Nevsun Resources Limited, Bisha Polymetallic Operation, Eritrea, Africa
NI 43-101 Technical Report

Prepared for:
Nevsun Resources Limited

Prepared by:
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Project No. 162460

Effective Date: 01 January 2011

CERTIFICATE OF QUALIFIED PERSON

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I, David G. Thomas, P.Geo., am employed as a Principal Geologist with AMEC Americas Ltd.

This certificate applies to the technical report entitled "Nevsun Resources Limited, Bisha Polymetallic Operation Eritrea, Africa, NI 43-101 Technical Report" (the "Technical Report"), dated 1 January 2011.

I am a Professional Geologist registered in British Columbia (P.Geo NRL # 149114) and a member of the Australasian Institute of Mining and Metallurgy (MAusIMM # 225250). I graduated in 1993 from Durham University, in the United Kingdom with a Bachelor of Science degree and in 1995 from Imperial College, University of London, in the United Kingdom with a Master of Science degree.

I have practiced my profession for over 15 years. In that time I have been directly involved in review of exploration, geological models, exploration data, sampling, sample preparation, quality assurance-quality control, databases, and mineral resource estimates for a variety of mineral deposits, including volcanic hosted massive sulphide deposits.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Bisha Polymetallic Operation (the "Project") from 7 to 9 May, 2009.

I am responsible for Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13, 14, 15, 17.1, 17.3, 19, 20, 21, 22, and 23 of the Technical Report.

I am independent of Nevsun Resources Limited as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Project since 2009 as part of mineral resource estimation, and for the purposes of preparing this Technical Report.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

“Signed and sealed”

David G. Thomas P.Geo

Dated: 28 March 2011

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I, Jay Melnyk, P.Eng, am employed as a Principal Mining Engineer with AMEC Americas Ltd.

This certificate applies to the technical report entitled “Nevsun Resources Limited, Bisha Polymetallic Operation Eritrea, Africa, NI 43-101 Technical Report” (the “Technical Report”), dated 1 January 2011.

I am a Professional Engineer in the province of British Columbia (P.Eng #25975). I graduated from the Montana Tech of the University of Montana with a Bachelor of Mining Engineering degree in 1988, and from the British Columbia Institute of Technology with a Diploma in Mining Technology in 1984.

I have practiced my profession for 22 years. I have been directly involved in open pit mining operations, and design of open pit mining operations in Indonesia, Canada, the United States, Chile, Peru and Mexico.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Bisha Polymetallic Operation (the “Project”).

I am responsible for Sections 17.2, 17.3, 18.1, 18.4 to 18.10, and those portions of the Summary, Conclusions and Recommendations that pertain to those sections of the Technical Report.

I am independent of Nevsun Resources Limited as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Project since November 2010, during preparation of this Technical Report.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and sealed"

Jay Melnyk, P.Eng.

Dated: 28 March 2011

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I, Alexandra J. Kozak, P.Eng., am employed as Manager, Process Engineering with AMEC Americas Limited.

This certificate applies to the technical report entitled "Nevsun Resources Limited, Bisha Polymetallic Operation Eritrea, Africa, NI 43-101 Technical Report" (the "Technical Report"), dated 1 January 2011.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC). I graduated from the University of Alberta with a Bachelor of Science degree in Mineral Process Engineering in 1985.

I have practiced my profession continuously since 1985 and have been involved in operations in Canada and Guyana and preparation of scoping, pre-feasibility, and feasibility level studies for gold, base metals and diamond properties in Canada, United States, Peru, Mexico, Mongolia, Ghana, and New Guinea. I am currently a Consulting Engineer and have been so since September 1996.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Bisha Polymetallic Operation (the "Project).

I am responsible for Sections 16 and 18.2 to 18.3 and those portions of the Summary, Conclusions and Recommendations that pertain to those sections of the Technical Report.

I am independent of Nevsun Resources Limited as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Project since November 2010 as part of preparation of this Technical Report.

I have read NI 43–101 and this Technical Report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

"Signed and sealed"

Alexandra J. Kozak, P.Eng.

Dated: 28 March 2011

CERTIFICATE OF QUALIFIED PERSON

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I, Vikram Khera, am employed as a Financial Analyst with AMEC Americas Limited.

This certificate applies to the technical report entitled "Nevsun Resources Limited, Bisha Polymetallic Operation Eritrea, Africa, NI 43-101 Technical Report" (the "Technical Report"), dated 1 January 2011.

I am a member of Professional Engineers and Geoscientists of BC. I graduated from the University of British Columbia in 2002 with a Bachelor of Applied Science degree in Chemical Engineering.

I have practiced my profession for 8 years since graduation.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Bisha Polymetallic Operation (the "Project").

I am responsible for Sections 18.11, 18.12 and 18.13 and those portions of the Summary, Conclusions and Recommendations that pertain to those sections of the Technical Report.

I am independent of Nevsun Resources Limited as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Project since November 2010, during preparation of this Technical Report.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and sealed"

Vikram Khera, P.Eng.

Dated: 28 March 2011

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Nevsun Resources Ltd. (Nevsun) by AMEC Americas Limited (AMEC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AMEC's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Nevsun subject to the terms and conditions of its contract with AMEC. This contract permits Nevsun to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, *Standards of Disclosure for Mineral Projects.* Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk.

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1.0 SUMMARY

AMEC Americas Limited (AMEC) was commissioned by Nevsun Resources Ltd. (Nevsun) to prepare an independent Qualified Person's Review and NI 43-101 Technical Report (the Report) for the Bisha polymetallic operation (the Project) located in Eritrea.

Nevsun will be using the Report in support of the company's Annual Information Form (AIF) filing with the Canadian Securities Administrators and in support of a press release entitled "Nevsun Increases Mineable Reserves at Bisha by 40% Using Updated Metal Prices" and dated 28 March 2011.

The Bisha Project is held within an Eritrean company, Bisha Mining Share Company (BMSC). The shareholder structure of BMSC is 60% Nevsun and 40% Eritrean National Mining Corporation (ENAMCO); with the ENAMCO shareholding comprising a 30% paid participating interest and a 10% free participating interest as provided by the country's mining legislation.

1.1 Location and Access

The Project is located 150 km west of Asmara (233 km by road), 43 km southwest of the regional town of Akurdat (Agordat on some maps), and 50 km north of Barentu, the regional, or Zone Administration Centre, of the Gash-Barka District, in Eritrea, East Africa.

Access to the Bisha Project is by paved road from Asmara to Akurdat, a distance by road of 181 km. From Akurdat, access is via an all-weather unpaved road. Dirt roads and tracks cross the Project but no paved roads exist west of Akurdat.

1.2 Mineral Tenure, Surface Rights, and Royalties

The Project comprises a mining licence covering an area of 16.5 km^2, a mining agreement area covering an area of 39 km^2, and an exploration licence covering an area of 71 km^2 for a total surface area of 110 km^2. Licences are held in the name of BMSC, and BMSC is the operator for the Project.

BMSC has surveyed the boundaries of the Mining License Area in accordance with the Mining Proclamation law. BMSC is not required to survey the Mining Agreement Area or to place Mining Agreement Area boundary markers. Exploration licences also do not require survey.

Under the terms of the Mining Agreement, BMSC has the exclusive right of land use in the Mining License Area that is granted within the Mining Agreement Area. This right is subject to the acquisition and settlement of any third-party land-use rights by payment of compensation and/or relocation at the expense of BMSC.

Royalties payable include an Eritrean Government royalty of 5% of precious metal net smelter return (NSR) and 3.5% of base metal NSR.

1.3 Permits

BMSC holds all the necessary permits to support a mining operation.

1.4 Environment

The Terms of Reference (ToR) for the project environmental and socio-economic and environmental impact assessment (SEIA) were approved by the Eritrean Ministry of Energy and Mines (the Ministry) in March 2006. An SEIA report was completed in December 2006 and submitted to the Ministry.

A Mining Licence for the Project was issued on May 26, 2008. Issuance of the licence is accepted by Nevsun as SEIA approval by the Impact Review Committee, as required under the Eritrean Mining Regulations.

The port facilities for concentrate export are not finalized, and concentrate would not be shipped until the projected third year of production; therefore the environmental impact assessment (EIA) of these facilities will constitute a separate application.

1.5 Geology and Mineralization

Mineralization found to date within the Project is typical of precious and base metal-rich volcanogenic massive sulphide (VMS) deposits.

Eritrea is divided into several north or northeast trending Proterozoic terranes, which are separated by major crustal sutures. The Nacfa Terrane comprises low-grade metamorphosed calc-alkaline volcanics and sediments, and hosts base metal mineralization in the region surrounding the city of Asmara, and in the Gash-Barka district, including the Bisha polymetallic mineralization.

The VMS deposits at Bisha are hosted by a tightly and complexly folded, intensely foliated, bimodal sequence of generally weakly stratified, predominantly tuffaceous metavolcanic rocks. Felsic lithologies appear to directly host the mineralization, predominate overall, and form the hanging wall stratigraphy. A significant component of mafic metavolcanic rocks occurred in the more obviously bimodal footwall, which is exposed mainly to the east of the known mineralized zones.

The Bisha Main Zone deposit extends for over 1.2 km along a north-trending strike, and has been folded (and overturned, dipping to the west) into an antiform so that there are two western and one eastern lenses. The thickness of the lenses is variable from 0 m to 70 m. The primary sulphide zone is below the weathering zone. The massive sulphide lenses can locally exceed 70 m in true thickness and show typical copper-rich bases and zinc-rich tops.

Deep weathering has affected Bisha Main Zone lenses that occur in low-lying areas by removing most of the sulphide and producing high-grade supergene blankets enriched in gold, copper, and lead in particular. The gossan zone can vary in composition from highly siliceous and somewhat ferruginous to a massive goethite–hematite–jarosite gossan. The depth of oxidation appears to be on the order of 30 m to 35 m in outcrop areas, but is variable in sand-covered areas.

The oxidation of the massive sulphides generated strong acid solutions that have progressively destroyed the sulphides and host rock. A horizon of extremely acid-leached material or “soap” has developed between the oxide and supergene/primary domains.

The Harena deposit has been traced over a strike length of 400 m, and is interpreted to be a northwest-dipping, tabular massive sulphide body, closed off by drilling to the northwest, but open to the southeast. Host rocks to the Harena deposit are a bimodal, hydrothermally-altered suite of basalts and rhyolite-dacite volcanics. Surficial weathering processes have produced three distinct zones of mineralization. These include a surface oxide/gossan overlaying a secondary supergene horizon, which grades into a primary massive sulphide horizon at depth. The gossanous horizon contains frequently anomalous levels of gold and silver. Oxide and sulphide mineralized zones are approximately 400 m in length and vary in thickness between 5 m and 15 m. The average grades of the oxides are 1.2 g/t gold, 14.2 g/t silver, 0.1% copper and 0.21% zinc, and the average grades of the massive and semi-massive sulphides are 0.84% copper, 0.41 g/t gold, 23.75 g/t silver, and 3.72% zinc.

Additional prospects are known within the Project area; the most advanced is the Northwest (NW) Zone, located approximately 1.5 km north of the Bisha Main Zone.

1.6 History and Exploration

Nevsun has no record of any previous exploration or mining activities on the Project or surrounding areas prior to 1996.

Work performed by Nevsun since 1996 has included geological mapping, trenching, geophysics (airborne and ground), geochemical sampling, petrographic work, bulk density measurements, geotechnical work, environmental baseline work, metallurgical testwork, and mineral resource and mineral reserve estimation. A preliminary assessment (PA) was completed in 2005, and a feasibility study in 2006. The costs associated with the 2006 Feasibility Study were updated in 2008, and mine construction commenced the same year.

1.7 Drilling

Drilling on the Project has been undertaken in a number of core and one RC campaign from 2002 to 2011. Drilling comprised a total of 578 drill holes (78,880.96 m), of which 545 were core drill holes (76,783.66 m) and 33 were RC drill holes (2097.3 m). Drill programs have been completed primarily by contract drill crew, supervised by Nevsun geological staff.

A total of 356 drill holes support estimation at the Bisha Main Zone. Much of the massive sulphide mineralization in the Bisha Main Zone has been well defined by drilling patterns of 25 m spaced holes on sections spaced 12.5 or 25 m apart. This density decreases with depth on the deepest portions of the primary mineralization. The deposit remains open at depth in the south, with the deepest intersections obtained to date returning long lengths of medium- to high-grade zinc mineralization

Core was logged for geological and geotechnical parameters, and photographed. Drill collar locations have been verified by survey. Down-hole surveys were not performed for the first 20 drill holes on the Project; all subsequent drilling has been down-hole surveyed using acid tests, Sperry-Sun Single-Shot and Reflex instrumentation.

Average recoveries are 71% in the oxides, 71% in the breccia, 61% in the “soap” lithological unit, 91% in the supergene domain, and 98% in the primary sulphides. As most of the low-recovery assays were associated with the gold-rich oxidized portion of the deposit, a decision was made to remove all of the assays with core recoveries of less than 60% from the database that supported Mineral Resource estimation.

RC samples were 2 m in length. The maximum core sample length is 12.00 m (only within wall rock away from mineralized intervals) and the minimum is 0.15 m. Within the zones of mineralization, samples lengths are generally between 1.00 and 3.00 m. Sample intervals are determined based upon mineralogical and lithological contacts. Samples are considered to be adequately representative of the true thicknesses of mineralization.

One-metre grade control samples weighing approximately 6 kg are split in-pit and submitted to the on-site laboratory for sample preparation.

The bulk density determination procedures are consistent with industry-standard procedures. Nevsun-determined density estimates were confirmed by an independent laboratory re-measuring the same core. There are sufficient density determinations to support the density values utilized in waste and mineralization tonnage interpolations.

1.8 Sample Preparation and Analyses

Sample analytical procedures that support Mineral Resource and Mineral Reserve were performed by independent analytical laboratories without company involvement from 1998 to the present. Analytical laboratories used in the period 2002–2011 include Intertek Testing Services Bondar Clegg Laboratory, Genalysis Laboratory Services Pty Ltd., ALS Chemex Ltd., and ACME Laboratory. The laboratories are ISO-registered, are internationally recognized analytical facilities, and independent of Nevsun. The run-of-mine laboratory was established by SGS Mineral Services, who trained BMSC staff as operators.

Sample preparation and analytical methods employed on the Project are in accordance with industry norms. Sample security was appropriate to the Project location.

Typically, drill programs included insertion of blank, duplicate and SRM samples. The QA/QC program results do not indicate any significant problems with the analytical programs that would preclude use of the data, therefore the analyses from the drill programs are suitable for inclusion in Mineral Resource and Mineral Reserve estimation.

1.9 Data Verification

A number of data verification programs and audits have been performed over the Project history to verify that data collected were sufficiently reliable for the purposes of Mineral Resource and Mineral Reserve estimation, and in support of detailed engineering studies and technical reports. No significant errors or biases that would materially impact the Mineral Resource and Mineral Reserve estimates were identified in the data reviewed.

1.10 Metallurgical Testwork

In 2005, metallurgical testwork was undertaken at SGS Lakefield (SGS) to support feasibility-level studies for the oxide, supergene and primary mineralization material types. Work performed included comminution, variability, cyanidation, and flotation tests, as well as petrographic studies. These tests defined the optimal process flowsheet used for the 2006 Feasibility Study.

In 2010, metallurgical testwork was conducted at Mintek on behalf of SENET to provide plant design and performance expectations of the Bisha supergene mineralization as part of ongoing development of the Bisha Project. As the results of those tests indicated that the ore sample delivered to Mintek did not behave as the previously tested samples at SGS, it was decided that the concentrate production be put on hold and instead conduct scouting investigations on the reasons for the variable response. The approach was modified to use a simplified reagent scheme to enhance copper flotation kinetics and this met with limited success. Poor reproducibility resulted in the concentrate generation program being suspended to investigate the possible reasons for the different results.

Following the testwork at Mintek, Maelgwyn Mineral Services Africa (Pty), South Africa (Maelgwyn) was contracted to duplicate the test program previously attempted at Mintek. The objective of the test program at Maelgwyn was to advance the testwork initiated at Mintek and demonstrate that the results could be reproducible at the given conditions. At the time of reporting batch flotation tests on supergene samples have been completed and further testing is on-going.

Recoveries used in the Mineral Resource and Mineral Reserve estimations are consistent with metallurgical testwork on the various mineralization types. Increased conservatism was used in recovery estimates where the amount of testwork is limited. As a result, the proposed plant design and associated recovery factors are considered appropriate to support Mineral Resource and Mineral Reserve estimation.

1.11 Mineral Resource Estimation

The resource model was prepared in 2005 by AMEC (S. Blower, P.Geo. under the supervision of Mr D. Reddy, P.Geo), and reviewed in 2009 by David Thomas, P.Geo.

The Project database was closed for Mineral Resource estimation purposes as at December 2005. Geological interpretations were completed by Nevsun based on lithological, mineralogical and alteration features logged in drill core, and were digitized by AMEC to form three-dimensional solids for six domains. A block size of 5 m x 5 m x 5 m was used.

Data were composited to 5 m down-hole lengths. Capping was applied to restrict outlier grades. Correlograms were computed assess appropriate distances for search ellipsoid radii.

Estimation was performed using ordinary kriging (OK). To ensure local reproduction of composite grade trends, and to help control grade smearing, the resource model was interpolated by multiple passes of OK within successively larger search radii. A total of three passes were required to fill all of the blocks in the Primary Zn and Primary domains. All of the other domains were completely estimated with two passes. Two different search ellipse orientations were used, one for the oxide and supergene domains, and another for the primary domains. For all passes, a minimum of three and a maximum of 12 composites were used. A maximum of two composites were allowed per drill hole to ensure that multiple holes would contribute to block values.

Mineral Resources were confined within a Lerchs–Grossmann optimized pit shell to assess reasonable prospects of economic extraction.

Mineral Resources are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources are inclusive of Mineral Reserves and do not include external dilution. The Mineral Resource estimate has an effective date of 01 January, 2011. David Thomas, P.Geo, an AMEC employee is the Qualified Person (QP) for the estimate.

The classified Bisha Mineral Resource estimate is summarized in Table 1-1. The oxide stockpile inventory at 01 January 2011 is shown in Table 1-2.

The Northwest Zone prospect is at an early stage of data collection, and insufficient information is available to support mineral resource estimation. However, the number of drill holes, and the dimensions of the mineralized area are sufficient to permit estimation of an Exploration Target of:

- 4 Mt to 11 Mt with grades ranging from 20 g/t to 45 g/t Ag, 0.3 g/t to 0.5 g/t Au, 0.5% to 1.1% Cu and 0.1% to 0.2% Zn.

Resource modeling has been initiated at the Harena deposit.

Table 1-1: Bisha Mineral Resource Estimate – Effective 01 January, 2011 (David Thomas, P.Geo.)

Category	Zone	NSR Cut-Off	Tonnes ('000 t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Au ('000 oz)	Ag ('000 oz)	Cu ('000 lb)	Pb ('000 lb)	Zn ('000 lb)
Measured	Oxides	$29.03	735	6.43	28.37	0.11	0.71	0.11	152	670	1,838	11,518	1,706
	Supergene Cu	$26.57	854	0.77	43.33	4.98	0.16	0.24	21	1,189	93,796	3,081	4,490
	Primary	$26.68	535	0.76	50.72	0.86	0.33	7.68	13	872	10,088	3,900	90,557
	Subtotal Measured		2,124	2.72	40.01	2.26	0.40	2.07	186	2,732	105,722	18,499	96,753
Indicated	Oxides	$29.03	3,671	7.56	32.46	0.08	0.55	0.07	893	3,832	6,767	44,544	5,816
	Supergene Cu	$26.57	6,830	0.73	30.74	3.75	0.10	0.10	160	6,751	564,024	14,385	15,503
	Primary	$26.68	17,224	0.71	43.34	0.95	0.19	5.12	393	23,999	362,493	71,679	1,944,427
	Subtotal Indicated		27,726	1.62	38.79	1.53	0.21	3.22	1,446	34,582	933,284	130,608	1,965,747
M+I	Oxides	$29.03	4,406	7.38	31.78	0.09	0.58	0.08	1,045	4,502	8,605	56,062	7,522
	Supergene Cu	$26.57	7,684	0.73	32.14	3.88	0.10	0.12	181	7,941	657,820	17,466	19,993
	Primary	$26.68	17,759	0.71	43.56	0.95	0.19	5.20	406	24,871	372,581	75,579	2,034,984
	Subtotal M+I		29,849	1.70	38.88	1.58	0.23	3.14	1,632	37,314	1,039,006	149,107	2,062,500
Inferred	Oxides	$29.03	44	3.46	21.93	0.02	0.06	0.02	4.86	31	21	56	18
	Supergene Cu	$26.57	197	0.48	22.09	1.97	0.05	0.03	3.06	140	8,537	210	120
	Primary	$26.68	10,330	0.66	48.38	0.90	0.24	5.80	218.39	16,068	203,942	54,065	1,320,452
	Subtotal Inferred		10,570	0.67	47.78	0.91	0.23	5.67	226.32	16,239	212,500	54,330	1,320,590

Table 1-2: Bisha Oxide Stockpile Inventory Effective January 1, 2011

Category	Zone	Tonnes ('000 t)	Au (g/t)	Au ('000 oz)
Measured	Stockpile	197,235	3.64	23,111

Notes to accompany Mineral Resource Tables:

1. Mineral resources that are not mineral reserves do not have demonstrated economic viability
2. Mineral Resources are inclusive of Mineral Reserves and do not include external dilution.
3. A Lerchs–Grossmann pit shell was used to constrain the Mineral Resources to assess reasonable prospects of eventual economic extraction
4. Mineral Resources are reported using assumed long-term prices as follows: gold price of $1,170/oz, silver price of $18.20/oz, copper price of $2.76/lb, and zinc price of $1.05/lb
5. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
6. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper, lead and zinc pounds as imperial pounds

1.12 Mineral Reserves

The Proven and Probable Mineral Reserves at the Project have been classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves, incorporated by reference in NI 43-101. Mineral Reserves are defined within a mine plan, with open pit phase designs guided by Lerchs–Grossmann optimized pit shells, generated using a gold price of $825/oz, silver at $12.50/oz, copper at $2.00/lb, and zinc at $0.75/lb and considering diluted Measured and Indicated resources. The Qualified Person for the estimate is Jay Melnyk, P.Eng., an AMEC employee.

1.13 Mine Plan

The pre-strip for the Project commenced in March 2010. To 31 December 2010, approximately 4,500 kt of waste had been mined and the ore stockpiles totalled approximately 150 kt. The commissioning of the plant was well advanced with the first gold pour of 26 kg taking place on 30 December 2010. Nevsun declared commercial production on 22 February 2011. Commercial production was defined by Nevsun to be at least 90% of planned throughput and at least 90% of planned recovery for a period of at least 30 days. During the period up to 28 February 2011, the Project had gold proceeds of approximately $57 M.

The mining method is conventional selective open pit mining, with mining rates of approximately 22 kt/d during the oxide phase. Mining rates increase to a peak of just under 50 kt/d later in the mine life as deeper primary mineralization is mined with higher strip ratios.

The Bisha open pit mining operation features a single pit consisting of nine internal phases, and is currently being developed. The first three phases target the oxide ore, phases four to six target supergene ore, and phases seven to nine target the primary ore. The mine has an estimated life of approximately 12 years at the currently planned mill thoughputs of 2.0 Mt/a for oxide, 2.4 Mt/a for supergene and 2.4 Mt/a for primary materials. Mill feed constitutes ore transported directly from the mine plus ore reclaimed from the blending stockpiles.

Table 1-3: Bisha Mineral Reserves, Effective Date: 01 January, 2011, Jay Melnyk, P. Eng.

Ore Type	Ore (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Oxide (above $29.03/t NSR cut-off)					
Proven	933	5.75	21.90		
Probable	3,719	7.39	31.48		
Sub-total Combined	4,651	7.06	29.56		
Supergene (above $26.57/t NSR cut-off)					
Proven	844	0.80	43.47	4.92	
Probable	6,537	0.77	31.29	3.77	
Sub-total Combined	7,382	0.78	32.68	3.90	
Primary (above $26.68/t NSR cut-off)					
Proven	521	0.78	52.51	0.91	8.09
Probable	15,759	0.72	44.12	0.97	5.31
Sub-total Combined	16,279	0.72	44.40	0.97	5.40
Total Proven	2,298	2.80	36.77	2.07	1.98
Total Probable	26,015	1.69	39.09	1.55	3.26
Total Combined	**28,313**	**1.78**	**38.90**	**1.60**	**3.15**

Notes to accompany Mineral Reserve Table:

1. Mineral Reserves are defined within a mine plan, with pit phase designs guided by Lerchs–Grossmann (LG) pit shells, generated using a gold price of $825/oz, silver at $12.50/oz, copper at $2.00/lb, and zinc at $0.75/lb and considering diluted Measured and Indicated resources. Metallurgical recoveries by ore type are 87% Au and 36% Ag for oxide mineralization reporting to dore, 56% Au, 54% Ag and 92% Cu for supergene mineralization reporting to Cu concentrate, 36% Au, 29% Ag and 85% Cu for primary copper mineralization reporting to copper concentrate, and 9% Au, 20% Ag and 83.5% Zn for primary zinc mineralization reporting to zinc concentrate. The mining cost was $1.46/t, plus $0.01/t/5 m bench below the reference elevation of 560 m. The total ore based costs (process, G&A and stockpile rehandle) are $29.03/t for oxide, $26.57/t for supergene and $26.68/t for primary ores. Overall pit slopes varied from 43º to 55.5º in rock and 18.5º in overburden.
2. Mineral Reserves are reported within the above mentioned pit phase designs, using an NSR grade item, where the marginal cut-off is the ore based cost stated above. After completion of the pit designs, the NSR was recalculated using a gold price of $1,015/oz, silver at $15.85/oz, copper at $2.40/lb, and zinc at $0.92/lb. Recoveries used for the NSR calculation are as above with the exception of the supergene copper recovery which was reduced from 92% to 88% based on recent metallurgical testwork.
3. Tonnages are rounded to the nearest 1,000 tonnes, grades are rounded to two decimal places.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper, lead and zinc pounds as imperial pounds
6. The life of mine strip ratio is 4.16
7. Proven oxide Mineral Reserves are inclusive of 197 kt at 3.64 g/t Au in stockpile as of 01 January 2011

A series of bench scale failures have occurred in the central part of the northeast highwall, an area that was lacking geotechnical borehole information due to the challenging topography. Additional geotechnical drilling is underway. The resulting design recommendations will be used to design a layback of the wall, to allow mining of the resources below a safety step-out. Other phase design modifications may be required to allow access to all areas. The mining of the new highwall pushback would likely start in the third or fourth quarter of 2011. However, as this layback has not been designed, it is not included in the current long range mine plan. AMEC does not consider the northeastern wall slope instabilities to be a fatal flaw. Remediation should not be insurmountable and such events are commonly encountered in the early phase of operations while mining companies better understand the geotechnical competence of the rock mass

The current mine plan does not reflect potential changes to the northeast highwall and so is an interim plan. Upon completion of the 2011 geotechnical and hydrological study, the oxide pit phases will be re-designed, and a new, more detailed life-of-mine (LOM) plan will be carried out. This work is expected to result in a higher strip ratio for the oxide portion of the pit.

1.14 Waste Rock Storage

Waste rock will primarily be placed in external waste dumps located to the east and south of the ultimate pit. Over the life of the mine, approximately 146 Mt of waste rock will be produced.

1.15 Process

The oxide plant facilities include a primary crusher, SAG and ball grinding mills, cyanide leach/carbon-in-pulp (CIP) circuit, cyanide destruction circuit, refinery to produce doré, tailings thickener, tailings discharge system and the necessary reagent, water and air systems. The 2 Mt/a oxide processing facility achieved commercial production in February 2011. Over a 30 day period spanning January and February 2011, plant throughput averaged approximately 5,250 t/d with a peak of 6,560 t/d.

The plan outlined in the 2006 Feasibility Study was to mine and process each zone in succession starting with the top oxide zone now in production. Before the oxide mineralization is exhausted, the additional supergene mineralization process equipment will be installed and commissioned, so a smooth transition can be made from the oxide mineralization to the supergene mineralization. There will be some transitional material that is a mixture of oxide and supergene mineralization that will be mined as the oxide is depleted.

The current plan is to treat this material in the copper flotation circuit designed for the supergene material prior to treating the tailings in the cyanide leach circuit with the intent of maximizing copper recovery and preventing any gold losses that may be present in the mixed material. The cyanide leach circuit will not be operated after the transitional ore is depleted.

Similarly, before the supergene mineralization is exhausted, the additional equipment required to process the primary mineralization will be installed and commissioned to permit a smooth transition from supergene mineralization to primary mineralization with minimum interruption and shutdowns.

Additional equipment for the supergene plant will include the supergene mineralization roughing and cleaning flotation circuits, a copper concentrate regrind mill, copper concentrate thickener and pressure filter, copper concentrate load-out building, copper flotation reagent systems, flotation air blowers and pressure-filter air compressors. According to the lasted mine plan, some stockpiling of supergene and primary mineralization will occur, however, due to the previously indicated highly reactive nature of the supergene and primary material noted in the 2005 SGS test program, it is recommended that the oxidation effects on flotation recovery be evaluated.

For treatment of primary mineralization, additional equipment will include zinc roughing and cleaning flotation circuits, zinc concentrate regrind mill, zinc concentrate thickener and pressure filter, zinc concentrate load-out building, zinc flotation reagent systems, additional zinc flotation air blower and zinc pressure filter air compressor.

Based on the current mine plan, the process plant feed rate will transition from Oxide to Supergene feed during the second quarter of 2013, with an associated increase in throughput for the remainder of the year. The throughput rate in 2014 is projected to be 2.4Mt/a with 100% supergene mineralization as feed material to the mill. As this represents a 20% increase in the feed rate, a number of modifications to the current plant equipment will be required. The engineering for the copper phase expansion, is nearing completion. Construction of the copper phase is planned to start later this year.

1.16 Equipment

Mining will proceed with the use of a conventional truck and shovel fleet. The fleet size was revised in 2011, to accommodate a northeastern wall layback. It appears that this fleet will be sufficient for the oxide phase of the mine life, including a very high level estimate of the potential tonnage in the northeastern wall layback. Further fleet additions and replacements will be required later in the mine life.

1.17 Markets

Nevsun has negotiated contracts with two refineries for the sale of the gold–silver doré. Negotiations are underway for the sale of concentrates to be produced from the future phases of the Project. Current contracts for doré are typical of, and consistent with, standard industry practice. The expected contracts for the supply of copper and zinc concentrates are expected to be in line with industry norms.

1.18 Financial Analysis

The results of the economic analysis represent forward-looking information (cashflows, net present value, internal rate of return, production rate, and total metal produced) that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

For the Bisha Project, the majority of the capital cost has already been spent; therefore, the cash flows from 2011 and onwards were discounted. The Project was equity financed and no debt was required.

The analysis incorporated the following key assumptions:

- Mineralization will be processed at an average rate of 2,177 kt/a over a planned mine life of approximately 13 years
- A 5% royalty is payable, for precious metals, on the net smelter return; a 3.5% royalty is payable, for base metals, on the net smelter return
- The following general smelter terms:
  - Average doré treatment charge for gold oxide ore: US$0.30/oz
  - Gold refining charge for gold oxide ore: US$0.20/oz Au
  - Copper treatment charge for supergene copper ore: US$75.00/dmt
  - Copper refining charge for supergene copper ore: US$0.750/lb
  - Zinc pay factor for zinc primary ore: 85%
  - Gold pay factor for zinc primary ore: 70%.
- Long-term base case metal price assumptions of:
  - Gold: US$1,015/oz
  - Copper: US$2.40/lb
  - Silver: US$15.85/oz
  - Zinc: US$0.92/lb.

- Life-of-mine average operating costs assumptions of:
  - Mining: US$2.11/t mined (includes labour, fuel and consumables from 2011 to 2023)
  - Process (includes ore re-handling costs):
    - Oxide Ore: US$24.35/t milled
    - Supergene Ore: US$21.89/t milled
    - Primary Ore: US$22.00/t milled.
  - G&A: US$4.68/t milled.
  - Average annual port operating costs of US$1.8 M.
  - Land Concentrate Freight: US$65/wmt
  - Ocean Freight: US$43.75/wmt
  - Port charges are assumed to be US$7.00/wmt.
- Capital cost assumptions of:
  - From 2009–2010, the capital costs were US$ 237.2 M
  - From 2011–2015, the capital cost estimate is US$119.6 M.
  - Sustaining capital: US$27.5 M
  - Closure and reclamation costs: US$19.3 M. In Year 5 of operations, US$1.82 M is required for reclamation of the southeast waste rock facilities, and the majority of the remaining costs will be spent at closure.
  - Salvage value: US$8.9 M. Assumed to be 2.5% of total capital costs.
- AMEC does not provide expert advice on taxation matters. An income tax rate of 38% was applied over the life-of-mine. The total tax payable over the life of mine is estimated US$739 million based on this assumption. It was assumed that historic costs were deductible for tax purposes. These costs included $28 M for historic exploration and $8.6 M in deferred financing costs.

The base case results pre-tax and post-tax are indicated in Table 1-4. From 2011 to 2015 US$120 M is required for development of the plant phases to treat supergene and primary ores.

Table 1-4: Pre-Tax and Post-Tax Results, Financial Analysis

Pre-Tax		NPV (2011 and onwards)	Capital Costs (2009–2010)	Pre-Production Operating Costs (2010)
NPV 10%	US$ million	**1395**	**($240)**	**($18)**
Payback	Years	**0.8**		
After Tax		**NPV (2011 and onwards)**	**Capital Costs (2009–2010)**	**Pre-Production Operating Costs**
NPV 10%	US$ million	944	($240)	($18)
Payback	Years	1.2		

Sensitivity analysis was performed on the pre-tax base case, taking into account ±30% variations in metal prices, grades, and operating costs. The results of the analysis showed that the Project is most sensitive to changes in grades, then metal prices, and is relatively less sensitive to changes in operating expenditure. Sensitivity analysis to capital expenditures was not conducted as the majority of capital costs have already been spent.

1.19 Conclusions

In the opinion of the QPs, the Project that is outlined in this Report has met its objectives. Mineral Resources and Mineral Reserves have been estimated for the Project, and a mine is under construction, and has produced the first doré. This indicates the data supporting the Mineral Resource and Mineral Reserve estimates were appropriately collected, evaluated and estimated, and the original Project objective of identifying mineralization that could support a proposed mining operation has been achieved. There are likely to be modifications to the long-term mine plan based on geotechnical input; and additional metallurgical testwork has been recommended.

1.20 Recommendations

A two-phase work program was devised for the Project. Recommendations are based on Project review conducted by Nevsun and AMEC. Work in Phase 2, apart from the exploration drilling, is contingent upon the results of Phase 1.

The recommended Phase 1 work program for the Project is to update the database, complete additional exploration and infill drilling, and conduct geotechnical, hydrological and engineering studies. This is estimated to cost about $8.9 M.

The recommended Phase 2 work program for the Project is to update the Project Mineral Resource and Mineral Reserve estimates. This is estimated to cost about $2.8 M.

2.0 INTRODUCTION

AMEC Americas Limited (AMEC) was commissioned by Nevsun Resources Ltd. (Nevsun), to provide an independent Qualified Person's Review and Technical Report (the Report) for the Bisha Polymetallic Project (the Project) located in Eritrea, Africa (Figure 2-1).

Nevsun holds a 60% interest in the Project, through a 60% interest in the Bisha Mining Share Company (BMSC). The remaining 40% interest is held by the Eritrean National Mining Corporation (ENAMCO). BMSC is the operator for the mining licence and the mining agreement area.

The Report was prepared in compliance with National Instrument 43–101, *Standards of Disclosure for Mineral Projects* (NI 43–101) and documents the results of ongoing exploration work on the Project. AMEC understands that this Report will be used by Nevsun in support of the company's Annual Information Form (AIF) filing with the Canadian Securities Administrators and in support of the Nevsun press release dated 28 March 2011, entitled "Nevsun Increases Mineable Reserves at Bisha by 40% Using Updated Metal Prices.

Unless specified, all measurements in this Report use the metric system. The Report currency is expressed in US dollars; the Report uses Canadian English. The currency used in Eritrea is the Nacfa. The exchange rate for US$1.00 is equal to 15 Nacfa.

2.1 Qualified Persons

The Qualified Persons (QPs), as defined in NI 43–101 (the Technical Report), responsible for the preparation of the Report include:

- David Thomas, P.Geo., Principal Geologist (AMEC, Vancouver)
- Jay Melnyk, P.Eng., Principal Mining Engineer (AMEC, Vancouver)
- Alexandra Kozak, P.Eng., Manager, Process Engineering (AMEC, Vancouver)
- Vikram Khera, P.Eng., Financial Analyst (AMEC, Oakville).

2.2 Site Visits

AMEC QPs have conducted site visits to the Project as shown in Table 2-1.

Figure 2-1: Location Map of Eritrea

Note: Map north to top of plan. Distance from Asmara to Bisha is approximately 150 km.

Table 2-1: Dates of Site Visits and Areas of Responsibility

QP name	Site Visit Date	Area of Responsibility
David Thomas	7 to 9 May, 2009	Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13, 14, 15, 17.1, 17.3, 19, 20, 21, 22, and 23.
Jay Melnyk	None	Sections 17.2, 17.3, 18.1, 18.4 to 18.10, and those portions of the Summary, Conclusions and Recommendations that pertain to those sections.
Vikram Khera	None	Sections 18.11, 18.12, and 18.13 and those portions of the Summary, Conclusions and Recommendations that pertain to those sections.
Alexandra Kozak	None	Sections 16 and 18.2 to 18.3 and those portions of the Summary, Conclusions and Recommendations that pertain to those sections.

During 2010, Ryan Ulansky, an AMEC mining engineer inspected the pre-strip operations and the facilities under construction, and Gustavo Gonzaga, an AMEC senior geotechnical engineer inspected early highwall excavations and drill core stored at the Project core shack adjacent the Project site.

Both AMEC staff members provided input to the QP authors for the Report on information from the site.

2.3 Effective Dates

The Report has a number of effective dates.

- The Mineral Resources have an effective date of 01 January, 2011
- The Mineral Reserves have an effective date of 01 January, 2011
- Drill data and information on the mining operation is current to 28 February 2011.

The overall Report effective date is taken to be that of the Mineral Resources and Mineral Reserves and is 01 January 2011.

There were no material changes to the scientific and technical information on the Project between the effective date and the signature date of the Report.

2.4 Previous Technical Reports

A number of previous Technical Reports were filed by Nevsun on the Project:

Waller, S., Reddy, D., Melnyk, L., 2006: Nevsun Resources (Eritrea) Ltd, 43-101 Technical Report On The Feasibility Assessment, Bisha Property, Gash-Barka District, Eritrea: unpublished technical report to Nevsun Resources Ltd., effective date 5 October 2006.

Yu., F., Reddy, D., Brisebois, K., and Melnyk, L., 2005: Nevsun Resources (Eritrea) Ltd. Bisha Property, Gash-Barka District, Eritrea, 43-101 Technical Report and Preliminary Assessment, 30 December 2005: unpublished technical report to Nevsun Resources Ltd., effective date 30 December 2005

Reddy, D., and Brisebois, K., 2004: Technical Report on the Bisha Property and Resource Estimate of the Bisha Deposit, Gash-Barka District, Eritrea, 1 October 2004: unpublished technical report to Nevsun Resources Ltd., effective date 18 November, 2004.

2.5 References

AMEC has sourced information from these reports and other reference documents are as cited in the text and summarized in Section 22 of this Report. Additional information was sourced from, and provided by, Nevsun. AMEC has relied upon other experts in the fields of mineral tenure, surface rights, and permitting outlined in Section 3.

2.6 Technical Report Sections and Required Items under NI 43-101

Table 2-2 relates the sections as shown in the contents page of this Report to the Prescribed Items Contents Page of NI 43-101F1.

Table 2-2: Contents Page Headings in Relation to NI 43-101 Prescribed Items—Contents

NI 43-101 Item Number	NI 43-101 Heading	Report Section Number	Report Section Heading
Item 1	Title Page		Cover page of Report
Item 2	Table of Contents		Table of contents
Item 3	Summary	Section 1	Summary
Item 4	Introduction	Section 2	Introduction
Item 5	Reliance on Other Experts	Section 3	Reliance on Other Experts
Item 6	Property Description and Location	Section 4	Property Description and Location
Item 7	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Section 5	Accessibility, Climate, Local Resources, Infrastructure and Physiography
Item 8	History	Section 6	History
Item 9	Geological Setting	Section 7	Geological Setting
Item 10	Deposit Types	Section 8	Deposit Types
Item 11	Mineralization	Section 9	Mineralization
Item 12	Exploration	Section 10	Exploration
Item 13	Drilling	Section 11	Drilling
Item 14	Sampling Method and Approach	Section 12	Sampling Method and Approach
Item 15	Sample Preparation, Analyses and Security	Section 13	Sample Preparation, Analyses and Security
Item 16	Data Verification	Section 14	Data Verification
Item 17	Adjacent Properties	Section 15	Adjacent Properties
Item 18:	Mineral Processing and Metallurgical Testing	Section 16	Mineral Processing and Metallurgical Testing
Item 19	Mineral Resource and Mineral Reserve Estimates	Section 17	Mineral Resource and Mineral Reserve Estimates
Item 20	Other Relevant Data and Information	Section 19	Other Relevant Data and Information
Item 21	Interpretation and Conclusions	Section 20	Interpretation and Conclusions
Item 22	Recommendations	Section 21	Recommendations
Item 23	References	Section 22	References
Item 24	Date and Signature Page	Section 23	Date and Signature Page
Item 25	Additional Requirements for Technical Reports on Development Properties and Production Properties	Section 18	Additional Requirements for Technical Reports on Development Properties and Production Properties
Item 26	Illustrations		Incorporated in Report under appropriate section number, on page immediately following first citation in text

3.0 RELIANCE ON OTHER EXPERTS

The QPs, authors of this Report state that they are qualified persons for those areas as identified in the appropriate QP "Certificate of Qualified Person" attached to this Report. The authors have relied upon and disclaim responsibility for information derived from the following reports pertaining to mineral rights, surface rights, and permitting issues.

3.1 Mineral Tenure

AMEC QPs have not reviewed the mineral tenure, nor independently verified the legal status or ownership of the Project area or underlying property agreements. AMEC has relied upon Nevsun experts for this information through the following document:

- Davis, C., 2011: Bisha Project: letter from Nevsun Resources Ltd to AMEC Americas Limited, dated 23 March 2011.

3.2 Surface Rights, Access, and Permitting

AMEC QPs have relied on information regarding the status of the current Surface Rights, Road Access and Permits through opinions and data supplied by Nevsun experts through the following document:

- Davis, C., 2011: Bisha Project: letter from Nevsun Resources Ltd to AMEC Americas Limited, dated 23 March 2011.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located 150 km west of Asmara (233 km by road), 43 km southwest of the regional town of Akurdat (Agordat on some maps), and 50 km north of Barentu, the regional, or Zone Administration Centre, of the Gash-Barka District (Figure 4-1), in Eritrea, East Africa.

The Project is at approximate latitude 15°28'N and longitude 37°27'E. The UTM coordinates of the centre of the Project are 1,711,000 N and 334,500 E (UTM Zone 37).

4.1 Property and Title in Eritrea

The only mine operating currently in Eritrea is Bisha.

During Italian colonial times, metal mining occurred at sites such as Okreb and Augaro but ceased once the British took control in 1941. Mining was conducted for gold at Augaro on a very limited basis in the 1950's.

The Ethio-Nippon Company was mining the Debarwa (Cu, Pb, Zn) massive sulphide deposit in the early 1970's and the ore was sent directly to Japan for refining. Operating mines, such as Debarwa, were halted in 1974 as hostilities between the governing Ethiopian regime and Eritrean independence groups increased.

In 1995, the Eritrean government presented the Proclamation to Promote the Development of Mineral Resources (No. 68/1995) in association with the Regulation of Mining Operations (Legal Notice 19/1995). Additional regulations and proclamations have been presented regarding environmental protection, land use, water use, and heritage.

Information in the following subsections is sourced from the 2006 Bisha Feasibility Study.

4.1.1 Mineral Property Title

The State of Eritrea has provided several key documents relating to mineral property title and regulations.

Property titles are granted in Agreements with the State of Eritrea under the provisions of *Proclamation No.68/1995 a Proclamation to Promote the Development of Mineral Resources.*

Licences are granted and identified according to the level of exploration work completed on a property. Properties are granted under the following licence types: Prospecting Licences, Exploration Licences or Mining Licences. Properties can be obtained under one type of licence and can be converted to the subsequent type if all obligations are met and the titleholder is not in breech of any provisions of the Proclamation and the appropriate application (with fees) are submitted.

A Mining Licence entitles the licensee a 90% interest and the State of Eritrea holds the remaining 10% interest, without cost. The State may acquire up to an additional 30% (total not exceeding 40%) by agreement with the licensee and by funding their share of the development and operating costs.

Under the *Regulation of Mining Operations (Legal Notice 19/1995),* the holder of a Mining Licence shall pay the Eritrean government:

- Royalty for all minerals produced (see below).
- Income tax in accordance with the Proclamation No.69/1995.
- Licence renewal fee.
- Annual rental fees for licence areas (as described above).

Additionally, the holder of a licence and his contractors shall pay a 0.5% customs duty on all imports into Eritrea of equipment, machinery, vehicles and spare parts (excluding sedan style cars and their spare parts) necessary for mining operations.

The royalty to be paid by a licensee pursuant to Article 34 (1) of the proclamation shall be as follows:

- For precious minerals the royalty is 5%.
- For metallic and non-metallic minerals including construction minerals the royalty is 3.5%.
- For geothermal deposits and mineral water the royalty is 2%.

Notwithstanding this law, a lesser rate of royalty may be provided by agreement with the licensing authority, when it becomes necessary to encourage mining activities.

Taxation rates are described in the *Proclamation No. 69/1995 Proclamation to Provide for Payment of Tax on Income from Mining Operations.* A holder of a mining licence shall pay income tax on the taxable income at a rate of 38%. Taxable income is to be computed on a historical accrual accounting basis by subtracting from gross income for the accounting year by taking into consideration all allowable revenue, expenditure, depreciation, which, for tax purposes, is deducted straight-line over four years,, re-investment deduction and permitted losses.

If any licensee transfers or assigns, wholly or partially, any interest in the licence, the proceeds shall be taxable income to the extent that such consideration exceeds the amount of his un-recovered expenditure.

Withholding taxes and personal income taxes of non-residents of Eritrea are identified within the proclamation. If the licensee contracts a company or person, who is not resident in Eritrea for services in Eritrea, the licensee will pay taxes on behalf of such a person. Taxes will be paid at the rate of 10% on the amount paid. For the purposes of this article in the proclamation, a person is temporarily present in Eritrea if he performs work in the country for more than 183 days in any accounting year. The compensation received by an expatriate employee of the licensee or his contractor shall be subject to an income tax at a flat rate of 20%.

The holder of a Mining Licence producing exportable minerals can open and operate a foreign currency account in Eritrea and retain abroad a portion of his earnings to be able to pay for importation of machinery, pay for services, for reimbursement of loans and for compensation of employees and other activities that may contribute to enhancement of the mining operations.

4.1.2 Environmental Regulations

Environment

In the absence of specific environmental legislation to co-ordinate and manage the issues related to the environment, the Eritrean Government's mining legislation includes two provisions for the Environmental Impact Assessment (EIA) of mining projects:

- A Proclamation to Promote the Development of Mineral Resources: No. 68/1995, Article 43
- Regulations on Mining Operations: Legal Notice No. 19/1995, Article 5.

Both of these legal instruments require that an environmental assessment is completed, submitted and subject to review before a mining licence can be granted. Guidance on preparation of an impact assessment is provided by The National Environmental Assessment Procedures and Guidelines (NEAPG) for undertaking EIA for all development projects, introduced by the Eritrean Ministry of Land, Water, and Environment. NEAPG provides mechanisms for ensuring an integrated approach to sustainable development.

Land Use

Land use regulations are described in the *Land Proclamation, No. 58/1994* which provides that all land is owned by the State and citizens have use rights only. Under this Proclamation, peasant farmers have the right to use land for a lifetime, and if significant investment has been made on the land then priority is given for closer relatives to inherit the property and to continue farming the land. This proclamation has not yet been implemented, at least with respect to land distribution to peasant farmers. *Legal Notice No. 31/1997* was introduced to speed up the land law implementation process, which provided the legal basis for methods of land allocation and land administration. This legal notice mandates the Ministry of Land, Water and Environment, in collaboration with other ministries, to prepare land use and area development plans. The plans are still pending, due to institutional and technical limitations.

Water Resources

The Ministry of Land, Water and Environment (Water Resources Department) has drafted a *Water Law* and efforts are being made to finalize and have it pass into legislation. The draft law deals with the institutional and regulatory issues, water use, water rights, environmental issues and water quality. Currently water use is subject to the overlapping of water development interests of the Ministries of Agriculture, Public Works, and local Government.

National Heritage

There is no integrated law that deals with National Heritage. *The Cultural Assets Rehabilitation Project (CARP)* has made studies on various aspects of National Heritage in Eritrea and has drafted a National Heritage law and efforts are being made to finalize and have it pass into legislation. The draft law deals with institutional and regulatory issues, heritage sites, preservation, and rehabilitation.

The National Museum, which forms an integral part of the University of Asmara, has the responsibility to educate the public, conduct research into critical issues that pertain to Eritrea's past, its natural history, its social configurations, and its social and military history. The museum must also manage its diverse collections, and is responsible for management of heritage sites (natural and cultural), and on-site museums, the dispensation of advice to owners of heritage objects, and the enforcement of laws and regulations pertaining to heritage resources of all kinds.

4.2 Project Ownership

The Bisha Project is held within an Eritrean company, Bisha Mining Share Company (BMSC). The shareholder structure of BMSC is 60% Nevsun and 40% ENAMCO; with the ENAMCO shareholding comprising a 30% paid participating interest and a 10% free participating interest as provided by the country's mining legislation.

ENAMCO agreed in October 2007 to purchase the 30% paid participating interest. The purchase price will be determined by an independent valuer within 90 days of first gold shipment from the mine and shall be based on the net present value of 30% of the project, using the 2006 feasibility study mineral reserves. The determination of net present value will be based on the 2006 feasibility study financial model updated for metals prices, actual pre-production capital expenditure and a discount rate appropriate for the mine as at first gold shipment. Once the purchase price has been determined, the amount owing by ENAMCO will be the purchase price less amounts already paid to Nevsun, including a $25,000,000 provisional payment made in the first quarter of 2008 and a $20,000,000 advance to Nevsun in the third quarter of 2009. Settlement of the remaining purchase price will be paid to Nevsun out of the cash flows generated by the Bisha Project.

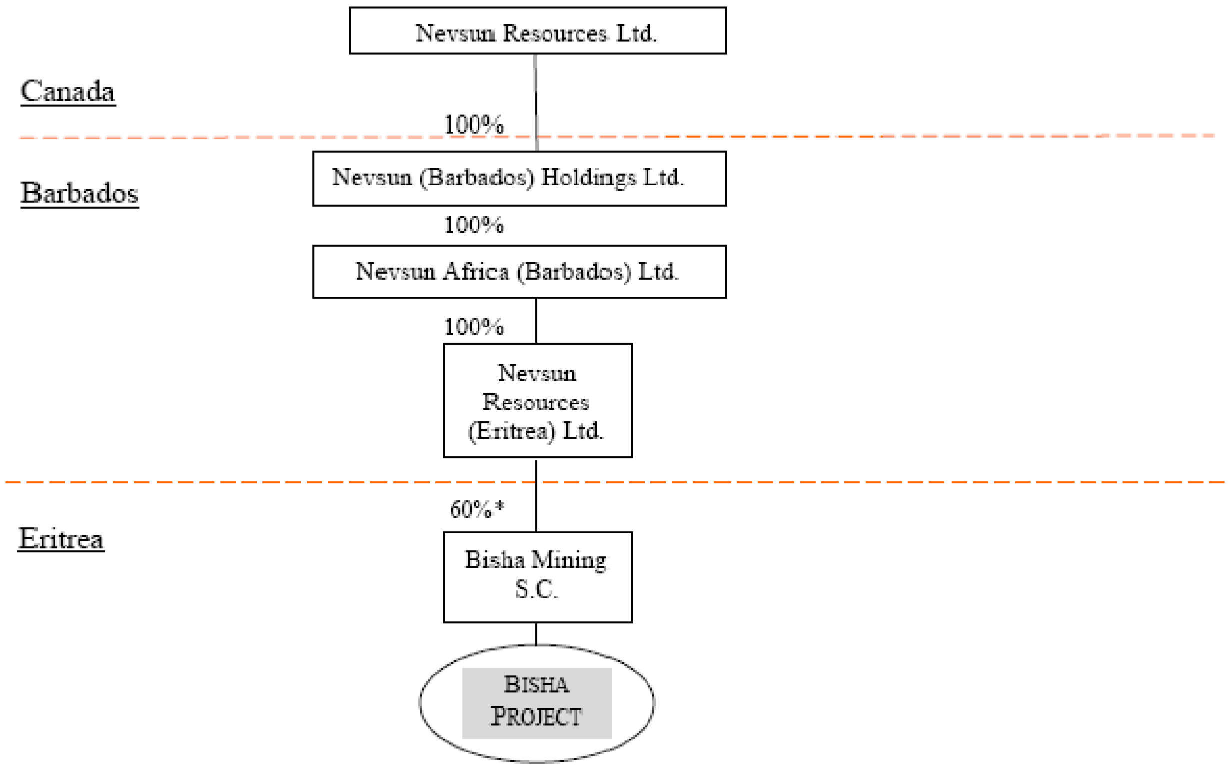
The Nevsun interest in the Project and Nevsun inter-corporate holdings are summarized in Figure 4-1.

4.3 Mining Agreement with Government

In December 2007 BMSC concluded a confidential Mining Agreement with the Government of the State of Eritrea containing all the normal provisions governing the future development and operations for the Bisha Project.

AMEC has reviewed the confidential document and is satisfied there are no terms that are out of the ordinary for agreements of this type, or that negatively impact AMEC's views of the Project.

Table 4-1: Nevsun Project Ownership Diagram



*10% free carry and 30% contributing interest by state-owned Eritrean National Mining Corporation (refer to notes to audited financial statements for financial arrangements)

4.4 Mineral Tenure

The Project comprises a mining licence covering an area of 16.5 km^2, a mining agreement area covering an area of 39 km^2, and an exploration licence covering an area of 71 km^2. BMSC is the operator for all of the licenses. The combined license area that form the Project cover a total surface area of 110 km^2 (Figure 4-1). UTM coordinates for the mining leases are listed in Table 4-1; Table 4-2 shows the boundary UTM co-ordinates for the exploration licence.

Figure 4-1: Location of the Bisha Mining and Exploration Licences and the Mining Agreement Area

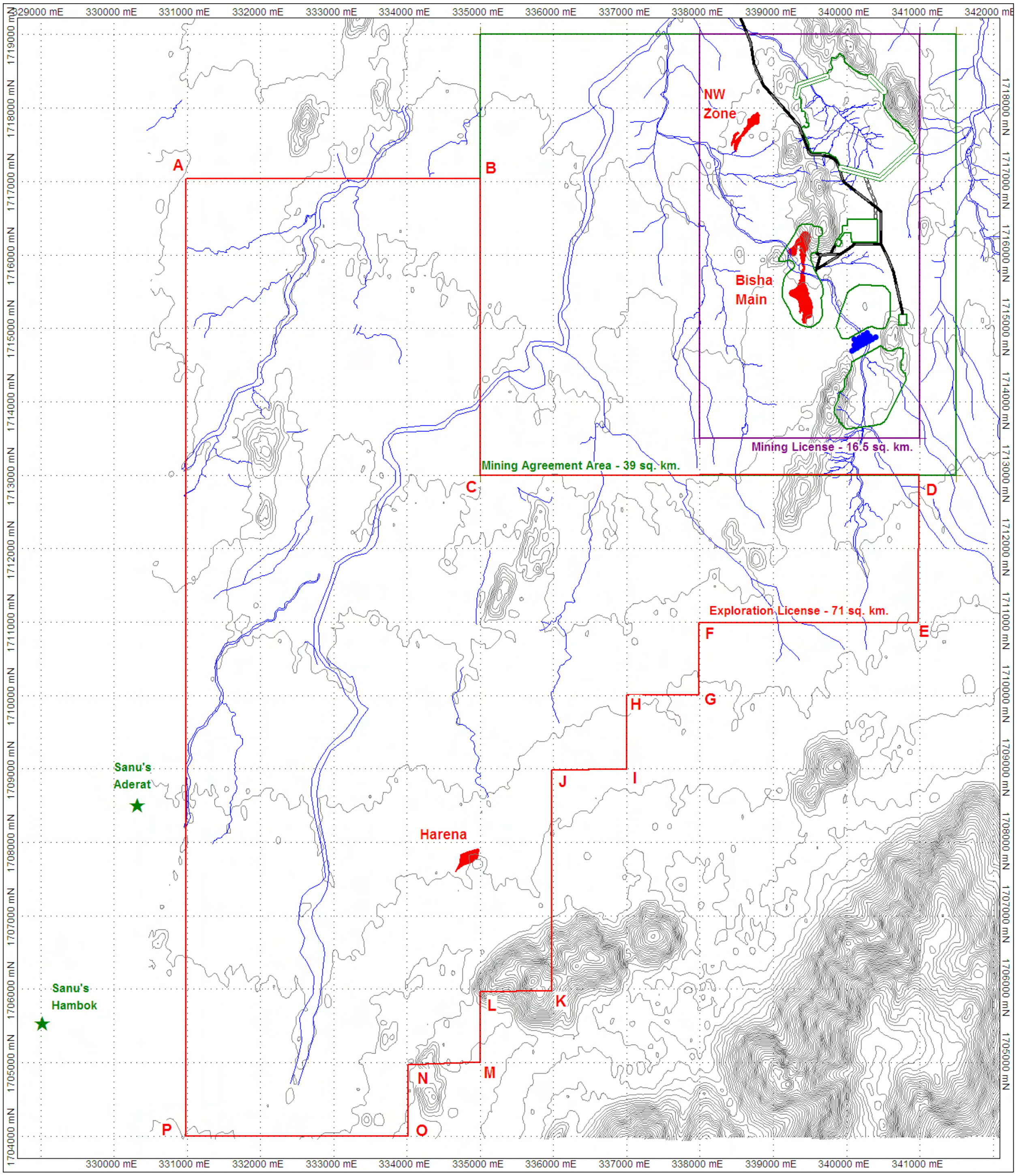


Table 4-2: UTM Coordinates of the Bisha Mining Licence Areas (UTM Zone 37)

Mining License		
Corner Point	Easting	Northing
A	338,000	1,719,000
B	341,000	1,719,000
C	341,000	1,713,500
D	338,000	1,713,500
	Area (sq km)	16.5
Mining Agreement Area		
Corner Point	Easting	Northing
A1	335,000	1,719,000
B1	341,500	1,719,000
C1	341,500	1,713,000
D1	335,000	1,713,000
	Area (sq km)	39

Table 4-3: UTM Coordinates of the Bisha Exploration Licence Area (UTM Zone 37)

Turning Point	Easting	Northing	Turning Point	Easting	Northing
A	331000	1717000	I	337000	1709000
B	335000	1717000	J	336000	1709000
C	335000	1713000	K	336000	1706000
D	341000	1713000	L	335000	1706000
E	341000	1711000	M	335000	1705000
F	338000	1711000	N	334000	1705000
G	338000	1710000	O	334000	1704000
H	337000	1710000	P	331000	1704000
				Area (km^2)	**71**

BMSC have the exclusive right to apply for and be granted multiple Mining Licenses within the Mining Agreement Area. The Mining Agreement entitles BMSC to apply for a mining licence valid for a period of up to 20 years, with renewal periods of up to 10 years providing that:

- Sufficient ore has been defined to demonstrate continued economic viability of Mining Operations
- BMSC has fulfilled the obligations specified in the Mining License and the Mining Agreement

- BMSC is not in breach of any provision of the Mining Proclamation and which would constitute grounds for suspension or revocation of the Mining License.

The Bisha Mining License was granted by the Eritrean Ministry of Energy and Mines during 2008.

The Exploration Licence may be converted to a Mining Licence upon the acceptance by the State of Eritrea of an appropriate Feasibility Study and environmental impact assessment (EIA) report.

The annual rental fee for the Exploration Licence is 53,200 Nakfa, and the annual licence renewal fee is 6,000 Nakfa (about US$3,500 and US$400 respectively). The Exploration Licence is valid until 10 May 2011, with a right of renewal upon application and payment of a minor fee. Nevsun has confirmed that the application renewal was lodged in February, 2011, and the appropriate fee paid.

BMSC has surveyed the boundaries of the Mining License Area in accordance with the Mining Proclamation law. BMSC is not required to survey the Mining Agreement Area or to place Mining Agreement Area boundary markers. Exploration licences also do not require survey.

4.5 Surface Rights

Under the terms of the Mining Agreement, BMSC has the exclusive right of land use in the Mining License Area that is granted within the Mining Agreement Area. This right is subject to the acquisition and settlement of any third-party land-use rights by payment of compensation and/or relocation at the expense of BMSC, in accordance with Eritrean Government Proclamation No. 68/1995, “Proclamation to Promote the Development of Mineral Resources and the Mining Agreement”.

4.6 Royalties

Royalties payable include an Eritrean Government royalty of 5% of precious metal net smelter return (NSR) and 3.5% of base metal NSR.

4.7 Permits

Nevsun has maintained all of the necessary work permits for conducting exploration programs.

For the mining operations, grant of the mining lease provides permission to construct and operate the Bisha mine. A permit has been granted for use of water from the Mogoraib River. Nevsun has also applied for a permit for a water diversion dyke.

These permits are sufficient to ensure that mining activities at Bisha are conducted in accordance with the appropriate National laws.

Nevsun commenced the process for the mine development portion of the Project in 2004, undertaking environmental and socio-economic baseline studies and an environmental assessment.

The Terms of Reference (ToR) for the project environmental and socio-economic and environmental impact assessment (SEIA) were approved by the Eritrean Ministry of Energy and Mines (the Ministry) in March 2006. An SEIA report was completed in December 2006 and submitted to the Ministry.

During 2007, a review of the report was conducted by the Ministry of Land, Water and Environment, by an appointed "Impact Review Committee". Comments and queries raised by the latter were addressed by BMSC in 2007–2008. A Mining Licence for the Project was issued on May 26, 2008. Issuance of the licence is accepted by Nevsun as SEIA approval by the Impact Review Committee, as required under the Eritrean Mining Regulations.

As the transportation route has already been constructed as part of the national transportation system, an assessment of the environmental effects that were associated only with the transport of hazardous materials (i.e., cyanide and fuels) and increased traffic was addressed in the SEIA.

The port facilities for concentrate export are not finalized. Concentrate is not planned to be shipped until the third year of production, therefore the environmental impact assessment (EIA) of these facilities would constitute a separate application.

4.8 Environmental

The Eritrean Government's mining legislation outlines two key provisions for EIAs on projects. A *"Proclamation to Promote the Development of Mineral Resources", No. 68/1995, Article 43* and the *Regulations on Mining Operations, Legal Notice No. 19/1995, Article 5,* both state that an EIA must be completed and submitted before a mining licence is granted. The *"National Environmental Assessment Procedures and Guidelines, March 1999"* (NEAPG) outlines the procedure for undertaking environmental assessments and clearance of projects. Approvals are the responsibility of the Department of Environment (DoE) of the Ministry of Land, Water and Environment.

The SEIA was conducted so as to comply with Eritrean requirements and with the International Finance Corporation Performance Standards on Social and Environmental Sustainability (IFC Performance Standards, April 2006) where the latter are more stringent or comprehensive than national requirements. As noted in Section 4.6, the SEIA report was submitted in December 2006 and following review by the appropriate Eritrean Government agencies, a Mining Licence was issued in May 2008, signifying that environmental approval had been granted. Subsequently, Nevsun indicated that various environmental studies were proceeding in order to provide more information for the operational management of the Project, and an SEIA update was issued in early 2009. The update includes more detail on the implementation of the social and environmental management plans that will manage the impact of the project and ensure employment of the proposed mitigation/enhancement measures.

The key environmental issues assessed by the SEIA study and addressed in Project risk assessment and the proposed environmental management plan are as follows:

- Direct footprint disturbance of 442 ha with associated potential for loss of land use, habitat, soils loss and drainage disturbance
- Groundwater impacts from both extraction of Project supply water from new wells and excavation of an open pit
- Water quality impacts arising from potential for acid rock drainage (ARD), including the need to ensure that there is no post-closure problem
- Soil and water quality impacts arising from the storage and use on site of hazardous chemicals, including cyanide
- Changes to local surface drainage patterns due to construction of a site surface water management system, including flood control and diversion works
- Air quality impacts, most significantly from surface haulage on unsealed roads.

Nevsun has provided a remediation bond with the State of Eritrea in the amount of $2,000,000. The Company has also accrued as an asset retirement obligation of approximately $11,650,000 as at December 31, 2010, for the estimated present value of remediation costs.

4.9 Socio-Economics

Since exploration and environmental baseline data collection began, considerable effort was spent developing support for the Project by fostering local relationships, developing a strong local workforce, educating stakeholders about the Project and mining in general and providing stakeholders with regular Project updates and, where appropriate, site visits.

The key socio-economic issues assessed by the SEIA study and addressed in the proposed social management and related plans are as follows:

- Direct footprint disturbance of 442 ha with associated potential for displacement of people and their customary use of the land (although it is noted that the affected area is sparsely populated and only lightly used)
- Influx of people seeking employment with associated potential issues, including pressure on existing social infrastructure
- Inward investment and creation of direct and indirect employment opportunity.

4.10 Comment on Section 4

In the opinion of the QPs the following conclusions are appropriate:

- The mining tenure held is valid for the mining lease, and sufficient to support declaration of mineral resources and mineral reserves. The exploration licence will expire in May 2011; a renewal application has been lodged, and appropriate payments for renewal were made. AMEC considers that it is a reasonable expectation that the licence will be renewed
- The mining lease has been surveyed on the ground. The Mining Agreement Area and the exploration licence boundaries have not been surveyed
- Annual lease-holding fees have been paid to the relevant regulatory authority
- Nevsun has taken the appropriate steps, where required, to lodge either extensions or renewals for the licence, as such fell due
- Nevsun holds sufficient surface rights to support mining operations over the projected life-of-mine, and to support the declaration of mineral resources and mineral reserves
- Permits obtained by the company to explore and undertake project development are sufficient to ensure that activities are conducted within the regulatory framework required by the Eritrean government

- At the effective date of this report, environmental liabilities comprise those that would be expected to be associated with a mine under construction, and include an open pit, waste rock facilities, infrastructure construction, and roads
- Based on the permits and the current state of environmental knowledge for the Project, Mineral Resources and Mineral Reserves can be declared, and the proposed mine plans are appropriate and achievable.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Asmara is the capital city of Eritrea and is serviced by regular international flights.

Access to the Bisha Exploration Licence is by paved road from Asmara to Akurdat, a distance by road of 181 km. From Akurdat, access is via an all-weather unpaved road. The Bisha permanent camp is located 5 km south of the village of Mogoraib beside the Mogoraib River, and approximately 1.5 km north of the Bisha Exploration Licence boundary (refer to Figure 5-1). The main work site at Bisha is located 4 km to the south of the camp along a track across a flat alluvial plain. The drive from Asmara to the Bisha camp takes approximately 4 hours. The main distances by road to the Bisha licence are summarized in Table 5-1.

The principal port for importation of heavy equipment is Massawa on the Red Sea coast, which is about 350 km by road via Asmara to the east.

5.2 Climate

The climate in the area is semi-arid with elevated temperatures year-round. During the hot season in April and May the average temperature is +42°C, although temperatures may rise to +50°C for short periods. The main rainy season is between June and September, and periodic flooding of the Mogoraib and Barka Rivers can result in spectacular flash floods. Occasional rain may also fall during April and May. Total rainfall is sparse with between 300 mm and 500 mm falling in the year.

The rainy season causes periodic, short-lived difficulty in travel off of the main highways, although exploration work is possible year round. During the period of exploration work by Nevsun, the precipitation has only occasionally been sufficient to flood the local rivers. All mining activities are planned on a year-round basis.

Figure 5-1: Project Access and Topography Map

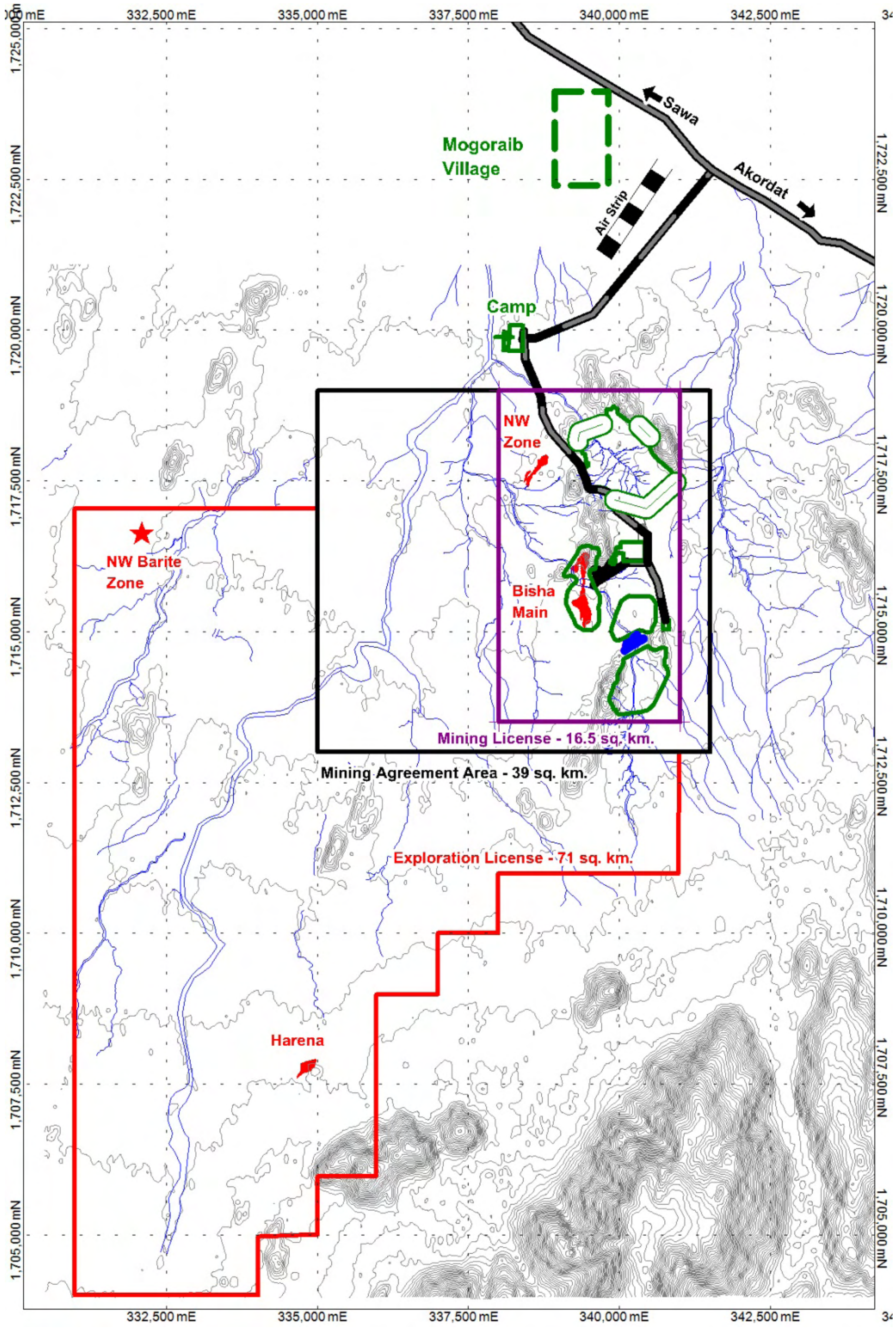


Table 5-1: Distances by Road to the Bisha Mining Licence

From	To	Distance (km)	Condition
Asmara	Akurdat	181	Paved, all weather road
Akurdat	Adi Ibrahim	28	Unpaved, all weather road
Adi Ibrahim	Hashakito	19	Unpaved, all weather road
Hashakito	Bisha Camp	5	Unpaved, all weather road
Asmara	Bisha Camp	233	4 hour drive
Bisha Camp	Main Gossan	4	Unpaved, all weather road

5.3 Local Resources and Infrastructure

The following subsections detail the local resources and the existing and projected infrastructure associated with the Project.

5.3.1 Local Resources

There are few local resources in the Bisha area.

A preliminary land use survey in the vicinity of the proposed mine site was conducted by Klohn Crippen in 2004. It was determined that approximately 96% of the area was used by local herders as pasture for livestock and used seasonally for activities including agriculture, domestic livestock migration and accessing wells and burial sites. Currently the land is overgrazed, which is related to ongoing drought conditions and pressures from livestock foraging. This study was conducted in consultation with people from local communities.

An additional survey was conducted in February 2006 for the six communities within the Bisha area; these included Tekeret, Adi-Ibrihim, Hashakito, Jimel, Adorat-Harenay and Takawda. Most of the people in the region are located in permanent settlements but utilize the Bisha area as one of the many areas for grazing livestock, planting crops and accessing watering areas, which in some cases involves migrating distances up to 200 km, as herders move through the region in search of suitable grazing lands.

The village of Mogoraib is the local administration centre for the Dige Sub-zone within the Gash-Barka District. The village has a small refugee resettlement site and subsidiary military and commercial interests. The village contains a well-equipped, eight person health centre capable of taking care of small medical problems by nursing staff in preparation for referral of patients to larger, better equipped hospitals in Akurdat and Keren. Camp Mogoraib is a military training site located just outside the village boundaries. With the presence of the mine development and exploration project at Bisha, this camp has been re-activated as a security post from its previous care/maintenance basis.

Few basic goods are commercially available in the region, either in Mogoraib or Akurdat. The main centre for support of exploration and project development is from the capital city, Asmara.

The local population has no exploration or mining culture, and training of local staff would be required.

5.3.2 Infrastructure

Current onsite Project infrastructure includes:

- An open pit,
- Process plant,
- Tailings and waste rock storage facilities,
- Offices,
- Maintenance and laboratory facilities,
- Fuel storage areas,
- On-site power plant
- Airstrip.

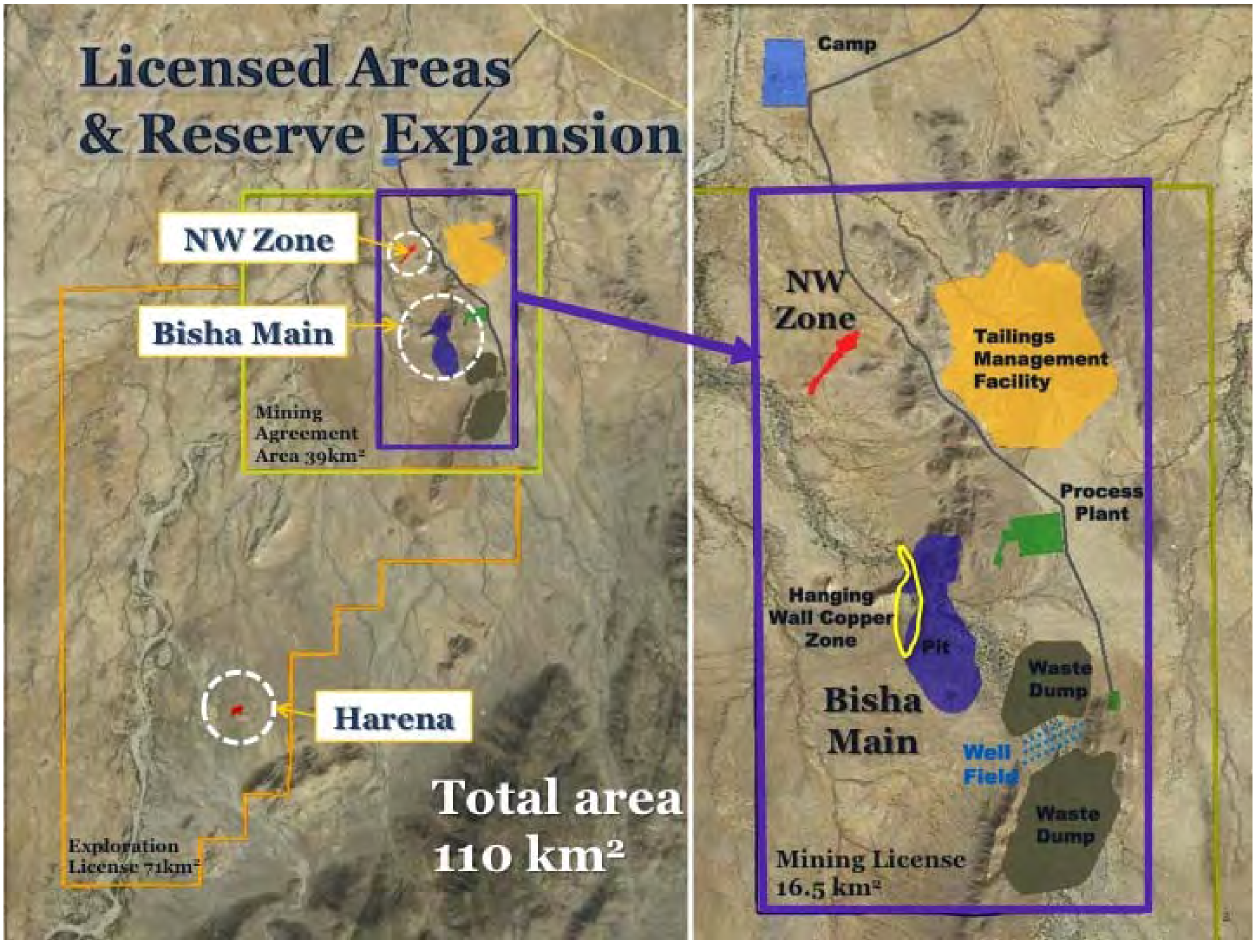
The key areas are indicated in Figure 5-2.

In addition, there are offsite infrastructure that includes port and ship-loading facilities at the port of Massawa. The 2006 Feasibility Study was based use of a port facility at the site of an existing cement production facility, adjacent to an existing jetty on the north shore of Khor Dakliyat Bay. The cement plant is owned by the Eritrean government and it was noted in the 2006 Feasibility Study that, according to correspondence from the Eritrean government, the site will be made available for use by Nevsun. The agreement for use of the facility has yet to be finalised.

5.3.3 Power

Electric power for the mine and processing plant site is supplied from a new diesel-fuelled power station located adjacent the process facilities. The port concentrate load-out facility near Massawa receives power from the local utility.

Figure 5-2: Project Layout



5.3.4 Water

Process water is sourced from recycling within the planned plant and additional needs will be supplemented from freshwater sources. The process was designed to maximize the recycle of process water and included the installation of a tailings slurry thickener to recover process water prior to pumping to the tailings containment system. This approach served to minimize the evaporation losses that result with the typically large water surface area in tailings containment systems. Even though evaporation rates in this region are very high, a tailings management facility supernatant water reclaim pumping system is installed to reclaim seasonal decant water from this source.

Freshwater is supplied to project from groundwater. Two well farms have been established by BMSC, the first approximately 1 km south of the open pit on the western bank of the non-perennial Fereketatet River, and the second 5 km to the west adjacent to the Mogoraib River (Figure 5-3). Potable water sourced from the well fields is pumped to a potable water plant utilizing chlorination filtration and ultraviolet radiation treatment. In addition, water from the pit is pumped to water storage facilities.

5.3.5 Communications

Current site communication is via radio, cellular service, and a satellite communications system.

5.4 Physiography, Flora and Fauna

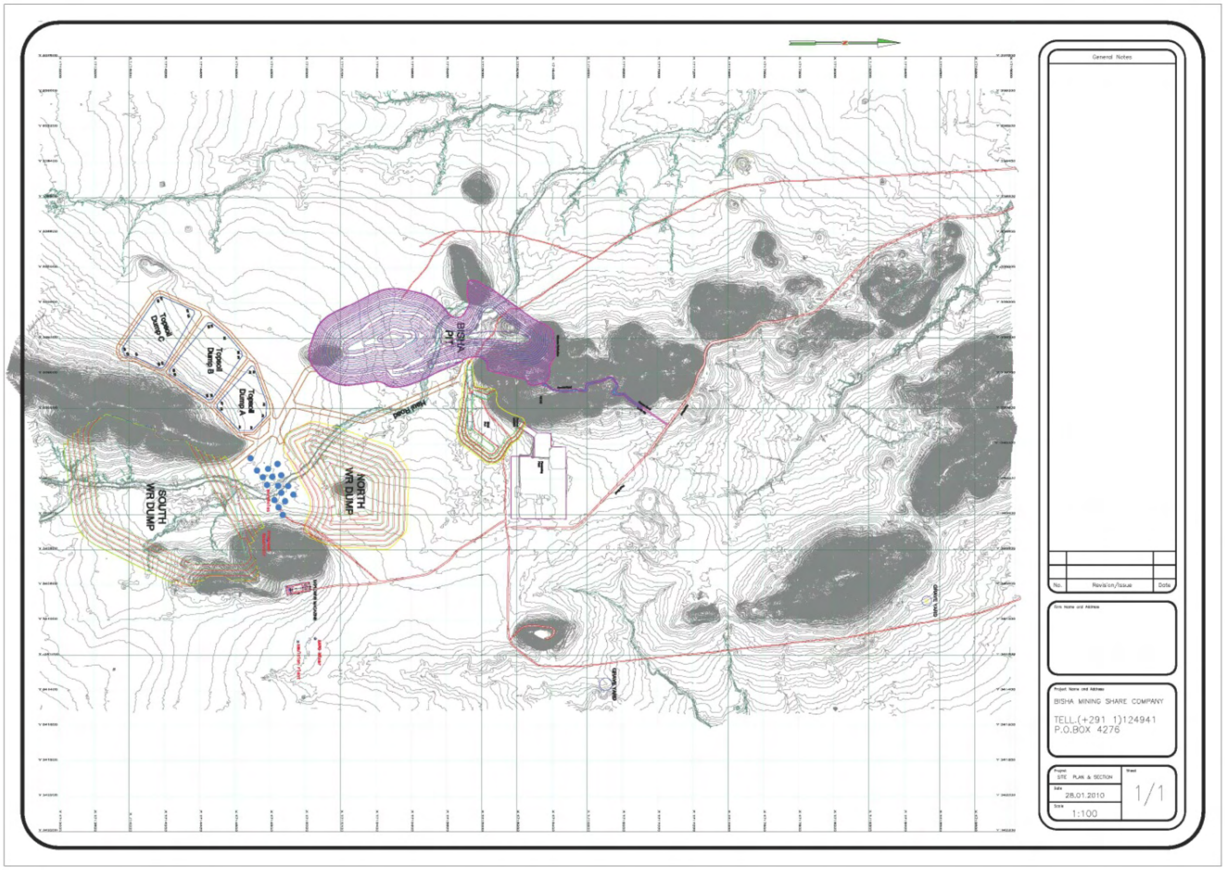
The Project is located on a flat to rolling, desert-like plain along the western foot of the Central Highlands. The plain is at 560 masl (refer to Figure 5-1), and contains scattered vegetation and few trees. Steep hills and ridges rise above the plain; the Bisha, Wade, and Neve peaks reach elevations of up to 1,226 masl above the alluvial plain at the southern boundary of the Project.

Abundant seasonal streams cross the area and flow northward from the Project into the Barka River (6 km north of the Project boundary), and continue north and northeast into Sudan. The Project is crosscut by the Mogoraib River, a tributary to the Barka River that flows northwards along the western side of the Project (refer to Figure 5-1). A smaller seasonal tributary, the Fereketatet River, flows north-northwest into the Mogoraib River. The Fereketatet River crosses the Project and passes immediately west of the Bisha Gossan Zone.

5.5 Comment on Section 5

In the opinion of the QPs the information discussed in this section supports the declaration of Mineral Resources and Mineral Reserves through documentation of the availability of staff, the existing and planned power, water, and communications facilities, the methods whereby goods are transported to and from the mine, and consideration of planned additions, modifications, or supporting studies. Mining activities will be carried out on a year-round basis.

Figure 5-3: Site Layout Plan



Note: Figure courtesy Nevsun.

6.0 HISTORY

Nevsun has no record of any previous exploration or mining activities on the Project or surrounding areas prior to 1996. A single colonial mine, dating from the Italian era, is situated at Okreb, 7 km south of the village of Adi Ibrahim.

In late 1996, a private Canadian company, Ophir Ventures, conducted prospecting in the Bisha area and collected samples from the gossan outcrops. Although this work resulted in the discovery of the surface exposure of the Bisha deposit in the Bisha Gossan Zone, the actual deposit was not recognized until drilling commenced in 2002.

Nevsun commenced work in 1997. During in the period 1998–1999, work conducted by Nevsun included reconnaissance-scale and 1:5,000 scale geological mapping, ground geophysical surveys (MaxMin, magnetometer), a multi-element stream sediment sampling survey, and limited “orientation” soil sampling. This work outlined gossanous areas with elevated base metal values, and in particular, the Bisha Gossan Zone was found to be highly anomalous in lead with significant values of copper, zinc, and silver.

Work was suspended between 1999 until 2002 due to the border war with Ethiopia.

From 2002 to 2005, work comprised diamond drilling, mapping, sampling, trenching, geophysics (airborne and ground), geochemical sampling, petrographic work, bulk density measurements, geotechnical work, environmental baseline work, metallurgical testwork, and mineral resource estimation.

A first time mineral resource estimate was completed by AMEC on behalf of Nevsun in 2004 (Reddy and Brisebois, 2004), and a preliminary assessment (PA), with an updated mineral resource estimate, was finalized by AMEC on behalf of Nevsun during 2005 (Yu et al., 2005). Results of the PA were positive.

During 2005–2006, a feasibility study (the 2006 Feasibility Study) was undertaken by AMEC on behalf of Nevsun, which included an updated mineral resource estimate, and first-time estimation of mineral reserves (Waller, et al., 2006). The study envisaged a 2 Mtpa conventional open pit mine that would produce mill feed at a rate of 5,500 t/d for approximately 10 years. A conventional cyanide leach and flotation process plant, designed to successively treat oxide, supergene, and sulphide (primary) mineralization was envisaged, producing doré and base metal concentrates. Under the set of assumptions used in the 2006 Feasibility Study, and using copper, gold, silver and zinc prices that varied by projected production period, the project base case scenario showed positive economics, and a payback period of 2.6 years.

Since completion of the 2006 Feasibility Study, BMSC has completed prospecting, pitting, trenching, mapping, and gravity geophysical surveys in the Mine Agreement Area, and the exploration licence area.

During 2008, an update to the capital costs (2008 Cost Update Study) used in the 2006 Feasibility Study was undertaken (Senet, 2008). The pre-production capital expenditure estimate was raised to $250 M, and the future expansion capital was re-estimated to $115 M versus the $92 M in the 2006 Feasibility Study; the revised estimates are within the sensitivity analysis included in the 2006 Feasibility Study.

The pre-strip for the Project commenced in March 2010. To 31 December 2010, approximately 4,500,000 tonnes of waste had been mined and the ore stockpiles totalled approximately 150,000 tonnes. The commissioning of the plant was well advanced with the first gold pour of 26 kg taking place on 30 December 2010. Nevsun declared commercial production on 22 February 2011. Commercial production was defined by Nevsun to be at least 90% of planned throughput and at least 90% of planned recovery for a period of at least 30 days. During the period up to 28 February 2011, the Project had gold proceeds of approximately $57 million. All critical elements of the facilities had been constructed by that date, with some less significant components to be completed in the first half of 2011.

7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The regional geology of Eritrea and the adjacent countries of the Horn of Africa are not well documented and geological mapping within Eritrea has been limited due to the armed conflicts since the 1960s. Eritrea is underlain by the western or Nubian portion of the Arabian-Nubian Shield, which is composed of accreted Archaean and Proterozoic rocks, which were reactivated during the Pan-African Orogeny in the Late Proterozoic–Early Palaeozoic Era (1,000 to 500 Ma; Berhe, 1990 in Chisholm et. al., 2003). Granitoids intruded and metamorphosed older rock sequences.

The age of the volcano-sedimentary rocks in the Arabian–Nubian Shield is not well known. In Eritrea, the units are considered to be approximately 850 Ma for the Tsaliet Group volcano-sedimentary rocks and >650 Ma for the overlying Tambien Group sedimentary rocks.

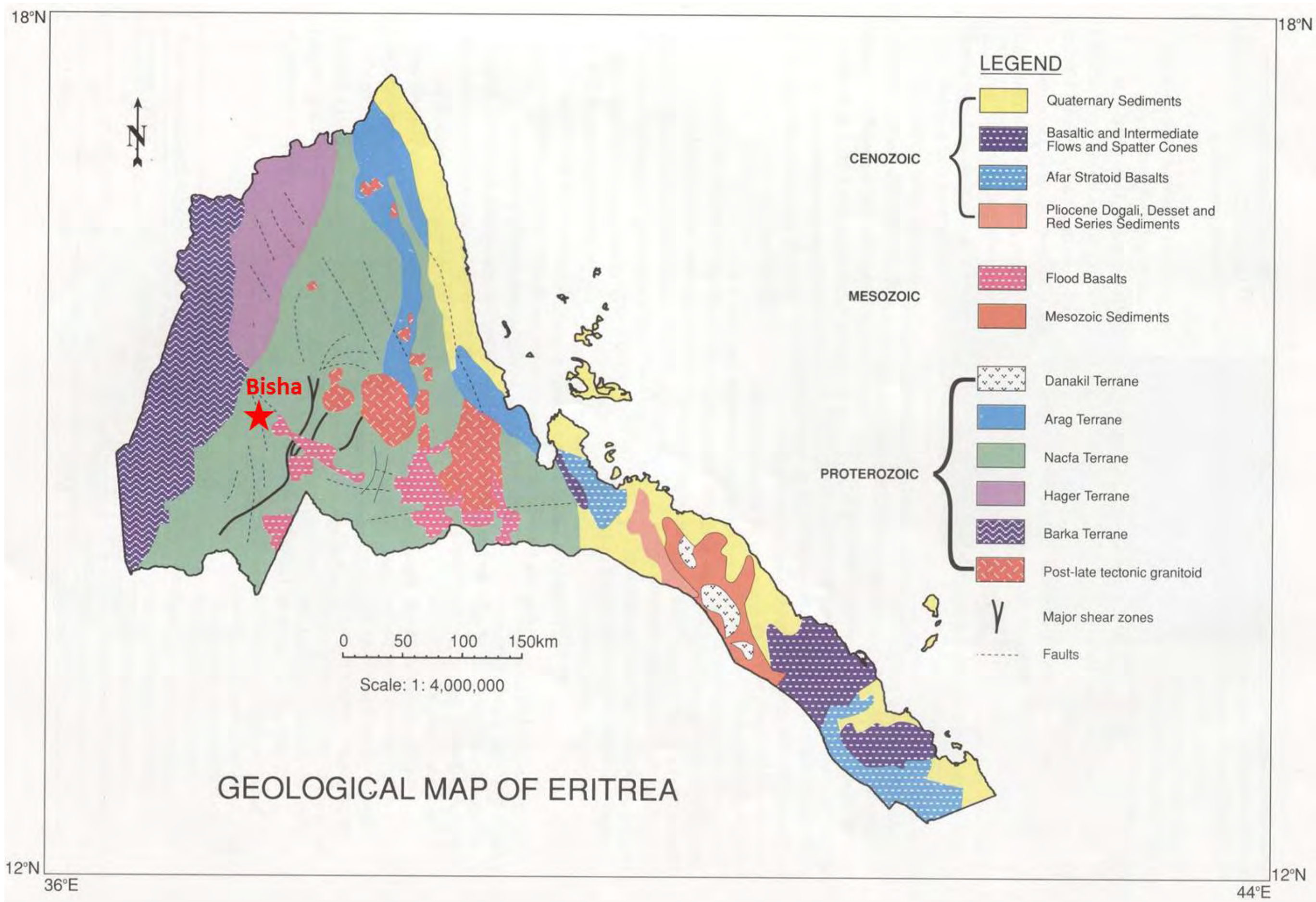
Eritrea is divided into several north or northeast trending Proterozoic terranes, which are separated by major crustal sutures. The terranes are, from west to east: Barka Terrane, Hagar Terrane, Nacfa Terrane, Arag Terrane, and Danakil Terrane (see Figure 7-1). The Nacfa Terrane comprises low-grade metamorphosed calc-alkaline volcanics and sediments, and hosts base metal mineralization in the region surrounding the city of Asmara, and in the Gash-Barka district, including the Bisha polymetallic mineralization.

7.2 Project Geology

The Bisha Project is underlain by low-grade metamorphosed (upper greenschist to lower amphibolite facies) volcanics and sedimentary units on the western margin of the Nacfa Terrane. Figure 7-2 shows the Project-scale geology, whereas Figure 7-3 presents the deposit-scale lithologies and structures.

The precious metals-enriched massive sulphide (VMS) deposits at Bisha are hosted by a tightly and complexly folded, intensely foliated, bimodal sequence of generally weakly stratified, predominantly tuffaceous metavolcanic rocks (Greig, 2004). Felsic lithologies appear to directly host the mineralization, predominate overall, and form the hanging wall stratigraphy. The felsic lithologies are mainly exposed to the west and southwest of the mineralized zones, and grade upward into a sequence of generally fine-grained volcaniclastic rocks. A significant component of mafic metavolcanic rocks occurred in the more obviously bimodal footwall, which is exposed mainly to the east of the known mineralized zones.

Figure 7-1: Geological Terrane Map of Eritrea



Note: Figure courtesy Nevsun

Figure 7-2: Project-Scale Geology Map

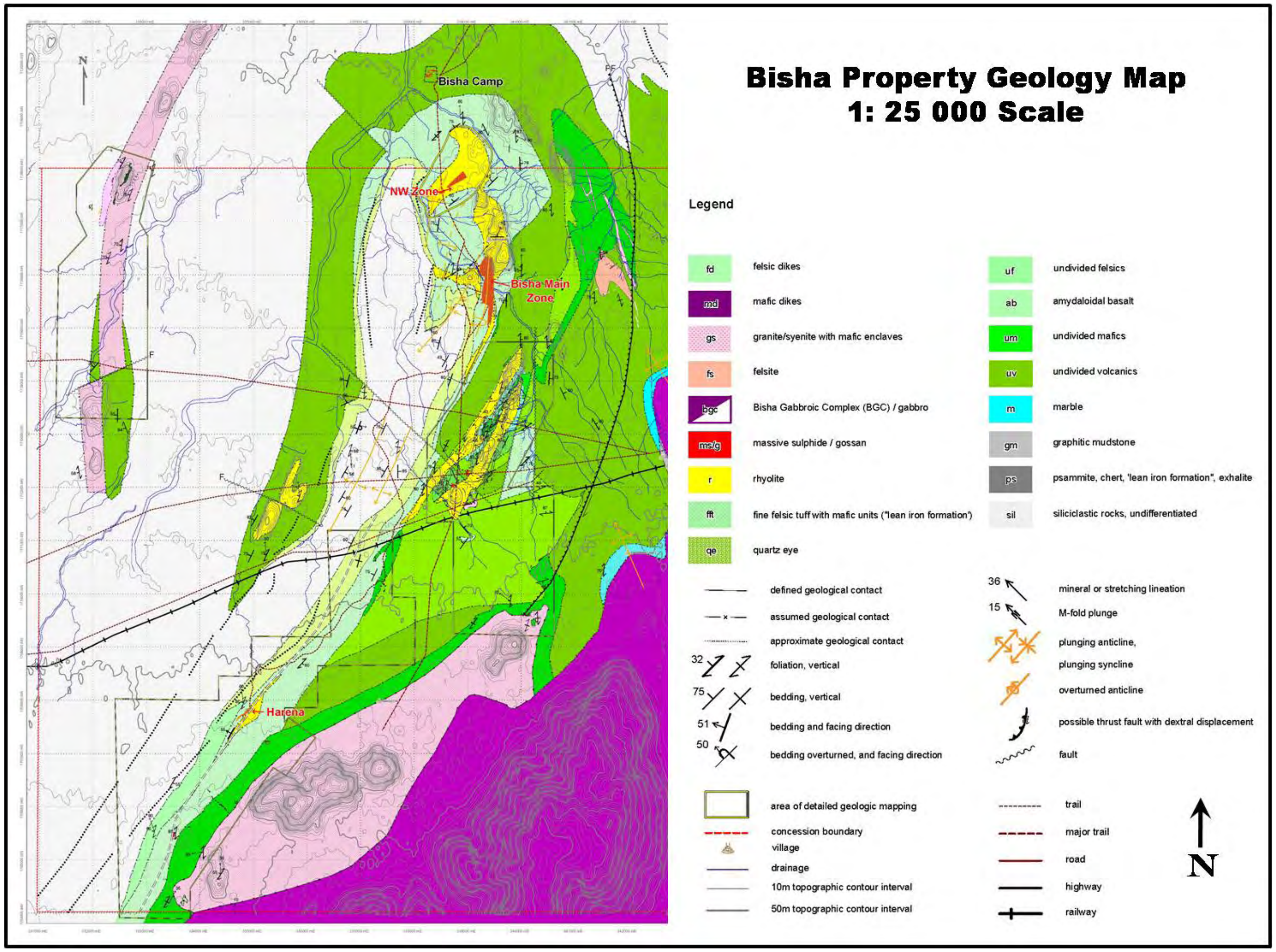
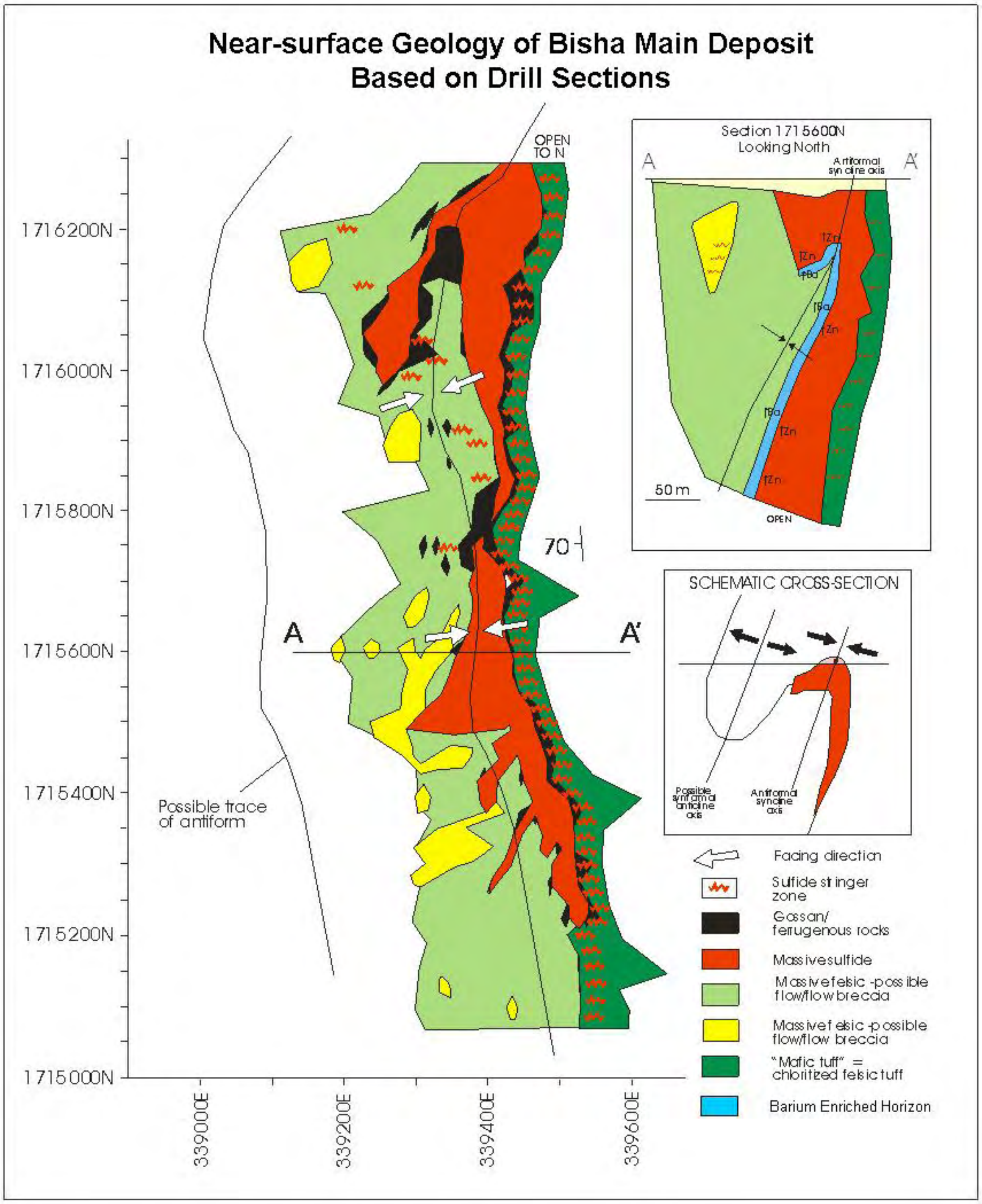


Figure 7-3: Deposit-Scale Geology Map (from Barrie, 2004, 2005)



To the east and south, the metavolcanic rocks are intruded by felsic to mafic intrusive rocks, now foliated, including those of the aerially extensive Bisha Gabbroic Complex. Sedimentary rocks overlie the felsic component, and have been mapped to the west of, and parallel to, the stratigraphic units that host mineralization.

7.3 Stratigraphy

The sedimentary rocks consist primarily of greywacke, siltstone, shale, marble, and feldspathic arenites with less common conglomerate, magnetic ironstone, quartzite and massive sulphide lenses. The volcanic sequence includes fine-grained pyroclastic rocks of mafic to intermediate composition and pillowed mafic flows, felsic ash and lapilli tuffs (Figure 7-4).

In general, stratified rocks at Bisha can be divided into two parts: an upper, predominantly felsic volcanic part that is capped by sedimentary rocks; and a lower volcanic part that is clearly bimodal, at least in the south and east. This lower bimodal volcanic part appears to be capped by the stratiform mineralized horizons at Bisha Main, Bisha South and the mineralization at the Northwest Zone.

The stratigraphic section near the Bisha deposit comprises, from the base (at the Bisha Gabbroic Complex contact) to top: carbonates and fine-grained siliciclastic rocks including siliceous iron formation; felsic lapilli and ash crystal lapilli tuffs with intercalated minor mafic flows and hyaloclastite; and fine-grained volcaniclastic/siliciclastic rocks. Volcanic rocks comprise ~50% of the stratigraphic section ±2.5 km from the deposit horizon.

Rhyolites are the predominant volcanic rock type. The rhyolites are mostly tuffs, with minor blocky flows and agglomerates present immediately west of the Bisha Main and Northwest deposits. Dacites comprise only approximately 5% of the volcanic strata; other volcanic rocks include tholeiitic basalts. The strata are cut by Neoproterozoic granite-syenite intrusions and minor mafic dykes/sills; and by Cenozoic felsic and mafic dykes. One suite of quartz and feldspar phyric rhyolite/granite dykes is texturally and chemically distinctive from the other felsic strata. They occur as rhyolite porphyry or as granitic rocks;

Carbonates, quartzites and siliceous iron formation are present in the lower section to the east of Bisha. The presence of carbonates indicates a relatively shallow depositional environment.

The Bisha Gabbroic Complex is a large (225 km^2) partly-layered gabbro–gabbro-norite intrusion that forms high hills in the central and southern part of the Project. The complex extends in a north–northeast–south–southwest orientation for 25 km, and has a maximum width of approximately 12 km (immediately south of the Project). The complex appears to cut strata, but has undergone penetrative deformation and is presumable coeval, or nearly coeval with the strata. The Bisha Gabbroic Complex is tholeiitic, and compositionally similar to the basalts.

7.4 Structure

In the Bisha area, the rock units generally trend north–northeast with moderate to steep dips to the east and west. The Bisha Gabbroic Complex broadly forms a north-plunging antiform that appears overturned, with dips generally steep to the east. It appears that volcanic and sedimentary strata were thrust against this buttress from the west-southwest, forming a nappe-like structure, with internal antiforms and synforms on a scale of hundreds of metres, which contain the VMS deposits.

The stratigraphy and principal tectonic fabrics at Bisha have been disrupted at least locally by late-stage brittle faults. Because of the relatively poor exposure in the area, these are expressed in the main as well-developed topographic lineaments.

Folds are typically adpressed, with narrow hinge regions and long limbs, and are generally upright to slightly overturned. Axial trends are generally to the north–northeast or north, although in the northeast part of the area mapped, folds appear to trend to the north–northwest.

Fold axes are variably plunging, and plunge reversals appear to be common; although marker units for outlining the resulting hinge-line culminations are scarce, several domes and basins that reflect such culminations are clearly apparent. This is perhaps best displayed at the Northwest Zone, where a basin, and the doubly-plunging Northwest Zone syncline, are outlined by resistant rhyolitic rocks. At its north end, a short distance north of the Project boundary, the Northwest Zone syncline plunges moderately to the south, and at its south end, near the road between the camp and Bisha Main, it plunges gently to the north (Figure 7-2).

The largest-scale fold structure apparent on Figure 7-2 is the north-plunging antiform outlined by the contact of the Bisha Gabbroic Complex. It has a wavelength of 10–15 km and a probable amplitude of up to several kilometres. Folds such as the Bisha and Northwest Zone synclines, are an order of magnitude smaller, with wavelengths of up to a kilometre, and amplitudes of hundreds of metres. The distance along the trend of these folds between adjacent culminations and depressions appears to be somewhat greater than their amplitude, perhaps on the order of 2 km or more; this is consistent with the adpressed nature of the folds.

Folds at Bisha are likely en-echelon in style, with one fold, or fold pair, terminating or relaying into another fold or fold pair—this may well be the case at Bisha Main, where hinges of synthetic folds on the western limb of Bisha anticline apparently end along trend to the north–northeast. It is likely that these folds may pass into, or over, a culmination, which plunges southerly on the southwest and northerly on the northeast.

7.5 Metamorphism

Nacfa Terrane greenstone belt rocks such as the volcanic and sedimentary units at the Bisha Project exhibit upper greenschist to lower amphibolite facies metamorphism. The presence of chlorite, fine grained amphibole, and local garnet in the mafic rocks supports that the grade of metamorphism has been reached (Greig, 2004).

7.6 Alteration

Footwall alteration is typically pervasive chloritic alteration of tuffs, which may extend for tens of metres below massive sulphide units. Immediately below the massive sulphides there is a thin but variable (< 3 m thick) zone of silicification and K-feldspar replacement (Chisholm et. al., 2003). This zone is more variable in intensity and thickness than the chlorite alteration and in some cases is entirely absent.

7.7 Deposits

7.7.1 Bisha Main Zone

The Bisha Main Zone deposit extends for over 1.2 km along a north-trending strike (Figure 7-2), and has been folded (and overturned, dipping to the west) into an antiform so that there are two western and one eastern lenses. The thickness of the lenses is variable from 0 m to 70 m. However, the Bisha Main Zone deposit is deformed, and exhibits limb attenuation and thickening at the fold hinge, which distorts original dimensions. The eastern lens can be traced along the entire strike length, whereas the western lenses are present for approximately half of the strike length.

Weathering along the fold axis extends to a depth of 60 to 70 m. The primary sulphide zone is below the weathering zone. The massive sulphide lenses can locally exceed 70 m in true thickness and show typical copper-rich bases and zinc-rich tops. In places, the stratigraphic tops of lenses are texturally and compositionally layered, with gradations from coarse- to fine-grained material. The host rocks above and below the sulphide lenses are variably altered felsic lapilli, and lapilli ash crystal tuffs, with minor felsic dykes.

The eastern lens is a continuous sheet of mineralization that extends for over 1.2 km (although the lens thins out considerably at line 1715750 N). This lens faces west and dips 65–70° to the west. The deepest drill holes in the southern half of this lens have high zinc grades in two separate layers, suggesting the presence of stacked massive sulphide lenses in this area. The southwestern lens ("the Wedge") is connected to the eastern lens for over 200 m along strike and is therefore an extension of the eastern lens at the same stratigraphic level over an antiformal structure.

The eastern massive sulphide lenses are generally hosted within tuffaceous rocks, but may abut more massive felsic flows with autoclastic aprons to the west. Gossan closely overlies massive sulphide, with most gossan units being no more than 25 m away from the massive sulphides.

The primary characteristics of the western lenses are less clear due to their proximity to the surface and unusual geometry. Primary metal zonation is nearly non-existent due to oxidation, with Zn stripped from the massive sulphide, and Cu, Pb, Au and Ag sporadically enriched by supergene processes. A few deeper massive sulphide intersections in the western lenses, for example in section 1715500N and nearby sections, have higher Zn grades near the base of the lens (the Wedge lens). The zonation of the Zn grades suggests that the western lens faces east, on the western side of an antiform.

Deep weathering has affected Bisha Main Zone lenses that occur in low-lying areas by removing most of the sulphide and producing high-grade supergene blankets enriched in gold, copper, and lead in particular. The gossan zone can vary in composition from highly siliceous and somewhat ferruginous to a massive goethite–hematite–jarosite gossan. The depth of oxidation appears to be on the order of 30 m to 35 m in outcrop areas, but is variable in sand-covered areas. Supergene sulphides are present at 35 m to 65 m depth, with accompanying carbonate, sulphate, phosphate, silicate, halide, and native base metal minerals.

The oxidation of the massive sulphides generated strong acid solutions that have progressively destroyed the sulphides and host rock. A horizon of extremely acid-leached material, or "soap" has developed between the oxide and supergene/primary domains.

7.7.2 Harena

The Harena deposit has been traced over a strike length of 400 m (Figure 7-2), and is interpreted to be a northwest-dipping, tabular massive sulphide body, closed off by drilling to the northeast, but open to the southwest.

The host rocks to the Harena deposit are a bimodal, hydrothermally-altered suite of basalts and rhyolite-dacite volcanics. The stratigraphic succession has appreciable siliciclastic rocks up-section to the west, and has predominantly rhyolite and dacite tuffs proximal to the deposit, with minor intercalated basaltic rocks. The deposit has a distinct footwall that contains kyanite and andalusite, and both minerals are often noticeably chloritized. The kyanite and andalusite are interpreted to have formed following metamorphism of aluminum-bearing seafloor sediments. There is an obvious graphitic component to some of the massive sulphide intersections. A number of late dykes have cut the near surface mineralization at Harena, making determinations of the actual widths of the zone difficult.

Surficial weathering processes have produced three distinct zones of mineralization. These include a surface oxide/gossan overlaying a secondary supergene horizon, which grades into a primary massive sulphide horizon at depth. The gossanous horizon contains frequently anomalous levels of gold and silver. The depth of oxidation appears to be on the order of 45 m to 50 m.

Both the oxide and sulphide mineralized zones are approximately 400 m in length and vary in thickness between 5 m and 15 m. The average grades of the oxides are 1.2g/t gold, 14.2 g/t silver, 0.1% copper and 0.21% zinc, and the average grades of the massive and semi-massive sulphides are 0.84% copper, 0.41 g/t gold, 23.75 g/t silver, and 3.72% zinc.

7.8 Prospects

7.8.1 Hanging Wall Copper Zone

To the west of the massive sulphide lenses of the Bisha Main Zone, there is a zone of copper mineralization, the Hanging Wall Copper Zone, which is located in the structural hanging wall of the Bisha Main Zone. Due to folding deformation, the mineralization is located in the stratigraphic hangingwall to the massive sulphide lenses. The Hanging Wall Copper Zone is, to date, largely restricted to the supergene horizon. The intersections extend from section 1715600N to section 1716275N, a distance of about 675 m. At section 1715800N, the zone is 200–250 m away from the hanging wall contact of the Bisha Main Zone. The Hanging Wall Copper Zone has a north–northeast strike, and converges towards the Bisha Main Zone towards the north. It would appear that the zones are open to the north and to the south. There are 60 drill hole intercepts with down-hole thicknesses varying from 2.7 m to 63.5 m. The geometry of the Hanging Wall Copper Zone is insufficiently known to calculate true thicknesses.

7.8.2 Northwest (NW) Zone

The Northwest (NW) Zone, located approximately 1.5 km north of the Bisha Main Zone (Figure 7-2), is interpreted by Greig (2004) to be another exposure of the same mineralized horizon that hosts the Bisha Main Zone.

The NW Zone has been traced over a strike length of 650 m and is open to the northeast and southwest (Figure 7-4). The mineralized zone is currently interpreted to be a doubly-plunging, northeast–southwest-trending, tight antiform. The axis of the antiform is interpreted to be at a vertical depth of 30 m to 50 m. The thickness of the massive sulphide lenses intercepted in the drill holes is variable, ranging from less than 5 m to 70 m. The massive sulphide mineralized horizon appears to have been thickened in the nose of the antiform. The width of the deposit measured between the interpreted limbs of the antiform, varies from 100 meters to 175 meters.

The host rocks to the mineralization are predominantly rhyolites and altered rhyolites . Basalts occur to the east of the deposit, similar to the stratigraphy identified at the Bisha Main Zone. The rhyolites occur principally as tuffs, with minor blocky flows and agglomerates found immediately west of the NW Zone. These rocks are cross-cut by several generations of mafic and felsic dykes, some of which contain significant gold content.

Oxide and supergene mineralization is lacking or limited in extent with the majority of the base metal mineralization occurring between 30 m and 150 m of surface. The majority of the massive sulphide intersections typically contain 0.3% to 1% copper mineralization, with the supergene zone generally having a higher copper content than the primary zone. The average grades of the main pyritic massive sulphide lens are 0.5% copper, 0.1% zinc, 0.5 g/t gold and 20 g/t silver. Zinc mineralization is not significant over most of the NW Zone, except on the southwestern most part of the prospect. Geological interpretation indicates that a zinc-rich lens of massive sulphides intersected in drill holes NW-08, NW-015, NW-023, NW-024, NW-025 and NW-026 is likely a separate, mineralized body in the hanging wall of the main, largely pyritic, body. The zinc-rich lens has a length of 200 m and has a thickness of 10 m to 15 m along the limbs and 25 m to 40 m in the nose of the antiform. The average grades are 0.9% copper, 3.6% zinc, 0.3 g/t gold and 34 g/t silver.

The discontinuous nature of the massive sulphides and the limited drilling completed to date makes interpretation of true widths difficult.

Figure 7-4: Geological Cross-Section, NW Zone

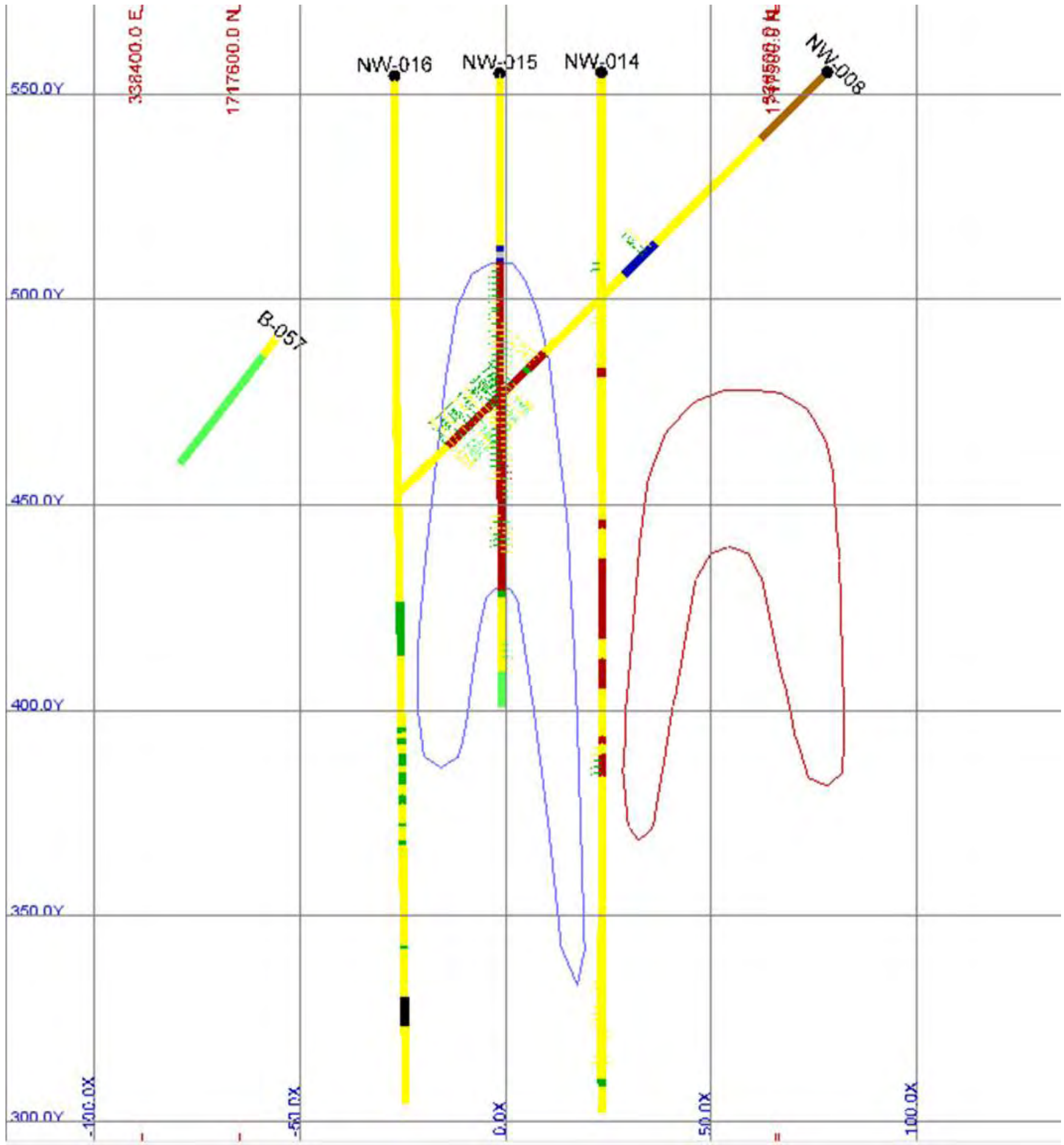


Figure courtesy Nevsun

7.8.3 NW Barite Hill Prospect

Geological mapping in the area in the northwestern corner of the Project has revealed a significant amount of barite mineralization scattered over a relatively large surface area.

Extrapolations of the geophysical trend of the strike of the Hambok massive sulphide occurrence, discovered in 2006, and held by Sanu Resources, and outside the Project boundary, (Giroux and Barrie, 2009) indicate that the NW Barite Hill Prospect and areas to the southwest of the barite occurrence are along strike from the Hambok deposit.

In June 2010 three diamond drill holes (408 m) were drilled at the NW Barite prospect to test gravity anomalies with coincident soil geochemical anomalies. No significant mineralization was intercepted in any of the holes. No further exploration will be conducted in this area.

7.9 Comment on Section 7

In the opinion of the QPs, the deposit settings, lithologies, and structural and alteration controls on mineralization in the Bisha Main Zone are well understood, and the geological understanding is sufficient to support Mineral Resource and Mineral Reserve estimation. The understanding of the Harena deposit is sufficient to support Mineral Resource estimation.

The Hanging Wall Copper Zone is at an earlier stage of exploration, and the lithologies, structural, and alteration controls on mineralization are currently insufficiently understood to support estimation of mineral resources. The NW Barite zone is considered to be sufficiently explored and no further exploration will be conducted in this area.

8.0 DEPOSIT TYPES

The Bisha Main Zone is a large precious and base metal-rich volcanogenic massive sulphide (VMS) deposit. The NW Zone and Harena are small to medium size VMS deposits and prospects. Pertinent deposit model types would be Noranda/Kuroko (Franklin et. al., 1981) or bimodal-siliciclastic VMS deposits (Barrie, 2004). The Matagami deposit in the Matagami VMS District in Quebec is a relevant and comparable deposit given the size (25 Mt), host rocks, proximity to a mafic complex, and several other features (Barrie, 2004).

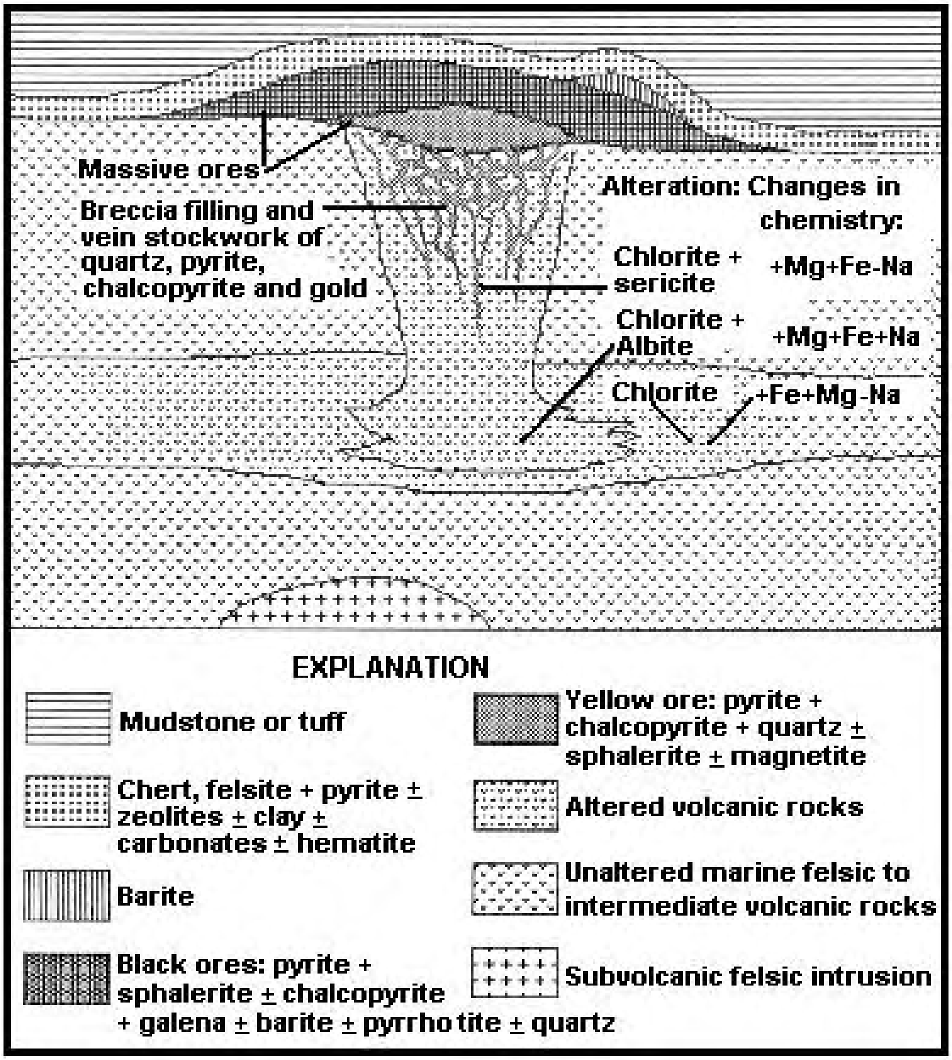
8.1.1 Noranda/Kuroko VMS Deposit Model

Noranda/Kuroko style volcanogenic massive sulphide deposits are noted for their high grade polymetallic nature, associated precious metal content, moderate to large tonnages, and occurrence of multiple lenses or horizons within mineralized districts. Key characteristics of Noranda/Kuroko-style volcanogenic massive sulphide deposits (after Höy, 2004) are:

- Marine volcanism, formed during period of felsic volcanism in an andesite or basalt dominated succession
- Associated with faults, grabens, and prominent fractures
- Associated with felsic or intermediate (or both) volcanic rocks including epiclastics
- Polymetallic (copper, lead, zinc plus gold and silver) massive sulphide deposits
- Massive to well-layered sulphides, sedimentary textures
- Quartz, chlorite, sericite alteration near the deposit centre to clay, albite, carbonate minerals further out
- One or more lenses within felsic volcanic rocks in a calc-alkaline bimodal arc succession
- Cu-rich base, Pb-Zn rich top
- Low-grade stockwork zones underlie lenses
- Barite and chert layers, lateral gradation into chert horizons
- Each of these features is present at Bisha with the exception of the host volcanic rock geochemistry, which is subalkaline (Greig, 2004).

Franklin et. al. (1981) presented a schematic section of the Kuroko volcanogenic massive sulphide deposit model (Figure 8-1).

Figure 8-1: Kuroko-style VMS Deposit Model



Note: Modified from Franklin et. al., 1981 by Singer and Mosier 1986.

The model is simple relative to the geological model of Bisha; the deformation, near-surface oxidation, and regional metamorphism at Bisha could easily have masked the Kuroko-style alteration pattern and stockwork zone shown in the figure.

Tonnage and grade estimates to date for the Bisha Main Zone deposit indicate the deposit is larger than most of the typical Kuroko VMS deposits based on grade–tonnage models by Singer and Mosier (1986; Figure 8-2A). The precious metal and base metal grades of the Bisha Primary Domain mineralization are in the higher percentiles for the grade-tonnage models (Figures 8-2B and 8-2C; base metal grade-tonnage models are not shown).

8.1.2 Bimodal Siliciclastic VMS Deposit Model

Barrie and Hannington (1999) have proposed a five-part classification system for VMS deposits based on host rock composition. Two of the types potentially describe the Bisha Main Zone deposit: bimodal siliciclastic, or mafic siliciclastic. Barrie (2004) visited the Project and concluded that the bimodal siliciclastic model was most appropriate. Mapping by Greig (2004) also indicated that the host rock is principally felsic volcanic rock (variably altered felsic lapilli, lapilli ash tuffs, crystal tuffs, and minor felsic dykes).

Many characteristics of the Kuroko VMS deposit model also apply to the bimodal siliciclastic VMS model. Bimodal siliciclastic deposits form in lithological sequences composed of roughly equal proportions of volcanic and siliciclastic rocks. Typically, felsic volcanic rocks are more abundant than mafic rocks, and are calc-alkalic in composition, while mafic rocks are of tholeiitic composition. Deposits are generally of Phanerozoic age, and are typified by the deposits of the Iberian Pyrite Belt and the Bathurst camp of New Brunswick. Barrie (2004) considers the Bisha Main Zone deposit to be similar to those of the Iberian Pyrite Belt.

Barrie (2004) developed a VMS model for the Bisha Main Zone deposit as shown in Figure 8-3. The model incorporated local features such as the Bisha Gabbroic Complex and dominantly felsic and siliciclastic host rocks.

Figure 8-2: Kuroko Style VMS Grade and Tonnage Model (Singer and Mosier, 1986)

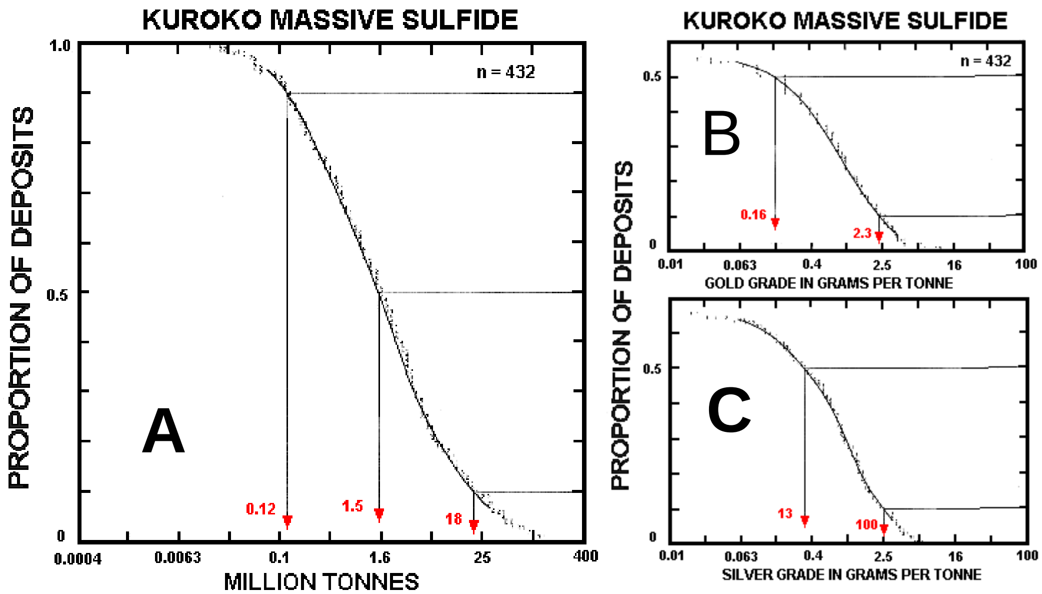
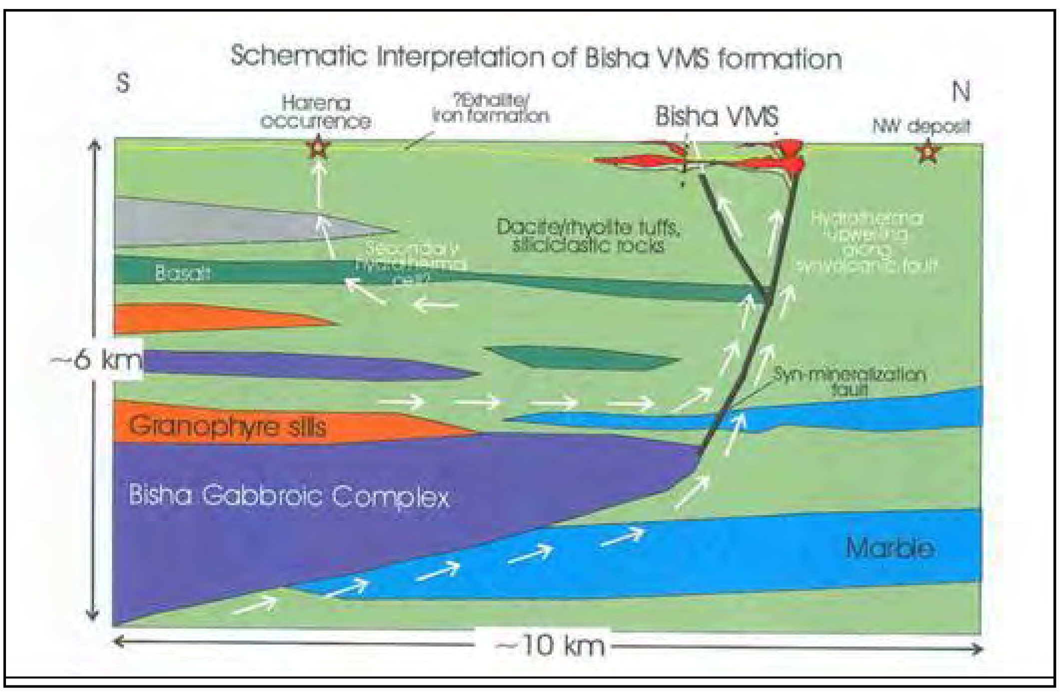


Figure 8-3: Bisha Bimodal Siliciclastic VMS Model Schematic



Note: Figure from Barrie (2004)

Bimodal siliciclastic deposits represent the largest VMS tonnage. However, on average, they have the lowest Cu (1%) and the highest Pb (1.8%) metal content of the five deposit types (Barrie and Hannington, 1999), while also having relatively high Zn (4%), high Ag (90 g/t), and low Au (1 g/t) contents.

Franklin (1998) described deposits of the Iberian Pyrite Belt and noted that they are characterized by great lateral continuity of mineralization, as well as lack of extensive alteration. Both characteristics appear to have relevance to the Bisha Main Aone deposit. The Iberian Pyrite Belt deposits range from small lenses of a few million tonnes to very large bodies that contain over 100 Mt.

8.2 Comment on Section 8

In the opinion of the QPs, either VMS deposit model is applicable to the Bisha Main Zone and Harena deposits.

9.0 MINERALIZATION

A discussion of the mineralization styles and related depth, width (thickness), orientation, and continuity is presented by deposit in Section 7 of this report. The discussion below relates to the mineralization type, character, and mineralogy.

9.1 Bisha Main Zone

The four principal domains of mineralization within the Bisha Main Zone include:

- A near-surface oxide/gossan
- A horizon that has been subjected to extreme acidification[1] (acidified)
- A supergene copper-enriched horizon
- A primary massive sulphide horizon.

The ferruginous to massive goethite–hematite–jarosite gossan is the remnant of surface oxidation of the massive sulphides. It consists of a large mound of red–brown oxide material ranging from fine sand to dense cobbles and boulders, distributed randomly or as groups or possible remnants of stratigraphic “horizons”. The boulders and cobbles are usually extremely siliceous. The depth of oxidation is variable but appears to be in the order of 30 m to 35 m in outcrop areas. The unit has a high gold content; the relatively low base metal values (copper, zinc) are due to leaching during oxidation. Banded, white, opaque, quartz veins ranging up to 0.5 m in thickness and several metres in length may occur in some of the gossans.

Flanking the gossan is a breccia unit, which appears to be a product of oxidation, lateritic weathering, and desegregation of the original rock as opposed to being a structural feature. The unit is mostly quartz breccia or silicified fragments within oxidized material.

The acidified horizon is the result of the extremely acidic nature of the oxidation of the massive sulphides that caused development of a highly leached “front”, causing very friable remnants consisting of mostly of clay and silica. The thickness of the “soap” horizon is variable ranging from 0.5 m to 6 m, and averaging 3 m in thickness. This unit has high gold and silver values but is usually devoid of significant base metal mineralization (with the exception that it can locally contain appreciable amounts of supergene copper mineralization)

1 Acidification of the massive sulphides and host rock results in remnant clay and silica, which is logged as ACID or SOAP rock codes.

The supergene mineralization is copper-enriched and occurs between 35 m to 65 m depth. As in the supergene enrichment of porphyry deposits, oxidation of the massive sulphides caused the descending waters to become acidic and leach copper and other metals. The metals were deposited generally as covellite and some chalcocite at the base of the acid and oxide domains. Sooty secondary sulphides coat and replace primary sulphides.

Primary sulphide mineralization occurs typically below a vertical depth of 60 m to 70 m. Sulphide minerals are predominantly pyrite, with some sphalerite and chalcopyrite. Sphalerite appears to be more abundant at the south end of the Bisha Main Zone deposit.

Textures include semi-massive, massive, banded/laminated, minor folds, clasts, and disseminated sulphides within chloritized volcanics.

Figure 9-1 shows a typical section through the northern portion of the Bisha Main Zone deposit. The section shows the angle that the drill holes intercept the mineralization, the different mineralization domains, and representative copper and gold composite assays. Figure 9-2 presents an isometric section view of the Bisha Main Zone deposit, showing the mineralized zones. Figure 9-3 shows the mineralized zones as a perspective view, from above.

Figure 9-1: Section 6025 N

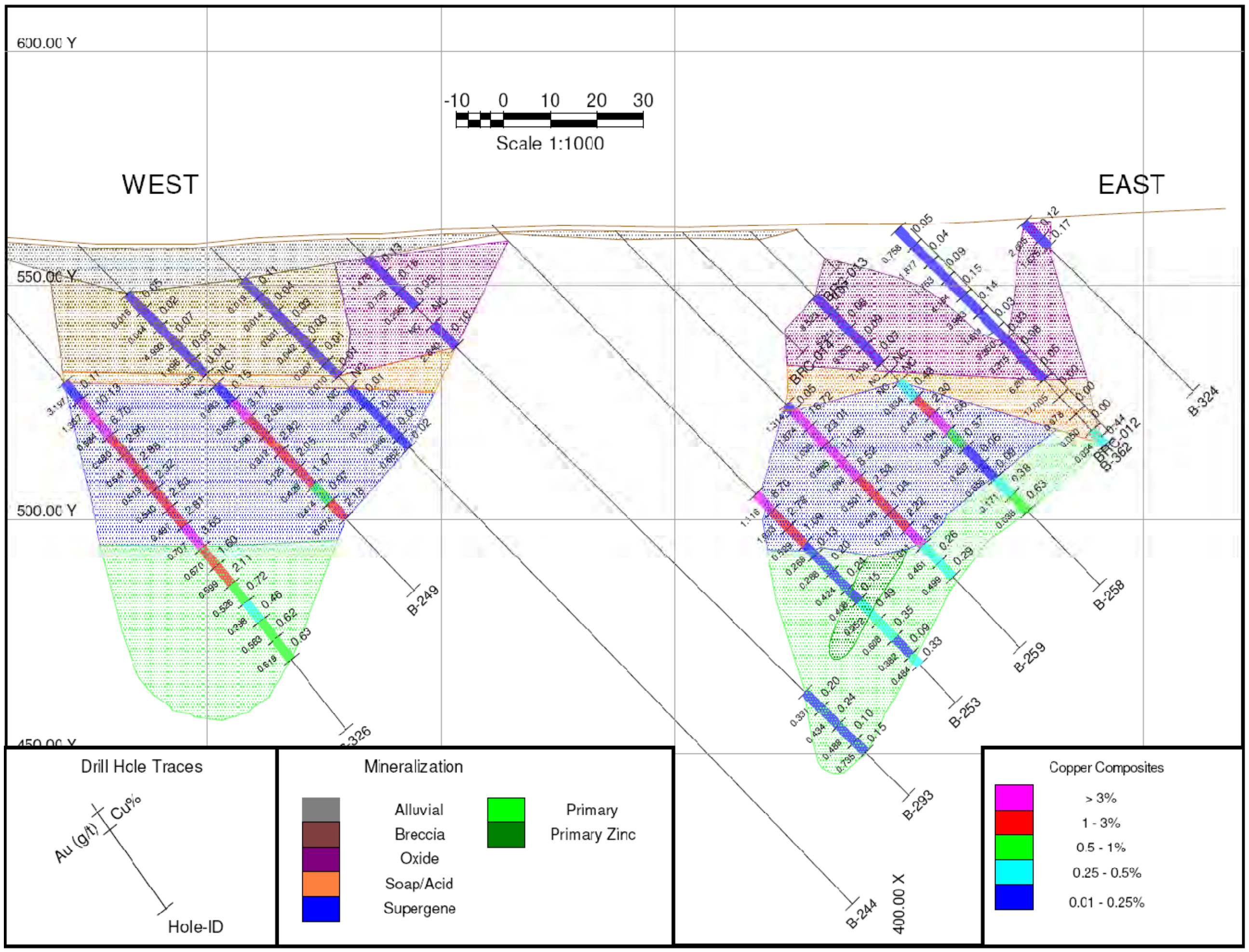


Figure courtesy Nevsun

Figure 9-2: Isometric View of the Bisha Deposit Facing West

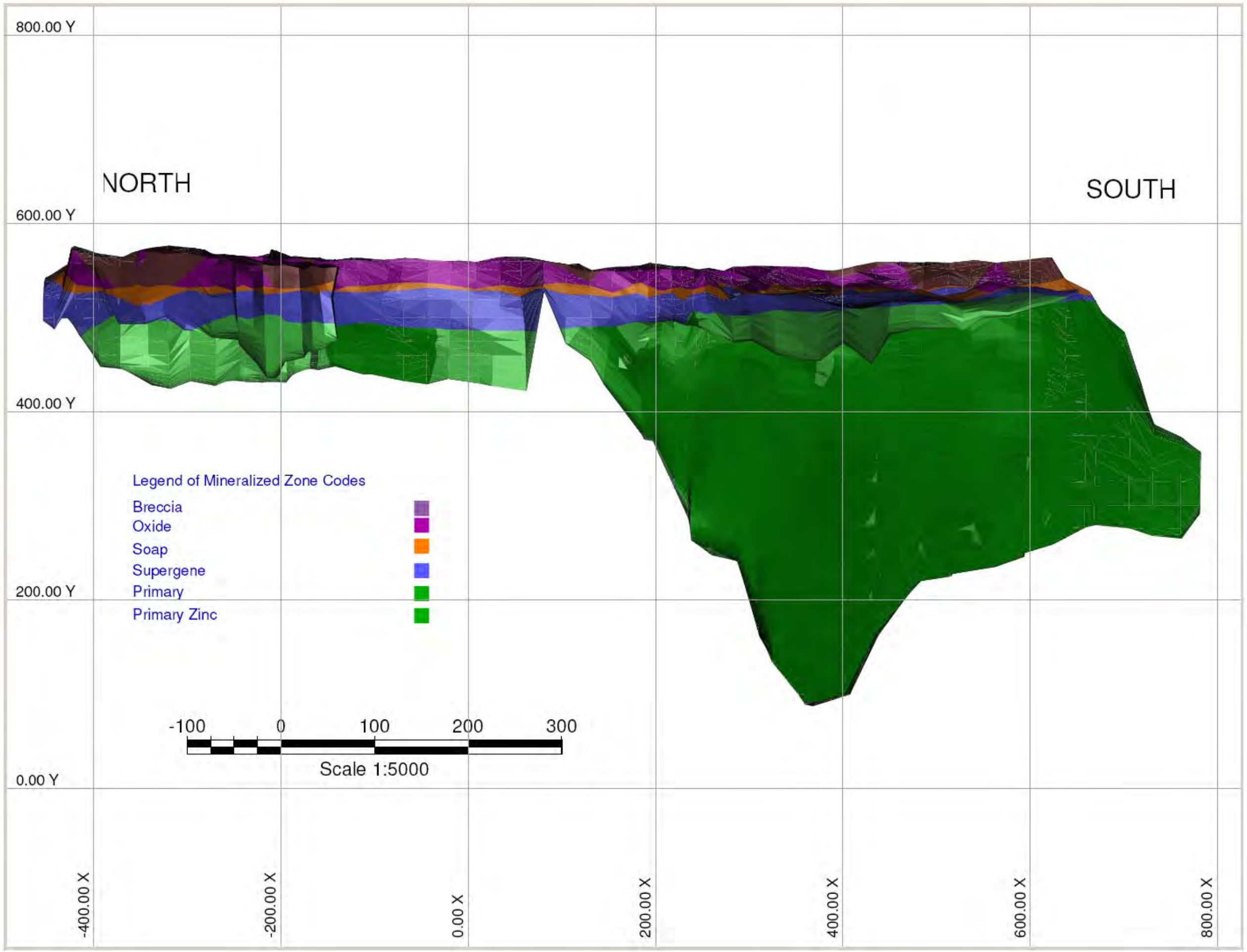


Figure courtesy Nevsun

Figure 9-3: Bisha Main Zone Outline (perspective view from above)

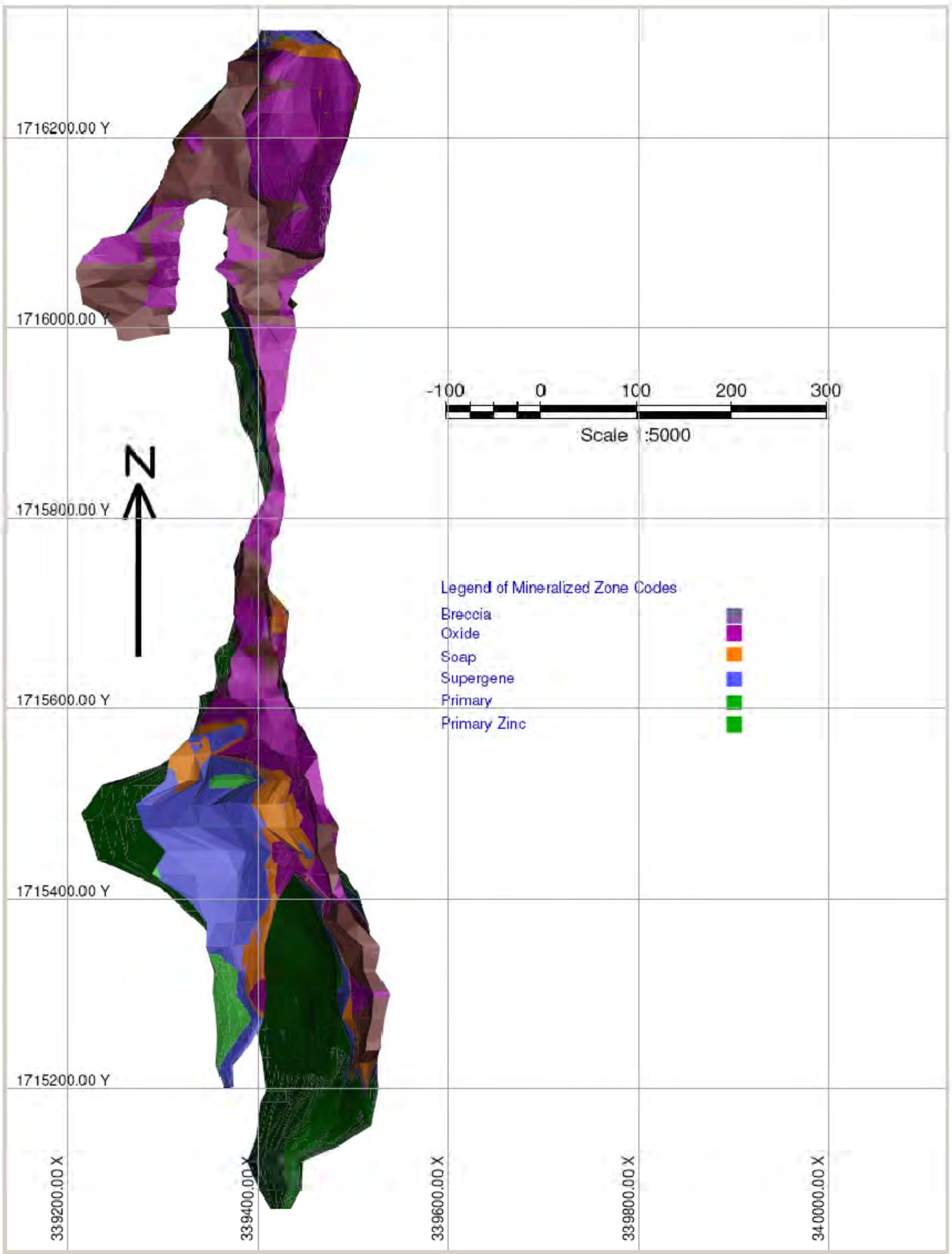


Figure courtesy Nevsun

9.1.1 Harena

Surficial weathering processes have produced three distinct zones of mineralization within the massive sulphide deposit at Harena. These include a surface oxide/gossan, overlaying a secondary supergene horizon, which grades into a primary massive sulphide horizon at depth.

The depth of oxidation appears to be on the order of 45 m to 50 m. For the most part, copper and zinc appear to be leached from the gossan material, although secondary lead is often present.

At Harena, there is a narrow supergene zone, with the primary sulphide zone generally encountered below a vertical depth from surface of 55 m. The deepest sulphide intersection to date is at a vertical depth of approximately 175 m. The deposit has a very high total sulphide content, in excess of 95%, typically with very little gangue.

The primary massive sulphides are predominantly made up of fine- to medium-grained subhedral to anhedral pyrite, with interstitial and/or enriched layers of sphalerite and chalcopyrite, and occasional gypsum veins or fracture-fill. The massive sulphide lenses also commonly contain patchy or narrow graphitic mudstone beds.

Figure 9-4 shows a typical cross section through the mineralized zone.

9.1.2 Hanging Wall Copper Zone

In the Hanging Wall Copper Zone, the mineralization is predominantly supergene copper mineralization consisting of chalcocite and covellite sulphidic stringers in chloritic-altered rocks and chalcocite, covellite and hematite in the “soap” rocks. Mineralized zones proximal to the massive sulphides appear to be quite erratically distributed. Supergene copper minerals tend to occur as a horizon gently dipping away from the massive sulphide lenses. The geometry of the mineralization is likely a result of underground fluid flow and fluctuations in the position of the paleo water-table.

Figure 9-4: Harena Northwest-Southeast oriented Section 3, looking Northeast

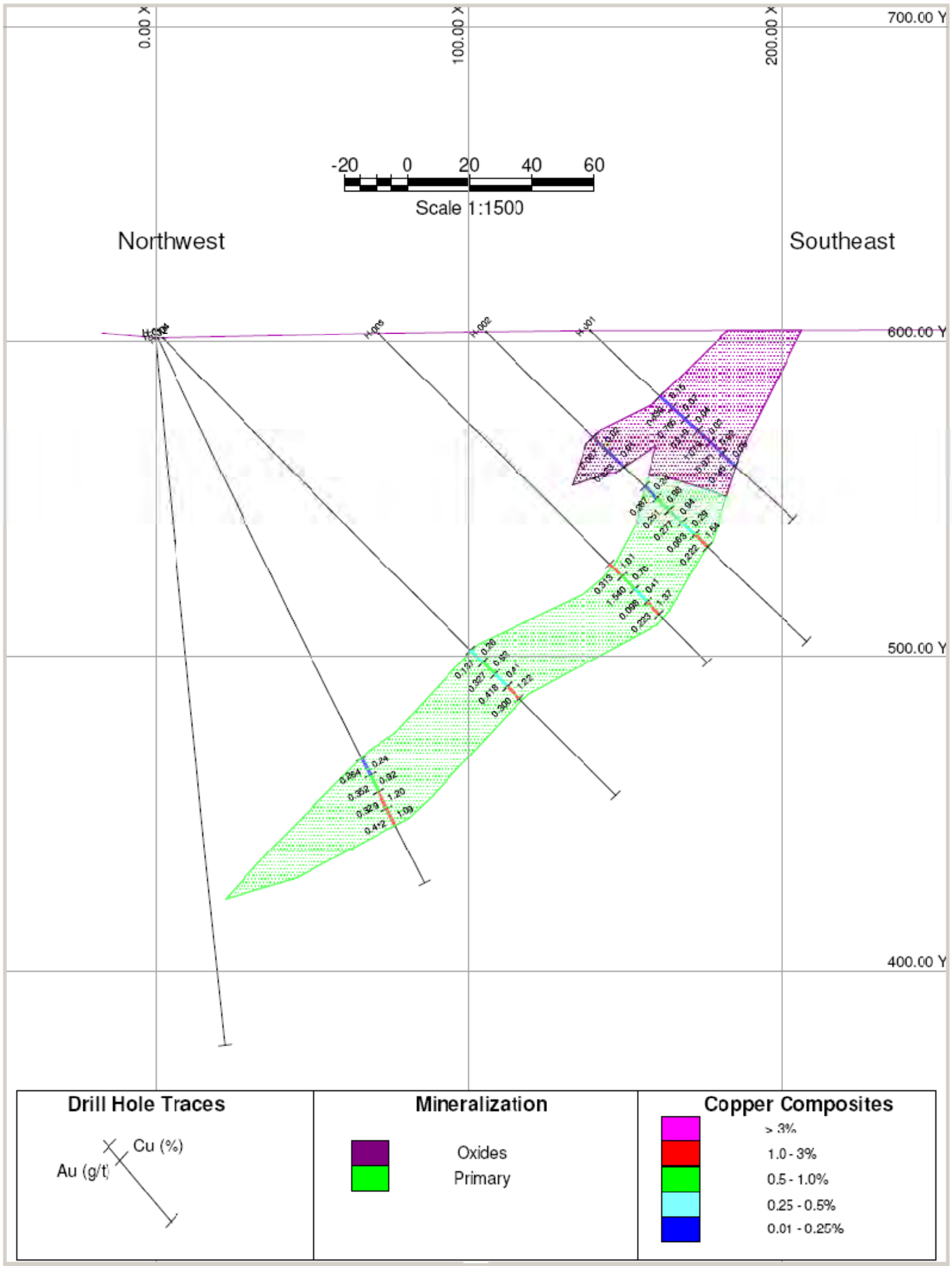


Figure courtesy Nevsun

9.1.3 Northwest Zone

The configuration of the Northwest Zone is such that oxide and supergene mineralization is lacking, or limited in extent, with the majority of the base-metal mineralization occurring between 30 m and 150 m of surface. The majority of the massive sulphide intersections typically contain 0.3–1% Cu, with the supergene zone generally having a higher copper content than the primary zone. Zinc mineralization is not significant over most of the NW Zone, except on the southwestern most drill section completed (NW-008 and NW-015). Additional drilling is required to further define the extent of the zinc mineralization to the southwest. Figure 9-5 shows a cross-section through the Northwest Zone.

The supergene zone at the NW Zone contains secondary base-metal oxide, carbonate, sulphate, phosphate, silicate, halide, and native base-metal minerals. This zone is characterized by dissolution of carbonate minerals, thus creating voids and increasing porosity, the coating of primary sulphides with sooty secondary sulphides, and the replacement of primary sulphides by secondary sulphides.

The primary sulphide zone is generally encountered below a vertical depth from surface of 60 m to 80 m. The primary massive sulphides predominantly comprise subhedral to anhedral pyrite with very fine interstitial chalcopyrite and lesser sphalerite. Minor galena is noted locally. Sphalerite banding, similar to what is seen at the Bisha Main Zone was encountered on the southwestern-most section drilled (NW-008 and NW-015). Gypsum veins are frequently intersected within the massive sulphide body.

Gold mineralization has been intersected in at least four different rock types (quartz veins, at the massive sulphide contact, mafic, and felsic dykes) associated with the supergene and primary sulphide horizons. In all instances, strong chlorite alteration is apparent, and disseminated sulphides are present.

Figure 9-5: Northwest Zone, East-West oriented Section 7 looking North

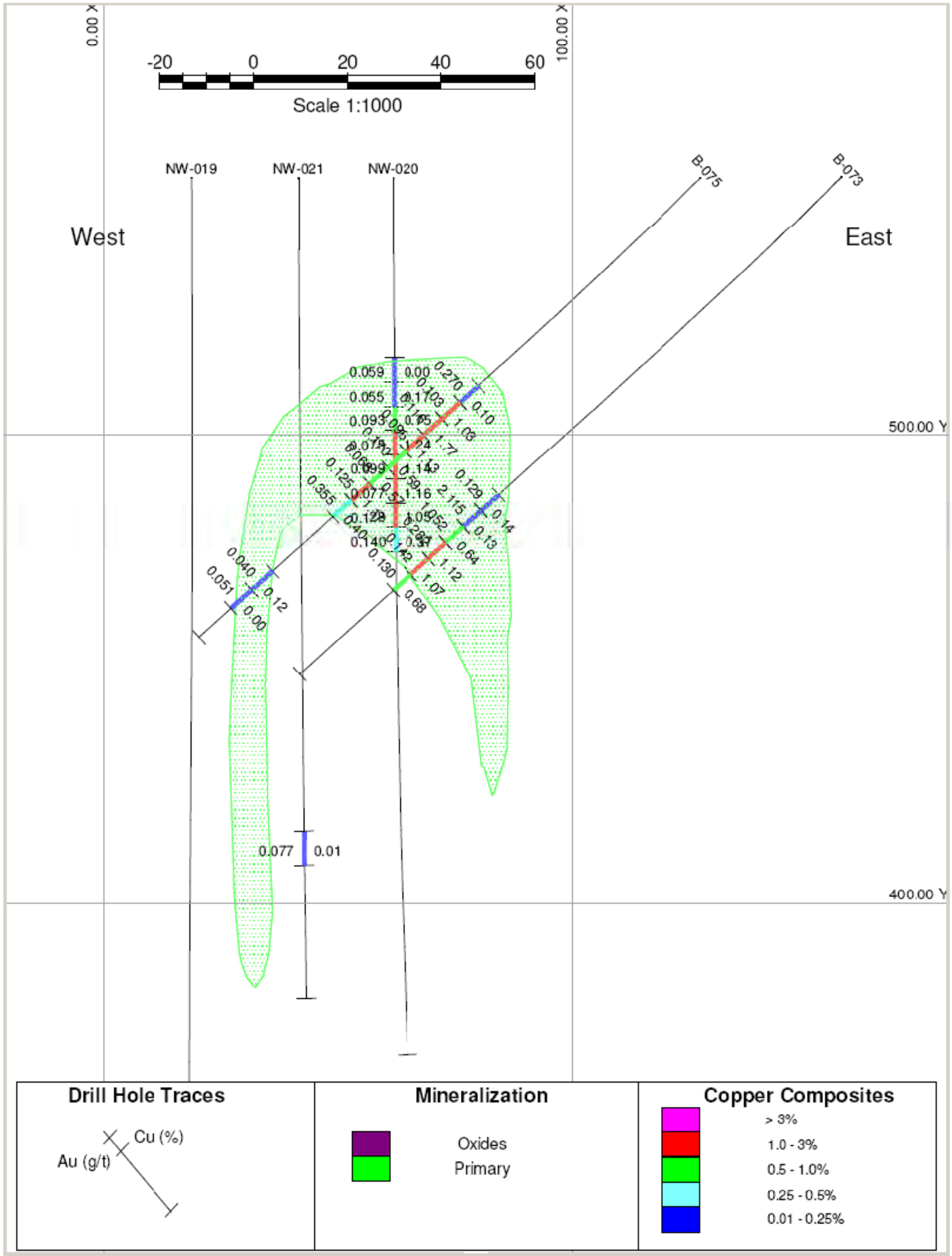


Figure courtesy Nevsun

9.2 Comment on Section 9

In the opinion of the QPs:

- The mineralization style and setting of the Bisha Main Zone is sufficiently well understood to support Mineral Resource and Mineral Reserve estimation
- The presence of deleterious elements within the Bisha mineralization for the process route is known (refer to Section 16)
- The mineralization style and setting of the Harena deposit is sufficiently well understood to support Mineral Resource estimation
- The mineralization style and setting of the Hanging Wall Copper Zone and Northwest Zone are currently insufficiently defined to support mineral resource estimation.

10.0 EXPLORATION

Exploration activities on the Project have included geological mapping, geochemical sampling, geophysical surveys, and drilling, and are summarized in Table 10-1. Exploration activities and results are discussed in the following subsections; drilling results are discussed in Section 11.

Activities were conducted either by Nevsun personnel, or by consultants and contractors appointed by Nevsun.

10.1 Survey

The coordinate system used for all data collection and surveying is the Universal Transverse Mercator (UTM) system, Zone 37 and geographic coordinates in WGS84[2] (World Geodetic System 1984).

During the 1999 exploration program, Nevsun established a local grid (not based on UTM coordinates) over the gossan area with a base line 5.9 km in length oriented at an azimuth of 010° from magnetic north. Individual lines were usually spaced 200 m apart and were of variable length.

The local grid constructed at the beginning of the 2003 program conforms to the UTM coordinate system with a baseline oriented at 0° and cross-lines oriented 090°. Cross-lines were usually spaced 100 m apart, except over the Bisha Main Zone where gridlines were spaced 25 m apart for drilling.

10.2 Geological Mapping

Geological mapping was completed to provide information on outcrop and gossan extents, geological units, and structure, at scales ranging from regional to project, including 1:50,000; 1:10,000; 1:5,000; 1:,2,500, and 1:1,000.

2 WGS84 is the datum to which all GPS positioning information is referred by virtue of being the reference system of GPS satellites. WGS84 is an earth-fixed Cartesian coordinate system.

Table 10-1: Summary of Work Completed

Year	Phase	Company	Type of Work	Description
1996		Ophir Ventures	Regional Grassroots Exploration	Prospecting, mapping and sampling
1998		Nevsun	Property Evaluation	Property examination and acquisition
1998		Nevsun	Property Grassroots Exploration	Reconnaissance scale geological mapping (1:50,000), geochemical stream sediment sampling
1999		Nevsun	Geophysical Surveys	Geophysical surveys - MaxMin horizontal loop EM and magnetometer
			Geological Mapping	Property scale (1:5,000)
			Geochemical Sampling	Soil sampling on three grid lines
2002		Nevsun	Drilling	6 core drill holes (B-01 to B-06) totalling 759.0 m
			Geological Mapping	Discovery outcrop area (1:1,000)
2003	I	Nevsun	Drilling	47 diamond drill holes (B-07 to B-53, B-02a) totalling 6722.6 m
			Trenching	36 trenches sampled and mapped
			Geophysical Surveys	Airborne EM and magnetometer (325 sq km), pulse EM and ground magnetometer (73.5 line km), gravimetric survey (40 km)
			Geological Mapping	Deposit scale (1:1,000), property scale (1:2,500) and regional scale (1:10,000) geological mapping
			Geochemical Sampling	Stream sediment (165 samples), soil (39 samples), termite mound (115 samples), auger and pit (33 samples)
			Petrographic Study	11 thin sections by Vancouver Petrographics
			Metallurgical Testing	2 oxide samples, 2 copper supergene mineralization and 2 primary mineralization samples
			Bulk Density	260 samples determined on site, 44 samples sent to ALS Chemex for determination
2003	II	Nevsun	Drilling	93 core drill holes (B-54 to B-146, & deepen B-40) totalling 11,750.8 m
			Drilling	2 air blast holes for water wells completed by Eritrean Drilling
			Geophysical Surveys	Pulse EM, horizontal loop EM (151 line km), gravimetric survey (107.6 km)
			Geochemical Sampling	pH soil survey, soil sampling (40.3 line km), whole rock (REE), regional prospecting
			Metallurgical Testing	2 oxide samples and 2 copper supergene mineralization samples at PRA in Vancouver, some minor work at Kappes Cassidy in Nevada
			Petrographic Study	13 thin sections by Vancouver Petrographics
			Bulk Density	611 samples determined on site, 68 samples sent to ALS Chemex for determination
2004	I	Nevsun	Drilling	163 core drill holes (B-147 to B-309) totalling 28,879.50 m
			Drilling	42 reverse circulation drill holes (BRC-01 to BRC-42) totalling 2,097.3 m
			Drilling	9 combination reverse circulation with drill core holes tails (BRCD-26,27, 32 to 34, 37,38, 41 and 42) totalling 308.70 m
			Drilling	15 reverse circulation holes for water wells totalling 768 m.
			Geophysical Surveys	Gravimetric survey, 65.2 line km
			Geological Mapping	Deposit scale (1:1,000) mapping and regional prospecting
			Geochemical Sampling	Soil sampling (111.6 line km), Whole Rock (REE), prospecting
			Petrographic Study	16 thin sections, 2 polished sections
			Bulk Density	311 samples determined on site, 697 samples sent to ALS Chemex for determination
			Geotechnical Work	All drill core oriented
			Environmental	Base line study implemented
			Metallurgical Testing	2 primary sulphide samples tested at PRA in Vancouver
			Hydrological	Studies commenced
			Archaeological	Studies commenced
			Physical Properties Tests	On selected core samples of massive sulphide by JVX Geophysics
2005		Nevsun	Drilling	112 diamond drill holes totalling 16,074.3 m
			Petrographic Study	10 thin sections by Vancouver Petrographics

Year	Phase	Company	Type of Work	Description
			Geochemistry	Whole rock analyses, petrographic studies, soil sampling
			Feasibility Studies	Metallurgical sampling, testwork, geotechnical studies, other studies.
			Geological mapping	Deposit scale (1:1,000) Harena and proposed tailings containment area
			Geophysical surveys	Gravity, HLEM, ground magnetometer
			Hydrological	Ongoing studies
			Archaeological	Ongoing studies
			Trenching	5 trenches at Harena (166.5 line meters)
			Geotechnical Work	All drill core oriented, point load and packer testing
			Metallurgical Testing	1 bulk sample of primary sulphide tested at SGS Lakefield in Canada
			Bulk Density	Harena and NW Zone prospects
			Geotechnical Pit Excavation	59 pits in areas of the proposed processing plant, accommodation and tailings dam
2006		Nevsun	Drilling	8 core drill holes totalling 1,680 m
			Feasibility Studies	Metallurgical sampling, testwork, geotechnical studies, other studies.
			Geochemistry	Soil geochemistry on the NW Barite, HW copper zone, target 4 to target 9 geophysical anomalies.
			Geophysical Surveys	IP / Resistivity survey, Harena, NW zone, NW Barite Hill and south of Bisha Main deposit.
			Trenching and Pitting	9 trenches on gold in soil anomalies and HW copper zone, 14 pits excavated HW copper zone
			Geological mapping and prospecting	Regional scale
			Exploration of a source of Aggregate	42 test pits in a basaltic dike
			GPS Surveys	Proposed mine site, roads to Massawa and port area
2007		BMSC/Nevsun	Geophysical Surveys	Gravimetric survey, 13.5 line km. Target 9 area
2008		BMSC/Nevsun	Prospecting	Targets 4 and 9
			Pitting	Targets 4 and 9, Bisha South and NW Barite Hill. 10 pits
			Trenching	Targets 4 and 9, Bisha South and NW Barite Hill. 6 trenches for 466 line meters.
			Geological Mapping	Target 9 area 1: 2000 scale
2009		BMSC	Geophysical Surveys	Gravimetric Survey, 32 line km
			Drilling	17 diamond drill holes totalling 2,163.5 m
			Geological Mapping	Northwest of T9 area 1 : 5000 scale
			Geotechnical Work	9 oriented drill core holes and point load testing
			Metallurgical Testing	Bulk sampling of supergene material tested at Mintek in South Africa
2010		BMSC	Drilling	47 core drill holes totalling 4,366 m
			Geophysical Surveys	Gravimetric survey, 70 line km. South Tabakin (33 line km) and NW Barite Hill (37 line km)
			Petrographic Studies	6 thin sections by Vancouver Petrographics & 11 polished thin sections by Carleton University
2011		BMSC	Drilling	20 core drill holes totalling 2,590.9 m

10.3 Remote Sensing and Satellite Imagery

Two remote sensing studies have been performed.

Nevsun prepared Landsat images of the area in 1998. Using the Landsat image with a translucent (30%) topographic overlay, a preliminary interpretation was made of the immediate Bisha occurrence area. The interpretation focused on structural features that were subsequently plotted on the geology map as well as the preliminary gravity maps.

During 2003, Earth Resource Surveys Inc. (ERSI) based in Vancouver, completed a remote sensing investigation for the Bisha Project and western Eritrea. The survey mapped alteration types and interpreted major structural features using different Landsat bands, and highlighted alteration and structural trends (Chisholm et. al., 2003).

10.4 Geophysical Surveys

10.4.1 Ground Geophysics

Horizontal loop and pulse electromagnetic (EM) surveys were completed in 1999, 2003, and 2005. Most of the electromagnetic surveys were successful in delineating mineralization and other features.

Magnetometer surveys were completed on portions of the Project in 1999, 2003, and 2005. The magnetometer surveys showed lithological contrasts and anomalies over the Bisha Main Zone but generally features were less distinct than the electromagnetic surveys. Chisholm et al., (2003) observed north–northeast-trending features that were interpreted to show that a fault zone that transects the Bisha Main Zone deposit.

Gravity surveys were carried out during 2003, 2004, 2005, 2007, and 2009. The filtered residual gravity surveys provided good definition of the Bisha Main Zone, NW Zone and Harena massive sulphide mineralization.

In December 2005, Nevsun conducted a reinterpretation of previous airborne geophysical surveys and ground geophysical surveys. Induced polarization (IP) and resistivity surveys were carried out over the Harena, NW Zone, and NW Barite Hill prospects, and over the area to the south of the Bisha Main Zone. The results of the IP/resistivity surveys showed a distinct low resistivity anomaly coincident with the massive sulphide body at the NW Zone. At Harena, there is a low amplitude resistivity anomaly associated with the sulphide mineralization. The IP and resistivity responses at the NW Barite Hill area are characterised by weak to moderate chargeability highs with associated resistivity lows.

In April 2007, BMSC initiated a 13.5 line-km gravity survey over a newly interpreted target area on the Project referred to as Target 9. The gravity survey was conducted by MWH Geo-Surveys Inc. The survey lines were spaced 200 meters apart with stations spaced at 25 meter intervals along the survey lines. The gravity survey identified a weak residual gravity anomaly with coincident EM conductors. This, combined with previously-identified soil geochemical anomalies, outlined an area of potential volcanic massive sulphide (VMS) mineralization that required follow-up investigation.

In 2009, the gravity survey was extended to the southwest corner of the exploration licence, with an additional 32 line-km of data collected (refer to Figure 10-1). The survey provided further definition of gravity highs to the southwest and along strike of the Harena deposit. Ongoing analysis of the data should help further identify exploration targets in the area that may have potential for VMS mineralization.

In 2010, gravity surveys were initiated in two areas; a 37 line-km survey over the NW Barite occurrence and a 33 line-km survey in an area directly south of the Tabakin hills, south of Bisha. The survey lines were spaced 200 m apart with stations spaced at 25 m-intervals along the survey lines. Results of the survey south of the Tabakin hills identified two gravity anomalies associated with mafic intrusions. The survey at the NW Barite occurrence identified two strong gravity anomalies which were drilled in June 2010 intersecting mafic tuffs with no significant mineralization. Survey results are shown in Figure 10-1.

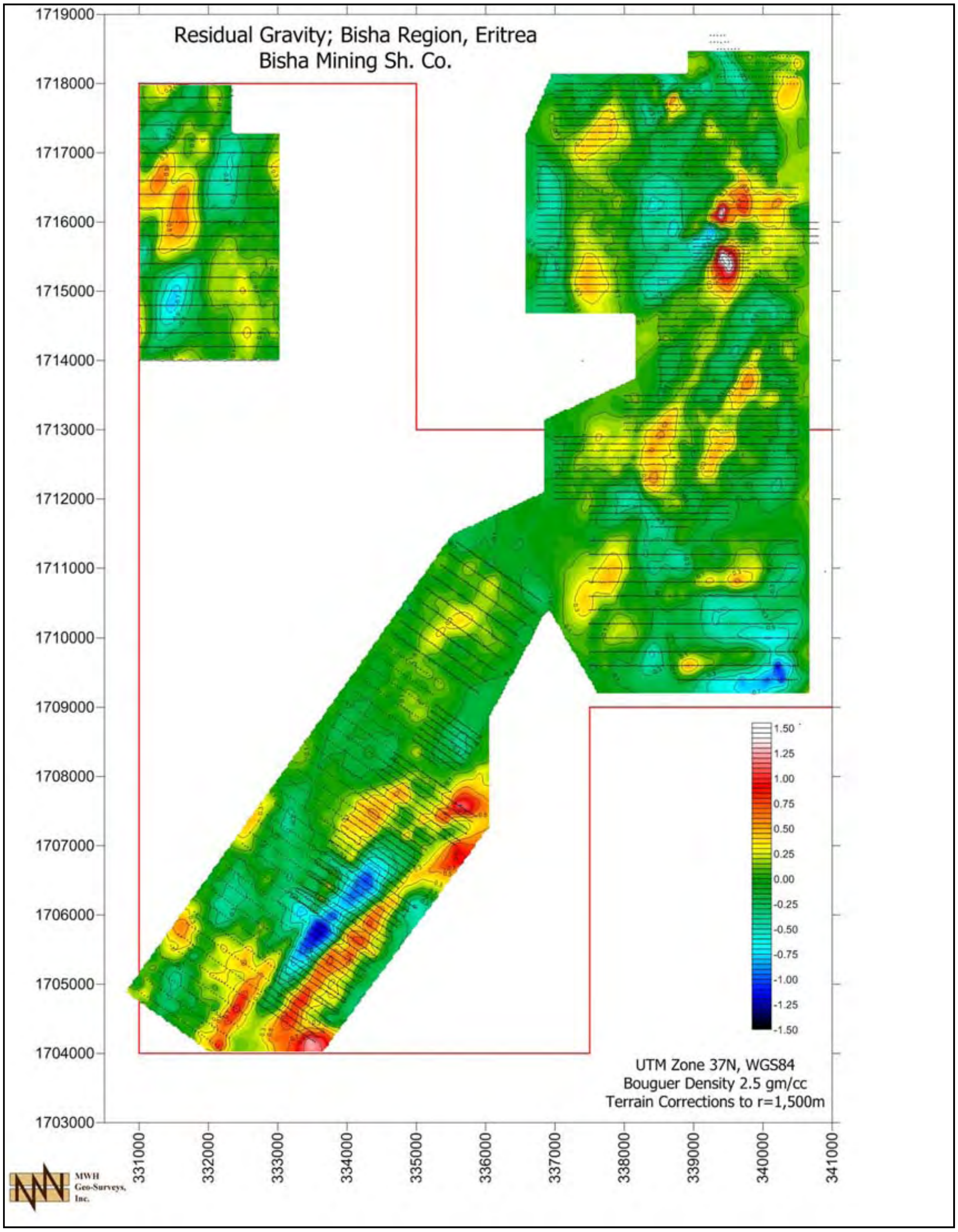
MWH Geo-Surveys Inc. performed the gravimetric surveys in 2009 and 2010.

10.4.2 Aerial Geophysics

In March 2003, a combined airborne EM and magnetometer fixed-wing survey was conducted over an area of approximately 325 km^2. The survey identified a number of horizons that were considered prospective for VMS mineralization.

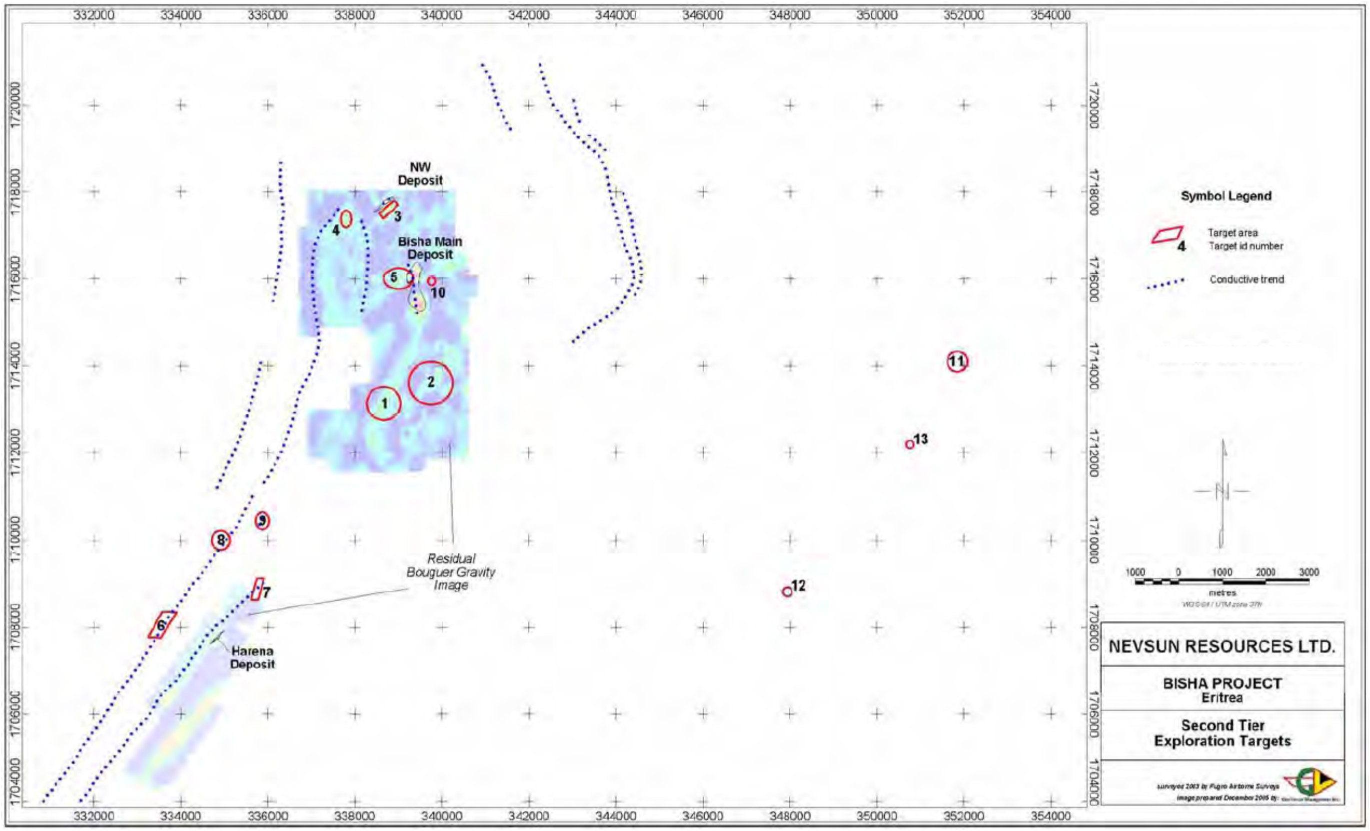
In December 2005, Nevsun reinterpreted the 2003 airborne EM survey, resulting in several priority areas being demarcated for further work. Several priority VMS targets were identified (marked as second-tier anomalies on the map), as shown in Figure 10-2.

Figure 10-1: 2010 Gravity Survey Results



Note: Figure courtesy Nevsun

Figure 10-2: Map Showing Interpreted Location of Airborne Magnetic Anomalies



Note: Figure courtesy Nevsun

The Target 4 area, located immediately to the west of the NW Zone, exhibits a coincident airborne electromagnetic and residual gravity anomaly. The position of the anomaly is marked on the ground by an abundance of gossanous boulders, suggesting the presence of massive sulphide mineralization.

Target 9 is located approximately 5 km to the southwest of the Bisha Main Zone deposit. Soil sampling over the area indicated by the airborne electromagnetic responses returned anomalous results from multiple elements. The area is considered prospective for VMS mineralization.

10.5 Geochemistry

10.5.1 Stream Sediment Sampling

Nevsun carried out stream sediment sampling in 1998, covering an area of 100 km². In 2003, a total of 165 stream sediment samples were collected at an approximate density of one sample per square kilometre.

The stream sediment surveys were considered to be an effective method of delineating areas with potential for base and precious metal mineralization. The main anomalous areas for copper, lead, zinc and gold based on the combined 1998 and 2003 results were the Okreb area (outside of the current Bisha Exploration License), the Bisha Main Zone southwards towards the Harena area, and the NW Barite Hill area.

10.5.2 Rock Chip Sampling

A total of 461 of rock chip samples were collected during mapping and prospecting of the Project area. The samples were used to help vector in on prospective areas for VMS mineralization.

10.5.3 Soil Geochemical Sampling

In 1999, soil samples were collected over the Bisha gossan outcrop (Nevsun 2003) and showed a distinct base metal anomaly. The samples were not analyzed for gold.

A total of 14,069 soil samples were collected between 2003 and 2006. Soil sampling was used to investigate geophysical anomalies, often in areas with minimal or no outcrop, and defined geochemical anomalies over selected geophysical targets and zones of known mineralization.

Initial soil sampling was performed in 2003, comprising orientation soil, termite mound, auger, and pit sampling. During the second exploration campaign in 2003, 40.3 line km of soil sampling was completed over the NW Barite Hill, Bisha Main and Harena areas. Based on the results of the orientation soil sampling, soil samples were collected at a shallow depth of no more than 10 cm at intervals spaced 25 m apart along grid lines spaced 50 m to 100 m apart. The grid lines were surveyed using a differential GPS unit.

Soil sampling in 2004 comprised 111.6 line-km completed over the Bisha Main Zone, Harena, and NW Barite Hill areas. The soil sampling resulted in the definition of a coincident multi-element Au, Ag, and Pb, anomaly over the Bisha Main Zone deposit.

At Harena, anomalous Cu, Pb, Zn, Ag, and Au values were identified in the south–central portion of the gridded area. At NW Barite Hill, the analytical results of the soil geochemical survey defined a widespread coincident Cu/Zn soil anomaly with subdued, weakly coincident, Au, Ag, and Pb.

As part of the 2005 exploration program, additional soil sampling (5,005 samples) was completed southwest of the Harena prospect, east of the Bisha Main Zone, and over the proposed tailings containment area in order to complete area coverage, or close-off previously-defined anomalous geochemical results. A

A total of 843 soil samples were collected over the NW Zone grid. For many of the contoured elemental plots, the highest values were present immediately west of the prospect; this represents an area where the footwall rhyolites rise to the surface, as the massive sulphide component of the deposit is at 30–50 m depth, trending north–northeasterly, and plunging shallowly to the north.

The Harena area was covered by a grid for soil and geophysical surveys. In total, 2,724 soil samples were collected over this target. Three targets were identified on the Harena soil sample grid; these three soil anomaly areas may indicate the presence of VMS mineralization at depth:

- At 1707000N/335450E, or ~750 m in the stratigraphic hanging wall from the Harena deposit, highlighted by all of the metals analyzed, except Cu and Fe
- At 1705300N/333850E, or 2.7 km southwest along strike from the Harena deposit, highlighted by Cu, Zn, Hg, S, and U
- At 1710000N/336750E, highlighted weakly by Cu, Fe, and Mo.

During 2006, soil samples were taken on the Target 4, 6, 7, 8, and 9 geophysical anomalies (refer to Figure 10-2). Targets 4 and 9 returned anomalous geochemical signatures; Targets 6, 7, and 8 did not.

Additional sampling was performed over the Northwest Zone, NW Barite Hill, and the Hanging Wall Copper Zone.

At the Bisha Main Zone deposit, additional soil samples were collected on a 100 m x 100 m grid spacing to complete the geochemical coverage of the interpreted location of the Hanging Wall Copper Zone. The limited add-on sampling at the Bisha Main Zone did not enhance the soil geochemical picture of the area to any great extent.

Additional soil sampling was completed over the NW Zone, extending the soil coverage westward. Grid lines were placed using hand-held GPS units in combination with chaining and back sight methods. Soil samples were collected at a shallow depth of no more than 10 cm along the grid lines at 25 m station intervals. Results from the samples collected show a continuation of the previously-defined mineralization trends. The Au-in-soil geochemical trend appeared to continue to the southwest and remains open in this direction. The Cu-in-soil geochemical anomaly appears to trend towards the Cu-in-soil anomaly defined over the Bisha Main Zone deposit. Additional, infill sampling remains to be completed to provide complete coverage between the Bisha Main Zone and NW Zone soil grids.

Soil sample coverage over the NW Barite Hill was resurveyed northward from UTM 1716800N to the northern exploration licence boundary, to provide coverage on 100 m x 100 m grid lines. The area was resampled because the previous analytical method used had a higher detection limit than the remainder of the survey data for the Project, making levelling of the analytical data problematic. The coverage area was also extended to conform to the soil geochemical coverage to the south. There were no significant soil geochemical anomalies detected.

10.5.4 Termite Mound Sampling

A total of 107 termite mound samples (including four duplicates) and 8 auger samples of the mounds were collected over an approximate area of 55 km^2. The termite sampling provided some additional geochemical information but was limited to areas with mounds.

10.5.5 Soil pH Geochemical Sampling

Soil geochemical surveys that collected pH signatures were conducted over the known Bisha Gossan to test the theory that a pH measurement can identify the change in pH related to the presence of massive sulphides (and related generation of acid conditions related to oxidation of the sulphides) even below alluvial cover. Nevsun found that the pH technique works very well in the delineation of known sulphide mineralization in alluvial-covered areas and considered that it may be used to define new targets. Unfortunately, the survey reacts to a wide variety of types of underlying chemical differences and thus produces a large number of anomalies that need to be prioritized.

10.5.6 Auger Geochemical Sampling

A total of 39 samples (23 soil samples and 16 auger samples) were collected during 2003 in order to test the hand auger as a geochemical sampling tool. Auger sampling was not determined to be advantageous and therefore was not continued.

10.6 Trenching and Pitting

In 2003, a series of 36 trenches were excavated over various parts of the Bisha Main and Northwest Zones. Rock samples returned elevated base metal and precious metal results.

In 2005, five trenches (166.5 m) were excavated at the Harena prospect, nine trenches were excavated in the Hanging Wall Copper Zone and the Northwest Zone, and 14 pits were excavated in the Hanging Wall Copper Zone. The trenches in the Hanging Wall Copper Zone intersected significant intervals of copper mineralization with rock samples returning grades ranging from 0.27% to 2.68% Cu and 0.3% to 1.33% Zn.

In 2008, Targets 4 and 9, Bisha South, and NW Barite Hill were subject to trenching and pitting, with ten pits and six trenches completed. The trenches at Targets 4 and 9 intersected mudstones and shales with varying amounts of graphite. The Bisha South trenches did not intersect any significant mineralization. At NW Barite Hill, the trenches intersected chlorite–sericite-altered schist with anomalous lead values up to a maximum of 0.23% Pb.

The trench data were not used in geological modelling or resource estimation.

10.7 Drilling

Drill programs are discussed in Section 11 of this Report.

10.8 Bulk Density

Bulk density data collected to date on the Project are discussed in Section 12 of this Report.

10.9 Geotechnical and Hydrology

Geotechnical and hydrological programs conducted were sufficient to support feasibility-level studies, and are ongoing (see Section 18).

10.10 Petrology, Mineralogy and Other Research Studies

Petrographic studies have been undertaken on mineralization samples, including thin and polished sections during 2003–2004. Thin section samples were collected from different types of lithology that Nevsun personnel had problems identifying during drill core logging. Two polished section samples were prepared from the primary massive sulphide lens (Bisha South), representing the zinc-rich zone and the copper-rich zone. The results of the study generally confirmed the lithological types logged by Nevsun personnel.

Over 500 whole-rock analyses were completed during 2003 and 2004 (Daoud, 2004). Winchester-and-Floyd trace-element diagrams show that the mineralization host rocks have a mafic to intermediate affinity (basaltic to dacitic chemistry). Alkali–silica diagrams (Lebas et al., 1986) show that the Bisha volcanic rocks vary from picrites (highly enriched in MgO) to rhyolites; however, the diagrams are affected by alteration, and therefore rocks plotting in the rhyolite field are probably silica-altered (enriched) intermediate rocks.

In 2005, 18 whole-rock analyses were completed on samples from the Harena prospect. The results of the study show that there are two populations of basaltic (mafic) and rhyolitic (felsic) rocks.

In April 2010, Brett Atkinson, a student at Carleton University, Canada, completed a thesis entitled “A Petrographic and Microprobe Study of Oxide Gold-Silver Mineralization and Gangue at Bisha, Eritrea.” The results of the study show that the Bisha oxide zone contains very pure native gold with some silver minerals (naumanite, acanthite and chloroargyrite). These ore minerals are hosted by a matrix of colloform goethite and hematite, quartz, traces of other silicates, barite, trace anglesite and jarosite (mainly plumbojarosite with trace natrojarosite).

10.11 Exploration Potential

The Project remains prospective for VMS mineralization in addition to the deposit and prospects discussed in Section 7 of this Report. Nevsun has identified gravity anomalies with coincident airborne EM anomalies southwest of Harena as well as areas of interest along the stratigraphic horizon between Harena and Bisha Main.

10.12 Comment on Section 10

The exploration programs completed to date are appropriate to the style of the VMS deposits and prospects within the Project. The research work supports the genetic and affinity interpretations.

11.0 DRILLING

Drilling on the Project has been undertaken in a number of core and one RC campaign from 2002 to 2011, as summarized in Table 11-1 and shown in Figure 11-1. Drilling comprised a total of 578 drill holes (78,880.96 m), of which 545 were core drill holes (76,783.66 m) and 33 were RC drill holes (2097.3 m).

Drill programs have been completed primarily by contract drill crew, supervised by Nevsun geological staff.

11.1 Drilling Contractors and Equipment

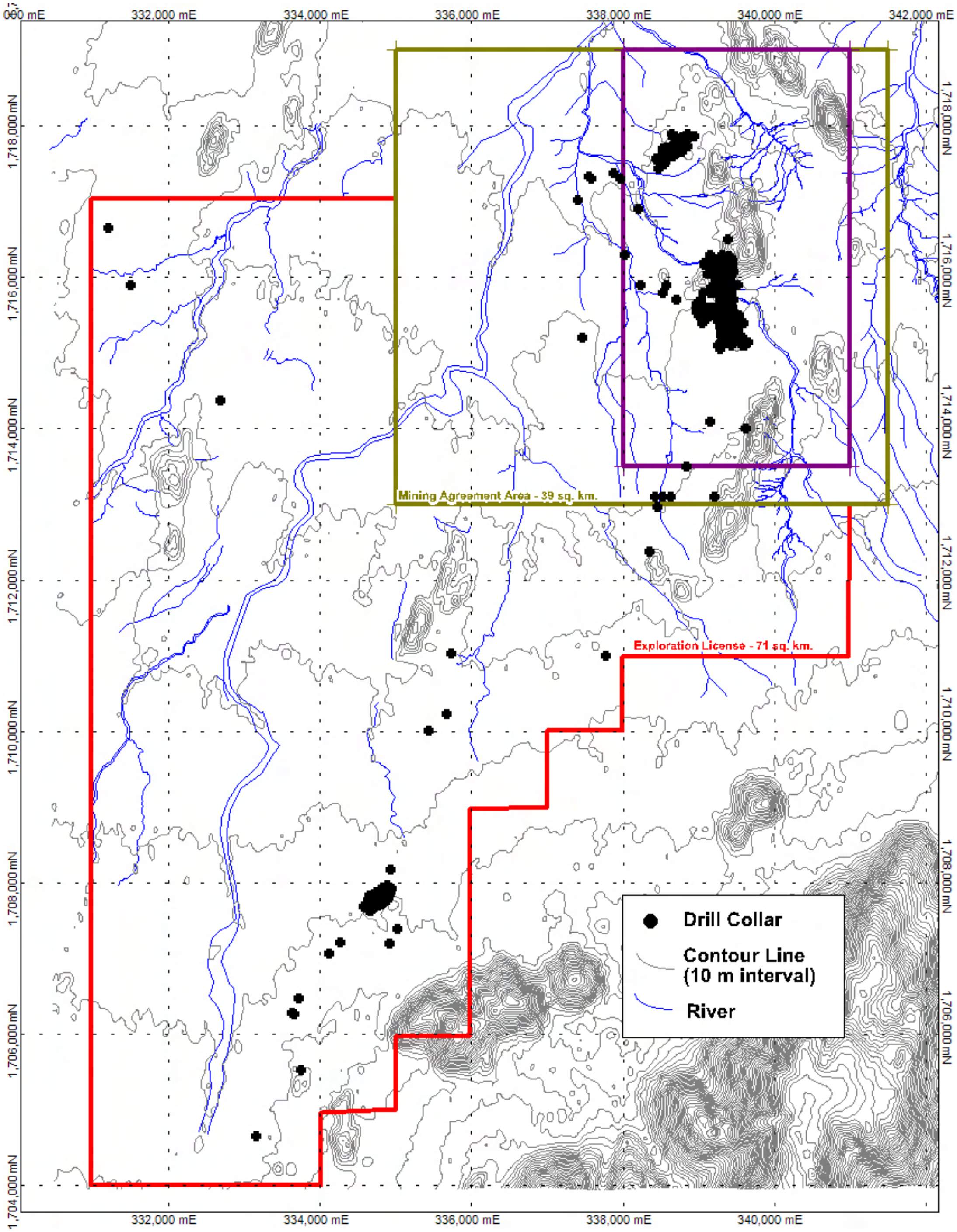
Contractors used on the Project include:

- Kluane International Drilling, a contractor based in Vancouver, BC, Canada. (2002–2003). Kluane used a “man-portable” drill rig. The unit uses a 1.51 m (5') long NTW core barrel (55.1 mm diameter core), which was reduced in bad ground to a BTW sized (41.0 mm diameter core) 1.52 m or 3.04 m (10') long core barrel
- Eritrean Core and Water Well Drilling, a local Asmara contractor, completed water bore drilling in 2003. The equipment included an Atlas Copco Aquadrill R5C and separate XRHS 385 compressor with a working pressure of 16 bars. Each hole was drilled with an 20.3 cm (8") hammer bit and lined with 15.2 (6") plastic perforated pipe. The space between bore and casing was filled with -0.5 cm size screened gravel
- Boart Longyear, a contractor based in North Bay, Ontario, Canada was used from 2003 and are still being used in 2011. Boart Longyear used two Longyear 44 skid-mounted wire-line rigs. Each hole was collared with HQ core (63.5 mm diameter) until ground conditions necessitated a reduction to NQ sized core (47.6 mm diameter). Not all holes were reduced if the ground conditions permitted reasonable penetration or if ground conditions were not favourable for reduction to a smaller core diameter (for example, if the ground was badly fractured). In 2005 holes were collared with PQ (85.0mm) then reduced to HQ core in order to improve recovery in the oxide zone at Bisha and Harena. In 2010 it was determined that although this method improved recoveries at Bisha, it was not successful at Harena so drilling returned to starting with HQ collars reducing to NQ.

Table 11-1: Drill Hole Summary Table

Year	Phase	Range of Hole #	# of DDH Holes	Length of DDH (m)	# of RC Holes	Length of RC (m)	Total # of Holes	Total Length (m)
2002	-	B-001 – 6	6	810.90	-	-	6	810.90
2003	I	B-002a, 7 – 53	48	6,724.76	-	-	48	6,724.76
2003	II	B-054 to 146	93	11,894.50	-	-	93	11,894.50
2004	-	B-147 – 309	163	28,879.50	-	-	163	28,879.50
2004	-	BRC-001 – 40*	-	-	33	1,814.40	33	1,814.40
2004	-	BRCD-026 – 42*	9	308.80	-	282.90	9	591.70
2005	I	B-310 to 367, GT-01 to 05, 04A, H-001 to 020, MET-05-01 to 04, NW-001 to 022, extend B-158	109	15,867.5	-	-	109	15,867.50
2005	II	MET-05-05 to 08, BH01 to 15, H-021 to 027 & H-021 extension	26	2,185.5	-	-	26	2,185.50
2006	-	B-368 to B-371, B-368b, NW-023 to NW-026	9	1,680	-	-	9	1,680.00
2009	-	GT-06 to GT-14, H-028 to H-044, MET-09 to MET-11	29	3,587.5	-	-	29	3,587.50
2010	I	B-372 to B-378, H-045 to H-047, NWB-001 to NWB-003, MET-12 to MET-17	19	2396			19	2396
2010	II	H-048 to H-081	34	2,448.7			34	2,448.70
2011	I	B-379 to B-399	20	2,590.9			20	2,590.9
Total			**545**	**79,374.56**	**33**	**2097.3**	**578**	**81,471.86**

Figure 11-1: Project Drill Hole Location Map



- Major Pontil Pty Ltd., an Australian subsidiary of Major Drilling Inc., based in Queensland, Australia performed the 2004 RC drilling. The company used a universal drill rig (UDR-650-P35 combination drill) with a centre-sample return, triple-wall system to drill holes with a diameter of 136 mm. Major Pontil also undertook water bore drilling during 2004.

11.2 RC Drilling

A total of 2,097.3 m of RC drilling in 42 drill holes (BRC-01 to BRCD-042) was completed during 2004; 282.9 m of this was RC pre-collars completed on nine of the drill holes.

Sample discharge and sample splitting equipment consisted of cyclone collectors mounted above Jones splitters for both wet and dry drilling. RC samples are stored in plastic chip trays for geological logging. Each chip tray represented about 2 m of drilling. Recovery data are not recorded for RC drilling. The sample volume, weight and split used was recorded for each sample, both at the drill (wet sample) and later in camp (dry sample) in an effort to determine sample loss.

Chips are logged by project geologists or geological contractors. The chip trays are labelled and stored in a locked storage container located at the Nevsun exploration camp. Logging was performed using standardized geological logging codes with data recorded on hardcopy logging forms that were later transferred to electronic format.

Digital back-up copies of the geologic logs are stored at BMSC's site office, BMSC's Asmara office, Nevsun's head office in Vancouver and Nevsun's off-site data storage in Vancouver. All hardcopy logs are archived in files, labeled, and stored at BMSC's site office.

A discussion of the collar and downhole survey methods and hole marking techniques used in the Project is included with the core drilling program, in Section 11.3.

11.3 Core Drilling

Core drilling, performed between 2002 and 2011 was undertaken to provide geological, mineralogical, metallurgical, hydrological and geotechnical information on the VMS deposits.

Core barrels were retrieved by wire line. Upon retrieval, the split tube was opened by the driller's helper, who transfers the core into a galvanized steel core box. The core was marked where it was manually broken to fit into the box. Drilled intervals were marked with wooden or plastic blocks. Core recovery and rock quality description (RQD) were measured at the drill rig as the core was placed in the core boxes.

The core was taken to the Bisha exploration camp for logging. The core logging and storage facility includes a large covered area for logging, handling, splitting and storing of the core within the camp perimeters. The core was logged by a geologist for geological and geotechnical elements, which captured lithological, alteration, mineralization, structural and geotechnical information.

The core was laid out for digital photography and then removed to the core storage area if no samples are marked or if samples are marked, the core was sent to the core-cutting area.

The geologist determines the sampling intervals and adheres to lithological intervals. The sample intervals were identified by waterproof tyvek tags indicating a sample interval and lumber crayon marks on the core for the beginning and end of each sample. The geologist also marked the cut line for the core cutters to follow if there was potential for an apparent bias in mineralization.

11.3.1 Drill Surveys

The drill collars were surveyed using a Trimble Pro-Mark 2 global positioning system (GPS) instrument consisting of a base station and rover unit with a radio link. The GPS unit is capable of sub-meter accuracy. Nevsun placed a drill rod within cement at the collar of each drill hole to identify the hole location for all programs. The drill hole number was marked in the cement base.

The first 18 drill holes completed at the Project have no down hole surveys. Later drill holes were down hole surveyed using acid tests, Sperry-Sun Single-Shot and Reflex instrumentation. Typically, measurements were taken at an initial 20 m depth down the drill holes and subsequently every 50 m thereafter unless hole conditions dictated otherwise.

11.3.2 Recovery

No recovery measurements were collected during the first drilling program (2002) or until hole B-13 at the beginning of Phase I drilling in 2003. Since the 2003 program, recoveries were measured for all core holes.

Given the nature of the Bisha mineralization, there are significant differences in recovery for the rock types due to the changes in lithology, alteration, and rock hardness. These factors result in poorer recovery for the near-surface mineralization of the oxide material and excellent recovery in the competent supergene and primary massive sulphide mineralization. High core loss was also common at the abrupt change from hard and relatively competent oxide material to the extremely soft "soap" lithological unit.

Average recoveries are 71% in the oxides, 71% in the breccia, 61% in the "soap" lithological unit, 91% in the supergene domain, and 98% in the primary sulphides.

During the 2006 Feasibility Study, AMEC noted that there was a negative correlation between gold grade and core recovery (Reddy et al., 2006). Conversely, there is no relationship between copper or zinc grades and core recovery. As most of the low-recovery assays were associated with the gold-rich oxidized portion of the deposit, a decision was made to remove all of the assays with core recoveries of less than 60% from the database.

11.4 Grade Control Drilling

Grade control drilling is performed using RC by Atlas Copco ROC L8 drill rigs. Drill holes are drilled on a staggered 8 m x 5 m pattern, and are inclined at an angle of 60º towards the east to define ore mineralization for a 10 m mining bench height. The sample length is usually 1 m and is monitored with a gauge in the drill cabin by the driller.

11.5 Geotechnical and Hydrological Drilling

Twenty-one drill holes have been completed for water wells, and to provide information on groundwater levels and flow data for the Project.

11.6 Drilling Used to Support Mineral Resource Estimation

A total of 356 drill holes support estimation at the Bisha Main Zone. Much of the massive sulphide mineralization in the Bisha Main Zone has been well defined by drilling patterns of 25 m spaced holes on sections spaced 12.5 or 25 m apart. This density decreases with depth on the deepest portions of the primary mineralization. The deposit remains open at depth in the south, with the deepest intersections obtained to date returning long lengths of medium- to high-grade zinc mineralization.

11.7 Comment on Section 11

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar, and downhole survey data collected in the exploration and delineation RC and core drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation for the Bisha Main Zone:

- Drill hole orientations are appropriate to the orientation of the mineralization. A cross section that demonstrated typical drill hole orientations in relation to the mineralization grade shells was included in Section 9
- Drilling is normally perpendicular to the strike of the mineralization, but depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths are typically greater than true widths. Examples of drill hole traces are included in the cross-sections in Section 9
- Drill hole intercepts adequately reflect the nature of the gold and base metal mineralization (Section 7 and Section 9). Figures included in Section 9 that display downhole composite data indicate areas of higher-grade and lower-grade mineralization, and waste material within the deposits;
- Figures included in Section 9 of the Report display sample composite intervals. The deposit has been well drilled, with approximately 545 drill holes completed at the date of the Report and 356 drill holes supporting Mineral Resource and Mineral Reserve estimation. Through interpretation and aggregation of the drill hole data, the sections provide a representative estimation of the true thickness of the mineralization for the deposit in relation to conceptual pit boundaries that are used to constrain the Mineral Resources and Mineral Reserves;
- Core logging meets industry standards for gold and base metal exploration;
- Geotechnical logging meets industry standards for planned open pit operations;
- Collar surveys have been performed using industry-standard instrumentation;
- Downhole surveys have been performed using industry-standard instrumentation.

No factors were identified with the data collection from the drill programs that could affect mineral resource or mineral reserve estimation.

RC drilling was noted by AMEC in 2006 to contain downhole contamination and RC drill holes were removed from the dataset used to estimate mineral resources and mineral reserves. The RC drill hole data are less than 3.5% of the total data, and therefore have no significant impact on the amount of data used to estimate mineral resources (see Section 14).

Samples with core recoveries below 60% in the Bisha Main Zone were noted by AMEC as having the potential to incorporate a positive bias in estimation of gold grades and those samples were therefore removed from the data prior to the calculation of composites.

12.0 SAMPLING METHOD AND APPROACH

12.1.1 Geochemical Sampling

Sample collection procedures for the geochemical sampling comprised:

- The rock chip sample database lacks documentation of the type of sample for most of the samples (i.e., whether the samples were channel chip, grab, or float samples)
- During 1998, stream sediment samples were sieved using a –28 mesh size, and an unknown quantity of sample was shipped for analysis. In 2003, the samples were collected in pits across the active bed of the stream and sieved at the sampling site at approximately 1 mm size fraction. Approximately 25 kg of composite sample was then put in a rice bag
- Soil samples were typically collected approximately 10 cm below surface regardless of the material type at the target depth. The samples were screened using a –60 mesh and 100 to 200 g of sample was placed in a pulp or kraft sample bag and labelled with the grid coordinates
- Termite mound samples were collected in the upper part of the mound (the more recent material deposited). Approximately 8 kg of sample was then put in a bag
- Approximately 2 to 3 kg of sample material was collected at the bottom of the auger holes
- Two sets of pH survey samples were collected from the same sample location, one near surface at less than 10 cm depth and the other at approximately 25 cm depth. After evaluation of the results, Nevsun completed the remaining sampling on the other five lines at a depth of 10 cm. The samples were collected using a hoe or shovel tool and sieved with a –28 mesh. Approximately 100 g to 200 g of sample was collected in a kraft sample bag.

12.2 Trench Sampling

The trenches were excavated to a depth of 0.5 m to 3.5 m depending on the difficulty of excavation or breaking the rock. A total of 707 samples were collected from a total of 1,402 m of excavated trenches. The trenches were mapped and then sampled by the same geologist. Channel samples were taken at 2.0 m intervals and respected lithological contacts.

12.3 RC Sampling

RC samples were 2 m in length. While drilling, the sample that passed through a conventional cyclone, was collected in pails and then passed through a riffle splitter (two-stage SP-2 Porta Splitter). Approximately 10% of the original sample (2 kg) was obtained after the riffle splitting. The remaining sample was discarded. At the end of each drill shift the samples were transported to Nevsun's camp and deposited at the sample laboratory on the gravel pad. The samples were sorted and dried in the sun in a secure area before placing in the preparation laboratory oven. The Nevsun-operated preparation laboratory took control of the samples prior to shipping.

12.4 Core Sampling

The maximum core sample length is 12.00 m (only within wall rock away from mineralized intervals) and the minimum is 0.15 m. Within the zones of mineralization, samples lengths are generally between 1.00 and 3.00 m. Sample intervals are determined based upon mineralogical and lithological contacts.

Standard diamond cutting blades flushed with water are used to halve the core. Highly broken core pieces are cut along the axis if possible or the core is split using a trowel down the middle of the tray row and hand picked or scooped to ensure representative samples are obtained. Cutting lines are not drawn on the core because of the massive nature of the mineralization. Generally the mineralization is lacking any significant banding or veining. The remaining half core is returned to the core storage area and stacked in the numerical order of the core box numbers.

The core splitters place half of the core in double-lined plastic bags with the sample tag placed inside of the bag and the sample number labelled on the outside of the bag. The open bags are placed outside of the laboratory in a secure area on a gravel pad to dry in the sun. For samples from holes B-54 onwards, the preparation laboratory operated by Nevsun took control of sample preparation prior to shipping of the samples.

From 2009 onward half core samples were sent to the Horn of Africa laboratory in Asmara, operated by Genalysis Laboratory Services for sample preparation prior to shipping of the samples.

12.5 Grade Control Sampling

One-metre samples weighing approximately 6 kg are split in-pit and submitted to the on-site laboratory for sample preparation.

12.6 Quality Assurance and Quality Control

Current QA/QC procedures are discussed in Section 13 of this Report.

12.7 Density Determinations

A total of 1,991 bulk density determinations (also referred to as specific gravity, or SG determinations), were performed on drill core by Nevsun and at ALS Chemex in Vancouver.

Data was collected from a representative suite of all rock types, mineralization types and grade ranges.

The bulk density determinations by Nevsun in the Phase I 2003 drill program were made on air-dried, unsealed core using a beam microbalance and a standard immersion-displacement technique, from which the bulk densities were calculated. From the 260 samples collected at site, 44 of the same samples were also sent to ALS Chemex, Vancouver, Canada for an independent measurement and comparison of the results.

The method in use by ALS Chemex was to oven dry the samples, coat in paraffin wax, weigh the sample in air and in water to determine the bulk density value. The comparison of the bulk density values resulted in procedural changes. Principally, the method used small pieces of core and was considered to be prone to over-estimation of the density values.

Nevsun subsequently improved the quality of the bulk density measurements by using larger samples and purchasing more accurate equipment.

The bulk density measurements collected at site during the Phase II 2003 drill program were made on oven-dried, unsealed core using an electronic balance and standard water immersion displacement technique. A total of 611 samples were collected from site during this program, of which 68 were sent to ALS Chemex for independent measurements and subsequent comparison with the Nevsun measurements.

This indicated that there were significant differences in density values with the porous rock types from the oxide domains, such as "soap" and gossan. Nevsun therefore made further changes to equipment and procedures for the Phase I 2004 drill program. The changes included the purchase of a bulk density scale and coating the samples with paraffin wax. During the 2004 program, Nevsun measured 311 samples and submitted 697 samples to ALS Chemex, of which 35 were from the original 311 samples measured at site as an independent check. The measurements collected at site and at ALS Chemex compared well.

In 2004, a suite of 40 samples was sent to Process Research Associates (PRA) for dry bulk density measurements using both wax and non-wax immersion methods to examine the difference between the measurements. The moisture content was also measured for these samples.

12.8 Comment on Section 12

A description of the geology and mineralization of the deposit, which includes rock types, geological controls, and widths of mineralized zones is given in Section 7 and Section 9.

A description of the sampling methods, location, type, nature, and spacing of samples is included in Section 10 and Section 12.

A description of the drilling programs, including sampling and recovery factors, are included in Section 11 and Section 12. Section 11 presents a drill hole location map that shows the size of the sampled areas for the Bisha Main Zone deposit.

The drill hole location plan included in Section 11 show that there is sufficient sampling to adequately represent the tenor and thickness of the mineralization, and that the drill spacing is representative of the distribution and orientation of the mineralization.

Bias issues were identified with the 2003–2004 RC drill program. However, this material constitutes about 3.5% of the total database supporting mineral resources, and in AMEC's opinion did not have a significant impact on the estimation.

Data validation of the drilling and sampling program is discussed in Section 14, and includes review of drill hole orientation, potential sampling biases and database audit results. Drill sample representivity, widths and grades are validated by AMEC's independent sampling and data verification programs which are discussed in Section 14.

The QPs consider that the sampling methods to be acceptable, meet industry-standard practice, and are adequate for Mineral Resource and Mineral Reserve estimation as follows:

- Data are collected following site-approved sampling protocols
- Sampling has been performed in accordance with industry standard practices
- Sample intervals were 2 m for RC and on average were 1.5 m for core drilling, broken at lithological and mineralization changes in the core, and are considered to be adequately representative of the true thicknesses of mineralization
- Grade control drill intervals are 1 m, which is considered appropriate for grade control purposes
- Submission of QA/QC samples is routine, and QA/QC is appropriately monitored
- The bulk density determination procedures are consistent with industry-standard procedures. Nevsun-determined density estimates were confirmed by an independent laboratory re-measuring the same core
- There are sufficient density determinations to support the density values utilized in waste and mineralization tonnage interpolations.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Nevsun staff have been involved with, or responsible for, throughout the duration of the Project to 2009, the following:

- Sample collection
- Core splitting
- Sample preparation of geochemical, pit, trench, RC, core, and grade control samples
- Delivery of samples to the analytical laboratory
- Specific gravity determinations
- Sample storage
- Sample security.

Sample analytical procedures that support Mineral Resource and Mineral Reserve were performed by independent analytical laboratories without company involvement from 1998 to the present. The run-of-mine laboratory was established by SGS Mineral Services, who trained BMSC staff as operators.

13.1 Analytical Laboratories

Exploration work during 1998 and 1999 used the Intertek Testing Services Bondar Clegg Laboratory (ITS Bondar Clegg), based in Asmara. There is little documentation provided in the reports reviewed by AMEC that documents the sample preparation and the analytical protocols from the earlier work.

All trench, rock chip and geochemical samples, including soil and auger, stream sediment, pit and termite mound samples collected during the 2003 Phase I program were shipped to the Horn of Africa Preparation Laboratory, in Asmara, which provided provides preparation services for Genalysis Laboratory Services Pty (Genalysis) of Perth, Australia. The preparation laboratory produced pulp samples that were subsequently shipped to Genalysis for analysis. Following the 2003 Phase I program, geochemical and rock chip samples were shipped to ALS Chemex Ltd. (ALS Chemex), in Vancouver, Canada.

The primary laboratory used by Nevsun for analytical work on the drilling programs was ALS Chemex. Nevsun used ALS Chemex for both sample preparation and analyses since the initiation of the first drill program in 2002. During the 2002 and

Phase I of the 2003 drilling program, samples were shipped as half-core from the Bisha camp to Asmara and forwarded to ALS Chemex in Vancouver via Lufthansa Airlines. After establishing a sample preparation facility[3] at camp in September 2003, Nevsun sent coarse crushed and split material (–2 mm) for core, RC, and rock samples to ALS Chemex for subsequent pulverization and analyses. All assay data contained in the database that supports Mineral Resource estimation was assayed by ALS Chemex.

Check analyses on pulp duplicate materials were performed by ACME Laboratory, in Vancouver, Canada.

In 2009 due to deterioration of the on-site prep laboratory, Nevsun had the core crushed at the Horn of Africa prep laboratory run by Genalysis. Samples were then shipped via courier to ALS Chemex in Romania for gold analysis. The Romania laboratory then forwarded the samples to ALS Chemex in Vancouver for multi-element analysis including base metals.

ALS Chemex, Genalysis, and ACME are ISO-registered and are internationally recognized analytical facilities that are independent of Nevsun and BMSC. The ITS Bondar Clegg Laboratory, based in Asmara is no longer in existence. Internationally, ALS Chemex took over Bondar Clegg in December 2001.

13.2 Sample Preparation

13.2.1 Geochemical Samples

The following sample preparation methods were used for geochemical samples:

Stream sediment samples were sieved in the field to –28 and –80 mesh sizes for the period 1998–1999 and 2003, and to –60 mesh during 2002–2003. Typically for samples taken from 2003 forward, a riffle splitter quartered the samples, the quarter retained was sieved at –80 mesh (–180 µm), pulverized (the specification for final pulverization was > 90% of the sample must be less than 200 mesh or 75 µm), and a 150 g split of the –80 mesh size fraction analysed for Au, platinum group elements, and a multi-element suite.

All of the soil samples were sieved at –80 mesh (–180 µm), pulverized, and a 150 g split of the –80 mesh size fraction analysed for Au, and a multi-element suite.

3 The sample preparation facility was designed and assembled by ALS Chemex for Nevsun.

All of the termite mound samples were sieved at –80 mesh (–180 µm), pulverized, and a 150 g split of the –80 mesh size fraction analysed for Au, and a multi-element suite.

Trench samples were sieved and then followed one of two sample preparation methods. One set of samples (identified as A and B samples), were sieved at –80 mesh (–180 µm), pulverized, and a 150 g split of the –80 mesh size fraction analysed for Au, and a multi-element suite. Samples labelled as "C" samples were crushed (> 75% of the sample must pass 10 mesh or 2 mm screen). A crushed split of approximately 2 kg was derived from the crushing process using a riffle splitter. The rest of the sample was discarded. The crushed portion was pulverized (the specification for final pulverizing is that >90% of the sample must be less than 200 µm), and a 150 g split analysed for Au, and a multi-element suite.

13.2.2 Horn of Africa Preparation Laboratory

Rock chip and trench samples processed at the Horn of Africa Preparation Laboratory followed the following procedures:

- Samples sorted and ordered numerically after receipt
- Placed in a drying oven for 12 to 18 hours at between 80°C and 100°C
- Samples passed through a jaw crusher to > 75% of the sample passing 10 mesh or 2 mm screen
- Sample split using a riffle style splitter to a sub-sample size of between 200 g to 250 g
- Sub-sample pulverized with ring and puck pulverizer to >85% of the sample passing 75 µm
- Samples were shipped to Genalysis. Samples collected during the 2007 to 2009 period were shipped to ALS Chemex.

13.2.3 Core and RC Samples

ALS Chemex

Rock and core samples sent to ALS Chemex prior to the implementation of the on-site sample preparation facility were prepared in the ALS Chemex Vancouver preparation facility.

The samples were dried at 110°C to 120°C for 10 hours to 12 hours, and then crushed with either an oscillating jaw crusher or a roll crusher. The ALS Chemex quality control specifications for crushed material require that > 70% of the sample must pass a 2 mm (10 mesh) screen.

The entire sample was crushed and typically 250 g was subdivided from the main sample by using a riffle splitter and carried through to the pulverizing stage. Generally ALS Chemex retains a 1 kg to 2 kg split of the reject in storage.

The 250 g split is pulverized using a ring mill. The ALS Chemex quality control specifications require that final pulverizing is >85% of the sample must pass 75 µm (200 mesh).

Nevsun Sample Preparation Laboratory

Nevsun purchased a fully equipped, containerized sample preparation laboratory from ALS Chemex in July 2003. The laboratory is controlled, operated, and monitored by Nevsun staff and workers. The personnel working in the laboratory are typically university educated and have prior experience working in laboratory conditions.

Sample preparation procedures were similar for grade control, core, and RC samples, and included:

- Samples are received in plastic sample bags. The sample bags are laid out in numerical order with the plastic bags open at the top to aid in drying
- Samples are crushed using T.M. Rhino Jaw crushers
- Sample splitting is carried out in a Jones-type riffle splitter; samples are split down to a sub-sample weight of approximately 200 g to 300 g.
- The Nevsun QA/QC procedures require the insertion of one blank, two preparation laboratory pulp duplicates (two splits of a crushed sample), and one quartered core duplicate.

Groups of approximately 20 RC or core samples are packed in large plastic bags that are then placed into plastic shipping barrels. When samples are ready to be shipped the sample lists are combined with an ALS Chemex sample submission form and enclosed with the samples in the plastic drums. Samples are shipped approximately weekly.

The sample information with required analytical procedures is emailed to ALS Chemex in Vancouver so that the sample shipments can be tracked and the Vancouver laboratory is made aware of the pending arrival of the samples.

Grade control samples are submitted to the run-of-mine laboratory on site.

13.3 Sample Analysis

13.3.1 Genalysis Laboratory Services

All samples analyzed for gold at the Genalysis Laboratory by 50 g fire assay standard fusion method (Au by solvent extraction and flame AAS) with a 1 ppb detection limit[4].

All samples analyzed for a 25 multi-element suite analysis used a 1 g aqua-regia digestion, followed by inductively coupled plasma (ICP) optical emission spectroscopy (OES) analyses. The multi-element suite (with detection limits in parentheses) included: Ag (0.5 ppm), Al (20 ppm), As (2 ppm), Ba (2 ppm), Bi (2 ppm), Ca (0.01%), Cd (0.5 ppm), Co (1 ppm), Cr (2 ppm), Cu (1 ppm), Fe (0.01%), K (20 ppm), Mg (0.01%), Mn (1 ppm), Mo (2 ppm), Ni (1 ppm) P (20 ppm), Pb (2 ppm), S (10 ppm), Sb (10 ppm), Sc (1 ppm), Te (5 ppm), Ti (5 ppm), V (2 ppm) and Zn (1 ppm). The aqua-regia acid digestion is "total" for most base metals but is only "partial" for some of the major and minor elements.

A series of 25 stream sediment samples collected during the 2003 Phase I work by Mercier were also analyzed for platinum group elements. The method used 25 g fire assay nickel sulphide collection followed by ICP mass spectroscopy (MS). The nickel sulphide button was pulverized and sample is digested with hydrochloric acid. The platinum group elements (with detection limits in parentheses) included: Ru (2 ppb), Rh (1 ppb), Pd (2 ppb), Os (2 ppb), Ir (2 ppb) and Pt (2 ppb) (Mercier, 2003).

13.3.2 ALS Chemex

All core and RC samples were sent to ALS Chemex for analyses. All samples were analyzed for gold by a 30 g fire assay fusion (Au AAS23) and determined analytically using an atomic absorption spectroscopy (AAS) finish. Assays that were greater than the detection level (i.e., over limits) of the AAS finish (i.e., greater than 10,000 ppb) were re-assayed by a 30 g fire assay fusion (Au GRA21) and determined analytically using a gravimetric finish.

4 Mercier (2003) states that a detection limit has an uncertainty of ±100%. In other words, a detection limit of 1 ppb implies an uncertainty of 1 ppb ±1 ppb).

Multi-element analyses were completed with 41 elements inductively coupled plasma - atomic emission spectroscopy (ICP-AES) with Nitric-HCl Digestion (ME-ICP41A). This is the method used to determine the copper, zinc, lead, and silver values. Copper, lead, zinc and silver samples that were greater than the detection level (i.e., over limits) of 50,000 ppm were re-assayed with aqua regia digestion and AAS. The 2002 drill core samples used the trace level ICP package (ME-ICP21) and were followed up with AAS for those samples that were over limits.

The few samples that were greater than the 30% detection level of AAS for base metals were assayed by wet assay titrometric methods.

Soil geochemical samples were tested using ICP-MS to achieve ultra-trace detection levels on base metals and minor and major elements while gold determinations were completed with ICP-AES on a fire assay fusion (Au-ICP21) for ultra-trace detection levels.

13.3.3 Grade Control

All grade control samples are fire-assayed, using a 30 g sample.

13.4 Nevsun Quality Assurance/Quality Control Protocols for Drill Programs, 2002–2006

Nevsun implemented quality assurance and quality control (QA/QC) protocols for all exploration from the beginning of work on the Bisha Project, including work in 1998 and 1999. The QA/QC samples used for geochemical sampling have little documentation and no presentation of the results or any corrective actions taken (if required). These samples were for the compilation of the exploration database and not part of the database supporting Mineral Resource estimation.

All of the core and RC drilling programs included certified reference materials (CRMs) and also included blanks, twin sample duplicates, and coarse preparation duplicates. Each drill program report documented the protocols and results of the QA/QC program. Nevsun did not submit pulp duplicates and external check samples during the drilling program. AMEC recommended that approximately 5% of the sample pulps be submitted to a second laboratory as a check on the primary laboratory. Nevsun submitted 656 pulps to ACME laboratory.

Nevsun purchased the CRMs from Geostat Sample and Assay Monitoring Service, located in Australia. The reference material includes a range of low-grade, mid-grade and high-grade precious and base metal standards with certified values and statistically-acceptable limits.

The QA/QC program for the 2002 drilling included 11 insertions of a CRM. Four of the 11 insertions were not within the accepted limits. These CRMs were not used for subsequent sampling programs, after significant mineralization encountered at the Bisha Main Zone required a more substantial and thorough QA/QC program.

The QA/QC sample insertion protocol employed by Nevsun for all core and RC drill sampling subsequent to the 2002 program includes the following samples:

- Six certified CRM control samples per 100 samples; three gold (B, D, and F) and three base metal (A, C, and E)
- One coarse blank sample of barren material per 100 samples; as well as, barren material randomly inserted in mineralized zones
- One quartered core "twin" duplicate sample per 100 samples
- Two coarse preparation duplicates per 100 samples.

During the 2003 Phase I and II and 2004 drilling programs, a total of 1,299 insertions of CRMs were made into the sample sequence of 20,545 core and RC samples. In addition to the CRMs were 352 blanks, 225 twin duplicates, and 372 coarse preparation duplicates. In total the QA/QC samples comprise 11% of the total sample analyses.

In 2005 a total of 837 control samples were inserted within a sequence of 7,845 core samples which made up 10.7% of the total sample population. AMEC reviewed the CRM values during 2006, and concluded that the Cu, Au, Ag, Pb and Zn accuracy at the ALS Chemex laboratory during the 2005 exploration campaign was acceptable.

The coarse blank material was sourced from near the Project. This material usually consists of limestone and/or dolomite, considered to consist of barren rock without any appreciable precious metal or base metal content. After review of the logs, sample batches, and data, AMEC noted that the blank material is not barren, and thus the true values of the blank for each metal are not known. AMEC recommended that Nevsun purchase a commercial blank for use. If the use of a coarse blank material was continued, then AMEC suggested it should be of a clean, barren material with no obvious oxidation surfaces or patches iron oxides such as limonite or hematite.

From the 2005 program, 83 samples of duplicate twin quarter core were submitted for check analysis. AMEC considered the results acceptable for twin samples and that the sampling variance for Cu, Au, Ag, Pb and Zn during the 2005 drilling exploration campaign was satisfactory.

For the 2005 program, a total of 165 coarse preparation duplicates were collected. Results for silver, copper, lead and zinc are all within acceptable limits in AMEC's opinion. Gold had a somewhat lower correlation coefficient indicating a nugget effect for this element.

In total, 656 check pulp duplicate samples were sent for external check to ACME Laboratory. The samples were assayed by ICP for 24 elements, including Au, Ag, Cu, Pb and Zn. On review, AMEC noted a satisfactory level of bias for each metal (greater or equal to –5% or greater or equal to 5%). Both Ag and Pb demonstrate higher variability since the bias for each of these metals worsens or remains the same with the removal of outliers.

Check sample batches included a certain number of control samples: 21 pulp duplicates, 21 CRMs and 42 pulp blanks, to assess analytical precision, accuracy and contamination at ACME. AMEC reviewed the results and considered that the results were within acceptable ranges.

13.5 Nevsun Quality Assurance/Quality Control Protocols for Geochemical Programs, 2006–2009

Soil, pit, and trench geochemical samples were taken in the period 2007–2009, and the programs employed a similar QA/QC method to that documented in Section 13.4. Duplicate samples were taken every 25 samples. The samples were sent to the African Horn Services sample preparation laboratory in Asmara and were subsequently shipped to ALS Chemex in Vancouver for analysis. AMEC did not review the QA/QC data associated with the geochemical samples as they are not material to the disclosure of mineral resources and mineral reserves.

13.6 Nevsun Quality Assurance/Quality Control Protocols for Drill Programs, 2006–2010

All of the drilling programs included certified reference materials (CRMs) and also included blanks, twin sample duplicates, and coarse preparation duplicates.

Nevsun used the same CRMs purchased from Geostat as those used in the 2002-2005 drilling campaigns. The reference material includes a range of low-grade, mid-grade and high-grade precious and base metal standards with certified values showing statistically-acceptable confidence limits.

During the 2006 drill program, a total of 26 insertions of CRMs were made into the sample sequence of 502 core samples. In addition to the CRMs were 10 blanks, three twin duplicates, and 12 coarse preparation duplicates. In total the QA/QC samples comprise 10% of the total sample analyses.

On review, AMEC noted a satisfactory level of bias for each metal (greater or equal to –5% or greater or equal to 5%).

Nevsun did not submit pulp duplicates or external check samples during the drilling program. AMEC reviewed the precision of the CRM samples analysed by ALS Chemex and found the analytical precision values to be satisfactory (within ±10% at the 90th percentile) with the exception of gold. Gold typically does not show high analytical precision in the presence of coarse gold or when the gold grades analysed are close to the analytical detection limit of the fire assay method.

During the 2009 drill program, a total of 50 CRM insertions were made into the sample sequence of 548 samples. In addition to the CRMs were 10 blanks, 12 twin duplicates, and nine coarse duplicate samples. In total the QA/QC samples comprise 15% of the total sample analyses.

AMEC reviewed the results and considers the accuracy of the results to be acceptable. The sub-sampling precision values obtained from the coarse duplicates are unacceptably low, although there are a low number of duplicates. The absolute relative difference precision values are shown in Table 13-1.

AMEC recommends that Nevsun:

- Carefully monitor the sub-sampling precision performance of duplicates prepared at the Horn of Africa Preparation Laboratory
- Send 5% of the samples for check assay at an independent second laboratory
- Insert pulp duplicates into the sample submissions sent to the laboratory for analysis.

13.7 Databases

Entry of information into databases has utilized a variety of techniques and procedures to check the integrity of the data entered. Geological data are entered into Microsoft

Excel® templates. Nevsun perform visual checks of the entered data and check the database for extreme values or codes that are not in the list of accepted codes. Geological data were validated for overlapping intervals by software routines.

Analytical data are uploaded from digital sources.

Survey data uploads are completed by the project geologist from digital survey files. Down-hole surveys are read from Sperry Sun® disks or directly from the survey instrument and were recorded by the driller.

Table 13-1: Coarse Duplicate Analytical Precision, 2009 Drill Program

Metal	90^{th} Percentile ARD Value
Gold (g/t)	± 22.6%
Silver (g/t)	± 18.8%
Copper (%)	± 40.9%
Lead (%)	± 11.7%
Zinc (%)	± 59.8%

13.8 Sample Security

The chain-of-custody for core samples collected and being shipped from site is as follows:

- Core is transported to the Bisha camp by the drill contractors and placed in the core logging area
- Logging and sample preparation area and Bisha camp is a fenced and guarded compound
- Core samples are crushed and sub-sampled
- Prepared samples are placed in sealed barrels
- Each barrel has a list of samples written on the outside of the container
- A sample submission form accompanies each barrel
- Barrels are transported to Asmara in company-owned vehicles arranged by Nevsun.

The sample barrels are submitted to the Eritrean Ministry of Mines for inspection and submission to customs, a customs seal is placed on the barrels and the sample barrels are shipped via air transport directly to ALS Chemex in Vancouver, Canada or via courier air/ground transport to ALS Chemex in Romania

AMEC considers the security and chain-of-custody procedures to be reasonable and acceptable.

13.9 Sample Storage

Retained core character, pulp, and pulp duplicate samples are stored onsite at the Bisha camp facilities.

The 4 kg to 5 kg crushed residues from grade control sampling are stored by drill hole number for one month while pulp is kept in containers at the laboratory for a maximum of three months.

13.10 Comment on Sample Analysis, QA/QC and Security

The QPs believe that the quality of the gold analytical data are sufficiently reliable (also see discussion in Section 14) to support Mineral Resource and Mineral Reserve estimation and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards as follows:

- Sample preparation for samples that support Mineral Resource estimation has followed a similar procedure since 2003. The preparation procedure is in line with industry-standard methods for VMS deposits;
- Drill sampling has been adequately spaced to first define, then infill, base and precious metal anomalies to produce the prospect-scale and deposit-scale drill data. Drill spacings are discussed in Section 11;
- Nevsun has used a QA/QC program comprising blank, standard and duplicate samples since the early 1990s. QA/QC submission rates meet industry-accepted standards of insertion rates;
- Data that were collected were subject to validation, using in-built program triggers that automatically checked data on upload to the database;
- Analytical biases were identified with some drill data; these are discussed in Section 14;
- Verification is performed on all digitally-collected data on upload to the main database, and includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards;

- Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory;
- Current sample storage procedures and storage areas are consistent with industry standards.

14.0 DATA VERIFICATION

14.1 Laboratory Visits

The Nevsun sample preparation laboratory has been visited and inspected by Mr. Doug Reddy, an AMEC employee, during a site visit between 28 May and 1 June 2004.

14.2 External Data Reviews

14.2.1 AMEC, 2004–2005

During the 2004 Bisha site visit AMEC reviewed the available drilling and other exploration and project data. A database with a total of 288 diamond drill holes with a cumulative meterage of 45,216 m was available for review but the collar survey and assay portions of the database were incomplete. A total of 40 RC drill holes were recorded in the database, however collar surveys, assays and other information was incomplete at that time. AMEC reviewed onsite the core and RC databases, location of hole collars to topographic plans, resurvey of six drill hole collars, and a downhole survey review. AMEC also inspected the core logging process, sample preparation and storage facilities.

Additional verification activities included checking high values and relationships between grades and sample lengths. High values for Au, Ag, Cu, Pb, and Zn for each rock type were investigated and checked to confirm that the logged mineralization did concur with the assay results.

Nevsun was advised of all problems or inconsistencies that were noted during the AMEC's review and Nevsun rectified these items. AMEC considered the final database that supported the 2006 Feasibility Study was robust and verified.

A check of QA/QC data included a series of sieve checks on current samples, and also on reject material of samples that were pulled from storage. The current sample preparation was within the accepted protocol of 70% passing 2 mm. AMEC noted that the sample preparation personnel regular check that the crushed material is meeting the protocol.

Sieve checks on sample material that was pulled from storage returned variable results and many samples did not meet the current protocol. A subsequent check of the original assay versus an assay of the reject material showed relatively good agreement (most samples within ±20%) and therefore AMEC did not consider this to be of concern.

As a check on the quality of the data entry, AMEC completed a small double data entry check. Discrepancies were noted between hard copy and digital data, primarily due to a change in lithological and mineralization coding and additional detail of mineralized intervals that was added in the holes selected. The hard copy and digital information should match therefore a revision of the hard copy logs is advisable. AMEC recommended the use of either a double entry system or a data entry system with some form of validation of codes.

AMEC collected a series of 172 samples during the 2004 site visit, which were submitted to ALS Chemex for analysis. AMEC conducted or was present during the collection and preparation of the samples. The samples were placed in a randomized sample sequence and renumbered which would prevent any systematic tampering with the samples. AMEC accompanied the samples from the preparation laboratory to the Ministry of Mines office in Asmara. The sampling results were of the same tenor and nature as Nevsun's analytical results.

AMEC observed drilling in progress during 2004–2005 and was confident of the presence of base metal mineralization and concludes that the samples of quarter core and rejects provide confirmation of the grades and reproducibility of assay values.

Forty-two samples of quartered core were collected from 9 holes from the 2004 drilling program. Comparisons of half-core to quartered core are difficult due to the change in size of sample. However, AMEC considers these samples to show a reasonable reproducibility. Following review of results from 40 samples (two were found to be swapped and excluded from consideration) AMEC was of the opinion that results showed a reasonable reproducibility and provided assurance that the sample homogenization prior to splitting was reasonable.

As a further test of the sample homogenization during the sample preparation, AMEC collected the first and last splits that are normally rejected during the sub-sampling using the Jones splitter. Thirty samples were processed, and although results fell within acceptable limits, AMEC noted that the need for ensuring that the crushing protocols are being met is underscored. The first splits were also compared to the original sample and were found to have similar results for the sample pairs.

14.3 Comment on Section 14

The initial process of data verification for the Project was performed by Nevsun, and by external consultancies contracted by Nevsun staff. During the 2006 Feasibility Study, and as part of checks on data for this, and previous technical reports, AMEC reviewed drilling and other exploration and project data. AMEC also submitted independent samples for verification of mineralization tenor at the Project.

As a result, AMEC considers that a reasonable level of verification has been completed, and that no material issues would have been left unidentified from the programs undertaken.

The QPs, who rely upon this work, have reviewed the appropriate reports, and are of the opinion that the data verification programs undertaken on the data collected from the project adequately support the geologic interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

15.0 ADJACENT PROPERTIES

There are no properties immediately outside the Project that are at the same state of development as the Bisha Project.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Metallurgical Testwork

In 2005, metallurgical testwork was undertaken at SGS Lakefield to support feasibility-level studies for the oxide, supergene and primary mineralization material types. Two separate drilling programs were conducted to collect samples for the SGS metallurgical testwork, the first set to support scoping-level studies (Phase I) and the second set for the 2006 Feasibility Study (Phase II).

The core sampling supergene and primary mineralization were handled and prepared with care to minimize exposure to air following the recovery of the cores from the ground. The oxide mineralization was essentially sulphide free and as the cyanide leach testwork was not unduly affected by sulphide oxidation, the oxide mineralization drill core did not require the same care in handling as the sulphide cores. The cost of collecting these samples separately therefore placed a limit on the number of holes that could be drilled.

In 2010, metallurgical testwork was conducted at Mintek on behalf of SENET to provide plant design and performance expectations of the Bisha supergene mineralization as part of ongoing development of the Bisha project. The test program was conducted in two phases. The phase 1 objective was to replicate test conditions set by SGS Lakefield (SGS) in 2005, and use those conditions to generate a bulk concentrate for marketing purposes. As the results of those tests indicated that the ore sample delivered to Mintek did not behave as the previously tested samples at SGS, it was decided that the concentrate production be put on hold and instead conduct scouting investigations on the reasons for the variable response. For phase 2, the approach was to use a simplified reagent scheme to enhance copper flotation kinetics and this met with limited success. Poor reproducibility resulted in the concentrate generation program being suspended to investigate the possible reasons for the different results.

Following the testwork at Mintek, Maelgwyn Mineral Services Africa (Pty), South Africa (Maelgwyn) was contracted to duplicate the test program previously attempted at Mintek. The objective of the test program at Maelgwyn was to advance the testwork initiated at Mintek and demonstrate that the results could be reproducible at the given conditions. At the time of reporting batch flotation tests on supergene samples have been completed and further testing is on-going.

SENET are currently working in conjunction with Eurus Mineral Consultants (Eurus) to evaluate and optimize the proposed supergene flotation circuit design and expected performance. Eurus uses a proprietary simulation modeling technique using flotation rate kinetic data to construct a flotation circuit model. Both the laboratory and simulation work is ongoing.

16.1.1 Metallurgical Samples

Table 16-1 summarizes the metallurgical sample drill locations for the 2005 and 2010 metallurgical test programs. Figure 16-1 provides a spatial representation of the metallurgical drill holes within the pit outline and shows a reasonable distribution of samples throughout the pit.

Drill holes Met 05-01 to 05-04 were drilled in March 2005 and comprise the Phase I portion of the program, producing five tonnes of sample. The testing program conducted on these samples provided the metallurgical results for the scoping phase of the Bisha Project.

Drill holes Met 05-06 to 08 were drilled in October 2005 and produced four tonnes of sample. These samples were used to complete the Phase II metallurgical testing program to a level suitable for the 2006 Feasibility Study.

Drill holes Met-09 to 11 were drilled in October 2009 and produced 1.7 t of sample. These samples were used to supply samples for the metallurgical testwork at Mintek.

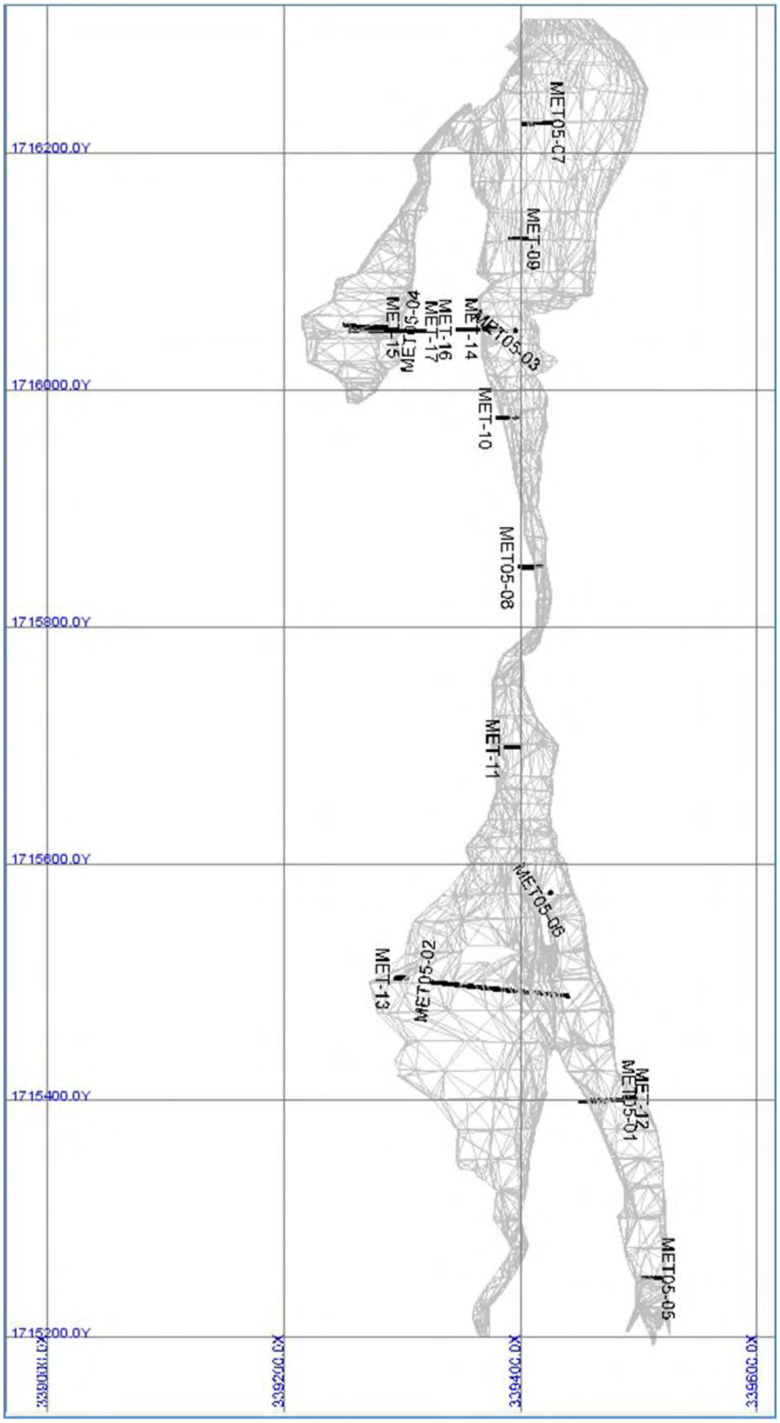
Drill holes Met-12 to 17 were drilled in July 2010. Batch flotation tests on these samples were conducted at Maelgwyn.

A number of intervals logged as ‘transition’ in Met-12, -16 and -17 represents material that is found within the gradational contact zone between the upper layer of supergene and the underlying primary mineralization. No testwork specific to this material was conducted.

Table 16-1: Metallurgical Sample Drill Hole Locations

Drill Hole Tag #	Drill Hole Co-ordinates	Azimuth	Dip	Ore Type	Depth From (m)	Depth To (m)
Met 05-01	1715400N, 339485E	270	-80	Oxide	10.5	36.0
				Supergene	39.0	67.5
				Primary	67.5	202.5
Met 05-02	1715500N, 339325E	90	-65	Primary	100.0	250.0
Met 05-03	1716050N, 339395E	vertical		Oxide	3.0	36.2
				Supergene	36.4	82.5
Met 05-04	1716050N, 339295E	290	-70	Oxide	-	36.0
				Supergene	36.0	84.0
Met 05-05	1715250N, 339520E	270	-80	Supergene	34.5	50.5
				Primary	50.5	115.0
Met 05-06	1715575N, 339425E	90	-90	Oxide	22.0	34.5
				Supergene	34.5	62.5
				Primary	62.5	152.5
Met 05-07	1716225N, 339425E	270	-80	Oxide	31.5	42.0
				Supergene	42.0	68.5
				Primary	68.5	124.3
Met 05-08	1715850N, 339400E	90	-70	Supergene	40.5	58.8
Met-09	1716128N, 339405E	270	-80	Supergene	36.15	84.00
Met-10	1715976N, 339381E	90	-75	Supergene	42.10	67.00
Met-11	1715698N, 339387E	90	-80	Supergene	38.80	68.45
Met-12	1715402N, 339497E	270	-80	Supergene	41.50	43.00
				Supergene	50.50	52.00
				Supergene	53.50	55.00
				Transition	61.00	62.50
Met-13	1715503N, 339294E	90	-80	Supergene	43.00	44.50
				Supergene	49.00	50.50
Met-14	1716052N, 339369E	90	-80	Supergene	38.50	40.00
				Supergene	52.00	53.50
				Supergene	61.00	62.50
Met-15	1716053N, 339286E	270	-70	Supergene	34.00	35.50
				Supergene	40.00	41.50
				Supergene	49.00	50.50
				Supergene	67.00	68.50
Met-16	1716051N, 339347E	90	-70	Transition	86.50	91.00
Met-17	1716050N, 339318E	270	-55	Transition	86.00	90.50

Figure 16-1: Metallurgical Drill Hole Locations within Pit



16.1.2 Composite Samples

For the 2005 SGS Phase I program, one master composite was made up from the variability composite samples for each of the following three main mineralization types:

- Oxide mineralization
- Supergene mineralization
- Primary mineralization.

For the 2005 Phase II program, two primary master composites, typical of primary mineralization, were produced, a “zinc-rich” master composite and a “low-zinc” master composite.

To the extent possible, the master composite samples were made to match the grades in the Mineral Resource estimated for each of the mineralization types.

Seven supergene variability composite samples were produced from the eight metallurgical sample drill holes ranging from a low of 1.82% Cu to a high of 15.9% Cu. From these variability composites, two master composite samples were produced, one for the 2005 SGS Phase I testwork with an assay of 1.93%Cu and the other for the 2005 SGS Phase II testwork assaying at 4.2% Cu.

Fifteen primary mineralization variability composite samples were made up from the eight metallurgical drill holes, seven for the Phase I testwork and eight for the Phase II testwork.

For the 2010 Mintek program a single composite was made up from the bulk of the drill samples sourced from supergene mineralization.

For the 2010 Maelgwyn program, the composite material from the Mintek program was used. In addition, rougher flotation tests were performed on each of the metallurgical drill core samples from Met-12 through 17 and additional rougher kinetic tests are being carried out at Report effective date.

16.1.3 Grinding Testwork

2005 SGS Test Program

Splits were taken from each of the three Phase I master composite samples for JK drop-weight tests, MacPherson autogenous mill tests, Bond rod mill and ball mill work indices and abrasion indices. Because the oxide ore was indicated to be the hardest of the three ore types, the grindability work on the oxide ore was expanded to include rod mill and ball mill Bond work index testing on the oxide ore variability samples and a MacPherson mill test on the oxide ore master composite sample.

In the context of the JK Tech mineralization database, the supergene material is in the very soft range and the oxide and primary ores are in the soft to very soft range of resistance to impact breakage.

As part of the JK Tech sample property assessment, the relative densities of 30 randomly selected particles for each mineralization type in the 26.5 to 31.5 mm size range were determined by weighing each particle in water and in air. The majority of the supergene and primary ore samples had bulk density values within the range of 4.6 to 5.0; however, the oxide ore samples displayed a bi-modal histogram with one set of values bracketing a bulk density of value 2.5 and the other set bracketing a bulk density value of 3.9. It was concluded the ferruginous gossans had higher densities, while the acid oxide, breccias and saprolite had lower densities.

Autogenous work index values were determined in accordance with the procedures used for the standard MacPherson grindability test. The MacPherson procedure compared the gross AWi resulting from the test against the MacPherson database of operating plants to provide a correlated AWi value. AMEC's experience with this conversion is that it is too low therefore AMEC added an additional factor between 1.4 and 1.6 to the correlated AWi. For the oxide ore, this resulted in a SAG mill work index range of 11.8 to 13.4 kWh/mt, which bracketed the 12.7 kWh/mt back-calculated from the JK Simmet semi-autogenous grind (SAG) mill model. As a result, AMEC considered that there was good agreement between the two methods in sizing the SAG mill for the oxide ore. The supergene and primary ores are much softer than the oxide ore therefore neither of these ores will have significant input into mill sizing. Similar comparative calculations were therefore not performed on those composite samples.

Standard Bond rod mill and ball mill work index determinations were conducted on the master composite samples and Bond ball mill work indices were run on the harder oxide, supergene and primary ore variability composite samples. The oxide ore proved to be the hardest of the three ore types, and was used to size the grinding mills for the proposed plant. The lower rod mill work indices for all three mineralization types indicated these mineralization types to be amenable to first stage SAG mill grinding.

2010 Test Program

No grinding testwork was conducted as part of the 2010 Mintek and Maelgwyn test programs.

16.1.4 Mineralogy

2005 SGS Test Program

Oxide Mineralization

A portion of Phase I oxide master composite sample was studied using optical microscope and X-ray diffraction tests for gold deportment to help evaluate potential metallurgical performance.

A portion of the oxide sample was concentrated on a superpanner and separated into three fractions. Results showed that gold in the oxide zone would not be amenable to gravity recovery and good leach extractions would require a comparatively fine grind.

Supergene Mineralization

Four of the Phase II supergene variability composite samples, the Phase II supergene master composite sample and four individually selected supergene core samples were submitted for mineralogical analysis using the method of bulk mineralogical analysis (BMA). The samples were analyzed by QEMSCAN microscope. The major minerals in the supergene mineralization were confirmed as pyrite (72–96%), the secondary copper minerals covellite (1–9%), chalcocite (0.2–3%) and enargite, the primary copper minerals chalcopyrite and bornite and non-sulphide gangue (0.5–7%). Of the total Cu mineralization, 40–80% occurred as secondary minerals and 20–60% occurred as the primary minerals. There was significant sphalerite in two of the samples with small quantities of molybdenite and galena in several samples.

Liberated mineral release curves were constructed for the supergene mineralization, and demonstrated that very high concentrate grades would be possible if the liberated copper minerals could be separated cleanly from the sulphide and non-sulphide gangue.

Primary Mineralization

Seven primary mineralization variability composite samples collected during the Phase II sample gathering program were submitted for QEMSCAN mineralogical study. Chalcopyrite is the most abundant copper mineral in the primary mineralization ranging between 1.5% and 6.0%. The range of sphalerite grade is wide, ranging from 2.4% to 26.6%. Pyrite comprises 65 to 95% of the samples. The samples display a range of liberated primary, secondary and mixed copper. Total liberated copper exceeds 80% in all samples. Pyrite is the major contaminant, being attached to 8–12% of the copper in the sample. Approximately 5% of the copper is locked in all the samples.

Most of the zinc in the composite samples is present as liberated sphalerite (80–94%) and only about 1% is locked. The sphalerite/pyrite association ranges between 5 and 15% and the sphalerite/copper association is 1 to 2%. The zinc mineral associations suggest that the zinc should be readily recovered to high-grade zinc concentrates. The biggest challenge to obtaining good zinc flotation results will be to minimize the recovery of zinc to the copper concentrate.

Limiting grade-recovery curves indicated that the actual separation by flotation, will be negatively affected by the secondary copper minerals activating the pyrite and sphalerite.

2010 Mintek Test Program

Quantitative mineralogical analysis was conducted on the flotation feed sample to compare the 2010 program flotation feed composite to the composites tested in 2005. Although the data output of the two separate testing facilities was not directly comparable, it was determined that the 2010 composite sample was lower in bornite and chalcopyrite with higher contents of covellite and chalcocite as compared to the samples tested in 2005. It was not confirmed by Mintek that the differences in the two sample responses in flotation could solely be as a result of mineralogy.

16.1.5 Cyanidation Testwork

The initial set of cyanidation tests examined order-of-magnitude grind size and cyanide solution strength versus leach extraction. Heap leaching was subsequently dismissed as a viable process due to the low leach extraction of 66% Au at the comparatively fine crush size of 10 mesh.

Reduced cyanide solution strengths indicated reduced leach extractions, the differences being less the longer the leach time was extended. Leach extraction times varied with different test series, resulting in the variables of grind, CN concentration and leach slurry density to be examined in more detail during the Phase II testing.

During Phase II testing, two oxide mineralization samples were subjected to a series of 24 hour leach tests with variations in grind P80 of 60 and 75 µm, variations in leach slurry density of 45 and 50% solids and variations in CN concentration of 0.25 and 0.5 g/L NaCN. The coarser grinds for both composite samples returned the higher leach extractions. In line with expectations, the higher cyanide concentration resulted in higher leach extraction and higher slurry densities resulted in lower leach extractions, although by only a small margin. To rationalize these grind versus extraction results, it was assumed that within the P80 range of 60 to 85 µm, there was no difference in leach extraction. The gold deportment study and grind versus recovery tests in the P80 range of 60 to 85 µm suggested that there was some porosity in the oxide mineralization matrix that allowed the cyanide to penetrate to the locked gold particles.

16.1.6 Flotation Testwork

2005 SGS Test Program

Oxide Mineralization

A single flotation test was conducted on the Phase I oxide mineralization master composite to determine if the gold could be economically recovered into a flotation concentrate. Results showed that the flotation selectivity for gold is poor with rougher concentrate grades ranging from 55 g/t Au to 12 g/t Au for gold recoveries between 44% and 77%. The test concluded that oxide mineralization was not a good candidate for upgrading by flotation.

Supergene Mineralization

Mineralogical studies confirmed the supergene mineralization had a high pyrite content, and that copper mineralization occurred primarily as covellite with minor chalcocite, chalcopyrite and bornite. Covellite and chalcocite are both slightly soluble therefore some copper ions from these minerals will have migrated and attached themselves to the surfaces of the pyrite, resulting in a portion of the pyrite being active to flotation. This phenomenon caused the flotation separation between copper and iron to be very difficult, and pyrite depression was the biggest challenge in evaluating the supergene mineralization flotation processing strategy.

During the Phase I program, over 30 batch flotation and two locked-cycle tests were conducted on the master composite sample. Grade–recovery curves for the supergene tests were essentially straight lines, with the slopes of the lines being mildly dependent on the degree of rougher concentrate regrinding. Conclusions were:

- A grind P80 of 75 µm will provide in excess of 90% Cu rougher recovery
- Little difference was observed in effectiveness between reagents SIPX and PAX collector, therefore PAX was eventually selected as it was the better reagent noted for primary ore flotation
- High lime additions were required in grinding, rougher flotation and cleaner flotation to obtain maximum recovery of copper
- The use of sodium sulphite in the primary grind and in regrinding limited the drop in redox potential and pH during the grinding stages, therefore less lime was required to maintain the pH through the course of the test.

The Phase II supergene master composite had a grade of 4.2% Cu, which was more comparable to the 4.4% Cu grade estimated for Mineral Resources. The higher copper grade as compared to the Phase I sample provided higher copper recoveries. An additional 27 batch tests and four locked-cycle tests were conducted on the Phase II sample, with the following results:

- Finer grinds provide an incremental increase in recovery of copper therefore the grind target was changed from P80 of 75 µm to 55 µm
- Regrinding all of the supergene rougher concentrate provided only marginal improvement in the position of the grade-recovery curve for the subsequent 3-stage cleaning as compared to the roughing only grade-recovery curve.

Roughing kinetics tests were conducted on several supergene variability composite samples. Only a single test was run for each composite, therefore the test conditions were not optimized.

Primary Mineralization

The initial tests confirmed the challenge in primary mineralization flotation to be the separation of a copper concentrate with minimum zinc and iron. Zinc flotation was relatively easy with recoveries of 95% or more of the remaining zinc to rougher concentrates which upgraded to 55% Zn grade in two cleaning stages. The majority of the work on the primary mineralization was therefore focused on copper flotation.

The initial Phase I flotation tests on the primary mineralization indicated copper rougher recoveries up to 90% are achievable with a grind P80 of 75 µm. Similar to the supergene mineralization, high levels of lime were required. Because of sample deterioration, the optimum copper flotation test conditions were not considered to be established during the Phase I program. One of the primary objectives of the Phase II test program was therefore to determine the conditions that would provide the optimum copper and zinc results from the primary mineralization.

Only a few zinc concentrate regrinding and cleaning tests were conducted during the Phase I program, but those tests generally produced good results.

The copper in the Phase II zinc-rich master composite sample was found to have a higher degree of dissemination than the copper in the Phase I sample and a grind of P80 of 55 µm was found to be necessary to obtain similar copper grade-recovery curves as obtained in the Phase I program. The grind target was therefore changed from P80 of 75 to 55 µm both for the remaining testwork and for the plant design.

Copper roughing kinetic tests showed similar flotation characteristics to the master composite samples with Cu recoveries ranging between 87 and 96% Cu. Zinc roughing kinetic tests were also conducted on the variability samples. The majority of these tests recovered 96% or more of the remaining zinc to the rougher concentrate.

As only single tests were conducted on the variability composite samples, there was no optimization of the test conditions, and were not used in any part to formulate the final feasibility grades and recoveries for the primary mineralization.

Initial flotation tests on both the zinc-rich and low zinc Phase II master composite samples used a depressant combination of zinc sulphate and sodium cyanide in both the primary grind and in the regrind. Later tests on the Phase II primary mineralization master composite centered on the substitution of the zinc sulphate/sodium cyanide depressant with the depressant sodium sulphite, in both the primary grind and the regrind. Sodium sulphite showed improved selectivity against pyrite, but inferior selectivity against sphalerite in copper roughing and cleaning. Combinations of both depressants were also tested, which showed some promise.

2010 Mintek Test Program

The objective of the Mintek test program was to replicate the supergene flotation testwork completed in 2005. As such, the program included the evaluation of various circuit configuration including primary and secondary rougher rates, primary and secondary cleaner rates and cleaning tests with and without regrinding. The results of these tests showed that the ore sample delivered to Mintek did not behave the same as the previously tested sample at SGS. The sample tested at Mintek seemed to have a slower flotation response.

The supergene test composite prepared for the Mintek testwork contained 4.6% copper and had a high iron content of 44%. The final cleaner concentrate produced using the SGS flowsheet and conditions resulted in a copper recovery of 87.2% and a copper grade of 31.6%. Copper recovery was lower than achieved in the 2005 SGS work at 93% recovery with a grade of 30% copper.

2010 Maelgwyn Test Program

The objective of the Maelgwyn test program was to optimize the flotation parameters for feasibility design of the supergene and primary mineralization. Flotation variables tested to determine the effect on flotation performance included particle grind size, flotation time, varying dosages in the reagent suite, mass pull, pH, effect of temperature, regrind and extent of oxidation. Repeat tests were conducted on select test to confirm reproducibility of results. Following optimization of a number of conditions additional batch flotation testwork was conducted on supergene sample composite material to generate kinetic data to allow for calculation of flotation rates in staged flotation including roughers, cleaners and cleaner scavengers. Rougher kinetic tests only were completed for primary mineralization composite samples. The kinetic information is used by Eurus to build simulation models of the circuit for further analysis and optimization. No locked cycle tests were conducted.

The supergene test composite prepared for the Maelgwyn testwork contained 4.4% copper, 0.50% zinc, 43.2% iron and 46.3% sulphur. The final cleaner concentrate produced a copper recovery of 89% and a copper grade of 30.8%.

The primary test composite prepared for the Maelgwyn testwork contained 1.28% copper, 7.69% zinc, 34.9% iron and 45.9% sulphur. Based on rougher flotation testwork and extrapolated results from secondary flotation results for copper and zinc from the supergene cleaner testwork, the Eurus simulation model estimates copper recovery of 61% and a copper grade of 22.4% from primary ore using the supergene flotation circuit. At the time of reporting, the testwork and simulation modeling work is ongoing.

16.2 Comment on Section 16

In the opinion of the QPs, the metallurgical testwork completed on the Project has been appropriate to establish optimal processing routes for the different mineralization styles encountered in the deposits:

- Sample testwork has been based on mineralization that is typical of the various mineralization types currently interpreted in the deposits.
- Recoveries used in the 01 January, 2011 mineral resource and mineral reserve estimations are consistent with metallurgical testwork on the various mineralization types . Increased conservatism was used in recovery estimates where the amount of testwork is limited.
- As a result, the proposed plant design and associated recovery factors are considered appropriate to support mineral resource and mineral reserve estimation.
- Based on metallurgical testwork completed in 2010 to February 2011 supergene copper recovery was reduced from 92% to 88%
- Locked cycle flotation testwork is recommended to support the design of the supergene flotation circuit and associated recovery factors.
- Additional batch and locked cycle flotation testwork is recommended to support the design of the primary ore flotation circuit and associated recovery factors.
- It is recommended that oxidation effects on supergene and primary flotation recovery be further evaluated.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Mineral Resources

The resource model was prepared in 2005 by AMEC (S. Blower, P.Geo. under the supervision of Mr D. Reddy, P.Geo), and reviewed in 2009 by David Thomas, P.Geo. Mr Thomas is the Qualified Person for the Mineral Resource estimate.

17.1.1 Database

The Project database was closed for Mineral Resource estimation purposes as at December 2005.

The database used to support mineral resource estimation contains 356 drill holes, comprising core holes, nine of which were pre-collared with reverse circulation (RC). Sections 11 through 14 of this Report discuss the drill hole data.

No RC sample data were used for interpolation, although geologic data from RC drill holes were used in conjunction with the core data for the geologic interpretation.

A total of 22,695 assays were available to support Mineral Resource estimation. Of these, a total of 9,712 are located within the six mineralized domains used in the estimate.

17.1.2 Resource Model

The geological interpretation was completed by Nevsun based on lithological, mineralogical and alteration features logged in drill core. Nevsun digitized outlines of the interpreted mineral domains on vertical cross sections using logged drill hole data displayed on drill hole traces as a reference. The interpretations were transferred to AMEC as electronic files and paper plots. During 2005–2006, AMEC created 3D solid model wireframes of the entire mineralized body incorporating all six geologic domains by recreating Nevsun's sectional polylines using Gemcom software. These were linked together to form one 3D wireframe model of the Bisha mineralization.

This mineralized shape was then subdivided into four preliminary units by clipping with modelled surfaces representing: (1) the bottom of the oxide domain, (2) the bottom of the acid domain, and (3) the bottom of the supergene domain. The oxidized domain above the oxide/acid contact surface was further subdivided into the oxide and breccia domains by clipping with another solid. Similarly, the primary domain beneath the surface representing the bottom of the supergene alteration was subdivided into the Primary and Primary Zn domains by clipping with the appropriate surface.

All solids were validated and checked for crossing errors, consistency, and closure prior to use. The solids and surfaces were used to assign the corresponding geological code to the 3D block models.

The size of the blocks (5 m cubes) was selected based on the data spacing, a visible inspection of the size and geometry of the geological domains, and consideration of the selective mining unit (SMU).

17.1.3 Composites

AMEC completed a review of composite length alternatives to determine the sensitivity of mean grade and other statistical properties to composite length. Based on:

- The insensitivity of mean composite grade to changes in composite length
- The 5 m x 5 m x 5 m size of the model blocks
- The 5 m x 5 m x 5m size of the anticipated SMU
- The relatively large width of the mineralization, commonly measured in tens of metres.

AMEC opted to use a 5 m downhole composite length, broken on the wireframe boundaries. Residual composites were created with lengths less than 5 m at the lower end of each domain solid. These were left in the data set if their lengths were greater than 2.5 m or if there was no other adjacent composite, and were recombined with the previous composite interval if their lengths were less than 2.5 m.

The composite data set was declustered before statistical analysis to avoid biases in the statistics caused by a spatially uneven data set, a feature inherent in most geologic data sets. AMEC declustered with a geologic cell declustering method.

The effect of core recovery on metal grades for the assays within the mineralized domains was reviewed with the aid of sorted scatter plots. There is a negative correlation between gold grade and core recovery, but no relationship between copper or zinc grades and core recovery.

As most of the low recovery assays are associated with the gold-rich oxidized portion of the deposit, a decision was made to remove from the database all of the assays with core recoveries of less than 60%. These were removed before they were composited for use in either exploratory data analysis (EDA) or interpolation.

Assays with core recoveries less than 60% were ignored during compositing and therefore composite lengths are based on the drill hole interval and not based on the length of assays within the composite.

17.1.4 Data Distribution

Histograms, scatter plots, cumulative frequency plots, and box plots were used for data analysis. These plots were useful for characterizing grade distributions, identifying multiple populations within a dataset, and identifying domains that required restriction of outlier samples.

Contact plots were created to assess the behaviour of metal grades at the domain boundaries. Soft contacts in interpolation allow composites from one side of a boundary to influence block grades on the other. Hard contacts are characterized by sharp grade transitions across boundaries, and do not allow blocks to be informed by composites from another domain.. The majority of contacts were hard.

17.1.5 Variography

Correlograms of the composites for each metal (Au, Ag, As, Cu, Pb, Zn) were computed for three domain groups: Oxide (combined Breccia, Oxide and Acid); Supergene, and Primary (combined Primary Zn and Primary) with using commercially-available Sage2001 software. Down hole experimental correlograms were fitted to determine the nugget effect.

17.1.6 Density

AMEC used two methods for the final block model bulk density assignments, one for the oxidized domains, and one for the supergene and primary domains. Multiple linear regression was used to establish a relationship between the high-quality waxed bulk density measurements and zinc, copper, lead, iron, sulphur and barium analyses.

This method required block grades for each of the metals in the regression equations, so AMEC constructed grade block models for iron, sulphur and barium with ordinary kriging. Bulk density was then calculated for each block. A final adjustment was made to correct calculated densities in an area of anomalously high barium values in the supergene domain, which were causing the calculated bulk density to be too low.

17.1.7 Estimation Methodology

Multiple passes of ordinary kriging were used to interpolate grades in blocks from composited drill hole data. For most metals and domains, restricted kriged grades, unrestricted kriged grades, and unrestricted nearest neighbour assignments were all stored in blocks.

To ensure local reproduction of composite grade trends, and to help control grade smearing, the resource model was interpolated by multiple passes of ordinary kriging within successively larger search radii. A total of three passes were required to fill all of the blocks in the Primary Zn and Primary domains. All of the other domains were completely estimated with two passes. The search radius of the first pass was roughly equal to one half of the variogram range. The radius of the second pass was twice the length of the first pass, and the radius of the third pass was set at two times the length of the second pass. Some adjustments were made to these radii (commonly involving an increase in the across strike radius) where necessary to prevent striping patterns in the resulting block grades, and to ensure that sufficient composites were captured to estimate all blocks at bends in the wireframes.

Two different search ellipse orientations were used, one for the oxide and supergene domains, and another for the primary domains. The search ellipse for the oxide and supergene domains is isotropic in plan view, with a vertical axis shorter than the horizontal axes. For the primary domains, the ellipse was oriented so that it was roughly parallel with the mineralization. The axes of the ellipse were isotropic within the dip plane, and the third axis (orthogonal to the plane of the mineralization) was shorter than the other two.

For all passes, a minimum of three and a maximum of 12 composites were used. A maximum of two composites were allowed per drill hole to ensure that multiple holes would contribute to block values.

17.1.8 Grade Capping (Metal at Risk)

High-grade restrictions were used in the resource model to limit the spatial extrapolation of occasional, isolated, anomalously high-grades. To determine the extent to which high-grade composites should be restricted, AMEC used a simulation approach.

The grade capping analysis utilized the entire population within the modelled geologic domains, and an assumed production rate of 5,500 t/d. At 5,500 t/d, a month-sized block of production would be equal to 167,000 t or 55,300 m^3. With a drill hole spacing of 25 m x 25 m x 3.5 m (reflecting the angled drilling pattern), the volume of mineralization per composite is 2,190 m^3. Therefore, 300 composites would be mined every year. Using these figures and the declustered composite distribution, a percentage metal-at-risk was calculated for each metal examined (Table 17-1).

Table 17-1: Metal-at-Risk Results for the Significant Metal and Domain Combinations

Metal	Domain	% Metal at Risk	Unit	Cap Level Suggested by Simulation
Au	Bx+Oxid+Acid	8.4	(ppm)	55
Ag	Bx+Oxid+Acid	20.9	(ppm)	900
Cu	Supergene	5.2	(%)	5.2
Zn	Primary Zn	3.3	(%)	18
Zn	Primary	7.1	(%)	8

Note: BX = breccia, Oxid = oxide

In AMEC's opinion, the amount of metal at risk in Table 17-1 was reasonable for this style of deposit at this stage of project development. The levels may decline somewhat as drilling continues on the Project. The amount of silver noted to be at risk is relatively high, but the very high grades are mainly restricted to the Acid domain, which is volumetrically contained and relatively small.

17.1.9 Dilution Considerations

The grade models were constructed with fixed length composites that honour wireframe boundaries and ignore lithological distinctions and grade. That is, every assay within the wireframe is used for estimation, providing it meets the recovery threshold described earlier. Therefore, AMEC considered the block grades to have already incorporated internal dilution, and no further internal dilution should be added to the blocks for mine planning.

However, external dilution was not added to this Mineral Resource model. The grades in the blocks and the tabulation of tonnage and grade in the Mineral Resource Summary Table only consider the mineralized portion of each block, as determined by the volume percent model item. Therefore, external dilution (and mining recovery) will need to be considered when using this block model for mine planning purposes.

17.1.10 Validation

Four validation exercises were completed on the Bisha resource model:

- Visual comparison of block and composite grades on sections and plans. No discrepancies between block and composite grades were observed

- Global statistical comparison of block and declustered composite grades the mean block grades are within 5% of the mean declustered composite grades for most of the metal-domain combinations. The only significant departures are for silver and zinc in the Acid domain, and gold in the breccia domain. The difference noted for silver in the Acid domain is due to the use of soft contacts with the lower-grade supergene and Oxide domains. The difference for zinc in the Acid domain is irrelevant due to the very low-grades there. Differences for gold in the breccia domain are caused by two outliers that severely impact the mean nearest neighbour grade. To test the sensitivity of the comparison to these outliers, the nearest neighbour grades were capped at 30 g/t and re-tabulated. The capped nearest neighbour grade was within 4% of the capped kriged grade. In AMEC's opinion, there is no global bias observed in the model
- Local comparison of block and declustered composite grades. In all three directions, the kriged blocks generally honour the distribution of declustered composite grades, indicating that no local bias is observed in the model. Any deviations noted correspond to areas where there are only a small number of blocks
- Herco change of support. The kriged gold distribution in the Oxide domain was similar to that of the transformed nearest neighbour target distribution, below a cut-off grade of < 2.5 g/t Au.

No errors were noted with the models that would affect Mineral Resource estimation.

17.1.11 Classification of Mineral Resources

In determining the appropriate classification criteria for Bisha, several factors were considered:

- The distribution of pierce points at the massive sulphide contacts. Massive sulphide contacts in the upper levels of the Bisha deposit are defined by pierce points that are almost always within 50 m of each other. The pierce point density decreases with depth in the lower half
- Observations of grade and geologic continuity on section and plan. The massive sulphide is very continuous when defined by holes drilled at 25 m intervals on sections spaced 12.5 m apart. Mineralization drilled at this density is confined to two small separate domains near the top of the deposit. The remainder of the upper half of the deposit has been drilled with holes at 25 m centres on sections spaced 25 m apart. For much of the remainder of the interpreted deposit, continuity can be reasonably assumed between holes spaced at distances greater

than 25 m x 25 m. An exception is the down-dip extension of the primary mineralization that has been extrapolated beyond distances where continuity can be reasonably assumed

- Confidence limit analysis results on grades. AMEC has generally found that for base and precious metal deposits, sampling must be sufficient to estimate the tonnage and grade on *quarterly* production increments ±15% at 90% confidence in order to define a Measured Resource. A resource should be classified as Indicated only if grades can be estimated with a ±15% accuracy on an *annual* basis at a 90% confidence limit. Idealized blocks approximating the production from one month (165,000 t at 5,500 t/d) were estimated by ordinary kriging using different grids of samples to calculate an ordinary kriging variance for the large block
  - The confidence limit analysis results suggest that Oxide resources are eligible for Measured status when the drill density is 12.5 m (between sections) x 25 m (on section) or less. Primary mineralization is eligible for classification as Measured if the drill spacing is 25 m (between sections) x 12.5 m (on section) or less
  - Oxide resources are eligible for Indicated status if they are within areas drilled with a 25 m x 25 m drill spacing or higher density. Primary mineralization is eligible for Indicated status with all drill spacings tested
  - After consideration of the continuity of the massive sulphide contacts on plan and section, AMEC considers that only those portions of the massive sulphide drilled at spacings of 12.5 m x 25 m or less are eligible for Measured status, and those portions drilled at spacings of 25 m x 25 m are eligible for Indicated status
  - Those portions defined by holes spaced up to 50 m apart are eligible for classification as Inferred
  - Blocks further than 50 m from a drill hole should be left unclassified and not included as part of the Mineral Resource.
- AMEC's experience with other VMS deposits
- NI 43-101/CIM requirements and guidelines.

17.1.12 Assessment of "Reasonable Prospects of Economic Extraction"

Mineral resources that could be extracted by open pit methods were confined within a Lerchs–Grossmann optimized pit shell using the following economic parameters:

- A mining cost of \$1.57/t, plus \$0.01/t/5 m bench below the reference elevation of 560 m
- Gold price of \$1,170/oz
- Silver price of \$18.20/oz
- Copper price of \$2.76/lb
- Zinc price of \$1.05/lb
- Total ore-based costs (process, G&A and stockpile rehandle) are \$29.03/t for oxide, \$26.57/t for supergene and \$26.68/t for primary ores
- Metallurgical recoveries by ore type are 87% Au and 36% Ag for oxide mineralization reporting to doré, 56% Au, 54% Ag and 88% Cu for supergene mineralization reporting to Cu concentrate, 36% Au, 29% Ag and 85% Cu for primary copper mineralization reporting to copper concentrate, and 9% Au, 20% Ag and 83.5% Zn for primary zinc mineralization reporting to zinc concentrate
- All applicable smelter terms including penalties for deleterious elements and other offsite costs
- Royalties for each payable metal; silver 5.0%, gold 5.0%, zinc 3%, and copper 3%
- Transportation costs of \$87/t of copper or zinc concentrate
- Overall pit slopes varied from 43º to 55.5º in rock and 18.5º in overburden.

17.1.13 Mineral Resource Statement

Mineral resources are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources are inclusive of Mineral Reserves and do not include external dilution. AMEC cautions that mineral resources that are not mineral reserves do not have demonstrated economic viability.

The Mineral Resource estimate has an effective date of 1 January, 2011. David Thomas, P.Geo, an AMEC employee is the QP for the estimate.

The classified Bisha Mineral Resource estimate is summarized by domain at the marginal NSR cut-offs in Table 17-2. The oxide stockpile inventory at 1 January 2011, is shown in Table 17-3. The sensitivity of the Mineral Resource to various NSR cut-off grades is shown in Table 17-4, with the base case for each mineralization type highlighted.

Table 17-2: Bisha Mineral Resource Estimate – Effective Date 01 January, 2011 (David Thomas, P.Geo.)

Category	Zone	NSR Cut-Off	Tonnes ('000 t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Au ('000 oz)	Ag ('000 oz)	Cu ('000 lb)	Pb ('000 lb)	Zn ('000 lb)
Measured	Oxides	$29.03	735	6.43	28.37	0.11	0.71	0.11	152	670	1,838	11,518	1,706
	Supergene Cu	$26.57	854	0.77	43.33	4.98	0.16	0.24	21	1,189	93,796	3,081	4,490
	Primary	$26.68	535	0.76	50.72	0.86	0.33	7.68	13	872	10,088	3,900	90,557
	Subtotal Measured		2,124	2.72	40.01	2.26	0.40	2.07	186	2,732	105,722	18,499	96,753
Indicated	Oxides	$29.03	3,671	7.56	32.46	0.08	0.55	0.07	893	3,832	6,767	44,544	5,816
	Supergene Cu	$26.57	6,830	0.73	30.74	3.75	0.10	0.10	160	6,751	564,024	14,385	15,503
	Primary	$26.68	17,224	0.71	43.34	0.95	0.19	5.12	393	23,999	362,493	71,679	1,944,427
	Subtotal Indicated		27,726	1.62	38.79	1.53	0.21	3.22	1,446	34,582	933,284	130,608	1,965,747
Measured	Oxides	$29.03	4,406	7.38	31.78	0.09	0.58	0.08	1,045	4,502	8,605	56,062	7,522
+	Supergene Cu	$26.57	7,684	0.73	32.14	3.88	0.10	0.12	181	7,941	657,820	17,466	19,993
Indicated	Primary	$26.68	17,759	0.71	43.56	0.95	0.19	5.20	406	24,871	372,581	75,579	2,034,984
	Subtotal M+I		29,849	1.70	38.88	1.58	0.23	3.14	1,632	37,314	1,039,006	149,107	2,062,500
Inferred	Oxides	$29.03	44	3.46	21.93	0.02	0.06	0.02	4.86	31	21	56	18
	Supergene Cu	$26.57	197	0.48	22.09	1.97	0.05	0.03	3.06	140	8,537	210	120
	Primary	$26.68	10,330	0.66	48.38	0.90	0.24	5.80	218.39	16,068	203,942	54,065	1,320,452
	Subtotal Inferred		10,570	0.67	47.78	0.91	0.23	5.67	226.32	16,239	212,500	54,330	1,320,590

Note: Mineral Resources are inclusive of Mineral Reserves

Table 17-3: Bisha Oxide Stockpile Inventory Effective Date January 1, 2011 (David Thomas, P.Geo.)

Category	Zone	Tonnes ('000 t)	Au (g/t)	Au ('000 oz)
Measured	Stockpile	197,235	3.64	23,111

Table 17-4: Resource Sensitivity at Various NSR Cut-Off Grades with Base Cases Highlighted, Effective Date 01 January 2011, (David Thomas, P.Geo.)

		Grade					Metal				
Cut-Off	**Tonnes ('000 t)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Au ('000 oz)**	**Ag ('000 oz)**	**Cu ('000 lb)**	**Pb ('000 lb)**	**Zn ('000 lb)**
Oxide											
Measured											
NSR > $20	751	6.31	27.93	0.11	0.70	0.11	152	674	1,865	11,662	1,743
NSR > $29.03	735	6.43	28.37	0.11	0.71	0.11	152	670	1,838	11,518	1,706
NSR > $40	719	6.55	28.70	0.11	0.72	0.11	151	664	1,801	11,349	1,664
NSR > $50	706	6.65	28.96	0.11	0.72	0.10	151	657	1,766	11,203	1,630
Indicated											
NSR > $20	3,856	7.23	31.37	0.08	0.54	0.07	897	3,888	7,093	45,900	6,210
NSR > $29.03	3,671	7.56	32.46	0.08	0.55	0.07	893	3,832	6,767	44,544	5,816
NSR > $40	3,483	7.92	33.64	0.08	0.56	0.07	887	3,767	6,397	43,064	5,457
NSR > $50	3,394	8.09	34.31	0.08	0.57	0.07	883	3,743	6,248	42,658	5,326
Measured and Indicated											
NSR > $20	4,607	7.08	30.81	0.09	0.57	0.08	1,049	4,563	8,958	57,562	7,953
NSR > $29.03	4,406	7.38	31.78	0.09	0.58	0.08	1,045	4,502	8,605	56,062	7,522
NSR > $40	4,202	7.68	32.80	0.09	0.59	0.08	1,038	4,431	8,198	54,413	7,122
NSR > $50	4,100	7.84	33.38	0.09	0.60	0.08	1,034	4,401	8,014	53,861	6,956
Inferred											
NSR > $20	55	2.87	18.41	0.02	0.06	0.02	5	33	30	76	25
NSR > $29.03	44	3.46	21.93	0.02	0.06	0.02	5	31	21	56	18
NSR > $40	40	3.70	23.68	0.02	0.05	0.02	5	30	19	47	16
NSR > $50	35	4.02	25.94	0.02	0.05	0.02	5	29	17	40	14
Supergene											
Measured											
NSR > $20	859	0.77	43.16	4.96	0.16	0.24	21	1,191	93,820	3,082	4,493
NSR > $26.57	854	0.77	43.33	4.98	0.16	0.24	21	1,189	93,796	3,081	4,490
NSR > $40	836	0.77	43.97	5.08	0.17	0.24	21	1,182	93,607	3,074	4,471
NSR > $50	821	0.78	44.46	5.16	0.17	0.25	21	1,174	93,384	3,066	4,444
Indicated											
NSR > $20	7,044	0.72	30.11	3.64	0.09	0.10	163	6,819	565,521	14,469	15,791
NSR > $26.57	6,830	0.73	30.74	3.75	0.10	0.10	160	6,751	564,024	14,385	15,503
NSR > $40	6,362	0.76	32.26	3.98	0.10	0.11	155	6,599	558,571	14,158	14,818
NSR > $50	5,995	0.78	33.50	4.18	0.11	0.11	151	6,457	552,326	13,930	14,254
Measured and Indicated											
NSR > $20	7,902	0.72	31.53	3.79	0.10	0.12	184	8,010	659,341	17,552	20,284
NSR > $26.57	7,684	0.73	32.14	3.88	0.10	0.12	181	7,941	657,820	17,466	19,993
NSR > $40	7,198	0.76	33.62	4.11	0.11	0.12	176	7,780	652,177	17,232	19,289
NSR > $50	6,816	0.78	34.82	4.30	0.11	0.12	171	7,631	645,710	16,996	18,697

				Grade					Metal		
Cut-Off	Tonnes ('000 t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Au ('000 oz)	Ag ('000 oz)	Cu ('000 lb)	Pb ('000 lb)	Zn ('000 lb)
Inferred											
NSR > $20	230	0.46	20.78	1.73	0.04	0.03	3	154	8,753	212	136
NSR > $26.57	197	0.48	22.09	1.97	0.05	0.03	3	140	8,537	210	120
NSR > $40	169	24.37	2.20	0.05	0.03	0.02	132	12	205	114	69
NSR > $50	148	0.54	26.79	2.40	0.06	0.03	3	128	7,825	199	111
Primary											
Measured											
NSR > $20	613	0.71	45.53	0.79	0.29	6.74	14	897	10,638	3,938	91,022
NSR > $26.68	535	0.76	50.72	0.86	0.33	7.68	13	872	10,088	3,900	90,557
NSR > $40	438	0.80	58.26	0.96	0.39	9.25	11	820	9,280	3,809	89,240
NSR > $50	393	0.82	62.06	1.02	0.43	10.18	10	785	8,804	3,752	88,274
Indicated											
NSR > $20	17,495	0.71	42.80	0.94	0.19	5.05	397	24,076	364,170	71,788	1,946,891
NSR > $26.68	17,224	0.71	43.34	0.95	0.19	5.12	393	23,999	362,493	71,679	1,944,427
NSR > $40	15,950	0.72	45.72	1.00	0.20	5.46	371	23,445	351,867	70,619	1,919,920
NSR > $50	13,974	0.75	49.45	1.08	0.22	6.03	335	22,215	332,267	67,390	1,856,775
Measured and Indicated											
NSR > $20	18,107	0.71	42.90	0.94	0.19	5.11	411	24,973	374,808	75,725	2,037,913
NSR > $26.68	17,759	0.71	43.56	0.95	0.19	5.20	406	24,871	372,581	75,579	2,034,984
NSR > $40	16,388	0.73	46.05	1.00	0.21	5.56	382	24,265	361,148	74,427	2,009,160
NSR > $50	14,367	0.75	49.79	1.08	0.22	6.14	345	23,000	341,071	71,142	1,945,050
Inferred											
NSR > $20	10,335	0.66	48.36	0.90	0.24	5.80	218	16,069	203,978	54,065	1,320,477
NSR > $26.68	10,330	0.66	48.38	0.90	0.24	5.80	218	16,068	203,942	54,065	1,320,452
NSR > $40	10,243	0.66	48.67	0.90	0.24	5.84	217	16,029	203,089	54,029	1,318,893
NSR > $50	9,829	0.67	49.84	0.91	0.25	6.04	211	15,752	197,831	53,622	1,308,192

Notes to accompany Mineral Resource and Sensitivity Tables:

1. Mineral resources that are not mineral reserves do not have demonstrated economic viability
2. Mineral Resources are inclusive of Mineral Reserves and do not include external dilution.
3. A Lerchs–Grossmann pit shell was used to constrain the Mineral Resources to assess reasonable prospects of eventual economic extraction
4. Mineral Resources are reported using assumed long-term prices as follows: gold price of $1,170/oz, silver price of $18.20/oz, copper price of $2.76/lb, and zinc price of $1.05/lb
5. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
6. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper, lead and zinc pounds as imperial pounds

17.1.14 Exploration Targets

The Northwest Zone prospect is at an early stage of data collection, and insufficient information is available to support mineral resource estimation. However, the number of drill holes, and the dimensions of the mineralized area are sufficient to permit estimation of an Exploration Target.

Drilling to date has comprised 26 drill holes in the Northwest Zone. The main massive sulphide lens at the Northwest Zone has dimensions of approximately 650 meters in length, varies in width from 100 m to 175 m in width (the inter-limb distance across the antiform) and varies in thickness from less than 5 m to 70 m. In addition there may be a zinc-rich area that has dimensions of 200 m in length, a thickness of 10 m to 15 m along the limbs and 25 m to 40 m in the nose of the antiform.

The average grades of the main pyritic massive sulphide lens are 0.5% copper, 0.1% zinc, 0.5 g/t gold and 20 g/t silver. The average grades of the zinc-rich area are 0.9% copper, 3.6% zinc, 0.3 g/t gold and 34 g/t silver.

Using this information an Exploration Target at the Northwest Zone would be in the range of:

- 4 Mt to 11 Mt with grades ranging from 20 g/t to 45 g/t Ag, 0.3 g/t to 0.5 g/t Au, 0.5% to 1.1% Cu and 0.1% to 0.2% Zn.

AMEC cautions that the potential quantity and grade are conceptual in nature, that there has been insufficient exploration to define the Exploration Target as a Mineral Resource, and that it is uncertain if further exploration will result in the targets being delineated as a Mineral Resource. The Exploration Target lies outside the Bisha Main Zone Mineral Resources.

17.2 Mineral Reserves

17.2.1 Dilution and Loss

Losses and dilution were evaluated by considering the size of the mining excavators, the bench height, the visual differentiation between different materials, blasting requirements, and the three-dimensional complexity of the contacts.

At the contact between oxide and supergene, the material is expected to be very friable, does not have to be blasted, and can be separated by colour. At the contact between supergene and primary, a geological technician needs to guide the blast hole drilling team to ensure that only supergene will be blasted. Based on this evaluation AMEC considered that the contacts will be able to be followed with an accuracy of ±0.5 m.

The dilution and mining losses were modeled by selecting all of the blocks that were not 100% one domain, then within each block taking 10% of the block's volume (0.5 m of a 5 m block is 10%) of each constituent component and combining it into a mixed material which was then added back to the remainder of the original domain. When a domain consisted of less than 10% of a block the entire domain was taken.

The ore zones and waste have different densities so their tonnages change when a volumetric loss/dilution calculation is applied. Of the total ore tonnage, 4.5% is mixed with an equal volume of waste with the resulting mixed material being split equally between ore and waste. Between each ore zone, material is also being mixed, 3.8% of the ore is involved in this process. Although the mixing of material between zones still results in the same plant feed tonnage the processing of material in the incorrect circuit results in increased losses and penalties.

For the current mining rate and projected geological conditions, AMEC believes that the dilution and mining recovery are reasonable. AMEC believes that with further geological information and with any revision in the mining rates the dilution should be re-estimated.

17.2.2 Mineral Reserves Statement

The Proven and Probable Mineral Reserves at the Project have been classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves, incorporated by reference in NI 43-101.

Mineral Reserves are defined within a mine plan, with pit phase designs guided by Lerchs–Grossmann optimized pit shells, generated using a gold price of $825/oz, silver at $12.50/oz, copper at $2.00/lb, and zinc at $0.75/lb and considering diluted Measured and Indicated Mineral Resources.

The Bisha Mineral Reserves are shown in Table 17-5.

Table 17-5: Bisha Mineral Reserves, Effective Date: 01 January, 2011 (Jay Melnyk, P. Eng.)

Ore Type	Ore (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Oxide (above $29.03/t NSR cut-off)					
Proven	933	5.75	21.90		
Probable	3,719	7.39	31.48		
Sub-total Combined	4,651	7.06	29.56		
Supergene (above $26.57/t NSR cut-off)					
Proven	844	0.80	43.47	4.92	
Probable	6,537	0.77	31.29	3.77	
Sub-total Combined	7,382	0.78	32.68	3.90	
Primary (above $26.68/t NSR cut-off)					
Proven	521	0.78	52.51	0.91	8.09
Probable	15,759	0.72	44.12	0.97	5.31
Sub-total Combined	16,279	0.72	44.40	0.97	5.40
Total Proven	2,298	2.80	36.77	2.07	1.98
Total Probable	26,015	1.69	39.09	1.55	3.26
Total Combined	**28,313**	**1.78**	**38.90**	**1.60**	**3.15**

Notes to accompany Mineral Reserve Table:

1. Mineral Reserves are defined within a mine plan, with pit phase designs guided by Lerchs–Grossmann (LG) pit shells, generated using a gold price of $825/oz, silver at $12.50/oz, copper at $2.00/lb, and zinc at $0.75/lb and considering diluted Measured and Indicated resources. Metallurgical recoveries by ore type are 87% Au and 36% Ag for oxide mineralization reporting to dore, 56% Au, 54% Ag and 92% Cu for supergene mineralization reporting to Cu concentrate, 36% Au, 29% Ag and 85% Cu for primary copper mineralization reporting to copper concentrate, and 9% Au, 20% Ag and 83.5% Zn for primary zinc mineralization reporting to zinc concentrate. The mining cost was $1.46/t, plus $0.01/t/5m bench below the reference elevation of 560m. The total ore based costs (process, G&A and stockpile rehandle) are $29.03/t for oxide, $26.57/t for supergene and $26.68/t for primary ores. Overall pit slopes varied from 43º to 55.5º in rock and 18.5º in overburden.
2. Reserves are reported within the above mentioned pit phase designs, using an NSR grade item, where the marginal cut-off is the ore based cost stated above. After completion of the pit designs, the NSR was recalculated using a gold price of $1,015/oz, silver at $15.85/oz, copper at $2.40/lb, and zinc at $0.92/lb. Recoveries used for the NSR calculation are as above with the exception of the supergene copper recovery which was reduced from 92% to 88% based on recent metallurgical testwork.
3. The life of mine strip ratio is 4.16
4. Proven oxide Mineral Reserves are inclusive of 197 kt at 3.64 g/t Au in stockpile as of 01 January 2011
5. Tonnages are rounded to the nearest 1,000 tonnes. Grades are rounded to two decimal places.
6. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
7. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper, lead and zinc pounds as imperial pounds

17.3 Comment on Mineral Resources and Mineral Reserves

The QPs consider that the estimations of Mineral Resources and Mineral Reserves for the Project conform to industry best practices, and meet the requirements of CIM (2010). Mineral Reserves have taken into account environmental, permitting, legal, title, taxation, socio-economic, marketing and political factors and constraints, as discussed in Section 4 and Section 18 of this Report. Mineral Reserves are acceptable to support mine planning.

Factors which may affect the estimates include variations in assumed commodity prices and exchange rates.

At the time of this Report, resource modelling work has been initiated on the Harena prospect, but no results are available.

18.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORT ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

During 2005–2006, a feasibility study was completed on the Bisha Project, with costs updated in 2009. The mine plan from that study is still the current mine plan, and is summarized in the following subsections.

18.1 Mining Operation

The Bisha open pit mining operation features a single pit consisting of nine internal phases is currently being developed. The first three phases target the oxide ore, phases four to six target the Supergene ore, and phases seven to nine target the primary ore. The mine has an estimated life of approximately 12 years at the currently-planned mill throughputs of 2.0 Mt/a for oxide, 2.4 Mt/a for supergene and 2.4 Mt/a for primary materials. Pre-stripping of the oxide area began in mid-2010, with ore delivery to the ROM crusher pad beginning in July, 2010.

The mining method is conventional selective open pit mining, with mining rates of approximately 22 kt/d during the oxide phase. Mining rates increase to a peak of just under 50 kt/d later in the mine life as deeper primary mineralization is mined with higher strip ratios.

Blast hole drilling is performed by conventional down-the-hole hammer drills drilling 140 mm and 102 mm diameter holes, depending on material properties. Material will be drilled and blasted on 10 m-high bench and where required, will be mined in two 5 m-high flitches.

Loading is performed by two 7 m^3 hydraulic excavators with backup provided by an 8.4 m^3 front-end loader. Ore is stockpiled on the run-of-mine (ROM) ore pad and re-handled to the crusher as required for blending purposes. Ore and waste are hauled by 62 Mt rigid body mining trucks.

Appropriately-sized support equipment maintains bench floors, roads and dump faces. The equipment is considered appropriately sized for the degree of selectivity required. The equipment fleet size (existing, plus 2011 sustaining capital purchases) is considered appropriate for the material to be moved. Equipment additions and replacements will be made as required by the mining schedule.

During pre-stripping in mid to late 2010, a series of bench scale failures occurred in the central part of the northeast highwall, an area that was lacking geotechnical borehole information due to the challenging topography. Wall mapping and monitoring was implemented and geotechnical assistance retained to investigate the mode of failure, assess newly-collected data and to advise regarding revisions to slope design parameters.

On 4 January, 2011, a moderately large failure occurred in the same region of the highwall (refer to Section 18.6). Additional geotechnical drilling is underway to support further geotechnical and hydrological analysis. The resulting design recommendations will be used to design a layback of the wall, to allow mining of the resources below the safety step-out. Other phase design modifications may be required to allow access to all areas. The mining of the new highwall pushback would likely start in the third or fourth quarter of 2011. However, as this layback has not yet been designed, it is not included in the current long-range mine plan.

18.1.1 Pit Optimization and Pit Phase Design

The pit optimization was performed by AMEC in 2010, using a gold price of \$825/oz, silver at \$12.50/oz, copper at \$2.00/lb, and zinc at \$0.75/lb and considering Measured and Indicated Mineral Resources. The reference mining cost of was \$1.46/t of material mined, plus an incremental cost with depth of \$0.01/t/5m bench below a reference elevation of 560 m elevation. A processing cost of US\$24.01/t for oxides, \$21.55/t for supergene, and \$21.66/t for primary was used plus an additional \$0.34/t to cover the cost of a loader tramming material at the ROM pad. A G&A cost of \$4.68/t was applied as an ore-based cost. The operating costs used for the pit optimization were provided by BMSC based on their internal estimates.

The process recoveries used are shown in Table 18-1. Since completion of the pit phase designs, the supergene copper recovery was reduced from 92% to 88% based on recent metallurgical test work.

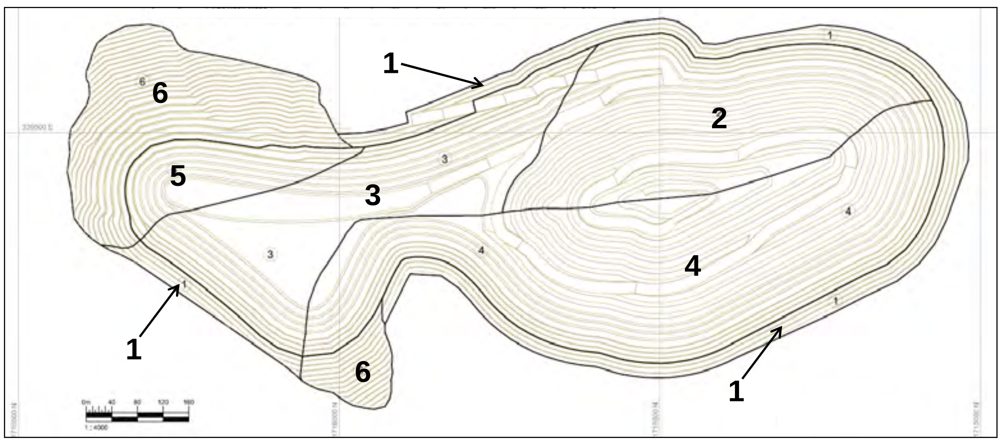
The overall pit slope angles applied ranged from 37° to 50°, based on the recommendations contained in Gonzalez (2010). The pit rock mass has been subdivided into six geotechnical domains as shown in Figure 18-1.

The overall slope angles were reduced for each of the geotechnical domains to flatten the slopes, allowing for the haulage ramps that would be included in the mine design.

Table 18-1: Process Recovery

Element	Au (%)	Ag (%)	Cu (%)	Zn (%)
Oxide				
Gold Dore	87.0	36.0	0.0	0.0
Supergene				
Cu Concentrate	56.0	54.0	92.0	0.0
Primary				
Cu Concentrate	36.0	29.0	85.0	0.0
Zn Concentrate	9.0	20.0	0.0	83.5

Figure 18-1: Geotechnical Design Sectors*



Note: Sectors are shown on 2006 Feasibility Study ultimate pit

Slope angle reductions were based on the haulage ramp width, the number of times a haulage ramp traversed a design sector, and the overall slope height of the sector. The flattened wall slopes are shown in Table 18-2.

Mineable pit phases were designed based on optimized nested pit shell guidance, geological boundaries, strip ratio, and access. The revenue factor 1.0 pit shell was used for guidance for the ultimate pit design. Ramps in final walls have a design width of 27 m and a gradient of 10%. A nominal minimum mining width of 25 m was used for phase design. The ultimate pit design is shown in Figure 18-2, and the individual phases are shown in Figure 18-3 in a bench plan at the 545 m elevation.

Table 18-2: Geotechnical Slopes Flattened for Ramps

Sector	Inter-ramp Angle (degrees)	Overall with Ramps (degrees)
1	43.0	37.0
2	47.5	39.0
3	51.5	50.0
4	55.5	47.0
5	51.5	43.0
6	51.0	46.0

Figure 18-2: Bisha Phase 9 (Ultimate Pit)

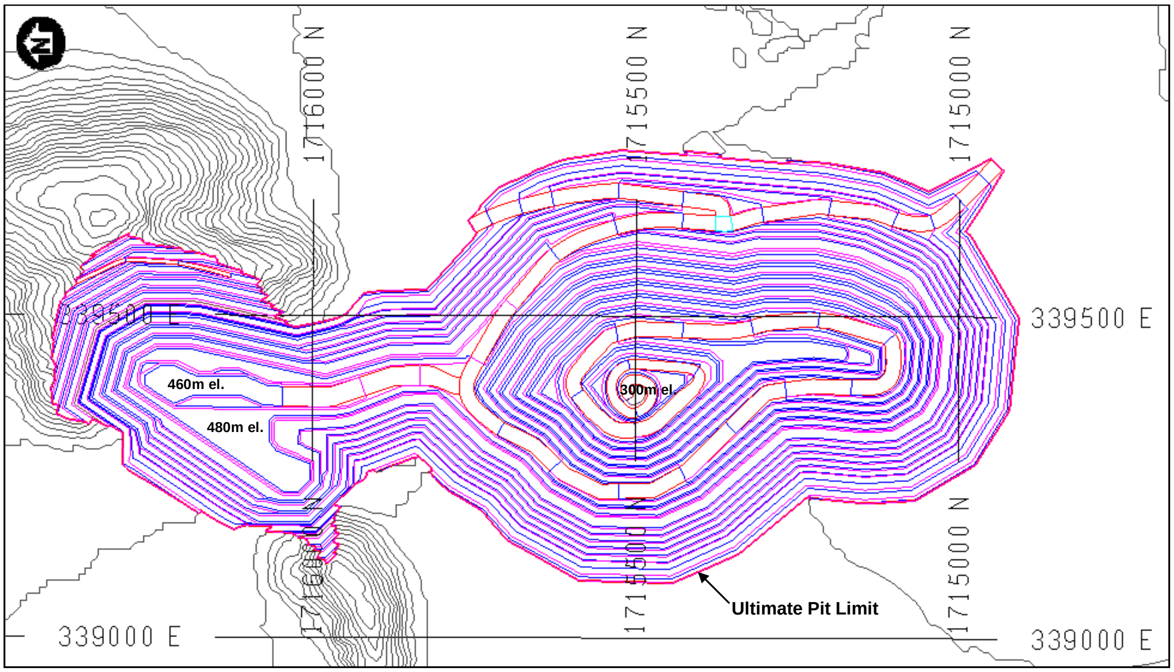
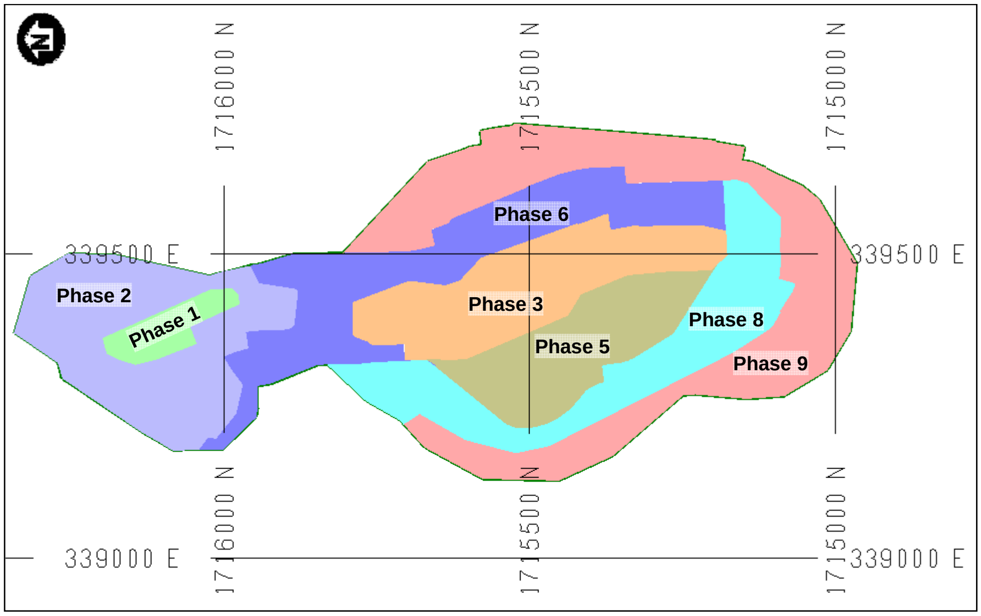


Figure 18-3: Pit Phases in Plan at 545 m Elevation



The current long-range mine plan was prepared by AMEC using the June 2010 pit phase designs, but with an updated net smelter return (NSR) calculation based on a gold price of $1,015/oz, silver at $15.85/oz, copper at $2.40/lb, and zinc at $0.92/lb and metallurgical recoveries shown in Table 18-3. The NSR calculation also considers, smelting and refining terms, transportation costs and royalties for the four different sellable products: oxide doré, supergene copper concentrate, primary copper concentrate and primary zinc concentrate.

When applying a cut-off to NSR grades expressed as $/t milled, the marginal cut-off is equal to the sum of the on-site ore based costs. For ore/waste delineation, a marginal cut-off approach was used as per client guidance. The ore-based costs, shown in Table 18-4, are the same as those applied during the pit optimization, and were provided by Nevsun in 2010, based on their internal estimates.

The following volumetrics are for grade ranges from $1/t less than the cut-off, up to $1/t more than the NSR cut-off, giving approximate average grades of material at the NSR cut-off. In the Oxide, material near the $29.03/t milled NSR cut-off has grades of approximately 0.94 g/t Au and 8.4 g/t Ag. In the Supergene, material near the $26.57/t NSR cut-off has grades of approximately 0.45% Cu, 0.43 g/t Au, 11.7 g/t Ag. In the Primary, material near the $26.68/t NSR cut-off has grades of approximately 0.35% Cu, 0.53 g/t Au, 12.5 g/t Ag and 0.72% Zn.

Table 18-3: Metallurgical Performance of the Three Ore Types as at 17 February 2011

	Au Recovery (%)	Ag Recovery (%)	Cu Grade (%)	Cu Recovery (%)	Zn Grade (%)	Zn Recovery (%)
Bullion from Oxide Ore	87	36	—	—	—	—
Cu Concentrate from Supergene Ore	56	54	30	88	—	—
Cu Concentrate from Primary Ore	36	29	25	85	3.9	2.1
Zn Concentrate from Primary Ore	9	20	0.3	3	55	83.5

Table 18-4: Ore-based Costs

		Oxide	Supergene	Primary
Process Cost	($/t Milled)	24.01	21.55	21.66
Stockpile Rehandle	($/t Milled)	0.34	0.34	0.34
G&A	($/t Milled)	4.68	4.68	4.68
Total Ore Based Cost	($/t Milled)	29.03	26.57	26.68

Although the operation has entered the production phase, current actual costs are not considered representative of long-term operating costs. AMEC considers these cost estimates reasonable for the purposes of this Report, but recommends they be updated based on actual data when the mine plan is updated following the current geotechnical assessment.

The re-calculated pit phase volumetrics using the marginal cut-offs for each rock type ($29.03/t for oxide, $26.57/t for supergene and $26.68/t for primary ores), and the 01 January, 2011 surveyed pit end of period surface, are shown in Table 18-5. This includes the BMSC-estimated low-grade oxide stockpile inventory as of 1 January, 2011.

18.1.2 Production Forecast

The mine plan presented in this Report was carried out using annual scheduling periods, targeting throughputs of 2.0 Mt/a for oxide, 2.4 Mt/a for supergene and 2.4 Mt/a for primary materials. The mine will work 360 d/a, with five days allowed for delays due to winter conditions. The plant is scheduled to operate 365 d/a.

The mine plan does not reflect potential changes to the northeast highwall and so is an interim plan. Upon completion of the 2011 geotechnical and hydrological study, the oxide pit phases will be re-designed, and a new, more detailed life-of-mine (LOM) plan will be carried out. This work is expected to result in a higher strip ratio for the oxide portion of the pit.

The mill feed constitutes ore transported directly from the mine plus ore reclaimed from the blending stockpiles. The annual mine plan is summarized in Table 18-6, and displayed graphically in Figure 18-4 and Figure 18-5.

18.1.3 Waste Rock Storage

Waste rock will primarily be placed in external waste dumps located to the east and south of the ultimate pit. Backfilling the northern portion of the pit with waste is planned in the later part of the mine life. The pit backfill design provides an additional access ramp to the pit, is a shorter haul then the external waste dumps, and is used to balance the truck fleet size.

The waste dumps are shown in Figure 18-6. The waste dump designs will be revised as required following the mine plan update.

18.1.4 Ore Stockpiles

The mine plan does not include use of elevated cut-offs and long term stockpiles. Short term stockpiling is performed for blending purposes on the designated ROM pad. The mine and mill will campaign Oxide ores first, followed by Supergene then Primary ores. At times, it may be necessary to stockpile small quantities of early exposures of the next ore type while the remainder of the current ore type is mined and processed. Space will be made available to temporarily store these materials as required.

In the 2006 Feasibility Study, there was significant concern that the supergene and primary ores would oxidize quickly once drilled and blasted. Partially driven by this concern, the 2006 Feasibility Study mine plan used relatively small equipment that could provide blending directly from multiple dig faces in the pit and did not incorporate blending stockpiles. Since the 2006 Feasibility Study, the approach has been taken that ore oxidization is not a significant concern, and the mining approach has changed, incorporating larger mining equipment and relying on re-handling from multiple short term stockpiles for blending.

Nevsun has commenced metallurgical testwork to verify if oxidation via stockpiling will affect flotation recovery. However, the testwork is at a preliminary stage, and no firm conclusions can be drawn from the limited data available.

Table 18-5: Pit Phase Volumetrics

Pit Phase	Ore (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Waste (kt)	Total Mined (kt)	Strip Ratio
Phase 1								
Oxide	828	7.58	19.85	0.10	0.08	659	1,487	0.80
Phase 2								
Oxide	2,332	8.83	30.56	0.09	0.06			
Supergene	124	1.65	46.92	2.04	0.03			
Total	2,456	8.47	31.39	0.19	0.06	6,153	8,608	3.5
Phase 3								
Oxide	1,107	3.81	32.71	0.18	0.11			
Supergene	75	1.21	68.10	6.38	0.09			
Total	1,182	3.64	34.96	0.57	0.11	6,675	7,857	6.6
Phase 4								
Oxide	83	9.41	75.74	0.69	0.02			
Supergene	2,328	0.88	25.11	3.40	0.07			
Primary	42	1.00	28.07	0.80	2.39			
Total	2,453	1.17	26.87	3.27	0.10	927	3,380	1.4
Phase 5								
Oxide	87	2.68	74.68	1.06	0.05			
Supergene	2,188	0.72	42.72	4.43	0.23			
Primary	357	0.67	37.93	0.89	4.25			
Total	2,632	0.78	43.13	3.84	0.77	6,376	9,007	3.4
Phase 6								
Oxide	17	1.77	45.66	0.48	0.09			
Supergene	2,220	0.71	30.66	4.32	0.12			
Primary	155	0.75	33.48	1.13	3.00			
Total	2,392	0.72	30.95	4.09	0.31	14,784	17,176	7.2
Phase 7								
Supergene	366	0.63	23.22	1.97	0.36			
Primary	4,339	0.73	34.95	0.85	4.09			
Total	4,705	0.73	34.04	0.93	3.80	3,997	8,702	1.8
Phase 8								
Supergene	81	0.43	22.96	1.56	0.43			
Primary	4,114	0.65	41.99	0.84	5.62			
Total	4,194	0.65	41.62	0.86	5.52	19,109	23,304	5.6
Phase 9								
Primary	7,273	0.75	52.04	1.11	6.17			
Total	7,274	0.75	52.03	1.11	6.17	58,974	66,247	9.1
Oxide Stockpile at Jan 01, 2011								
	197	3.64					197	
Total								
Oxide	4,651	7.06	29.56	0.14	0.07			

Pit Phase	Ore (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Waste (kt)	Total Mined (kt)	Strip Ratio
Supergene	7,382	0.78	32.68	3.90	0.15			
Primary	16,280	0.72	44.40	0.97	5.40			
Total	28,313	1.78	38.90	1.60	3.15	117,653	145,966	4.16

Table 18-6: Annual Mine Plan

Year		2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	Total
Oxide	(kt)	2,000	2,000	652											4,652
NSR	$/t	223.33	222.43	145.21											212.00
Au	g/t	7.69	7.35	4.24											7.06
Ag	g/t	16.07	39.73	39.70											29.56
Cu	%	0.08	0.13	0.33											0.14
Zn	%	0.07	0.08	0.05											0.07
Supergene	(kt)			1,616	2,400	2,400	965								7,381
NSR	$/t			163.84	178.53	192.33	122.70								172.50
Au	g/t			1.03	0.73	0.71	0.62								0.78
Ag	g/t			31.54	34.90	35.80	21.33								32.68
Cu	%			3.63	4.08	4.40	2.68								3.90
Zn	%			0.03	0.16	0.20	0.21								0.15
Primary	(kt)						1,435	2,400	2,400	2,400	2,400	2,400	2,400	445	16,279
NSR	$/t						79.06	84.51	64.28	99.21	110.03	110.49	103.46	106.76	94.21
Au	g/t						0.72	0.73	0.63	0.70	0.72	0.78	0.76	0.78	0.72
Ag	g/t						35.77	39.05	28.90	45.72	51.30	52.65	51.56	57.08	44.40
Cu	%						0.92	0.91	0.63	0.92	1.11	1.11	1.11	1.12	0.97
Zn	%						4.12	4.52	3.55	6.23	6.53	6.52	5.69	5.92	5.40
Total Ore	(kt)	2,000	2,000	2,268	2,400	2,400	2,400	2,400	2,400	2,400	2,400	2,400	2,400	445	28,313
NSR	$/t	223.33	222.43	158.48	178.53	192.33	96.61	84.51	64.28	99.21	110.03	110.49	103.46	106.76	133.97
Au	g/t	7.69	7.35	1.95	0.73	0.71	0.68	0.73	0.63	0.70	0.72	0.78	0.76	0.78	1.78
Ag	g/t	16.07	39.73	33.89	34.90	35.80	29.97	39.05	28.90	45.72	51.30	52.65	51.56	57.08	38.90
Cu	%	0.08	0.13	2.68	4.08	4.40	1.63	0.91	0.63	0.92	1.11	1.11	1.11	1.12	1.60
Zn	%	0.07	0.08	0.04	0.16	0.20	2.55	4.52	3.55	6.23	6.53	6.52	5.69	5.92	3.15
Waste	(kt)	5,419	6,300	5,481	7,802	9,379	14,521	14,128	16,464	15,182	14,944	5,474	2,474	86	117,653
Total Mined	(kt)	7,419	8,300	7,749	10,202	11,779	16,921	16,528	18,864	17,582	17,344	7,874	4,874	531	145,966
Strip Ratio	(W/O)	2.71	3.15	2.42	3.25	3.91	6.05	5.89	6.86	6.33	6.23	2.28	1.03	0.19	4.16

Figure 18-4: Annual Mill Feed

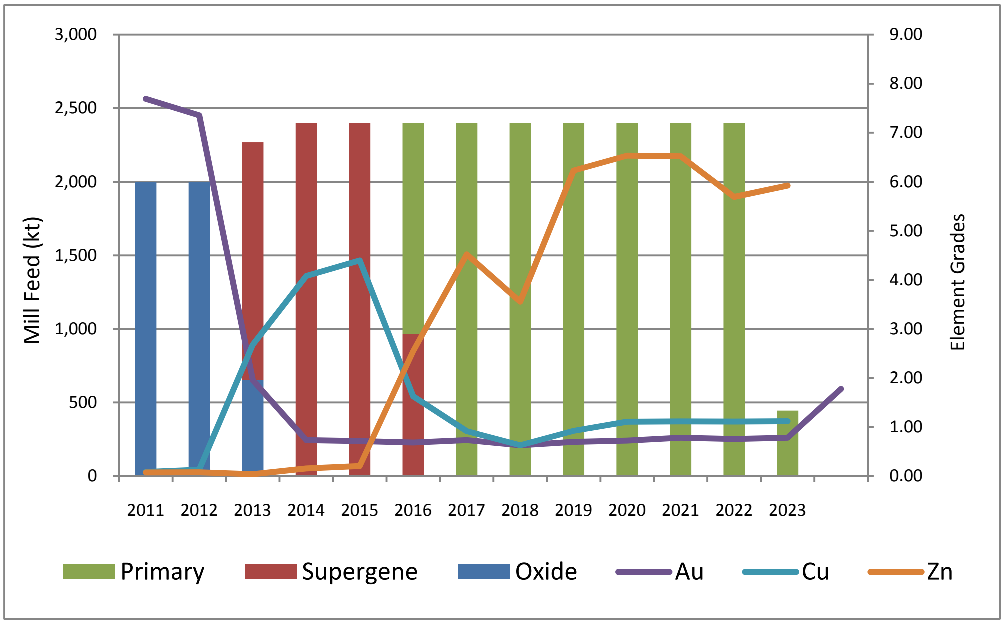


Figure 18-5: Annual Material Movement

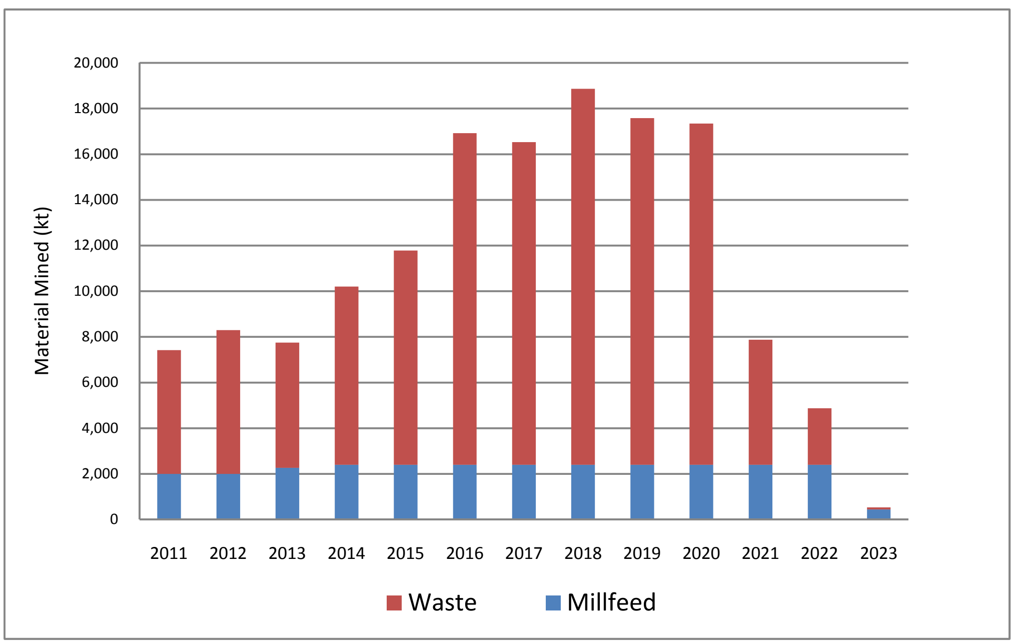
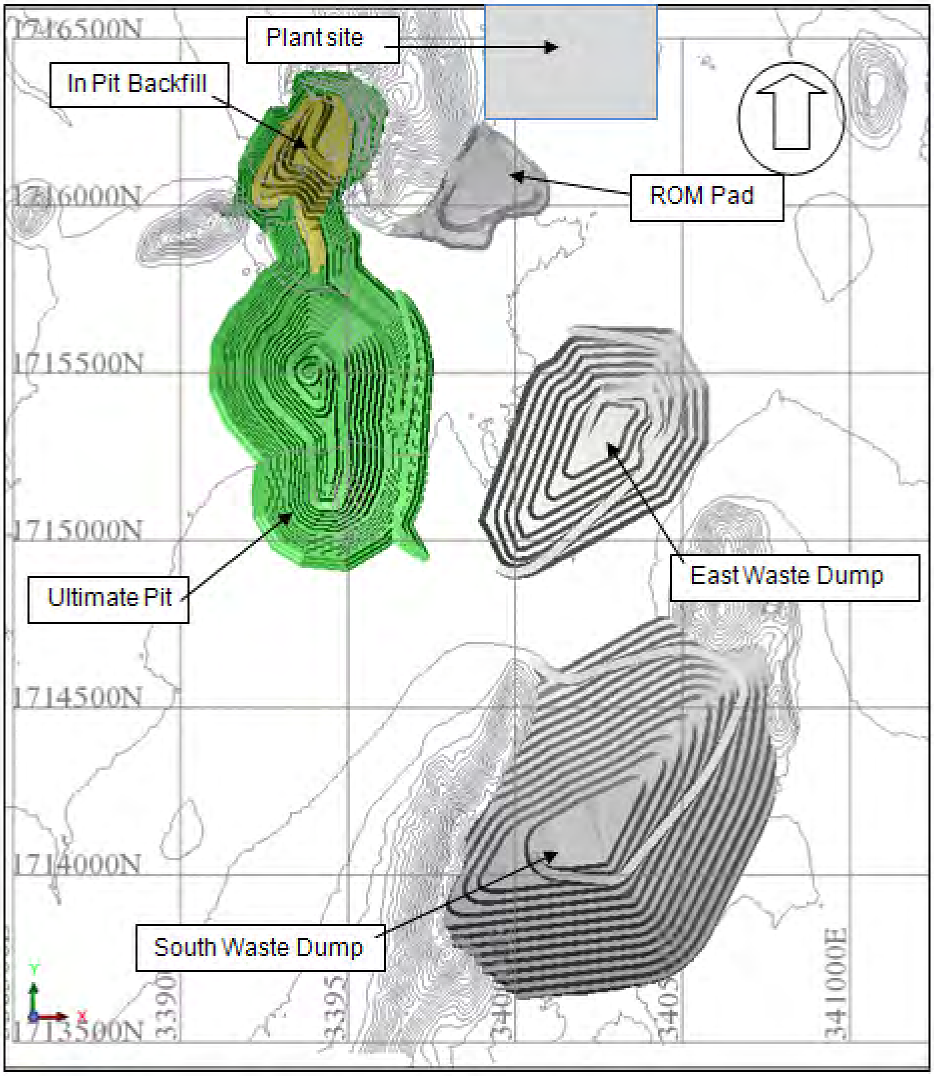


Figure 18-6: Location Plan, Bisha Pit and Waste Dumps



18.1.5 Mining Equipment

The current production equipment fleet as of 01 January 2011, with 2011 purchases are shown in Table 18-7. The 2011 purchases were planned with knowledge of the pending northeastern wall layback. It appears that this fleet will be sufficient for the oxide phase of the mine life, including a very high level estimate of the potential tonnage in the northeastern wall layback. The first of the scheduled additions and replacements occurs in 2014.

Table 18-7: Production Fleet on Site or Ordered

Equipment Unit	Current Fleet	2011 Purchases	Fleet at mid-2011
Drills			
ROC L8 Drill	2	0	2
Pantera 1500 Drill	0	1	1
Loading Equipment			
Terex RH40 Shovel	1	1	1
Terex RH40 Excavator	1	2	2
CAT 990H Loader	1	1	1
Haul Trucks			
CAT 775 Truck	6	4	10

To determine the number of additional equipment units required for each major fleet, productivities were calculated by first principles based on estimated annual operating hours and mechanical availabilities. To allow for inefficiencies, a 50-minute operating hour was applied to all equipment. In addition, a 90% truck availability was applied to the loading units. At peak production which occurs in year 2018, the equipment requirements are four excavators, one front-end loader, three drills, and eighteen trucks. Equipment replacements were based on the following projected equipment lives in thousands of operating hours: 60 for excavators, 50 for trucks and the front end loader, and 35 for drills. Production equipment additions and replacements beyond the current plus ordered fleet are shown in Table 18-8.

Projected ancillary equipment requirements included two road graders, four tracked dozers, three water trucks, one wheel loader, support trucks, utility loader and tyre manipulator, lighting plant, tractor-trailer, crew bus and pickup trucks.

18.2 Process Plant

The 2 Mt/a oxide processing facility achieved commercial production in February 2011. Over a 30 day period spanning January and February 2011, plant throughput averaged approximately 5,250 t/d with a peak of 6,560 t/d.

The oxide plant facilities include a primary crusher, SAG and ball grinding mills, cyanide leach/carbon-in-pulp (CIP) circuit, cyanide destruction circuit, refinery to produce doré, tailings thickener, tailings discharge system and the necessary reagent, water and air systems. A simplified schematic of the process flowsheet is provided in Figure 18-7.

Process plant throughput projections for the newly commissioned oxide plant designed by SENET are provided in Table 18-9. Projected oxide feed grades and recoveries are listed in Table 18-10.

Table 18-8: Production Equipment Additions and Replacements

Equipment Unit	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Drills										
ROC L8 Drill	1			1						
Pantera 1500 Drill										
Loading Equipment										
Terex RH40 Shovel		1								
Terex RH40 Excavator			1							
CAT 990H Loader										
Haul Trucks										
CAT 775 Truck		3	5	1	1	3	3			

Figure 18-7: Process Flowsheet

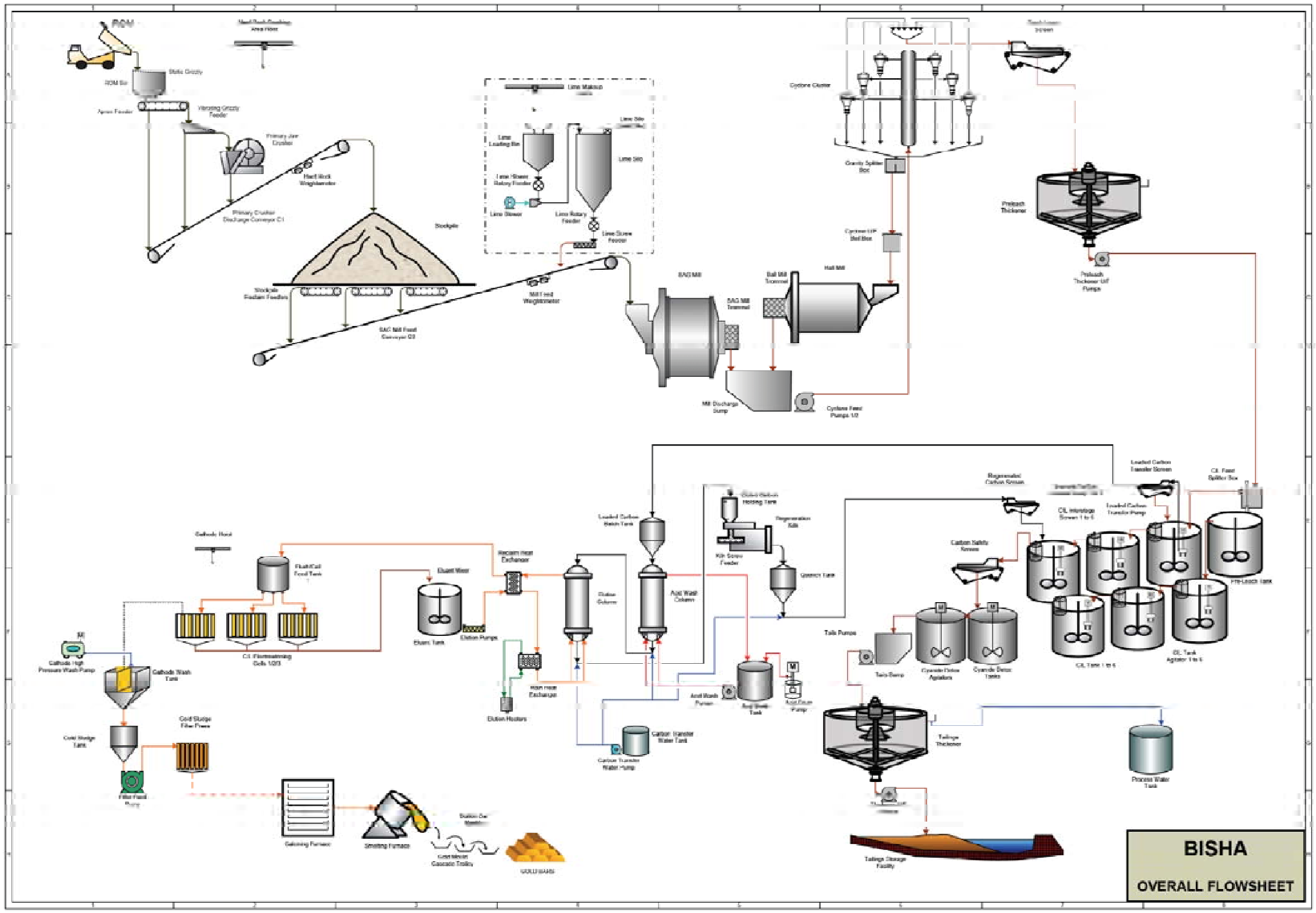


Table 18-9: Process Assumptions for Oxide Throughput

General	Units	Oxide
Yearly Throughput	(Mt)	2
Operating Days/Year	(d)	365
Overall Plant Availability	(%)	91.3
Throughput Rate	(t/ operating hour)	250
	t/ operating day (average)	5,479

Table 18-10: Oxide Treatment Process Plant Feed Assumptions (2008 SENET)

Run-of-Mine Ore	Units	Oxide
Ore Moisture Content	(% wt)	5
Max Lump Size	(Mm)	800
Ore Specific Gravity	(t/m$^{3)}$	3.17
Ore Bulk Density	(t/m$^{3)}$	1.90
Ore Feed Grades	(g/t Au)	8.0
	(g/t Ag)	32.9
	(%Cu)	0.10
	(%Zn)	0.08
Bullion Recovery	(% Au)	87
	(% Ag)	36

18.3 Proposed Additional Processing Facilities

At Bisha, there exist three different types of mineralization: oxide, supergene and primary; each requiring different processing techniques and equipment. The plan outlined in the 2006 Feasibility Study was to mine and process each zone in succession starting with the top oxide zone now in production. Before the oxide mineralization is exhausted, the additional supergene mineralization process equipment will be installed and commissioned, so a smooth transition can be made from the oxide mineralization to the supergene mineralization. There will be some transitional material that is a mixture of oxide and supergene mineralization that will be mined as the oxide is depleted.

The current plan is to treat this material in the copper flotation circuit designed for the supergene material prior to treating the tailings in the cyanide leach circuit with the intent of maximizing copper recovery and preventing any gold losses that may be present in the mixed material. The cyanide leach circuit will not be operated after the transitional ore is depleted.

Similarly, before the supergene mineralization is exhausted, the additional equipment required to process the primary mineralization will be installed and commissioned to permit a smooth transition from supergene mineralization to primary mineralization with minimum interruption and shutdowns.

Additional equipment for the supergene plant will include the supergene mineralization roughing and cleaning flotation circuits, a copper concentrate regrind mill, copper concentrate thickener and pressure filter, copper concentrate load-out building, copper flotation reagent systems, flotation air blowers and pressure-filter air compressors. According to the lasted mine plan, some stockpiling of supergene and primary mineralization will occur, however, due to the previously indicated highly reactive nature of the supergene and primary material noted in the 2005 SGS test program, it is recommended that the oxidation effects on flotation recovery be evaluated.

For treatment of primary mineralization, additional equipment will include zinc roughing and cleaning flotation circuits, zinc concentrate regrind mill, zinc concentrate thickener and pressure filter, zinc concentrate load-out building, zinc flotation reagent systems, additional zinc flotation air blower and zinc pressure filter air compressor.

Based on the current mine plan, the process plant feed rate will transition from oxide to supergene feed during the second quarter of 2013, with an associated increase in throughput for the remainder of the year. The throughput rate in 2014 is projected to be 2.4 Mt/a with 100% supergene mineralization as feed material to the mill. As this represents a 20% increase in the feed rate, a number of modifications to the current plant equipment will be required. Based on the work completed to date the following anticipated modifications have been identified:

- Increase in pipe size in the grinding classification circuit and pre-leach thickener/flotation feed line for the additional volume
- Increase size of cyclones in the grinding circuit
- Increase motor size on the grinding thickener/flotation feed pipeline
- Addition of two extra pumps installed for the tailings thickener underflow pipeline.
- Increase the pumping capacity in the gland seal water system.

These modifications are not viewed as extensive or capital intensive.

Updated life-of-mine process plant throughput projections are provided in Table 18-11 and recovery assumptions are provided in Table 18-12.

Table 18-11: LOM Process Plant Throughput

General	Units	Oxide	Supergene	Primary
Yearly Throughput	(Mt)	2	2.4	2.4
Operating Days/Year	(d)	365	365	365
Overall Plant Availability	(%)	91.3	91.3	91.3
Throughput Rate	(t/ operating hour)	250	300	300
	t/ operating day (average)	5,479	6,575	6,575

Table 18-12: Life-of-Mine Process Plant Feed, Recovery and Concentrate Grade Assumptions

Run-of-Mine Ore	Units	Oxide	Supergene	Primary
Ore Feed Grades	(g/t Au)	7.06	0.78	0.72
	(g/t Ag)	29.56	32.68	44.4
	(%Cu)	0.14	3.9	0.97
	(%Zn)	0.07	0.15	5.4
Bullion Recovery	(% Au)	87	—	—
	(% Ag)	36	—	—
Copper Concentrate	% Au Rec	—	56	36
	% Au Rec	—	54	29
	% Cu Grade	—	30	25
	% Cu Rec	—	88	85
	% Zn Grade	—	—	3.9
	% Zn Rec	—	—	2.1
Zinc Concentrate	% Au Rec	—	—	9
	% Au Rec	—	—	20
	% Cu Grade	—	—	0.3
	% Cu Rec	—	—	3
	% Zn Grade	—	—	55
	% Zn Rec	—	—	83.5

18.4 Site Infrastructure

Site plans covering site facilities were shown in Figure 5-2. Power and communication descriptions are included in Section 5 of this Report. Constructed on-site infrastructure includes:

- Administration building. Offices and cubicles are provided for the mine management and supervisory staff as well as for human resources, accounting, procurement, information technology and safety staff
- Camp. The permanent camp complex is located approximately 1 km to the north west of the plant site. The camp houses approximately 300 people and includes dormitories, kitchen and dining facilities, recreation facilities, laundry, water treatment, sewage treatment, incinerators and emergency power facilities

- Maintenance workshop, warehouse and laboratory complex. The complex is located at the south end of the plant area adjacent to mine access road
- Explosive magazine and ANFO mixing plant. The ANFO facility is located 1,000 m southeast of the process plant site and 450 m east of the ultimate footprint of the waste dump
- Fuel storage. Fuel is stored in an HDPE lined and bunded tank farm
- Concentrate storage and reclaim facility
- Power plant
- Process control system
- Communications system
- Water supply (potable and process).

Off-site infrastructure is required to support the export of copper and zinc concentrates. The Massawa port site is planned on the site of an existing cement production facility, adjacent to an existing jetty on the north shore of Khor Dakliyat Bay. Facilities required at the port site include storage areas for the zinc and copper concentrate, conveyor-and-plow system, travelling hopper, transfer tower, and shiploader.

18.5 Planned Manpower

In 2011, the Bisha operation is forecast to employ 750 personnel. Employment will peak at 810 in Year 2018. Senior plant operations personnel and supervisors will be expatriates for the first few years of operation; however the intention will be to train Eritrean personnel so that they may move into these positions.

18.6 Geotechnical Considerations

The Bisha open pit is expected to be approximately 1.4 km long and 550 m wide and consists of two coalescing pits, a shallow pit to the north and a much deeper pit to the south. The open pit is widest in the southern sector with a highwall approximately 200 m high. A wall with approximately the same height will be developed in the northeastern sector of the pit where a hill side will be partially excavated.

Rock mass characterization has indicated that excavation of the Bisha Pit will be challenging, with various materials types having to be excavated, from overburden and weak rocks to moderate competent rocks. Pit slopes will be controlled by structural features at the bench face level and rock mass strength at inter-ramp and overall pit scale level. Relative to bench stability, kinematic analyses indicate that wedges may form at steep benches causing partial bench loss and rubble accumulation.

This failure mechanism is expected to be particularly active at the south side of the midway "nose". Moreover, toppling failure may also occur particularly on walls facing east and west. The effect of toppling at the bench scale would be to the local hazard of falling blocks, and this should be controllable with good pit limit blasting and excavation.

At bigger scales such as inter-ramp scales, and overall, toppling failures (if present) are not expected to be catastrophic, tending to involve large, slow displacements. Therefore, toppling failures (if any) are not expected to disrupt operations and can be managed by matching the speed of excavation to the rate of deformation.

Additional investigation into the presence, characteristics and particularly continuity of major and intermediate scale structures and their impacts to pit wall (bench face, inter-ramp, overall slope) designs is strongly recommended.

Relative to overall pit stability, limit-equilibrium analyses have demonstrated stability for fully drained condition. The limit-equilibrium analysis also shows the detrimental effect of groundwater as the saturation condition increase from fully drained to completely saturated slope. Therefore, pit slope depressurization is major requirement in order to achieve the pit slope angles provided in this report. In some situations, natural drainage of the rock walls as excavation proceeds is adequate for achieving the desired goals for slope stability. However, in many cases the rate of pore pressure dissipation achieved by natural drainage, as the pit is excavated, is too slow and the target depressurization levels is not achieved. For this last case implementation of active pore pressure control, that is pumping wells and/or horizontal drains, is required.

There are several considerations for deciding whether to implement and active mine depressurization program or not. These considerations involve practical aspects (achievability of desired results within the mining time frame) and economical aspects (the impact of strip ratio reduction on the mine economics). In any case a detailed hydrogeological study is deemed necessary and strongly recommended.

18.6.1 Bench Failures

In April 2010, AMEC was informed by BMSC of planar and wedge bench failures that developed on the northeastern pit wall. As bench failures persisted, on August 2010

BMSC issued a memorandum summarizing various geotechnical aspects of these bench failures, including structural data collected at the bench level.

Following these events BMSC requested geotechnical support from AMEC. A site visit was completed from 3–7 October 2010. Conclusions arising were that the bench-scale stability of the northeastern wall is problematic as several bench failures have taken place. Field assessment have suggested that bench failures are associated with low dipping medium to high persistence shear planes that were not identified at the initial site investigation phase. This was because at the time of the initial site investigation supporting the 2006 Feasibility Study design, access to the northeastern sector hill formation was difficult and geotechnical drilling was therefore limited to the base of the hill. Failure development is facilitated by weak character of the rock mass and may also be reflecting blast damage.

Bench failures are generally located at the central portion of the northeastern wall. The north sector of the pit is less affected but still displays some bench failures. Bench geometry modification recommendations included increasing the bench width, reducing the bench face angle or a combination of both. Either of these measures results in a reduction of the inter-ramp and overall slope angles and may affect the mine plan. Two bench configurations were proposed to minimize the amount of back-break, leaving effective benches as catchment areas for rockfall events. Additionally, it was strongly recommended to monitor rock displacements as a preventive measure for overall slope failures. The following monitoring was recommended:

- Visual inspections
- Cross-crack measurements
- Slope survey monitoring.

On 04 January, 2011, a moderately large failure occurred in the same region of the highwall. The failure was successfully predicted from the wall movement monitoring data. A barricaded safety step-out restricted access to the area. No injuries or equipment damage occurred. Mining activities have continued behind the safety step-out. Additional geotechnical drilling is underway to support further geotechnical and hydrological analysis. The resulting design recommendations will be used to design a layback of the wall, to allow mining of the resources below the safety step-out.

AMEC does not consider the northeastern wall slope instabilities to be a fatal flaw. Remediation should not be insurmountable and such events are commonly encountered in the early phase of operations while as a mining company better understands the geotechnical competence of the rock mass and data are based on exposures rather than drill core.

18.7 Hydrological Considerations

18.7.1 Pit Dewatering

For the purposes of the 2006 Feasibility Study, hydrogeological data were derived from several packer tests that were conducted during one of the drilling programs in the proposed pit area. The testing indicated that permeabilities are generally low in the rock mass below a depth of 60 m, but are generally high enough that pit inflows can be expected.

Water quality in the pit area is not known in detail, but a single field test completed in 2005 indicated a pH of approximately 4.23 and a conductivity of over 4,000 µS. Some of the pit inflow water may need to be treated or blended with fresh water prior to use in the process circuit.

It was recommended that the use of pit water in the process be further evaluated before detailed designs were produced, to be able to incorporate the requirement of treatment facilities if necessary.

The pit could receive approximately 43,000 m^3 of direct precipitation and about 6,000 m^3 of runoff in a 200-year precipitation event. This volume would require 20 days at 30 L/s to pump out of the pit.

With mine development, additional hydrological investigations are currently underway.

18.7.2 Run-off Water

Run-off from waste dumps will potentially be unsuitable for release. Hence, all waste dump contact runoff will be collected. The footprint of waste dumps around the pits will reach their maximum extent at the end of Year 5. During a 200-year precipitation event, approximately 15,000 m^3 of water could be stored in the run-off collection ponds. The run-off water that enters the ponds will be pumped to the tailings facility where it will evaporate.

Diversions will be constructed in the vicinity of proposed infrastructure such as the plant site. Run-off will be collected in storm drains and will be collected for use in the process plant if the water is not contaminated. Facilities with the potential to spill contaminants require curbs or berms to contain any spills.

The tailings line that crosses the Shatera River will also require flood protection in areas where the line could be affected by moving water and any debris it may transport.

18.7.3 Fereketatet River Interception and Diversion

The hydrology of the project site is such that high intensity, short duration rainfall events occur during the rainy months, resulting in flash flooding situations. The open pit is located in the ephemeral drainage of the Fereketatet River. Diversion works are being constructed to intercept flow in the Fereketatet River during surface runoff events to prevent water entering the pit during development and operations. The diversion works will consist of a dyke across the river upstream (southeast) of the proposed pit, which ultimately forms part of the project long term southern waste dump. This dyke will intercept surface flow and, in extreme events, divert the flow over a dividing ridge and into the adjacent Shatera River to the east. Additional diversion channels will also be formed to divert upstream catchments west to the Mogoraib river catchment and east to the upper regions of the Shatera. Permits are in place and construction of the diversion is underway.

18.8 Waste Rock Storage Facilities

Over the life of the mine, approximately 146 Mt of waste rock will be produced and placed in two waste rock storage facilities (WRFs) to the northwest and southeast of the open pit (refer to Figure 18-6). The northwest WRF covers nominally 80 ha and the southeast WRF covers nominally 90 ha.

An operational scheduling plan has been prepared for placement of the rock within the dumps that allows potential acid rock drainage (ARD) issues to be appropriately managed during operations while providing some flexibility for closure.

Approximately 85% of the waste rock to be generated at the site is likely to be acid generating. The remaining 15% may be suitable for construction material if it can be separated. This material will be directed to a dedicated section of the waste dumps to be ultimately used for closure and rehabilitation works.

18.9 Tailings Management Facility

Approximately 17.8 Mt of tailings will be present in the TMF at mine closure, assuming a 10 year mine life. The final footprint area of tailings in the TMF will be approximately 162 ha. The current TMF design is optimized around the 2010 mine plan, and will require revision following the planned 2011 mine plan optimization.

AMEC considers it a reasonable expectation that even with the optimization for the 2011 mine plan, and the likelihood of additional mine life and production scenarios, that the tailings dam, when re-designed, can support the changes to the production plan.

The tailings impoundment facility is located approximately 1 km to the northeast of the plant site. The TMF is confined by local ridges to the west and east, and by starter embankments formed to the north and south. The geometry of the impoundment is based on projected tailings production and the provided site topography. The impoundment was created by construction of a granular/low permeability northern embankment and southern low wall abutting the western and eastern ridges.

The tailings impoundment site is underlain by weathered schists and low permeability bedrock (massive metavolcanic rock). The TMF has been fully lined with a HDPE geomembrane in addition to the provision of underliner and overliner drains. A seepage cut-off has also been provided below the main embankment.

The starter embankment was designed for two years of tailings storage capacity (4 Mt equivalent to 3.05 Mm^3), with a required fill volume of 1.3 Mm^3. The remaining life-of-mine tailings storage will be contained with initially downstream raised embankments requiring additional fill.

Quarried non-acid generating (NAG) rock and low permeability earthfill was used to construct the rockfill shell of the two-year starter dam and will also be used to raise the dam in subsequent years. Further work is required to confirm the source of NAG construction material to cap the tailings.

Each of the three ore types, oxide, supergene, and primary has different specific gravities, which for the purposes of the tailings dam design were selected as 3.41 for oxide, 5.17 for supergene, and 4.71 for primary, respectively; and will correspondingly result in different average in situ densities within the tailings impoundment.

Tailings will be thickened prior to being pumped to the tailings facility. The average thickened tailings product will be approximately 50% solids by weight for the initial two years of Oxide process production, changing to 46% and 58% respectively when the supergene and primary ores are processed. Tailings will be distributed by spigotting along the full length of the embankment, to create beaches adjacent to the dam. Free water from discharged tailings and seasonal run-off will collect in a small pond in the centre of the impoundment, from which it will be recovered to the process plant when not lost to evaporation.

A cyanide destruction circuit was included in the process plant design to reduce the amount of cyanide in the tailings pond so it will be non-toxic to birds and wildlife.

18.10 Environmental Considerations

The environmental reviews and supporting studies completed to date are discussed in Section 4 of this Report.

18.10.1 Closure

Knight Piesold, an independent consulting firm, developed a conceptual closure and reclamation plan (CCRP) in 2009. Closure considered:

- TMF: non-PAG cover to prevent erosion from wind and surface water run-off; tailings delivery pipelines, power lines, and associated infrastructure will be decommissioned and cleaned of potentially hazardous materials; roads required to access the TMF will be decommissioned unless required to provide access for post-closure inspection, maintenance, and monitoring
- WRFs: at closure the final slopes will be no steeper than 3:1; non-PAG cover to prevent erosion from wind and surface water run-off; for budgetary purposes revegetation has been included in the closure costs; drainage from both of the WRFs will be directed towards the open pit through lined channels
- Open pit: there will be a pit lake upon cessation of operations, and a preliminary estimate of water quality was developed to support the closure plan; the preferred closure option for the Bisha Pit is to berm the pit and monitor the water quality during filling to identify any potential risk to avian life during this time; man-made equipment and materials (e.g., mining equipment, dewatering apparatus, cables) with salvage value will be removed from the open pit; materials and equipment with no salvage value will be cleaned of any potentially hazardous substances and disposed of in the open pit or in the non-hazardous landfill; access into the open pit will be blocked by installation of a rock boulder barrier across the access ramp/s into the pit. At closure, a security fence or an earthwork berm or a combination of both will be constructed
- Process plant: demolition and removal of equipment and structures to leave the property in an environmentally sound condition that will sustain accepted post-closure land uses; as part of demolition of the process plant infrastructure, allowance is made for the handling and disposal of potentially hazardous materials (e.g. hydraulic oil, gasoline, etc.) encountered during the dismantling process. Non-hazardous debris will be disposed of within the open pit or in the non-

hazardous landfill. Potentially hazardous materials will be properly containerized and shipped offsite for recycling or disposal at an approved facility

- Ancillary facilities: will be decommissioned, cleaned, and removed from site following similar procedures to those envisaged for the plant; equipment and building systems will be inspected and cleaned of potentially hazardous materials; if practicable, materials will be salvaged, otherwise the facilities will be demolished and the debris will be disposed of in the open pit or in the non-hazardous land fill
- Roads: The main access road to the mine site and any other onsite access roads will be left in place for a minimum period of five years post-closure, to allow access to the site for post-closure maintenance and monitoring activities; roads that are no longer required will be permanently decommissioned.

Closure and post-closure monitoring will document the progress of the closure and reclamation effort. The elements of these monitoring programs will include:

- Inspection of the physical conditions (e.g., for evidence of erosion and landslides) at the end of the initial rainy season post-closure
- Inspection of the plantings after the first year post-closure
- After two years, evaluation of the effectiveness of the reclamation effort (e.g., number and type of plant species, plant heights, productivity)
- Demonstration that water quality objectives are met
- Assessment of the adequacy and performance of drainage structures and sediment control systems.

Closure and post-closure monitoring and control programs will be conducted twice per year (dry and wet seasons) for a period of five years after closure has been completed. In the event that deficiencies in vegetation establishment are identified, appropriate mitigation measures will be taken to correct these deficiencies.

Final reclamation of the pit, waste rock facilities, yards, and roads is estimated to cost approximately US$14,800 per ha, for a total of approximately US$3.97 M.

Reclamation of the TMF, which includes the costs for the placement of a 40 cm cover, an evaporation pond, and the closure spillway, is estimated to be approximately US$5.6 M. Revegetation is estimated to be approximately US$370,000.

No salvage credits have been assumed for the closure costing.

Demolition and reclamation of the plant and other infrastructure is estimated to be approximately US$2.5 M. Closure and post closure monitoring costs for a period of 10 years (five years of TMF drain-down and five years of post closure monitoring) is estimated at US$400,000.

The total reclamation cost is estimated at US$19.44 M including administrative costs and a 15% contingency (this includes the cost of reclaiming both the WRFs).

18.11 Markets

Nevsun has negotiated contracts with two refineries for the sale of the gold–silver doré. Negotiations are underway for the sale of concentrates to be produced from the future phases of the Project.

Normal commercial terms are included in the refinery contracts, and are similar to typical industry-standard doré contracts.

Terms contained within the concentrate sales contracts are likely to be typical of, and consistent with, standard industry practice, and be similar to such contracts elsewhere in the world.

18.12 Taxation

Information on the taxation for the Project is sourced from the 2006 Feasibility Study:

Taxation rates are described in the Proclamation No. 69/1995 Proclamation to Provide for Payment of Tax on Income from Mining Operations. A holder of a mining licence shall pay income tax on the taxable income at a rate of 38%. Taxable income is to be computed on a historical accrual accounting basis by subtracting from gross income for the accounting year by taking into consideration all allowable revenue, expenditure, depreciation which, for tax purposes is deductible straight-line over a four year period,, re-investment deduction and permitted losses.

If any licensee transfers or assigns, wholly or partially, any interest in the licence, the proceeds shall be taxable income to the extent that such consideration exceeds the amount of his un-recovered expenditure.

Withholding taxes and personal income taxes of non-residents of Eritrea are identified within the proclamation. If the licensee contracts a company or person, who is not resident in Eritrea for services in Eritrea, the licensee will pay taxes on behalf of such a person. Taxes will be paid at the rate of 10% on the amount paid. For the purposes of this article in the proclamation, a person is temporarily present in Eritrea if he performs work in the country for more than 183 days in any accounting year. The compensation received by an expatriate employee of the licensee or his contractor shall be subject to an income tax at a flat rate of 20%.

The holder of a Mining Licence producing exportable minerals can open and operate a foreign currency account in Eritrea and retain aboard a portion of his earnings to be able to pay for importation of machinery, pay for services, for reimbursement of loans and for compensation of employees and other activities that may contribute to enhancement of the mining operations.

Proclamations 64/1994, Sales and Excise Tax, and 117/2001 Amend the Sales and Excise Tax No. 64/1994, require the licensee to withhold 10% of invoiced amounts for services rendered within Eritrea by non-resident service providers.

18.13 Financial Analysis

The results of the economic analysis represent forward-looking information (cashflows, net present value, internal rate of return, production rate, and total metal produced) that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

18.13.1 Basis of Financial Analysis

Financial analysis of the Bisha Project was carried out as an update to the 2006 Feasibility Study, using a discounted cash flow (DCF) approach. This method of valuation requires projecting yearly cash inflows (or revenues) and subtracting yearly cash outflows (such as operating costs, capital costs, and royalties). The resulting net annual cash flows are discounted back to the date of valuation and totalled in order to determine the net present value (NPV) of the project at selected discount rates. For discounting purposes, cash flows are assumed to occur at the end of each period.

For the Bisha Project, the majority of the capital cost has already been spent; therefore, the cash flows from 2011 and onwards were discounted. The Project was equity financed and no debt was required.

The payback period is the time calculated from the start of mine production until all initial capital expenditures have been recovered. The economic analysis includes sensitivities to variations in operating costs, capital costs, and metal prices. All monetary amounts are presented in United States dollars (US$).

Key assumptions supporting the financial analysis were:

- Mineralization will be processed at an average rate of 2,177 kt/a over a planned mine life of approximately 13 years
- A 5% royalty is payable, for precious metals, on the net smelter return
- A 3.5% royalty is payable, for base metals, on the net smelter return
- The following general smelter terms:
  - Average doré treatment charge for gold oxide ore: US$0.30/oz
  - Gold refining charge for gold oxide ore: US$0.20/oz Au
  - Copper treatment charge for supergene copper ore: US$75.00/dmt
  - Copper refining charge for supergene copper ore: US$0.750/lb
  - Zinc pay factor for zinc primary ore: 85%
  - Gold pay factor for zinc primary ore: 70%.
- Long-term base case metal price assumptions of:
  - Gold: US$1,015/oz
  - Copper: US$2.40/lb
  - Silver: US$15.85/oz
  - Zinc: US$0.92/lb.
- Life-of-mine average operating costs assumptions of:
  - Mining: US$2.11/t mined (includes labour, fuel and consumables)
  - Process (includes ore re-handling costs):
    - Oxide Ore: US$24.35/t milled
    - Supergene Ore: US$21.89/t milled
    - Primary Ore: US$22.00/t milled
    - Process costs were provided by Nevsun Resources. Process costs are approximately 25% higher compared to the 2006 Feasibility Study. This represents a reasonable escalation from 2006 figures.
  - G&A: US$4.68/t milled. These costs were provided by Nevsun and, in AMEC's opinion, are reasonable for the Project.
  - Port operating costs were revised for the financial analysis in this Report. Total port operating costs were increased by 25% from the 2006 Feasibility

Study figures. Average annual port operating costs are estimated at US$1.8 M.

- Land transportation and Ocean Freight costs were increased by 25% to:
  - Land Concentrate Freight: US$65/wmt
  - Ocean Freight: US$43.75/wmt.
- Port charges are assumed to be US$7.00/wmt (17% increase from the 2006 Feasibility study).

- Capital cost assumptions of:
  - From 2009–2010, the capital costs were US$ 237.2 M (provided by Nevsun Resources Ltd.). These costs include Owners' costs and represent an increase of approximately 28% from the Feasibility study figures. The increase in costs is within the expected range.
  - From 2011–2015, the capital cost estimate is US$119.6 Million. This represents a 30% increase from the Feasibility Study estimate.
  - Sustaining capital: US$27.5 M
  - Sustaining costs were provided by Nevsun. The mining portion of the sustaining costs was updated by AMEC. The balance of the sustaining costs was escalated by 25% from the Feasibility Study, which represents a reasonable escalation.
  - Closure and reclamation costs: US$19.3 M. These costs are based on Knight Piésold figures. In Year 5 of operations, US$1.82 M is required for reclamation of the southeast waste rock facilities, and the majority of the remaining costs will be spent at closure.
  - Salvage value: US$8.9 M. Assumed to be 2.5% of total capital costs.
- A working capital allocation of one month of operating costs for the Oxide phase, and two months of operating costs for the base metal phase.
- The base-case economic analysis is based on 100% equity financing.
- The base case economic analysis included no inflation.
- AMEC does not provide expert advice on taxation matters. A simple tax model was generated for the Bisha Project. An income tax rate of 38% was applied over the life-of-mine. The total tax payable over the life of mine is estimated US$749 million based on this assumption. It was assumed that historic costs were deductible for tax purposes. These costs included $28 M for historic exploration and $8.6 M in deferred financing costs.

18.13.2 Results of Financial Analysis

The base case results pre-tax and post-tax are indicated in Table 18-13. Table 18-14 provides the full cashflow on an annualized basis. From 2011 to 2015 US$120 M is required for development of the supergene and primary ores. Total sustaining costs required are US$27.5 from 2011 to 2015.

Payback considerations were not undertaken, as for the purposes of the financial analysis, capital costs to 31 December 2010 were considered to be "sunk" capital. The financial analysis supporting the 2006 Feasibility Study indicated that at the time, payback would be achieved after 2.6 years of commencement of full production.

Table 18-13: Pre-Tax and Post-Tax Results, Financial Analysis

Pre-Tax		NPV (2011 and onwards)	Capital Costs (2009–2010)	Pre-Production Operating Costs (2010)
NPV 10%	US$ million	**1395**	**($240)**	**($18)**
Payback	Years	**0.8**		
After Tax		**NPV (2011 and onwards)**	**Capital Costs (2009–2010)**	**Pre-Production Operating Costs**
NPV 10%	US$ million	944	($240)	($18)
Payback	Years	1.2		

Table 18-14: Cashflow Summary Table

		Life of Mine	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Net Smelter Return																	
Copper	US$000	1,541,063	0	0	0	0	211,317	353,117	380,690	135,751	70,355	48,282	70,511	84,867	85,270	85,130	15,773
Gold	US$000	1,145,932	0	0	434,150	414,767	105,281	27,259	26,128	21,101	19,341	16,819	18,285	18,826	20,444	19,748	3,782
Silver	US$000	186,026	0	0	5,838	14,422	15,934	18,398	18,652	12,272	14,361	10,327	15,385	17,955	18,775	19,552	4,154
Zinc	US$000	751,820	0	0	0	0	0	0	0	50,554	92,838	72,987	127,974	134,186	133,887	116,859	22,537
Total net smelter return	***US$000***	***3,624,842***	***0***	***0***	***439,988***	***429,189***	***332,532***	***398,774***	***425,469***	***219,678***	***196,895***	***148,415***	***232,155***	***255,835***	***258,375***	***241,289***	***46,246***
Operating Costs																	
Mining																	
Mining Cost	US$000	317,384	0	13,950	23,203	19,782	16,714	20,029	23,205	30,605	31,755	32,838	33,580	32,832	20,910	16,362	1,620
Process																	
Process power Cost	US$000	26,172	0	1,939	24,234	0	0	0	0	0	0	0	0	0	0	0	0
Process Cost (includes power from 2012)	US$000	610,745	0	2,275	24,173	48,700	51,250	52,536	52,536	52,694	52,800	52,800	52,800	52,800	52,800	52,800	9,781
Port costs	US$000	19,749	0	0	0	0	1,771	1,843	1,863	1,774	1,782	1,755	1,821	1,834	1,799	1,814	1,694
Overhead																	
Precious Metal Royalty	US$000	66,598	0	0	21,999	21,459	6,061	2,283	2,239	1,669	1,685	1,357	1,683	1,839	1,961	1,965	397
Base Metal Royalty	US$000	80,251	0	0	0	0	7,396	12,359	13,324	6,521	5,712	4,244	6,947	7,667	7,670	7,070	1,341
Third party Royalty	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Management Fee	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Engineering Fee	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
General and Administration	US$000	137,832	0	0	14,689	9,360	10,614	11,232	11,232	11,232	11,232	11,232	11,232	11,232	11,232	11,232	2,081
Total Site Operating Costs	***US$000***	***1,258,731***	***0***	***18,164***	***108,298***	***99,301***	***93,806***	***100,282***	***104,400***	***104,494***	***104,966***	***104,227***	***108,063***	***108,204***	***96,372***	***91,243***	***16,913***
Operating Cash Flow		**2,366,111**	**0**	**-18,164**	**331,691**	**329,888**	**238,726**	**298,492**	**321,070**	**115,185**	**91,929**	**44,188**	**124,092**	**147,631**	**162,004**	**150,047**	**29,333**
Capital Costs																	
Mining and PP Capital	US$000	356,777	115,718	121,459	7,956	65,637	5,967	0	36,036	4,004	0	0	0	0	0	0	0
Ore I Gold Oxide	US$000	237,177	115,718	121,459	0	0	0	0	0	0	0	0	0	0	0	0	0
Ore II Copper Supergene	US$000	79,560	0	0	7,956	65,637	5,967	0	0	0	0	0	0	0	0	0	0

		Life of Mine	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Ore III Zinc Primary	US$000	40,040	0	0	0	0	0	0	36,036	4,004	0	0	0	0	0	0	0
Owner's cost	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining Capital	US$000	27,529	0	0	0	-380	4,643	530	-1,702	8,700	2,333	5,679	3,382	3,872	473	0	0
Adjustment to Mining Sustaining Capex	US$000	6,804	0	0	0	-380	-833	530	-1,702	1,325	1,521	554	2,695	2,622	473	0	0
Salvage	US$000	-8,919	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-8,919
Closure	US$000	19,443	0	0	0	0	0	0	1,821	0	0	0	0	0	0	0	17,622
Working Capital (Recovery)	US$000	0	0	3,027	5,997	-750	7,359	1,079	686	16	79	-123	639	24	-1,972	-855	-15,207
Total Capital	***US$000***	***401,634***	***115,718***	***124,486***	***13,953***	***64,127***	***17,136***	***2,139***	***35,139***	***14,045***	***3,933***	***6,109***	***6,716***	***6,518***	***-1,026***	***-855***	***-6,505***
Pre-tax net Cashflow	***US$000***	***1,964,477***	***-115,718***	***-142,650***	***317,737***	***265,760***	***221,590***	***296,353***	***285,930***	***101,140***	***87,997***	***38,079***	***117,376***	***141,113***	***163,029***	***150,902***	***35,838***
Taxes and Other Fees																	
Eritrea Income Tax	US$000	739,178	0	0	92,376	92,393	56,744	79,405	111,487	38,243	30,192	11,562	44,866	54,649	60,273	56,269	10,719
After Tax Net Cashflow	***US$000***	***1,225,299***	***-115,718***	***-142,650***	***225,362***	***173,367***	***164,846***	***216,948***	***174,443***	***62,897***	***57,804***	***26,517***	***72,511***	***86,464***	***102,757***	***94,633***	***25,119***

18.13.3 Sensitivity Analysis

Sensitivity analysis was performed on the pre-tax base case, taking into account ±10% variations in metal prices, grades, and operating costs. The results are shown graphically for NPV 10% in Figure 18-8. The results of the analysis showed that the Project is most sensitive to changes in grades, then metal prices, and is relatively less sensitive to changes in operating expenditure.

Sensitivity analysis to capital expenditures was not conducted as the majority of capital costs have already been spent.

A separate sensitivity analysis was performed on the metal prices shown in Table 18-15 to evaluate the Project sensitivity to a range of long-term assumptions. Results are shown in Figure 18-9.

Figure 18-8: Sensitivity Analysis

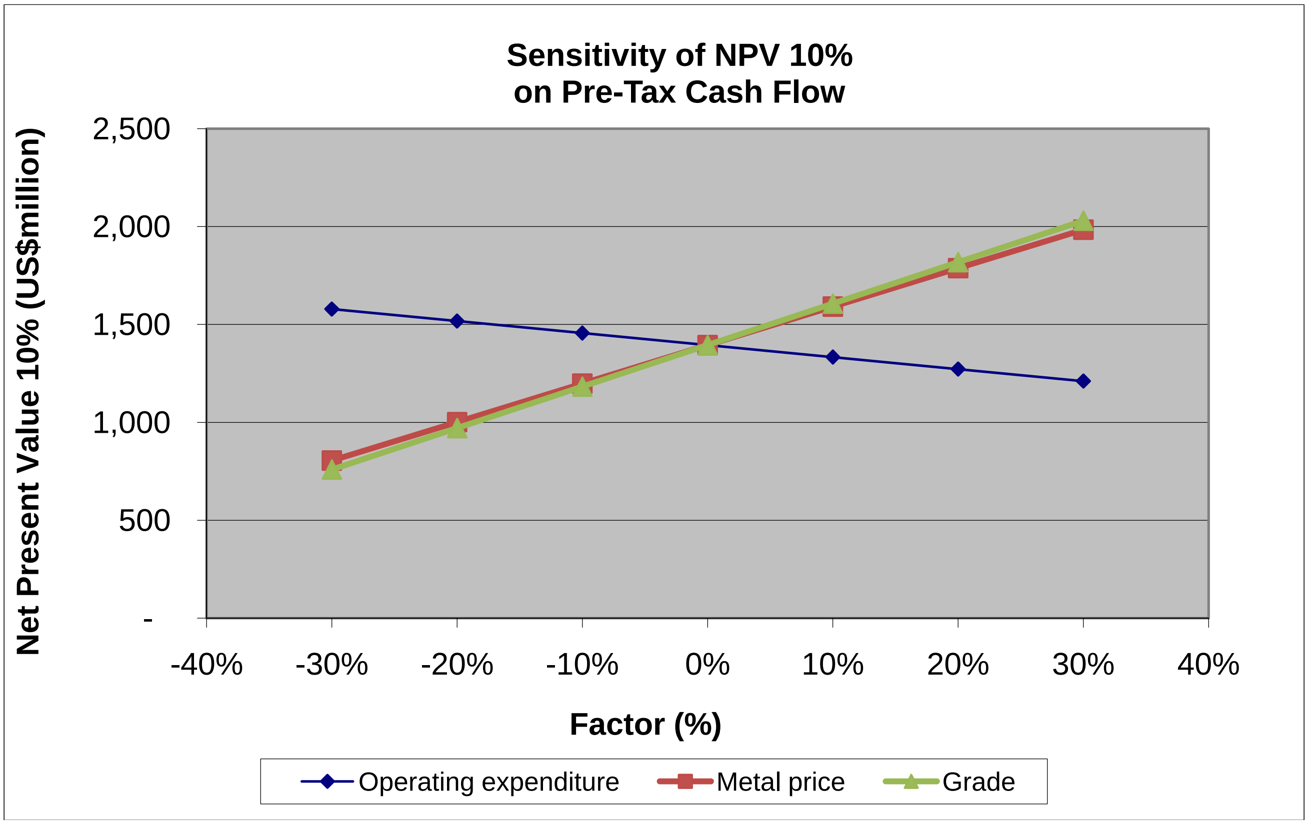


Table 18-15: Metal Price Ranges for Sensitivity Case (base case is highlighted)

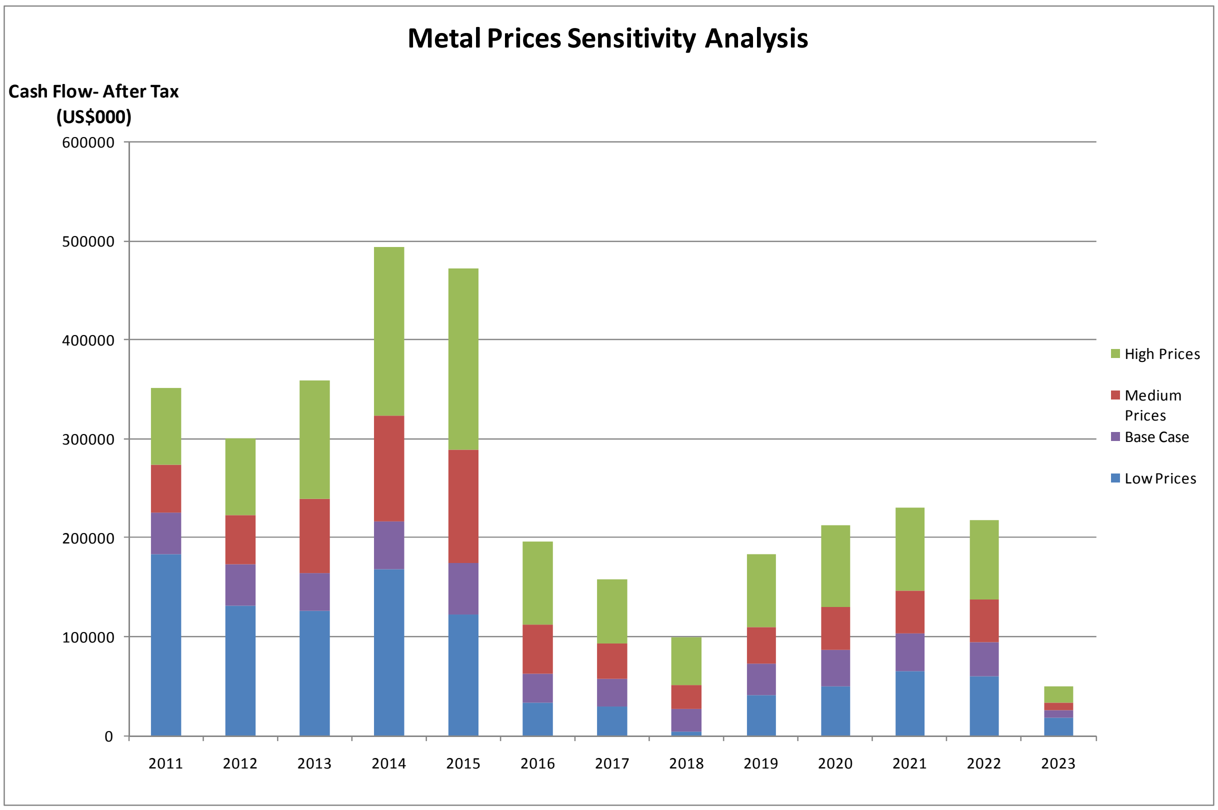
Metal	Unit	Low	Medium	High	Base Case Price
Copper	US$/lb	2.00	3.30	4.75	2.40
Gold	US$/oz	850	1200	1500	1,015
Silver	US$/oz	12.50	24.00	35.00	15.85
Zinc	US$/lb	0.75	1.00	1.25	0.92

Table 18-16: Net Future Cash Flows at Different Price Ranges

Sensitivity Scenarios	Net Future Cash Flows[1]
	(US$000)
Low Prices	1,034,000
Base Prices	1,484,000
Medium Prices	2,162,000
High Prices	3,325,000

[1]Net future cash flow is undiscounted after tax from 2011 onwards, and after all expansion and sustaining capital costs for 100% of the Bisha Project.

Figure 18-9: Metal Price Sensitivity Analysis



19.0 OTHER RELEVANT DATA AND INFORMATION

There are no other data that are relevant to the Report.

20.0 INTERPRETATION AND CONCLUSIONS

Following evaluation of the Project, the QPs are of the opinion that:

- Based on information provided by Nevsun to AMEC, the mining tenure held is valid, and sufficient to support declaration of Mineral Resources and Mineral Reserves
- Based on information provided by Nevsun to AMEC Nevsun holds sufficient surface rights to support mining operations over the planned life-of-mine, and the declaration of Mineral Resources and Mineral Reserves
- Royalties payable include an Eritrean Government royalty of 5% of precious metal net smelter return (NSR) and 3.5% of base metal NSR
- Permits obtained by the company to operate the mine and undertake Project development are sufficient to ensure that mining activities are conducted within the regulatory framework required by the Eritrean government; a permit is currently being sought for diversion structure construction, and permits will be required for the port area.
- At the effective date of this report, environmental liabilities are considered to be typical of a mine that is under construction, and which will produce base metals and doré, and include an open pit, tailings and waste rock facilities, construction of mining infrastructure, and roads
- Environmental impacts were assessed for the Project as part of the 2006 Feasibility Study; the appropriate environmental permits for operation have been issued; environmental bonding has been lodged
- Closure requirements were assessed as part of the 2006 Feasibility Study, and re-assessed by Knight Piésold during 2010
- The existing and planned infrastructure, availability of staff, the existing power, water, and communications facilities, the methods whereby goods are transported to the mine, and any planned modifications or supporting studies are well-established, or the requirements to establish such, are well understood by Nevsun; additional studies are underway for the port facilities required to support base metal production
- The geologic understanding of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support estimation of Mineral Resources and Mineral Reserves
- The mineralization style and setting is well understood and can support declaration of Mineral Resources and Mineral Reserves

- The exploration programs completed to date are appropriate to the VMS style of the Bisha deposit
- The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration, delineation, and grade control drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation
- Sampling methods are acceptable, meet industry-standard practice, and are adequate for mineral resource and mineral reserve estimation and mine planning purposes
- The quality of the gold, silver and base metal analytical data is reliable. Sample preparation, analysis, and security are currently performed in accordance with industry exploration practices and industry standards
- Metallurgical testwork completed on the Project has been appropriate to establish optimal processing routes for the different mineralization styles encountered in the deposit. Testwork was conducted on mineralization that is typical of the deposit. Increased conservatism was used in recovery estimates where the amount of testwork is limited. Additional testwork has been recommended (see Section 21). Since completion of the pit phase designs, the supergene copper recovery was reduced from 92% to 88% based on recent metallurgical test work.
- Estimations of Mineral Resources and Mineral Reserves for the Project conform to industry best practices, and meet the requirements of CIM (2010). The Mineral Reserves are adequate to support mine planning
- Reviews of the environmental, permitting, legal, title, taxation, socio-economic, marketing and political factors and constraints for the Project support the declaration of Mineral Reserves using the set of assumptions outlined
- The mine plans are appropriate for the style of mineralization
- Geotechnical instability of a portion of the northwestern highwall will require remediation in the form of a layback. Other phase design modifications may be required to allow access to all areas. The mining of the new highwall pushback would likely start in the third or fourth quarter of 2011. However, as this layback has not been designed, it is not included in the current long range mine plan. AMEC does not consider the northeast wall slope instabilities to be a fatal flaw. Remediation should not be insurmountable and such events are commonly encountered in the early phase of operations while mining companies better understand the geotechnical competence of the rock mass

- The process design uses a conventional flowsheet that is appropriate to the mineralization styles. Additional metallurgical testwork is recommended (see Section 21).
- Production from the Project can be marketed. Current contracts for doré are typical of, and consistent with, standard industry practice. The expected contracts for the supply of copper and zinc concentrates are expected to be in line with industry norms
- The Project economic analysis is positive under the sets of assumptions used.

In the opinion of the QPs, the Project that is outlined in this Report has met its objectives. Mineral Resources and Mineral Reserves have been estimated for the Project, and a mine is under construction, and has produced the first doré. This indicates the data supporting the Mineral Resource and Mineral Reserve estimates were appropriately collected, evaluated and estimated, and the original Project objective of identifying mineralization that could support a proposed mining operation has been achieved.

There are likely to be modifications to the long-term mine plan based on geotechnical input; and additional metallurgical testwork is recommended (see Section 21).

21.0 RECOMMENDATIONS

A two-phase work program was devised for the Project. Recommendations are based on Project review conducted by Nevsun and AMEC. Work in Phase 2, apart from the exploration drilling, is contingent upon the results of Phase 1.

21.1 Phase 1 Work Program

The recommended work program for the Project is to update the database, complete additional exploration and infill drilling, and conduct geotechnical, hydrological and engineering studies. This is estimated to cost about $8.9 M.

21.1.1 Database and Data Collection

As a result of the data verification program on data collected on the Project to date, AMEC has proposed a number of recommendations that should be incorporated into Project activities. AMEC recommends that Nevsun:

- Submit external pulp duplicates into the regular sample submissions
- Send 5% of the samples for check assay at an independent second laboratory.

These recommendations should be incorporated into regular day-to-day practices.

Additional SG measurements should be taken on Project core, and comprise measurements on both likely waste and mineralized materials. This program is estimated to cost about $10,000.

21.1.2 Exploration and Infill Drilling

The mineralization defined to date within the HW Copper Zone requires additional infill drilling to better delineate the extent of the copper mineralization. The ongoing drill program, which commenced in late January, 2011, is testing the HW Copper zone on 50 m-spaced sections. Subsequent planned infill drilling will test the mineralization on a 25 m drill hole spacing. Based on an inclusive drilling cost, including assays and deviation surveys, of $225/m, Nevsun estimates that approximately 8,400 m of drilling in 60 core holes is required, at an approximate total cost of $2.0 M.

AMEC recommends that infill drilling of the Inferred portions of the Bisha Main deposit be completed, and concurs with the approach proposed by Nevsun. Key aims of the exploration drilling program should be to have drill spacing sufficient to support re-classification of Inferred mineral resources higher confidence categories.

Initial exploration work programs proposed by Nevsun are likely to include about 14,000 m of drilling in approximately 35 core holes. Assuming an all-in drilling cost of $225/m, the total cost of the drilling program is approximately $3.0 M.

21.1.3 Geotechnical

Third-party consultants are currently reviewing the geotechnical designs for, in particular, the northeastern slope of the pit. The work will culminate in a set of updated recommendations for pit design, and is underway.

It is likely that additional geotechnical drilling will be required to support the evaluation. A preliminary estimation of the drilling is that it is likely that 8–10 drill holes (1,200 m) may be required, at an approximate cost of $300/m including tests, for a total cost of approximately $3.6 M.

21.1.4 Hydrogeology

Additional hydrogeological studies are planned, for an approximate cost of about $150,000–200,000.

21.1.5 Engineering Studies

The Phase 1 work program includes provision for engineering studies and completion of metallurgical testwork. This work is estimated to cost approximately $200,000, and include:

- Locked cycle flotation testwork is recommended to support the design of the supergene flotation circuit and associated recovery factors
- Additional batch and locked cycle flotation testwork is recommended to support the design of the primary ore flotation circuit and associated recovery factors
- It is recommended that oxidation effects on supergene and primary flotation recovery be further evaluated

21.2 Phase 2 Work Program

The recommended work program for the Project is to update the Project Mineral Resource and Mineral Reserve estimates. This is estimated to cost about $2.8 M.

21.2.1 Mineral Resource and Mineral Reserve Estimation

Once the Phase 1 campaigns are completed, and results are to hand, the mineral resource and mineral reserve estimates for the Bisha Main Zone should be updated to incorporate the findings of Phase 1. There is expected to be sufficient drill data to support estimation of mineral resources for the Harena and HW Copper Zone deposits. Depending on whether the work is undertaken by Nevsun or a third-party consultant, the program is estimated to cost between $250,000–350,000

21.2.2 Mine Plan Optimization

AMEC is of the opinion that through further optimization, the current mine plan can be improved upon. After completion of the 2011 geotechnical and hydrological study, the oxide pit phases will be re-designed to mitigate the northeast wall stability issues. At this time, an updated resource model, including information from the infill drill program should be available. AMEC recommends a full re-optimization of the mine plan be performed, including updated pit phase designs, cut-off analysis and detailed material movement scheduling, taking into consideration actual production performance of the mine and plant. Depending on whether the work is undertaken by Nevsun or a third-party consultant, the program is estimated to cost between $150,000–250,000

21.2.3 Exploration Drilling

The mineralization defined to date within the NW Zone requires additional infill drilling to better delineate the extent of the mineralization. Based on an inclusive drilling cost, including assays and deviation surveys, of $225/m, Nevsun estimates that approximately 10,000 metres of drilling in 50 holes may be required, at a total cost of $2.25 M. Nevsun plans that the drilling will be completed on a 25 m by 25 m grid spacing.

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23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical report, entitled "Nevsun Resources Limited, Bisha Polymetallic Operation Eritrea, Africa, NI 43-101 Technical Report" is 01 January 2011.

"Signed and sealed"

David Thomas, P.Geo. Dated: 28 March 2011.

"Signed and sealed"

Jay Melnyk, P.Eng. Dated: 28 March 2011.

"Signed and sealed"

Alexandra Kozak, P.Eng. Dated: 28 March 2011.

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Vikram Khera, P.Eng. Dated: 28 March 2011.