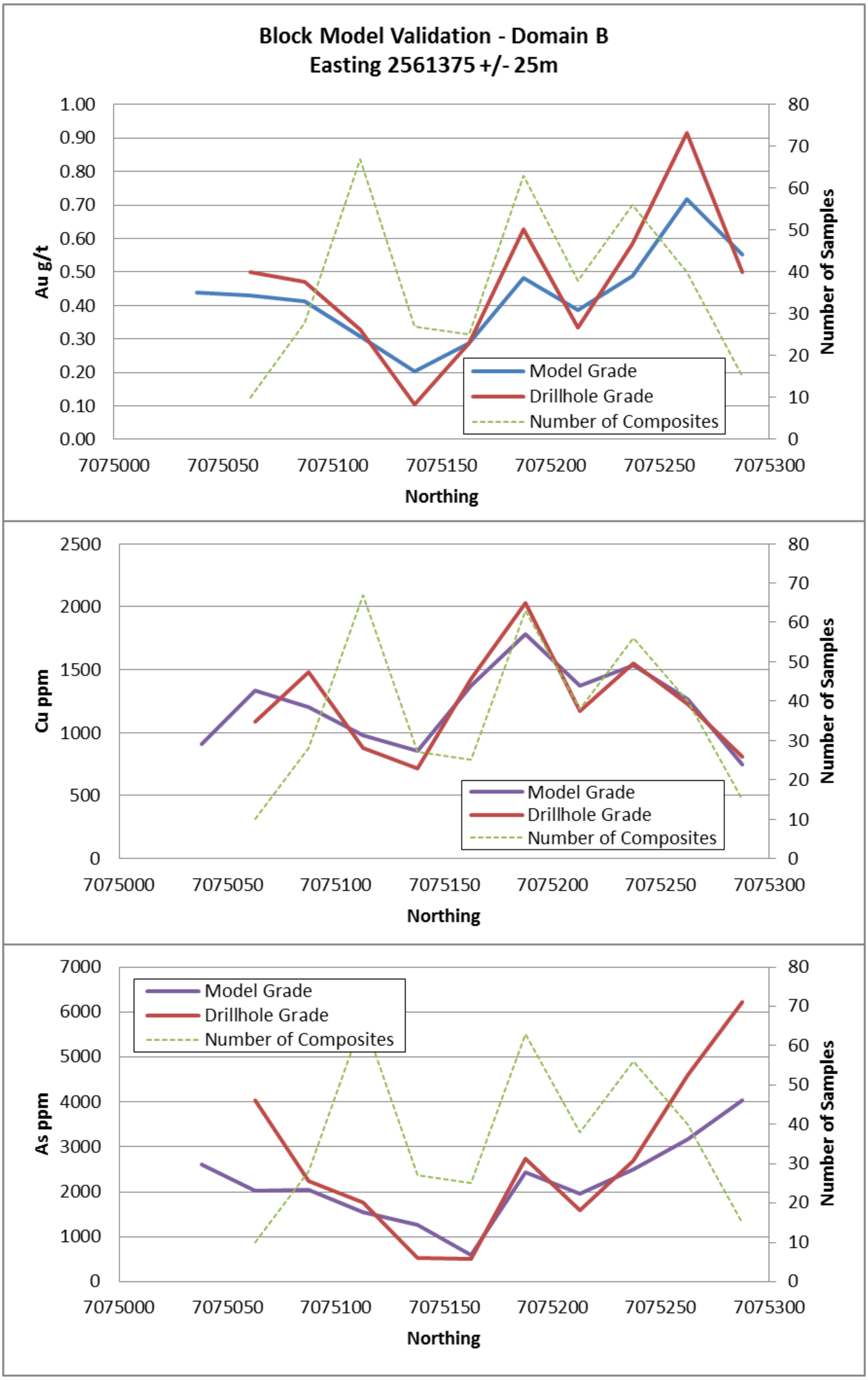

Figure 71 Swath Plots for Au, Cu and As in Domain B along line 2561375E ±25m

17.8.3 Visual Validation

The block model validation includes a visual inspection of block grades versus composite values on vertical sections. This did not show any unusual problem when compared with drillhole grade across sections. Examples of two of these sections are shown in Figure 72.

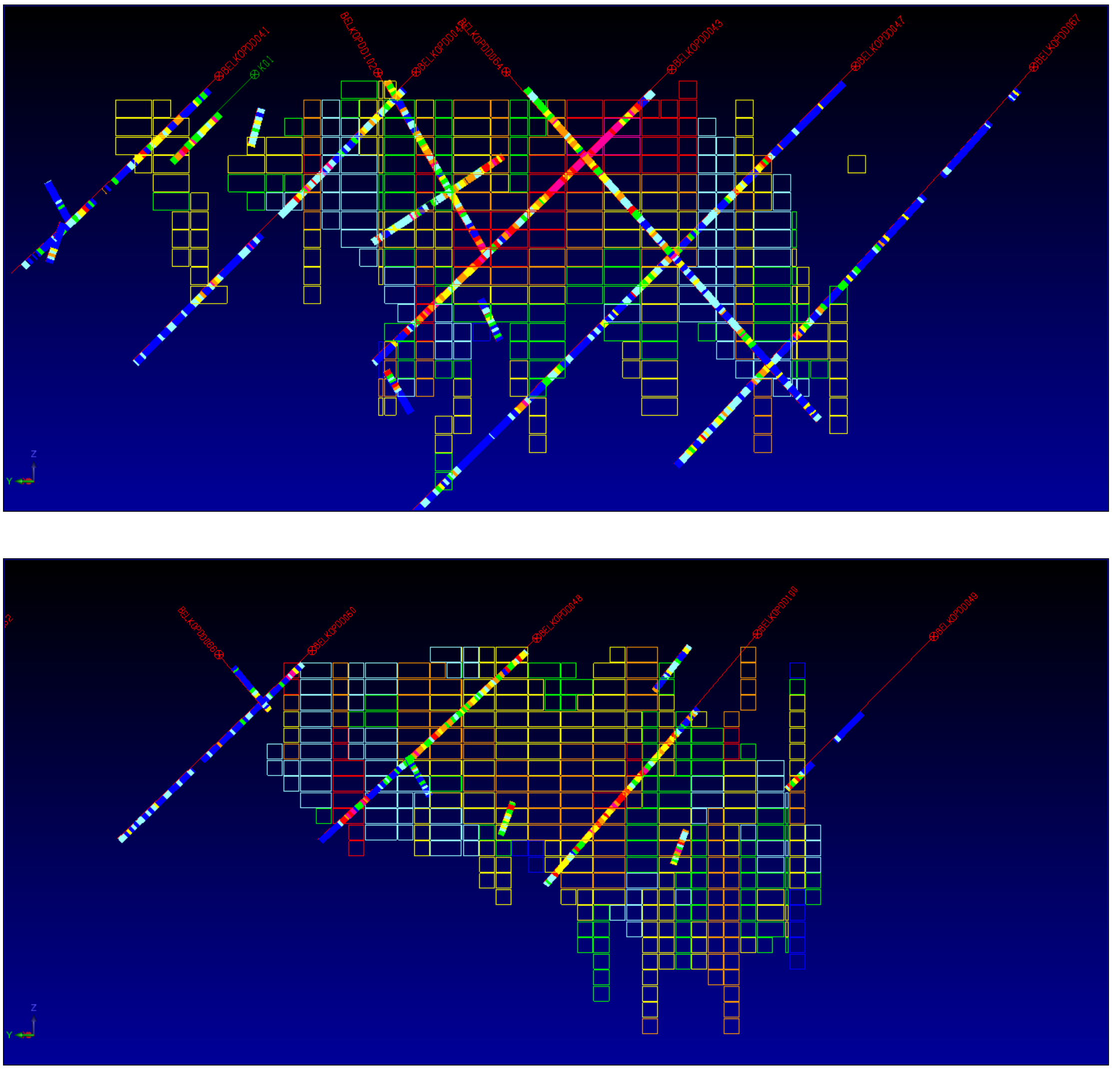

Figure 72 Visual validation of block model estimate compared to raw assay data along two sections. The colours for both the block models and the assays are the same: magenta > 4 g/t; red 2 – 4 g/t; orange 1 – 2 g/t; yellow 0.5 – 1.0 g/t; green 0.3 – 0.5 g/t; pale blue 0.1 – 0.3 g/t; dark blue <0.1 g/t

17.9 Mineral Resource Classification

Classification is based on the density of data and matching between the geological framework and grade continuity. Mineral resources were calculated following the guidelines of the Australasian Code for Reporting of Mineral Resources and Ore Reserves prepared by the Joint Ore Reserve Committee in 2004 (JORC Code, see http://www.jorc.org/).

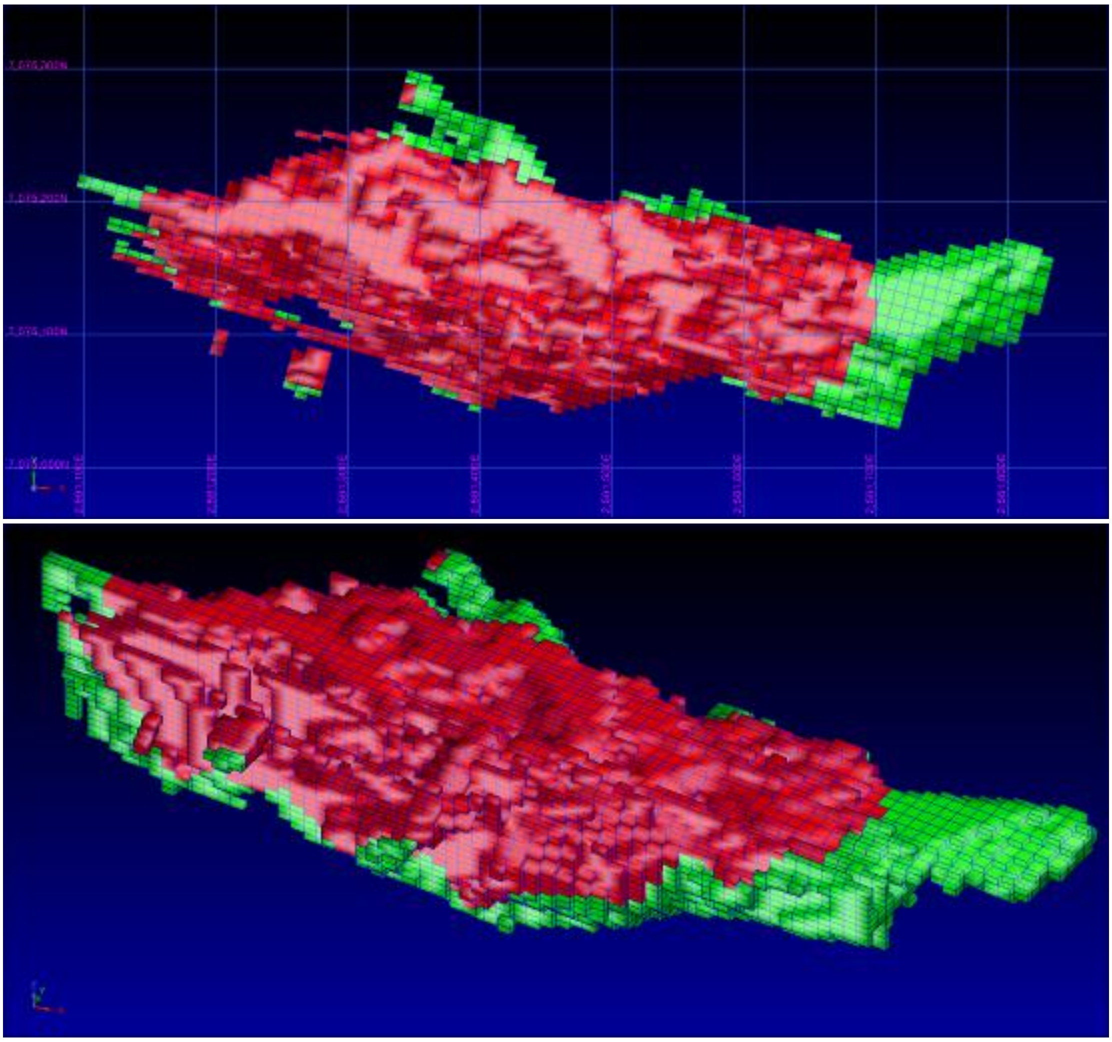

Figure 73 Blocks classified as Indicated (red) and Inferred (green) in the Kopsa deposit

17.10 Mineral Resource Estimate – 29th October 2012

The qualified Person considers the mineralisation contained within the Kopsa deposit to fulfil the criteria of “potentially economic” to be reported as a resource.

Tables 39 and 40 summarise the October 29th, 2012 Mineral Resource estimate based on several cut-off grades, a density of 2.73 g/cm3, and down to a maximum vertical depth of z = -15m, roughly equivalent to a vertical depth of 125 metres below bedrock surface. A reasonable cut-off grade for modelling and reporting the Kopsa resources has been set to 0.4 g/t Au. As previously stated, this cut-off grade is considered to be a reasonable starting-point should Kopsa be developed into an open pit mine. However this does not necessarily imply any economic feasibility at this time.

As copper is potentially an economic by-product of any mining at Kopsa, a gold equivalence has been reported to provide some indication of the possible economic significance a copper by-product may have. Gold equivalence calculations are based on a gold price of $ 1,200 /ounce and a copper price of $ 6,000 /tonne.

The October 29th, 2012 Mineral Resource estimate defines an Indicated Resource of 6.68 million tonnes at an average grade of 1.04 g/t Au and an Inferred Resource of 1.8 million tonnes at 0.76 g/t Au using a cut-off grade of 0.4 g/t Au (highlighted in blue). Tonnage and contained metal figures have been rounded to the appropriate levels after calculations. No recoveries or dilution factors have been considered in this estimate and the results should be considered strictly *in situ*, in accordance with NI 43-101 reporting guidelines for resources.

Indicated Mineral Resource at Kopsa						
Au Cut-off g/t	**Tonnes**	**Au g/t**	**Cu ppm**	**As ppm**	**Au Ounces**	**Au_equiv. g/t**
0.3	8,180,000	0.91	1463	4468	240,000	1.14
0.4	6,680,000	1.04	1526	4886	223,000	1.28
0.5	5,830,000	1.13	1562	5154	211,000	1.37
0.6	5,140,000	1.20	1584	5341	199,000	1.45
0.7	4,500,000	1.28	1610	5513	186,000	1.53
0.8	3,860,000	1.37	1643	5702	170,000	1.62
0.9	3,110,000	1.50	1698	5937	150,000	1.76
1.0	2,460,000	1.64	1750	6215	130,000	1.91
1.5	950,000	2.33	1963	6977	71,000	2.64
2.0	480,000	2.94	2123	7748	45,000	3.27

Table 39 Indicated Mineral Resources at Kopsa on October 29th, 2012

Inferred Mineral Resource at Kopsa						
Au Cut-off g/t	**Tonnes**	**Au g/t**	**Cu ppm**	**As ppm**	**Au Ounces**	**Au_equiv. g/t**
0.3	2,300,000	0.67	1753	4725	49,000	0.94
0.4	1,800,000	0.76	1761	5191	43,000	1.03
0.5	1,300,000	0.87	1711	5801	37,000	1.13
0.6	1,100,000	0.95	1734	6374	32,000	1.21
0.7	800,000	1.05	1821	6656	26,000	1.33
0.8	600,000	1.13	1872	6654	22,000	1.42
0.9	400,000	1.24	1955	6877	18,000	1.54
1.0	300,000	1.35	2056	7382	14,000	1.67
1.5	100,000	2.06	2119	10716	4,000	2.39
2.0	20,000	2.73	1797	15674	2,000	3.01

Table 40 Inferred Mineral Resources at Kopsa on October 29th, 2012

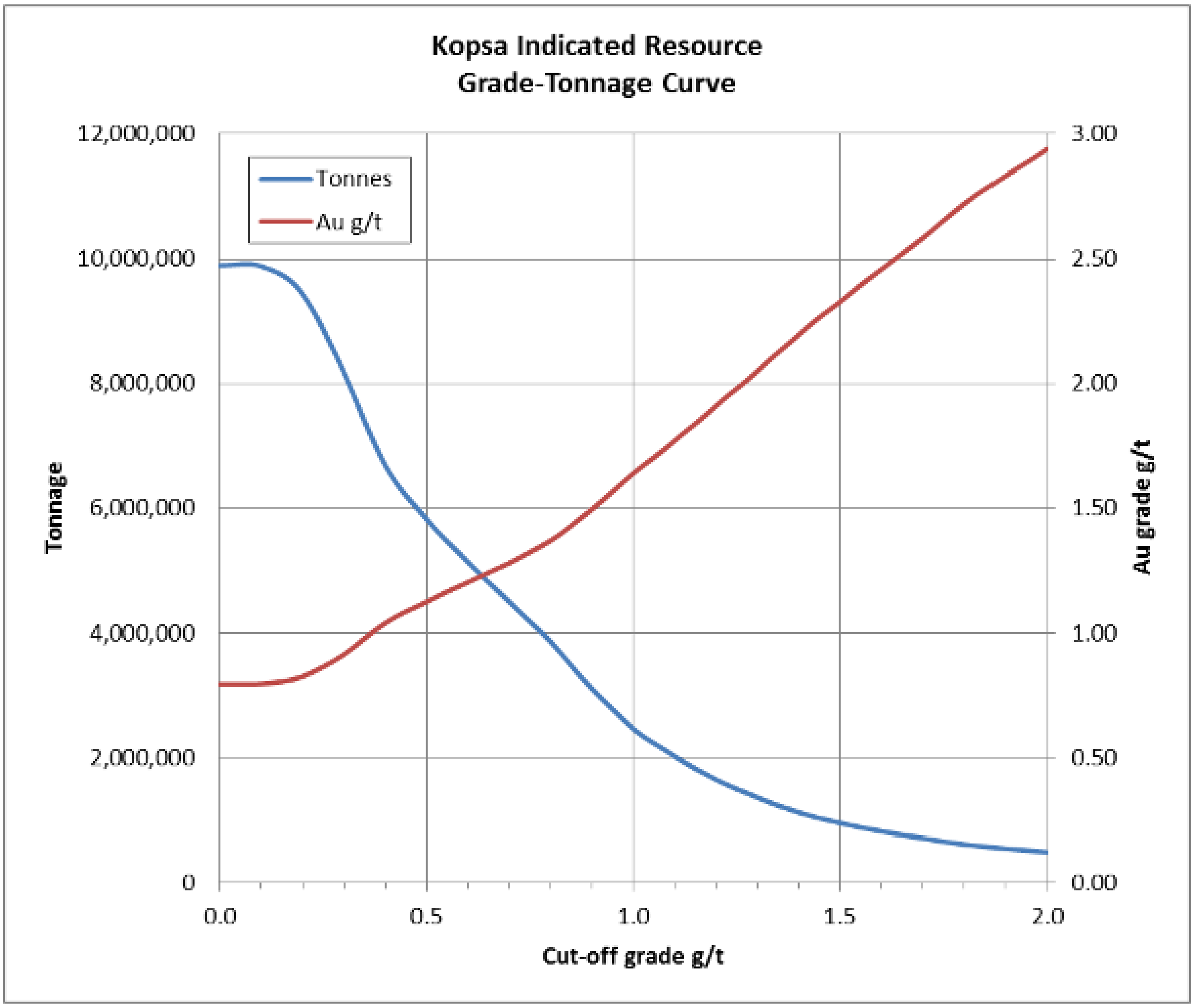

Figure 74 Grade-tonnage curve for the Indicated Mineral Resource at Kopsa

In preparation of the October 29th, 2012 Mineral Resource Estimate for the Kopsa deposit, the Qualified Person is not aware of any known environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues that may materially affect the Mineral Resource estimate

17.11 Exploration Potential

The Kopsa Au-Cu project resources have only been focussed on the Main Zone mineralisation. This remains open along strike, both to the east and west, although at the eastern end in particular, the mineralisation has become more copper-rich with lower gold grades. The Main Zone also remains open down-dip to the south, and at depth. However, some of the best exploration potential for the Main Zone exists in extending the hanging wall intersections of BELKOPDD0037, -68 and -69.

In addition to the Main Zone (Figure 75), relatively little exploration has been conducted on the North Zone, and virtually none on the IP anomaly to the south of the Main Zone (South Zone). All of these offer reasonable potential for delineating more resources.

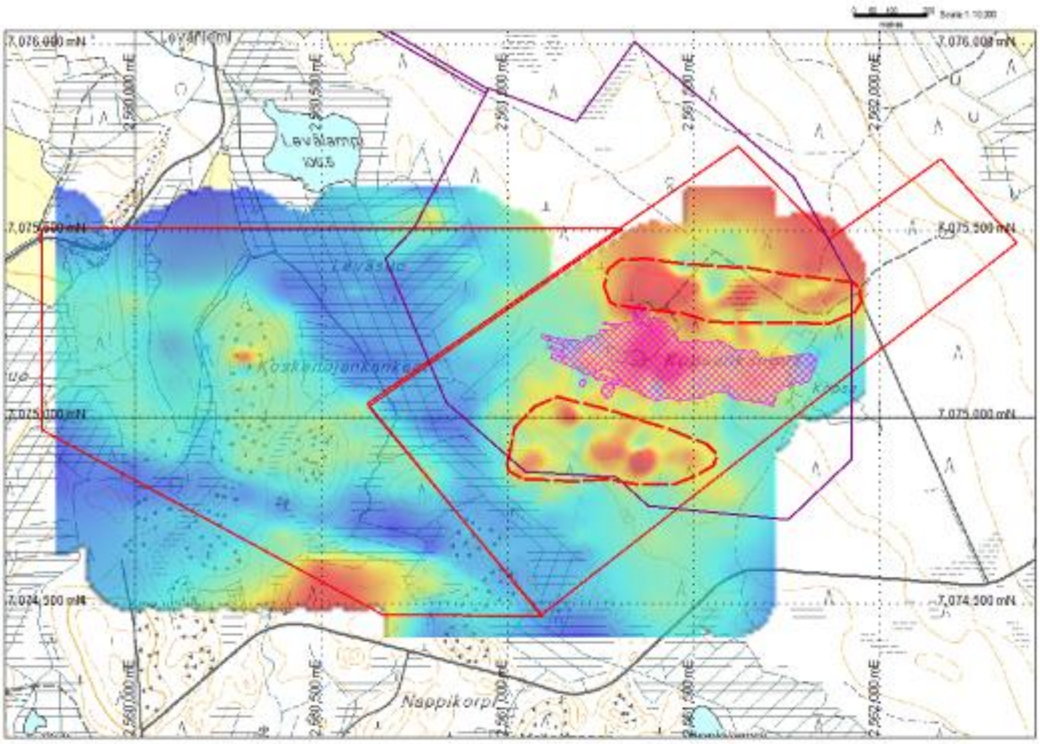

Figure 75 Compilation map of IP surveys with North and South Zone exploration targets

17.12 Mineral Reserve Estimate

Since no economic evaluation study has been completed to date on the Kopsa deposit, no conversion of Resources to Reserves can be done at this stage.

18 Other Relevant Data and Information

Mining projects in Finland need a Mining Concession as well as an Environmental Permit, the latter including a permit to take and discharge water. A Mining Concession is granted by the Mining Inspector. The Environmental Permit is granted after a permitting process that includes the approval of an Environmental Impact Assessment and a legal process at the Environmental Court.

Once the permit is approved the supervision is carried out by the regional environmental center. An Environmental permit is normally re-negotiated after a few years of operation.

Belvedere has applied for the Mining Concession for Kopsa, but has not yet applied for the Environmental Permit.

19 Conclusions and Recommendations

This initial resource estimate for the Kopsa gold-copper deposit is reported at a 0.4 g/t cut-off. This is considered to be potentially economic cut-off for Kopsa for a number of reasons:

Kopsa is only 20km by road from Belvedere's Hitura Nickel Mine. In 2010, Hitura was granted the environmental permit for extending the tailings area. This included a lined tailings area which could be utilised for gold ores, subject to the chemical parameters of the tailings material being supplied and accepted by the environmental agency responsible. The processing studies to date have shown that the Kopsa tailings could be made very clean with respect to arsenic and sulphur.

The Kopsa mineralisation is close to the surface with much of the mineralisation at or just below bedrock surface. Table 41 shows the breakdown of the Kopsa indicated resources according to depth below surface. From this it is apparent that most (>70%) of the resources, in terms of tonnages, grade and contained gold ounces, are in the top 50 metres. Consequently, it is possible to envisage an open pit mining scenario which would benefit from initial low stripping ratios and higher grade ores. This should be hugely beneficial to the economics of the project if developed.

Depth (vertical m)								**Cum %**
From	**To**	**Tonnes**	**Au g/t**	**Cu ppm**	**As ppm**	**Au_eq**	**Au Oz**	**Oz's**
0	10	460,692	1.13	1395	4726	1.35	16,712	7.5%
10	20	900,221	1.16	1573	4937	1.40	33,524	22.5%
20	30	1,039,452	1.19	1583	5036	1.44	39,775	40.3%
30	40	1,040,818	1.15	1534	5439	1.39	38,630	57.6%
40	50	980,757	1.05	1541	5468	1.28	32,977	72.4%
50	60	800,577	0.94	1508	4981	1.18	24,267	83.3%
60	70	600,602	0.82	1455	4089	1.04	15,795	90.4%
70	80	455,915	0.76	1460	3741	0.99	11,168	95.4%
80	90	260,035	0.79	1612	3991	1.04	6,609	98.3%
90	100	120,805	0.77	1574	4003	1.01	2,991	99.7%
100	110	24,572	0.94	1372	4312	1.15	746	100.0%
		6,684,446	1.04	1526	4886	1.28	223,193	

Table 41 Indicated Resources at 0.4 g/t Au cut-off subdivided by vertical depth from surface (z=110)

The proximity of the Hitura Nickel Mine with it's existing infrastructure, would therefore have enormous economic significance regarding potential capital expenditures. No studies have yet been done to quantify this, and it is recommended that on completion of the processing studies a full scoping study be completed to determine the optimum scale of the project, and the likely economics.

Beyond this, it is recommended that further drilling be conducted on the North and South Zones, and to fully close off the Main Zone.

20 References

Eilu P. 1999. FINGOLD - a public database on gold deposits in Finland. Tiivistelmä: FINGOLD - julkinen tietokanta Suomen kultaesiintymistä. Geologian tutkimuskeskus, Tutkimusraportti Geological Survey of Finland, Report of Investigation 146, 224 p.

Eilu P. 2007. FINGOLD : brief descriptions of all drilling-indicated gold occurrences in Finland - the 2007 data. Geologian tutkimuskeskus, Tutkimusraportti Geological Survey of Finland, Report of Investigation 166, 35 p.

Ekdahl, E. 1993. Early Proterozoic Karelian and Svecofennian formations and the evolution of the Raahe-Ladoga Ore Zone, based on the Pielavesi area, central Finland. Geological Survey of Finland. Bulletin 373. 137 p. + 2 app., 2 app. maps.

Gaál, G. 1986. 2200 million years of crustal evolution : the Baltic Shield. Bulletin of the Geological Society of Finland 58 (1), 149-168.

Gaál, G. 1990. Tectonic styles of Early Proterozoic ore deposition in the Fennoscandian Shield. In: G. Gaál, D. I. Groves (eds.) Precambrian ore deposits related to tectonics. Precambrian Research 46 (1-2), 83-114.

Gaal, G. 1997. Unpublished Structural Report for Glenmore Highlands Inc.

Groves, D.I. 1993. The crustal continuum model for late-Archaean lode-gold deposits of the Yilgarn Block, Western Australia. Mineralium Deposita 28, 366–374.

Koistinen, T. (comp.); Stephens, M. B. (comp.); Bogatchev, V. (comp.); Nordgulen, Ø. (comp.); Wennerström, M. (comp.); Korhonen, J. (comp.) 2001. Geological map of the Fennoscandian Shield, scale 1:2 000 000. Espoo : Trondheim : Uppsala : Moscow: Geological Survey of Finland : Geological Survey of Norway : Geological Survey of Sweden : Ministry of Natural Resources of Russia.

Kontoniemi, O. 1998. Geological setting and characteristics of the tonalite and shear zone hosted Paleoproterozoic gold deposit at Osikonmäki, Rantasalmi, southeastern Finland : synopsis. Espoo: Geological Survey of Finland. 15 p.

Kontoniemi, O. 2009. Ore mineralogy and structural features of the Kopsa deposit. Kokkola. Geological Survey of Finland. Unpublished Report. 31 p.

Korsman, K. (ed.) 1988. Tectono-metamorphic evolution of the Raahe-Ladoga zone. Geological Survey of Finland. Bulletin 343. 96 p.

Kousa, J., Lukkarinen, H. & Lundqvist, T. 2000. Proterozoic crystalline rocks. In: Lundqvist, Th. & Autio, S. (eds.) Description to the bedrock map of central Fennoscandia (Mid-Norden). Geological Survey of Finland. Special Paper 28, 25-75.

Kretschmar, U. & Scott, S.D. 1976. Phase relation involving arsenopyrite in the system Fe-As-S and their application. The Canadian Mineralogists 14, 346-386.

Lehtinen, M. (ed.); Nurmi, P. A. (ed.); Rämö, O. T. (ed.) 2005. Precambrian geology of Finland: key to the evolution of the Fennoscandian Shield. Developments in Precambrian Geology 14. Amsterdam: Elsevier. 736 p.

Moore, J. 2004. Petrographic Report on 5 Samples from Kopsa Deposit, Finland. Belvedere Resources Ltd Unpublished Report. 9pp.

Nurmi, P. A., Lestinen, P. & Niskavaara, H. 1991. Geochemical characteristics of mesothermal gold deposits in the Fennoscandian Shield, and a comparison with selected Canadian and Australian deposits. Geol. Surv. Finland, Bulletin 351. 101 p.

Ojala J. & Iljina M. (eds.) 2007. Metallogeny and tectonic evolution of the Northern Fennoscandian Shield : field trip guidebook. Geologian tutkimuskeskus, Opas Geological Survey of Finland, Guide 54, 98 p., 2 apps.

Pym, D. 2003. Structural analysis of oriented drilling at Kopsa. Belvedere Resources Ltd Unpublished Report. 5pp.

Pym, D 2003b. Preliminary interpretations on structure and vein paragenesis, Kopsa. Belvedere Resources Ltd Unpublished Report. 7pp

Sorjonen-Ward, P. 2005. Appraisal of structural geometry and history at the Kopsa prospect. Unpublished Report. 28pp.

Strauss, T. 1999. Kopsa Claims, Report on Exploration Activities. Glenmore Highlands Inc. Unpublished Report for Ministry of Trade and Industry, Finland.

Strauss, T. 2010 Summary of Kopsa structures and interpretation. Belvedere Resources Ltd Unpublished Report 16pp.

21 Certificate of Qualified Person

Pekka Lovén
Puolikkotie 8, Po Box 515
FI-02201 ESPOO, FINLAND
Telephone: +358 40 8200517
Fax: +358 (0)20 529 2549
Email: pekka.loven@outotec.com

1. I am Senior Technology Adviser - Mining:

Outotec (Finland) Oy
Puolikkotie 8, Po Box 515
FI-02201 ESPOO, FINLAND

2. I graduated from the Helsinki University of Technology with a Master of Science (Mining) in 1980.

3. I am a member of the Australian Institute Of Mining and Metallurgy (AusIMM) with Chartered Professional accreditation.

4. I have worked as a mining engineer for a total over 30 years since my graduation from the university. I have been involved in mining engineering roles at open pit and underground copper-zinc-cobalt, iron-vanadium, iron, zinc-lead, nickel-cobalt, copper, chromium and precious metals mines in Finland, Sweden and Ireland.

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

6. I am responsible for the NI 43-101 Technical Report titled "Kopsa Au-Cu deposit, central Ostrobothnia, Finland" dated 29th October 2012. The mineral resource estimation was completed by myself together with Markku Meriläinen (AusIMM) of Outotec (Finland) Oy. The Technical Report was prepared and written under my supervision by Toby Strauss (CGeol) and David Pym of Belvedere Resources Ltd.

7. I visited the Kopsa property on 27th May, 2011, and inspected the drill core at Belvedere's core repository on 2nd February, 2011.

8. I am not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not reflected in this report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of Belvedere Resources according to the definition in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101FI and the Technical Report has been prepared in compliance with that instrument and form.

11. 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 in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 29th Day of October, 2012

Pekka Lovén