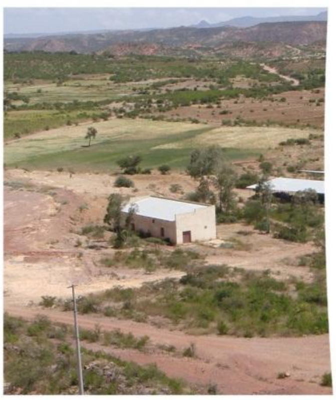






SENET

Asmara Project Feasibility Study

NI 43-101 Technical Report

16 May 2013

AUTHORS CONSENT TO FILING

Neil Senior

SENET (Pty) Limited Executive Director

Building 12, Greenstone Hill Office Park

Emerald Boulevard

Modderfontein, Johannesburg

Gauteng 1609 Republic of South Africa

n.senior@senet.co.za

CONSENT of QUALIFIED PERSON

I, Neil Senior, consent to the public filing of the technical report titled "Asmara Project Feasibility Study NI 43-101 Technical Report" dated effective May 16, 2013 (the "Technical Report") by SENET Pty Ltd. and in part by Snowden Inc, Blue Coast Metallurgy Ltd and Knight Piésold Ltd.

I also consent to any extracts from or a summary of the Technical Report in the May 16, 2013 and May 28, 2013 news releases of Sunridge Gold Corp.

I certify that I have read the news release that the report supports being filed by Sunridge Gold Corp. and that it fairly and accurately represents the information in the sections of the technical report for which I am responsible.

Dated this 26th day of June 2013

"Signed and sealed by Neil Senior"

Neil Senior

Anthony Finch

Snowden Mining Industry Consultant Pty Ltd

Level 3, The Magdalen Centre, Robert Robinson Avenue,

The Oxford Science Park,

Oxford 0X4 4GA United Kingdom

afinch@snowdengroup.com

CONSENT of QUALIFIED PERSON

I, Anthony Finch consent to the public filing of the technical report titled "Asmara Project Feasibility Study NI 43-101 Technical Report" dated effective May 16, 2013 (the "Technical Report") by SENET Pty Ltd. and in part by Snowden Inc, Blue Coast Metallurgy Ltd and Knight Piésold Ltd.

I also consent to any extracts from or a summary of the Technical Report in the May 16, 2013 and May 28, 2013 news releases of Sunridge Gold Corp.

I certify that I have read the news release that the report supports being filed by Sunridge Gold Corp. and that it fairly and accurately represents the information in the sections of the technical report for which I am responsible.

Dated this 26th day of June 2013

"Signed and sealed by Anthony Finch"

Anthony Finch

Andrew F Ross

Snowden Mining Industry Consultants Pty Ltd

87 Colin Street

West Perth, Australia

aross@snowdengroup.com

CONSENT of QUALIFIED PERSON

I, Andrew F. Ross, consent to the public filing of the technical report titled “Asmara Project Feasibility Study NI43-101 Technical Report” dated effective May 16, 2013 (the “Technical Report”) by SENET Pty Ltd. and in part by Snowden Inc, Blue Coast Metallurgy Ltd and Knight Piesold Ltd.

I also consent to any extracts from or a summary of the Technical Report in the May 16, 2013 and May 28, 2013 news releases of Sunridge Gold Corp.

I certify that I have read the news release that the report supports being filed by Sunridge Gold Corp. and that it fairly and accurately represents the information in the sections of the technical report for which I am responsible.

Dated this 26th day of June 2013

“Signed and sealed by Andrew F. Ross”

Andrew F Ross

Scott David Rees

Knight Piésold Ltd.

1400 – 750 West Pender Street, Vancouver

British Columbia

srees@knightpiesold.com

CONSENT of QUALIFIED PERSON

I, Scott David Rees, consent to the public filing of the technical report titled "Asmara Project Feasibility Study NI 43-101 Technical Report" dated effective May 16, 2013 (the "Technical Report") by SENET Pty Ltd. and in part by Snowden Inc, Blue Coast Metallurgy Ltd and Knight Piésold Ltd.

I also consent to any extracts from or a summary of the Technical Report in the May 16, 2013 and May 28, 2013 news releases of Sunridge Gold Corp.

I certify that I have read the news release that the report supports being filed by Sunridge Gold Corp. and that it fairly and accurately represents the information in the sections of the technical report for which I am responsible.

Dated this 26th day of June 2013

"Signed and sealed by Scott David Rees"

Scott David Rees

Christopher John Martin

Blue Coast Metallurgy Ltd

1020 Herring Gull Way, Parksville

British Columbia, Canada

Chris.martin@bluecoastmet.com

CONSENT of QUALIFIED PERSON

I, Christopher John Martin, consent to the public filing of the technical report titled “Asmara Project Feasibility Study NI43-101 Technical Report” dated effective May 16, 2013 (the “Technical Report”) by SENET Pty Ltd. and in part by Snowden Inc, Blue Coast Metallurgy Ltd and Knight Piesold Ltd.

I also consent to any extracts from or a summary of the Technical Report in the May 16, 2013 and May 28, 2013 news releases of Sunridge Gold Corp.

I certify that I have read the news release that the report supports being filed by Sunridge Gold Corp. and that it fairly and accurately represents the information in the sections of the technical report for which I am responsible.

Dated this 26th day of June 2013

“Signed and sealed by Christopher John Martin”

Christopher John Martin

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Sunridge Gold Corp. by SENET. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SENET'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 to be used by Sunridge Gold Corp., subject to the terms and conditions of its contract with SENET. That contract permits Sunridge Gold Corp. to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

1 SUMMARY

1.1 Introduction

This technical report has been prepared to disclose relevant information about the feasibility study (FS) undertaken on the Sunridge Gold Corp. (SGC:TSX.V, SGCNF:OTCQX) Asmara Project deposits in Eritrea, Africa. This information has resulted from technical, economic, environmental and social studies.

1.2 Property Description and Ownership

This technical report refers to the Asmara Project, an advanced stage exploration property located within a 10 to 30 km radius from Asmara, the capital of Eritrea.

The Project consists of four known deposits, Debarwa, Emba Derho, Adi Nefas, and Gupo. Of these, Emba Derho, Adi Nefas, and Gupo are situated to the north, and Debarwa is roughly 30 km to the south of Asmara.

The Project extents that are the subject of this study are located on three licence areas, namely Adi Nefas, Medrizien and Debarwa, totaling 111 km^2.

Asmara is 100% owned by Sunridge Gold Corp (SGC). SGC owns 100% of the Asmara property. Upon the project completing of this feasibility study and conditional upon the granting of a mining license, the Government of Eritrea will have a 10% carried interest in the project and has exercised its option through the Eritrea National Mining Company (ENAMCO) to purchase an additional 30% working interest in the Project.

The economics and financial analysis of this Technical Report assumes a 100% interest by SGC.

1.3 Geology and Mineralization

The base metal deposits of SGC in the Asmara Project (Emba Derho and Adi Nefas) and Debarwa are examples of volcanogenic massive sulphide (VMS) deposits located in the Neoproterozoic Arabian-Nubian Shield (ANS). The ANS represents a composite granitoid-greenstone belt terrain that straddles the Red Sea and covers much of Eritrea, parts of Egypt, Sudan and Ethiopia, and the western part of Saudi Arabia.

The Gupo gold deposit shares similarities with other shear-hosted quartz-vein related gold deposits that are formed in Precambrian terrains and subsequently modified by tectonic events and near-surface weathering effects.

All deposits have been subjected to near-surface weathering effects such that the base metal VMS deposits exhibit oxidation and remobilization of metals. Typically the surface manifestation of the deposit is a gossan overlying zones of depletion or enrichment in oxide, supergene and transition zones. Primary sulphides are developed at depth.

1.3.1 Emba Derho

The Emba Derho deposit appears on surface as a prominently outcropping gossan developed over an area of 800 m by 220 m where it outcrops as a tightly folded unit with northwest oriented fold axial planes and steeply dipping limbs. The sulphide mineralization is hosted within variably but generally heavily sulphide-altered, and moderately sericite-, chlorite-, and quartz-altered predominantly felsic metavolcanic rocks.

The mineralised zones are:

- Gossan (oxide zone)
- Copper-enriched supergene zone
- Pyritic massive sulphide primary zone
- Zinc-rich primary massive sulphide zone
- Copper-rich primary massive sulphide zone

For the purposes of estimation, four distinct zones of mineralization are recognised. A surface oxide gold-rich zone strongly leached with respect to base metals, a zinc-poor supergene zone, overlying a zinc-rich primary sulphide zone which is in turn underlain by a copper-rich primary sulphide zone.

1.3.2 Adi Nefas

The Adi Nefas deposit occurs as an elongated north-northeast trending steeply east dipping massive sulphide layer that is hosted within an upright bimodal sequence of metavolcanic and derived metasedimentary rocks. The massive sulphide unit ranges in thickness from 5 m to 20 m and is largely hosted within a hydrothermally altered felsic quartz-sericite-chlorite-pyrite schist which in turn is flanked above and below by altered metabasaltic rocks. The altered felsic sequence ranges in thickness from 25 m to 60 m.

The Adi Nefas gossan comprises a silica, hematite and goethite-rich assemblage and averages some 10 m in width, and is mapped along strike for almost 2 km.

An upper oxide zone and underlying transition zone are leached and particularly depleted in copper and zinc relative to the primary sulphide mineralization. A slight enrichment in gold is reported for these zones. The base metal tenor increases slightly with depth in these zones. These zones typically extend to groundwater level at a depth of 20 m to 30 m. At and below the groundwater level, the supergene zone contains significantly enriched copper and gold and slightly enriched silver relative to the primary sulphide mineralization. Zinc is still depleted relative to the primary zone. The supergene zone is typically 20 m to 40 m thick.

1.3.3 Gupo

Locally, the highest grade gold mineralization occurs in crystallised, coarse-grained pyrite within quartz veins as well as lower grade gold mineralization in medium to coarse grained, euhedral to sub-euhedral pyrite within a sericite alteration halo. This alteration halo varies in width from few centimetres up to several metres depending of the thickness of the quartz veins, width of the shear zone and the porosity of the host rocks. The quartz veins form a complicated network of stockworks that pinch and swell, within the shear zone.

The Gupo gold deposit has been defined at surface and drilling over about a 1.6 km strike length and a 10 m to 20 m width. These zones are divided into two zones; Gupo North and Gupo South separated by a 400 m long barren zone which is interpreted as a late stage normal fault zone associated with the uplifting of these terrains of the Arabian-Nubian Shield during the opening of the Red Sea.

The Gupo North zone splits into eastern and western sub-zone half way to the south, probably representing the root system of the gold mineralization uplifted by the normal fault.

1.3.4 Debarwa

Three distinct vertical zones of mineralization are recognised at Debarwa. A surface oxide gold zone from which base metals have been predominantly leached, extends to approximately 80 m depth from the highest points and is underlain by an enriched copper supergene zone to around 110 m depth. The supergene zone is in turn underlain by a copper-rich primary sulphide zone which is between 5 to 20 m thick. A thin precious metals-enriched transition zone the oxide from the supergene zone. In addition, remobilised copper mineralization forms a halo surrounding the supergene zone. The geological interpretation identifies the following:

- Gossan (oxide zone)
- Transition zone
- Supergene zone
- Primary zone

1.4 Status of Exploration, Development and Operations

Each deposit has been subjected to programs of surface drilling and sampling, using accepted industry practices, in order that Mineral Resource estimates may be declared. Additionally, the Debarwa deposit has been subject to underground development for exploration purposes and bulk sampling by a prior operator.

There are currently no mining operations.

Mineral Resource estimates for Emba Derho, Adi Nefas, Gupo and Debarwa were disclosed by independent Qualified Persons on behalf of SGC. The estimates comply with the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards on Mineral Resources and Mineral Reserves (CIM, 2010), as required by National Instrument 43-101.

1.5 Mineral Processing and Metallurgical Testwork

Metallurgical testing to provide data for the feasibility study was conducted in eight laboratories from 2009 to 2013.

Gold and silver-bearing oxide composites representing the Debarwa transition and oxide zones, the Emba Derho oxide zone and the Gupo deposit were designed to provide good spatial and lithological representation of the respective deposits, while spanning a range of feed grades.

In addition to mineralogical characterisation, these are subjected to grindability, gravity, cyanide leaching and flotation testing. Heap leach cyanidation was the chosen process resulting from economic trade-off analysis of the candidate processes. A total of seven separate column leach tests were completed. Gold extractions for the various column tests ranged from 42% to 76%. The Gupo and Emba Derho Oxide composites showed the best and most consistent gold extractions ranging from 62% to 73%. The Debarwa Transition Composite #1 exhibited a relatively low gold extraction of 51%. Silver extractions ranged from 13% to 70%. Cyanide consumptions were relatively consistent and ranged from 0.82 kg/tonne to 1.35 kg/tonne.

Following mineralogical characterisation and grindability testing, composites representing supergene ores from the Debarwa and Emba Derho deposits were tested by flotation. Debarwa ores are typical high grade, fine-grained VMS secondary copper ores. Their modal composition is dominated by pyrite in addition to the copper sulphides, but the average grain size of 30 microns requires that these ores are finely ground, and the rougher concentrate very finely reground to achieve good metallurgy. The Emba Derho supergene ores are lower grade but coarser grained, hence they are easier to process. They are also more transition in nature, with a 50:50 mix of primary and secondary copper sulphides, and the intermittent presence of zinc as sphalerite. They are moderately soft, with a mean Bond Ball Mill Work Index (BBMWI)of 9.9 kWh/mt.

The process adopted included grinding to a k_{80} product size of roughly 65 microns, rougher flotation at pH11 with 35-50 g/t isopropyl xanthate. The rougher concentrate is reground to roughly 17 microns, and then cleaned in three stages at pH 11.5, again using isopropyl xanthate. The first cleaner includes a scavenger stage, the tails from which report to final tails. Locked cycle testing of blended Debarwa/Emba Derho supergene ores yielded concentrate grades of 24-25% copper at 85-86 % recovery. Mean gold and silver recoveries of 59 and 65 percent respectively left payable grades of gold (4 g/t) and silver (96 g/t) in the concentrate.

Composites for locked cycle testing were created of primary ores from Emba Derho, Adi Nefas and Debarwa. With the vast majority of tonnage stemming from Emba Derho, most of the testing was done on this material. For the feasibility study, cycle test composites representing the projected mine production schedule as developed in the pre-feasibility study were created, together with samples representing life of mine and end-member copper and zinc contents. Composites representing early years (Years 1-4) of production included representative components from Debarwa, Adi Nefas and Emba Derho. The BBMWI averaged 11.2, 10.5 and 8.5 kWh/mt for Emba Derho, Adi Nefas and Debarwa respectively.

Emba Derho mineralogy is straightforward with copper present as moderately fine chalcopyrite and zinc as slightly coarser sphalerite in a pyrite-dominant rock. Adi Nefas and Debarwa both contain a broader suite of copper minerals, some of which are more reactive so more depressants are needed when treating these ores. The flowsheet developed, and ultimately tested in locked cycle and batch variability mode included:

Copper flotation:

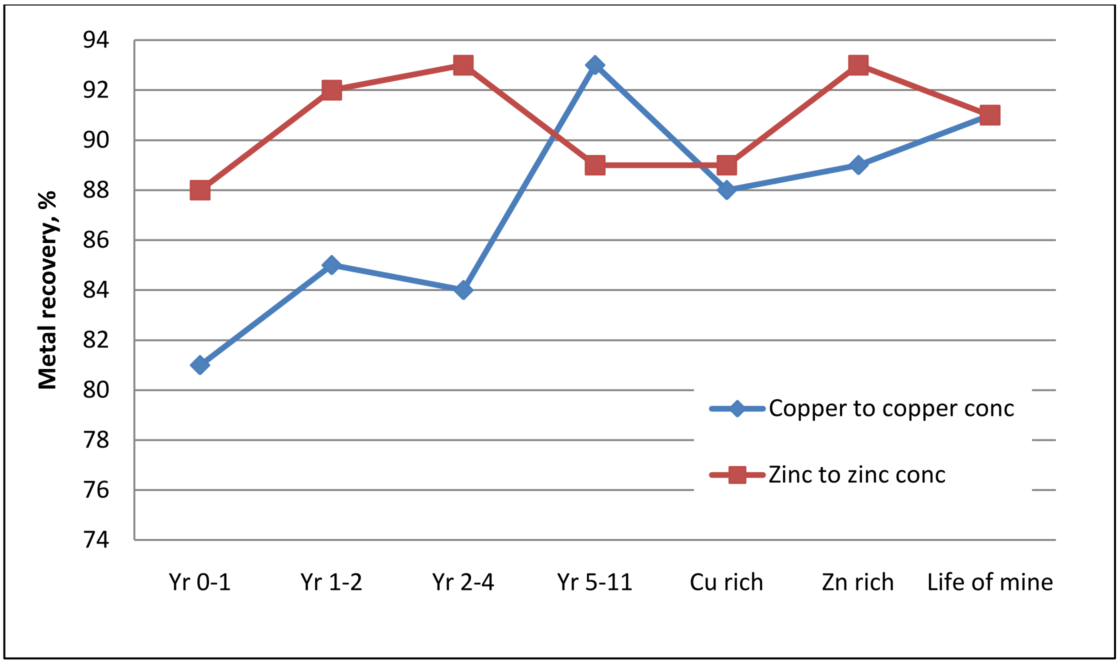
- Grinding to a k_{80} product size of 80 microns, with a pre-mix of zinc sulphate and sodium cyanide added as zinc depressants to the mill, run at pH 9-10 modified with lime (dose was dictated by the secondary copper content in the sample).
- Copper rougher flotation at pH10 using isopropyl xanthate.
- Copper rougher concentrate regrinding a k_{80} product size of 25-30 microns, again with the zinc depressants, then two or three stages of copper cleaning (three in the early years) at pH10, again using isopropyl xanthate collector.

Zinc flotation:

- Copper sulphate activation of the zinc feed using roughly 120 g/t copper sulphate per percent zinc in the feed, pH adjustment to 11.6, then zinc flotation using isopropyl xanthate
- Zinc concentrate regrinding to k_{80} of 25-45 microns (the regrind size being related to the zinc rougher mass pull), then two stages of cleaning using xanthate at pH 11.8 with a first cleaner scavenger, the scavenger tail being open circuited to final tails.

Locked cycle copper and zinc recoveries to the respective concentrates are shown in Figure 1.1. Precious metal recoveries from the chronology samples ranged from 35 to 62% with substantial payable precious metals in the early years' copper concentrates.

Figure 1.1: Copper and Zinc Recovery to their Respective Concentrates



Variability batch cleaner testing on 18 samples from Emba Derho demonstrated good consistency in metallurgical performance. Recoveries were linked to head grades, being quite consistent for head grades above 0.4% copper and 1% zinc, but dropping significantly at head grades below 0.3% copper and 0.5% zinc.

1.6 Mineral Resource Estimation

1.6.1 Emba Derho

The updated resource estimate for the Emba Derho deposit was completed by Snowden Mining Industry Consultants Inc. (Snowden), is as of 6 February 2012 and is based on geological interpretations and a drill database (current as at 9 September 2011) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed.

The mineralization on which the 2012 Emba Derho resource model is based extends over a strike length of 1,250 m and a width of 850 m and has been drilled to a maximum vertical depth from surface of approximately 500 m. The deposit has been explored using 322 exploration drillholes and 7 geotechnical drillholes. Two hundred and eighty (280) drillholes encountered mineralization and have been used in this estimation of resources. In this total were 236 diamond core drillholes and 44 reverse circulation drillholes.

Grades for copper, zinc, gold, silver, lead and iron were estimated within primary and weathered horizon control using Ordinary Kriging after compositing the assay intervals to nominal 1.5 m down-hole lengths.

The resource reporting was constrained by a conceptual pit shell and a conceptual assessment of underground mining extractability to identify those regions of the model that have reasonable prospects for eventual economic extraction.

Mineral Resource Estimates are reported in Table 1.1 and Table 1.2 .

Table 1.1: Measured and Indicated Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	0.07	0.04	1.06	4.3	1.74
Cu Supergene	0.5% Cu	0.94	0.38	0.17	12.2	1.64
Copper-rich Primary	0.3% Cu	0.83	0.93	0.17	7.7	49.8
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.14	2.80	0.31	9.9	16.8
TOTAL						**70.0**

Table 1.2: Inferred Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	-	-	-	-	-
Cu Supergene	0.5% Cu	-	-	-	-	-
Copper-rich Primary	0.3% Cu	0.87	0.89	0.25	10	13.28
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.20	1.94	0.39	11	1.77
TOTAL						**15.05**

1.6.2 Adi Nefas

The updated resource estimate for the Adi Nefas deposit was completed by Snowden, is as of 20 February 2012 and is based on geological interpretations and a drill database (current as at 19 September 2011) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed.

Grades for copper, zinc, gold, silver, lead and iron were estimated within primary and weathered horizon control using Ordinary Kriging after compositing the assay intervals to 1.5 m down-hole lengths.

The resource reporting was considered in the context of underground mining extractability and reasonable prospects for eventual economic extraction.

Mineral Resource Estimates are reported in Table 1.3 below.

Table 1.3: Indicated Mineral Resource Estimate – Adi Nefas

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Primary	2.0 % Zn	1.78	10.05	3.31	115	1.841

1.6.3 Gupo

The resource estimate for the Gupo deposit was completed by Snowden , as of 3 April 2012 and is based on geological interpretations and a drill database (current as at 12 March 2012) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed.

Grades for gold were estimated within grade shells using Ordinary Kriging after compositing the assay intervals to 1.0 m down-hole lengths within the mineralised shell contacts.

Mineral Resource estimates reported for Gupo are constrained by a conceptual pit shell in order to determine the potential quantity for eventual economic extraction. These resources are reported in Table 1.4.

Table 1.4: Indicated and Inferred Mineral Resource Estimate - Gupo

	Cut-off grade	Gold (g/t)	Gold (oz)	Mass (t)
Indicated	0.50 g/t Au	1.53	46,780	951,800
Inferred	0.50 g/t Au	1.83	106,340	1,808,550

1.6.4 Debarwa

The updated resource estimate for the Debarwa Project was completed by AMC Consultants (UK) Ltd, is as of 11 August 2011 and is based on geological interpretations and a drill database (current as at 20 April 2011) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed.

The mineralization on which the 2011 Debarwa resource model is based extends over a strike length of 1 250 m and dips westerly at approximately 50° and has been drilled to a maximum vertical depth from surface of 250 m. The deposit has been explored using 392 exploration holes of which 314 have been used in the estimation of resources, including 268 diamond core and 46 reverse-circulation holes.

Grades for copper, zinc, gold and silver were estimated under zonal and weathering horizon control using Ordinary Kriging for the two main enriched zones and by inverse distance squared weighting for the remainder.

The preliminary classified block model was then subjected to two levels of constraint to ensure that only those portions which demonstrated potential economic viability were retained. Firstly an optimised pit shell derived using metal price parameters at a premium above long term prices (copper $3.00 per pound, gold $1,200 per ounce, zinc $1.00 per pound and silver $20.00 per ounce) was used to identify potential open pit material, after which optimised stope shapes, based on the same prices, were used to incorporate further material considered to be potentially mineable by underground methods.

Mineral Resource estimates for Debarwa are reported in Table 1.5 and Table 1.6 below.

Table 1.5: Measured and Indicated Mineral Resource Estimate – Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5g/t	0.06	0.04	1.47	6	371
Transition	Au 0.5g/t	0.08	0.05	2.85	27	720
Supergene	Cu 0.5%	5.15	0.07	1.40	33	1,389
Primary (Cu)	Cu 0.5%	2.34	3.92	1.30	29	774
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.36	3.05	1.24	22	58
Total						**3 312**

Table 1.6: Inferred Mineral Resource Estimate - Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5g/t	0.1	0.1	1.1	5	239
Transition	Au 0.5g/t	0.1	0.0	1.4	22	138
Supergene	Cu 0.5%	2.7	0.1	0.6	31	144
Primary (Cu)	Cu 0.5%	1.2	3.6	2.6	41	154
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.4	3.3	1.1	21	6
Total						**681**

1.7 Mineral Reserve Statement

The study used the 2012 estimate of Measured and Indicated Resources for the Asmara Project as reported in previous technical reports. Table 1.7 provides an overall summary of the Mineral Reserves.

Table 1.7: Mineral Reserves

Classification	Mass (kt)
Total proven	**4,861**
Total Probable	**51,723**
Total Proven and Probable	**56,584**

1.8 Mining Methods

Emba Derho, Debarwa and Gupo will be mined using conventional open pit truck shovel techniques. The mining rate at Emba Derho is nominally 20 Mtpa total ore and waste. At Debarwa the mining rate peaks at 12 Mtpa, and at Gupo 1 Mtpa.

Adi Nefas is an underground mine; it will be mined using a professional international mining contractor. The underground mining method is long-hole bench retreat. Mine access is via a single decline, which also serves as an intake airway. Each level is accessed from the decline via a level access drive, which intersects the orebody in the approximate centre along the orebody strike. From the level access, strike drives are developed along the ore to the north and south. Between levels stopes are extracted on retreat back to the central decline. All stopes will be waste-filled after extraction. Peak ore production at Adi Nefas is approximately 400 Ktpa.

1.9 Environmental Studies and Social Impacts

Social and environmental baseline studies and stakeholder engagement programs are well advanced on all four deposits that are included in the FS. This work has been completed to comply with the Equator Principles and the International Finance Corporation Performance Standards for Social and Environmental Impact Assessment Studies, as well as the Eritrean Government "National Environmental Assessment Procedures & Guidelines". The work is being carried out by the Sunridge Gold Corp social and environmental staff and consultants (both international and national) and will lead to the submission of a Social and Environmental Impact Assessment (SEIA). It is expected that the SEIA will be completed and submitted to the Eritrean government in September 2013.

1.10 Recovery Methods

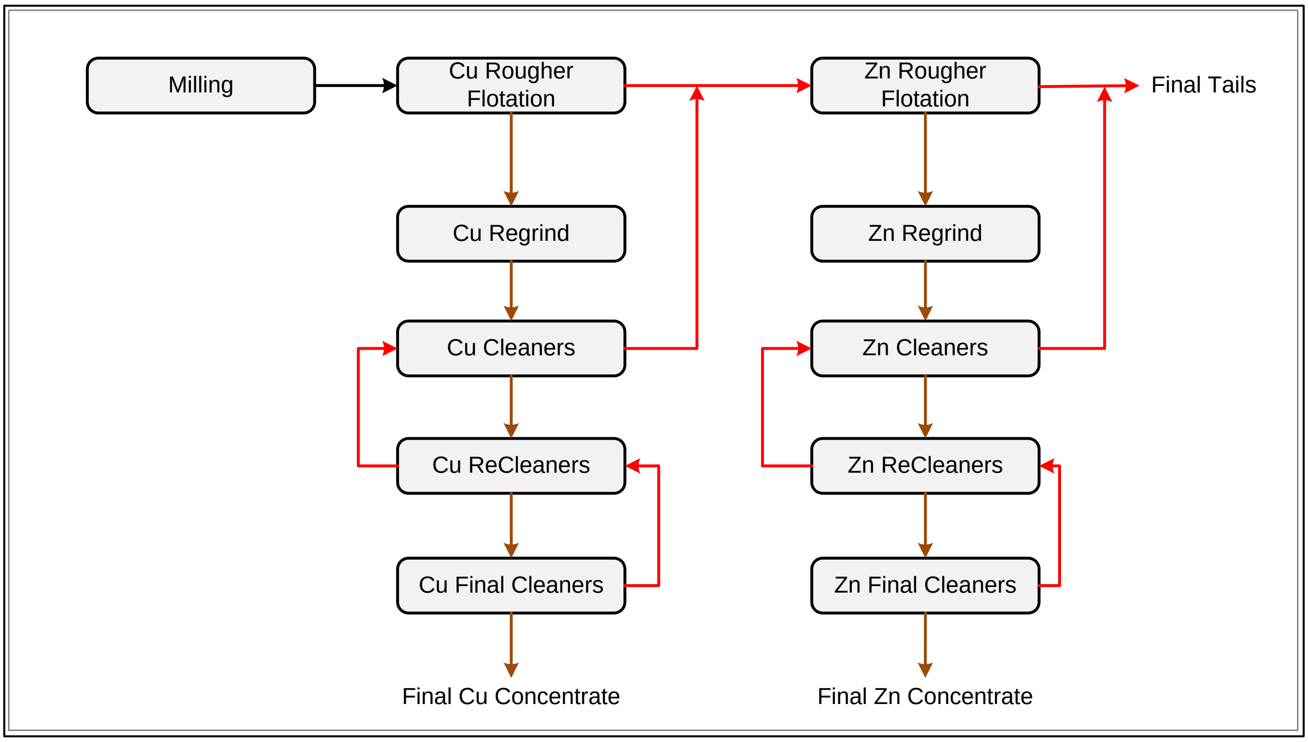
The Asmara Process Plant has been designed for the beneficiation of the following:

- 116,000 tonnes of copper (Cu) rich direct shipping ore (DSO) at a rate of 200,000 tonnes per annum (tpa)
- 1,400,000 tonnes per annum of fresh oxide/transition ore for the recovery & extraction of gold (Au) and silver (Ag)
- 2,000,000 tonnes per annum of fresh supergene ore for the recovery of copper concentrate
- 4,000,000 tonnes per annum of fresh primary ore for the recovery of copper (Cu) & zinc (Zn) concentrates

Proven heap leach & CIS technology will be employed for the recovery and extraction of gold and silver from the oxide and transition ore, and sulphide copper and zinc concentrates will be recovered via sulphide flotation process technology as shown in Figure 1.2. The DSO will be treated through the heap leach plant utilising the comminution circuit to produce a product size of ≤9.5 mm.

It is anticipated that the Asmara plant will be built in phases, with the proposed first phase being the heap leach and DSO plant. The second will include the flotation process plant designed for an initial 2 Mtpa throughput to treat the supergene ore, thereafter upgraded in phase three to 4 Mtpa to treat the primary ore.

Figure 1.2: Process Flowsheet Diagram for Flotation Circuit



Source: SENET

1.11 Project Infrastructure

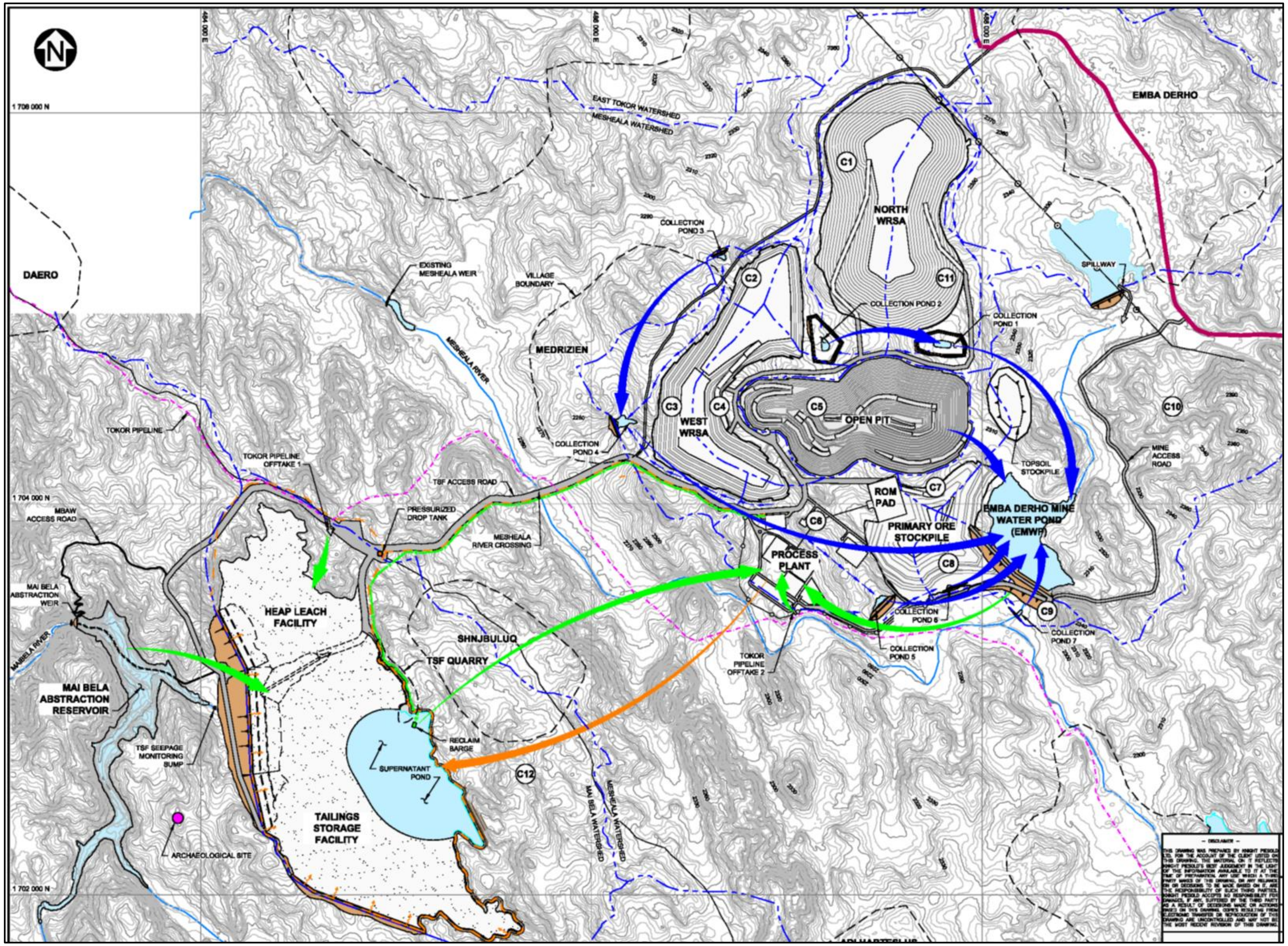
1.11.1 Emba Derho Site Infrastructure

The Asmara Project is spread over an extensive area consisting of four mine sites, with numerous components of surface infrastructure including the process plant, heap leach facilities, tailings storage facility (TSF), waste rock dumps, ore stockpiles, water supply and water management facilities, and other project infrastructure. The project consists of green-fields sites without existing mining infrastructure in place, with the exception of Debarwa which was previously mined using underground methods.

The proposed mine infrastructure will support the mine operations from mining to transportation to purchasers. The three sites of Debarwa, Adi Nefas and Gupo are small deposits relative to Emba Derho and will be mined as satellite operations. The Emba Derho mine site contains the largest mineral value and the majority of on-site infrastructure for the Asmara Project is based in close proximity to Emba Derho. The Emba Derho general arrangement is depicted in Figure 1.3, and includes the following major components:

- Emba Derho open pit
- NAG and PAG waste rock storage areas
- Mine access road
- Haulage roads
- Process plant and associated faciltiies
- Heap leach pad, gold plant, and associated facilities
- Emba Derho mine water pond (EMWP)
- Mai Bela abstraction reservoir (MBAR)
- Tailings storage facility (TSF)
- Saprolite and topsoil stockpiles
- Site water management ponds, pumps and pipeline

Figure 1.3: Emba Derho Site GA and Water Management Diagram



Source: KP

1.11.2 Emba Derho Tailings and Water Infrastructure

1.11.2.1 Tailings Storage Facility (TSF)

The TSF is situated in the Mai Bela River drainage area, a watershed that receives sewage effluent from the City of Asmara. The Mesheala River drainage, located to the north of the TSF, is a tributary catchment of the Tokor reservoir which provides roughly one third of the fresh water supplied to the people of Asmara and surrounding communities. One of the fundamental factors considered in TSF site selection studies was locating the TSF outside of this catchment, ensuring that surface water and groundwater protection in the Tokor reservoir is maximized.

1.11.2.2 Collection Ponds

There are a total of seven collection ponds, excluding those associated with the process plant which will be required at the Emba Derho mine site area to collect runoff from the waste rock storage areas, rom pad, stockpiles and roads. These collection ponds have been sized according to environmental and operational design requirements and will ensure mine contact water is retained on site and prevented from entering the Mesheala River.

1.11.2.3 Emba Derho Mine Water Pond (EMWP)

The EMWP will be located approximately 400 m from the Emba Derho open pit and will be the main control point for contact water at the Emba Derho site. Along with TSF recycle water, it will be used as a primary supply source for the process plant to sustain mill operations. The EMWP will also provide water for dust suppression until the project is closed, and after closure, will become a water supply reservoir for local agriculture and farming use. The EMWP impoundment incorporates 1.5 Mm^3 of operating water storage capacity, with an additional 0.5 Mm^3 of stormwater capacity and a spillway to handle extreme inflows exceeding the 1 in 100 year 24 hour storm event.

1.11.2.4 Mai Bela Abstraction Reservoir (MBAR)

The Mai Bela Abstraction reservoir will be a make-up water supply reservoir located on the Mai Bela River to the west of the TSF. It will provide make-up water for use in the HL facilities and the process plant in periods of shortfall that cannot be met by the TSF and EMWP. The facility has been designed with a capacity of 1.5 Mm^3 which has been calculated to exceed the estimated maximum make-up requirements for both average and dry climatic conditions. The structure will become a long term asset after mine closure, for the local people requiring water for agriculture and farming use.

1.11.2.5 Mine Site Infrastructure

The three sites of Debarwa, Adi Nefas and Gupo will be utilised solely for mining purposes. The extent of site infrastructure will be kept to a minimum as the life of these mines ranges between one year and five years. Each site will be provided with temporary infrastructure including workshop, offices and ablutions which will be moved on from site to site as each mine site is mined out.

1.11.3 Water Management

There are two main water management objectives: preventing mine-contact water from entering the receiving environment by surface and/or groundwater discharge, and impounding a sufficient quantity of water in the various water retaining structures to support processing, even during dry periods. The FS has focused on process design and site layout incorporating these priorities.

Protection of the regional surface and groundwater resources at Emba Derho is particularly important due to the Tokor reservoir, one of Asmara's major water supply reservoirs located roughly 3.5 km downstream of the project area. Similarly, a water supply reservoir is currently under construction downstream of the Debarwa Project, and a domestic water resource at is located downstream of the Adi Nefas project area, both necessitating a high degree of care in regards protection of the water resources.

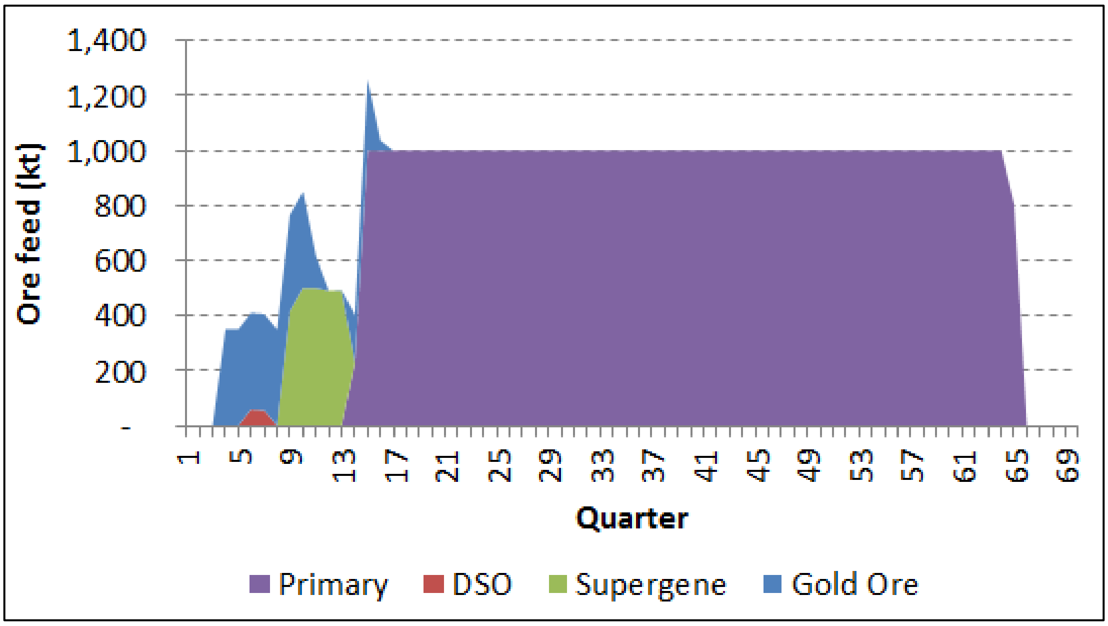
All contact water from site will be contained within the mine site operational water management plan at each of Emba Derho, Debarwa, and Adi Nefas (due to the short mine life and absence of surface infrastructure at Gupo Gold this is not required). This includes major water management infrastructure (the TSF, EMWP, and Debarwa Mine Water Pond (DMWP)), plus a series of water management ponds, pumps and pipelines that will transfer water as appropriate to achieve the goals of zero discharge and continuous water availability. Water supply systems at Emba Derho will include two pipeline taps (the Tokor pipeline offtakes) to supply fresh water and make-up water to the project, and the MBAR for make-up water supply.

1.12 Project Schedule

The overall processing schedule is shown in Figure 1.4 and Table 1.8.

Processing commences in quarter 4 with heap leach loading. The heap leach continues for three years. DSO commences in quarter 5 and is completed withiin six months. Supergene ore processing commences in quarter 9 and continues for five quarters. The floatation plant is then modified to enable processing of primary ore in quarter 14 which is processed for a further 13 years.

Figure 1.4: Overall Processing Schedule



Source: Snowden

Table 1.8: Processing Schedule by Stream

	Gold Ore			DSO Ore				Supergene Ore				Primary Ore				
Year	Mass (kt)	Au (g/t)	Ag (g/t)	Mass (kt)	Cu (%)	Au (g/t)	Ag g/t	Mass (kt)	Cu (%)	Au (g/t)	Ag (g/t)	Mass (kt)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)
1	350	2.0	12.3													
2	1,400	1.6	10.8	116	15.6	3.0	76.8									
3	817	0.8	6.9					1,905	2.6	0.9	24.9					
4	471	2.0	-					490	0.8	0.4	8.8	2,230	0.9	3.7	0.8	25.0
5												4,000	0.6	3.6	0.7	25.3
6												4,000	1.0	2.8	0.5	20.0
7												4,000	1.4	1.6	0.4	16.2
8												4,000	1.1	1.6	0.4	13.1
9												4,000	0.5	2.4	0.4	15.0
10												4,000	0.5	2.5	0.2	9.2
11												4,000	0.6	1.9	0.4	9.1
12												4,000	0.7	1.5	0.2	7.0
13												4,000	1.2	0.6	0.1	7.0
14												4,000	0.5	1.2	0.2	7.7
15												4,000	0.3	1.1	0.2	7.4
16												4,000	0.3	0.8	0.2	6.1
17												805	0.2	0.8	0.2	6.8

1.13 Capital and Operating Costs

1.13.1 Operating Costs

The LOM operating costs are shown in Table 1.9.

Table 1.9: Life of Mine Operating Costs

LOM Costs	Unit	LOM
Mining Operating Costs	$/t	12.8
Process Plant Operating Costs	$/t	13.4
G&A Costs	$/t	2.21
Assay Costs	$/t	0.48
TSF & Water Management Costs	$/t	0.29
LOM Operating Costs	**$/t**	**29.19**
Royalties and transport	$/t	7.22
Total LOM Cash Costs	**$/t**	**36.41**

1.13.2 Capital Costs

The LOM capital costs are provided in Table 1.10. for each of the three phases split into gold, copper and zinc.

Table 1.10: Life of Mine Capital Costs

	Phase I $ million	Phase II $ million	Phase III $ million	Total $ million
Pre-strip mining and mining equipment [1]	0	116.0	0	116.0
Phase I Plant and Equipment	49.5	0	0	49.5
Copper circuit facility	0	113.8	0	113.8
Zinc circuit facility	0	0	22.8	22.8
Site development, utilities and facilities	3.8	55.5	5.5	64.8
Water facilities	0.04	19.4	0	19.44
Tailings facilities	11.2	18.3	0.2	29.7
Debarwa facilities	0	9.8	0	9.8
Adi Nefas facilities	0	3.2	0	3.2
Gupo facilities	1.1	0	0	1.1
Adi Nefas development	0	17.0	17.1	34.1
EPCM costs	4.1	29.8	5.2	39.1
First fills (fuel, reagents)	0.03	1.7	0	1.73
Owner's costs	1.0	22.7	0	23.7
Contingency	5.5	21	3.6	30.1
SUBTOTALS	**76.3**	**428.2**	**54.4**	**558.9**
Sustaining Costs				**56.0**
Social Costs				**14.8**
Closure Costs				**36.6**
TOTAL				**666.3**

[1] Includes all mining costs incurred until copper ore is mined (quarter 5)This excludes HL & DSO Opex.

1.14 Economic Analysis

The economic analysis was completed using four metal price scenarios, shown in Table 1.11.

Table 1.11: Metal Price Assumptions for Each Case

Metal	Base Case Prices	Low Copper Metal Price	Low Metal Prices	Current Metal Prices (May 10 2013)
Copper ($/lb)	3.25	3.00	2.75	3.35
Zinc ($/lb)	1.00	1.00	0.80	0.83
Gold ($/oz)	1,400	1,400	1,250	1,449
Silver ($/oz)	25.00	25.00	21.00	24.00

The life of mine net revenue for each metal and the total for the project are detailed in Table 1.12, and the life mine metal recovered from the entire project is shown in Table 1.13.

Table 1.12: LOM Net Revenue

Metal	Base Case Prices ($M)	Low Copper Metal Price ($M)	Low Metal Prices ($M)	Current Metal Prices (May 10, 2013) ($M)
Copper	2,459	2,257	2,055	2,539
Zinc	1,301	1,301	1,005	1,051
Gold	537	537	479	555
Silver	221	221	182	211
Total	**4,517**	**4,315**	**3,721**	**4,356**

Table 1.13: LOM Metal Production

Item	Unit	Amount
Copper in concentrate	Millions of pounds	841
Zinc in concentrate	Millions of pounds	1,874
Gold in concentrate	Thousands of ounces	339
Silver in concentrate	Thousands of ounces	10.927
Gold doré from heap leach	Thousands of ounces	97
Silver doré from heap leach	Thousands of ounces	295

The base case uses constant metal prices of $3.25/lb copper, $1.00/lb zinc, $1,400/oz gold and $25.00/oz silver for the LOM. The pre and post-tax financial results are provided for four different cases as detailed in Table 1.14. All prices are reflected as $ million.

Table 1.14: Financial Results

	Base Case Prices	Low Copper Metal Price	Low Metal Prices	Current Metal Prices May 10 2013
Pre-tax				
NPV @ 10% discount ($M pre-tax)[2]	692	595	309	623
IRR %[3]	34%	31%	22%	33%
Payback (years)[4]	4.1	4.3	5.1	4.2
Post-tax				
NPV @ 10% discount ($M post- tax)	345	275	69	296
IRR %	27%	24%	17%	26%
Payback (years)	4.6	4.8	5.6	4.7

1.15 Interpretation and Conclusions

There is a significant combined Mineral Resource on the project that has been estimated at a confidence whereby much of it can be converted into a substantial Mineral Reserve

The processing methods are in common use and the materials react favourably to the treatment methods proposed for each rock type and deposit. The recovery of the contained metals by the selected processes is as expected for these types of ore and concentrates. Commercially acceptable grades can be produced. The processing risk is low due to the use of proven equipment and technologies.

At the current level of the study the project shows robust economic returns, and it is recommended that the project be moved forward to a detailed design wherein project economics are further refined and financing can be sought for construction on that basis.

1.16 Recommendations and Opportunities

At the current level of the study the project shows robust economic returns, and it is recommended that the project be moved forward to a detailed design wherein project economics are further refined and financing can be sought for construction on that basis.

[2] The NPV (Net Present Value) is the total of all the period net cashflows discounted at the nominated rate to the start of the pre-production period

[3] The IRR (Internal Rate of Return) is the discount rate when applied to the NPV calculation brings the NPV of all the periodic net cashflows to zero.

[4] The Payback period is at the time in the production profile that the cumulative negative net cashflow begins to turn positive. From this time forward there will generally be periodic positive cashflows and the total cost of bring the project into production will be paid back

1.17 Cautionary Note Regarding Forward-Looking Information and Statements

This Technical Report contains or incorporates by reference “forward-looking statements” within the meaning of the United States Private Securities Litigation Reform Act of 1995 and applicable Canadian securities legislation. Except for statements of historical fact relating to the SGC the information contained herein constitutes forward-looking statements, including any information as to the SGC’s strategy, plans or future financial or operating performance. Forward-looking statements are characterized by words such as “plan”, “expect”, “budget”, “target”, “project”, “intend”, “believe”, “anticipate”, “estimate” and other similar words, or statements that certain events or conditions “may” or “will” occur. Forward-looking statements are based on the opinions, assumptions and estimates of the authors and are considered reasonable at the date the statements are made, and are inherently subject to a variety of risks and uncertainties and other known and unknown factors that could cause actual events or results to differ materially from those projected in the forward-looking statements. These factors include SGC’s expectations in connection with the project and herein being met, the impact of general business and economic conditions, global liquidity and credit availability on the timing of cash flows and the values of assets and liabilities based on projected future conditions, fluctuating metal prices (such as gold, copper, silver and zinc), currency exchange rates (such as the Eritrean Nakfa, the Canadian Dollar versus the United States Dollar), possible variations in ore grade or recovery rates, changes in accounting policies, changes in the project’s corporate Mineral Resources (as defined herein), risks related to non-core mine disposition, changes in project parameters as plans continue to be refined, changes in project development, construction production and commissioning time frames, risk related to joint venture operations, the possibility of project cost overruns or unanticipated costs and expenses, higher prices for fuel, steel, power, labour and other consumables contributing to higher costs and general risks of the mining industry, failure of plant, equipment or processes to operate as anticipated, unexpected changes in mine life, final pricing for concentrate sales, unanticipated results of future studies, seasonality and unanticipated weather changes, costs and timing of the development of new deposits, success of exploration activities, permitting time lines, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, title disputes or claims, limitations on insurance coverage and timing and possible outcome of pending litigation and labour disputes, as well as those risk factors discussed or referred to herein and in the SGC’s annual management’s discussion and analysis filed with the securities regulatory authorities in all provinces of Canada and available under the Company’s SEDAR profile at www.sedar.com.

Although the Authors have attempted to identify important factors that could cause actual actions, events or results to differ materially from those described in forward-looking statements, there may be other factors that cause actions, events or results not to be anticipated, estimated or intended. There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. The reader is cautioned not to place undue reliance on forward-looking statements.

The forward-looking information contained herein is presented for the purpose of assisting investors in understanding the projects expected financial and operational performance and SGC’s plans and objectives and may not be appropriate for other purposes. The Authors do not undertake to update any forward- looking statements contained herein or incorporated by reference herein, except in accordance with applicable securities laws.

Cautionary Note to United States Investors Concerning Estimates of Measured, Indicated and Inferred Mineral Resources: This Technical Report uses the terms “Measured”, “Indicated” and “Inferred” Mineral Resources. United States investors are advised that while such terms are recognized and required by Canadian regulations, the United States Securities and Exchange Commission does not recognize them. “Inferred Mineral Resources” have a great amount of uncertainty as to their existence, and as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Mineral Resource will ever be upgraded to a higher category.

Under Canadian rules, estimates of Inferred Mineral Resources may not form the basis of feasibility or other economic studies. United States investors are cautioned not to assume that all or any part of Measured or Indicated Mineral Resources will ever be converted into Mineral Reserves (as defined herein). United States investors are also cautioned not to assume that all or any part of an Inferred Mineral Resource exists, or is economically or legally mineable.

2 INTRODUCTION

This technical report has been prepared by SENET (Pty) Ltd (SENET) for Sunridge Gold Corp (SGC) in compliance with the disclosure requirements of Canadian National Instrument 43-101 (NI 43-101), to support disclosure of the results of a feasibility study (FS) of the Asmara Project located in Eritrea, Africa. SGC is a mine development and exploration company listed on the Toronto Stock Exchange (SGC-TSX-V).

The effective date of the Asmara Project Technical Report is 16 May 2013. The economic analysis to determine the appropriate cut-off grades for reporting mineral resources and reserves and for the subsequent mining study was completed on 1 May 2013. No new material information has become available between these dates and the signature date given on the certificate of the qualified persons.

This FS is definitive in nature and its purpose is to establish economic potential and define the preferred option to take forward to permiting and detailed engineering, procurement and construction. Mineral resources and reserves have been estimated as part of the study in accordance with CIM guidelines and best practice. There is no certainty that the project economics will be realised as the study only considers a nominal ± 15% level of accuracy on operating and capital cost inputs.

Unless otherwise stated, information and data contained in this report or used in its preparation have been provided by SGC. This technical report has been compiled from sources listed in the References section and cited in the text by the following:

- Mr. David Chambers, P.Eng. (MBA) of SENET (Pty) Ltd (SENET), Study Manager
- Neil Senior, P.Eng. MSc Mech.Eng, FSAIMM
- Mr. Anthony Finch, P.Eng., MAusIMM (CP Mining), Study Manager of Snowden
- Mr. Andrew F. Ross FAusIMM (CP Geo), Snowden Senior Principal Consultant
- Mr Christopher J. Martin, BSc (Hons) ACSM, M.Eng, MIMMM, C.Eng President and Principal Metallurgist, Blue Coast Metallurgy (BCM)
- Mr. Scott Rees, P.Eng. of Knight Piésold (KP).

Mr Senior, Mr. Finch, Mr. Ross, Mr. Martin and Mr Rees are qualified persons as defined by NI 43-101 and are independent of SGC. The responsibilities of each qualified person are provided in Table 2.1.

Mr. Chambers - visited the project sites in June 2012, January 2013 and March 2013 where he reviewed the location of the deposits, topography, potential sites for the processing facilities and surface infrastructure.

Mr. Senior - visited the project sites in September 2012.

Mr. Finch - visited the project site in February 2011, February 2012 and March 2013, where he reviewed the location of the deposits, the general infrastructure and the topography.

Mr. Rees – visited the project site in November and December of 2010 and May 2012, where he conducted site investigation programs, and reviewed topography and the location of waste and water management mine infrastructure at Debarwa, Emba Derho, and Adi Nefas.

Mr. Ross - visited the project site in February 2011, where he reviewed the geology and mineralization of the deposits, sampling practices, geology field procedures, and general infrastructure.

Mr. Martin - visited the project site in February 2012 and March 2013, where he reviewed the location of the deposits, the general infrastructure and the topography.

Table 2.1: Summary of Responsibilities of Qualified Persons

Company & QP	Section of Responsibility	Attendance on site
Knight Piésold (KP)	TSF, Water Management and Supply, Conceptual Closure and Environmental	
Scott Rees	1, 2, 3, 5, 18, 20,21, 25, 26 & 27	November & December 2010, May 2012
Snowden	Geology and Mining	
Anthony Finch	1, 2, 3,15, 16, 19, 21, 22, 24, 25, 26 & 27	February 2011, February 2012 and March 2013
Andrew Ross	1, 2, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25, 26 & 27 `	February 2011
Blue Coast Metallurgy	Metallurgical Testwork	
Chris Martin	1, 13, 25 & 26	February 2012 and March 2013
SENET	Process Plant and Infrastructure	
Neil Senior	1, 2 & 3, 4, 17, 18, 19 and 21 to 27	September 2012

3 RELIANCE ON OTHER EXPERTS

SENET has relied upon documentation provided by SGC in respect of the status of the exploration licenses that cover the Emba Derho, Adi Nefas, Gupo and Debarwa deposits. This is described in Sections 4.1 to 4.5.

Snowden has relied upon documentation reported in Hopley *et al.,(*2011) in respect of the status of Mineral and Environmental Legislation.This is described in Section 4.2.

KP relied on documentation and input provided by Ryan Stinson (KP), Debra Stokes (SGC) and Angela Reeman (Reeman Consultants) for environmental and social aspects of the Project.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 General

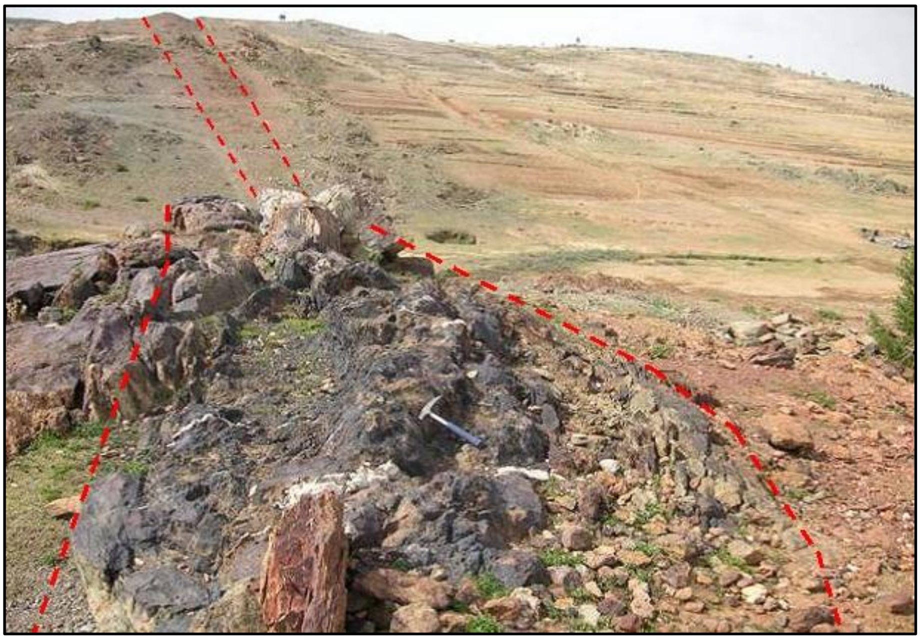
The Asmara Project covers 111 square kilometres in central Eritrea, located immediately to the south, north and north-west of the capital city of Asmara.

Figure 4.1: Asmara Project Location

Source: SGC

The terrain has moderate elevation relief and limited vegetation. Figure 4.2 and Figure 4.3 show typical landscape and terrain near the Adi Nefas and Emba Derho deposits respectively.

Figure 4.2: Typical terrain near Adi Nefas



Source SGC

Figure 4.3: Typical terrain near Emba Derho

Source: SGC

The infrastructure servicing the project area is excellent. Services such as accommodation, roads, and an international airport are close at hand. Asmara North encompasses a district of base and precious metals VMS deposits as well as near surface gold mineralization.

The project encompasses four separate sites in Eritrea. The largest deposit, Emba Derho, is approximately 12 km north-west of Asmara. The proposed process plant would also be located at the Emba Derho site, and would be the sole treatment facility for all of the deposits.

Other deposits are Adi Nefas and Gupo, which are located 8 km and 10 km respectively east of Emba Derho. The fourth deposit, Debarwa, is located approximately 30 km south of Asmara.

4.2 Mineral Tenure

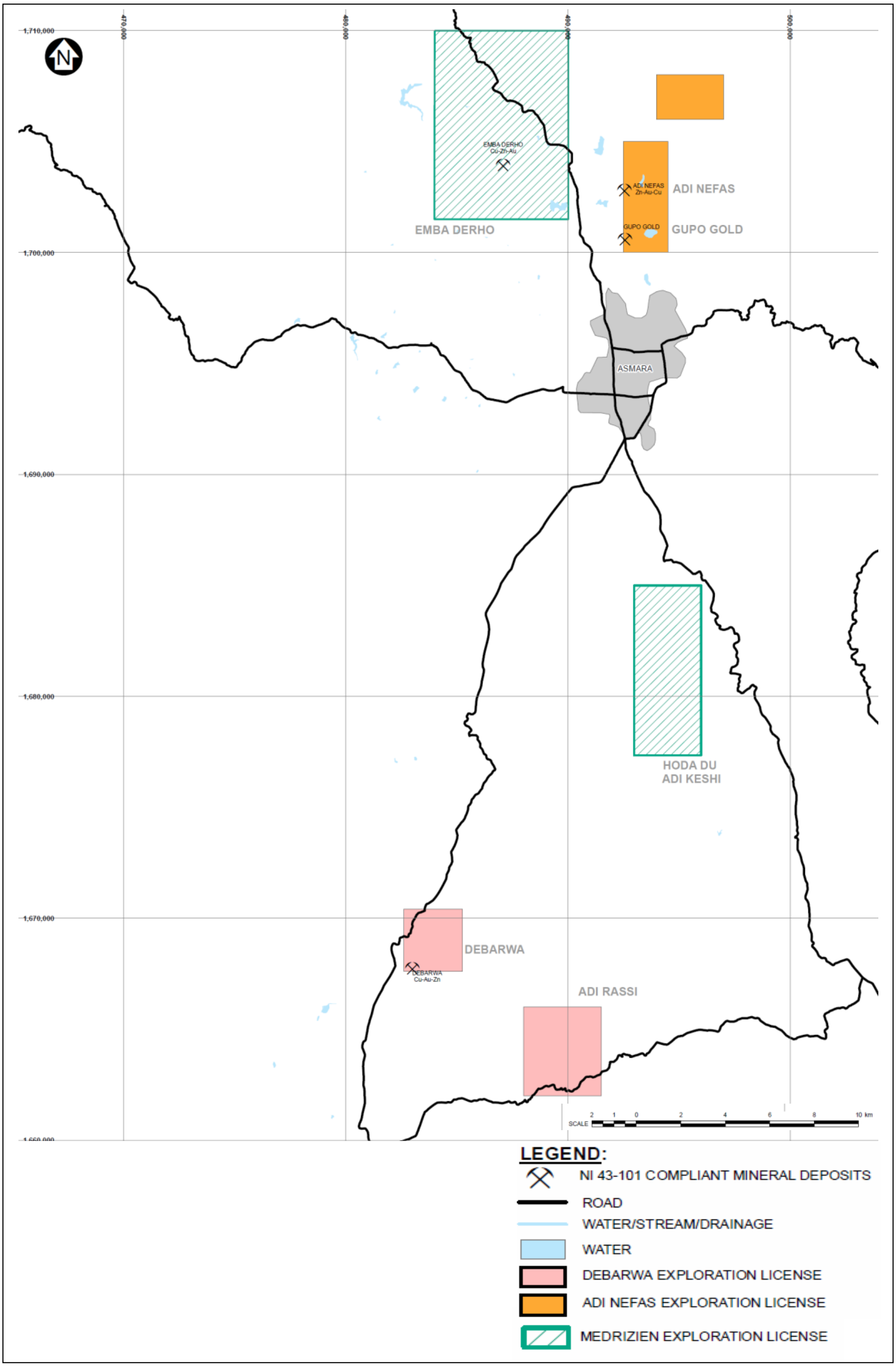
The deposits explored by SGC are held under tenure through Sunridge Gold Eritrea, a branch of SGC. These are shown in Figure 4.4.

The Emba Derho deposit is contained within the Medrizien exploration license (“EL”) located near the capital city of Asmara, Eritrea. Following reductions in the EL area, the Medrizien EL currently occurs in two parts. Emba Derho occurs in the north of the larger part of the EL which lies immediately north of Asmara. The central coordinate location of the Emba Derho deposit is 1,704,400N; 487,200E (UTM coordinates Projection UTM Zone 37N; Datum WGS84).

The Adi Nefas EL is located immediately north of Asmara. The central coordinate location of the Adi Nefas deposit within this EL is 1,703,300N; 493,700E (UTM coordinates Projection UTM Zone 37N; Datum WGS84). The central coordinate location of the Gupo deposit within this EL is 1,701,700N; 493,200E.

The Debarwa EL is located 30 km south of Asmara. The Debarwa deposit is 30 km south of Asmara and 2 km east of the town of Debarwa. The Debarwa deposit is located on the southern part of the Debarwa EL at approximate latitude 15°26’N and longitude 37°30’E. The UTM coordinates of the centre of the property are 1,668,700N and 483,400E.

Figure 4.4: The Asmara Gold and Base Metal Deposits of SGC



Source: SGC

4.2.1 Medrizien

4.2.1.1 History of Mineral Tenure

SGC, through its branch office Sunridge Gold Eritrea, is the holder of the Medrizien EL following assignment of the license from Sub-Sahara Resources (Eritrea) Limited on 13 April 2007. The Assignment Agreement, signed by the Minister of Energy and Mines of the State of Eritrea, transfers and assigns all title and obligations to SGC. Gribble *et al*., (June 2009) were provided copies of the license rental and renewal advice and receipts, as well as copies of the signed Assignment Agreement. The ownership history of the Medrizien EL is covered in more detail in Section 6.

4.2.1.2 Status of Exploration License

In a letter dated 3 June 2012, the Eritrean Ministry of Energy and Mines has approved an application by Sunridge Gold Eritrea for an extraordinary extension of the Medrizien (Emba Derho) EL until 23 May 2013 subject to the requirement that Sunridge Gold Eritrea complete a JORC or equivalent definitive feasibility report within the extension period and complete a JORC or equivalent standard compliant resource estimate on Kodadu-Adi Kessi mineral deposit. SENET understands that Sunridge Gold Eritrea is the operating entity of SGC in Eritrea. The Medrizien EL covers an area of 74 square km.

SGC is currently waiting for the letter from Eritrean Department of Mines confirming renewal of the exploration license for another year. SGC has fulfilled all the work obligations during the previous year. The letter of confirmation for renewal can take 4 to 6 weeks after the expiration date.

4.2.2 Adi Nefas

4.2.2.1 History of Mineral Tenure

SGC, through its branch office Sunridge Gold Eritrea, is the holder of the Adi Nefas EL following assignment of the license from Sub-Sahara Resources (Eritrea) Limited on 13 April 2007. The Assignment Agreement, signed by the Minister of Energy and Mines of the State of Eritrea, transfers and assigns all title and obligations to SGC. Hall et al., (2008) were provided copies of the license rental and renewal advice and receipts, as well as copies of the signed Assignment Agreement. The ownership history of the Adi Nefas EL is covered in more detail in Section 6.

4.2.2.2 Status of Exploration License

In a letter dated 3 June 2012, the Eritrean Ministry of Energy and Mines has approved an application by Sunridge Gold Eritrea for an extraordinary extension of the Adi Nefas EL until 23 May 2013 subject to the requirement that Sunridge Gold Eritrea complete a JORC or equivalent definitive feasibility report within the extension period. SGC is currently waiting for the letter from Eritrean Department of Mines confirming renewal of the exploration license for another year. SGC has fulfilled all the work obligations during the previous year. The letter of confirmation for renewal can take 4 to 6 weeks after the expiration date.

The Adi Nefas EL covers an area of 16 square km.

4.2.3 Debarwa

4.2.3.1 History of mineral tenure

SGC, through its branch office Sunridge Gold Eritrea, is the holder of the Debarwa EL following assignment of the license from Sub-Sahara Resources (Eritrea) Limited on 13 April 2007. The Assignment Agreement, signed by the Minister of Energy and Mines of the State of Eritrea, transfers and assigns all title and obligations to SGC. Hall et al., (2008) were provided copies of the license rental and renewal advice and receipts, as well as copies of the signed Assignment Agreement. Part of the Debarwa EL may be converted to a Mining License upon the acceptance by the State of Eritrea of an appropriate Feasibility Study and social and environmental impact assessment (SEIA) report. The ownership history of the Debarwa EL is covered in more detail in Section 6.

4.2.3.2 Status of Exploration License

The annual rental fee and the annual license renewal fee for the Debarwa EL is approximately 42,800 Nakfa (about US$2,853). Based on a letter dated June 3, 2012 from the Minister of Energy and Mines the Debarwa EL is valid until 23 May, 2013 subject to the complete a JORC or equivalent definitive feasibility report within the extension period and complete a JORC or equivalent compliant resource estimate on Adi Rassi mineral deposit. The Debarwa EL is a single contiguous exploration license covering a total surface area of 21.42 square km.

SGC is currently waiting for the letter from Eritrean Department of Mines confirming renewal of the exploration license for another year. SGC has fulfilled all the work obligations during the previous year. The letter of confirmation for renewal can take 4 to 6 weeks after the expiration date.

4.3 Mineral Legislation

Information provided below has been excerpted from Hopley *et al*., (2011).

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). A copy of Proclamation No. 68/1995 and related legislation (Mining Income Tax Proclamation No. 69/1995; Legal Notice No. 19/1995 – Regulations of Mining Operations) was provided to Gribble et al., (2009) by SGC. Additional regulations and proclamations have been presented regarding environmental protection, land use, water use and heritage. The following summary of the mining legislation and environmental regulations is largely sourced from AMEC's Bisha Feasibility Study (2006).

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.

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

A Prospecting License (PL) grants an exclusive right to prospect for minerals within the license area, is valid for a period of one year, and may not be renewed. Upon discovery of indications of minerals within the license area, the licensee shall have the right to be granted an EL. An Exploration License (EL) grants an exclusive right to explore for all minerals within the area specified in the license other than construction material, mineral water and geothermal deposits. An EL is valid for an initial period of three years and may be renewed twice for additional terms of one year each. Further extensions of renewal periods may be allowed where the licensee documents the necessity for additional advanced exploration or provides other circumstances which justify an extension of the duration of the license. On each renewal the licensee shall relinquish a minimum of 25% of the original license area.

A Mining License (ML) entitles the licensee to 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. A ML is valid for a maximum period of 20 years or the life of the deposit, whichever is shorter. The license may be renewed for a maximum period of ten years on each renewal; subject to the licensee being able to demonstrate the continued economic viability of mining the deposit and that the licensee has fulfilled the obligations specified in the license and is not in breach of any provision of Proclamation No. 68/1995.

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

- Royalty for all minerals produced (see below)
- Income tax in accordance with the Proclamation No.69/1995 x License renewal fee
- Annual rental fees for license areas (as described above)
- Additionally, the holder of a license 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 net smelter 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 net smelter 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. 69/1 995 Proclamation to Provide for Payment of Tax on Income from Mining Operations. A holder of a ML 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, re-investment deduction and permitted losses.

If any licensee transfers or assigns, wholly or partially, any interest in the license, 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 ML 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.

The licensee may enter and occupy the land covered by the license during its term, and may use the land for activities in support of mining operations. Further the licensee may use surface and subsurface water found in the license area for consumption including mining operations, provided that such use does not result in a substantial reduction in the amount of water available to satisfy the needs of other users.

4.3.1 Environmental

The Eritrean Government's mining legislation outlines two key provisions for social environmental impact assessments (SEIA) 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 SEIA must be completed and submitted before a mining license 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) and the Ministry of Land, Water and Environment.

A SEIA will be conducted 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. The most relevant environmental Eritrean Proclamations and Legal Notices to the Asmara Project are summarized below.

4.3.2 Land use

Land use regulations are described in Land Proclamation, No. 58/1994 which provides that all land is owned by the State and citizens have land use rights only. Under this Proclamation, farmers have the right to use the land for a lifetime with priority given to relatives to inherit the property if significant investment has been made on the land.

In further elaborating the implementation process of Proclamation No. 58, the Eritrean Government also introduced:

- Proclamation No. 95/1997, A Proclamation to Provide for the Registration of Land and Other Immovable Property
- Legal Notice No. 31/1997, A Regulation to Provide for the Procedure for Land Allocation and Administration

Proclamation No. 95/1997 repealed previous legislation regarding registration of land and immovable property, established a Cadastral Office under the Ministry of Land, Water and Environment to administer land, and addressed the registration of land and transfer of immovable property erected over land.

Legal Notice No.31/1997 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 fundamental lines of the land reform were reiterated in the official Macro-policy document. Its ultimate confirmation was the Eritrean Constitution, which was ratified during May 1997, Article 23, which states:

1. Subject to the provisions of Sub-Article 2, of this article, any citizen shall have the right, anywhere in Eritrea, to acquire and dispose (of) property, individually or in association with others, and to bequeath the same to his heirs or legatees.
2. All land and all natural resources below and above the surface of the territory of Eritrea belong to the State. The interested citizens shall have land, which shall be determined by law.
3. The State may, in the national or public interest, take property, subject to the payment of just compensation and in accordance with due process of law.

Although there are laws in place to manage the allocation of land, the land allocation process currently in use in the highlands of Eritrea, is based on the traditional system, rather than the legal system. All of the land is allocated to families or groups of families within each village for residential, farming or herding. The administration of each village manages this allocation with a rotation of lands every 12 years. The overall amount of land available for agriculture for each family or group of families is often reduced due to the requirement for the national government to utilize land for new residences, new infrastructure, or for industrial development, or as a direct result of the increase in population. The intent of the Land Use regulations is to allow peasant farmers to use the land on a more permanent basis.

4.3.3 Water Resources

The Ministry of Land, Water and Environment (Water Resources Department) has drafted a water law and efforts to finalise and pass into legislation are in progress. The law manages institutional and regulatory issues, water use, water rights, environmental issues and water quality.

More recently, Eritrean Water Proclamation No. 162/2010, has resulted in some of the objectives of the above noted water policy to be brought into law. These are:

- The conservation and protection of water resources from pollution
- The identification of owners and priority users of the resource
- Promotion of integrated water management and studies to support management
- Promotion of public awareness and participation in water conservation, protection and management of water
- Establishment of the Minister’s mandate and responsibilities, such as:
    - the power to develop standards/guidelines for water quality dependent upon use (drinking, irrigation) and wastewater discharge water quality standards
    - review Environmental Impact Assessments regarding water issues, and approve, suspend, or cancel permits
    - issue new water regulations
- A description of the permitting process and various possible conditions of such permits
- Establishment of who and under what conditions fees and charges will be applied to water use
- Outlines monitoring functions of the State and consequences of failure to comply with permit and or the Water Proclamation

4.3.4 National Heritage

There are no integrated laws that deal 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 with efforts being made to finalise the law in legislation. The law will take care of 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

4.4 Issuer's Interest

SGC owns 100% of the Asmara property. Upon the project completing a feasibility study and conditional upon the granting of a mining license, the Government of Eritrea will have a 10% carried interest in the project and exercised the option on July 4, 2012 to purchase up to a 30% working interest. Sunridge and the Eritrean National Mining Corporation (ENAMCO) are in negotiation for the cost of their acquisition and structure of the joint venture company.

The economics and financial analysis of this Technical Report assumes a 100% interest by SGC.

4.5 Royalties, Back-In Rights, Payments, Agreements, and Encumbrances

The granted tenements are subject to an agreement with Mr. A. Perry entitling him to a 2% net profit interest in any profits derived from mining operations conducted on the three exploration licenses that comprise SGC's Asmara North project, and there is an unregistered charge over the first US$900,000 of revenue derived from the sale of any minerals recovered from the Debarwa EL to WMC (Overseas) Pty. Limited ("WMC"), a subsidiary of the Australian mining and exploration company Western Mining Corporation, itself subsequently owned by BHP Billiton.

SENET has relied on mineral tenure information supplied by SGC. SENET has not undertaken, nor is qualified to undertake, an independent verification of this information.

4.6 Environmental liabilities

Preliminary site inspections and baseline characterization of soil and water quality have been completed for the proposed mine sites. No obvious pre-existing environmental liabilities have been identified on the Emba Derho, Adi Nefas or Gupo mine sites. On the Debarwa mine site, pre-existing environmental liabilities include two exposed waste rock dumps (approximately 25,000 m^3 in total volume) developed during previous mining operations, and fuel storage, fuel use and sanitary waste treatment associated with the exploration camp. Preliminary characterization of the waste rock dumps indicates that they are a source of acid and metal leaching.

4.7 Permits

4.7.1 Exploration License

Sunridge Gold Corp. (Eritrea) is the holder of the Medrizien, Adi Nefas and Debarwa Exploration Licenses following assignment of the license from Sub-Sahara Resources (Eritrea) Limited on 13 April 2007. The Assignment Agreement, signed by the Minister of Energy and Mines of the State of Eritrea, transfers and assigns all title and obligations to SGC. The Medrizien License covers the Emba Derho while Adi Nefas covers Adi Nefas and Gupo deposit areas, whereas the Debarwa License covers the Debarwa deposit area.

4.7.2 Mining License and SEIA Permitting Process

The application for a mining license from the Eritrean Government includes the submission and review of a Feasibility Study and the SEIA Report. Although the permitting process is not formalized or officially documented, both the Mining License and environmental approval were required for the previously approved mining license for the Bisha Property and Zara Property. SGC will be submitting a permit application post Feasibility Study and the production of an SEIA is scheduled for third or fourth quarter 2013.

4.7.3 Other Permitting Requirements

Water use, land use and construction permits are likely to be required under the Water and Land Proclamations of Eritrea. Although the process and requirements are evolving with the mining industry in Eritrea, it is noted that these permits/licenses have previously formed part of the Mining Agreement for operating mines in Eritrea.

4.8 Significant Factors and Risks

The Property is subject to the same risks as any other mining project in newly industrialised nations, including under-estimation of capital and/or operating costs, delays or inability in securing land agreements, the inability to attract qualified personnel, personal security risks, poor communication with stakeholders, over-estimation of mineral resources and metal prices, inflation of capital and/or operating costs, complicated geotechnical conditions, and changes in the Project mine plan. There are no other known significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Asmara is the capital city of Eritrea and hosts the country's main international airport which is serviced by a number of international airlines. A recently refurbished narrow gauge railway links Asmara with the port of Massawa on the Red Sea coast, 75 km east of Asmara. The Asmara Project is located in the Central Highlands region of Eritrea.

The Emba Derho project has good road access all year round, with a network of sealed bitumen roads and well-maintained secondary dirt roads, which lead to smaller settlements. A paved road north of the city traverses the north-eastern corner of the Medrizien license area within 2 km of the Emba Derho deposit. Emba Derho is located 12 km northwest of the Asmara city centre.

The Adi Nefas and Gupo deposits are located 6 km north and 4 km north of the Asmara city centre, respectively, and are accessed by well-maintained dirt-roads.

Access to the Debarwa EL is by sealed bitumen road and well-maintained secondary access road to the proposed mine site over a distance of 25 km from Asmara. A paved road south of the town of Debarwa traverses the Debarwa EL within 1 km of the deposit.

5.2 Climate

Climatic conditions are described separately for Debarwa and the Asmara North properties (Emba Derho, Adi Nefas, and Gupo Gold deposits). Climate and hydrology studies in both the Asmara North region and at Debarwa have been on-going since 2008. The Asmara North region has an arid climate, with a mean annual temperature of 16 °C. The lowest temperatures tend to occur in January, and highest temperatures in May. Debarwa also has an arid climate, with a mean annual temperature of 19 °C. Similar to Asmara North, the lowest temperatures tend to occur in January, and the highest in May. Annual lows and highs range between roughly 5 and 30 °C for both regions. Freezing conditions have not been recorded and are not expected based on the regional and site specific data available.

The majority of precipitation in both regions occurs in July and August, the season called 'the big rains'. The mean annual precipitation for the Asmara North deposits is estimated to be 534 mm, of which roughly 65% falls during July and August, 17% in June and September, and 12% in April and May. The mean annual precipitation at Debarwa is estimated to be 440 mm, of which roughly 68% falls during July and August, 15% in June and September, and 11% in April and May. The remaining precipitation occurs as minor rainfall events that do not occur with enough intensity, duration, or frequency to result in measurable surface water runoff. Evaporation is estimated to be approximately 1,200 mm.

5.3 Local Resources and Infrastructure

Asmara has a population of approximately 550,000 people and is the main commercial centre of the country. The potential for training local personnel in mining and related activities is considered good as there is a well-educated and willing population in close vicinity to the deposits.

A national grid power system is in operation with the main thermo-electric power plant located at Hirgigo adjacent to Massawa on the Red Sea coast. An additional smaller 15 MW plant at Beleza, contains three Wartsila heavy fuel oil (HFO) generators, one of which is operational on-demand and the other two are either not operational or in need of repair.

Essentially all drinking and process water is obtained from wells and the provision of water in large volumes would need to be addressed in the event of an industrial-scale mining operation.

5.4 Physiography

The Asmara Project area lies at an altitude between 1,900 m and 2,400 m AMSL, with the property forming the higher lying ground near the edge of the Red Sea escarpment. Numerous deeply incised ravines and gullies define the escarpment edge. Vegetation cover is sparse in terms of tree coverage, with most of the arable land under cultivation.

6 HISTORY

Information in this section has been excerpted from Hall *et al*., (2008), Gribble *et al*., (June 2009) and Hopley *et al*., (2011) and updated.

6.1 Introduction

The early pre-modern history of mining in Eritrea is documented in detail in Blackburn and Chisholm (2004). The early mining history in the region was largely directed towards gold-bearing quartz vein systems, particularly during the Italian era. Old workings are numerous and widespread. Mining and exploration experienced a hiatus for over 20 years as a result of the struggle for Eritrean independence. Potential remains for the discovery of new gold and base metal deposits as evidenced by the Bisha Project of Nevsun Resources Ltd. and other discoveries since the formation of the independent State of Eritrea.

6.2 Ownership History

The mineral tenements comprising the Asmara Project were originally held by Ashanti Gold Fields (Medrizien License), Western Mining (Debarwa License) and La Source (Adi Nefas License). Phelps Dodge Exploration Corporation (“Phelps Dodge”) subsequently acquired the Debarwa and Medrizien Licenses in 1997 and 1998 respectively. Phelps Dodge transferred and assigned all titles and obligations to Sub-Sahara, a wholly owned subsidiary of Sub-Sahara Resources NL. This title was defined by an exploration agreement signed by Sub-Sahara, Phelps Dodge and the government of the State of Eritrea in November 2001. The properties were also covered by a separate joint venture agreement between Sub-Sahara and Africa Wide Resources on an 80:20 ownership basis respectively. At the same time, Sub-Sahara Resources was also granted the Adi Nefas License in November 2001.

SGC signed a Letter of Intent with Sub-Sahara on August 21, 2003, whereby SGC was granted an option to earn up to a 90% joint venture interest in the Asmara Project by funding staged exploration expenditures. Sub-Sahara remained the operator of the ensuing Joint Venture until 2 September 2004 when all exploration work was halted following an edict from the government. SGC became operator of the Joint Venture following resumption of exploration in January 2005. The Sub-Sahara/SGC Joint Venture received notice from the Minister of Energy and Mines in Eritrea in February 2005 advising that the renewal date of the three ELs that make up the Asmara Project had been extended to 30 May 2006.

On 16 January 2006, SGC announced that it had provided notice to both its partners of its intention to exercise its option to acquire their combined 60% interest in the Asmara Project for shares of SGC such that SGC would own 100% of the Asmara Project. On 1 March 2006 SGC announced that negotiations among the parties were unable to establish a fair value for the project, acceptable to all parties. As a consequence and pursuant with the Option Agreements between the parties, SRK Consulting (South Africa) (Pty) Limited (SRK Consulting) was appointed to complete an independent valuation of the project. The resulting valuation was effective as of 13 January 2006 and reported on in compliance with the VALMIN Code 2005 in Van der Merwe *et al* (2006). TSX-V regulatory approval for the purchase of the aggregate 60% interest held by Sub-Sahara and Africa-Wide was obtained in October 2006. The Asmara Project ELs were extended for a further year until 28 May 2007 being the third anniversary date. On 13 April 2007 the Minister of Energy and Mines of Eritrea signed the Assignment Agreement allowing for the transfer of 100% of the Asmara Project ELs to SGC. Under the terms of the Sale Agreement dated 22 August 2006, SGC has issued a total of 9.5 M common shares of the company to Sub-Sahara and Africa-Wide.

6.3 Exploration History

6.3.1 Emba Derho

Emba Derho is the largest known mineral deposit within the Medrizien EL. The Emba Derho gossan is exposed at surface approximately 2,000 m southwest of the Emba Derho village. Most of the early mining history in the area is related to gold exploitation. It is likely that local inhabitants mined gold on various scales for much of the last 3,000 years.

The 20th century exploration history in the region is described in Blackburn and Chisholm (2004). By 1902, the English-owned company Société per Le Miniére d'Oro was active in the Asmara-Medrizien area. MacLaren (Blackburn and Chisholm, 2004) reported a production estimate of approximately 100 kilograms (kg) of gold for Eritrea for 1907. Official mining appears to have ceased at the onset of the First World War in 1914. Mining started up again in the early 1930s and by 1940 some 12,000 ounces (oz.) of gold per annum were being produced. The Second World War (1939-1945) again curtailed mining activities.

The Medrizien EL essentially covers the core of the old Hamasien Goldfield and most of the more significant historical Italian gold mines are contained within the current license area. These include the Medrizien, Sciumagalle, Regina de Saba and the Hara Hot mines. The Medrizien mine was one of the largest and is located some 11 km northwest of Asmara. Both open pit and underground mining was focused on a 2 km long gold-bearing sulphidic quartz vein and took place from 1902 to 1914 and again between 1931 to at least 1963 (Blackburn and Chisholm, 2004). The old Sciumagalle mine is located 5 km to the south of the Medrizien mine. As at Medrizien, mining was focused on a prominent gold-bearing quartz vein within a large vein field. Apparently no historical production records are available for these operations.

The prominently outcropping Emba Derho gossan and its potential significance was first discovered by the Ethiopian Geological Survey who investigated the gossan with 7 core holes during the period 1969-1970. More recently Ethio-Nippon carried out exploration programs in the Asmara region during the 1970s, including drilling of two holes at the Emba Derho gossan. This work was abandoned after 1974. All of the above drillholes intersected massive sulphides with low but potentially significant base metal grades.

Ashanti (Eritrea) carried out exploration during the period 1996-1998 including drilling 3 core holes along a section line across the Emba Derho massive sulphide gossan, which returned similar values to those recovered by Ethio-Nippon. Limited surface geochemical work (trenching and soil sampling) was also carried out at some of the old colonial gold mine sites within the license. The primary data for this work does not appear to be available (Blackburn and Chisholm, 2004). Ashanti relinquished the license in April 1998.

Phelps Dodge acquired a license within the current Medrizien EL in May 1998. Work undertaken included stream sediment, rock chip and trench sampling, geological mapping, and a Time-Domain Electromagnetic (TDEM) survey. Trenching results revealed large widths of low grade (approximately 0.1 to 0.5 grams per tonne (g/t)) gold values with some higher grade samples ranging up to 30 g/t. Zinc values were uniformly low suggesting intense leaching, while copper values of approximately 1,000 to 5,000 parts per million (ppm) across the horizon indicated considerable dispersion (Blackburn and Chisholm, 2004). A single diamond hole was drilled to test a TDEM anomaly located about 200 m south of and along strike of known mineralization. A number of apparently sub-parallel massive sulphide lenses were intersected. The drilling results were reviewed and compiled by Blackburn and Chisholm (2004).

The Adi Nefas EL was granted to Sub-Sahara in late 2001, at the same time the rights to the Debarwa and Medrizien ELs were transferred to Sub-Sahara from Phelps Dodge, via Africa Wide

Resources (Stokes *et al*., 2007). Sub-Sahara undertook a digital data compilation of previous work as well as trenching and rock chip sampling over the various properties. An airborne electro-magnetic (EM) and magnetic survey was flown over the Adi Nefas, Debarwa and Medrizien license areas in late-2003.

Sub-Sahara entered into an option agreement with SGC in August 2003, but remained the operator until a government shutdown of exploration in September 2004. SGC became operator of the Joint Venture following resumption of exploration in 2005. In early 2006, SGC exercised its option to purchase a 100% interest in the Asmara Project from Sub-Sahara and since that time has focused on drill definition of the Emba Derho and other deposits. The exploration history of the Emba Derho deposit is summarised in Table 6.1.

There are no significant historical mineral resource estimates to be reported for the Emba Derho property.

6.3.2 Adi Nefas

The Adi Nefas gossan is exposed at surface in the immediate vicinity of the Adi Nefas village and can be traced more or less continuously along strike for some 700 m. The gossan was reportedly discovered in 1967 and was investigated from 1968-69 by the Ethiopian Geological Survey who completed 9 diamond drillholes for a total of 954 m. Although the original data are no longer available, the results of this work appear in secondary reports and have been quoted by Blackburn and Chisholm (2004).

The UN Mineral Survey conducted geophysical surveys from 1970 to 1971 and outlined anomalies coincident with the Adi Nefas gossan. Ethio-Nippon Mining Company undertook geochemical sampling and diamond drilling at Adi Nefas from 1971 to 1973, coincident with their advanced exploration at Debarwa. A total of 24 diamond drillholes were completed and an “ore reserve” reported that pre-dates NI 43-101.

The Adi Nefas deposit was included within the license granted to LaSource Développment SAS (“La Source”) in 1996. Although much of their effort was directed towards the Gupo gold deposit, La Source undertook soil and stream sediment geochemistry over the Adi Nefas VMS deposit and outlined soil geochemical anomalies coincident in part with the altered rocks and gossans (Stokes *et al*., 2007) Fourteen trenches were excavated across the Adi Nefas gossan Channel sample results from the trenches were reported to range up to 8.5 g/t Au with an average of 2.54 g/t Au, over widths of 2 m to 8.5 m. La Source did not drill test the gossan. The project was farmed out to Rift Resources Ltd (“Rift”) in May 1999, who completed only two diamond drillholes before the outbreak of hostilities related to the struggle for Eritrean independence forced the cessation of work Both drillholes intersected massive sulphides and the results of the drilling were in line with previous work. La Source relinquished the Adi Nefas license in 1999.

The Adi Nefas license was granted to Sub-Sahara Resources (Eritrea) Ltd (“Sub-Sahara”) in late 2001, at the same time the rights to the Debarwa and Medrizien ELs were transferred to Sub-Sahara from Phelps Dodge, via Africa Wide Resources (Stokes *et al*., 2007). Sub-Sahara undertook a digital data compilation of previous work and trenching and rock chip sampling to the south of the Adi Nefas gossan.

Sub-Sahara entered into a joint venture (JV) option agreement with SGC in August 2003, but remained the operator until a government shutdown of exploration in September 2004. Until this point, Sub-Sahara had completed only 6 new drillholes for 1,200 m on Adi Nefas.

SGC became operator of the JV following resumption of exploration in 2005, and completed 50 new drillholes for almost 7,000 m during the course of 2006. In early 2006, SGC exercised its option to purchase a 100% interest in the Asmara Project from Sub-Sahara and since that time has focused on drill definition of zones of known mineralization. The exploration history of the Adi Nefas deposit is summarized in

Table 6.2. There are no significant historical mineral resource estimates to be reported for the Adi Nefas deposit

6.3.3 Gupo

The Gupo deposit, also known historically as Adi Nefas Doop, was exploited by Italian miners during the colonial period in Eritrea and is within the Adi Nefas Exploration License. La Source undertook soil sampling, mapping, geophysical investigations and drilling between 1996 and 1998. Approximately 10,500 m of reverse circulation drilling (119 drillholes) and 460 m of core drilling (6 drillholes) was completed (Coumoul, 1998).

Thomas and Marais (2009) concluded that La Source reverse circulation (RC) drill sampling posed a risk in estimation of mineral resources. There are no significant verifiable historical mineral resource estimates to be reported for the Gupo deposit.

6.3.4 Debarwa

A detailed account of the exploration history at Debarwa is given in Blackburn and Chisholm (2004). Much of the discussion below has been taken from this source. The Debarwa gossan and associated barite are exposed at surface. The gossan was reportedly discovered in 1955, however due to the lack of secondary copper minerals, the significance of the gossan was not realised at the time. The Geological Survey of Ethiopia undertook a 38 hole (4,586 m) diamond drilling program from 1966 following the discovery of significant copper mineralization in a hole designed to test the barite potential. The drill program delineated a mineralised zone having a strike length of 1,300 m and a width up to 200 m.

In the early 1970s the Ethio-Nippon Mining Company (a joint venture between the Ethiopian Government and the Nippon Mining Company) completed a 75-hole (12,005 m) diamond drilling program over a strike length of 1,500 m at Debarwa. Approximately 50 of these holes were drilled over a strike length of 200 m in the area of best gossan development. A “productive ore reserve” estimate was reported and a feasibility study undertaken into mining the deposit.

In 1972 a vertical timber-lined shaft was sunk to a depth of 136 m (of a planned vertical depth of 240 m) and 379 m of development driven. The shaft is also intersected by an adit, which is at a level above the supergene zone and at the lower limit of the gossan. A 2,000 tonne bulk sample was extracted and sent to Japan for metallurgical testwork. The bulk sample is reported to have had a grade of 13.8% Cu (in Blackburn and Chisholm (2004). The Debarwa “mine” was abandoned in March 1974 following an attack by Eritrean independence forces during the struggle for Eritrean independence. The head frame and underground workings were observed by the authors during the site visit. The mine shaft is reportedly flooded to 50 m below surface, however the upper part of the shaft and the adit are dry and the shaft is accessible via the adit.

Piles of supergene and primary massive sulphide showing copper oxide alteration were noted alongside two large jaw crushers on the old crusher pad, and are likely remnants of the material reported to have been shipped as part of the metallurgical bulk sample in 1973-1974. Drill core from the above historical campaigns was disturbed during the War of Independence and is contained within a dump in an old core storage building at the Debarwa Camp.

Apparently no further field work was completed at Debarwa from 1974 (when work was abandoned due to the War of Independence) until Eritrea gained independence from Ethiopia in 1993. The newly formed Eritrean Government called for expressions of interest and in April 1996, WMC was granted the license to explore and develop the Debarwa deposit. Over the next year WMC spent US$981,000 on a re-interpretation of the original drilling and various surface geochemical and geophysical surveys. Detailed rock chip channel sampling was carried out over the surface exposure of the gossan, leading to an average gold grade of 4.82 g/t.

WMC relinquished the property in April 1997 and the license was granted to Phelps Dodge in July 1997. Phelps Dodge spent a total of US$1,161,215 on exploration on the license from that time until March 1999. During this period they carried out various geochemical and geophysical programs aimed at evaluating the known mineralization. Phelps Dodge also drilled a total of 12 diamond drillholes for 1,944 m into the Debarwa mineralised horizon.

Not all of the original data from the above programs is still available.

The Debarwa EL was granted to Sub-Sahara in late 2001, at the same time the rights to the Debarwa and Adi Nefas ELs were transferred to Sub-Sahara from Phelps Dodge, via Africa Wide Resources (Stokes *et al* ., 2007). Sub-Sahara undertook a digital data compilation of previous work as well as trenching and rock chip sampling over the various properties. A TEMPEST airborne EM and magnetic survey was flown over the Asmara Project, comprising the Adi Nefas, Debarwa and Medrizien license areas in late-2003.

Sub-Sahara undertook a 9 drillhole diamond drilling program totalling 450 m focused on the Debarwa Main zone up to August 2003.

Since acquiring 100% interest in the Asmara projects from Sub-Sahara, SGC has focused on drill definition of the Debarwa and other VMS deposits, as well as a property-wide exploration program in search of new drill targets. The exploration history of the Debarwa deposit is summarised in Table 6.3.

Table 6.1: Exploration Historical Summary – Emba Derho

Geological Survey of Ethiopia (1967-1971)	
"Discovery" of gossan outcrops	
Drilling	7 diamond drillholes
Ethio-Nippon Mining Company (1971-1974)	
Drilling	2 diamond drillholes
Ashanti (Eritera) (1966-1998)	
Surface Geochemistry	Limited soil sampling in the vicinity of historical Au prospects
Trenching	
Drilling	
Phelps Dodge (1998)	
Surface Geochemistry	Stream sediment and rock chip sampling
Trenching	11 trenches
Geological mapping	
Ground Geophysics	Electromagnetic Survey
Drilling	1 diamond drillhole
Sub-Sahara Resources (Eritera) Ltd. ("Sub-Sahara") (2001-August 2003)	
Data Intergration	Digital database of previous exploration work established
Geochemistry	Rock chip sampling
Drilling	2 diamond drillholes
Sunridge/Sub-Sahara Joint Venture (August 2003-22nd August 2006 [date of sale Agreement with Sub-Sahara])	
Geophysics	TEMPEST airborne EM and magnetics flown in late 2003 by Fugro Airborne Surveys (Australia) over Adi Nefas,Debarwa and Medrizien (5,441 line km at 200 m line spacing)
	Regional gravity survey covering Asmara Project area by MWH Surveys (300 m x 300 m diamond shaped grid)
	Ground gravity surveys (43.3 line km; 80 m x 20 m grid)
	Ground gravity surveys (3.6 line km); Ground Pulse Electro-magnetic (EM) surveys. Down-hole pulse Electro-magnetic surveys; Audio -Magneto -Telluric (AMT) surveys
Geochemistry	5 samples for whole rock geochemistry (Barrie)
Mapping	1:3500 geological mapping by C.J Greig
Drilling	51 diamonds holes (10,592 m) drilled between december 2005 and August 2006
Sampling and Assay	4,894 drill core samples submitted for assay
Metallurgy	Ongoing metallurgical testwork
Environmental	Baseline studies initiated

Table 6.2: Exploration Historical Summary – Adi Nefas

Geological Survey of Ethiopia (1967-1971)	
Discovery of gossan outcrops	
Drilling	9 diamond drillholes for 954 m
Historical resources	1.0 Mt at: • 2.0% Cu • 15.0% Zn • 0.8% Pb • 120 g/t Au
UN Mineral Survey (1970-1971)	
Geophysics	Outlined anomalies coincident with the Adi Nefas gossan
Ethio-Nippon Mining Company (1971-1973)	
Geochemistry	Soil and rock chip sampling
Drilling	24 diamond drillholes
Historical resources (non NI 43-101 compliant)	1.0 Mt at: • at 1.4% Cu • 13.0% Zn • 1.6% Pb • 4.0g/t Au • 160 g/t Ag (4.5m width; S.G. = 4.0)
La Source and Rift Resources (1996-1999)	
Geochemistry	Soil and stream sediment geochemistry
Trenching	• 14 trenches for 2 523 m • sidewalls sampled at 3 m intervals-samples assayed for gold only
Drilling	2 diamond drillholes (449.4m)
Sub-Sahara Resources (Eritrea) Ltd. ("Sub-Sahara") (2001-August 2003)	
Data integration	Digital database of previous exploration work established
Geochemistry	Rock chip sampling of old pits to the south of the Adi Nefas gossan (26 samples)
Trenching	1 trench, 54 samples, Adi Nefas south area
Geophysics	Ground magnets (approximately 50 line km)

Table 6.3: Exploration Historical Summary – Debarwa

Geological Survey of Ethiopia (1966-1971)	
"Discovery" of gossan and barite outcrop	In 1955
Drilling	38 diamond drillholes (4,586m) delineated a zone of mineralization with strike length 1,300 m and width up to 200 m
Historic Mineral Resources/Reserves*	"Ore Reserve Estimate" of 1.5Mt at 1.62% Cu*
Ethio-Nippon Mining Company (1971-1974)	
Drilling	75 diamond drillholes (12,005 m) over a strike length of 1,500m
Historic Mineral Resources/Reserves*	"Productive Ore Reserve Estimate" of 0.6Mt at 7.63% Cu, 1.29g/t Au, 30g/t Ag, 1.8% Zn
Feasibility Study	New "Ore Reserve" estimate of 2.6Mt at 5.72% Cu, 1.45g/t Au, 37.81g/t Ag
Mining	Shaft sinking; adit and underground development
Metallurgy	2,000 tonne underground bulk sample extracted
WMC (Overseas) Pty. Ltd. (April 1996-April 1997)	
Mapping	1:50 000 (250 km^2); 1:10 000 (3.4 km^2); 1:4 000 (2 km^2); 1:1 000 (10.4 km^2)
Geochemistry	Regional stream sediment survey (370 samples); orientation stream sediment survey (38 samples)'; follow-upstream sediment survey (74 samples); soil geochemistry orientation survey (39 samples); detailed soil geochemistry survey (992 samples); lag sampling survey (166 samples); gossan sampling survey (22 samples).
Geophysics	Ground magnetic survey (35.3 line-km); gravity survey (35.3 line-km); TEM survey (531 readings); Induced polarization (IP) chargeability/resistivity surveys
Historic Mineral Resources/Reserves*	1.62 Mt at an average grade of 4.91% Cu and 1.22 g/t Au (re-estimation based on Ethio-Nippon data)
Phelps Dodge Exploration Corporation (July 1997-November 2001)	
Detailed mapping	1.2km^2
Geochemistry	Soil geochemical sampling (236 samples); rock chip and channel sampling (185 samples); 4 trenches for 178 m and 89 samples; 12 pits (30 samples)
Geophysics	Ground magnetic surveys; TEM, CSAMT and combined TEM/CSAMT surveys; IP chargeability/resistivity surveys; downhole TEM and downhole radial IP surveys.
Drilling	12 diamond drillholes for 1,945 m (538 samples)
Historic Mineral Resources/Reserves*	1.65 Mt at an average grade of 5.10% Cu and 1.40 g/t Au
Sub-Sahara Resources (Eritrea) Ltd. ("Sub-Sahara") (November 2001-August 2003)	
Data integration	Digital database of previous exploration work established
Geochemistry	4 trenches over previously unsampled gossans NE of Debarwa Main returned significant anomalous Zn and Cu (600 m; 302 samples)
Geophysics	Reinterpretation of previous ground geophysical surveys
Drilling	9 diamond drillholes for 450 m in the Debarwa Main zone

** Non-NI 43-101 compliant*

7 GEOLOGICAL SETTING AND MINERALIZATION

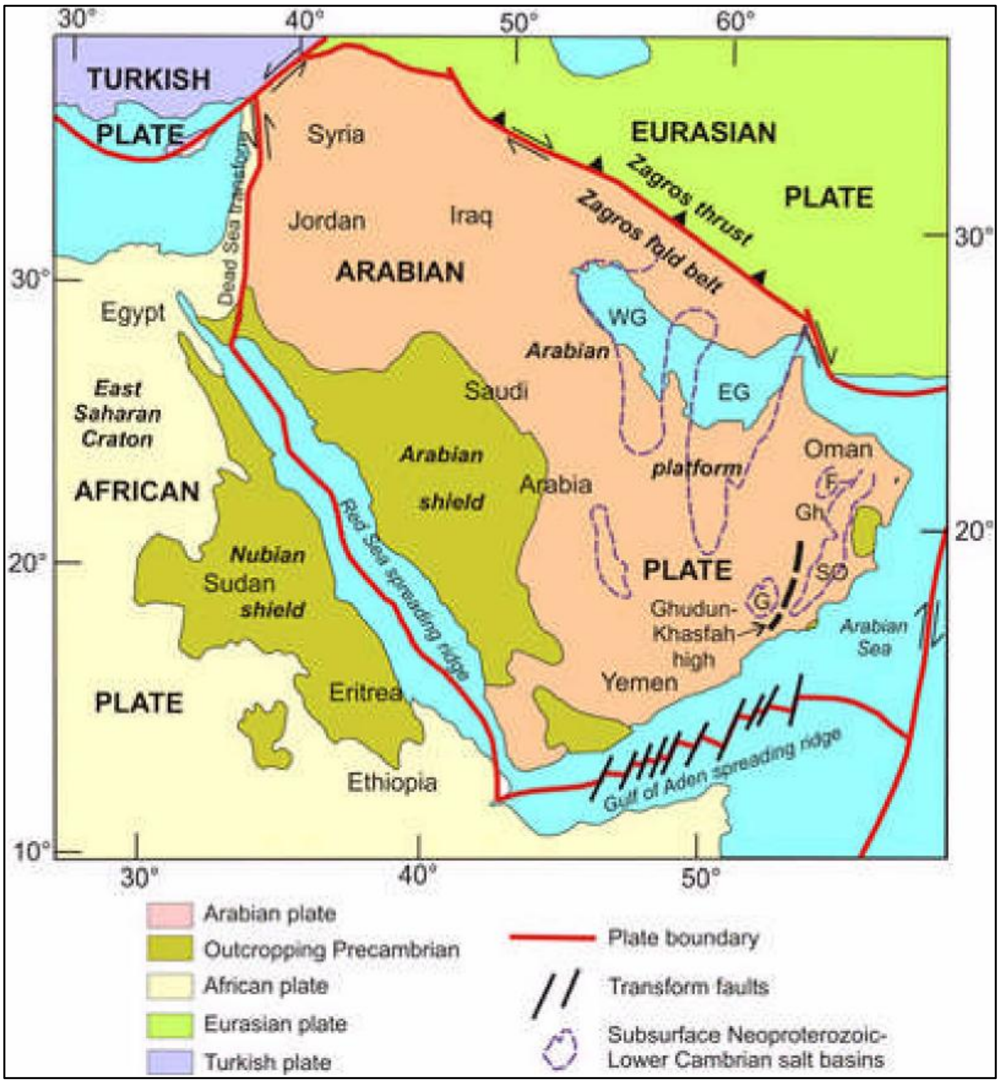
Information in this section has been excerpted from Hall *et al*., (2008), Gribble *et al*., (June 2009) and Hopley *et al*., (2011) and updated.

The regional and local geological setting of volcanogenic massive sulphide (VMS) deposits in Eritrea has been reviewed by Barrie (2004). The following sections describe the regional to property scale geological setting and mineralization relevant to this study.

7.1 Regional Geology

The Neoproterozoic Arabian-Nubian Shield (ANS) represents a composite granitoid-greenstone belt terrain that straddles the Red Sea and covers much of Eritrea, parts of Egypt, Sudan and Ethiopia, and the western part of Saudi Arabia (Figure 7.1). Eritrea is underlain by the western or Nubian portion of the ANS. The ANS is geologically similar to other granitoid-greenstone belt terrains in Canada and Australia, which contain significant VMS and Au deposits (Barrie, 2004). The Shield is distinguished from other such granitoid-greenstone belt terrains by the relatively high proportion of intermediate to felsic volcanic rocks and derived siliclastic rocks.

Figure 7.1: Geological Setting of the Arabian Nubian Shield

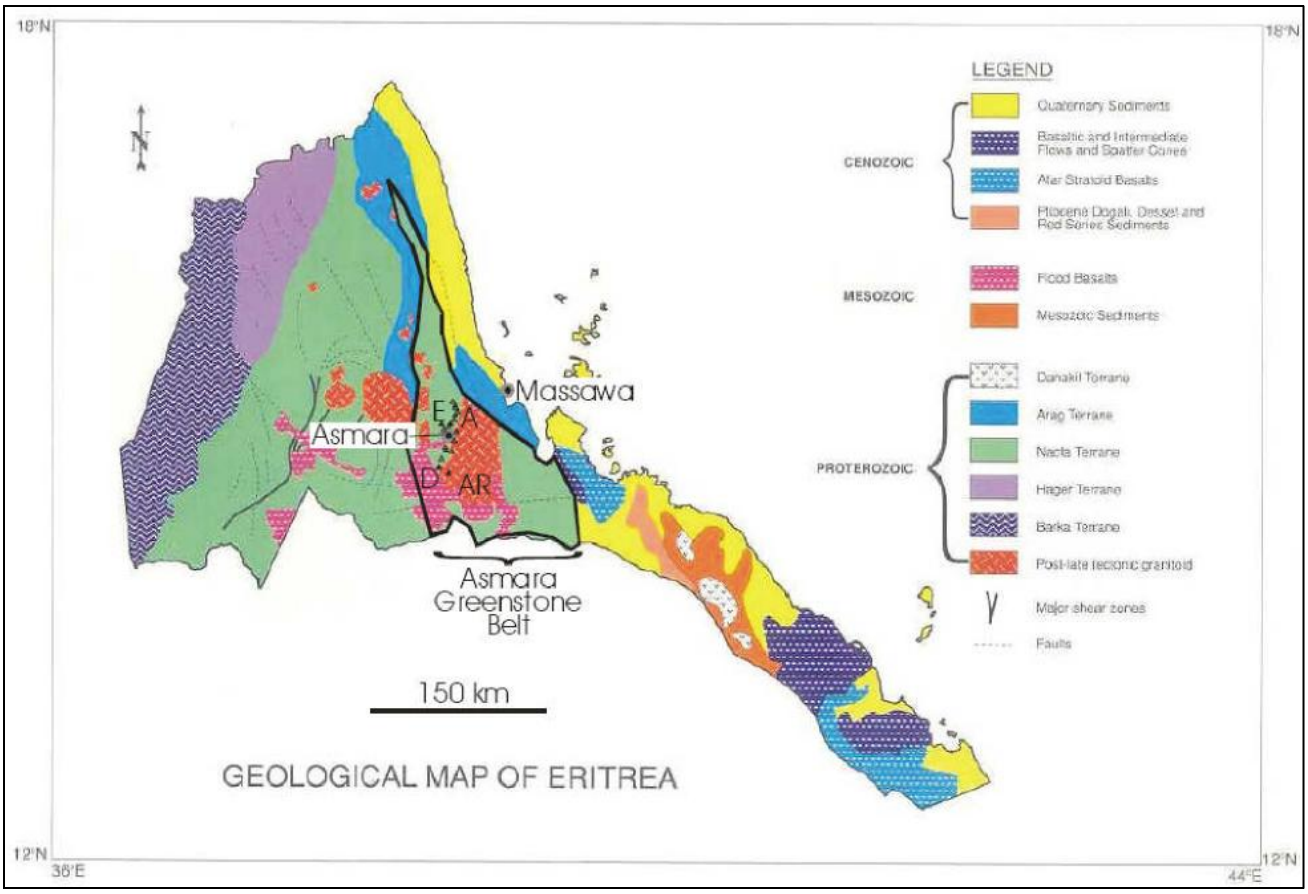


Source: Saudi Geological Survey

The ANS is recognised as a collage of distinct Precambrian Tectonostratigraphic terrains believed to have converged and amalgamated between 870 Million years (Ma) ago and 650 Ma ago in a fashion similar to accreted magmatic arc terrains in the Phanerozoic. This amalgamation, deformation, metamorphism and uplift culminated in the Nabitah orogeny which was accompanied by intrusion of late-to post-tectonic granitoids, and partially covered by overlap assemblages deposited in sag and rift basins. The ANS subsequently separated into two with the rifting and opening of the Red Sea which began around 26 Ma. During this period the ANS was partly covered by subaerial flood basalts resulting in the creation of the Red Sea escarpment.

Four Tectonostratigraphic terrains are identified in the ANS in Eritrea. The westernmost Barkan terrain comprises upper amphibolite to granulitic predominantly metasedimentary rocks. The Hagar terrain is made up of basaltic and siliclastic rocks including minor mafic-ultramafic blocks; while the narrow Adobha Abiy terrain comprises highly deformed metasedimentary rocks (including carbonates). The Nakfa terrain underlies much of the central part of the country and is made up of mixed volcanic and metasedimentary (siliclastic and carbonate) rocks. The Nakfa terrain also contains the Asmara greenstone belt VMS deposits and the Bisha VMS deposit. The location of the SGC Asmara Project within the Asmara Syncline of the Nakfa terrain is shown in Figure 7.2.

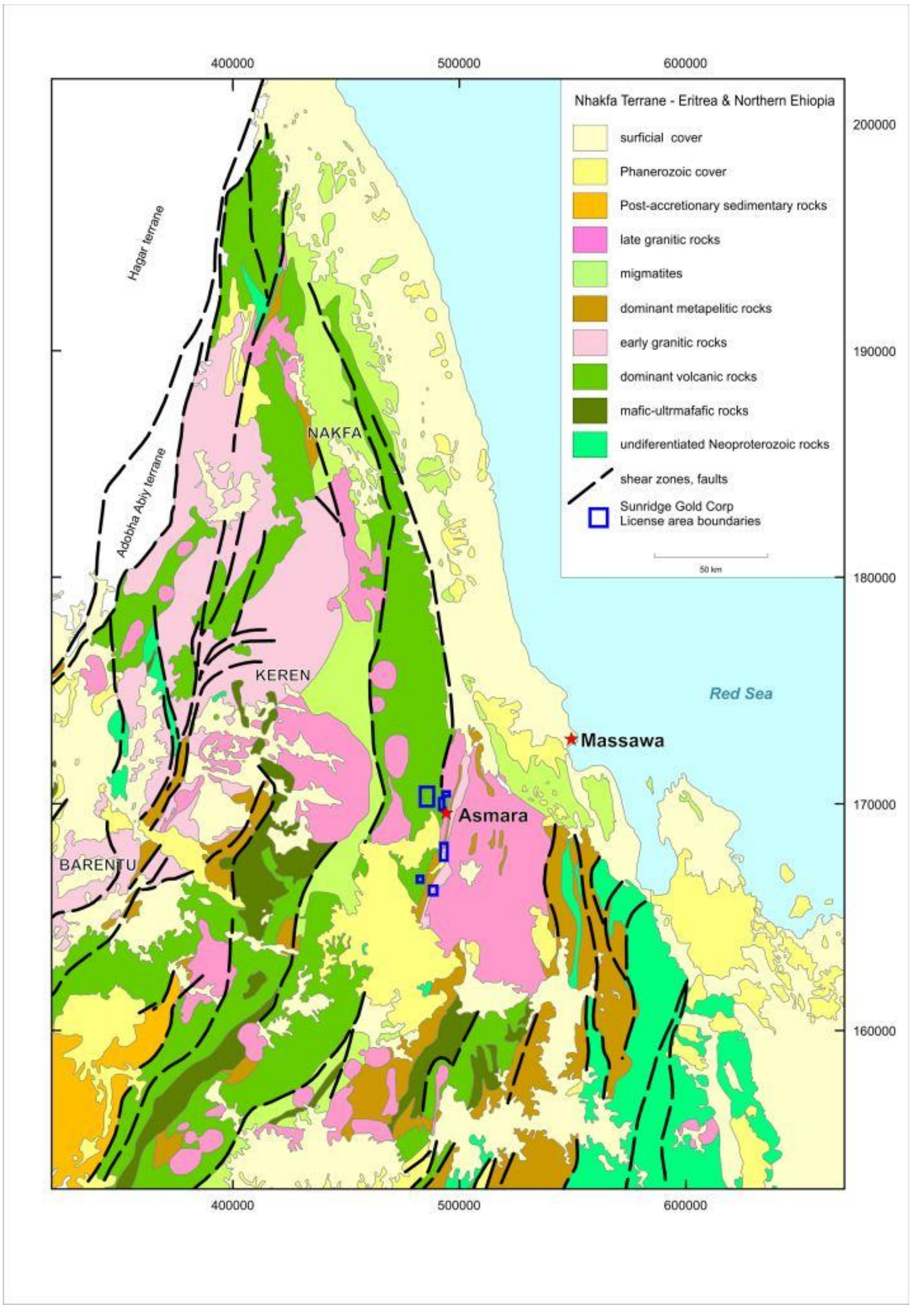
Figure 7.2: Simplified Geological Map of Eritrea



(after Barrie, 2004). E - Emba Derho; A – Adi Nefas; D – Debarwa; AR – Adi Rassi)

The so-called Asmara greenstone belt comprises several Tectonostratigraphic blocks including the Adi Neared block, the Central Steep belt, the Asmara Syncline, and other blocks to the north. That part of the granitoid-greenstone belt in the vicinity of Asmara and within the license areas represents a moderately evolved belt typical of the granitoid-greenstone terrains of the ANS. The sequence is dominated by mafic to felsic flows and tuffs with a predominant calc-alkali affinity and lesser siliclastic rocks, typical of moderately evolved island or continental arcs. In contrast Archean and Paleoproterozoic granitoid-greenstone belts that contain abundant VMS and gold deposits (for example, Abitibi and Norseman-Wiluna belts) contain appreciably more tholeiitic basalts, and are generally more primitive and oceanic (Barrie, 2004).

Figure 7.3: Main Structural and Geological Features in Eritrea and Ethiopia

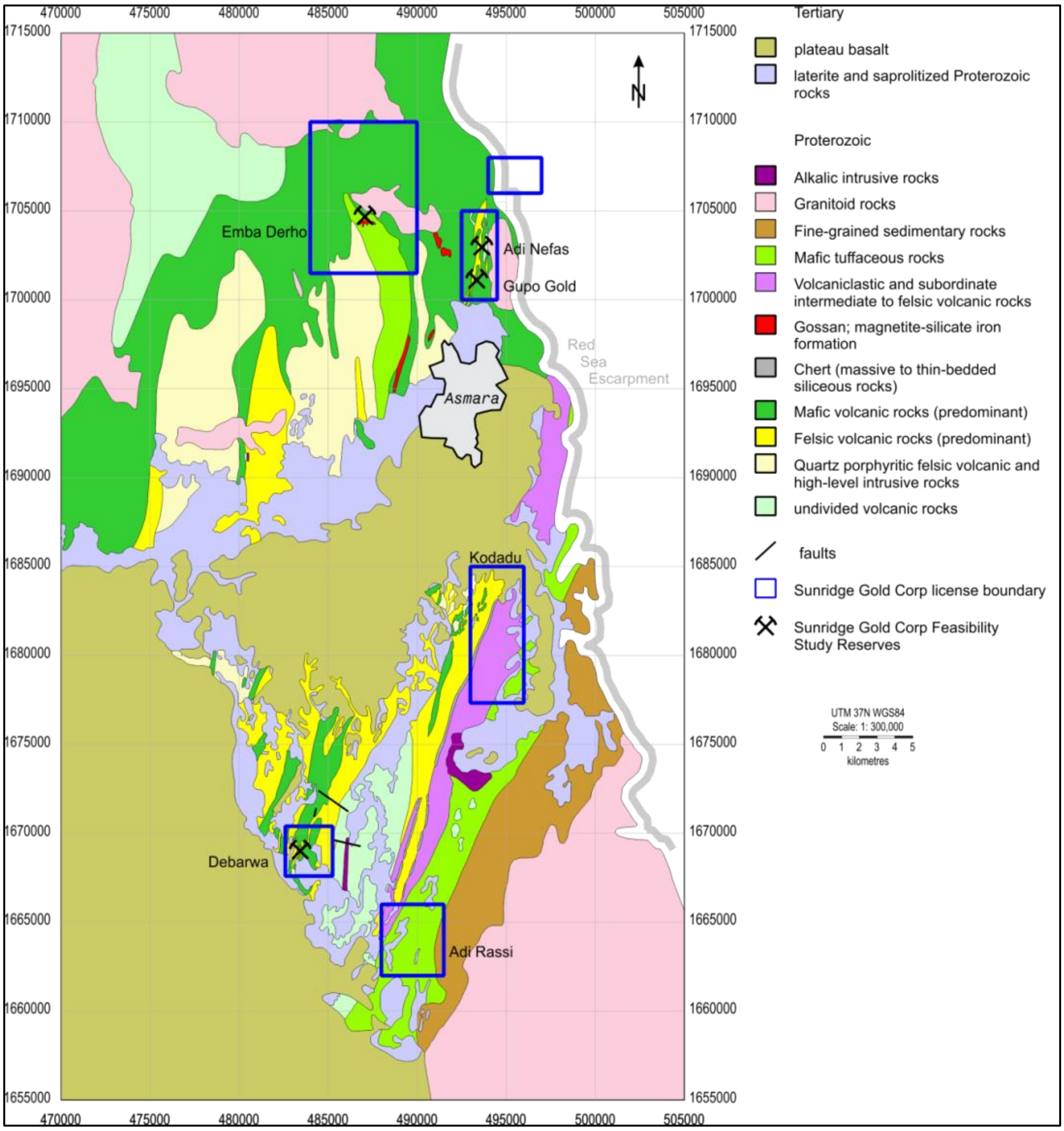


Source: SGC

7.2 Deposit Scale Geology

The Asmara Project is underlain by rocks assigned by Drury and de Souza Filho (1998) to the Asmara Syncline of the Nakfa terrain (Figure 7.3). The improvement in the understanding of the local stratigraphy is in large part due to the activities of exploration companies working in the area. A simplified geological map of the Asmara Project area has been compiled by SGC (Figure 7.4). These rocks are generally east-facing, tightly folded about northerly trending fold axes, generally dip to the east or east-southeast at 45º to 85º and are preserved at or below lower greenschist facies. They consist primarily of a bimodal suite of volcanic and derived volcaniclastic rocks that are overlain to the east by a metasedimentary sequence. A well-developed foliation parallels the regional structural-stratigraphic grain which generally trends north-northeast. A change in the foliation trend is observed north of Asmara in the vicinity of a number of syn- to post-tectonic intrusions that vary in composition and age of emplacement.

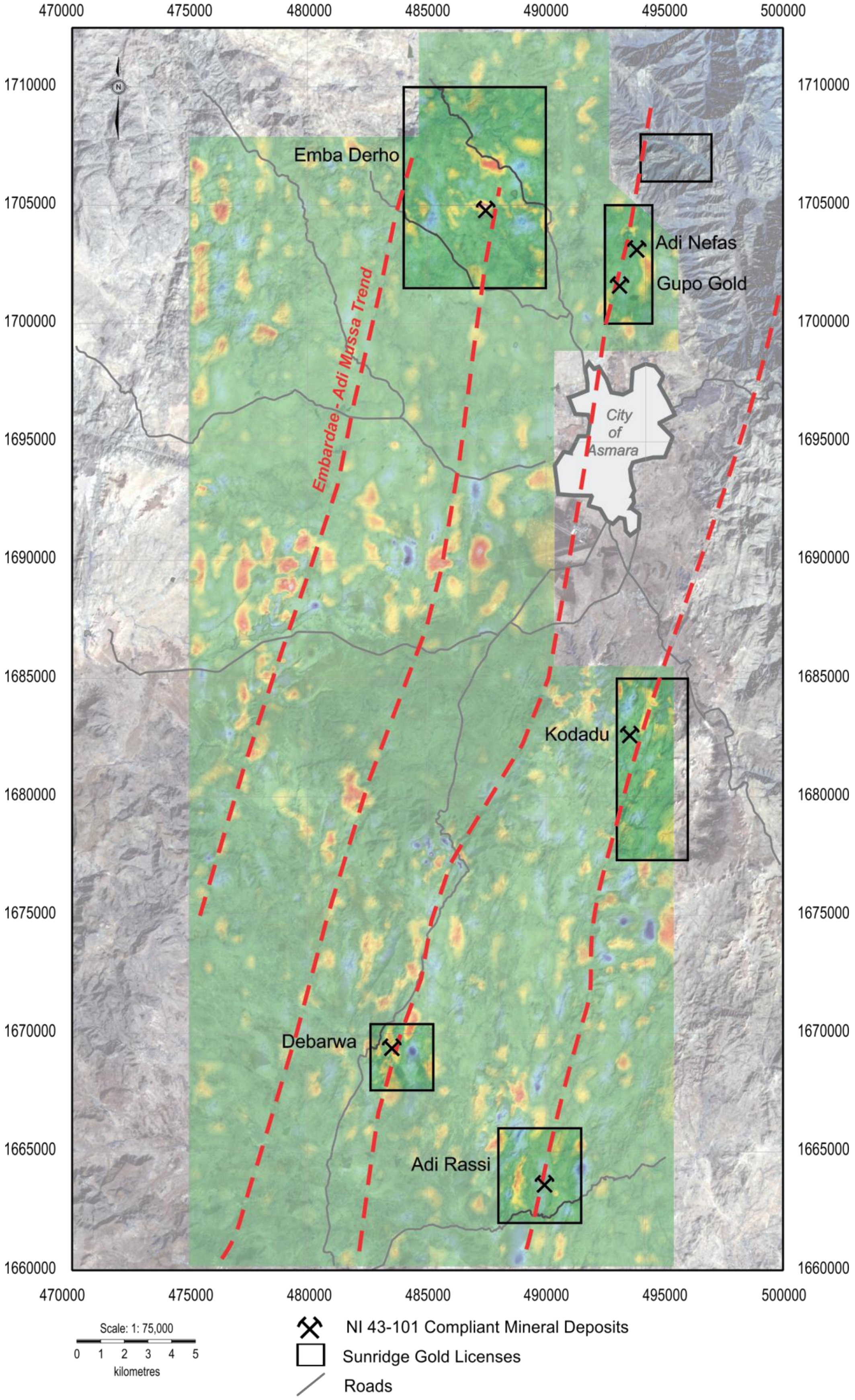
Figure 7.4: Simplified Geology of the Asmara Project Area



Source :SGC

The Neoproterozoic rocks are overlain by flat-lying Tertiary olivine basalts which reach thicknesses of over 200 m. The basalts overlie a well-developed paleo-weathering horizon along which locally thick laterites have developed and the underlying Neoproterozoic rocks are extensively saprolitised. Three mineralised trends are observed, all oriented north-northeast (Figure 7.5).

Figure 7.5: Mineralization Trends in the Asmara Project Area



Source: SGC

These are: the Emba Derho trend to the west of Asmara that includes the Dairo Paulos occurrence, Woki Duba occurrence and Emba Derho deposit; the Debarwa-Adi Nefas trend extends at least 25 km south and 5 km north of the capital and includes the Debarwa, Shiketi, Adi Lamza, Adi Nefas deposits and Adi Adieto occurrence; and the Adi Rassi-Kodato trend to the east (Barrie, 2004). The latter two trends are defined in part by a chert/exhalite unit whereas the Emba Derho trend becomes less-well delineated to the southwest where many of the metal occurrences are hosted within granitoids.

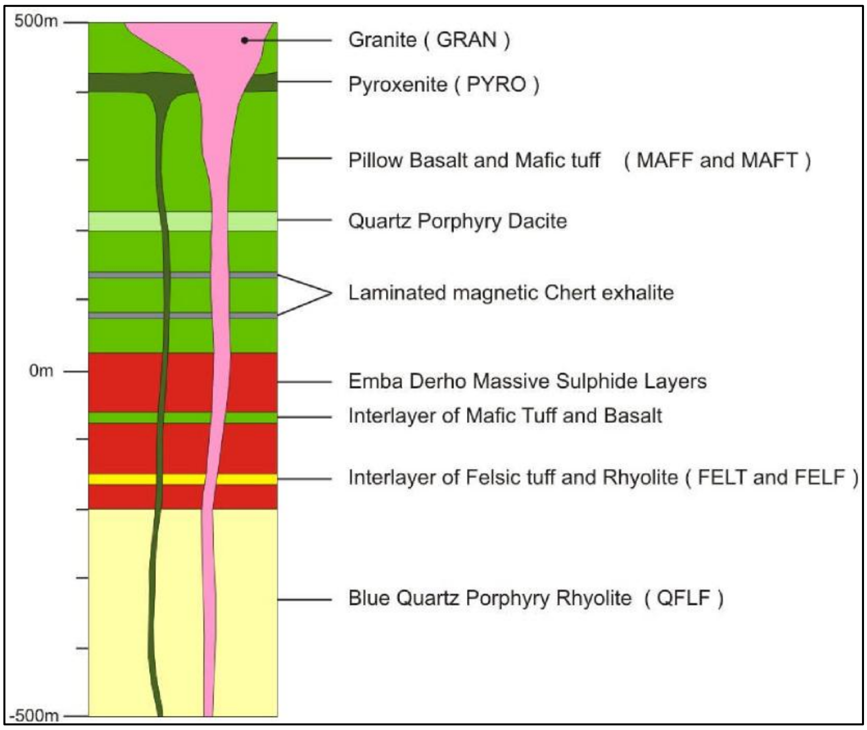
7.3 Property Geology

7.3.1 Emba Derho

The geology of the deposit and surrounding rocks is known from mapping and drilling carried out by SGC. A description of the geology of the deposit is contained in Daoud and Greig (2007) and Barrie (2004).

A stratigraphic section compiled by Barrie (2004) is shown schematically in Figure 7.6. The footwall to the massive sulphide horizons comprises blue quartz or phyric rhyolite flows, flow breccia, and associated felsic fragmental tuffaceous rocks. These are locally altered to sericite chlorite schists. The overlying mineralised zone consists of stacked layers of semi-massive to massive sulphide with numerous partings of tuffaceous and volcanic flow material (the interlayers) and one barite layer. These are all cut by various phases of post-deformation felsic dykes typically 1 m to 5 m thick. The hangingwall sequence comprises a pillow basalt and pillow breccia units with significant epidote-silica alteration. Several manganiferous, siliceous exhalite units are noted within the altered mafic volcanics just above the massive sulphide layers. A small sill of altered and deformed coarse-grained pyroxenite is also noted within the mafic volcanic flows. The entire package generally dips steeply to the north.

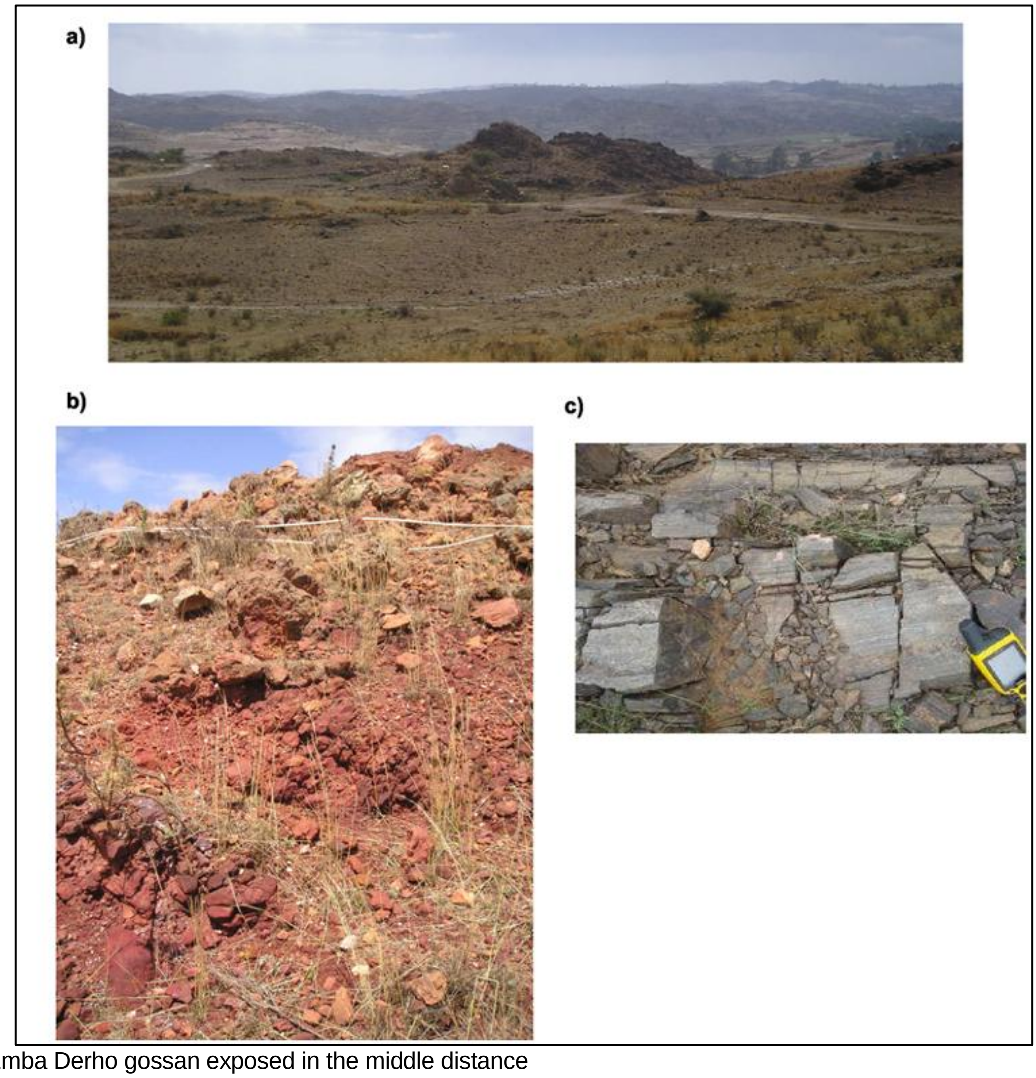
Figure 7.6: Emba Derho Stratigraphic Section



(Barrie, 2004)

The deposit is characterised on surface by a prominently outcropping gossan that measures approximately 800 m east-west by 220 m north-south. The gossan comprises oxidised and acid-altered (supergene altered) felsic tuffaceous rocks and flows, which are most likely the most abundant lithology; weathered massive to semi-massive sulphides; and orange-brown weathering rhyolite dykes (or possibly sills), which are relatively thin, being on the order of 1 m to 2 m thick at most. Surrounding the gossan are relatively poorly exposed rocks which may be considered an integral element of the gossan itself. These are typically well foliated acid-altered predominantly fine tuffaceous rocks of both mafic and felsic composition, and which, for lack of exposure, have been grouped as undivided acid-altered fine tuffaceous rocks. Other elements, also identifiable locally, but largely obscured by the processes of surficial weathering, include post-deformation granitic dykes of various compositions.

Figure 7.7: Aspects of Emba Derho Surficial Geology

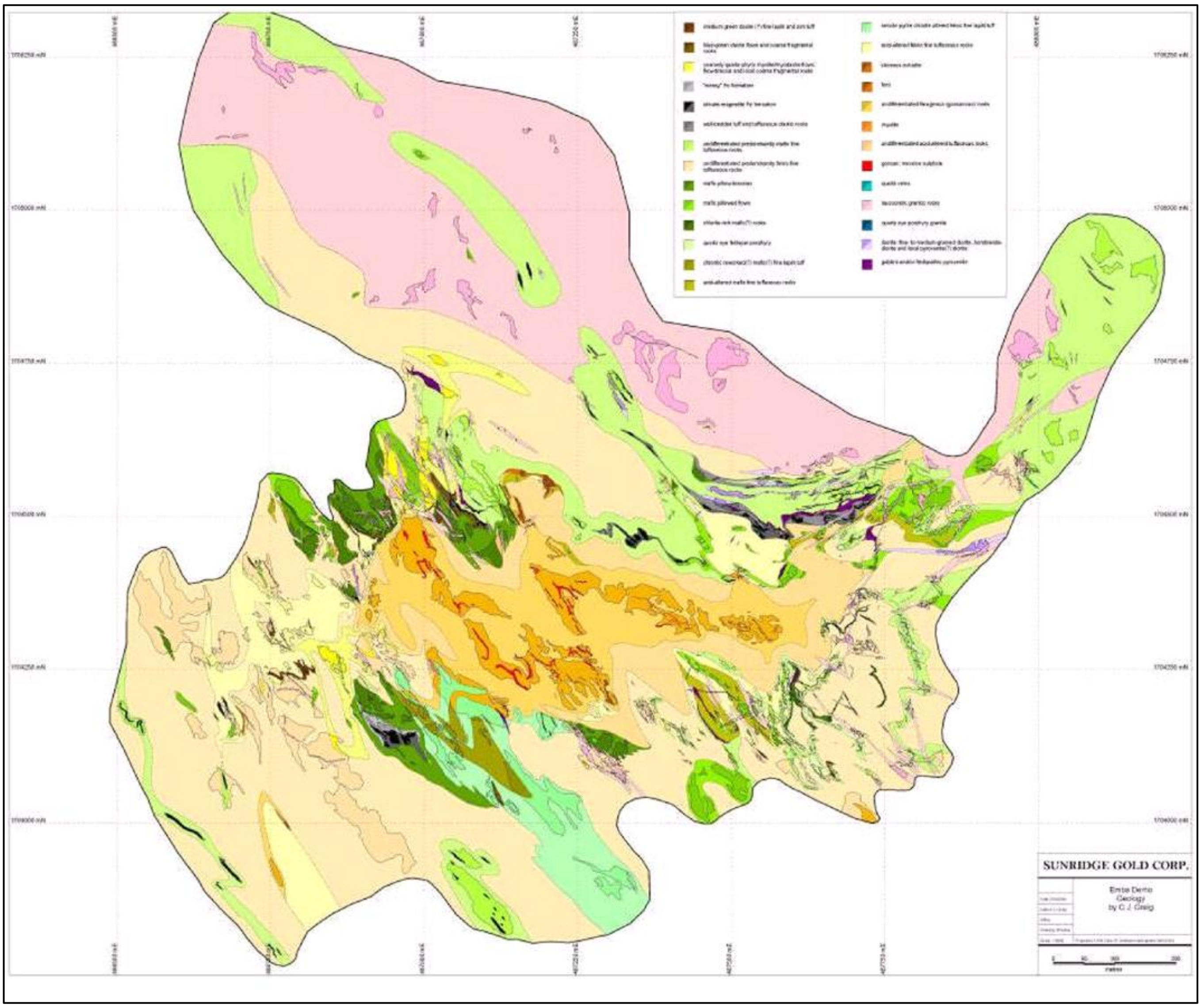


a) Folded Emba Derho gossan exposed in the middle distance
b) Close-up of the Emba Derho gossan;
c) Outcropping chert exhalite, Emba Derho.
(Source MSA)

The host rocks have been subject to at least two phases of tight folding forming a "W" fold which faces northwest, with fold axes plunging moderately to steeply to the northwest. An additional fold on the eastern extremity of the deposit plunges to the northeast. Within these fold structures, the gossan and chert exhalite units form prominent markers

The mineralization and host stratigraphy are cut by a number of phases of felsic intrusives. The most prominent of these is a suite of fine- to medium-grained post-kinematic leucocratic rocks, probably ranging from tonalite to granite in composition, which are thought to be genetically related to the high-level intrusion of similar composition which is exposed to the north of the Emba Derho deposit. In general, the granitoid dykes parallel the structural-stratigraphic trends of the host rocks they intrude, with northwest trends most common, but with north-northeast trends common in the northeast part of the area mapped. An earlier but much less voluminous suite of post-kinematic diorite dykes is also present, and they generally trend to the north.

Figure 7.8: Geology of the Emba Derho Deposit



Source: Snowden

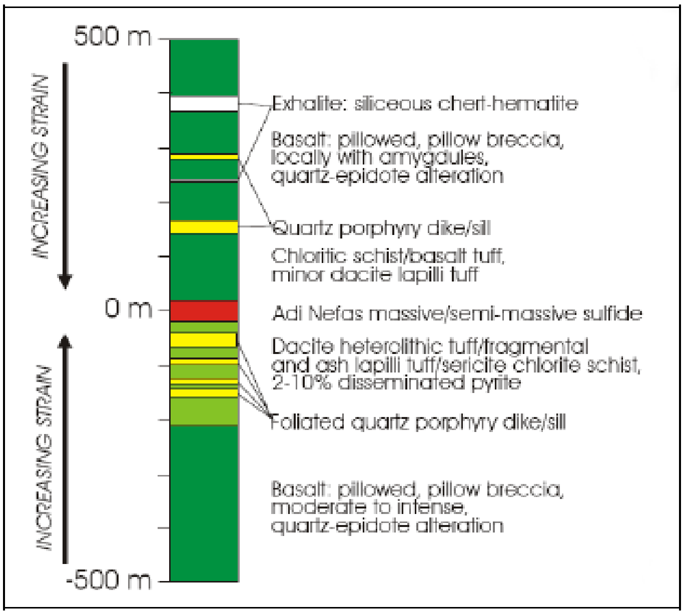
7.3.2 Adi Nefas

The geology of the deposit and surrounding rocks is known from mapping and drilling carried out by SGC and others. A description of the geology of the Adi Nefas deposit is contained in Barrie (2004) and Zeremariam (2005).

The Adi Nefas deposit is a bimodal-felsic (Kuroko) type VMS deposit with both mafic and felsic volcanic rocks present, an enriched polymetallic signature, and significant barium enrichment within the deposit itself (Barrie, 2004). The stratigraphy and dominant structural grain trend north-northeast and the sequence is subvertical to steeply east dipping (Figure 7.10). The footwall basalts on the western side are strongly epidotised and are locally epidosites. The stratigraphic section on the eastern hangingwall side comprises mainly pillow basalts and foliated equivalents, intruded by minor post mineral quartz-porphyry dykes and sills. The pillowed basalts and associated mafic metavolcanics are overlain by undifferentiated tuffaceous sediments containing silicate-magnetite exhalite lenses. A simplified stratigraphic section for Adi Nefas appears in Barrie (2004) and is reproduced in Figure 7.9.

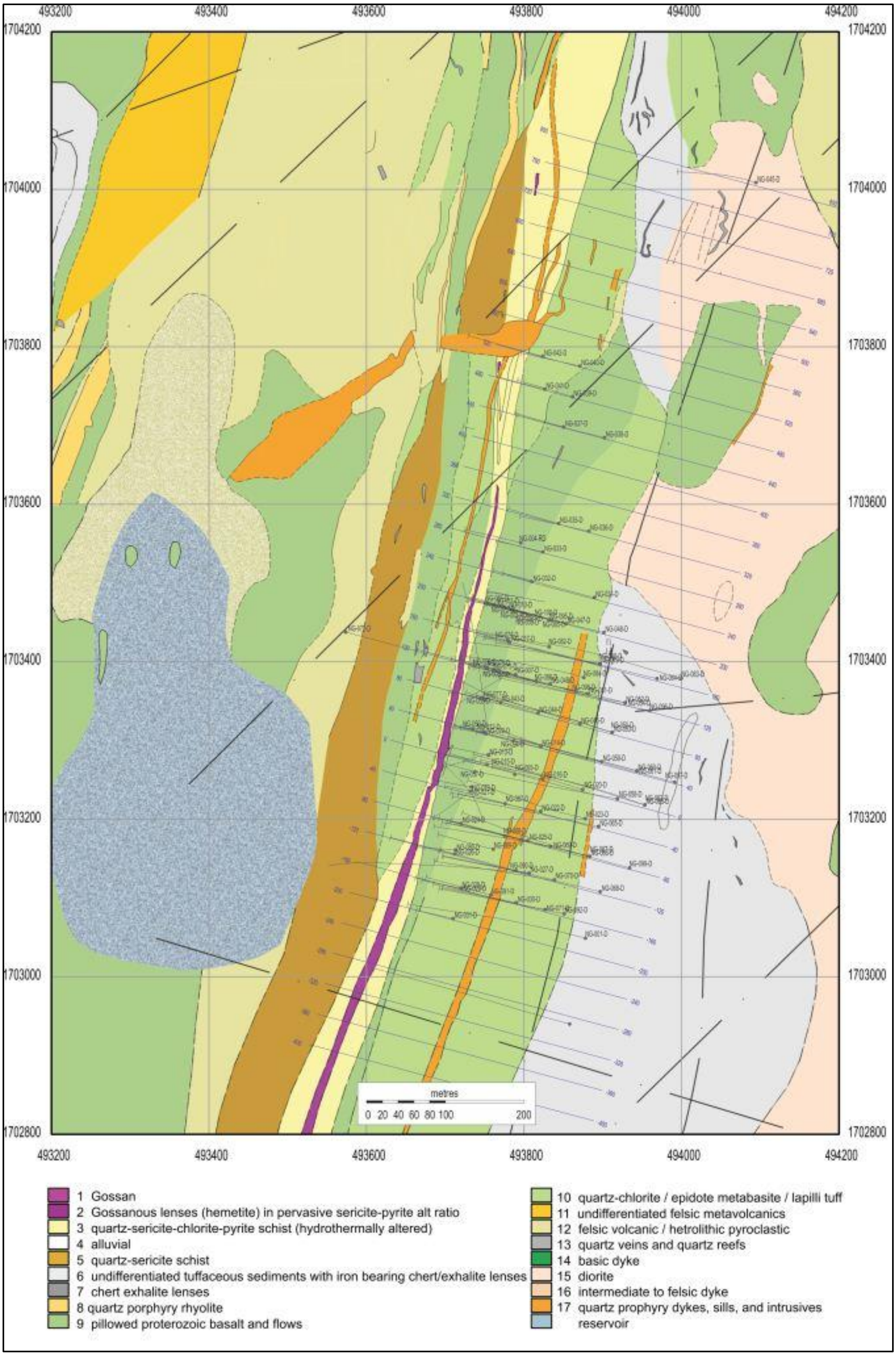
Zeremariam (2005) interpreted the property-scale structural setting to one of a tight moderately northerly plunging westerly-verging antiform, with the felsic rocks in the fold core and the massive sulphide lenses occurring on the eastern limb of the fold. The Adi Nefas gossan is well-exposed to the immediate north of Adi Nefas village and extends further northward for several hundred metres. In total the gossan can be traced more or less continuously for approximately 700 m. The Adi Nefas VMS deposit is located approximately 1.5 km north of the Gupo gold deposit. These two deposits appear to be along strike from each other and located within the same stratigraphy, although gold mineralization at Gupo is structurally controlled and related to quartz veining.

Figure 7.9: Adi Nefas Stratigraphic Section



Source: Barrie, 2004

Figure 7.10: Geology of the Adi Nefas Deposit Area



Source: SGC

Figure 7.11: Adi Nefas Gossan

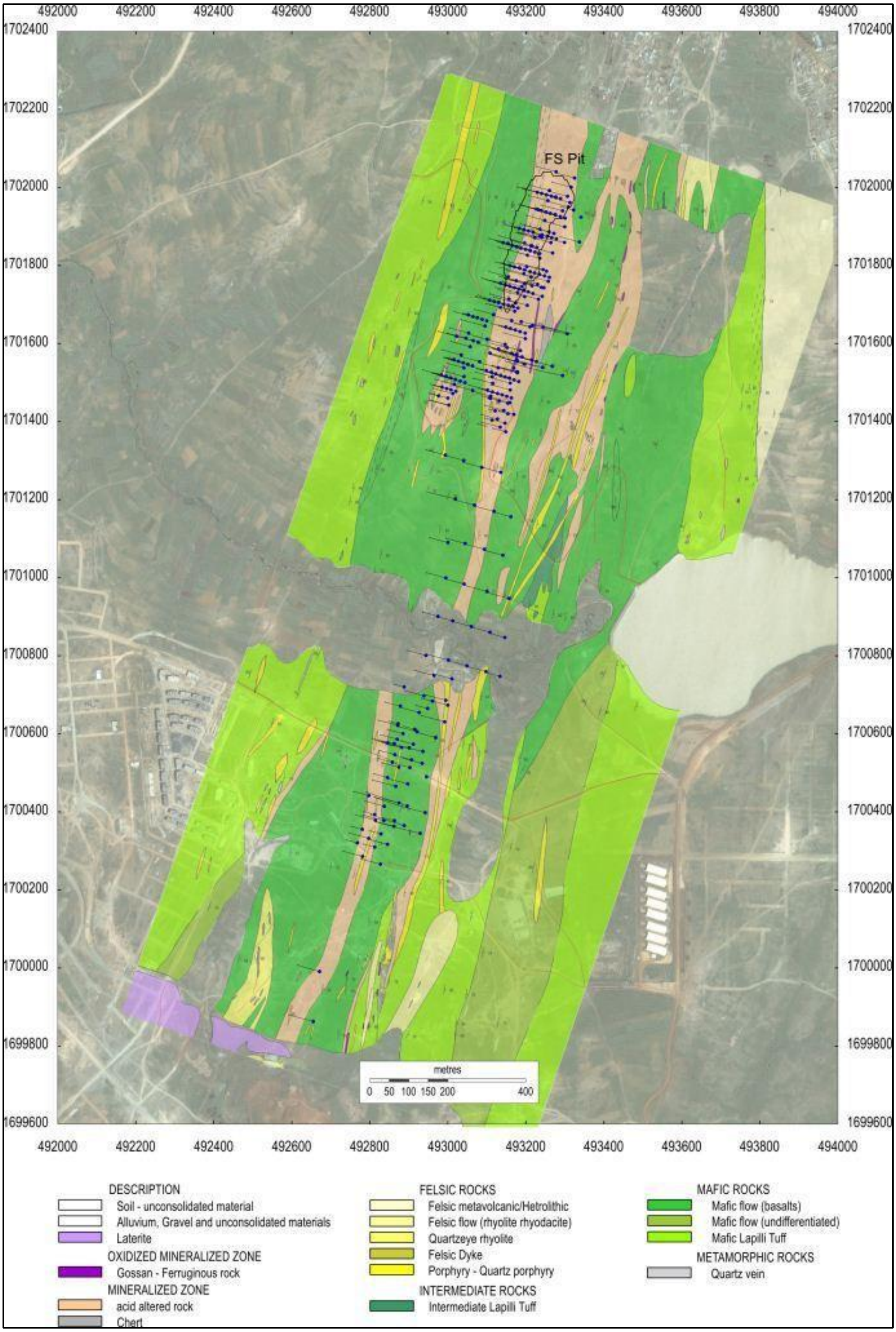
View looking south-southwest along the strike of the Adi Nefas gossan

View looking northeast towards Adi Nefas village, with felsic metavolcanics forming the immediate footwall to the gossan.

Source: MSA

The rocks hosting Gupo gold deposit are composed of highly deformed and strongly foliated mafic flows and mafic tuffs, intercalated and cross cut by thin layers of quartz phyric felsic flows and subordinate dykes. The mafic volcanic rocks show a pervasive chlorite alteration throughout, with locally epidote alterations. However within the shear zone this alteration changes to hydrothermal alteration of a combination of sericite, pyrite and carbonates (Figure 7.11).

Figure 7.12: Gupo Drillhole Paths and Geology



Source: SGC

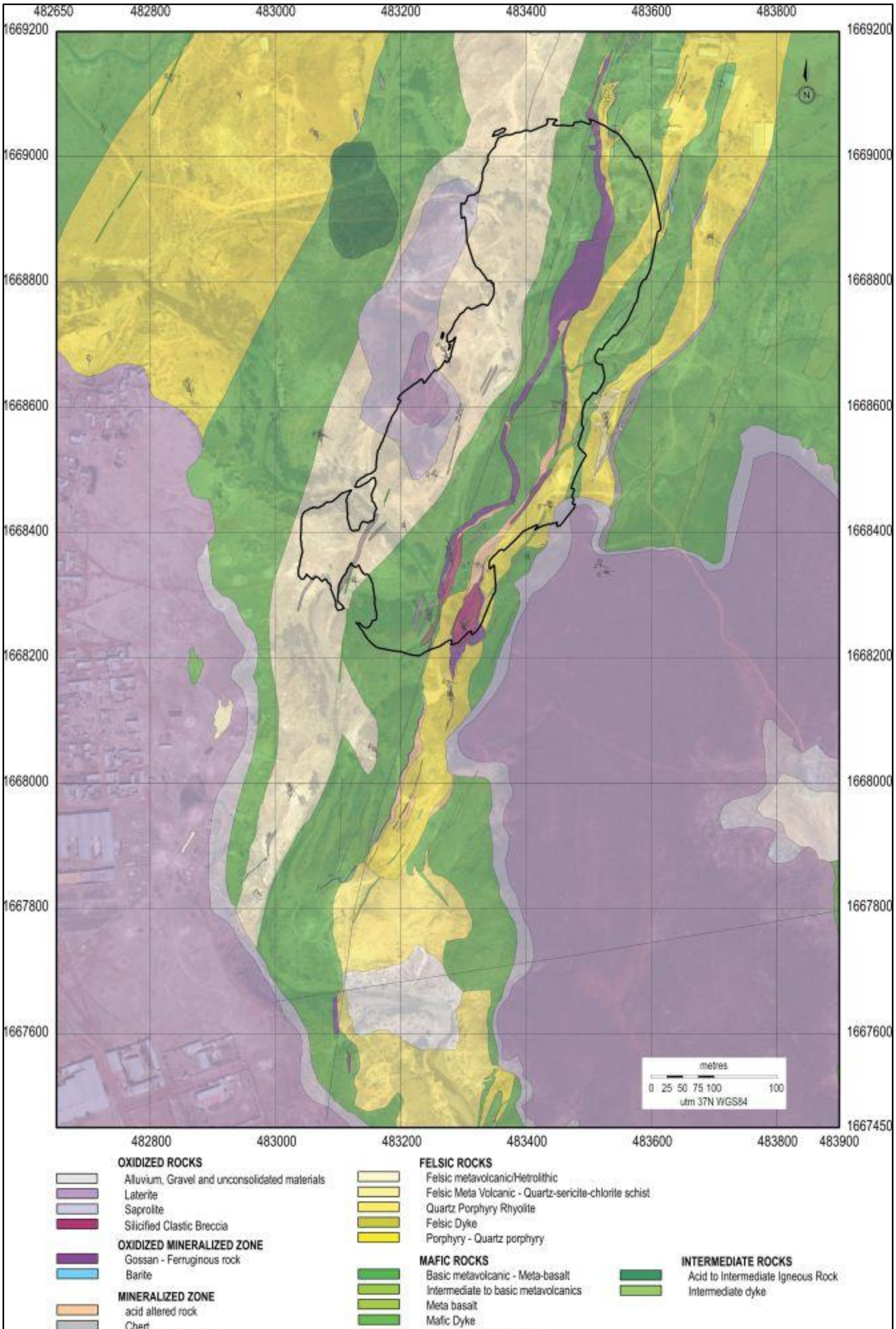
7.3.3 Debarwa

At the Debarwa deposit mineralization is hosted within an overturned sequence of variably and intensely altered, bimodal, low-K tholeiitic basalt-rhyolite that was deposited in a submarine environment along with locally interbedded volcaniclastic rocks and exhalite (Mauritsen and Arafine, 1998). The immediate host rocks to mineralization are altered felsic rocks, although mafic rocks locally mark the immediate stratigraphic footwall to mineralization. Mafic rocks also predominate within the north-northeast trending 6 km belt of prospective stratigraphy from Debarwa to the Shiketi Gossan.

The Debarwa gossan, the surface expression of the massive sulphide mineralization, has a mapped strike length of approximately 1.2 km. At the Debarwa camp, the gossan lies at the crest of a sharp, west facing ridge flanking the Gual Mereb River. Two main zones of massive sulphides are recognised: a larger lens known as Debarwa Main which is approximately 830 metres long and a smaller lens to the south, known as Debarwa South, which is approximately 285 m long. Mineralization between these two zones is narrow and intermittent – see Figure 7.13.

At Debarwa Main, the main mineralised zone, which is the most westerly, most continuous, and thickest of at least three sub-parallel mineralised horizons, dips approximately 50º to 60º to the west. This zone is approximately 8 m to 30 m wide and has been defined by SGC drilling over a strike length of more than 800 m and to a depth of about 250 m from surface. Massive sulphide mineralization is confined to the main zone and varies in thickness from less than one metre to approximately 22 m. The overlying supergene and oxide zones attain thicknesses up to 50 m. At Debarwa South the dip of the massive sulphide mineralised zone steepens from approximately 35º to 45º to the west in the north to around 60º to the west in the south, over a strike length of approximately 280 m. In general grades of mineralization at Debarwa South are more variable than at Debarwa Main, and they are generally somewhat lower. The width of mineralization is also more variable and in general, thinner, particularly in the primary sulphide zone.

Figure 7.13: Debarwa Property Geology



Source: SGC

7.4 Mineralization

7.4.1 Emba Derho

The Emba Derho deposit appears on surface as a prominently outcropping gossan, the surface expression of the massive sulphide deposit. The gossan is centrally developed over an area of 800 m by 220 m where it outcrops as a tightly folded unit with northwest oriented fold axial planes and steeply dipping limbs. Discontinuous gossan lenses have been mapped to the west of this zone. The sulphide mineralization is hosted within variably but generally heavily sulphide-altered, and moderately sericite-, chlorite-, and quartz-altered predominantly felsic metavolcanic rocks.

The primary VMS deposit is composed of several stacked and folded massive sulphide layers or lenses, which range in thickness from 5 m up to 40 m. Mineralization is partly truncated by a granitic intrusion towards the northeast. In addition the mineralised sequence is crosscut by at least five phases of felsic intrusives (monzonite granite, feldspar porphyry dyke, quartz feldspar porphyry dyke, alkali feldspar granite and felsic dykes). These intrusives typically vary in thickness from 1 m to 5 m. The following mineralised zones have been recognised by SGC geologists (Daoud and Greig, 2007):

Gossan (oxide zone): Dense, dark red-brown “true” gossan; hematite-, limonite-, goethite-, and locally magnetite-rich rocks; derived from surface weathering (oxidization) of massive sulphides. This rock type forms discontinuous and commonly folded layers which are mainly restricted to the area of the main gossan (800 m x 220 m); within this area these layers may be up to several m or more in thickness; elsewhere they occur locally as decimetre to centimetre scale layers and veins, principally within acid-altered host rocks. Locally the gossan has been remobilised to form ferruginous “ferricrete” deposits with very delicate textures. The oxide zone typically extends to groundwater level at a depth of 20-30 m.

Copper-enriched supergene zone: Fine to medium-grained, vuggy, locally sandy, massive pyrite with interstitial covellite, digenite, and minor bornite. The supergene zone occurs at or below groundwater level. Zinc is depleted relative to the primary mineralization.

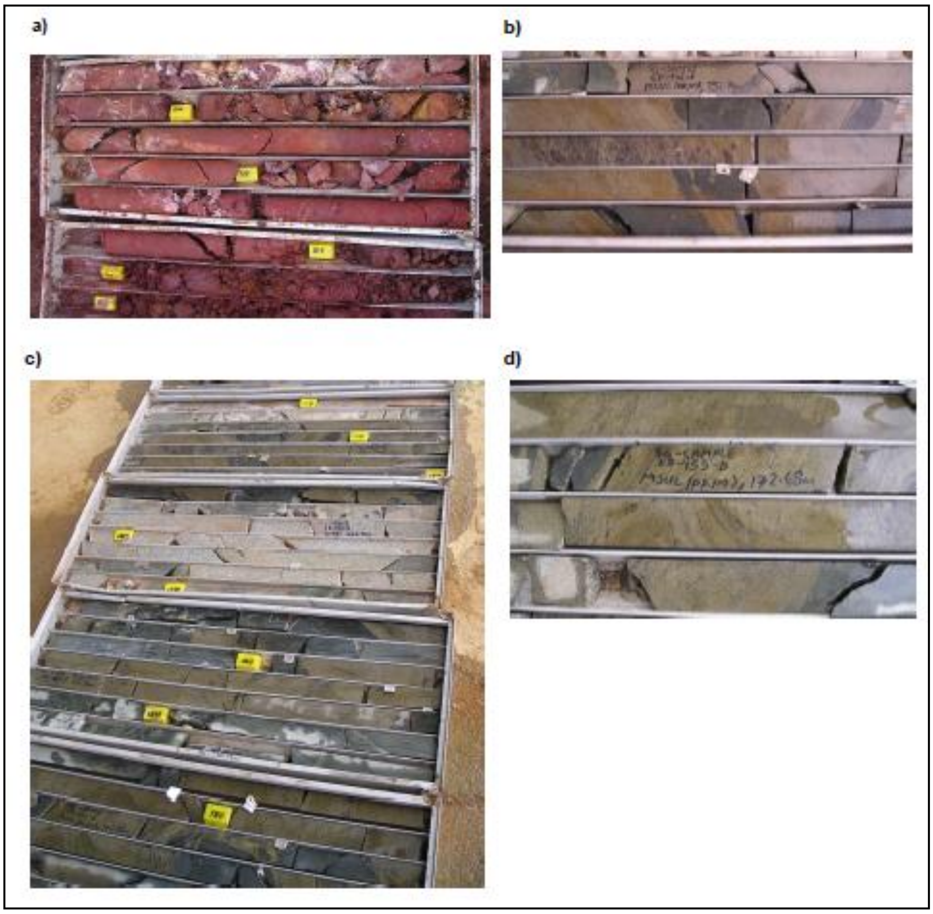
Pyritic massive sulphide primary zone: This massive sulphide zone occurs immediately below the chert and silica rich exhalite and the mafic flows; it represents the stratigraphic top of the massive sulphide event. The sulphides are mainly fine to medium-grained massive pyrite with disseminated fine-grained magnetite and very minor chalcopyrite and sphalerite.

Zinc-rich primary massive sulphide zone: The zinc-rich mineralised zone is well developed in the southern and western part of the VMS deposit. The mineralization consists of fine to medium-grained massive pyrite with interstitial sphalerite. Sphalerite also occurs as thin bands and laminae within the pyrite. At least three types of sphalerites are noted by colour variations; a rusty-brown sphalerite (probably iron (Fe)-rich and represents high temperature sphalerite), honey-yellow sphalerite, and whitish-grey sphalerite (a lower temperature phase).

Copper-rich primary massive sulphide zone: This zone is well defined in the northern part of the deposit and consists of medium to coarse-grained massive pyrite and pyrrhotite with interstitial chalcopyrite and magnetite. Chalcopyrite also occurs as massive bands, stringers, and blebs.

For the purposes of modelling, four distinct zones of mineralization are recognised. A surface oxide gold-rich zone strongly leached with respect to base metals, a zinc-poor supergene zone, overlying a zinc-rich primary sulphide zone which is in turn underlain by a copper-rich primary sulphide zone.

Figure 7.14: Mineralization Styles in Drill Core in the Emba Derho VMS Deposit



a) Gossan (oxide zone) with friable transition zone
b) Close-up of massive sulphide mineralization styles
c) Typical massive sulphide mineralization intersection with cross cutting dykes
d) Close-up of massive sulphide mineralization styles
Source: MSA

7.4.2 Adi Nefas

The Adi Nefas deposit occurs as an elongate north-northeast trending steeply east dipping massive sulphide layer that is hosted within an upright bimodal sequence of metavolcanic and derived metasedimentary rocks. The massive sulphide unit ranges in thickness from 5 m to 20 m and is largely hosted within a hydrothermally altered felsic quartz-sericite-chlorite-pyrite schist which in turn is flanked above and below by altered metabasaltic rocks. The altered felsic sequence ranges in thickness from 25 m to 60 m.

The Adi Nefas gossan comprises a silica, hematite and goethite-rich assemblage that represent the surface expression of the massive sulphide unit and the immediate sulphide-rich host rocks (Figure 7.14). The gossan averages some 10 m in width, and is mapped along strike for almost 2 km. The Adi Nefas deposit exhibits a vertical zonation due to weathering, as shown in Figure 7.15.

An upper oxide zone and underlying transition zone are leached and particularly depleted in copper and zinc relative to the primary sulphide mineralization. A slight enrichment in gold is reported for these zones. The base metal tenor increases slightly with depth in these zones. These zones typically extend to groundwater level at a depth of 20 m to 30 m. At and below the groundwater level, the supergene zone contains significantly enriched copper and gold and slightly enriched silver relative to the primary sulphide mineralization. Zinc is still depleted relative to the primary zone. The supergene zone is typically 20 m to 40 m thick. Examples of drillhole intersections from the transition and primary sulphide zones are shown in Figure 7.15. Mineralization within the primary sulphide zone attests to Adi Nefas being a zinc-copper-silver-gold VMS deposit. Although Adi Nefas is similar in style to the Debarwa deposit, it is distinguished from the latter by having zinc-rich primary sulphide mineralization (Figure 7.16).

Figure 7.15: Section Through the Adi Nefas Deposit Showing Vertical Mineral Zonation

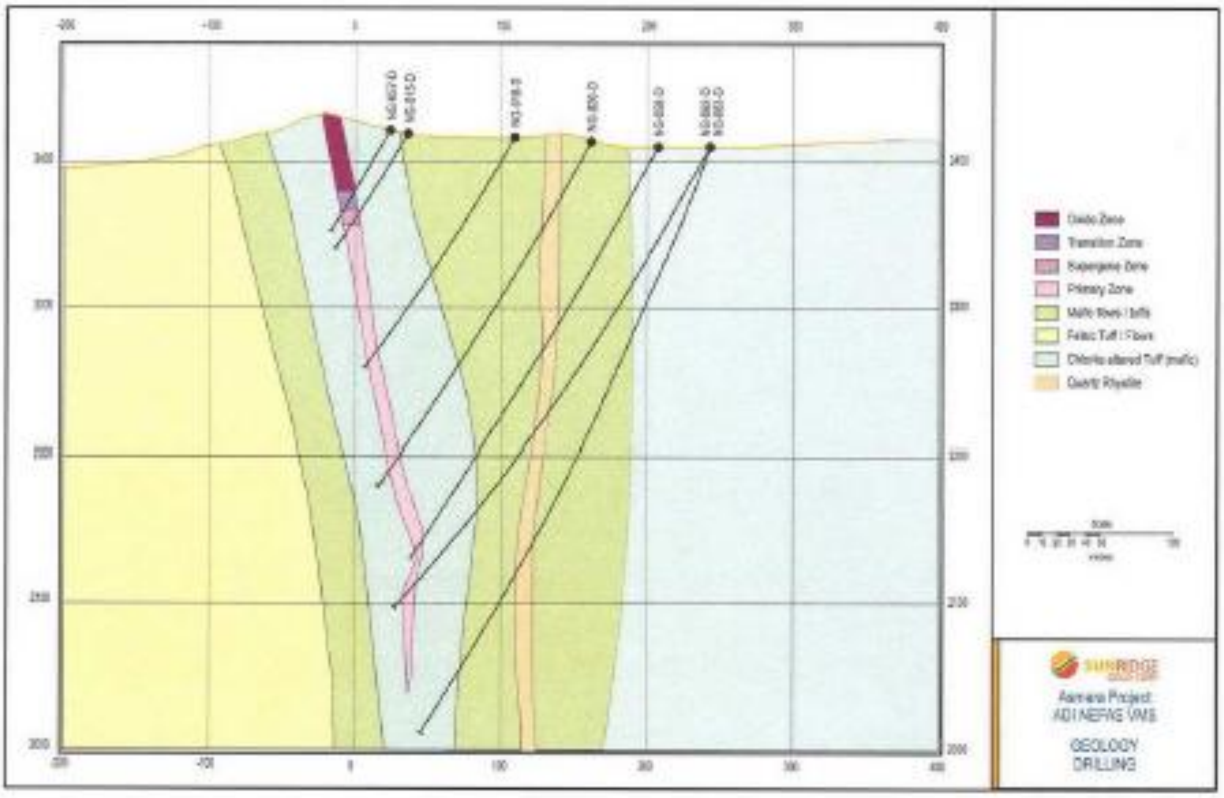


Figure 7.16: Mineralization styles – Adi Nefas VMS Deposit

Transition zone (Drillhole NG-057-D)

Zinc-rich primary massive sulphide zone (Drillhole NG-020-D)

Close-up of laminated (light-coloured) sphalerite mineralisation (Drillhole NG-070-D)

Source: MSA

7.4.3 Gupo

Locally, the highest grade gold mineralization occurs in crystallised, coarse-grained pyrite within quartz veins as well as lower grade gold mineralization in medium to coarse grained, euhedral to sub-euhedral pyrite within a sericite alteration halo. This alteration halo varies in width from few centimetres up to several metres depending of the thickness of the quartz veins, width of the shear zone and the porosity of the host rocks. The quartz veins form a complicated network of stockworks that pinch and swell, within the shear zone.

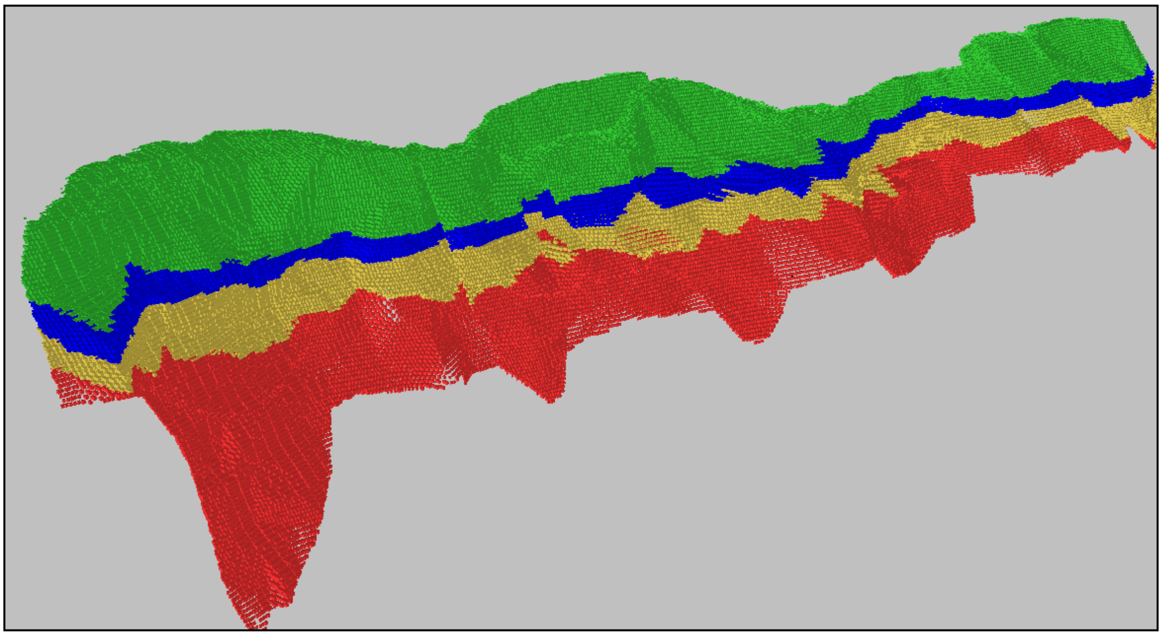
The Gupo gold deposit has been defined at surface and drilling over about a 1.6 km strike length and a 10 m to 20 m width. These zones are divided into two zones; Gupo North and Gupo South separated by a 400 m long barren zone which is interpreted as a late stage normal fault zone associated with the uplifting of these terrains of the Arabian-Nubian Shield during the opening of the Red Sea.

The Gupo North splits into eastern and western sub-zone half way to the south, probably representing the root system of the gold mineralization uplifted by the normal fault.

7.4.4 Debarwa

Three distinct vertical zones of mineralization are recognised at Debarwa (Figure 7.17). A surface oxide gold zone (green colour), from which base metals have been predominantly leached, extends to approximately 80 m depth from the highest points (between 35 m and 50 m below the floor of the Gual Mereb River valley) and is underlain by an enriched copper supergene zone (yellow colour) to around 110 m depth. The supergene zone is in turn underlain by a copper-rich primary sulphide zone (red colour). The oxide zone is typical of massive sulphide deposits exposed at surface in arid climatic regions. A thin precious metals-enriched transition zone (blue colour) separates the oxide from the supergene zone. In addition, remobilised copper mineralization forms a halo surrounding the supergene zone

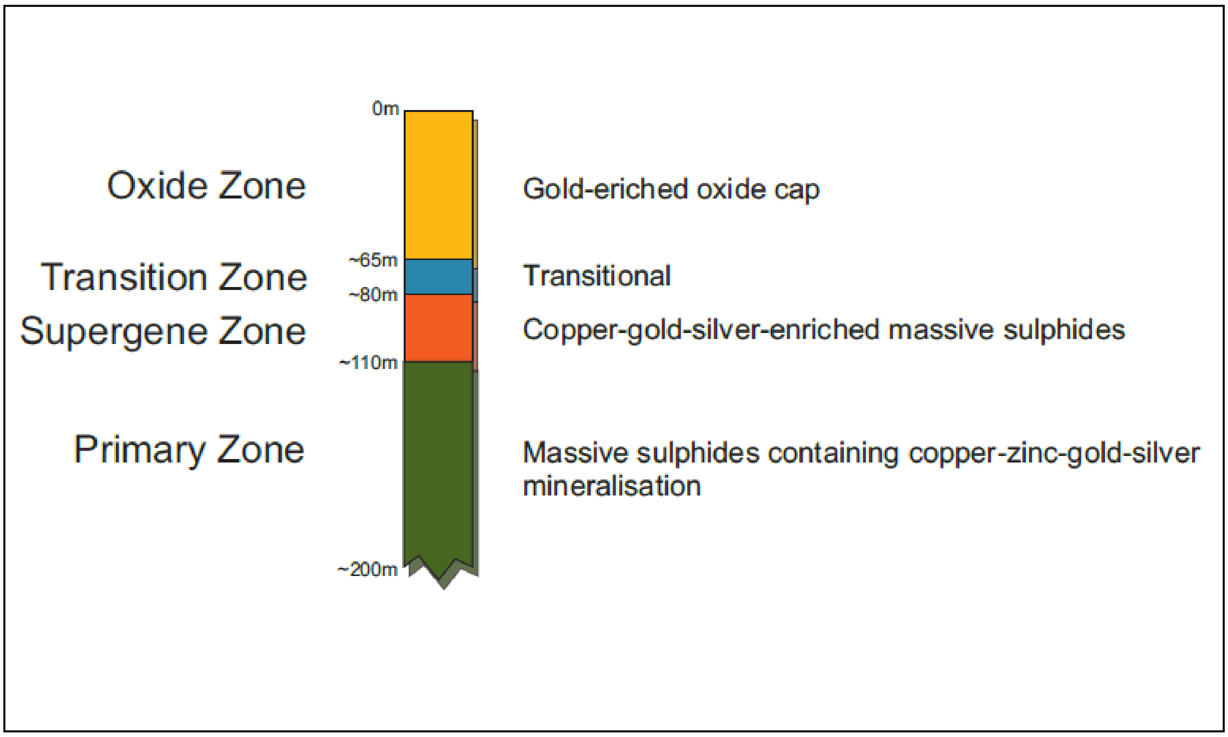
Figure 7.17: Isometric View of the Debarwa Enrichment Zones Facing East



Source: SGC

These zones are shown schematically in Figure 7.18 and are described below.

Figure 7.18: Simplified Representations of Mineral Zones at Debarwa



Source: SGC

Gossan (oxide zone): The oxide zone at Debarwa, which is exposed at surface as the Debarwa gossans, consists largely of iron oxides and hydroxides (hematite, limonite, goethite, jarosite (minor)), silica, and remnant clay. It may vary in colour from deep brick red to black and may include a variety of lithologies, from siliceous botryoidal limonite-hematite, to jasporoid, to greenish impure barite layers or beds.

At Debarwa Main, the oxide zone typically yields erratic but commonly high gold grades. Drill intersections vary between 0.40 and 14 g/t gold, with values commonly greater than 4 g/t gold over widths of between 7 m and 17 m. Silver values range between 0.4 and 183 g/t, and are typically greater than 15 g/t, while copper and zinc values are relatively insignificant.

Transition zone: The transition zone at Debarwa is about 10 m to 15 m thick and occurs at the transition between the oxide and supergene zones where the water table level fluctuates. This zone is the most enriched in precious metals, gold and silver and depleted with base metals especially copper and zinc, such as in drillhole DEBR-023-D, from 32 m to 40 m depth (8 m) 39.1 g/t gold, 519.5 g/t silver, 0.07% copper and 0.02% zinc.

Supergene zone: The supergene zone is characterised by extreme copper enrichment and is developed by oxidation of sulphide minerals in the overlying oxide and transition zones and the subsequent downward migration of copper within acidic fluids that also leach the wall rock they migrate through. The metals are then re-deposited at the groundwater table as enriched copper sulphides and oxides (principally digenite, chalcocite, tenorite, covellite, and possibly bornite). The secondary sulphides, which comprise the main mineralization in the supergene zone, replace and form coatings around primary sulphides, such as chalcopyrite, bornite, and pyrite and as crystallizing in the voids left by sphalerite, which is instable in these acidic conditions. The top of the supergene zone therefore represents the paleo-groundwater table.

At the Debarwa Main zone, the supergene zone yields higher grades, with one drillhole yielding a 20 m intercept of 12.8% copper, 4.6 g/t gold, and 69 g/t silver. This intercept occurred directly downhole from a 6 m intercept at the base of the oxide zone (transition zone) which assayed to 5.2 g/t gold, 185 g/t silver. More typical supergene zone intercepts range between 2 and 26 metres, with copper grades ranging from 0.9% to 32% (the high was over an interval of 6 m). Gold grades in holes from the supergene zone range between 0.5 and 4 g/t gold (typically between 1.5 and 3 g/t gold), and silver grades range between 16 and 144 g/t (typically in 30-80 g/t silver range). Zinc is generally very low and has been almost completely stripped out.

Primary zone: Primary sulphide or hypogene mineralization at Debarwa is preserved below the oxide, transition, and supergene zones, at depths ranging between 65 m and 90 m beneath the valley floor of the Mereb and Gual Mereb rivers. The major sulphide mineral phases consist predominantly of pyrite and chalcopyrite, with common bornite and sphalerite, in massive, semi-massive, and stringer vein zones that range in thickness up to 15 m. Typically, drill intersections of primary zone mineralization at Debarwa Main yield copper grades of between 2.0 and 9% copper (typically 2-4%), 0.5 and 7 g/t gold (typically <2 g/t), 6 and 150 g/t silver, and 1 to 12% zinc (typically 2-3%).

8 DEPOSIT TYPES

Information in this section has been excerpted from Hall *et al* ., (2008), Gribble *et al* ., (June 2009) and Hopley *et al* ., (2011) and updated.

8.1 VMS Deposit Model

VMS deposits occur in submarine volcanic environments as lenses of polymetallic massive sulphide that form at or near the seafloor through the focused discharge of hot, metal-rich hydrothermal fluids. VMS deposits are major sources of zinc (Zn), copper (Cu), lead (Pb), silver (Ag) and gold (Au) having produced approximately 22% of the world's zinc, 6% of the world's copper, 9.7% of the world's lead, 8.7% of its silver and 2.2% of its gold as at 1995 (Singer, 1995). VMS deposits typically comprise two components. A mound-shaped to tabular stratabound body composed principally of massive (>40%) sulphide, quartz, subordinate phyllosilicates and iron oxide minerals as well as altered silicate wall rock typically overlies a stockwork feeder zone enveloped in distinctive alteration halos, which may extend into the hangingwall strata above the deposit.

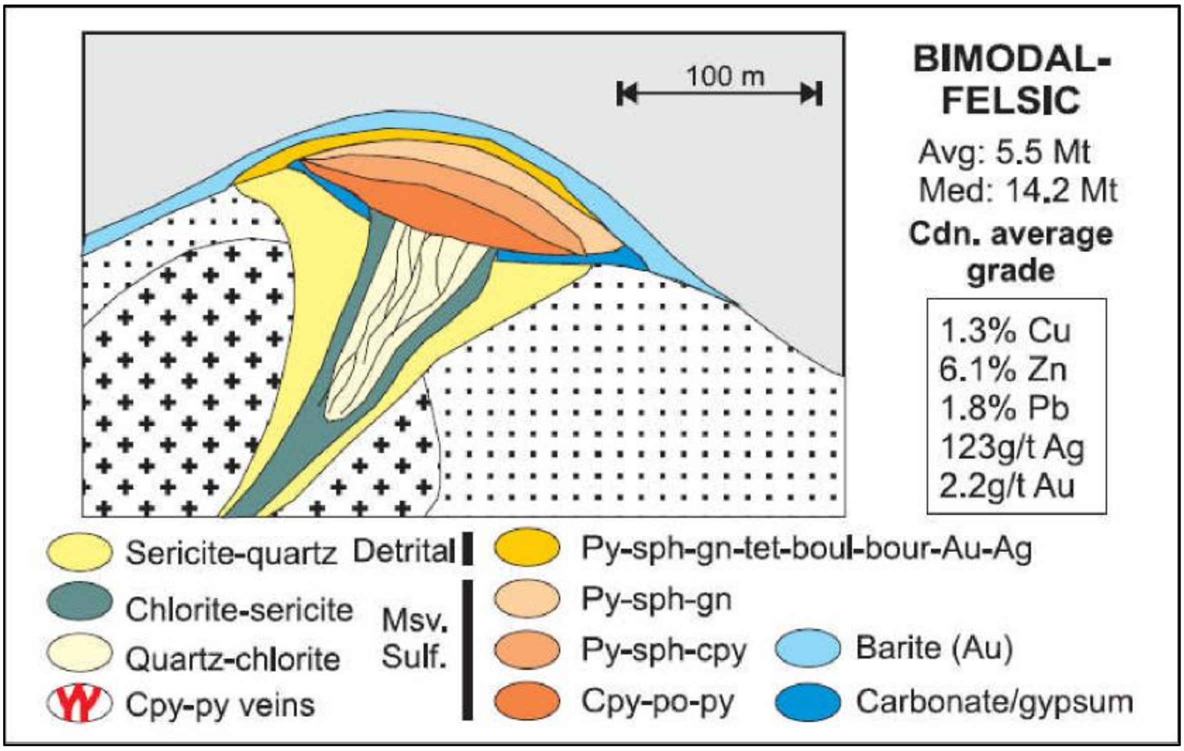
VMS deposits typically occur in clusters with one or more "giant" deposits in association with numerous smaller deposits. They are generally classified according to dominant host-rock lithology, as well as base-metal and gold content. Five groups are recognised namely mafic-dominated, bimodal-mafic (Archean copper-zinc type), bimodal-felsic (Kuroko type), siliclastic-mafic (Besshi type) and bimodal-siliclastic (Bathurst type). This lithological association correlates with tectonic setting where mafic volcanic and volcaniclastic strata tend to occur in oceanic arcs and spreading centres, whereas those dominated by felsic strata and more common in arc-continental margin and continental arc settings. They further tend to occur at specific stratigraphic horizons, commonly boundaries between contrasting lithologies within volcanic successions.

8.2 Kuroko/Noranda VMS Deposit Model

VMS deposits in the Asmara Project area are regarded by Barrie (2004) as being of the Kuroko type. A diagram illustrating the basic features of the Kuroko type model is shown in Figure 8.1. Kuroko/Noranda-style VMS deposits are known for their high grade polymetallic character, associated precious metal content, moderate to large quantities, and occurrence of multiple massive sulphide lenses. These deposits tend to form mineralised districts, and have the following key characteristics (after Galley *et al* , 2004):

- Marine volcanic geological setting, commonly during a period of more felsic volcanism within an andesite or basalt dominated succession
- Island arc tectonic setting, typically in a local extensional setting or rift environment within, or perhaps behind, a calc-alkaline bimodal arc succession
- Concordant polymetallic (copper, zinc, lead, plus gold and silver) massive to banded sulphide lenses which are typically metres to tens of metres thick and tens to hundreds of metres in horizontal dimension
- Quartz, chlorite, sericite alteration near the deposit centre to clay, albite, carbonate minerals further out
- Low-grade underlying crosscutting "stringer" zone of intense alteration and stockwork veining
- Massive to well- layered sulphides typically zoned vertically and laterally
- Copper-rich base, zinc-lead-rich top
- Barite or chert (exhalite) layers may overly the mineralised zone

Figure 8.1: Schematic Model for Kuroko-type VMS Deposits



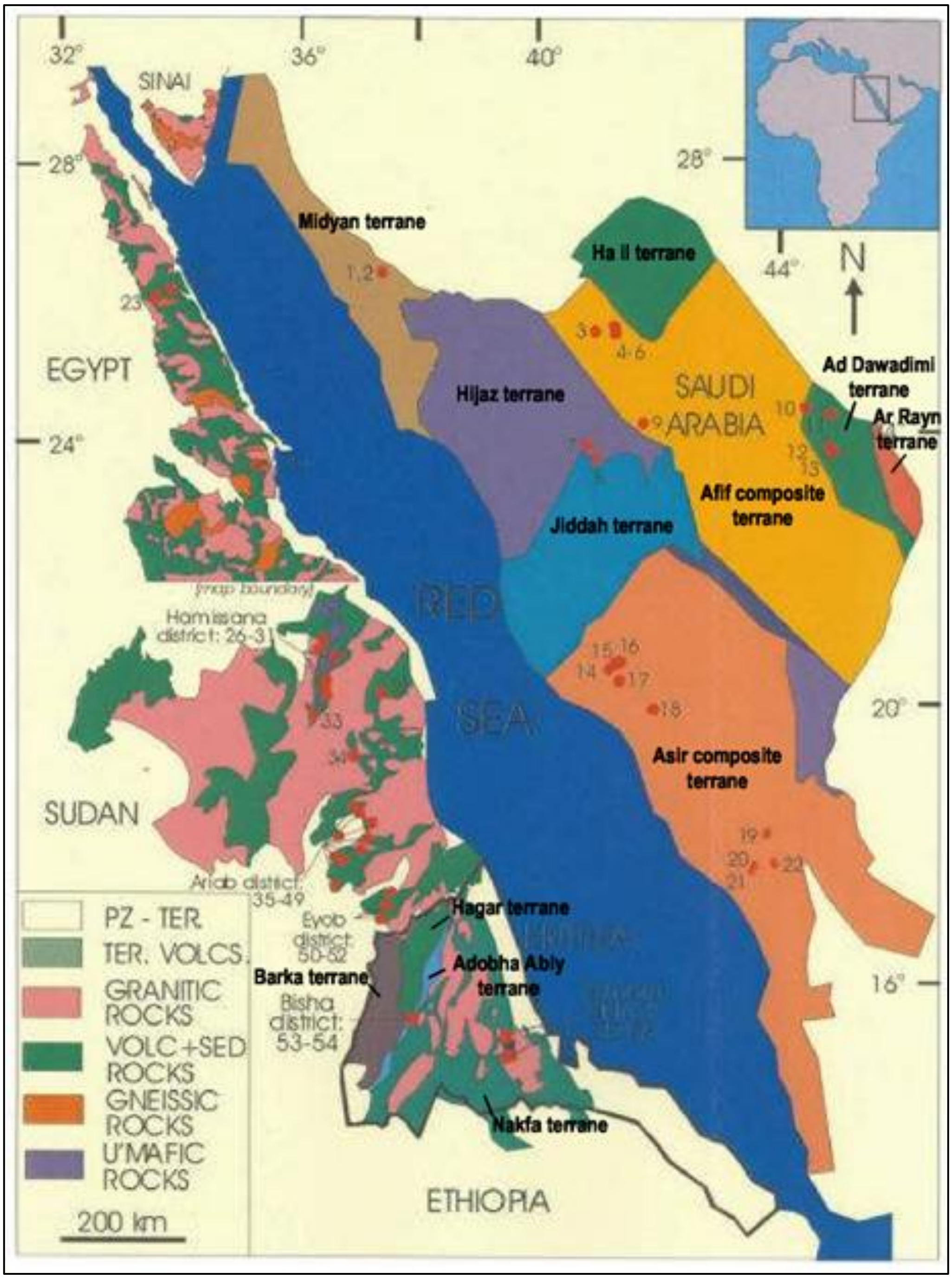
(Source: Galley et al 2004)

8.3 VMS deposits in the Arabian-Nubian Shield

The ANS contains at least 52 known VMS deposits and occurrences distributed throughout Eritrea, Sudan, Egypt and Saudi Arabia (Barrie, 2004). According to Barrie, most of these deposits and occurrences are concentrated in ten districts, with five districts on either side of the Red Sea (Figure 8.2). The majority of these deposits have a significant felsic volcanic component in their host rocks, which is consistent with the felsic more evolved character of the ANS generally. Twenty seven of these deposits have been classified as Kuroko subtype (bimodal-felsic association), nine as Bathurst subtype (bimodal-siliclastic), five as Archean copper-zinc subtype (bimodal-mafic), one as Besshi subtype (mafic-siliclastic) with the remainder unclassified due to insufficient information (Barrie, 2004).

All the VMS deposits and occurrences within the Asmara Project area have been classified by Barrie (2004) as Kuroko-type. The Emba Derho VMS deposit may be regarded as a large deposit spatially associated with a field of smaller deposits.

Figure 8.2: Occurrences of VMS Deposits in the Arabian Nubian Shield



Source: Barrie, 2004

8.4 Gold Deposit Model

The Gupo gold deposit shares similarities with other shear-hosted quartz-vein related gold deposits that are formed in Precambrian terrains and subsequently modified by tectonic events and near-surface weathering effects. The association of Gupo with a broad zone of hydrothermal alteration that is along-strike from the Adi Nefas VMS deposit suggests that the precursor mineralization to the Gupo gold deposit was related to volcanogenic activity.

9 EXPLORATION

For information and disclosure on this topic the reader is referred to previously filed documents for each deposit, as follows:

9.1 Emba Derho

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Emba Derho Deposit, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 110 pages. Effective date 6 February 2012.

9.2 Adi Nefas

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 82 pages. Effective date 20 February 2012.

9.3 Gupo

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Gupo Gold Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 97 pages. Effective date 3 April 2012.

9.4 Debarwa

Hopley, M.J., Arnold, C.G., Martin, C.J. (2011). Debarwa Copper Gold Deposit Eritrea Technical Report on Additional Drilling and Revised Mineral Resource Estimates NI 43-101 Technical Report prepared by and for Sunridge Gold Corp., with contributions by AMC Consultants (UK) Limited and Blue Coast Metallurgy Ltd. 188 pages. Effective date 18 August 2011.

10 DRILLING

For information and disclosure on this topic the reader is referred to previously filed documents for each deposit, as follows:

10.1 Emba Derho

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Emba Derho Deposit, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 110 pages. Effective date 6 February 2012.

10.2 Adi Nefas

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 82 pages. Effective date 20 February 2012.

10.3 Gupo

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Gupo Gold Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 97 pages. Effective date 3 April 2012.

10.4 Debarwa

Hopley, M.J., Arnold, C.G., Martin, C.J. (2011). Debarwa Copper Gold Deposit Eritrea Technical Report on Additional Drilling and Revised Mineral Resource Estimates. NI 43-101 Technical Report prepared by and for Sunridge Gold Corp., with contributions by AMC Consultants (UK) Limited and Blue Coast Metallurgy Ltd. 188 pages. Effective date 18 August 2011.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

For information and disclosure on these topics the reader is referred to previously filed documents for each deposit, as follows:

11.1 Emba Derho

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Emba Derho Deposit, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 110 pages. Effective date 6 February 2012.

11.2 Adi Nefas

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 82 pages. Effective date 20 February 2012.

11.3 Gupo

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Gupo Gold Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 97 pages. Effective date 3 April 2012.

11.4 Debarwa

Hopley, M.J., Arnold, C.G., Martin, C.J. (2011). Debarwa Copper Gold Deposit Eritrea Technical Report on Additional Drilling and Revised Mineral Resource Estimates.NI 43-101 Technical Report prepared by and for Sunridge Gold Corp. with contributions by AMC Consultants (UK) Limited and Blue Coast Metallurgy Ltd. 188 pages. Effective date 18 August 2011.

12 DATA VERIFICATION

For information and disclosure on these topics the reader is referred to previously filed documents for each deposit, as follows:

12.1 Emba Derho

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Emba Derho Deposit, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 110 pages. Effective date 6 February 2012.

12.2 Adi Nefas

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 82 pages. Effective date 20 February 2012.

12.3 Gupo

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Gupo Gold Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 97 pages. Effective date 3 April 2012.

12.4 Debarwa

Hopley, M.J., Arnold, C.G., Martin, C.J. (2011). Debarwa Copper Gold Deposit Eritrea Technical Report on Additional Drilling and Revised Mineral Resource Estimates. NI 43-101 Technical Report prepared by and for Sunridge Gold Corp., with contributions by AMC Consultants (UK) Limited and Blue Coast Metallurgy Ltd. 188 pages. Effective date 18 August 2011.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction and Background

Metallurgical testwork has been ongoing at eight different laboratories under the guidance of Sunridge Gold Inc and Blue Coast Metallurgy Ltd, for the Asmara Project ores since 2009. The flowsheet development efforts have focused on the use of conventional mineral processing strategies to maximise the recovery of the various pay metals while producing saleable products and minimising capital and operating costs.

An initial scoping study was conducted for Sunridge on Debarwa supergene material (the dominant ore type) at SGS Vancouver in 2009. A supergene composite assaying 6.65% copper and 39% sulphur was used for this study, the composite being prepared using aging drill core from six different holes. The textural character of the sample was ultrafine, with copper sulphide liberation remaining poor, even at grind k_{80} sizes of less than 30 microns. The best results achieved from this study were concentrates assaying slightly over 20% copper at recoveries of roughly 70% recovery.

A feasibility study on Debarwa followed, including testwork on oxide, transition, direct shipping ore (DSO), supergene sulphide and primary sulphide samples. Key findings were that the oxide and transition ores would respond adequately well to heap leaching, with agitation leaching yielding better recoveries, but not sufficiently better to warrant inclusion of this more expensive processing option.

The supergene zone comprises most of the value in the Debarwa deposit, the highest grade material being DSO grade, the remainder assaying about 4.5% Cu. This is very fine-grained and requires fine regrinding for reasonable metallurgy to be achieved. Stable locked cycle performance was achieved on the low and mid-grade master composites, with copper concentrates assaying 19 and 21 percent copper floated at 78 and 84 percent respectively.

The relatively small primary zone was not extensively tested. High doses of zinc sulphate and sodium cyanide succeeded in the depressing the sphalerite, allowing for sequential copper and zinc flotation. The results are shown in Table 13.1.

Table 13.1: Summarized Locked Cycle Test Metallurgy from Debarwa Primary Composites

			Copper flotation				Zinc flotation			
	Feed %		Grade %		Recovery %		Grade %		Recovery, %	
	Cu	Zn	Cu	Zn	Cu	Zn	Cu	Zn	Cu	Zn
Average Grade	2.0	4.2	30	6	67	7	4	53	10	83
High Zinc	1.7	6.2	31	7	57	4	2	61	11	80
Low Zinc	1.9	1.5	28	5	77	20	12	34	15	66

Testing for the Asmara North pre-feasibility study was commissioned in 2011, and focused on Emba Derho samples. Rougher flotation was at pH 11 to depress the pyrite and mostly followed a fine grind k_{80} of 60 microns. The optimum rougher float yielded 12% Cu at 87% recovery when using 24 g/t dithiophosphate collector. No cleaner tests were run in this study.

Testing on Emba Derho primary ores focused on flowsheet development and closed with a series of locked cycle tests on average grade, and copper rich composites. Some key findings made in the pre-feasibility study were.

- A primary grind in the order of 70-80 microns was established as optimal
- Zinc sulphate/sodium cyanide worked very well for Emba Derho, indeed only modest doses were needed in the absence of the more soluble copper-rich Adi Nefas and Debarwa feed materials.
- The copper concentrate regrind sizes were established based on regrind tests on the CuZn and Zn rich composites

The locked cycle metallurgy achieved in the PFS is summarized in Table 13.2.

Table 13.2: EmbaDerho Primary PFS Locked Cycle Test Metallurgy

	Feed (%)		Copper flotation				Zinc flotation			
			Grade (%)		Recovery (%)		Grade (%)		Recovery (%)	
	Cu	Zn	Cu	Zn	Cu	Zn	Cu	Zn	Cu	Zn
Average Grade	0.9	1.4	25	4	89	4	2	60	5	87
High Copper	1.4	0.7	28	2	95	11	1	61	1	71

13.2 Feasibility Sample Selection

Eight oxide composites were selected by Sunridge Gold Corp and Blue Coast Metallurgy for heap leach testwork at Kappes Cassidy & Associates (KCA), using half drill core material, shown in Table 13.3.

Table 13.3: Oxide Composite Sample Head Grades

Sample	Au (g/t)	Ag (g/t)	Cu (ppm)
Debarwa Transition Comp 1	1.03	21.96	427
Debarwa Transition Comp 2	2.45	10.95	4103
Emba Derho Oxide Comp 1	0.55	1.99	793
Emba Derho Oxide Comp 2	0.99	3.39	459
Gupo Comp 1	1.7	0.62	179
Gupo Comp 2	2.02	0.62	164
Debarwa Oxide Comp 1	1.36	7.1	1797
Debarwa Oxide Comp 2	0.91	21.7	679

Supergene material from both Emba Derho and Debarwa was used during the feasibility study, including three Emba Derho-Debarwa supergene composites, four Debarwa supergene composites, one Emba Derho supergene composite and one Gupo sample. The origin and head assays of the composites are presented inTable 13.4.

Table 13.4: Head Assays of Asmara North Supergene and Gupo Samples

Composite ID	Cu (%)	Zn (%)	Au (gt)	Ag (gt)	S (%)	Origin and Make Up
ED-DEB Composite 1 (ED-DEB Composite)	3.29	0.18	0.85	22.9	28.2	Made from 50/50 Emba Derho Supergen Master cpmposite and Debarwa Supergen MAGC frpm XPS
ED-DEB Composite 2 (new ED-DEB Composite)	2.37	0.19	0.56	10.9	29.4	Made from 51.4% of debarwa supergene from XPS and 48.6% of Emba Dergo dill cores from minesite
ED-DEB Composite 3 (ED DEB high Zn composite	1.34	0.43	0.34	11.8	34.8	Made from Emba Derho supergene composites 2,3 and 4 from XPS and ED-DEB composite 2
Debarwa Supergene MAGC	5.17	0.09	1.26	26.0	25.0	From XPS
Debarwa Supergene MLGC	2.59	0.05	1.02	12.1	18.5	From XPS
Debarwa Supergene MHGC	8.77	0.13	2.93	49.4	33.4	From XPS
Debarwa Supergene Master Composite	3.98	0.07	1.28	19.2	22.2	Made from 2kg of MHGC,6kg of MAGC and 12kg of MLGC of XPS samples
Emba Derho Supergene Low Copper Master Composite	0.83	0.15	0.13	10	30.8	Made from Emba Derho cores from mines
Gupo Master Composite			1.01		0.48	Made XPS samples

Seven master composites of primary ores were prepared. The make-up of the composites is summarized in Table 13.5, each composite either designed to represent material processed through a specific period in the mine life, or to represent life of mine or specific end-member compositions, either copper or zinc rich.

A further two composites were prepared for use to prepare parameters for solid-liquid separation circuit design and for regrind circuit sizing. These used a combination of material from specific drill holes and spare composite material from the PFS phase of the study. Results from tests on these composites were not used for recovery predictions for the Project.

Table 13.5: Composition, Source and Head Assays of the Primary Master Composites

	YR0-1	YR1-2	YR2-4	YR5-11	Cu rich	Zn rich	LOM	Settling comp	Regrind comp
Debarwa	13%	0%	0%	0%	0%	0%	0%	0%	0%
Adi Nefas	8%	11%	7%	0%	0%	0%	0%	10%	14%
Emba Derho	79%	89%	93%	100%	100%	100%	100%	90%	86%
	Composite source holes/PFS comps in Emba Derho							**Settling comp**	**Regrind comp**
	YR0-1	**YR1-2**	**YR2-4**	**YR5-11**	**Cu rich**	**Zn rich**	**LOM**		
	226-D	221-D	216-D	183-D	202-D	183-D	224-D		224-D
	231-D	224-D	235-D	224-D	268-D	271-D	268-D		226-D
	278-D	279-D	268-D	225-D	271-D	273-D	271-D		268-D
	279-D		273-D	268-D	273-D	279-D	273-D		271-D
				279-D			279-D		
								AN MC	AN MC
								EDPCuZn	EDPCuZn
								EDPCu	EDPCu
								EDPZn	EDPZn
	Assays							**Settling comp**	**Regrind comp**
	YR0-1	**YR1-2**	**YR2-4**	**YR5-11**	**Cu rich**	**Zn rich**	**LOM**		
Cu (%)	0.63	0.42	0.41	0.74	1.12	0.44	0.73	0.85	0.97
Zn (%)	3.31	3.76	3.28	1.39	0.94	3.21	1.82	2.57	2.89
Ag (g/t)	19.6	19.8	20.6	10.6	5.2	12.6	5.4	9.86	15.65
Au,(g/t)	0.77	0.60	0.44	0.27	0.15	0.49	0.15	10.6	4.6

In addition, 7 Cu Zn composites, 14 Cu composites and 15 Zn variability composites were selected from different areas of the Emba Derho resource in order to cover spatial representivity, as well as grade and lithology variability.

13.3 Hardness Testwork and Characterisation

Hardness testwork was undertaken on samples from all zones to characterise the various orebodies and provide design parameters for comminution (crushing and grinding) circuit design. This data is summarised in Table 13.6.

Table 13.6: Summarised Hardness Characterisation Testwork Results

Material type		JK Parameters		CWI	RWI	BWI	AI
	Relative density	(A x b)	(t/a)	(kWh/t)	(kWh/t)	(kWh/t)	(g)
Emba Derho Primary	4.26	171	1.04	10.2	n/a	11.2	0.217
Emba Derho Supergene	2.53	n/a	n/a	n/a	n/a	9.9	0.023
Adi Nefas		n/a	n/a	n/a	9.3	10.5	0.144
Emba Derho Oxide	2.16	206	2.55	8.7	7.2	9.6	0.023
Gupo	2.78	43	0.41	11.6	12.7	11.9	0.203

In addition, an HPGR amenability test was run by Koppern. Koppern concluded that this particular sample was amenable to HPGR comminution, however follow-up trade-off studies led to the conclusion that SAG-ball milling was the more favourable choice for comminution in the project.

13.4 Gold Ores Metallurgical Testwork

Bottle roll testing (96 hours) was conducted on each of the eight composites, following grinding to 80% passing 75 microns, and using a 1.0g/L sodium cyanide solution with the pH maintained between 10.5 - 11. In addition, two coarse bottle roll tests were performed on the Debarwa Transition Composite #2 at a topsize of 9.5 mm and 12.5 mm to evaluate the sample’s amenability to heap leaching. The results of these tests are summarized inTable 13.7 below.

Table 13.7: Cyanide Leach Test Results on Asmara Oxide Samples

Sample	Size	Leach	CN cons	Head grades, Au		Extract'n,	Tails
	(k_{80} mm)	(Time hours)	(kg/ton)	(Assay g/t)	(Recon g/t)	(%)	(g/t Au)
Deb Transition Comp 1	0.075	96	2.1	1.03	1.16	89%	0.13
Deb Transition Comp 2	7.5	144	3.1	2.45	2.59	32%	1.68
Deb Transition Comp 2	5.8	144	3.3	2.45	2.59	32%	1.76
Deb Transition Comp 2	0.075	96	7.4	2.45	2.64	45%	1.46
ED Oxide Comp 1	0.075	96	0.3	0.52	0.39	74%	0.14
ED Oxide Comp 2	0.075	96	0.2	0.95	0.78	82%	0.17
Gupo Comp 1	0.075	96	0.8	1.65	1.53	92%	0.13
Gupo Comp 2	0.075	96	0.9	1.98	1.83	93%	0.15
Debarwa Oxide Comp 1	0.075	96	1.1	1.13	0.93	82%	0.20
Debarwa Oxide Comp 2	0.075	96	1.4	0.95	0.68	71%	0.28

In general, the gold extraction ranged from 71-93% for the 75 micron bottle roll tests with the exception of Transition Composite #2 which had yielded relatively poor gold extraction at 45% (this also contained the most copper). The coarse bottle roll tests on Transition Composite #2 also exhibited poor gold extractions at 32% for both tests.

A total of seven separate column leach tests were completed using material from the Asmara Project. Gold extractions for the various column tests ranged from 42% to 76%. The Gupo and Emba Derho Oxide composites showed the best and most consistent gold extractions ranging from 62% to 73%. The Debarwa Transition Composite #1 exhibited a relatively low gold extraction of 51%. Silver extractions ranged from 13% to 70%. Cyanide consumptions were relatively consistent and ranged from 0.82 kg/tonne to 1.35 kg/tonne.

13.5 Supergene Ores Metallurgical Testwork

Debarwa supergene ores are typical fine-grained supergene VMS copper ores. Their modal composition is relatively straightforward, a mix of (mostly) secondary copper sulphides and pyrite, with quartz being the dominant third component. In the case of some samples (as in the low grade composite described below) the abundance of clay and micaceous material is somewhat higher, but still low on an industry-wide basis. The ultrafine textures that dominate the copper sulphide mineralisation represent the most challenging mineralogical component of these ores. The average grain size of the Debarwa copper sulphides is less than 30 microns, and becomes finer for the lower grade materials (Figure 13.1).

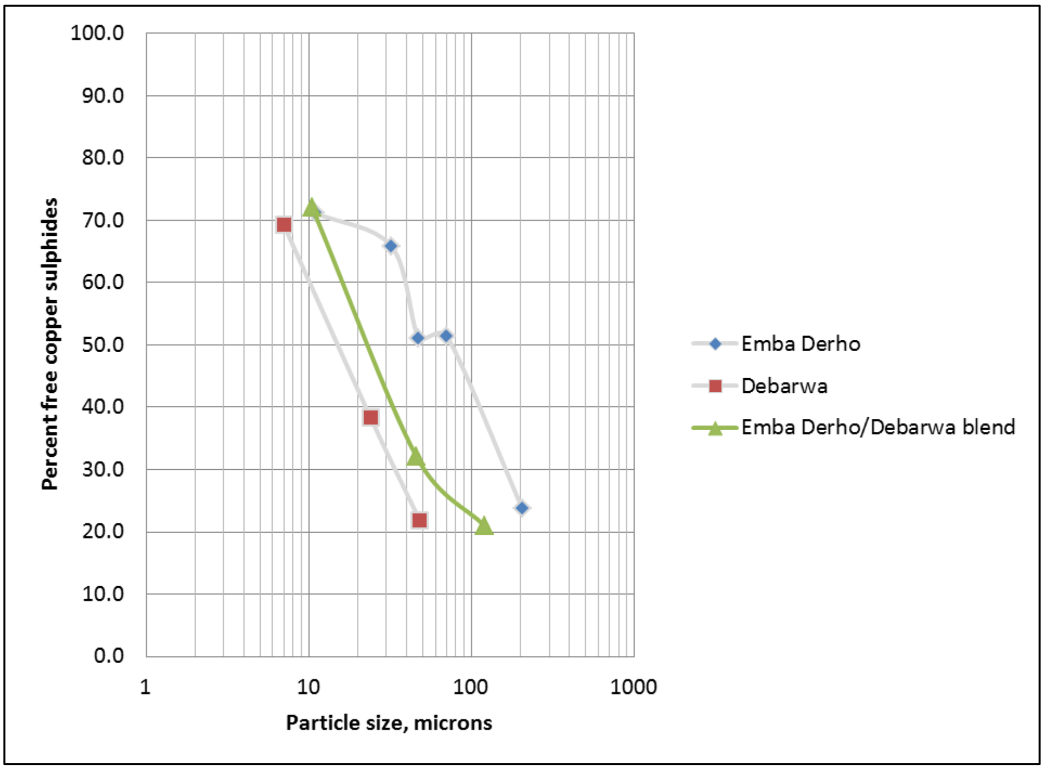
Two different treatment philosophies were explored. The first, developed for the Debarwa feasibility study, included flotation of all the sulphides at natural pH. The sulphides were then fine-ground to a k_{80} size of 15 microns and cleaned twice at high pH to reject the pyrite. The second flowsheet incorporated a much finer primary grind, and a strategy of floating the copper sulphides somewhat more selectively in the roughers at higher pH. This flowsheet, though more power-intensive, had the key advantage that the pyrite was depressed throughout, so it was more amenable to overall process stability. This flowsheet was adopted for the Asmara feasibility study, and yielded 84% copper recovery to a 27% copper concentrate grade. Some 58% and 70% of the gold and silver respectively were floated to the copper concentrate, which assayed 5 g/t gold and 116 g/t silver.

Emba Derho's supergene ore is perhaps somewhat of a misnomer as mineralogically it more resembles a supergene/hypogene transition material – reflecting that there is not a well-formed supergene zone in the Emba Derho deposit. The copper mineralization is a blend of primary and secondary copper sulphides, with the proportion of copper in primary and secondary form being close to 50:50. There is also a small amount of sphalerite in the zone, not enough to create concentrate quality problems, though it can be expected to report to the copper concentrate and care will be needed in practice to keep the mill feed Cu:Zn ratio high enough to ensure good quality copper concentrates.

The remaining ore is mostly a mix of pyrite and quartz, iron oxides and aluminium silicates. The non-sulphide components would be mostly benign in sulphide flotation. Copper sulphide textures in the Emba Derho ores are described through the release analysis in Figure 13.1. Emba Derho copper sulphides are substantially coarser grained than Debarwa which bodes well for their processing.

Only two tests were conducted on the Emba Derho composite, the second a cleaner flotation test. This open circuit test yielded a copper concentrate assaying 26% copper at 70% copper recovery. Assuming 50% of the copper in the 2nd and 3rd cleaner tails would report to the concentrate, a locked cycle test on this material may have been expted to yield a recovery of roughly 75% to a concentrate assaying 23-25% copper. Relatively little gold and silver floated to the final concentrate, which assayed 1.6 g/t Au and 113 g/t Ag.

Figure 13.1: Copper Sulphide Grain Size Distribution, Emba Derho/Debarwa Blended Feed



In reality, the mine plan consistently calls for a blended feed of Emba Derho and Debarwa feed materials. On average, this blended material would be mostly supergene in nature, with over 80% of the copper being present as fine-grained chalcocite/digenite.

Approximately 17 batch flotation tests and 2 locked cycle flotation tests were run on the blended Emba Derho/Debarwa master composites. The developed flowsheet, shown below, employed a primary grind k_{80} of 66 microns and a regrind size of 15 microns, together with flotation in a conventional rougher, three cleaner and cleaner scavenger configuration. The flowsheet stabilized in locked cycle testing of the two blended composites tested, yielding copper, gold and silver recoveries of 85%, 58% and 64% respectively to a concentrate assaying 24% copper and containing payable gold and silver. Details are shown in Figure 13.2 and Table 13.8.

Figure 13.2: Supergene Processing Flowsheet for the Emba Derho/Debarwa Blended Feed

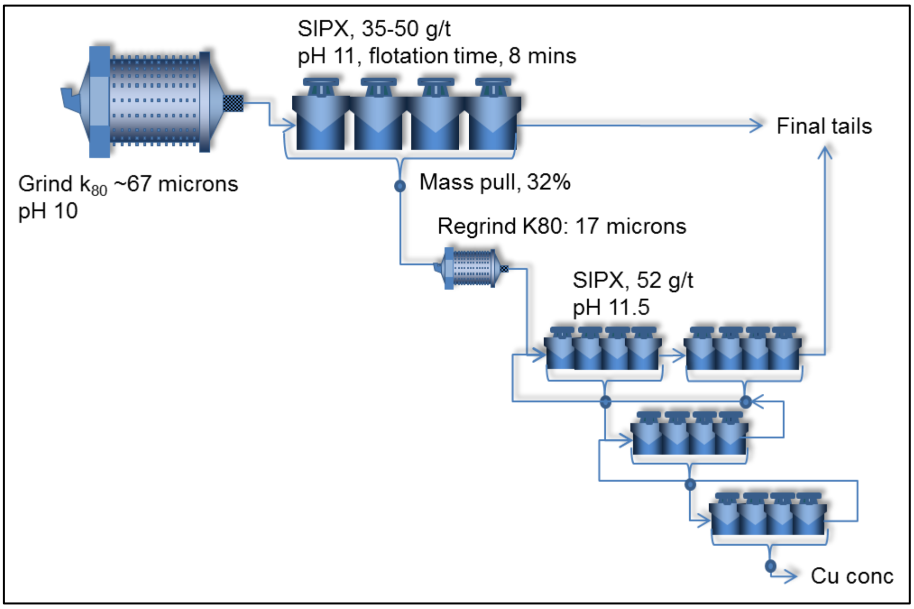


Table 13.8: Blended Emba Derho/Debarwa Processing Locked Cycle Metallurgy

	Assays, % g/t				Distribution %			
Composite 1	Cu	Zn	Au	Ag	Cu	Zn	Au	Ag
Cu Clnr 3 conc	24	1.5	3.6	82	85	79	58	64
Cu Clnr 1 tails	1.4	0.05	0.7	11	10	5	23	18
Cu Rougher Tails	0.26	0.05	0.2	4	6	16	19	18
Feed (Calc.)	3.1	0.21	0.7	14	100	100	100	100
Composite 2	Cu	Zn	Au	Ag	Cu	Zn	Au	Ag
Cu Clnr 3 conc	25	1.4	4.8	110	86	64	60	66
Cu Clnr 1 tails	0.85	0.03	0.5	12	9	4	22	22
Cu Rougher Tails	0.15	0.08	0.2	2	5	32	19	13
Feed (Calc.)	2.2	0.17	0.6	13	100	100	100	100

A third blended composite was tested to explore the effect of a lower copper head grade and a higher zinc head grade on flotation performance. Such material may occur from time to time when Zn-bearing marginal primary ore is inadvertently mixed with the Emba Derho transition material. This sample assayed 1.5% copper and 0.43% Zn. This yielded a 24% copper concentrate, floated at 74% copper recovery. This compares with 80% and 82% copper recovery achieved in batch testing with a very similar flowsheet for Composites 1 and 2, and suggests a small drop in recovery; roughly 6-8% can be expected when processing this material.

13.6 Primary Ores Metallurgical Testwork

13.6.1 Chronological Samples

Seven master composites of primary ores were prepared for flowsheet optimization and locked cycle testing. Their total modal mineral abundances are shown below:

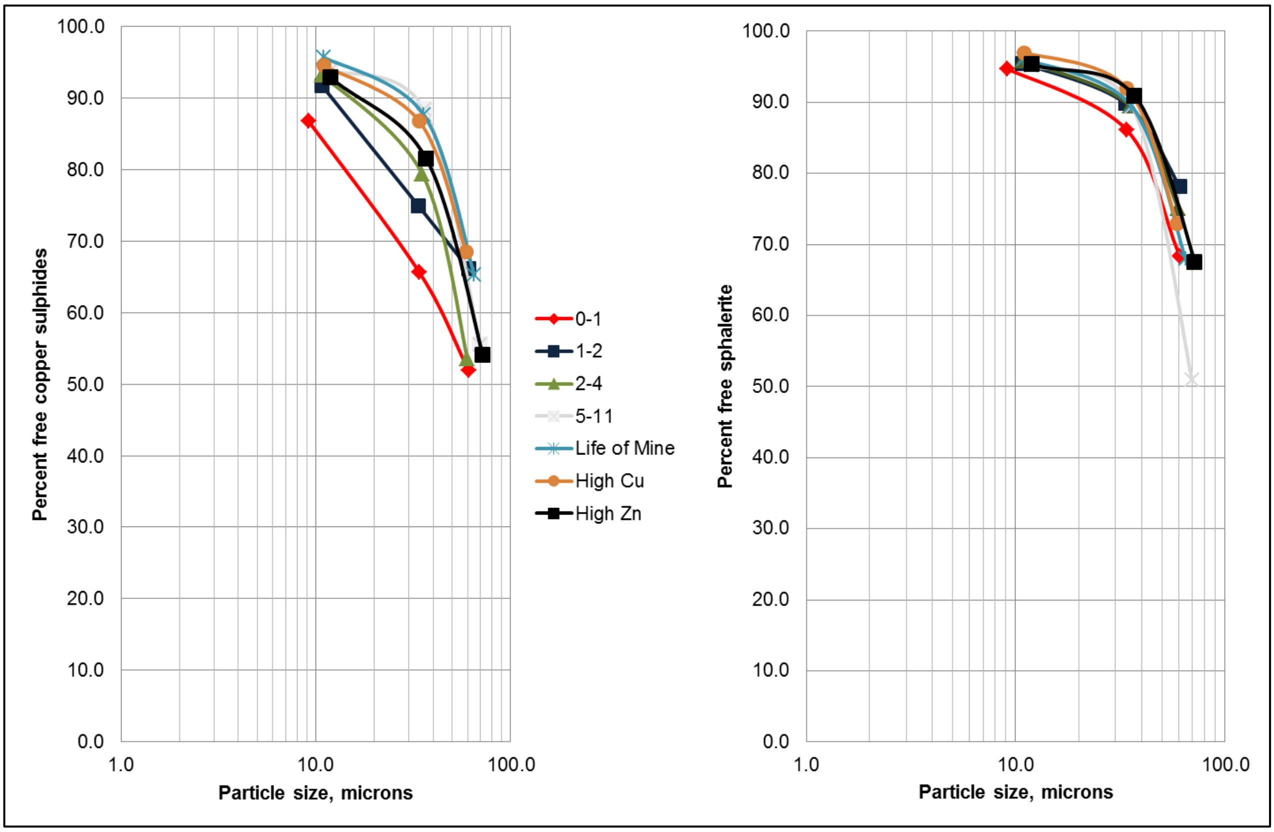
Table 13.9: Modal Mineralogical Analyses of the Primary Composites

	YEAR OF PRIMARY PRODUCTION						
	0-1	**1-2**	**2-4**	**5-11**	**LOM**	**High Cu**	**High Zn**
Chalcopyrite	1.4	1.3	1.2	2.4	2.4	3.7	1.4
Bornite	0.2	0.0	0.0	0.0	0.0	0.0	0.0
Other copper sulphides	0.1	0.1	0.1	0.0	0.0	0.0	0.0
Cu Textures	0.2	0.1	0.1	0.1	0.0	0.1	0.1
Iron sulphides	40.7	43.4	62.1	56.5	63.7	79.0	63.5
Sphalerite	5.4	6.0	5.6	2.3	3.2	1.8	5.4
Quartz	13.2	12.2	7.2	8.4	7.6	3.3	4.0
Barite	8.7	4.4	8.8	3.4	1.3	0.2	1.1
Carbonates	1.3	2.0	1.2	1.0	1.9	1.3	3.1
Kaolinite & Mica	3.5	2.8	1.6	1.3	0.6	0.2	0.4
Fe Oxides	2.4	5.5	2.7	4.3	6.0	6.0	6.0
Others (mainly silicates)	23.0	22.2	9.5	20.3	13.2	4.5	15.0

The most complex primary ores are processed in the earlier years, especially in Year 1 where the secondary copper sulphides are most abundant. The presence of substantial secondary copper will increase the need for zinc depressants in the primary grind to ensure production of saleable copper concentrate. The need for these depressants can be expected to decline with the reducing secondary copper content as the mine life advances.

The textural analyses of the copper and zinc sulphides in the primary composites are shown below. The copper sulphides in the primary ores are substantially coarser-grained than those in the supergene and, for the most part, tend to coarsen in the latter years of the operation. The sphalerite tends to be coarser than the copper sulphides throughout.

Figure 13.3: Copper Sulphide and Sphalerite Textural Analysis, Primary Ore Composites



Locked cycle tests were conducted on all composites, the conditions used shown in Figure 13.4 below.

All tests reached an acceptable level of stability. Copper recoveries rose through the life of the mine, starting at a low of 81% in the first year of primary ore processing to reaching a high of 93% in years 5-11. The life of mine sample yielded 91% copper recovery based on the locked cycle test data, slightly higher than the weighted average copper recovery (weighted for number of years of production represented by each composite) of 89%. Concentrate quality, in the early years, will be enhanced by a substantial by-product credit of gold and silver, this dropping away in the latter years and while in those same early years the zinc grade approaches a limit where smelter would impose penalties, the substantial precious metal credits would ensure these concentrates would remain attractive. On average, the copper concentrate assay 25% copper, being perhaps slightly lower in early years to enhance precious metal recovery, and higher in latter years as the precious metal component drops away. Zinc recoveries also averaged close to 90% both based on a weighted average of the composite recoveries, and the life of mine sample. The concentrate grade was slightly lower in year 0-1 (likely a consequence of over-dosing the copper sulphate in testing) but averaged (weighted for tons of mill feed) 59% zinc through the rest of the life of the operation. This agreed well with the life of mine result.

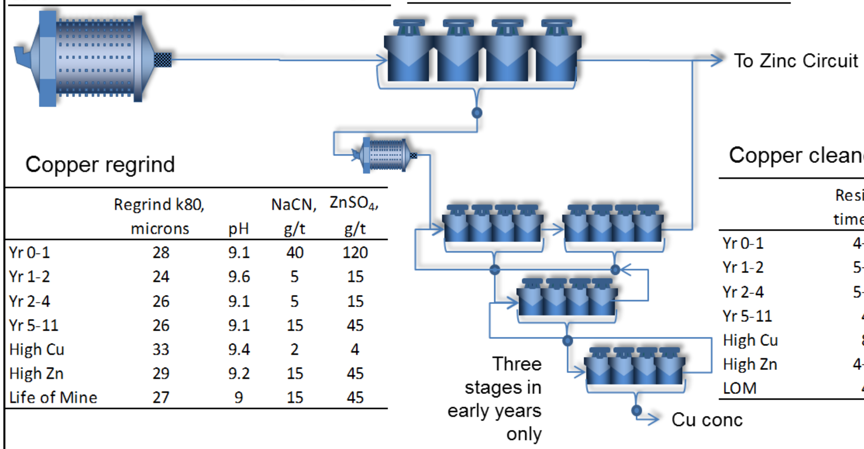
Figure 13.4: Copper and Zinc Flotation Locked Cycle Conditions, Primary Ore Composites

Primary grind

	grind k80, microns	pH	NaCN, g/t	$ZnSO_4$, g/t
Yr 0-1	71	8.3	400	1200
Yr 1-2	70	9.3	150	450
Yr 2-4	69	9.8	100	300
Yr 5-11	70	10.2	30	90
High Cu	70	10	5	15
High Zn	70	9.8	30	90
LOM	70	9.6	30	90

Copper rougher flotation

	Residence time, mins	pH	SIPX, g/t	MIBC, g/t
Yr 0-1	6	10	47	35
Yr 1-2	6	10	45	35
Yr 2-4	6	10	55	30
Yr 5-11	6	10	35	30
High Cu	10	10	95	37
High Zn	6	10	55	28
LOM	6	10	55	30



Copper regrind

	Regrind k80, microns	pH	NaCN, g/t	$ZnSO_4$, g/t
Yr 0-1	28	9.1	40	120
Yr 1-2	24	9.6	5	15
Yr 2-4	26	9.1	5	15
Yr 5-11	26	9.1	15	45
High Cu	33	9.4	2	4
High Zn	29	9.2	15	45
Life of Mine	27	9	15	45

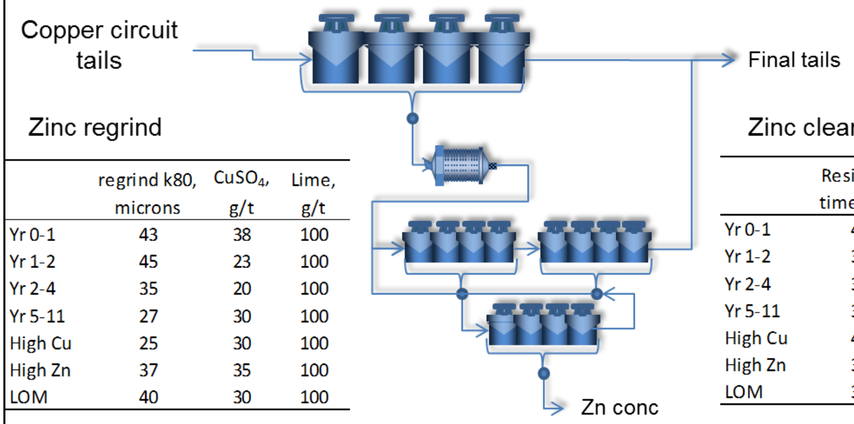
Copper cleaner flotation

	Residence time, mins	pH	SIPX, g/t	MIBC, g/t
Yr 0-1	4+4+3	10	22	10
Yr 1-2	5+4+2	10	15	7
Yr 2-4	5+4+3	10.5	22	10
Yr 5-11	4+3	10	13	9
High Cu	8+5	10.5	20	8
High Zn	4+4+3	10	18	7
LOM	4+3	10	16	4

(a) Copper Flotation

Zinc rougher flotation

	Residence time, mins	$CuSO_4$, g/t	SIPX, g/t	MIBC, g/t	pH
Yr 0-1	4	380	75	30	11.6
Yr 1-2	8	225	75	30	11.6
Yr 2-4	7	200	75	30	11.6
Yr 5-11	8	175	60	40	11.6
High Cu	6	300	85	30	11.6
High Zn	10	350	70	30	11.6
LOM	8	200	60	30	11.6



Zinc regrind

	regrind k80, microns	$CuSO_4$, g/t	Lime, g/t
Yr 0-1	43	38	100
Yr 1-2	45	23	100
Yr 2-4	35	20	100
Yr 5-11	27	30	100
High Cu	25	30	100
High Zn	37	35	100
LOM	40	30	100

Zinc cleaners

	Residence time, mins	pH	SIPX, g/t	MIBC, g/t
Yr 0-1	4+4	11.8	15	15
Yr 1-2	3+4	11.8	15	15
Yr 2-4	3+3	11.8	12	15
Yr 5-11	3+3	11.7	10	10
High Cu	4+4	11.7	15	15
High Zn	3+2	11.8	12	15
LOM	3+2	11.8	10	12

(b) Zinc Flotation

Table 13.10: Locked Cycle Metallurgy on the Primary Sulphide Ore

	COPPER FLOTATION						
	LOM	**Yr 0-1**	**Yr 1-2**	**Yr 2-4**	**Yr 5-11**	**Cu rich**	**Zn rich**
Mass (%)	2.7	2.1	1.4	1.4	2.7	4.1	1.6
CONCENTRATE GRADES							
Cu (%)	25	25	24	24	27	26	25
Zn (%)	1.3	6.7	6.4	4.5	2.4	0.7	3.7
Au (g/t)	1.4	16.0	18.4	n/a	4.1	0.7	0.7
Ag (g/t)	74	458	671	n/a	134	27	25
RECOVERIES							
Cu (%)	91	81	85	84	93	88	89
Zn (%)	2	4	2	2	5	3	2
Au (%)	35	59	62	n/a	51	20	40
Ag (%)	36	55	57	n/a	43	18	50
	ZINC FLOTATION						
	LOM	**Yr 0-1**	**Yr 1-2**	**Yr 2-4**	**Yr 5-11**	**Cu rich**	**Zn rich**
Mass (%)	2.9	5.4	6.0	5.4	2.1	1.6	5.1
CONCENTRATE GRADES							
Cu (%)	0.7	0.9	0.5	0.5	0.5	2.6	0.2
Zn (%)	59	54	56	57	60	58	57
Au (g/t)	0.3	1.2	0.6	n/a	0.5	0.5	0.7
Ag (g/t)	25	58	49	n/a	43	31	25
RECOVERIES							
Cu (%)	3	8	7	7	1	4	7
Zn (%)	91	88	92	93	89	89	93
Au (%)	7	12	8	n/a	5	6	8
Ag (%)	13	19	18	n/a	11	8	13

Concentrates from the locked cycle tests were evaluated for marketability. Arsenic is present in the copper concentrate at levels that may attract penalties in the early years, and zinc and lead are richer in the copper concentrates in the early years. Otherwise the concentrate qualities were good.

13.6.2 Variability Samples

Approximately 36 samples were studied using QEMSCAN as part of the Emba Derho studies. Variability studies on the Adi Nefas and Debawra sample have been described separately and are not included in this report.

Pyrite ranges from 35% to over 90% of the ore, and it is likely that the most pyrite-rich ores would require high pH levels to keep the pyrite depressed in copper and zinc flotation. Pyrrhotite is a minor feature of the zinc rich samples. Pyrrhotite is often quite readily copper-activated but in practice proved to be depressed well at high pH's. Non-sulphide mineralogy is consistently benign. The samples are consistently free from mica (or potentially illite) and kaolinite, suggesting consistently minimal silicate flotation or chemical interference with the sulphide float.

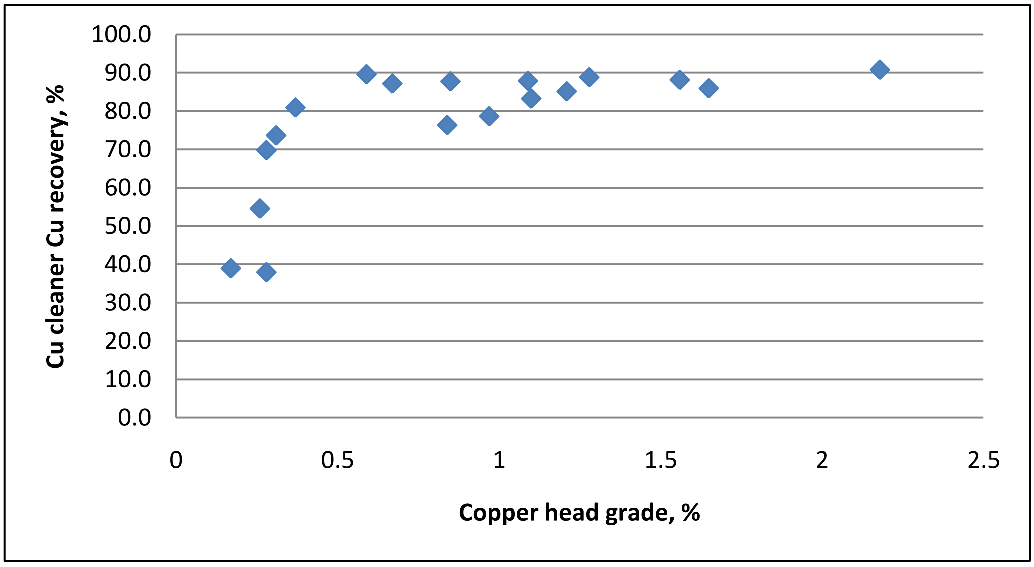
There is modest variability in the textural analysis of the copper sulphides with just one clear outlier in the zinc-rich suite and one marginal sample in the copper-rich suite. Otherwise, they show similar release characteristics to the locked cycle composites suggesting the chosen primary grind will have broad application throughout the deposit.

A general scarcity of the more soluble copper sulphides and textural release characteristics that span the range seen in the locked cycle test composites, suggest that if tested in locked cycle mode these samples likely would behave quite similarly to the locked cycle composites.

The zinc mineralogy appears to be still more consistent with the sphalerite release characteristics showing minimal variability from sample to sample, at least for the zinc-rich samples. They varied only marginally for the zinc poor samples, even though some of these samples contained virtually no zinc.

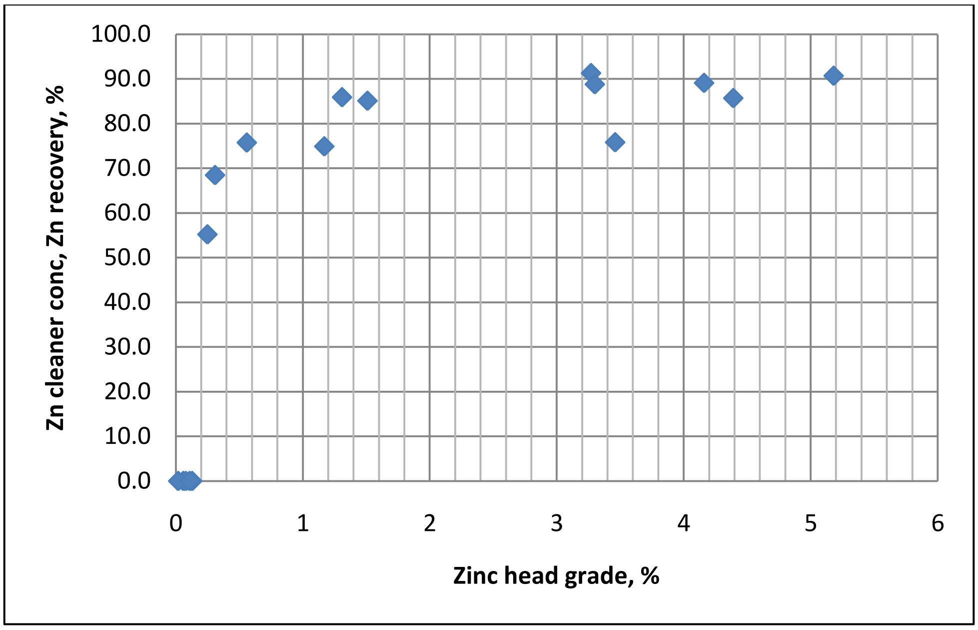
Eighteen of the variability samples were tested metallurgically. For all but one sample, the flowsheet was essentially identical to that used in the locked cycle tests (the one outlier included a small pre-float). The copper recoveries are plotted below. The samples assaying above 0.35% copper yielded batch cleaner recoveries averaging 86%, to concentrates assaying 23% copper or higher. Typically, testing in locked cycle mode has yielded recoveries 4-5% higher than in batch mode, suggesting these would have averaged roughly 90% recovery in locked cycle mode, much the same as the weighted average 89% recovery from locked cycle testing. Those with head grades below 0.35% copper yielded poorer recoveries.

Figure 13.5: Copper Head Grade vs Cleaner Circuit Recovery: ED Variability Testing



Zinc flotation performance in the variability study is shown below. Zinc concentrates assaying on average 55% zinc were floated at recoveries averaging 84%. The recoveries would have been 6-7% higher in locked cycle mode, where some of the zinc caught in batch testing, in the copper circuit middlings streams, and some in the zinc circuit middlings would have been recovered to final concentrate. This is consistent with the locked cycle data.

Figure 13.6: Zinc Head Grade vs Recovery to Final Concentrate: ED Variability Studies



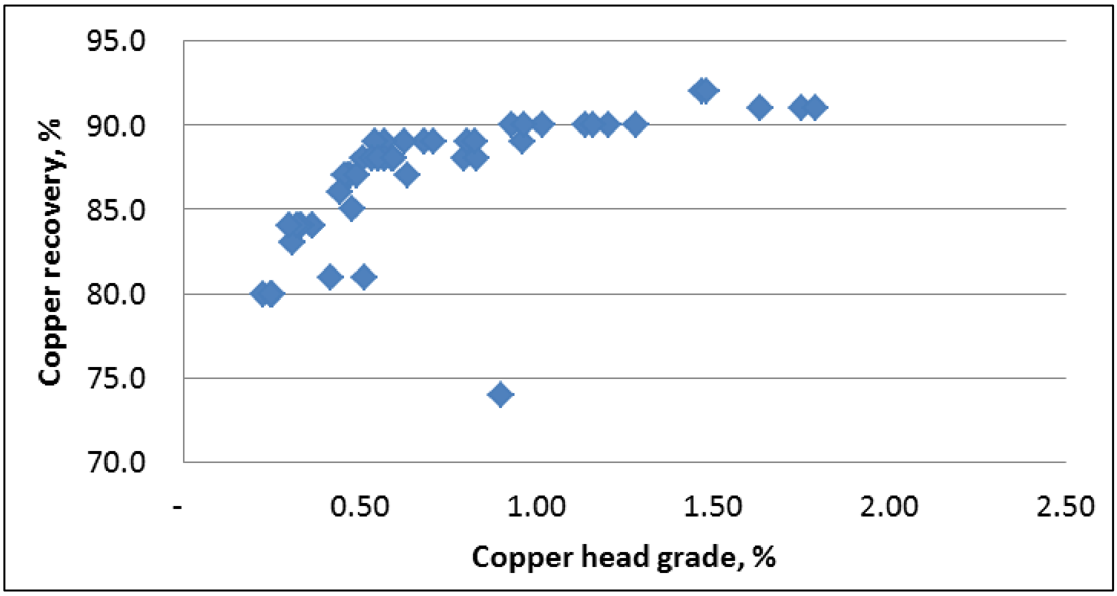
13.6.3 Final Primary Flowsheet Selection and Metallurgical Projections

The final process selected for primary ore processing was derived from the locked cycle flowsheets and conditions shown earlier.

Metallurgical projections from this study were derived from recovery correlations with the head grades and ore blends resulting from the Asmara project mine planning exercise. Early primary ore production quarters include material from Debarwa Primary (quarters 14 to 17) and Adi Nefas (quarters 15 to 36) blended with Emba Derho Primary.

The first quarter (quarter 14) of primary ore processing contains 85% Debarwa material, therefore LCT data from the Debarwa feasibility study were used to derive the metallurgical parameters for this period (hence the low projected recoveries). A discount of 5% was applied to the copper and zinc recoveries to account for ramp up reflecting the likely challenges associated with achieving selective flotation from these secondary copper-bearing ore sources. No further discounts in recovery due to “ramp up” were applied in subsequent periods. An discount, derived from past project experience, in copper recovery of 7% was applied to quarter 15 as no testwork data was conducted at the blend ratios provided (30% Debarwa, 10% Adi Nefas, 60% Emba Derho) and copper-zinc head grade provided. The head grades for this quarter were very close to the Year 0-1 composite tested in the lab so these recoveries with a downward adjustment to copper recovery were considered appropriate. Otherwise, projected recoveries followed the head grade/recovery relationships described in this section (adjusted for closed circuit processing).

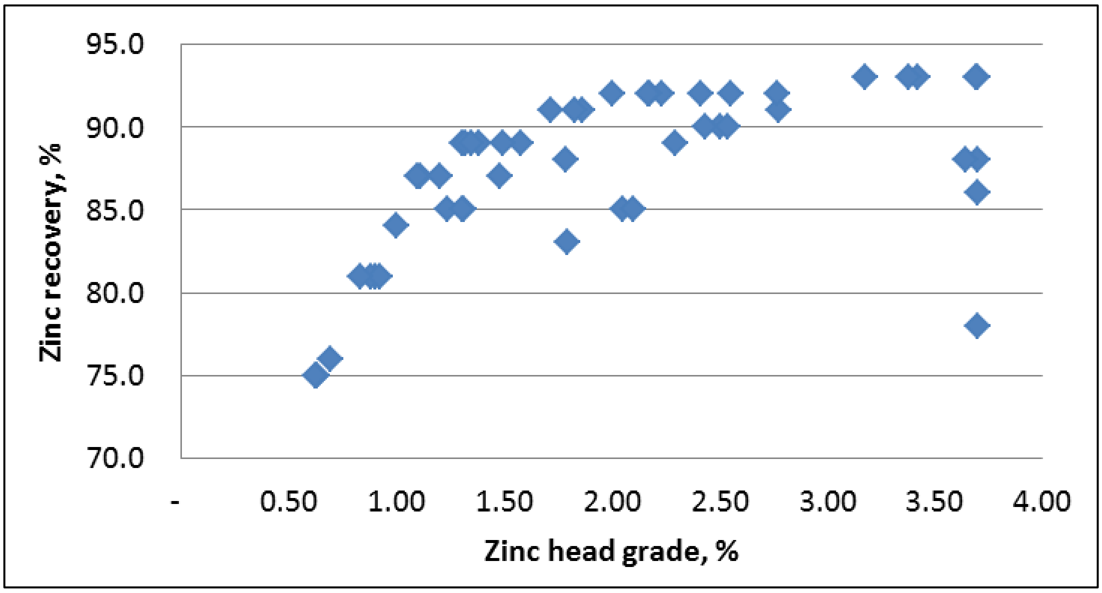
Figure 13.7: Projected Copper Head Grade vs Recovery



Note that outliers from the basic grade/recovery envelope are mostly due to unfavourable blends of ore sources (addition of Debarwa and/or Adi Nefas materials).

The projected zinc recoveries also follow the locked cycle data, supported where possible by the batch data. Typically recoveries are close to 90% for head grades of 1% or higher, but drop off quite steeply at levels below 1%. Note that where the recoveries deviate from the standard grade/recovery envelope, the ore mixes are different from those tested, or ramp up is expected to lead to poorer recoveries.

Figure 13.8: Projected Zinc Head Grade vs Recovery



13.7 QP Comment on Metallurgical Testwork

All the metallurgical and mineralogical testwork completed for the Emba Derho 43-101 has been completed at reputable Canadian or US testwork laboratories under the direct supervision of metallurgists with Blue Coast Metallurgy. Hence the data provided in this report can The Asmara project is comprised of a number of zones or ore types, mostly having their own individual processing requirements to achieve maximum return on investment. In reality, the Asmara Project is a series of VMS deposits and a single, smaller gold deposit and all ores are to be processed at a single processing facility.

Broken down into its simplest form, the Asmara Project is comprised of the following ore types:

- Gold ores that will be heap leached,
- Copper Supergene ores that will be processed via flotation to produce a copper concentrate,
- Copper/Zinc Primary ores that will be processed via flotation to produce a copper concentrate and a zinc concentrate.

The Asmara Project is comprised of the following deposits:

- Emba Derho – high tonnage, lower grade VMS
- Debarwa – low tonnage, high grade VMS
- Adi Nefas – low tonnage, high grade VMS
- Gupo – low tonnage, gold only deposit

The Emba Derho deposit is a sizeable copper/zinc/gold deposit located close to Asmara, Eritrea, and the cornerstone of the project. The deposit contains three distinct zones, oxide, supergene sulphide and primary sulphide. The primary sulphide zone is by far the largest in both tonnage and contained value. Therefore, the bulk of the metallurgical testwork effort has been focused on thoroughly understanding the primary zone metallurgy. In parallel with the testwork on the primary ores, the project team has also established the optimum process routes, and associated process conditions and metallurgical response for the supergene and oxide ores.

The primary zone can be considered to consist of a classic medium grained primary volcanogenic massive sulphide material. The presence of copper solely as chalcopyrite and zinc as a mostly high grade sphalerite helps make the flotation chemistry simple and relatively low cost. This allows for robust metallurgy and reliable locked cycle testing. The medium grained textures allow for a moderately fine primary grind requirement and moderately fine regrind requirements for the copper and zinc concentrates. All the above allows for good metallurgy, with copper recoveries averaging approximately 90% to concentrates typically assaying about 25% copper; and zinc recoveries also of 90% to concentrates assaying 58% zinc using a conventional, low technical risk flowsheet. Full element scans on the concentrates yielded no evidence of penalty elements at a level that would attract major smelter penalties – while the gold and silver contents were at levels that would attract a small pay from most smelters. Blending of the more mineralogically complex primary ores from Adi Nefas and Debarwa could lead to higher reagent costs and potentially higher smelter penalties on elements such as zinc and arsenic in the copper concentrate. However, these concentrates should attract far higher precious metal credits which should more than off-set the penalties.

Processing the supergene sulphides will also follow a conventional, low risk route. The richest supergene ore material from Debarwa is direct shipping quality and hence will be shipped without beneficiation. The rest can be processed through the plant as designed for the primary ores, at a lower tonnage allowing for a slightly finer primary grind and the use of the copper and zinc regrind capacity will provide sufficient power for regrinding the supergene copper rougher concentrates. Copper recoveries of roughly 85% will be achieved to concentrates assaying 25% copper.

Column leach tests using material from the Asmara Project yielded gold extractions ranging from 42% to 76%. The Gupo and Emba Derho Oxide composites showed the best and most consistent gold extractions ranging from 62% to 73%. Silver extractions ranged from 13% to 70%. Cyanide consumptions were relatively consistent and ranged from 0.82 kg/tonne to 1.35 kg/tonne. Agitation leaching yielded better metallurgy but given the modest size of the deposits, and the need to process the material early to improve project cashflow, the low-capital heap leaching option proved to be more attractive.

14 MINERAL RESOURCE ESTIMATES

14.1 General

Mineral Resource estimates for Emba Derho, Adi Nefas, Gupo and Debarwa were disclosed by independent Qualified Persons on behalf of SGC. The estimates comply with the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards on Mineral Resources and Mineral Reserves (CIM, 2010), as required by National Instrument 43-101.

The resource classification definitions used for these estimates are:

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

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

***Inferred Mineral Resource:** that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.*

Mineral Resource estimates reported below include Mineral Reserve estimates.

14.2 Emba Derho

The deposit is characterised on surface by a prominently outcropping gossan that measures approximately 800 m east-west by 220 m north-south. The gossan comprises: oxidised and supergene altered felsic tuffaceous rocks and flows, which are most likely the most abundant lithology; weathered massive to semi-massive sulphides; and orange-brown weathered rhyolite dykes (or possibly sills). Surrounding the gossan are typically well foliated altered fine tuffaceous rocks of both mafic and felsic composition. Other units, largely obscured by the processes of surficial weathering, include post-deformation granitic dykes of various compositions.

Massive sulphide lenses are developed beneath the gossanous, oxidised and supergene zones and their extents have been tested by SGC's' core drilling programs. Metals of potential economic significance are copper, zinc, gold and silver. Lead content is low.

The host rocks have been subject to at least two phases of tight folding forming a "W" fold which faces northwest, with fold axes plunging moderately to steeply to the northwest. An additional fold on the eastern extremity of the deposit plunges to the northeast.

The mineralization and host stratigraphy are cut by a number of phases of felsic intrusives.

The geological interpretation has identified four subsidiary zones in a strongly folded primary sequence intruded by post-mineralization felsic dykes and subjected to weathering; from top to bottom these are the gold oxide, copper supergene, zinc-rich and copper-rich primary zones. The oxide and supergene are characterised by strong weathering-related vertical zonation of depletion and enrichment, resulting in a sequence, from top-down, of near surface gold oxide and transition zones, through a supergene copper zone and a lowermost horizon of primary zinc and copper mineralization.

The updated resource estimate for the Emba Derho deposit was completed by Snowden Mining Industry Consultants Inc. is as of 6 February 2012. All resource modelling and grade estimation was undertaken by Andrew F. Ross, FAusIMM (CP Geo), PGeo, Snowden Senior Principal Consultant, a Qualified Person as defined by NI 43-101, based on geological interpretations and a drill database (current as at 9 September 2011) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed.

The mineralization on which the 2012 Emba Derho resource model is based extends over a strike length of 1,250 m and a width of 850 m and has been drilled to a maximum vertical depth from surface of approximately 500 m. The deposit has been explored using 322 exploration drillholes and 7 geotechnical drillholes. Two hundred and eighty (280) drillholes encountered mineralization and have been used in this estimation of resources. In this total were 236 diamond core drillholes and 44 reverse circulation drillholes.

The interpretations of mineralised zones from drilling and surface mapping were modelled using three-dimensional wire framing techniques based on a distribution of drill intersections ranging from less than 25 m spacing on 40 m drill section intervals (or closer) through to drill spacings in excess of 80 m by 50 m. The wireframe interpretations formed the basis for the construction of a block model as well as the constraining of samples for geostatistical analysis and grade estimation.

Due to the structural complexities inherent in the geological interpretation, a prototype block model was constructed with cells of 5 m x 5 m x 5 m (XYZ). This allowed the identification of the various mineralised zones and distinguished the broader non-mineralised felsic dykes that are expected to be selected as waste during mining.

Grades for copper, zinc, gold, silver, lead and iron were estimated within primary and weathered horizon control using Ordinary Kriging after compositing the assay intervals to nominal 1.5 m down-hole lengths. Where necessary, a limited number of high grade caps, as determined from statistical and spatial distributions, were applied to reduce the impact of grade bias during estimation. Search ellipsoid dimensions and orientations were determined on structural geological and geostatistical information. Density values were calculated for mineralised blocks based on regression formulae for iron, copper, zinc and lead estimates. Reporting of the resource estimates is based upon a 15 m x 15 m x 5 m (XYZ) resolution

The resource reporting was constrained by a conceptual pit shell and a conceptual assessment of underground mining extractability to identify those regions of the model that have reasonable prospects for eventual economic extraction.

Mineral Resource Estimates are reported in Table 14.1,Table 14.2, Table 14.3 and Table 14.4.

Table 14.1: Measured Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper| (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	-	-	-	-	-
Cu Supergene	0.5% Cu	-	-	-	-	-
Copper-rich Primary	0.3% Cu	0.97	1.50	0.23	11.3	3.64
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.19	2.68	0.32	12.5	0.78
TOTAL						**4.42**

Table 14.2: Indicated Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	0.07	0.04	1.06	4.3	1.74
Cu Supergene	0.5% Cu	0.94	0.38	0.17	12.2	1.64
Copper-rich Primary	0.3% Cu	0.81	0.89	0.16	7.44	46.19
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.14	2.81	0.31	9.82	15.97
TOTAL						**65.55**

Table 14.3: Measured & Indicated Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	0.07	0.04	1.06	4.3	1.74
Cu Supergene	0.5% Cu	0.94	0.38	0.17	12.2	1.64
Copper-rich Primary	0.3% Cu	0.83	0.93	0.17	7.7	49.8
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.14	2.80	0.31	9.9	16.8
TOTAL						**70.0**

Table 14.4: Inferred Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	-	-	-	-	-
Cu Supergene	0.5% Cu	-	-	-	-	-
Copper-rich Primary	0.3% Cu	0.87	0.89	0.25	10	13.28
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.20	1.94	0.39	11	1.77
TOTAL						**15.05**

For information and disclosure on the key assumptions, parameters and risks of the Mineral Resource estimates for Emba Derho the reader is referred to the previously filed document, as follows:

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Emba Derho Deposit, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Inc. with contributions by Blue Coast Metallurgy Ltd. 110 pages. Effective date 6 February 2012.

14.3 Adi Nefas

The Adi Nefas deposit occurs as a prominent outcropping gossan at surface that is the expression of deeper VMS mineralization. The gossan is traceable over a strike length of 700 m and is north-northeast trending. The deposit is a steeply east dipping massive sulphide layer that is hosted within a sequence of felsic meta-volcanic and meta-sedimentary rocks. The massive sulphide layer has an average width of 6 m to 12 m and is largely hosted within a hydrothermally altered felsic quartz-sericite-chlorite pyrite schist which is flanked in the hanging wall and footwall by altered meta-basaltic rocks. The deposit is partitioned into upper oxide and transition zones with precious metal enrichment and base metal depletion, underlain by a zinc and copper rich supergene zone, and a lower primary zinc-rich sulphide zone.

The primary zone is typically developed beyond a depth of 40 m from surface and is the focus of the prefeasibility study. The primary zone includes a unit of low grade chert and exhalite which has been modelled separately to the massive sulphides. Previous mineral resource estimates in 2008 did not partition this rock unit from the massive sulphides and as a result were lower grade and higher quantity than the current estimate.

The updated resource estimate for the Adi Nefas deposit was completed by Snowden Mining Industry Consultants Inc. is as of 20 February 2012. All resource modelling and grade estimation was undertaken by Snowden and reviewed by Andrew F. Ross, FAusIMM (CP Geo), PGeo, Snowden Senior Principal Consultant, a Qualified Person as defined by NI 43-101, based on geological interpretations and a drill database (current as at 19 September 2011) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed.

The mineralization on which this new Adi Nefas resource estimate is based extends over a strike length of 455 m and a width of up to 12 m and has been drilled to a maximum vertical depth from surface of approximately 400 m. The deposit has been explored using 101 exploration diamond core drillholes and 8 geotechnical drillholes. Seventy (70) drillholes encountered mineralization and assays of split core have been used in this estimation of resources.

The interpretations of mineralised zones were modelled in Gemcom mining software using three-dimensional wireframing techniques based on a distribution of drill intersections ranging from less than 25 m to up to 135 m spacing's on 40 m drill section intervals. The wireframe interpretations formed the basis for the construction of a block model as well as the constraining of samples for geostatistical analysis and grade estimation. A block model was constructed in the Datamine mining software with a parent cell dimension of 7.5 m (X) x 10 m (Y) x 5 m (Z) sub-celled to 0.5 m x 5 m x 2.5 m (XYZ) for accurate coding with the wireframe interpretations.

Grades for copper, zinc, gold, silver, lead and iron were estimated in the Datamine mining software within primary and weathered horizon control using Ordinary Kriging after compositing the assay intervals to 1.5 m down-hole lengths. No grade caps were applied.

Search ellipsoid dimensions and orientations were determined on structural geological and geostatistical information. Density values were calculated for blocks based on regression formulae for iron, copper, zinc and lead estimates.

The interpreted mineralised zones were categorised for resource classification as Indicated or Inferred in a series of steps. Each zone was reviewed in the context of the spatial distribution of drill intersections used to model and estimate grades for that zone, with due consideration for the known geological and geostatistical continuities and confidences in the base data and geological interpretations. On this basis the relatively densely drilled (40 m section spacing) primary zone received Indicated status.

The resource reporting was considered in the context of underground mining extractability and reasonable prospects for eventual economic extraction. Mineral Resource Estimates are reported in Table 14.5 below:

Table 14.5: Indicated Mineral Resource Estimate – Adi Nefas

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Primary	2.0 % Zn	1.78	10.05	3.31	115	1.841

For information and disclosure on the key assumptions, parameters and risks of the Mineral Resource estimates for Adi Nefas the reader is referred to the previously filed document, as follows:

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Inc. with contributions by Blue Coast Metallurgy Ltd. 82 pages. Effective date 20 February 2012.

14.4 Gupo

Immediately south of Adi Nefas village, approximately 1.5 km south of the VMS deposit, the Gupo gold deposit is related to quartz veining in a shear zone within heterolithic metadacite fragmental rocks. A strong structural control to mineralization is apparent with gold concentrated in second and third order structures. Mineralization is accompanied by a 200 m to 300 m wide quartz-feldspar-carbonate-sericite-pyrite hydrothermal alteration zone. Gold anomalism extends along strike for a distance of 1.6 km.

Low grade gold mineralization is associated with weak pyritic content.

The resource estimate for the Gupo deposit was completed by Snowden Mining Industry Consultants Inc. is as of 3 April 2012. All resource modelling and grade estimation was undertaken by Snowden and reviewed by Andrew F. Ross, FAusIMM (CP Geo), PGeo, Snowden Senior Principal Consultant, a Qualified Person as defined by NI 43-101, based on geological interpretations and a drill database (current as at 12 March 2012) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed.

The mineralization on which the 2012 Gupo resource estimate is based extends over a strike length of 1.6 km and a width of up to 60 m and has been drilled to a maximum vertical depth from surface of approximately 175 m. The resource has been estimated using 176 exploration drillholes consisting of 18 diamond drill core holes and 158 reverse circulation drillholes. Assays of split core and reverse circulation samples have been used in this estimation of resources.

The interpretations of mineralised zones as 0.1 g/t gold grade shells were modelled using three-dimensional wireframing techniques based on a distribution of drill intersections ranging from less than 15 m to up to 135 m spacing's on 20 m drill section intervals. The wireframe interpretations formed the basis for the construction of a block model as well as the constraining of samples for geostatistical analysis and grade estimation. A block model was constructed in Vulcan mining software with a parent cell dimension of 15 m (X) x 15 m (Y) x 5 m (Z) sub-celled to 5 m x 5 m x 2.5 m (XYZ) for accurate coding with the wireframe interpretations and surface topography.

Grades for gold were estimated within grade shells using Ordinary Kriging after compositing the assay intervals to 1.0 m down-hole lengths within the mineralised shell contacts. Search ellipsoid dimensions and orientations were determined on structural geological and geostatistical information. Density (specific gravity) values of 2.70 and 2.80 for oxide and primary horizons respectively were assigned to blocks based on 171 density measurements provided by SGC.

The interpreted mineralised zones were categorised for resource classification as Indicated or Inferred in a series of steps. Each zone was reviewed in the context of the spatial distribution of drill intersections used to model and estimate grades for that zone, with due consideration for the known geological and geostatistical continuities and confidences in the base data and geological interpretations. On this basis the relatively densely drilled (20 m section spacing) north zone received Indicated status where geological continuity could be demonstrated. All other areas with less drill density were classified as Inferred.

Mineral Resource estimates reported for Gupo are constrained by a conceptual pit shell in order to determine the potential quantity for eventual economic extraction. Parameters used in the generation of the Whittle pit shell are: gold price = US$2,300/ounce; mining cost = $2.10/tonne; processing cost (including administration cost) = US$28.48/tonne; recovery = 92% (oxide) 93.4% (primary); pit slope angle = 45 degrees.

Mineral Resource Estimates are reported in Table 14.6 and Table 14.7 below.

Table 14.6: Indicated Mineral Resource Estimate- Gupo

Cut-off grade	Gold (g/t)	Gold (oz)	Mass (t)
0.50 g/t Au	1.53	46,780	951,800

Table 14.7: Inferred Mineral Resource Estimate- Gupo

Cut-off grade	Gold (g/t)	Gold (oz)	Mass (t)
0.50 g/t Au	1.83	106,340	1,808,550

For information and disclosure on the key assumptions, parameters and risks of the Mineral Resource estimates for Gupo the reader is referred to the previously filed document, as follows:

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Gupo Gold Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc. for Sunridge Gold Corp. with contributions by Blue Coast Metallurgy Ltd. 97 pages. Effective date 3 April 2012.

14.5 Debarwa

The Debarwa deposit appears on surface as a prominently outcropping gossan, the surface expression of the massive sulphide deposit. The gossan is developed over a strike length of approximately 1,350 m and is north-northeast trending, sub-parallel to the regional tectono-stratigraphic trend. Sulphide mineralization is hosted within variably but generally heavily sulphide-altered, and moderately sericite-, chlorite-, and quartz-altered predominantly felsic metavolcanic rocks. The mineralised zone at Debarwa dips approximately 50º to 60º to the west and is generally 8 m to 30 m wide. A surface oxide gold zone, from which base metals have been predominantly leached, extends to approximately 65 m depth from the highest points (between 25 m and 40 m below the floor of the Gual Mereb River valley). A higher grade sulphate rich gold and silver transition zone to 80 m depth is underlain by an enriched copper supergene zone to around 110 m depth. The supergene zone is in turn underlain by a copper/zinc-rich primary sulphide zone.

The updated resource estimate for the Debarwa Project was completed by AMC Consultants (UK) Ltd, is as of 11 August 2011. All resource modelling and grade estimation was undertaken by Chris Arnold, MAusIMM (CP Geo), AMC Principal Geologist, a Qualified Person as defined by NI 43-101, based on geological interpretations and a drill database (current as at 20 April 2011) provided by SGC. The database was subjected to various validation steps and the SGC sampling and assaying QA/QC procedures and results were reviewed. Snowden has reviewed this work and finds it appropriate for the purpose of this study.

The mineralization on which the 2011 Debarwa resource model is based extends over a strike length of 1,250 m and dips westerly at approximately 50° and has been drilled to a maximum vertical depth from surface of 250 m. The deposit has been explored using 392 exploration holes of which 314 have been used in the estimation of resources, including 268 diamond core and 46 reverse-circulation holes.

The geological interpretation of the Debarwa VMS deposit has identified two upper oxide gold-enriched zones (the oxide and transition zones), a copper enriched supergene zone, as well as enriched stringer zones and seven relatively small subsidiary zones. The oxide, transition and copper supergene zones exhibit strong weathering-related vertical zonation of depletion and enrichment, resulting in a sequence, from top-down, of near surface gold oxide and transition zones, through a supergene copper zone and a lowermost horizon of primary zinc and copper mineralization.

The interpretations of mineralised zones were modelled using three-dimensional wireframing techniques, based on a distribution of drill intersections ranging from less than 10 m spacing on 10 m drill section intervals through to drill densities in excess of 40 m by 40 m. The wireframe solids formed the basis for the construction of a block model as well as the constraining of samples for zonal analysis and grade estimation. The mineralised zones were further differentiated on the basis of weathering zone.

Grades for copper, zinc, gold and silver were estimated under zonal and weathering horizon control using ordinary kriging for the two main enriched zones and by inverse distance squared weighting for the remainder. Where necessary, a limited number of high grade caps, as determined from statistical and spatial distributions, were applied to reduce the risk of extreme grade bias during estimation. Search ellipsoid dimensions and orientations were determined on geological and geostatistical information. Where sufficient measurements existed, density values were interpolated into blocks by zone and enrichment horizon using inverse distance squared weighting, or elsewhere applied as calculated mean values.

The interpreted mineralised zones were categorised for resource classification as Measured, Indicated or Inferred in a series of steps. Each zone was reviewed in the context of the spatial distribution of drill intersections used to model and estimate grades for that zone, with due consideration for the known geological and geostatistical continuities and confidences in the base data and geological interpretations. On this basis the relatively densely drilled (approximately 10 m x 10 m) northern main zone received Measured status while, in general, areas with 20 m x 20 m spacing, and in some cases greater, were allocated to the Indicated category.

The preliminary classified block model was then subjected to two levels of constraint to ensure that only those portions which demonstrated potential economic viability were retained. Firstly an optimised pit shell derived using metal price parameters at a premium above long term prices (copper $3.00 per pound, gold $1,200 per ounce, zinc $1.00 per pound and silver $20.00 per ounce) was used to identify potential open pit material, after which optimised stope shapes, based on the same prices, were used to incorporate further material considered to be potentially mineable by underground methods.

Mineral Resource Estimates are reported inTable 14.8, Table 14.9, Table 14.10 and Table 14.11 below. .

Table 14.8: Measured Resources - Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (gt)	Mass (kt)
Oxide	Au 0.5 g/t	0.01	0.01	1.03	4	3
Transition	Au 0. 5g/t	0.07	0.03	4.59	90	103
Supergene	Cu 0.5%	11.63	0.07	2.58	65	321
Primary (Cu)	Cu 0.5%	2.39	5.97	1.32	26	7
Primary (Zn)	ZN 2.0% (Cu<0.5%)					
Total						**434**

Table 14.9: Indicated Resources – Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (gt)	Mass (kt)
Oxide	Au 0.5g/t	0.06	0.05	1.47	6	368
Transition	Au 0.5g/t	0.08	0.06	2.55	17	617
Supergene	Cu 0.5%	3.21	0.08	1.04	23	1,068
Primary (Cu)	Cu 0.5%	2.34	3.90	1.30	29	767
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.36	3.05	1.24	22	58
Total						**2 878**

Table 14.10: Measured and Indicated Resources – Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5g/t	0.06	0.04	1.47	6	371
Transition	Au 0.5g/t	0.08	0.05	2.85	27	720
Supergene	Cu 0.5%	5.15	0.07	1.40	33	1,389
Primary (Cu)	Cu 0.5%	2.34	3.92	1.30	29	774
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.36	3.05	1.24	22	58
Total						**3 312**

Table 14.11: Inferred Resources – Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5g/t	0.1	0.1	1.1	5	239
Transition	Au 0.5g/t	0.1	0.0	1.4	22	138
Supergene	Cu 0.5%	2.7	0.1	0.6	31	144
Primary (Cu)	Cu 0.5%	1.2	3.6	2.6	41	154
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.4	3.3	1.1	21	6
Total						**681**

For information and disclosure on the key assumptions, parameters and risks of the Mineral Resource estimates for Debarwa the reader is referred to the previously filed document, as follows:

Hopley, M.J., Arnold, C.G., Martin, C.J. (2011). Debarwa Copper Gold Deposit Eritrea Technical Report on Additional Drilling and Revised Mineral Resource Estimates. NI 43-101 Technical Report prepared by and for Sunridge Gold Corp. with contributions by AMC Consultants (UK) Limited and Blue Coast Metallurgy Ltd. 188 pages. Effective date 18 August 2011.

15 MINERAL RESERVES

Mineral Reserves, which are a subset of the Mineral Resources described in Section 14, have been prepared for the Asmara project as part of the Feasibility Study.

In accordance with the CIM Definition Standards on Mineral Resources and Mineral Reserves (as adopted and amended), Mineral Reserves are classified as either “probable” or “proven” Mineral Reserves and are based on Indicated and Measured Mineral Resources only. No Mineral Reserves have been estimated using Inferred Mineral Resources.

15.1 Mineral Inventory Summary

The study used the recently completed estimate of Measured and Indicated Resources for the Asmara project as described in section 14. Table 15.1 summarizes the Mineral Reserves included in the study reported by both ore type and classification.

Table 15.1: Mineral Reserves

Rock Type	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Tonnes (kt)
Proven					
Emba Derho Primary	0.9	1.7	0.2	11.6	4,337
Debarwa Oxide	-	-	1.0	6.7	1
Debarwa Transition	-	-	4.3	84.1	94
Debarwa Supergene	8.9	0.2	2.2	53.2	423
Debarwa Primary	1.6	2.8	0.6	15.6	6
Total Proven					**4,861**
Probable					
Emba Derho Supergene	1.0	0.4	0.3	14.9	1,200
Emba Derho Primary	0.7	1.6	0.3	9.2	44,497
Debarwa Oxide	-	-	1.6	8.2	163
Debarwa Transition	-	-	2.5	17.0	428
Debarwa Supergene	2.5	0.2	1.0	22.9	888
Debarwa Primary	1.9	4.0	1.1	25.4	514
Adi Nefas Primary	1.6	8.2	2.8	96.5	1,682
Gupo Oxide	-	-	1.9	-	399
Gupo Sulfide	-	-	2.4	-	66
Total Probable					**51,723**
Total Proven and Probable					**56,584**

The Mineral Reserves were estimated for Emba Derho, Debarwa and Gupo by generating Net Smelter Return (NSR) values (revenue minus royalty and smelting/selling costs) for each metal.

Metal prices used were \$2.80/lb copper, \$0.80/lb zinc, \$1,150/oz gold and \$18.50/oz silver. The net revenue of each block was compared to total cost. Each mining block becomes economical and included in the processing schedule and becomes part of the Mineral Reserve if it is above the total cost of processing, general administrative and applicable transport.

In the case of the Adi Nefas underground mine the Mineral Reserves were generated, using the same metal prices as above, through a sequential process of NSR calculation, stope optimization, stope design, and development design. Stope optimization was applied using Snowden's Stopesizor software which modifies the resource to reflect minimum mining width for the NSR. The outcome is a set of blocks that reflect this recoverable resource. Unplanned dilution was added to the model through adding a fixed width of over break waste into the planned stopes.

15.2 Material Factors Affecting Mineral Reserve Estimation

The Mineral Reserves could be affected by changes in metal price, capital and operating costs, metallurgical performance, infrastructure requirements, permitting or other factors. These factors are discussed below. The major risks to the Mineral Reserves are factors that either effects the costs incurred or the revenue received.

To mitigate the revenue risk conservative commodity prices have been used in the economic evaluation of the resource to provide a practical basis for revenue estimation. There always remains a risk that the Mineral Reserves could materially change, either increased or decreased, should significant adjustments occur in commodity prices over the life of the project.

The metallurgical testwork has indicated that the minerals are able to be economically recovered using existing technology and methodology. This follow up testwork will confirm the parameters to be used in future mine design and reserve estimation processes. Metallurgical performance has a direct effect on the revenue received and increase or decreases in performance will change the amount of metal recovered and hence the revenue received.

Permitting is not expected to be a material risk to the project as there have been no indications to date that there are any social, regulatory or community issues that cannot be managed through best practice operating standards and/or risk management planning and mitigation measures. Permitting remains a risk to the reserves until the granting of the mining license as part of the outcomes of the feasibility studies and the successful submission of the permitting and license to operate requirements that will be outcomes of the final social, environmental and community studies.

There are no perceived infrastructure risks to the reserve estimation process. The infrastructure is either existing or of a relatively standard type to install during construction of the project. The site is relatively clear of vegetation and has no extremes of temperature, elevation or climate that could adversely impact infrastructure installation, provided that is designed and constructed according to practical and statutory design codes and standards. Where local statutory best practice standards and codes do not apply, it is assumed that industry best practice would apply.

16 MINING METHODS

16.1 Open Pit

The three open pits (Emba Derho, Debarwa and Gupo) were designed through a standard process of pit optimization, waste dump design and pit design.

Pit optimizations were completed in Whittle 4X software, an industry standard package. This software determines the economic limits of each deposit after accounting for estimated revenues and costs associated with mining each resource and waste block and the maximum allowable slope angles. The results of the pit optimization were pit shells which were used for subsequent planning processes.

The pit optimization is used to derive volumes for waste dump placement. Meeting workshops were completed with relevant parties to decide on a waste disposal concept for each pit. After calculating the volumes of each waste type, waste dumps were designed to contain this material, meet environmental and social constraints, and minimise required haulage distances as much as possible.

Pit designs were completed in the general mine planning package, GEMS and Minesight. Pits were designed to minimise waste and ore deviation in comparison to the pit shells from the optimization as well as conform to the defined geotechnical wall parameters. Additionally, ramps were designed to provide easy access to the relevant dumping areas.

Stages were designed within the final pit design to expedite the winning of ore and delaying waste stripping. Various approaches were applied in determining the staging concept for each deposit. These are explained in the relevant sections below.

The Adi Nefas underground mine was designed through a sequential process of cut-off grade calculation, stope optimization, stope design, and development design.

The cut-off grade for design was calculated by approximating the mining and processing cost to determine a total ore cost. The net revenue of each block (revenue after selling costs, royalties and deductions) was compared to this total cost.

Stope optimization was applied using Snowden’s Stopesizor software which modifies the resource to reflect minimum mining width for the selected cut-off grade. The outcome is a set of blocks that reflect this recoverable resource. Unplanned dilution was added to the model through adding a fixed width of overbreak waste into the planned stopes.

Development for access, haulage and ventilation were designed in Datamine.

16.2 General Parameters

A number of parameters[5] used for pit optimization are common to all deposits and are listed in the following sections. Those parameters that are specific to a deposit are listed in the relevant sections.

16.2.1 Discount Rate

A discount rate of 10% per annum was applied to generate indicative net present values (NPV).

[5] These parameters were used for pit optimization and do not necessarily reflect the final numbers used for financial modelling. All changes to the final parameters were tested for materiality and were found to not affect the pit optimization result.

16.2.2 Administration Costs

Three general phases were assumed for applying administration costs (Table 16.1). The gold ore and DSO phases did not have administration costs applied as they are decoupled from the main plant and thus do not influence the length of the project (and the resultant fixed costs).

Table 16.1: Administration Costs

	Production rate (ktpa)	Total admin cost ($Mpa)	Unit cost ($/t ore)
Gold ore-[6]	1,300	-	-
DSO ore	240	-	-
Supergene ore	2,000	10.0	5.00
Primary ore	4,000	10.6	2.66

16.2.3 Commodity Prices

The commodities prices supplied by SGC for use in this study are shown in Table 16.2.

Table 16.2: Commodity Prices

Metal	Units	Price
Gold	$/oz	1,150
Silver	$/oz	18.50
Copper	$/t	6,160
Zinc	$/t	1,750

16.2.4 Royalties

Royalties are applied as a percentage of revenue (net of smelter deductions). These are shown in Table 16.3.

Table 16.3: Royalties

Metal	Units	Price
Gold	%	5.0
Silver	%	5.0
Copper	%	3.5
Zinc	%	3.5

[6] Gold ore is not allocated admin cost as this ore is decoupled from the main plant and does not affect the life of the project. Therefore costs associated with increasing project length should not be applied. However, administration costs will be applied in financial model during periods that gold ore is being processed.

16.2.5 Doré Smelter Terms

The smelter terms for doré are shown in Table 16.4.

Table 16.4: Doré Smelter Terms

Item	Units	GOLD	SILVER
Payable metal	%	99.5	99.5
Treatment cost	$/oz	0.25	0.25
Refining cost	$/oz payable	0.75	0.00

16.2.6 Copper Concentrate Smelter Terms

The smelter terms for copper concentrate are shown in Table 16.5.

Table 16.5: Copper Concentrate Smelter Terms

Item	GOLD	SILVER	COPPER
Minimum deduction	1 g/t	30 g/t	1% Cu
Payable	90%	90%	97%
Refining cost	$18.00/oz (payable)	$2.50/oz (payable)	$0.065/lb (payable)
Treatment cost			$65.00/t conc (dry)
Transport and shipping cost			$62.00/t conc (wet)
Port cost			$7.00/t conc (wet)
Moisture			9%

16.2.7 Zinc Concentrate Smelter Terms

The smelter terms for zinc concentrate are shown in Table 16.6.

Table 16.6: Zinc Concentrate Smelter Terms

Item	Zinc
Minimum deduction	8% Zn
Payable	85%
Refining cost	-
Treatment cost	$195/t conc (dry)
Transport cost	$73.50/t conc (wet)
Port cost	$7.00/t conc (wet)
Moisture	7.5%

16.3 Emba Derho Design

16.3.1 Mining Method

Open pit and underground mining of Emba Derho was considered. However, given the indicative strip ratio of the pit is approximately 2.5:1, open pit mining is clearly preferred. The open pit extracts most of the resource; however, the northwest extension of the orebody, which is under approximately 200 m of waste and sits outside the economic pit and has potential for future underground mining. Currently there is approximately 10 Mt of potentially economic material in this area. This part of the resource is currently all classified as Inferred Mineral Resources. Consequently, only open pit mining of Emba Derho is included in this study.

16.3.2 Pit Optimization

16.3.2.1 Basis

General parameters are shown in Section 16.2. In addition, there are a number of parameters specific to Emba Derho. These are described in the following sections.

16.3.2.2 Geotechnical

The PFS design for Emba Derho was analysed by AMEC with updated geotechnical information. A new division of geotechnical sectors was applied (Figure 16.1) with the slope parameters for each shown in Table 16.7.

Figure 16.1: Geotechnical Design Sectors

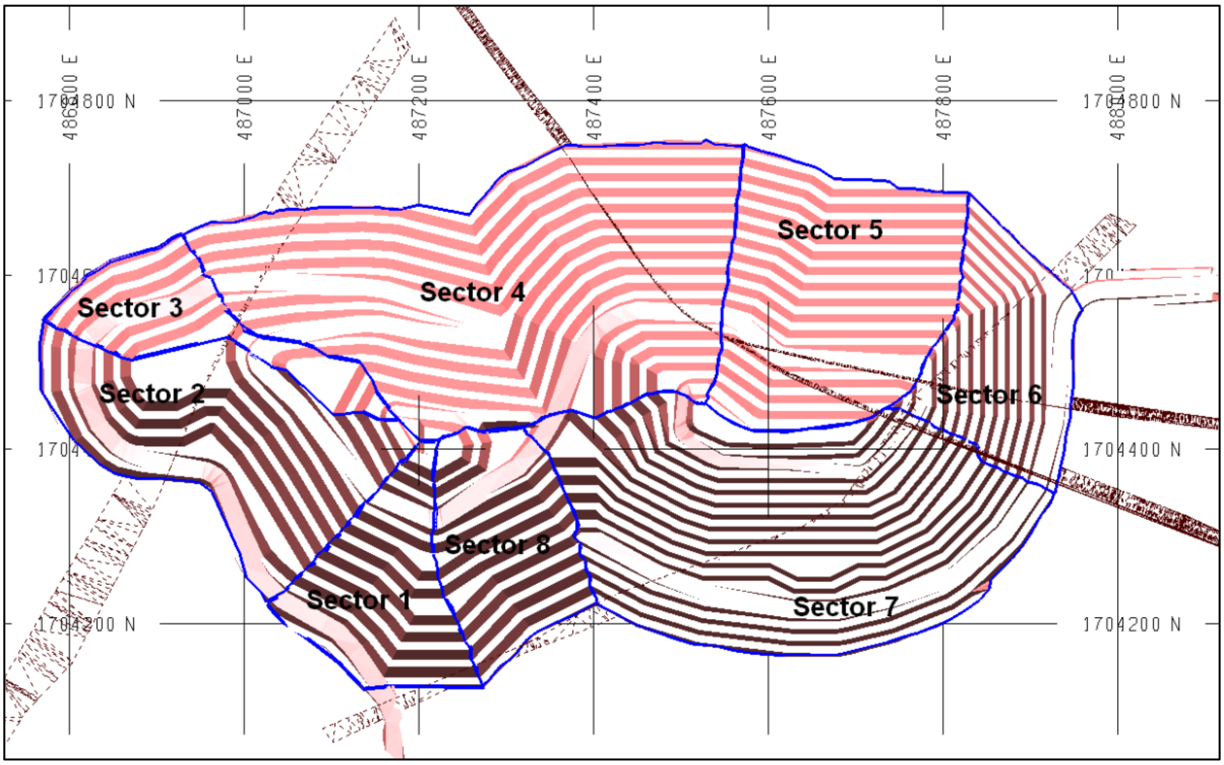


Table 16.7: Emba Derho Geotechnical Design Parameters

Design Sector	BFA[7] (deg)	Approximate Slope Height (m)	Bench width (m)	Bench height (m)	IRA[8] (deg)	OVA[9] (deg)	Geotechnical Berm or Ramp
1	67	200	6.5	10	42.9	42.7	15 m wide at each 150 m vertical interval
			8.5	20	49.7	49.2	20 m wide at each 150 m vertical interval
2	76	200	6.5	10	48.0	47.7	15 m wide at each 150 m vertical interval
			8.5	20	56.0	55.4	20 m wide at each 150 m vertical interval
3	60	200	6.5	10	39.2	38.9	15 m wide at each 150 m vertical interval
			8.5	20	44.9	44.5	20 m wide at each 150 m vertical interval
4	65	300	6.5	10	41.9	41.7	15 m wide at each 150 m vertical interval
			8.5	20	48.3	48.0	20 m wide at each 150 m vertical interval
5	70	300	6.5	10	44.6	44.4	15 m wide at each 150 m vertical interval
			8.5	20	51.7	51.4	20 m wide at each 150 m vertical interval
6	70	300	6.5	10	44.6	44.4	15 m wide at each 150 m vertical interval
			8.5	20	51.7	51.4	20 m wide at each 150 m vertical interval
7	76	300	6.5	10	48.0	47.8	15 m wide at each 150 m vertical interval
			8.5	20	56.0	55.6	20 m wide at each 150 m vertical interval
8	71	200	6.5	10	45.2	45.0	15 m wide at each 150 m vertical interval
			8.5	20	52.4	52.1	20 m wide at each 150 m vertical interval

The design assumed a 20 m bench spacing for all sectors. In the optimization the angles for each wall were reduced by 3 degrees for pit optimization to allow for ramps in the final design. For pit optimization, the bench face angles and berms were coded into the block model to ensure correct parameters were adhered to.

16.3.2.3 Mining

A cost of $2.31/t mined at surface (approx. 2,300 mRL) was applied, plus an incremental cost of $0.01/t per metre below the surface for pit optimization. In this way, the additional cost of hauling material to the surface is considered in the optimization.

16.3.2.4 Processing Parameters

Processing parameters used in the optimization for Emba Derho were provided by BCM (recoveries) and SENET (processing costs), and are shown in Table 16.8.

[7] BFA = Bench face angle
[8] IRA = inter-ramp angle, measured from toe to toe
[9] OVA = overall wall angle, measured from toe to crest

Table 16.8: Emba Derho Processing Parameters

		Cu concentrate/Dore				Zn concentrate	
	Processing cost ($/t ore)	Cu recovery (%)	Cu conc (%)	Au recovery (%)	Ag recovery (%)	Zn recovery (%)	Zn conc (%)
Oxide ore	11.88	-	-	70.0	53.0	-	-
Supergene ore	21.84	80.0	27.0	56.0	61.0	-	-
Primary ore	14.00	91.0	24.6	35.0	38.0	91.0	59.5

Ore Transport

No additional costs for ore haulage were added due to the location of the plant at Emba Derho. The haulage cost of transporting gold ore to the TSF leaching location was incorporated into the processing cost for this rock type.

Mining Block Model

The Emba Derho resource was completed by Snowden in February 2012. A summary of the resource is provided in Table 16.9. This resource is unchanged from the PFS.

Table 16.9: Emba Derho Resource Summary

Resource classification/ material type	Cut-off	(Mt)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)
Measured		**4.42**				
Primary Cu	Cu 0.3%	3.64	0.97	1.50	0.23	11.3
Primary Zn	Zn 1.0% (Cu<0.3%)	0.78	0.19	2.68	0.32	12.5
Indicated		**65.55**				
Oxide	Au 0.5 g/t	1.74	0.07	0.04	1.06	4.33
Supergene	Cu 0.5%	1.64	0.94	0.38	0.17	12.24
Primary Cu	Cu 0.3%	46.19	0.81	0.89	0.16	7.44
Primary Zn	Zn 1.0% (Cu<0.3%)	15.97	0.14	2.81	0.31	9.82
Inferred		**15.05**				
Primary Cu	Cu 0.3%	13.28	0.87	0.89	0.25	10.03
Primary Zn	Zn 1.0% (Cu<0.3%)	1.77	0.20	1.94	0.39	10.72

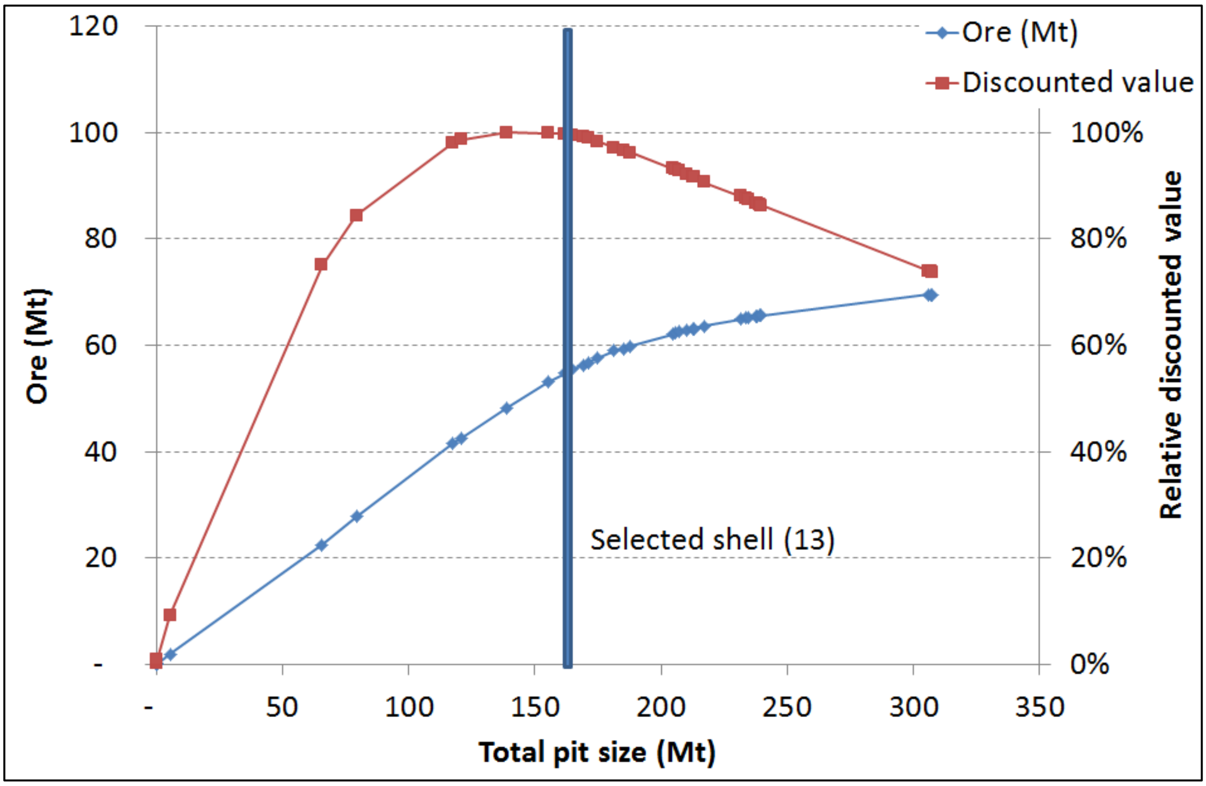
Importantly, the Inferred Resources are at depth and to the northwest. This material is not of immediate interest for open pit mining.

As the Emba Derho model was based upon 15 mE by 15 mN by 5 mRL blocks (reblocked from 5 mE by 5 mN by 5 mRL), it was assumed that this block size was sufficiently large to account for mining recovery and dilution and consequently no further losses or dilution were applied.

Results

A number of pit shells were generated, parameterized by revenue, with factors between 20% and 200% of the base case revenue assumptions. The discounted value for each of the pit shells was then estimated at the base case prices, assuming cash flows are spread evenly over the project life, with life determined by a nominal process feed rate of 4 Mtpa. The results are shown in Figure 16.2. A pit shell size of 13 (based on a revenue factor of 80%) was selected for design as this maximizes inventory while producing very close to optimal discounted value.

Figure 16.2: Emba Derho Pit Optimization Results



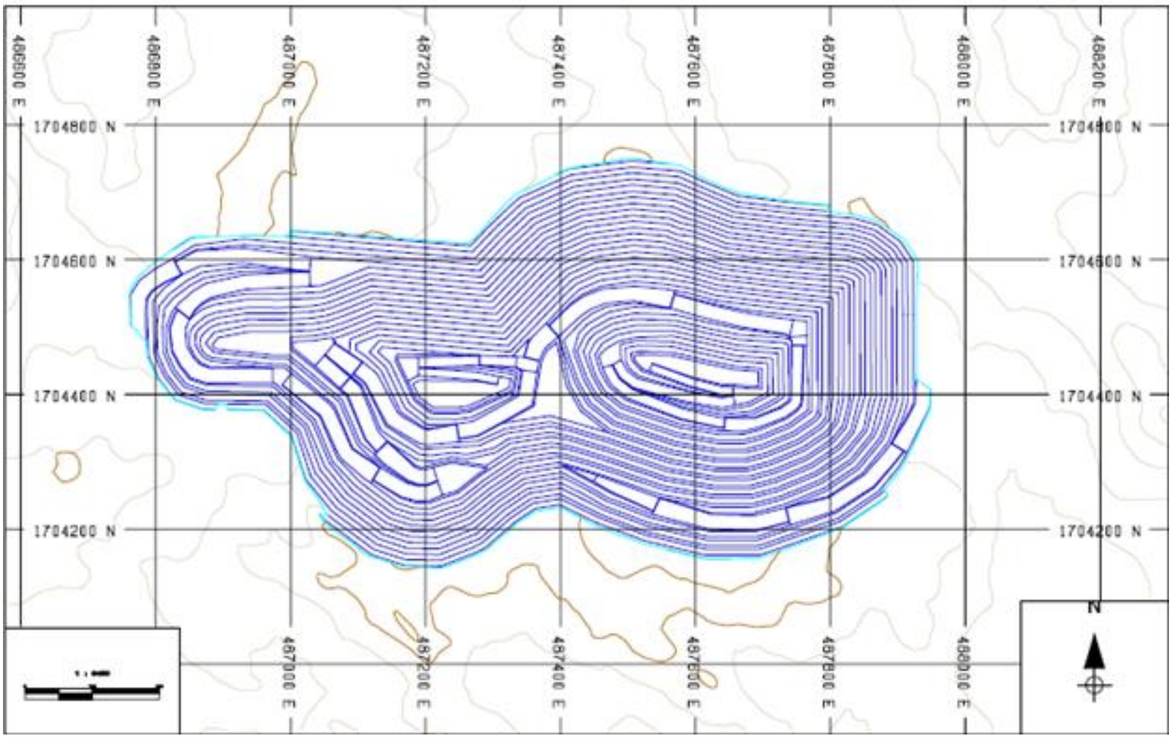
Source: Snowden

Pit Design

The design basis for bench face angles and berm widths are shown in Table 16.7. Haulage roads of 22 m were designed for dual lane access, and 15 m for single lane access at the base of the pit. Snowden has used ramps to satisfy the geotechnical berm requirements.

The final pit design is shown in Figure 16.3. The pit is 1,200 m long by 500 m wide and ranges in depth from 150 m in the west to 300 m in the east.

Figure 16.3: Emba Derho Final Pit Design



Source: Snowden

The reconciliation of volumes and value between the pit shell selected for design and the design itself is shown in Table 16.10.

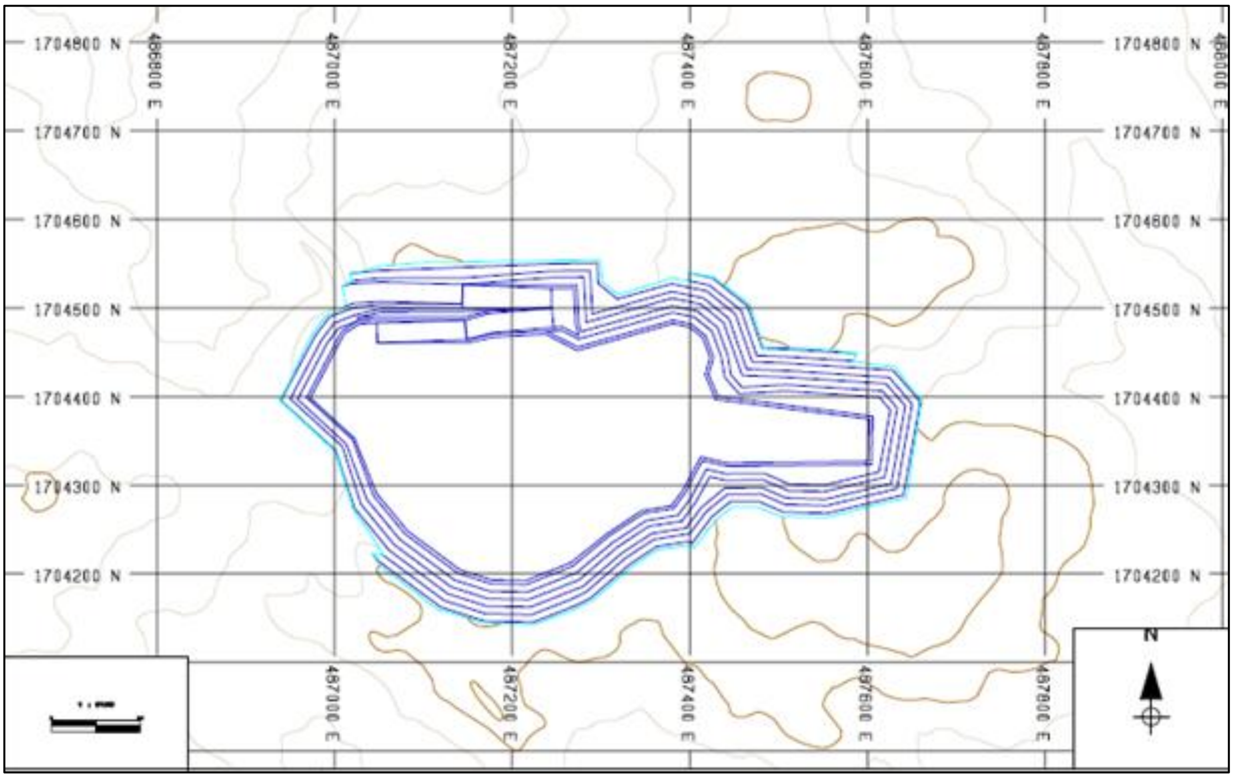
Table 16.10: Emba Derho Design Inventory Comparison[10]

	Design	Pit shell	Difference (%)
Total pit (Mt)	179.5	162.0	11%
Waste (Mt)	123.8	107.7	15%
Oxide ore (kt)	1,923	1,923	0%
Supergene ore (kt)	1,449	1,375	5%
Primary ore (kt)	52,354	51,012	3%
Strip ratio (w:o)	2.22	1.98	12%
Cash flow	1,167.9	1,214.4	(4%)

[10] "Ore" is reported as tonnes above a marginal cut-off grade. The final inventory is reduced due to considerations for blending, process timing, and stockpile size limitations. The material that is rejected is low grade.

Given the limited window of processing capability for oxide and supergene ores, a starter pit (ED1) was designed to extract this material (Figure 16.4).

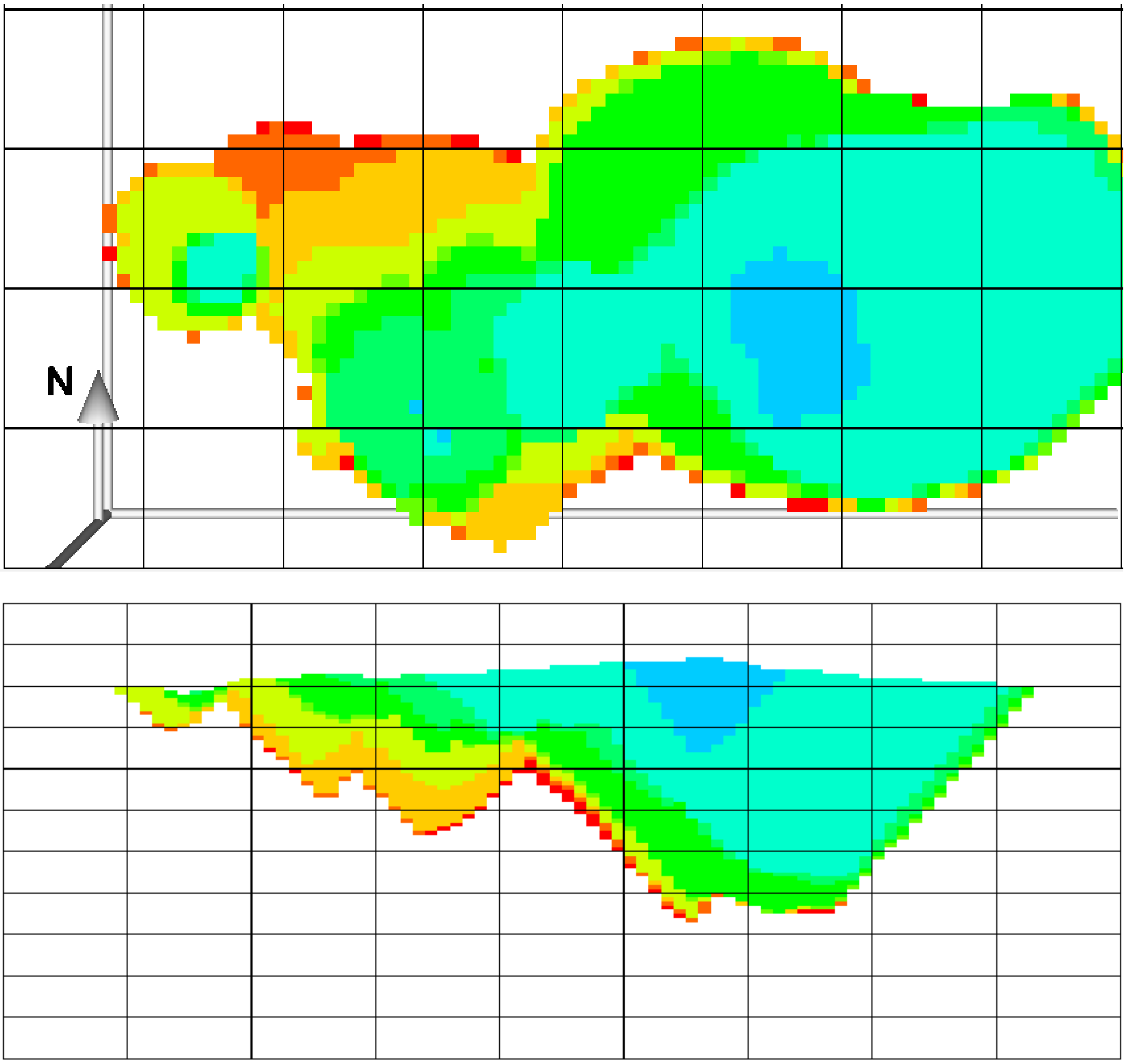
Figure 16.4: Emba Derho Stage 1 Design



Source: Snowden

Review of the lower revenue factor pit shells reveals that these pits tends to be driven on deep, higher grade copper ores in the east of the pit. Therefore this area was targeted for an interim stage (known as ED2).

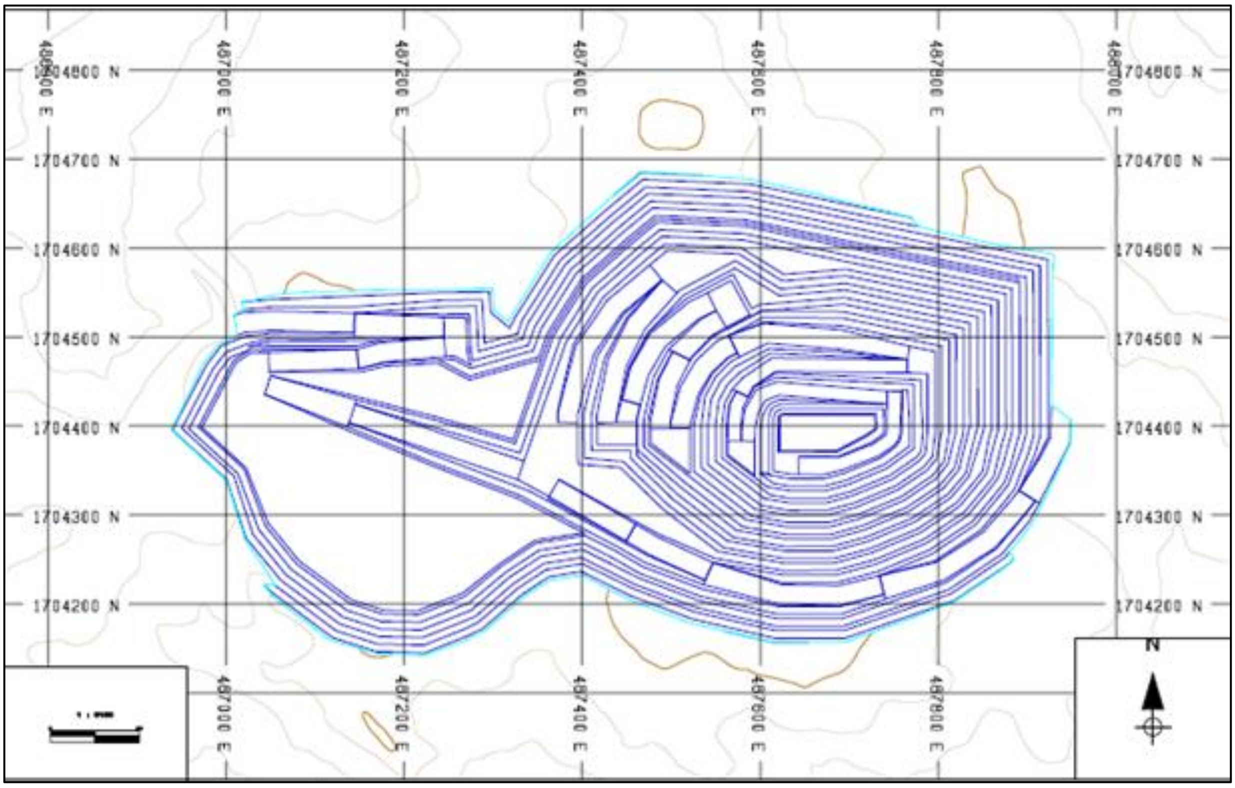
Figure 16.5: Emba Dehro pit Shell Progression [11]



Source: Snowden

[11] Cool colours, lower revenue factor in plan (top) and in long section looking north

Figure 16.6: Emba Derho Stage 2 Design



Source: Snowden

The final mining inventory for Emba Derho, after considering additional factors such as the cost of stockpiling, process scheduling and process plant availability is shown in Table 16.11.

16.3.3 Mining Site Layout Design

16.3.3.1 Overall

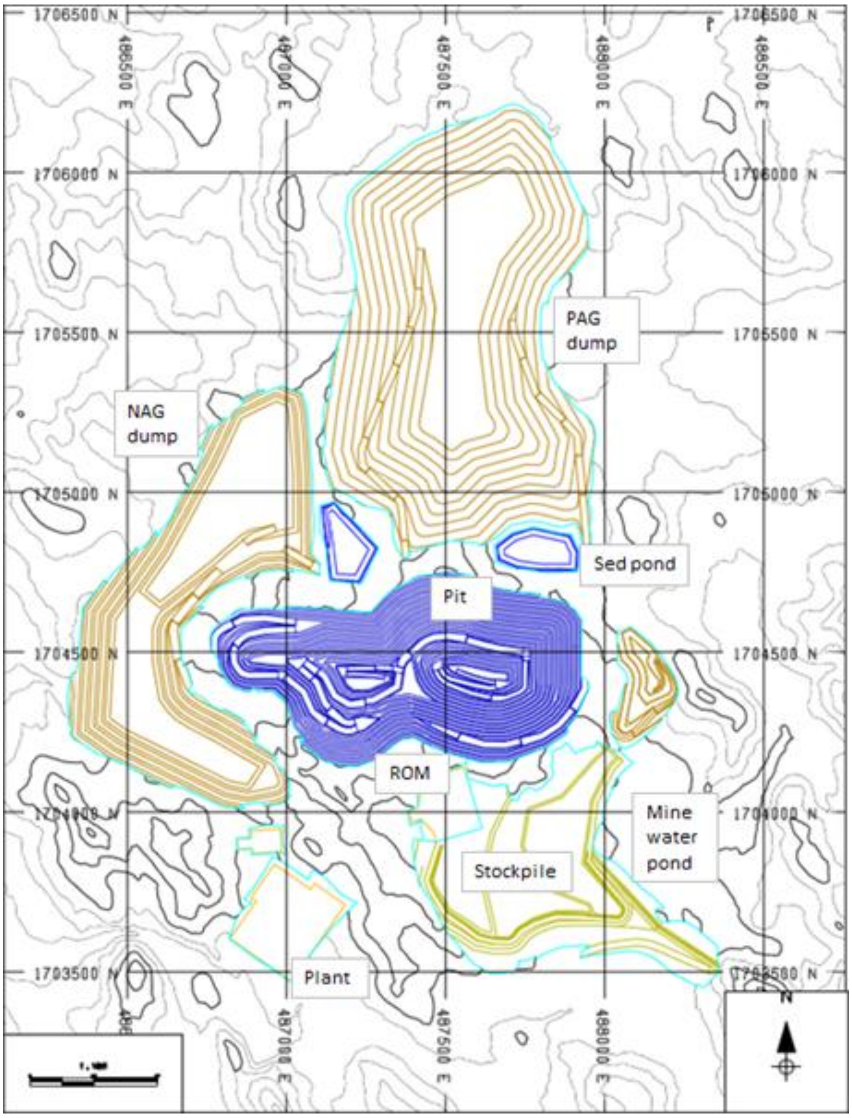
The mining layout was aimed to manage potentially acid generation material (PAG), and provide sufficient capacity to store non-acid generating material (NAG), as well as low grade and topsoil stockpiles. Assuming a 28% swell factor the required storage volumes for each storage type is shown in Table 16.11.

Table 16.11: Emba Derho Required Storage Volumes

Storage Type	Loose Cubic Metres (Mm^3)
Top soil	0.5
Low grade stockpile	4.4
PAG	28.1
NAG	30.6

The design of the site layout for Emba Derho is shown in Figure 16.7.

Figure 16.7: Emba Derho Mining Site Layout



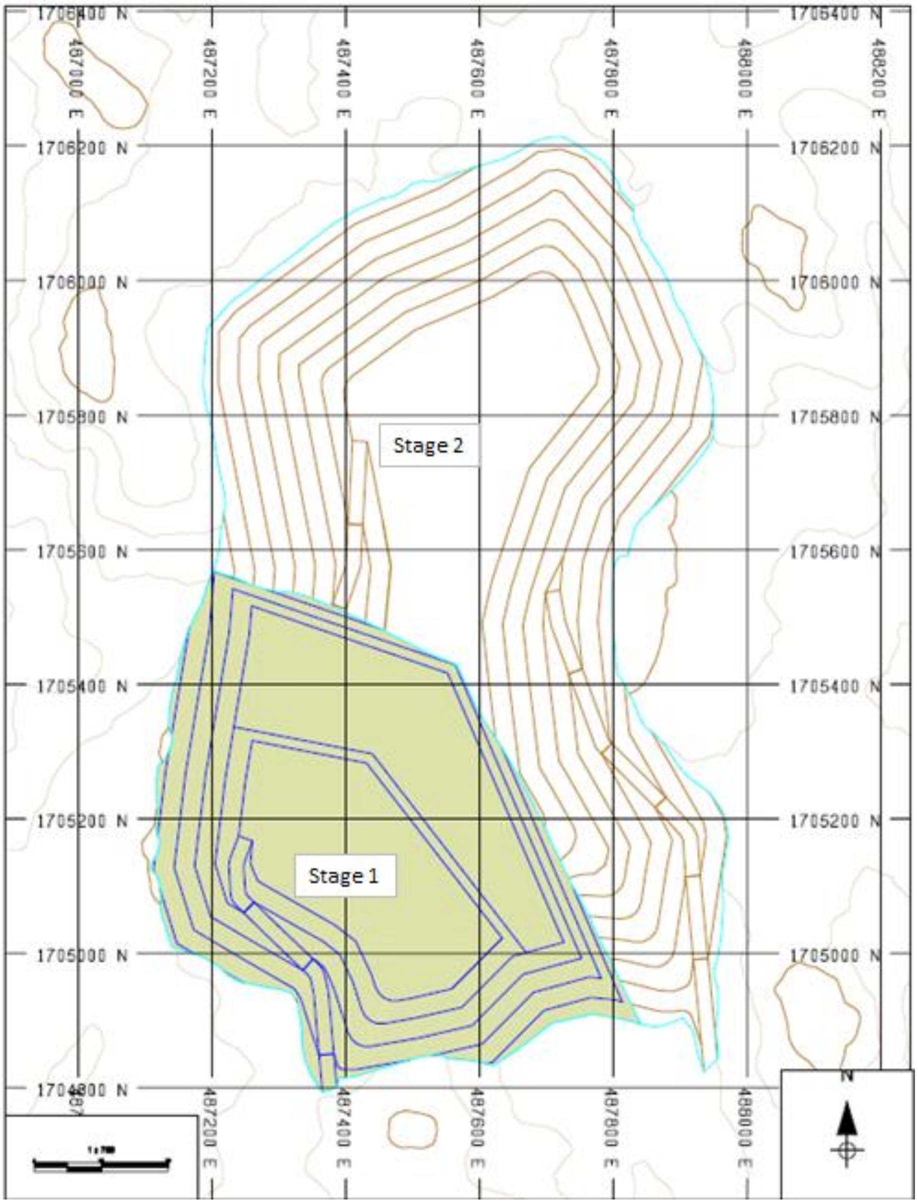
Source: Snowden

16.3.3.2 PAG Dump

The PAG dump was designed to cater for all of the PAG material and encapsulate it within a 12 m cap of NAG material. The footprint is contained within two drainage catchments that will ultimately drain towards the pit at closure, but drain into two sediment ponds during operations.

The slopes of the PAG dump were designed at 3:1 (H:V) overall slope without berms to reflect progressive shaping. A ramp of 22 m width is designed. An offset of 75 m to the pit has been provided to allow for pipelines, roads and potentially further reshaping of the waste dump at closure. The final dump is serviced by a ramp originating in the southwest and southeast corners of the dump. This is accessed by a pit exit to the west that is available for most of the mine life. A ramp from the east is used in an earlier stage to reduce haulage distance from material removed from the upper benches of ED2, where the western exit is not available. It is no longer used after it is cut-off. The dump is built in two stages (Figure 16.8).

Figure 16.8: Emba Derho PAG Dump Staging



Source: Snowden

The design capacities of the PAG dump stages are shown in Table 16.12.

Table 16.12: Emba Derho PAG Dump Design Volumes

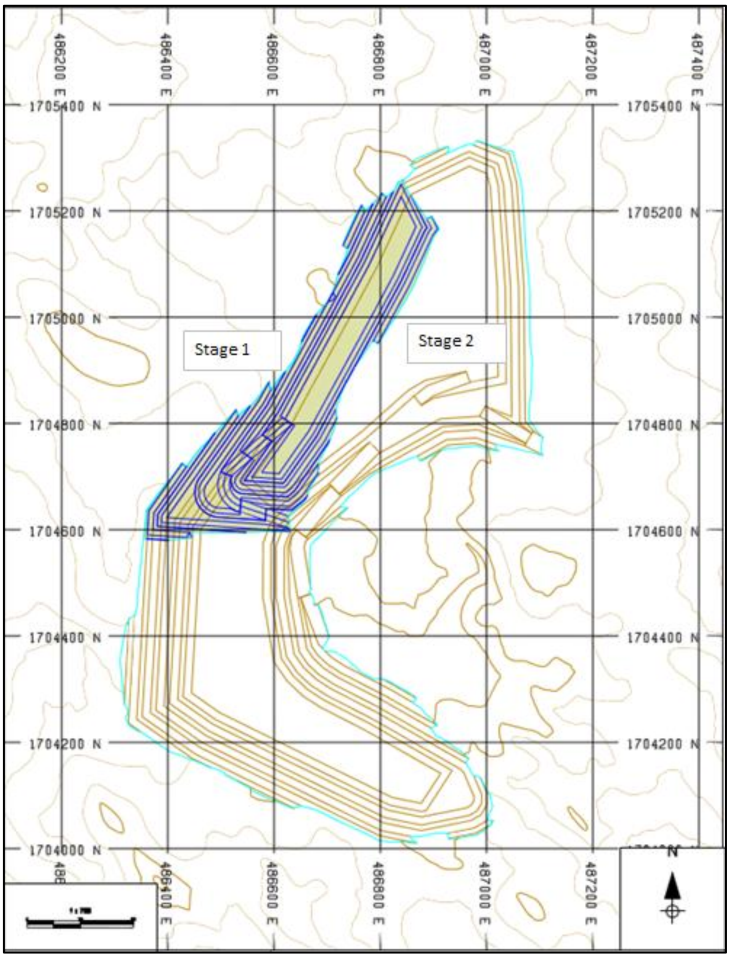
Stage	PAG (Mm³)	NAG (Mm³)	Total
Stage 1	5.6	1.5	7.1
Stage 2	22.7	8.3	31.0
Total	**28.3**	**9.8**	**38.1**

The dump will be progressively rehabilitated as NAG side capping is placed simultaneously with the PAG. The top capping will be placed as soon as the PAG has reached full height.

16.3.3.3 NAG Dump

The NAG dump has a design capacity of 11.8 Mm^3 of NAG waste (Figure 16.9). The dump is designed to an overall 2.5:1 (H:V) slope angle with 1.5:1 faces, and includes 10 m benches and 10 m berms. The road is 22 m wide. There is no capping requirement for the dump. Ramp entrances are provided from the north and south of the dump, with the south entrance servicing the west pit exits and the north servicing the northeast pit exit for ED2. This will enable ex-pit haulage to be minimised.

Figure 16.9: Emba Derho NAG Dump Staging



Source: Snowden

The dump is staged in such a manner that a visual barrier to the Medrizien village is constructed as soon as possible. This initial dump will be 500 m long and up to 30 m high. After this the dump will be constructed in a single stage, but will prioritise shortest hauls from material from relevant pit exits. The Stage 1 dump has a capacity of 1.4 Mm^3 and Stage 2 has a capacity of 10.1 Mm^3.

16.3.3.4 Primary Stockpile

The primary stockpile is designed to integrate with the ROM pad. It is constructed on a NAG waste platform to enable efficient reclaim, to store additional NAG waste, and to assist in the management of potential acid run-off from the stockpile. The design requirement for the stockpile is 12 Mt or 4.8 Mm^3 after considering swell. The stockpile is designed to an overall 2.5:1 (Horizontal : Vertical) slope angle with 1.5:1 faces, 10 m benches and 10 m berms. The stockpile ramp is 22 m wide. There is no capping requirement for the dump as it is planned to be entirely reclaimed and processed by the end of the mine life.

The dump provides adequate capacity for the primary ore and consumes 4.5 Mm^3 of NAG waste.

16.3.3.5 Platforms

NAG waste is used as fill for ROM and plant platforms. The ROM pad is designed to the 2328 mRL level and the plant to the 2280 mRL.

The total cut is 420,000 m^3 and total fill is 374,000 m^3 cut for a net cut of 46,000 m^3.

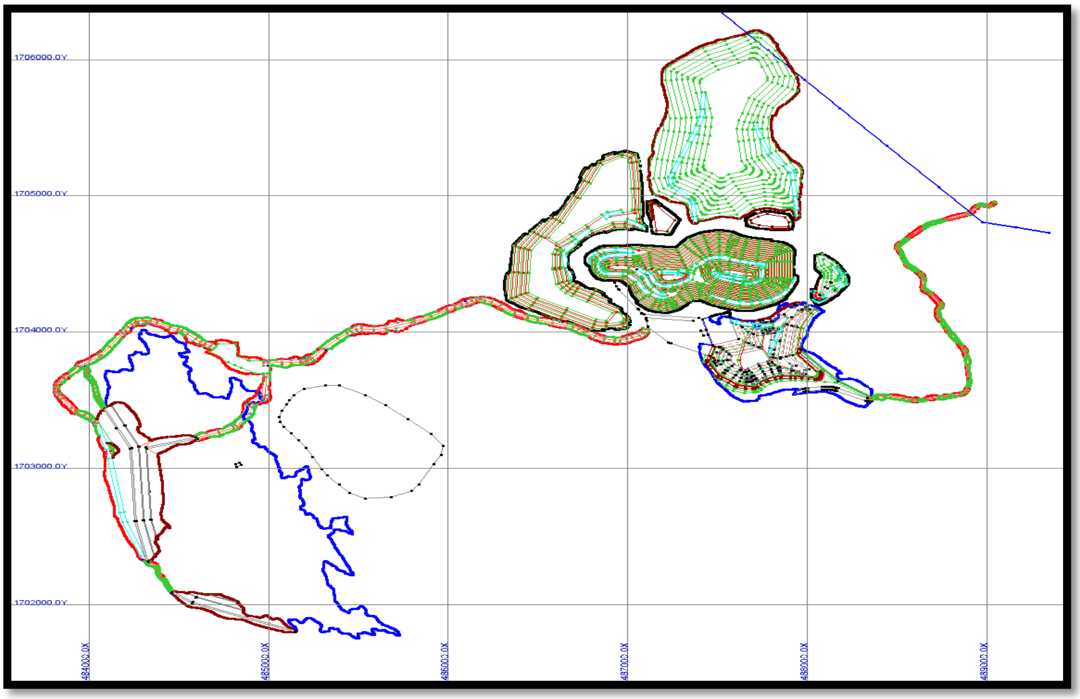
16.3.3.6 TSF Waste Requirements

The TSF requires 7.3 Mm^3 of NAG waste for construction and capping. The TSF design and construction is discussed in detail in Chapter 13.

16.3.3.7 Haulage Roads

The major designed roads are the access road to the mine from the east, and the haulage road between the pit and the TSF/heap leach area ,shown in Figure 16.10). It is assumed that a 1 m cap of NAG waste will be added to all designed roads. In total, 0.5 Mm^3 of NAG waste is required for fill with a minimal amount of cut.

Figure 16.10: Emba Derho Major Haul Roads*



**red = fill*
green = cut

Source: Snowden

16.3.3.8 Topsoil Stockpiles

A 300 mm topsoil layer for each of the disturbed waste dumps, platforms and stockpiled areas is removed and stockpiled for later use at closure. The total volume of topsoil to be stripped is 570 Km^3.

All topsoil is to be stored in one location. The design volume of the stockpile is 589 Km^3.

16.3.3.9 Overall Waste Balance

The overall waste volume balance is provided in Table 16.13.

Table 16.13: Emba Derho Overall Material Balance

Storage Area	Topsoil (Mm^3)	Low Grade Ore (Mm^3)	PAG (Mm^3)	NAG (Mm^3)	Total Design (Mm^3)
PAG dump			28.3	9.9	38.2
NAG dump				11.4	11.4
Primary stockpile		4.8		4.2	9.0
TSF construction				7.3	7.3
Roads				0.5	0.5
Topsoil stockpile	0.6				0.5
Total design	0.6	4.8	28.3	33.3	66.9
Total requirement	0.6	4.8	28.2	30.6	64.2

16.4 Debarwa Design

16.4.1 Mining Method

Both open pit and underground mining was considered at Debarwa. Preliminary economics indicated that open pit mining provided economic returns mainly due to the lower overall cost of mining; whereas, underground mining presented negative cashflows. Open pit mining supports the higher production rate required by the selected processing strategy. If additional resources are delineated at depth, then underground mining may be reconsidered. Only open pit mining of Debarwa is included in this study.

16.4.1.1 Basis

General parameters are shown in Section 16.2. In addition, there are a number of parameters specific to Debarwa. These are described in the following sections.

Geotechnical

AMC provided Snowden with a geotechnical design basis for Debarwa. The pit was divided into geotechnical sectors (Figure 16.1) with the slope parameters for each listed in Table 16.14.

Table 16.14: Debarwa Geotechnical Design Parameters

Design Sector	BFA[12]	Approximate Slope Height	Bench width	Bench height	IRA[13]	OVA[14]	Geotechnical berm or ramp
	(deg)	(m)	(m)	(m)	(deg)	(deg)	
Oxide	45	30	5.0	10	38.7	33.7	-
Fresh East	70	80-160	10.0	20	52.0	49.2	15 m every 80 m
Fresh West	75	80-160	8.5	20	58.1	55.3	15 m every 80 m

Snowden reduced the angles for each wall by 3 degrees in the pit optimization to account for ramps and geotechnical berms. For pit design, the bench face angles and berms were coded into the block model to ensure correct parameters were adhered to.

Mining

Snowden applied a cost of $2.31/t mined at surface (approx. 1,900 mRL) plus an incremental cost of $0.01/t per metre below the surface for pit optimization. This accounts for additional haulage cost.

[12] BFA = Bench face angle
[13] IRA = inter-ramp angle, measured from toe to toe
[14] OVA = overall wall angle, measured from toe to crest

Processing

Processing parameters for Debarwa, including recoveries and processing costs, are shown in Table 16.15 below.

Table 16.15: Debarwa Processing Parameters

		Cu Concentrate/Doré				Zn Concentrate	
	Processing cost ($/t ore)	Cu recovery (%)	Cu conc (%)	Au recovery (%)	Ag recovery (%)	Zn recovery (%)	Zn conc (%)
Oxide/Transition ores	11.88	-	-	60.0	21.0	-	-
DSO ore	50.00	100.0	15.5	100.0	100.0	-	-
Supergene ore	21.84	80.0	27.0	56.0	61.0	-	-
Primary ore	14.00	66.7	29.7	36.4	44.2	82.6	53.1

Ore transport

Costs of $3.25/t were added for transportation of Debarwa ore to the Emba Derho processing plant. This cost was based upon a quote supplied by the transport company Bukkehave.

Mining Block Model

The Debarwa resource assessment was completed by AMC on behalf of SGC for the Debarwa Feasibility Study in August 2011. A summary of the resource is provided in Table 16.16. This model was supplied to Snowden in the form of a Datamine model (debresmd.dm).

The Debarwa resource model is based on a parent cell size of 5 mE by 10 mN by 5 mRL with a smallest subcell of 0.5 mE by 0.5 mN by 0.5 mRL. To account for dilution and mining recovery factors associated with the selectivity of the assumed mining equipment, the model was re-blocked to the parent cell size (5 mE by 10 mN by 5 mRL) for mine planning work. A reconciliation of mass and grade is shown in Table 16.17. This generally shows that the re-blocking process has increased mass above the nominal mining cut-off grades by between 1% and 20% and reduced grade by between 13% and 22%. This result is as expected for a deposit containing narrow areas with potential mill feed of less than 2 m width. It is assumed that this reblocked model has accounted for mining recovery and dilution and thus no further factors (for mining) will be applied.

Table 16.16: Debarwa Resource Summary

Resource Classification/ Material Type		Cut-Off	Tonnes (kt)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)
Measured	Oxide	Au 0.5 g/t	3	0.01	0.01	1.03	4
	Transition	Au 0.5 g/t	103	0.07	0.03	4.59	90
	Supergene	Cu 0.5%	321	11.63	0.07	2.58	65
	Primary Cu	Cu 0.5%	7	2.39	5.97	1.32	26
	Primary Zn	Zn 2.0% (Cu<0.5%)					
	Total		**434**				
Indicated	Oxide	Au 0.5 g/t	368	0.06	0.05	1.47	6
	Transition	Au 0.5 g/t	617	0.08	0.06	2.55	17
	Supergene	Cu 0.5%	1,068	3.21	0.08	1.04	23
	Primary Cu	Cu 0.5%	767	2.34	3.90	1.30	29
	Primary Zn	Zn 2.0% (Cu<0.5%)	58	0.36	3.05	1.24	22
	Total		**2 878**				
Measured and Indicated	Oxide	Au 0.5 g/t	371	0.06	0.04	1.47	6
	Transition	Au 0.5 g/t	720	0.08	0.05	2.85	27
	Supergene	Cu 0.5%	1,389	5.15	0.07	1.40	33
	Primary Cu	Cu 0.5%	774	2.34	3.92	1.30	29
	Primary Zn	Zn 2.0% (Cu<0.5%)	58	0.36	3.05	1.24	22
	Total		**3,312**				
Inferred	Oxide	Au 0.5 g/t	239	0.1	0.1	1.1	5
	Transition	Au 0.5 g/t	138	0.1	0.0	1.4	22
	Supergene	Cu 0.5%	144	2.7	0.1	0.6	31
	Primary Cu	Cu 0.5%	154	1.2	3.6	2.6	41
	Primary Zn	Zn 2.0% (Cu<0.5%)	6	0.4	3.3	1.1	21
	Total		**681**				

Table 16.17: Debarwa Reblocked Model Comparison[15]

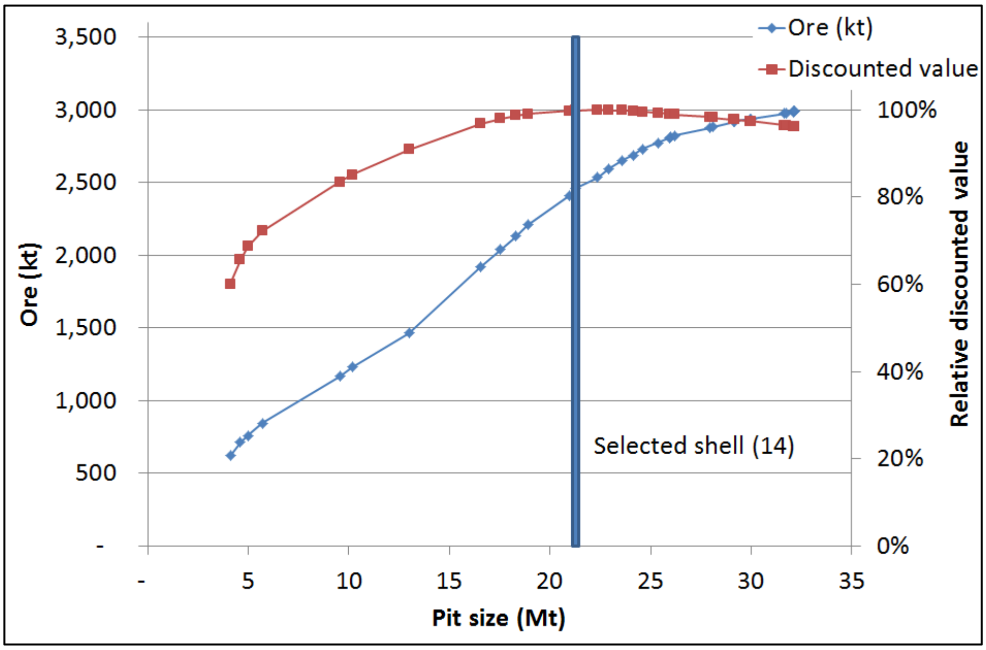
Rock Type	Cut-off	Mass (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Original						
OX	0.8 g/t Au	226	2.01	7.62	0.08	0.04
TR	0.8 g/t Au	631	3.16	29.75	0.07	0.05
SG	0.5% Cu	1,389	1.4	32.98	5.15	0.07
PR	0.5% Cu	774	1.3	29.16	2.34	3.92
Total		**2,905**				
5x10x5 Reblock						
OX	0.8 g/t Au	224	1.77	6.7	0.08	0.05
TR	0.8 g/t Au	649	2.68	24.92	0.11	0.04
SG	0.5% Cu	1,775	1.2	27.36	4.03	0.12
PR	0.5% Cu	944	0.99	22.78	1.89	2.93
Total		**3,592**				
Reconciliation						
OX	0.8 g/t Au	99%	88%	88%	100%	125%
TR	0.8 g/t Au	103%	85%	84%	157%	80%
SG	0.5% Cu	128%	86%	83%	78%	171%
PR	0.5% Cu	122%	76%	78%	81%	75%
Total		**124%**				

[15] For Measured and Indicated material only

16.4.1.2 Results

A number of pit shells were generated, parameterized by revenue, with factors between 15% and 200% of the base case revenue assumptions. The discounted value for each of the pit shells was then determined at the base case prices, assuming cash flows are spread evenly over the project life, with life determined by the project life at production rate of 1 Mtpa. The results are shown in Figure 16.11. Pit shell 14 (based on a revenue factor of 80%) was selected for design as this maximizes relative discounted value.

Figure 16.11: Debarwa Pit Optimization Results



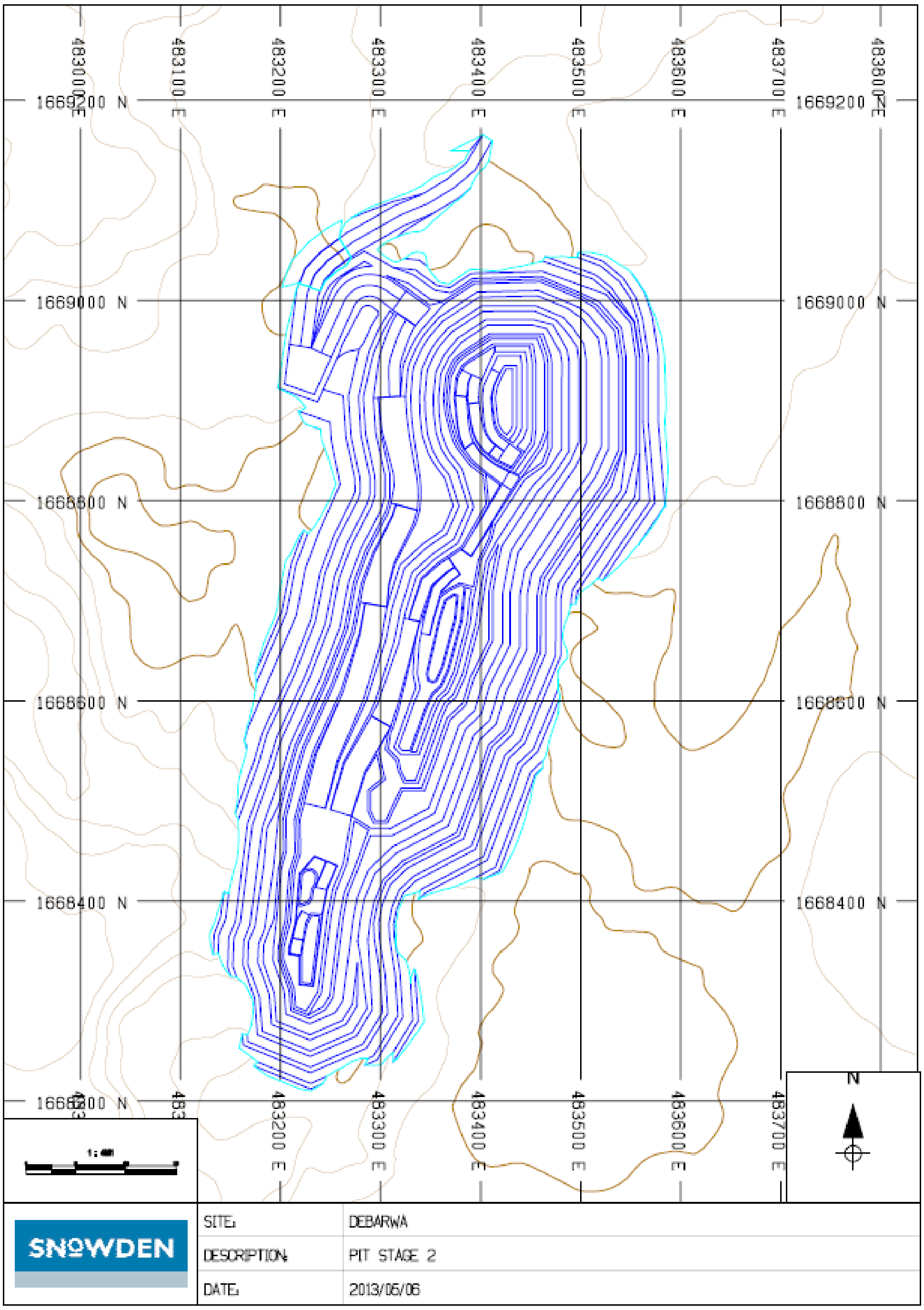
Source: Snowden

16.4.2 Pit Design

The design basis for bench face angles and berm widths are shown in Haulage roads of 22 m were designed for dual lane access, and 11 m for single lane access at the base of the pit, noting that smaller trucks are to be used on these roads. Ramps were used to satisfy the geotechnical berm requirements where possible, with the exception of the northeast of the pit where a 15 m stability berm has been added.

The final pit design is shown in Figure 16.12. The pit is 800 m long by 300 m wide and ranges in depth from 80 m in the south to 180 m in the north.

Figure 16.12: Debarwa Final Pit Design



Source: Snowden

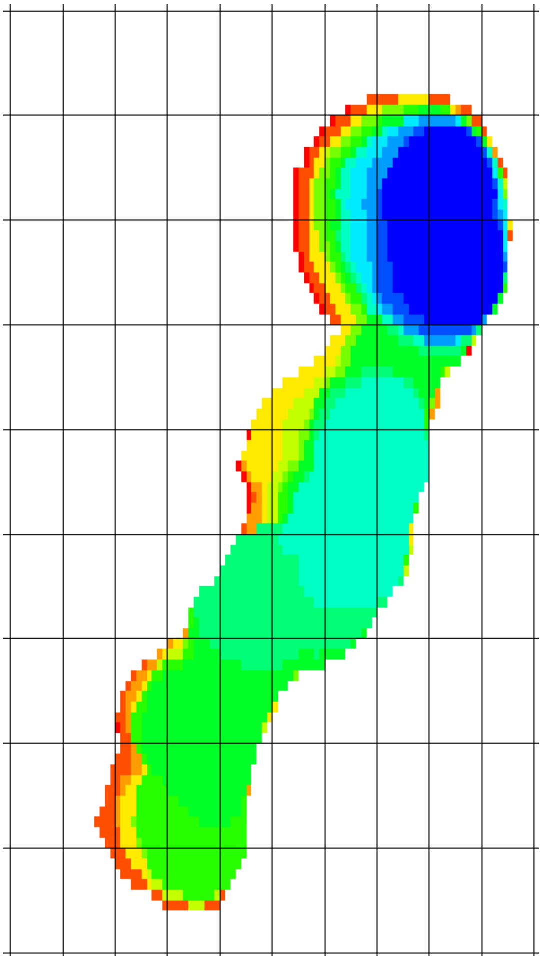
The reconciliation of volumes and value between the pit shell selected for design and the design itself is shown in Table 16.18.

Table 16.18: Debarwa Design Inventory Comparison[16,]

	Design	Pit shell	Difference (%)
Total pit (Mt)	24.3	22.4	8%
Waste (Mt)	21.7	19.9	9%
Gold ore (kt)	704	714	(1%)
Supergene/DSO ore (kt)	1,312	1,312	0%
Primary ore (kt)	598	502	19%
Strip ratio (w:o)	8.3	7.8	6%
Cash flow ($M)	267	276	(3%)

Analysis of the lower revenue factor pit shells (Figure 16.13) reveals that the pit tends to be driven on deep, higher grade copper ores deep in the north of the pit. Therefore this area was targeted for a starter pit (known as DW1) which aims to extract as much of the DSO material as possible and as little other material (waste and ore as possible) as the main processing plant will not be available when this is mined. The design of this stage is shown in Figure 16.14.

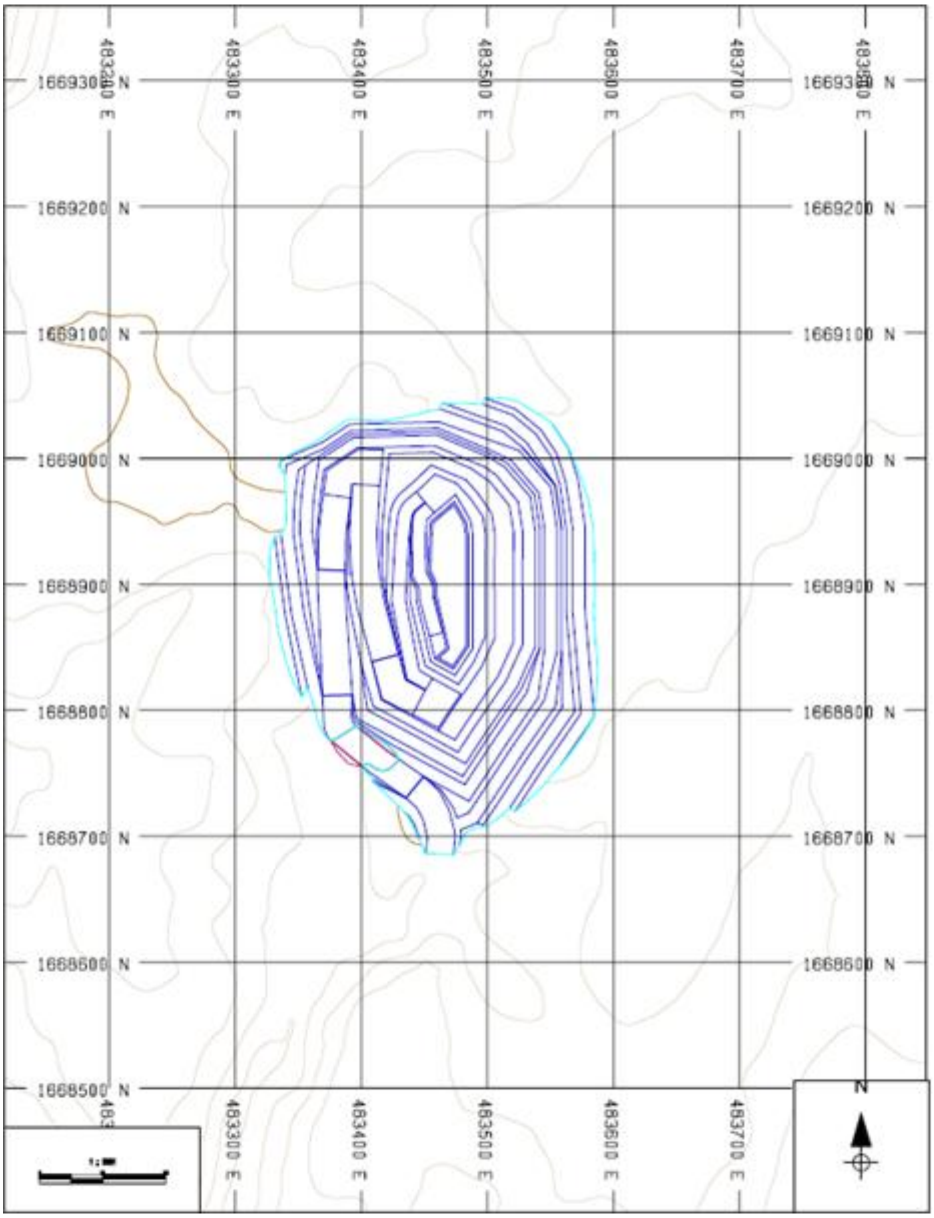
Figure 16.13: Debarwa Pit Shell Progression[17] (in plan view)



Source: Snowden

[16] "Ore" is reported as tonnes above a marginal cut-off grade. The final inventory is reduced due to considerations for blending, process timing, and stockpile size limitations. The material that is rejected is low grade.
[17] Cool colours, lower revenue factor

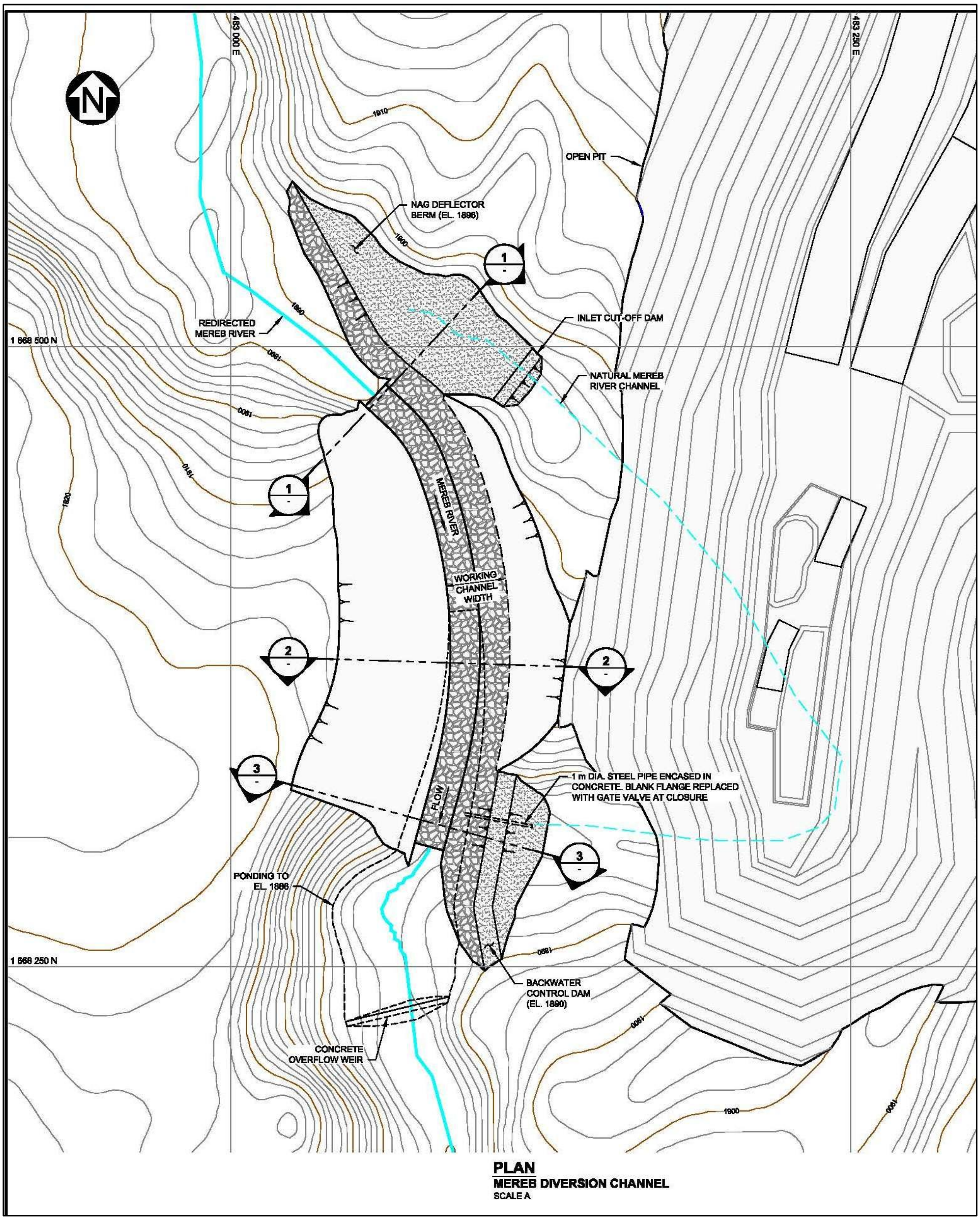
Figure 16.14: Debarwa Stage 1 Design



Source: Snowden

An additional stage for Debarwa has been added to reflect the mining out of the Mereb river diversion (DWR). This is shown in Figure 16.15.

Figure 16.15 Debarwa Mereb River Diversion



Source: KP

The final mining inventory for Debarwa, after considering additional factors such as stockpiling, scheduling and process plant availability is shown in Table 16.19.

Table 16.19: Debarwa Ore Inventory

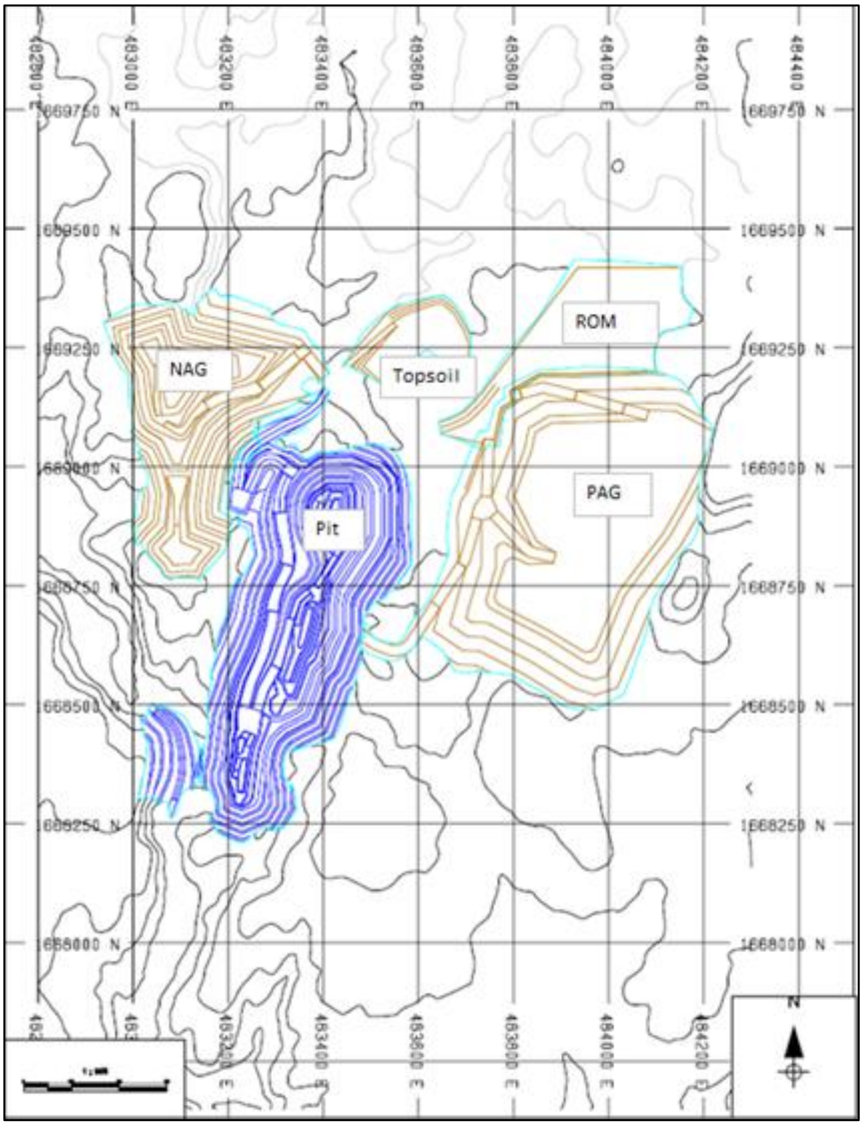
	DW1	DW2	DWR	Total
Total (kt)	5,849	18,501	404	24,753
Waste (kt)	5,030	16,802	404	22,236
Ore (kt)	819	1,698		2,517
Strip ratio (w:o)	6.1	9.9		8.8
Gold ore (kt)	320	366		686
Au (g/t)	2.7	2.3		2.5
Ag (g/t)	34.4	15.1		24.1
DS ore (kt)	116			116
Cu (%)	15.6			15.6
Au (g/t)	3.0			3.0
Ag (g/t)	76.7			76.7
Supergene (kt)	380	815		1,195
Cu (%)	4.4	3.1		3.5
Au (g/t)	1.3	1.2		1.3
Ag (g/t)	28.9	28.1		28.4
Primary (kt)	3	517		520
Cu (%)	2.3	1.9		1.9
Zn (%)	6.0	4.0		4.0
Au (g/t)	0.8	1.1		1.1
Ag (g/t)	17.4	25.4		25.3

16.4.3 Mining Site Layout Design

16.4.3.1 Overall

The mining layout was designed to manage potentially acid generation material (PAG), and provide sufficient capacity to store non-acid generating material (NAG), and ROM stockpile material. Additionally, designs were completed to store 300 mm of topsoil from each of the dumps and storages (500 mm for PAG dump). Incorporating a 35% swell factor the required storage volumes for each storage type is shown in Table 16.11. The design of the site layout for Debarwa is shown in Figure 16.16.

Figure 16.16: Debarwa Mining Site Layout



Source: Snowden

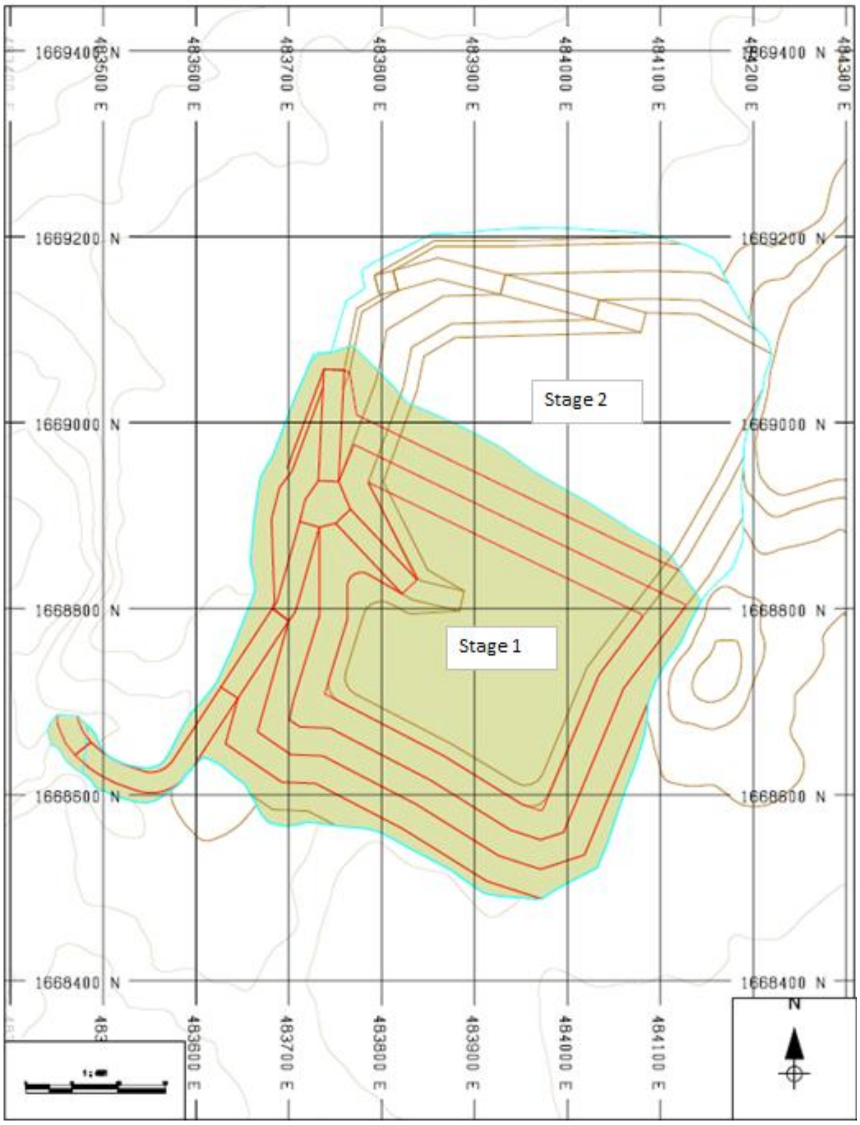
16.4.3.2 PAG Dump

The PAG dump was designed to cater for all of the PAG material and encapsulate it within a 12 m cap of NAG material. The footprint is contained within drainage catchments that will ultimately drain towards the pit at closure, but drains into a sediment pond during operations.

The slopes of the PAG dump were designed at 3:1 (H:V) overall slopes without berms to reflect progressive shaping. A ramp of 22 m width is included in the design. An offset of 100 m to the pit has been provided to allow for pipelines, roads and potentially further reshaping of the waste dump at closure. The final dump is serviced by a ramp originating from the north which splits into ramps travelling east and south. There is also a ramp from the south that is designed to provide access for the initial dump stage. As soon as material exits from the northwest, the south ramp becomes redundant, apart from access for rehabilitation efforts.

The dump is built in two stages. The first stage places material in low lying areas to the south, where access is provided through a low point in the topography. This stage is built to full height and capped, followed by filling of the north stage of the dump, when ramps are provided from the northwest. The staging is shown in Figure 16.17.

Figure 16.17: Debarwa PAG Dump Staging



Source: Snowden

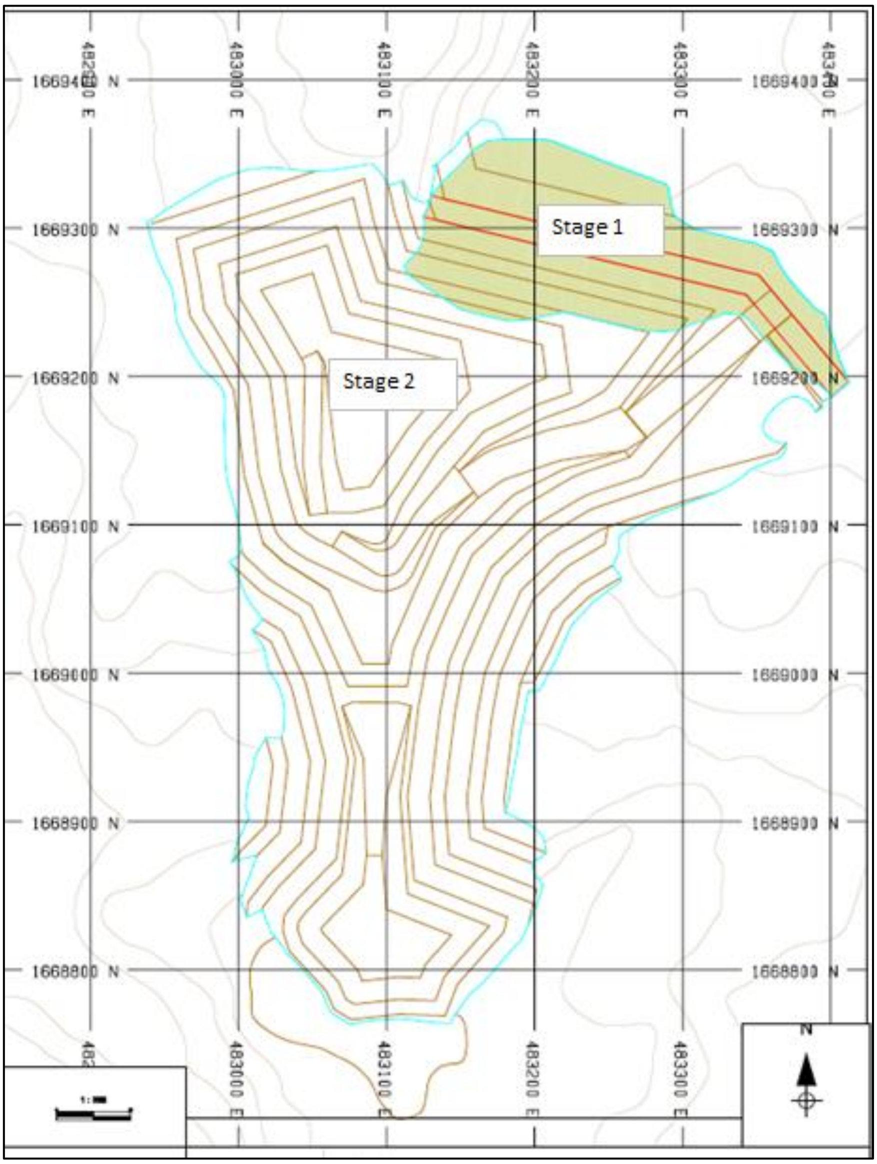
The PAG dump stores 4.4 Mm3 of PAG and includes 2.8 Mm3 of NAG capping. The dump will be progressively rehabilitated as NAG side capping is placed simultaneously with the PAG. The top capping will be placed as soon as the PAG has reached full height.

16.4.3.3 NAG dump

The NAG dump is designed to store 3.3 Mm^3 of NAG waste. The dump is designed to an overall 2.5:1 (H:V) slope angle with 1.5:1 faces, 10 m benches and 10 m berms. The haul road is 22 m wide. There is no capping requirement for the dump. The Ramp entrance integrates with the Stage 2 pit exit in the northwest.

The dump is staged to enable the dam for the mine water pond to be constructed as soon as possible (Figure 16.18, Stage 1). This starter dump will be approximately 200 m long and up to 30 m high. After this the dump will be constructed in a single stage (Stage 2).

Figure 16.18: Debarwa NAG Dump Staging



Source: Snowden

The Stage 1 NAG dump stores 0.2 Mm^3 and Stage 2 NAG dump stores 3.2 Mm^3.

16.4.3.4 Stockpile

The stockpile is provided to store material prior to being transported to Emba Derho by highway trucks. It is placed on top of the ROM and facilities platform. The design requirement for the stockpile is 700 kt or 300 000 m^3 after considering swell. The stockpile is designed to an overall 2.5:1 (H:V) slope angle with 1.5:1 faces, 10 m benches and 10 m berms. The road is 22 m wide. There is no capping requirement for the stockpile as it is planned to be sent to Emba Derho for processing.

16.4.3.5 Platforms

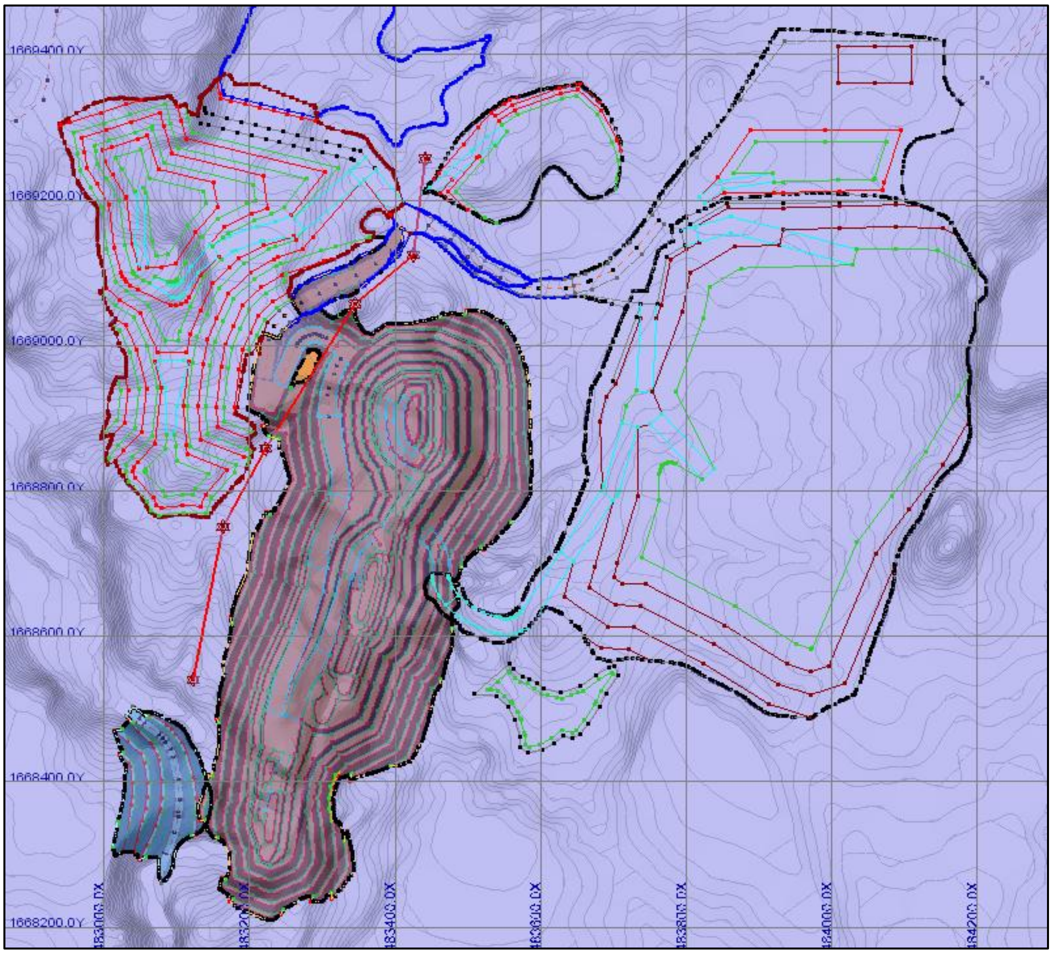
NAG waste is used as fill for ROM and the mine facilities platform. Each platform is designed with 2.5:1 (H:V) slopes for fill. The ROM pad is designed to 1 938 mRL.

The platforms provide storage for 0.3 Mm^3 of NAG waste.

16.4.3.6 Haulage roads

Very minor haulage roads are required for Debarwa to connect the pit exits with the ROM pad and waste dumps. These are shown in figure Figure 16.19. It is assumed that 1 m of waste will be added to cap all main roads. A total of 0.1 Mm^3 of NAG waste is required for this purpose.

Figure 16.19: Debarwa Layout with Haulage Roads



Source: Snowden

16.4.3.7 Topsoil stockpiles

A 300 mm topsoil layer for each of the disturbed waste dumps (500 mm for the PAG dump), and ROM platform is stockpiled for later use in closure. In total, 230 000 m^3 of topsoil is required to be stored.

A single topsoil stockpile was designed to the north of the pit to provide storage capacity for this material and to minimise haulage distances. The design volume is 230 000 m^3.

16.4.3.8 Overall waste balance

The overall volume balance is provided in Table 16.20. Expansion can be achieved through adding a lift to the PAG dump.

Table 16.20: Debarwa Overall Material Balance

Storage area	Topsoil (Mm³)	Low Grade Ore (Mm³)	PAG (Mm³)	NAG (Mm³)	Total Design (Mm³)
PAG dump				2.8	3.4
NAG dump			4.4	3.3	7.7
ROM pad platform and facilities		0.3		0.2	0.5
Roads				0.1	0.1
Topsoil	0.2				0.2
Total design	**0.2**	**0.3**	**4.4**	**6.4**	**11.9**

16.5 Gupo Design

16.5.1 Mining Method

The Gupo Indicated Resource is within 50 m of the surface and therefore only open pit mining was considered. There are some Inferred extensions at depth and along strike which may be considered for mining if additional resources are delineated.

16.5.1.1 Basis

General parameters are shown in Section 16.2. In addition, there are a number of parameters specific to Gupo. These are described in the following sections.

Geotechnical

a guidance of 57 degree overall slope angles were provided for Gupo for all material/zones. A 10 m bench height, 82 degree batter angle and 5 m berm were applied.

Mining

A cost of $3/t mined for pit optimization was applied, but with no depth increment as the pit is shallow.

Processing

Processing parameters for Gupo including recoveries and processing costs are shown in Table 16.21.

Table 16.21: Gupo Processing Parameters

		Copper concentrate/Dore				Zinc concentrate	
	Processing cost ($/t ore)	Cu recovery (%)	Cu conc (%)	Au recovery (%)	Ag recovery (%)	Zn recovery (%)	Zn conc (%)
Oxide/Transition	11.88	-	-	55.0	-	-	-

Ore Transport

Costs of $1.15/t were added for transportation of Gupo ore to the Emba Derho processing plant. This cost was based upon a quote supplied by Bukkehave for PFS quantities.

Mining Block Model

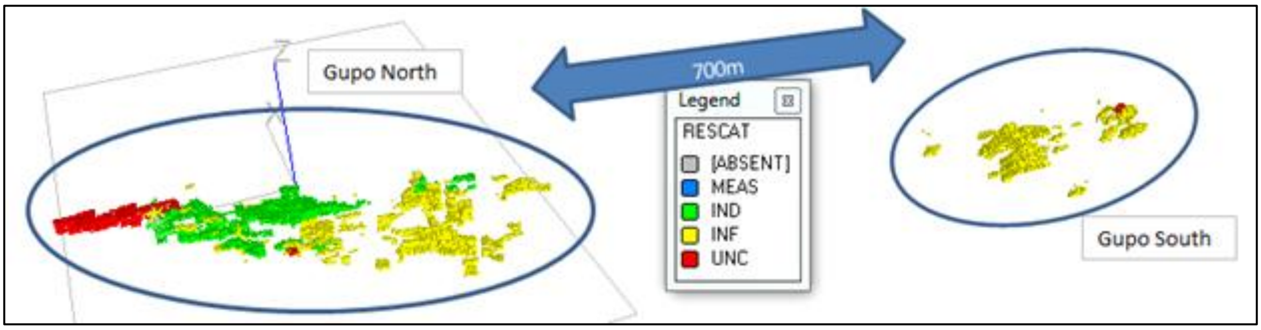
The Gupo resource estimate was completed in March 2012. A summary of the resource estimate is provided in Table 16.22. This model was supplied in the form of a comma separated value file (Gupo_NEW_March_2012_output.csv).

Table 16.22: Gupo Resource Summary

		Copper concentrate/Dore				Zinc concentrate	
	Processing cost ($/t ore)	Cu recovery (%)	Cu conc (%)	Au recovery (%)	Ag recovery (%)	Zn recovery (%)	Zn conc (%)
Oxide/Transition	11.88	-	-	55.0	-	-	-

A view of the orebody is shown in Figure 16.20. The orebody consists of two main areas, Gupo North and Gupo South, separated by approximately 700 m. Gupo South is currently wholly classified as Inferred. Gupo North includes Indicated and Inferred Resources.

Figure 16.20: View of Gupo Resource Looking East



Source: Snowden

The Gupo resource model is based on a parent cell size of 15 mE by 15 mN by 5 mRL with a smallest subcell of 5 mE by 5 mN by 2.5 mRL. To account for dilution and mining recovery factors associated with the selectivity of the assumed mining equipment, the model was re-blocked to the cell size (5 mE by 10 mN by 5 mRL) for mine planning work. A reconciliation of mass and grade is shown in Table 16.23. This shows that the re-blocking process has reduced the mass above the nominal mining cut-off grades by about 6% and reduced grade by about 2% It is assumed that this reblocked model has accounted for mining recovery and dilution and ttherefore no further factors (for mining) will be applied.

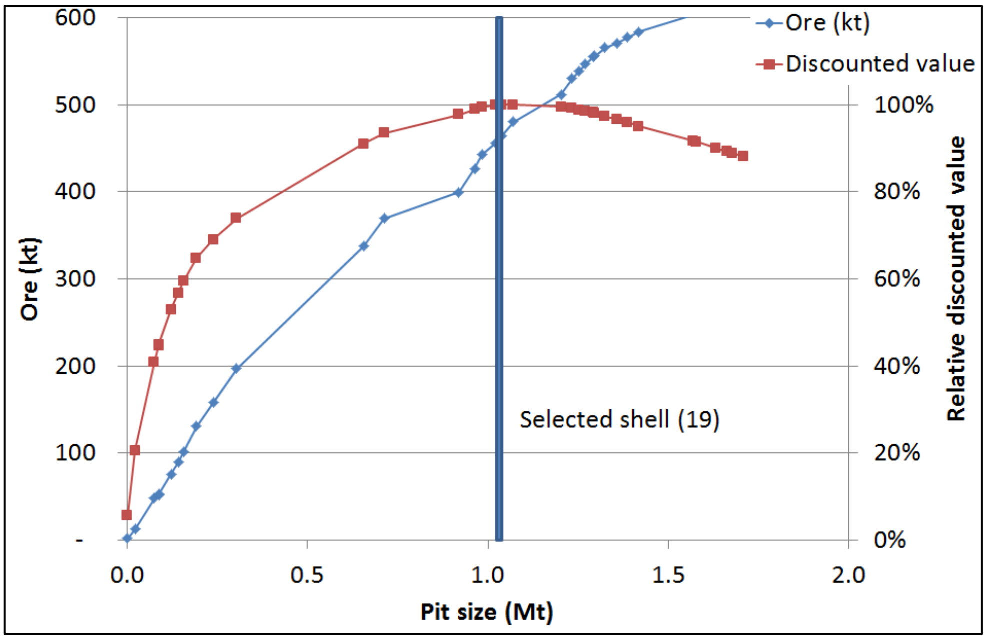
Table 16.23: Gupo Reblocked Model Comparison (for Indicated Resources only)

Rock Type	Cut-off	Mass (kt)	Au (g/t)
Original			
Oxide	0.8 g/t Au	432	1.97
Sulfide	0.8 g/t Au	235	1.83
Total		**667**	**1.92**
5 x 5 x 2.5 Reblock			
Oxide	0.8 g/t Au	419	1.93
Sulfide	0.8 g/t Au	208	1.81
Total		**627**	**1.89**
Reconciliation			
Oxide	0.8 g/t Au	97%	98%
Sulfide	0.8 g/t Au	89%	99%
Total		**94%**	98%

16.5.1.2 Results

A number of pit shells were generated, parameterized by revenue, with factors between 30% and 200% of the base case revenue assumptions. Snowden then estimated the discounted value each of the pit shells at the base case prices, assuming cash flows are spread evenly over the project life, with life determined by the project life at production rate of 1 Mtpa. The results are shown inFigure 16.21. Snowden selected pit shell 14 (based on a revenue factor of 100%) for design as this maximizes relative discounted value.

Figure 16.21: Gupo Pit Optimization Results



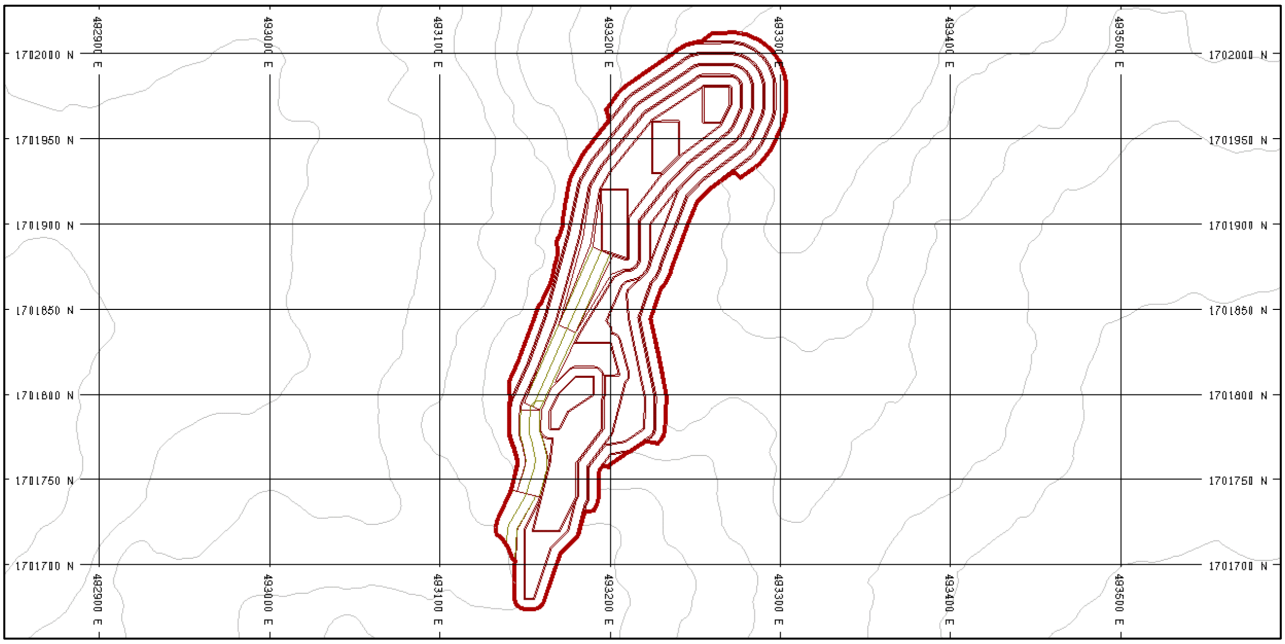
Source: Snowden

16.5.2 Pit Design

The design basis for bench face angles and berm widths are shown in Table 16.7. Haulage roads of 11 m were design for single lane access to the base of the pit, noting that smaller trucks are used for this pit. Many of the upper benches can be accessed via the topography.

The final pit design is shown in Figure 16.22. The pit is 300 m long by 60 m wide and ranges in depth from 20 m in the south to 60 m in the north.

Figure 16.22: Gupo Final Pit Design



Source: Snowden

The reconciliation of volumes and value between the pit shell selected for design and the design itself is shown in Table 16.24. A small section from the pit optimization to the south of the main pit was omitted from the comparison. This contained approximately 9 kt of material.

Table 16.24: Gupo Design Inventory Comparison

	Design	Pit shell	Difference (%)
Total pit (Mt)	1,244	1,032	21%
Waste (kt)	778	573	36%
Gold ore (kt)	466	459	2%
Strip ratio (w:o)	1.67	1.25	34%
Cash flow	7.8	9.2	(15%)

The final mining inventory for Gupo, after considering additional factors such as the cost of stockpiling, scheduling and process plant availability, is shown in Table 16.25.

Table 16.25: Gupo Inventory

	GP1
Total (Mt)	1,244
Waste (Mt)	778
Ore (Mt)	466
Strip ratio (w:o)	1.67
Gold ore (Mt)	466
Au (g/t)	1.97

16.5.3 Mining Site Layout Design

16.5.3.1 Overall

The layout for Gupo is straightforward: All ore and waste is transported to Adi Nefas directly from the pit. There are no waste dumps, stockpiles or platforms. There is also no topsoil stockpiling requirement. Incorporating a 35% swell factor, the required storage volume of NAG is 386,000m³.

16.5.3.2 Waste Dump

For the Gupo waste dump see the Adi Nefas surface.

16.5.3.3 Overall Waste Balance

The overall volume balance is provided in Table 16.13. There is sufficient space for all material. However, if the pit were to be expanded there would need to be an alternative storage area found.

Table 16.26: Gupo Overall Material Balance

Storage area	NAG (M^3)
Adi Nefas dump	297 000
Roads	89 000
Total design	386 000
Total requirement	386 000

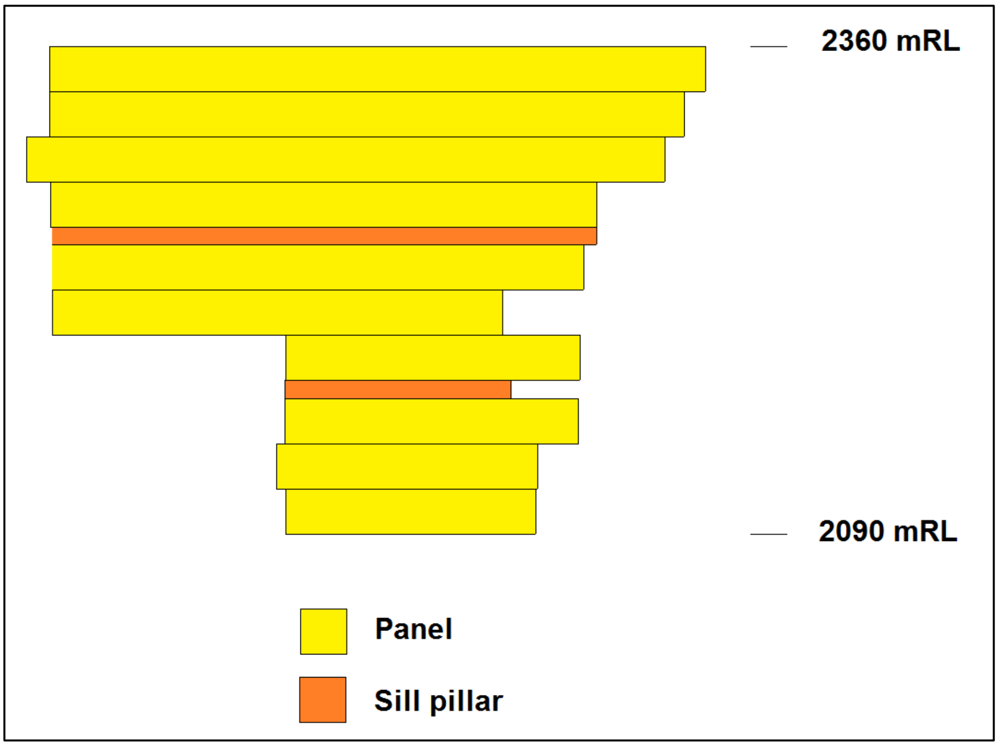
16.6 Adi Nefas Design

16.6.1 Mining Method and Layout

The Adi Nefas deposit will be mined using a bottom-up, retreat mining method with rockfill.

The mine will be divided into three panels separated by sill pillars, enabling each panel to be mined independently (Figure 16.23). Within each panel, levels will be developed at a 25 m level spacing (floor to floor).

Figure 16.23: Conceptual Panel Layout



Source: Snowden

Development drives will be developed on each level from a central access out towards the peripheries of the orebody. Each level will be serviced by an upper drive (for drilling and backfilling) and a lower drive (for loading). Once a stoping level has been completed and backfilled, the upper drive will serve as the loading drive for the stoping level above.

Each level will be divided into a number of stopes, each with a length (along strike) of no more than 45 m. Near-vertical rings will generally be drilled as downholes.

Each stope will commence with the firing of a longhole rise, which serves as a slot for subsequent firings. The stope rings will be fired in shots comprising several rings, with each firing followed by stope loading. While manual loading will be suitable for immediately after each firing, tele-remote loading will be required for the majority of stope loading.

Once firing of the stope has been completed and the stope has been loaded clean, backfilling with uncemented rock fill will commence. As development of the mine proceeds, more than one panel will operate concurrently.

16.6.2 Optimization

16.6.2.1 Basis

Based on the parameters presented in Section 16.2 and Table 16.27, Snowden's Stopesizor software was used to generate a series of potential stoping inventories for a range of NSR cut-off grades (based on practical stope shape and dimension constraints).

Table 16.27: Adi Nefas Processing Parameters

		Copper concentrate/Dore				Zinc concentrate	
	Processing cost ($/t ore)	Cu recovery (%)	Cu conc (%)	Au recovery (%)	Ag recovery (%)	Zn recovery (%)	Zn conc (%)
Primary	14.00	91.0	24.6	35.0	38.0	91.0	59.5

The total onsite operating cost for Adi Nefas was estimated to be $119.31/t based on quotations by mining contractors (Table 16.28). Consequently, the $120/t NSR cut-off grade Stopesizor inventory was selected for the purpose of mine design. Subsequent to the mine design and schedule being developed a more accurate cost model was developed. This is detailed in chapters 16 and 17.

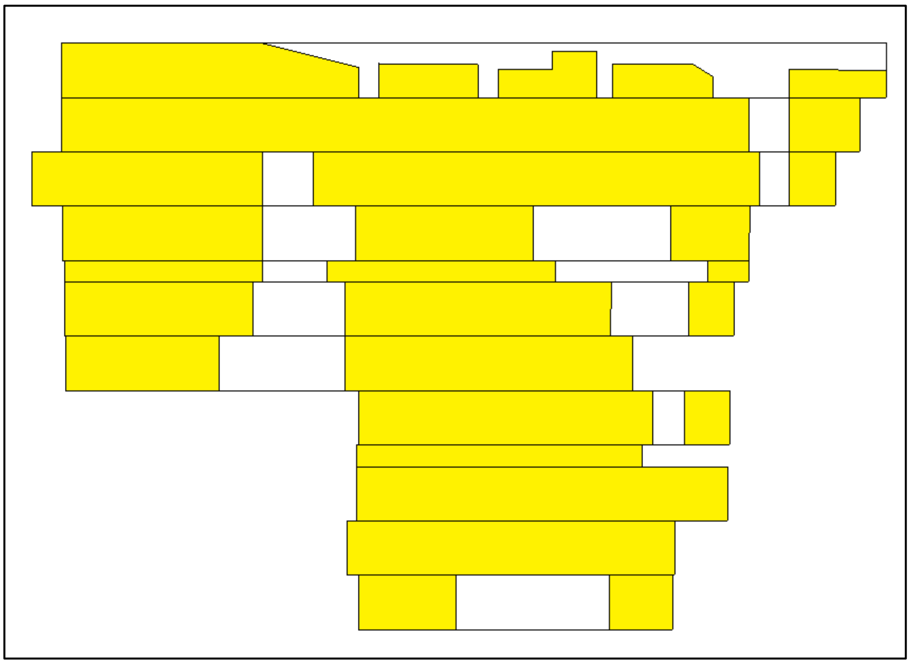
Table 16.28: Total Onsite Operating Cost Estimate for Adi Nefas Optimization

Component	Cost	Notes
	($/t)	
Mining	100.50	Based on PFS contractor quotations
Ore transport	1.15	
Processing	14.00	
G&A	2.66	
Total	119.31	

The proposed mining method does not incorporate regular, designed rib pillars, but a number of natural pillars exist in lower grade areas of the orebody (Figure 16.24). Consequently, there will be no ore losses due to rib pillars.

In those areas where there are adjacent stopes, there will be some rilling of backfill material from the filled stope into the newly fired stope. This can be managed one of two ways: accept the dilution and load more tonnes than fired at a lower grade, or reject the diluted portion and mine fewer tonnes than fired at the *in-situ* grade. For this study, the latter method was adopted.

Figure 16.24: Natural Pillars in Adi Nefas Stoping Design



Source: Snowden

Mining of the sill pillar stopes will consist of a recovery operation. Because there will be rockfill above and below the pillar stopes, it is expected that overall recovery will be low.

Table 16.29: Adi Nefas Mining Modifying Factors

	Tonnage factor	Grade factor
Lateral development	1.0	1.0
Vertical development	1.0	1.0
Backfill stopes	0.9	1.0
Open stopes	0.9	1.0
Pillar stopes	0.5	1.0

16.6.3 Mining Inventory

The mining inventory for Adi Nefas (including modifying factors) is summarised in Table 16.30. For stoping ore, a NSR cut-off grade of $120/t was applied. For development ore, a cut-off grade of 50 $/t was used.

Table 16.30: Adi Nefas Mining Inventory

	Mass	Au	Ag	Cu	Zn
	(kt)	(g/t)	(g/t)	(%)	(%)
Ore	1,681	2.82	96.53	1.56	8.19
Waste	426	-	-	-	-

16.6.4 Stope Design

Two types of stopes were designed (Figure 16.25):

- open stopes
- sill pillar stopes

Backfill stopes are the primary type of stope, and consist of 25 m high stopes which are rockfilled upon completion of mining.

Sill pillar stopes are those stopes which will be mined from the top of the stopes immediately below the sill pillars upon completion of mining of the panels above and below. These will be mined using an uphole retreat mining method, with no fill.

In order to maintain a stable hydraulic radius, backfill stopes will be mined to a maximum length of 40 m before being filled. While there are some natural rib pillars in areas of lower grade, no additional rib pillars will be left between abutting stopes.

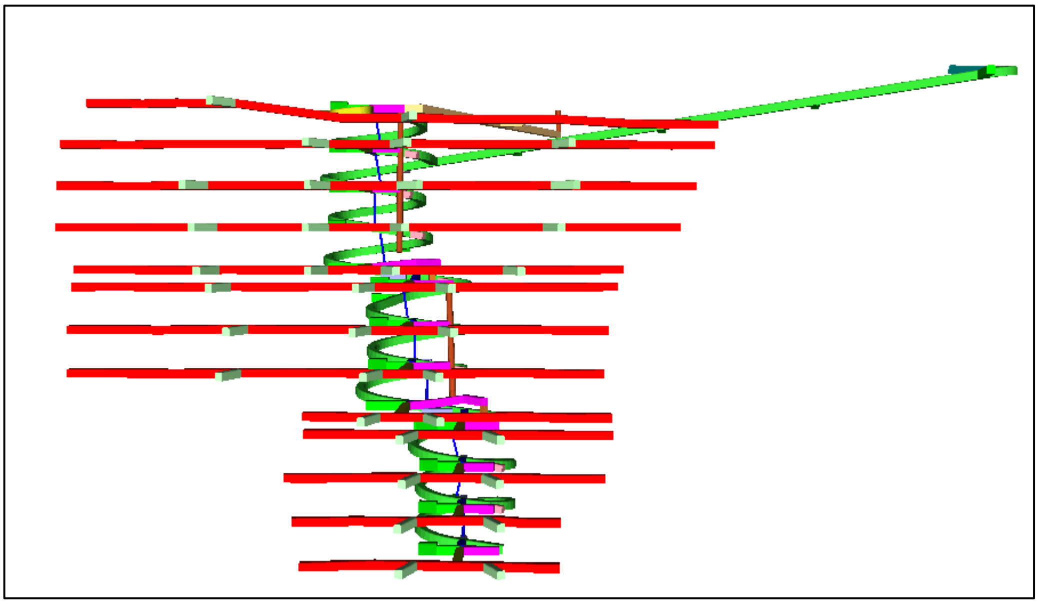
Figure 16.25: Adi Nefas Stope Designations

Source: Snowden

16.6.5 Mine Development

A view of the mine development is presented in Figure 16.26.

Figure 16.26: View of Adi Nefas Mine Development



Source: Snowden

The total development advance contained in the mine design is summarised in Table 16.31. (Note that all profiles will be arched).

Table 16.31: Adi Nefas Development Requirements

Development type	Width	Height	Advance
	(m)	(m)	(m)
Lateral Development			
Decline	5.5	5.5	2 426
Decline stockpile	5.0	5.5	315
Sump	4.5	4.5	120
Escapeway drive	4.5	4.5	229
Substation	5.0	5.0	15
Level access	5.0	5.0 to 5.5	721
Truck tipping stockpile	5.0	5.5 to 7.2	327
Return air drive	5.0	5.0	350
Ore drive	5.0	5.0	3 468
Loader stockpile	5.0	5.0	720
Return air connection	5.0	5.5	277
Refuge chamber drive	5.0	5.0	15
Total Lateral			**8 982**
Vertical Development			
Escapeway	1.5	-	235
Return air raise/rise	3.5	-	262
Total vertical			496

16.6.6 Mine Access

The mine is accessed via a 5.5 mH by 5.5 mW trucking decline developed at 1:7 gradient. Emergency access to the mine from surface can also be gained via the return air raise and return air connection.

16.6.7 Level Layout

Each level is accessed from the decline through a level access. The level access intersects the orebody centrally, and north and south ore drives are developed along the orebody strike. Loader stockpiles developed along the ore drives facilitate remote loading.

A truck tipping stockpile is located on each level access, opposite a return air drive. The return air drive provides a location for trucks to turn into, before reversing into the truck tipping stockpile.

Just off the decline and opposite the level access there is an escapeway drive, which provides access to the escapeway system.

16.6.8 Underground Materials Handling

16.6.8.1 Development Materials Handling

During decline development, stockpiles will be developed at intervals of approximately 100 m. Fired waste will be loaded from the face to the nearest stockpile. From there, it will be loaded on trucks and hauled to surface.

During of-decline development, fired ore/waste will be loaded from the face to the nearest stockpile or empty development heading. Trucks will park on the decline at the intersection of the level access, and will be loaded with the stockpiled ore/waste.

16.6.8.2 Stoping Ore Materials Handling

Following each stope firing, ore will initially be manually loaded back to the level truck tipping stockpile. When the loader has to proceed past the stope brow, loading will revert to remote or teleremote operation. Ore/waste will be remote/tele-remote loaded back to one of the level loading stockpiles, then later manually loaded to the level truck tipping stockpile.

Trucks will park on the decline at the intersection of the level access, and will be loaded with the stockpiled ore.

16.6.8.3 Backfill Materials Handling

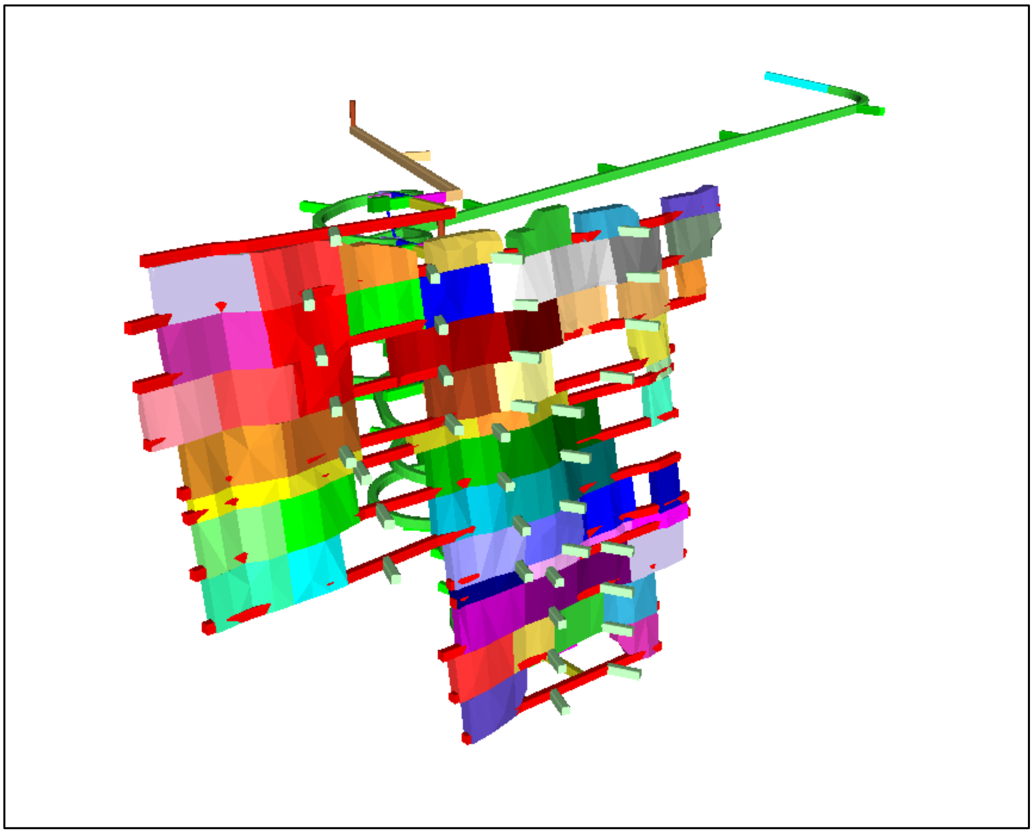
Waste rock for backfill will either be direct-hauled from mine development faces, or back-hauled underground from surface. Trucks will tip the rockfill material into truck tipping stockpiles located off the level accesses. It will then be transported and tipped into the stope void by a loader.

16.6.9 Ventilation

16.6.9.1 Primary Ventilation

The mine ventilation network is premised on an intake decline and a return air drive and raise (Figure 16.27). A primary fan station will be positioned atop the return air raise.

Figure 16.27: View of Adi Nefas Primary Ventilation Network



Source: Snowden

16.6.9.2 Level Ventilation

Each level is force-ventilated. Two 90 kW fans located in the decline will collect fresh air. It will then be ducted through a 1400 mm vent bag along the level access and the north and south ore drives to the development faces/stoping fronts. Note that one fan services the northern ore drive of a level, while the other services the southern vent drive of a level.

Used air flows back along the ore drives to the level access, and then via the return air drive into the return air raise system.

When return air reaches the top of the stoping panels at 2360 mRL, it flows along the return air connection drive to the return air raise and is exhausted to surface.

16.6.9.3 Duty Requirements

A VUMA-3D analysis was undertaken to determine the required fan duty based on the ventilation network, mining schedule and equipment fleet. The airflow requirements for diesel dilution are shown in Table 16.32.

Table 16.32: Adi Nefas Airflow Requirement for Diesel Dilution

Equipment	Quantity	Unit engine Rating	Total Engine Rating	Total Quantity Required
		(kW)	(kW)	(m^3/s)
Production fleet				
Production trucks (Sandvik TH550)	3	410	1,230	75
Production loaders (Sandvik LH 517)	2	285	570	35
Auxiliary fleet				
I.T (Cat 930H)	1	110	110	10
Secondary leakage (20% of loaders and auxiliary)				10
Primary leakage (10%)				15
Total				**145**

Table 16.33: Adi Nefas Airflow Requirement for Mining Activities

Item	No of levels	Required quantity per unit	Total quantity	Description
	(no)	(m^3/s)	(m^3/s)	
Loading levels	3	20	60	Allowance for 1 x 285 kW loader per level
Drilling levels	3	15	45	Minimum velocity 0.5 m/s
Backfill levels	1	20	20	Allowance for 1 x 285 kW loader per level
Development levels	1	20	20	Allowance for 1 x 285 kW loader per level
Primary leakage (10%)			15	
Total			**160**	

The overall ventilation requirement is dominated by mining activity criteria, and it is estimated that approximately 165 m^3/s will be required at maximum production. This is shown in Table 16.33.

16.6.9.4 Primary Fan Station

The primary fan station will consist of two axial flow fans in a bifurcated drift arrangement complete with adjustable guide vanes, inlet and discharge silencers, exhaust evase and airlock doors. The fans will operate as duty/duty units with the twin installation offering some redundancy in the case of a fan failure. The duty of the primary surface fan station is shown in Table 16.34.

Table 16.34: Adi Nefas Primary Fan Station Duty

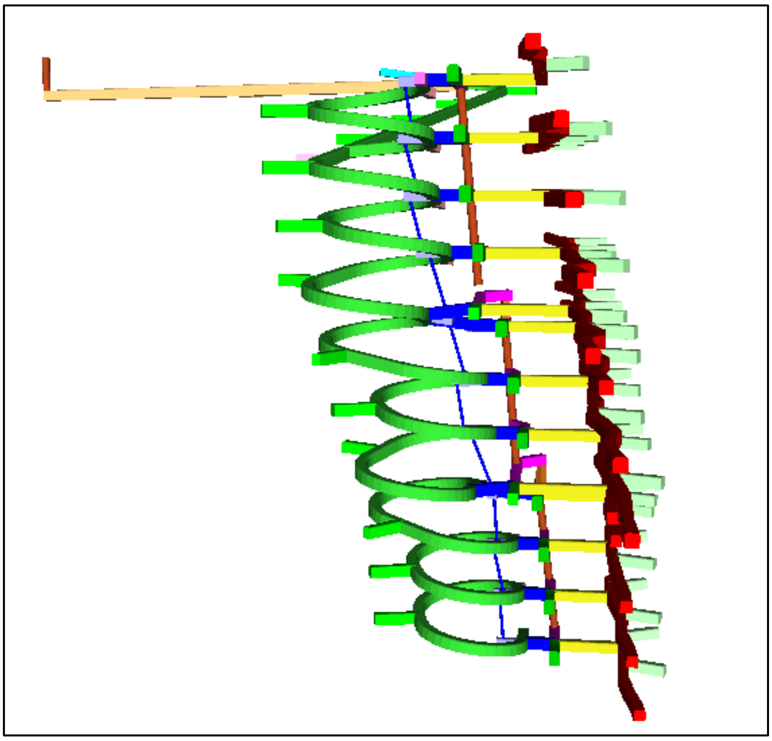
Description	Units	Duty
Airflow (total)	m^3/s	160
Airflow (per fan)	m^3/s	80
Static pressure (collar)	Pa	1,050
Absorbed power (per fan)	kW	120
Rated power (per fan)	kW	150

16.6.10 Emergency Egress

An escapeway system is accessed via escapeway drives located opposite to the entrances to the level access drives (Figure 16.28).

In the event that the decline becomes blocked or impassable, the escapeways provide a means for personnel to access the decline above or below the blockage. If the decline segment between the portal and 2330 mRL becomes blocked, personnel can use the escapeway to move up to the 2360 mRL, where they can shelter in the refuge chamber. After the main fan has been turned off, personnel can travel from the refuge chamber along the return air connection to the return air rise and finally to surface.

Figure 16.28: View showing Adi Nefas Emergency Egress System (in dark blue)

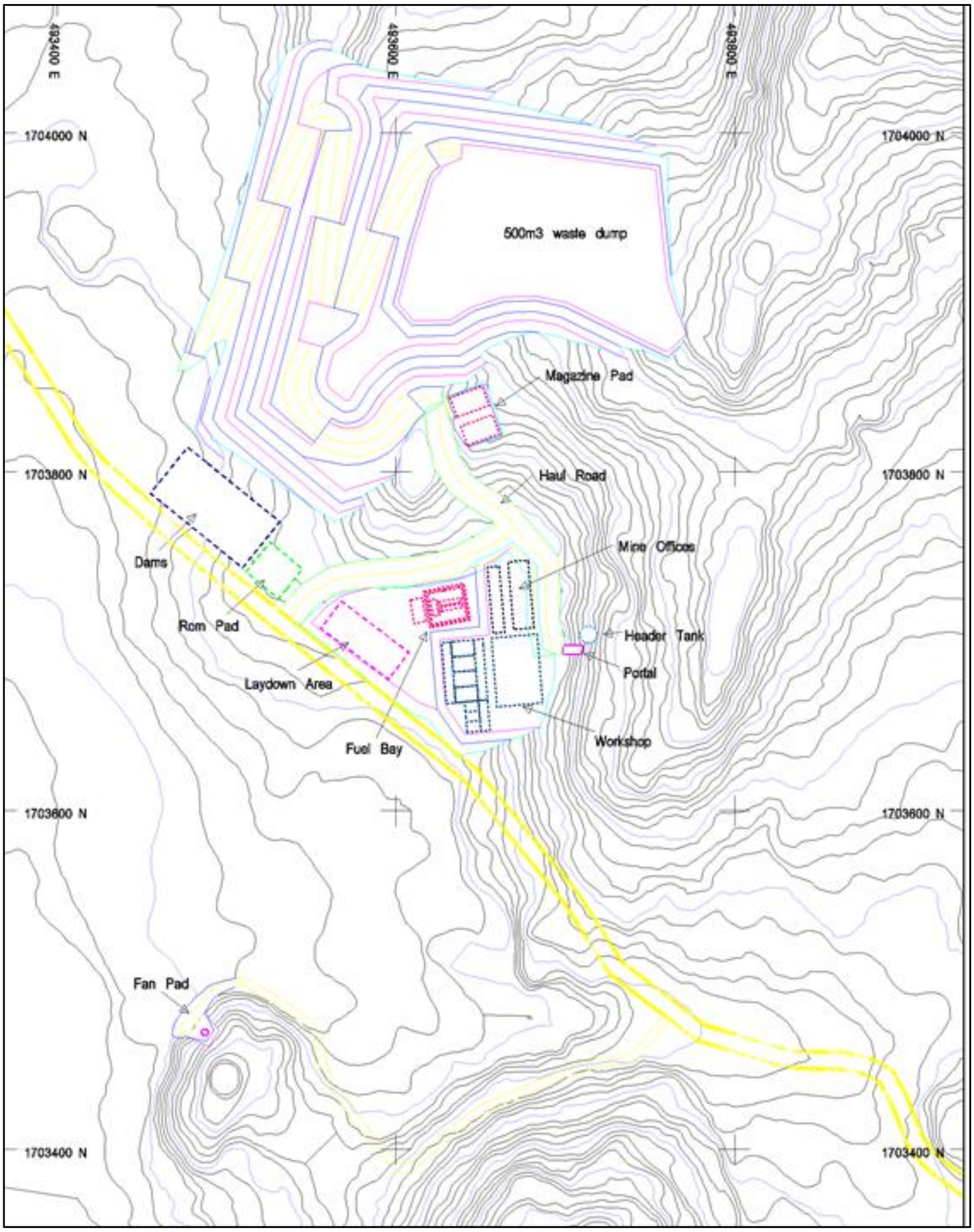


Source: Snowden

16.6.11 Adi Nefas Surface Facilities

The surface layout consists of offices, workshops, waste dumps, rom pad, fans and other associated facilities. A plan of the layout is shown in Figure 16.29 below.

Figure 16.29: Adi Nefas Surface Layout



Source: Snowden

Pads will be created for some of the structures as part of the initial development. As the decline develops, waste will be used to create pads for the remainder of the structures and the ROM pad. Excess waste will then be placed on the dedicated waste dump to the north. When underground stope backfilling commences the waste dump will be drawn down. The Adi Nefas waste dump will also receive waste from Gupo which will be used to supplement the underground backfill requirements.

16.7 Mining Schedule

16.7.1 Basis

16.7.1.1 Software

The mining schedule was completed in Snowden's Evaluator scheduling software, which is a Mixed Integer Linear programming based tool. The Adi Nefas underground schedule was completed in EPS with the results being integrated into the Evaluator schedule to ensure consistency.

16.7.1.2 Time Horizons

The production schedule was completed in quarterly increments over the life of the project. Quarter 1 commences two years before the commencement of production from the main processing facility at Emba Derho. Quarter 1 is January 2014.

16.7.1.3 Processing Constraints

A number of processing constraints were applied to the various streams (Table 16.35).

Table 16.35: Processing Constraints

Processing stream	Sources	Start quarter	Production rate (ktpa)	Ramp up	Grade constraints
Gold ore	ED oxide, DEB oxide, DEB trans, GP oxide, GP sulfide	4	1,400	No ramp up, although metal is recovered over time	-
Direct Ship ore	DEB supergene (Cu>12%)	5	240	No ramp up	-
Supergene ore	DEB supergene (Cu<12%)	9	2,000	60% production in Month 1, and 90% production in Month 2	Maximum Cu:Zn ratio of 6
Primary ore	ED primary, DEB primary, AN primary	14	4,000	Two month shutdown, followed by 70% production Month 3	Maximum Zn grade of 3.7%

16.7.1.4 Mining Constraints

A number of mining constraints were considered in the schedule:

- Maximum overall mining rate of 20 Mtpa (5 Mtpq) was arrived at in order to smooth the mining schedule and ensure sustained utilization of equipment. This applied only to the primary (larger) mining fleet (which is used in parts of Debarwa and Emba Derho).
- Narrow mining areas (mined by a smaller secondary fleet) were capped to 2.4 Mtpa (600 ktpa) in total. This mining occurs in Debarwa and Gupo.
- Maximum vertical advance rate of 60 vertical metres per year for all deposits. This advance is rarely required during the life of the project.
- At Adi Nefas the maximum single heading development rate was limited to 80 m/month and 300 m/month for multiple headings.

16.7.1.5 Other Constraints

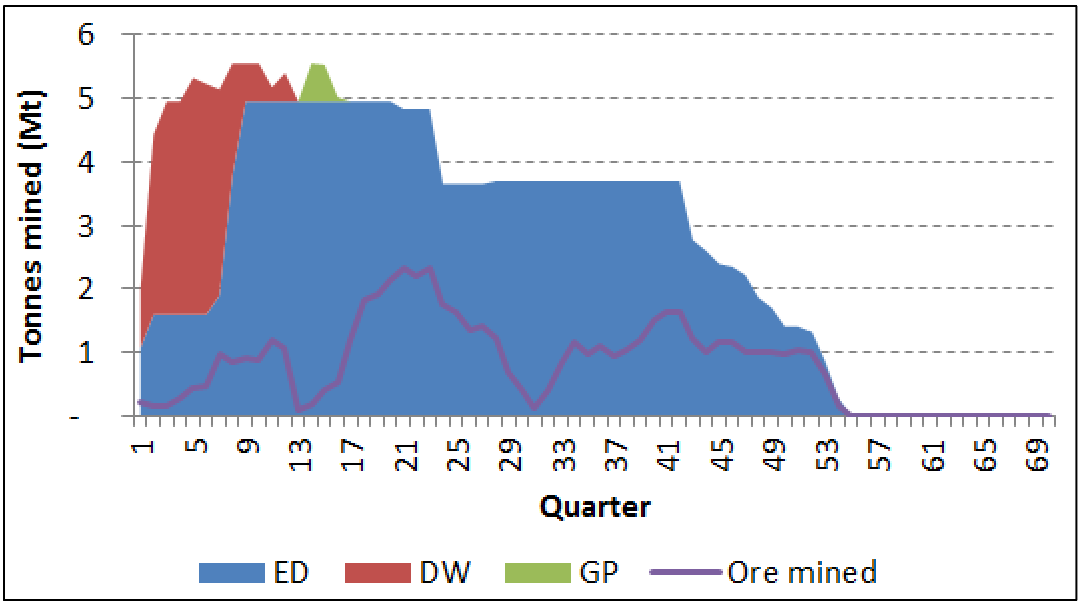
A number of other constraints were considered in the schedule:

- Maximum haulage from Debarwa to Emba Derho of 1 Mtpa (250 ktpq) with the objective of minimising the community impact of trucks on shared roads. This is not the physical capacity of these roads
- Haulage from Gupo/Adi Nefas to Emba Derho follows the completion of haulage from Debarwa to enable the loading equipment to be moved between sites
- A maximum Emba Derho stockpile size was set at 12 Mtpa

16.7.2 Overall Schedule

The overall open pit mining schedule is shown in Figure 16.30. Early mining focuses on the depletion of Debarwa, with some stripping of Emba Derho. As Debarwa is depleted, mining at Emba Derho ramps up. The smaller fleet that is used to mine narrow benches at the base of the Debarwa pit is moved to Gupo when Debarwa is completed.

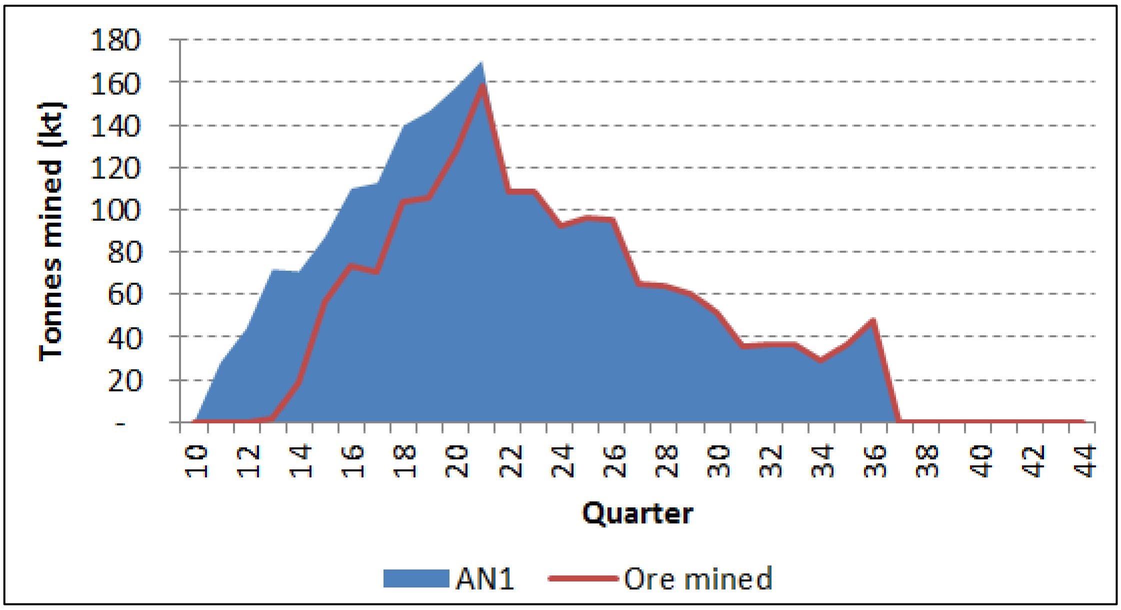
Figure 16.30: Overall Schedule



Source: Snowden

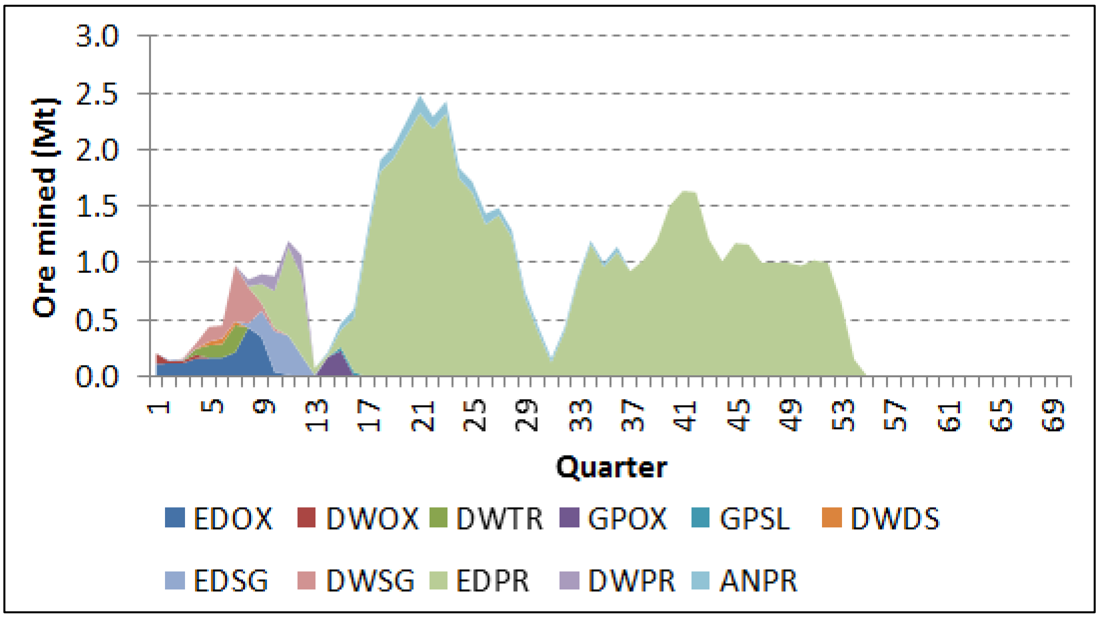
The underground schedule for Adi Nefas is shown in Figure 16.31. The first year is dominated by waste development. The remainder of the waste mining is completed by the end of the second year of production to minimise fixed costs and maximize utilisation of the development crew and equipment.

Figure 16.31: Underground Mining Schedule



The overall ore mining schedule is shown in Figure 16.32. This shows a wide range of rock types being mined over the first three years followed by almost exclusive mining of Emba Derho primary ore. The variation in ore mining is controlled through stockpiling the use of the lower grade excess.

Figure 16.32: Overall Ore Mining Schedule



A summary of the overall mining volumes is shown in Table 16.36.

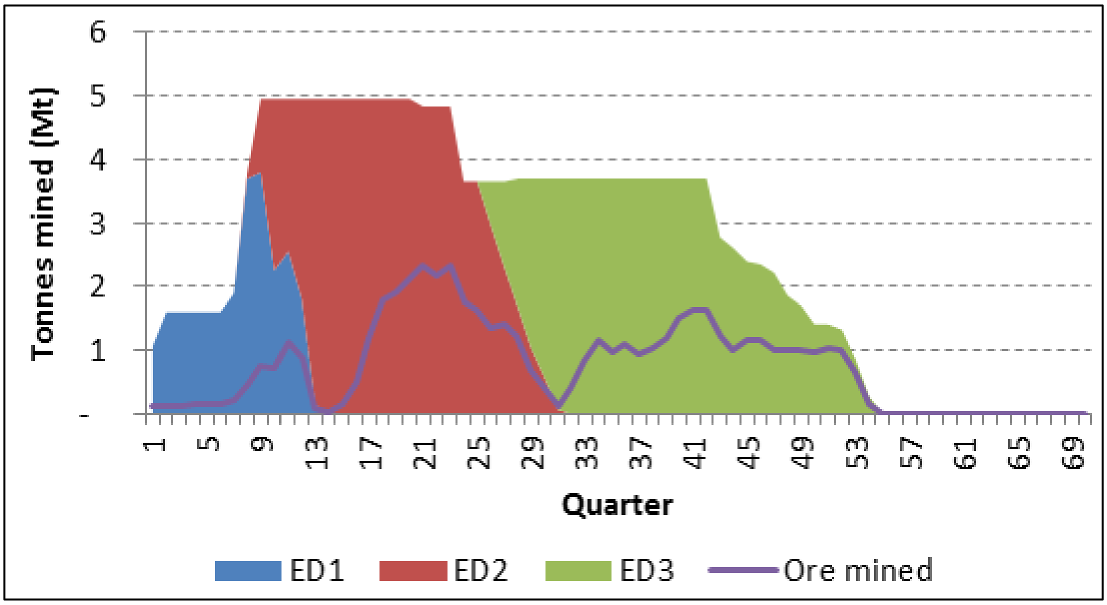
Table 16.36: Mining Schedule

	Emba Derho			Debarwa			Gupo			Adi Nefas			Total		
Year	Ore (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Waste (Mt)	Total (Mt)
1	0.6	5.3	5.9	0.3	10.2	10.5							0.7	15.6	16.3
2	1.0	7.9	8.9	1.7	10.7	12.4							2.8	18.5	21.3
3	3.5	16.3	19.8	0.6	1.3	1.9					0.1	0.1	4.1	17.7	21.8
4	0.7	19.1	19.8				0.4	0.8	1.2	0.1	0.2	0.3	1.4	20.0	21.4
5	7.0	12.8	19.8							0.5	0.1	0.6	7.5	12.9	20.4
6	8.6	9.6	18.2							0.5		0.5	9.0	9.6	18.6
7	5.7	9.0	14.7							0.3		0.3	6.0	9.0	15.0
8	1.6	13.2	14.8							0.2		0.2	1.8	13.2	15.0
9	4.1	10.7	14.8							0.1		0.1	4.3	10.7	15.0
10	4.6	10.2	14.8										4.6	10.2	14.8
11	5.5	7.3	12.8										5.5	7.3	12.8
12	4.3	4.5	8.8										4.1	4.5	8.8
13	4.0	1.8	5.8										4.0	1.8	5.8
14	0.8	0.3	1.1										0.8	0.3	1.1

16.7.3 Emba Derho Schedule

Emba Derho is mined in three stages over a 13 year period (Figure 16.33). Waste mining for each stage commences during ore mining of the previous stage. There are slight gaps in ore production which are managed through stockpile depletion.

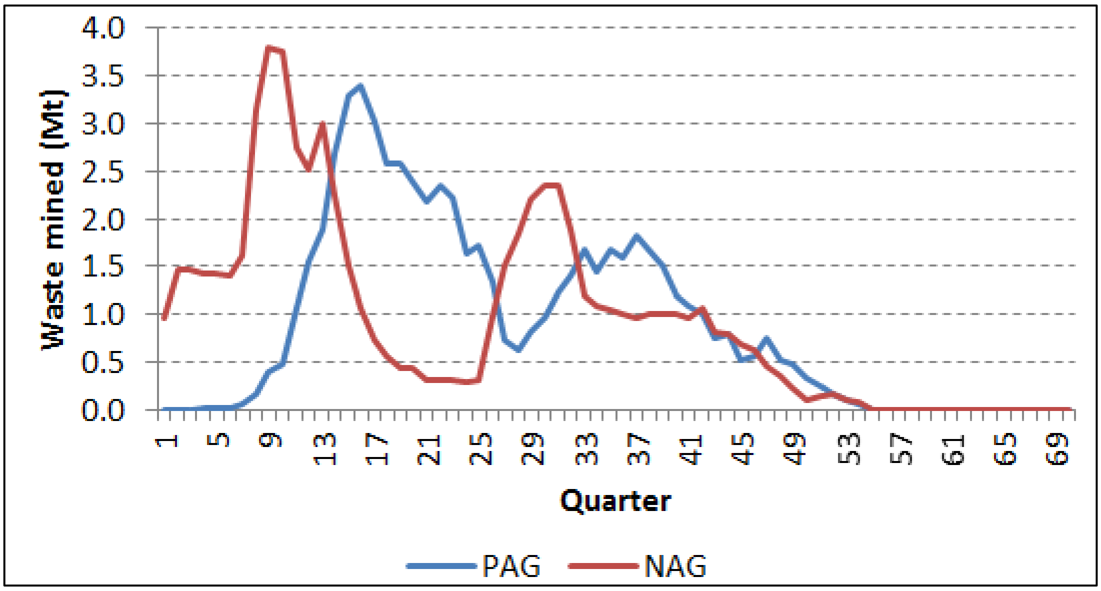
Figure 16.33: Emba Derho Mining Schedule



The Emba Derho waste rock contains both potentially acid generating (PAG) and non-acid generating (NAG) material. A schedule of the extraction of this material is shown in Figure 16.34. There are three key trend periods in production:

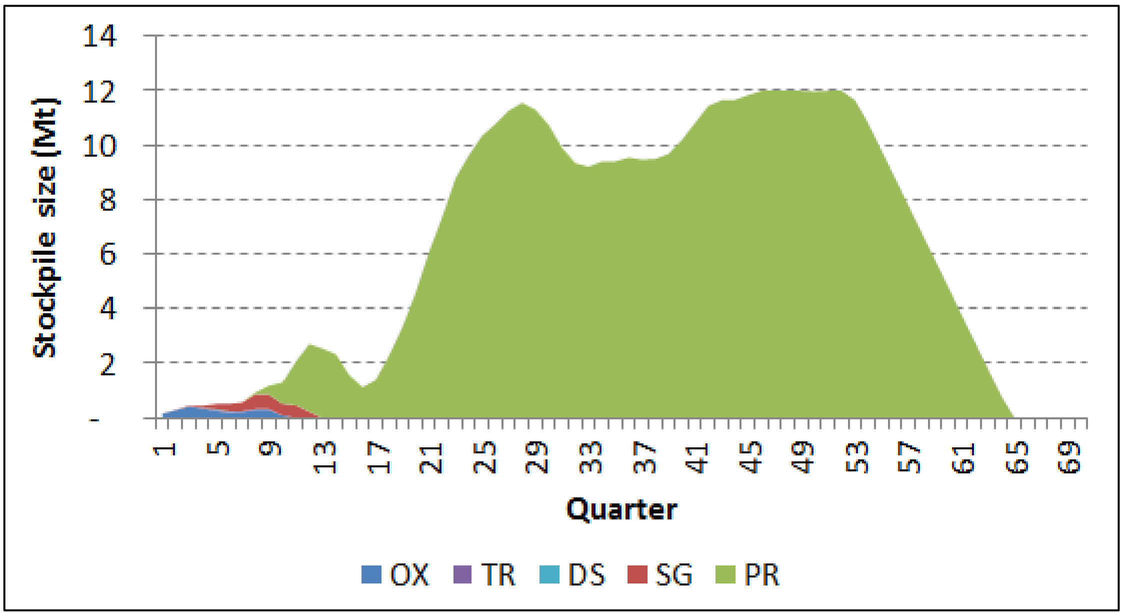
- An initial period dominated by NAG material as the oxide material is depleted (three years). Much of this material is used for initial construction purposes.
- An interim period of mostly PAG material (three years).
- A period of balanced PAG and NAG production (seven years).

Figure 16.34: Emba Derho Waste Schedule



The size of the Emba Derho stockpile is shown in Figure 16.35. In the initial periods the stockpile contains a number of rocktypes and includes material hauled from Debarwa to smooth out long haul production rates. After Year 3, the stockpile consists solely of Primary ores. The stockpile builds over time to a maximum size of 12 Mt in Year 13. It is then depleted over a period of three years towards the end of the mine.

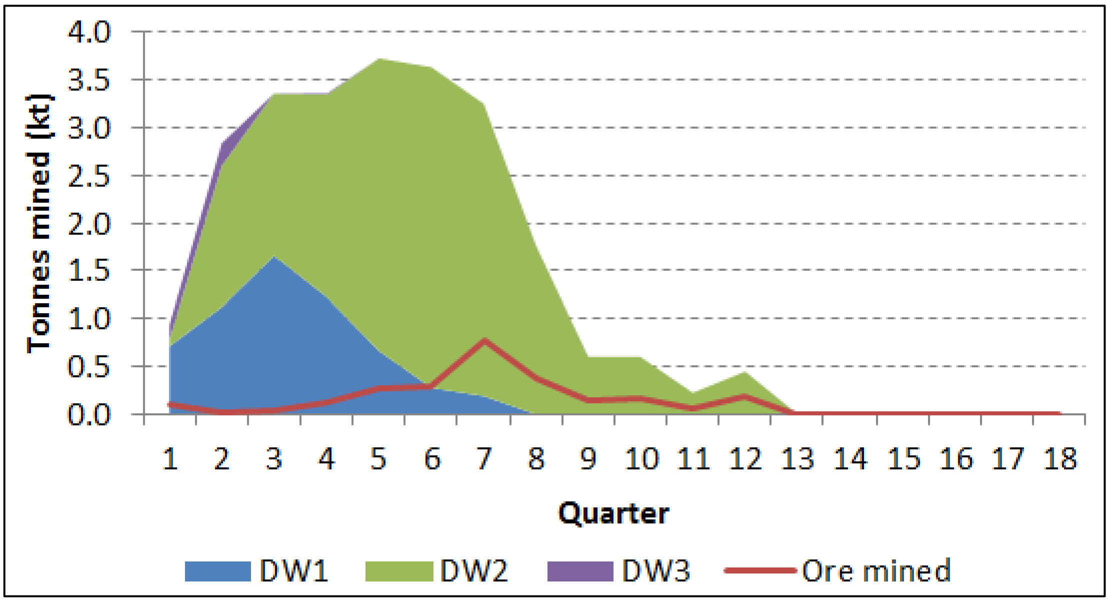
Figure 16.35: Emba Derho stockpile balance



16.7.4 Debarwa Schedule

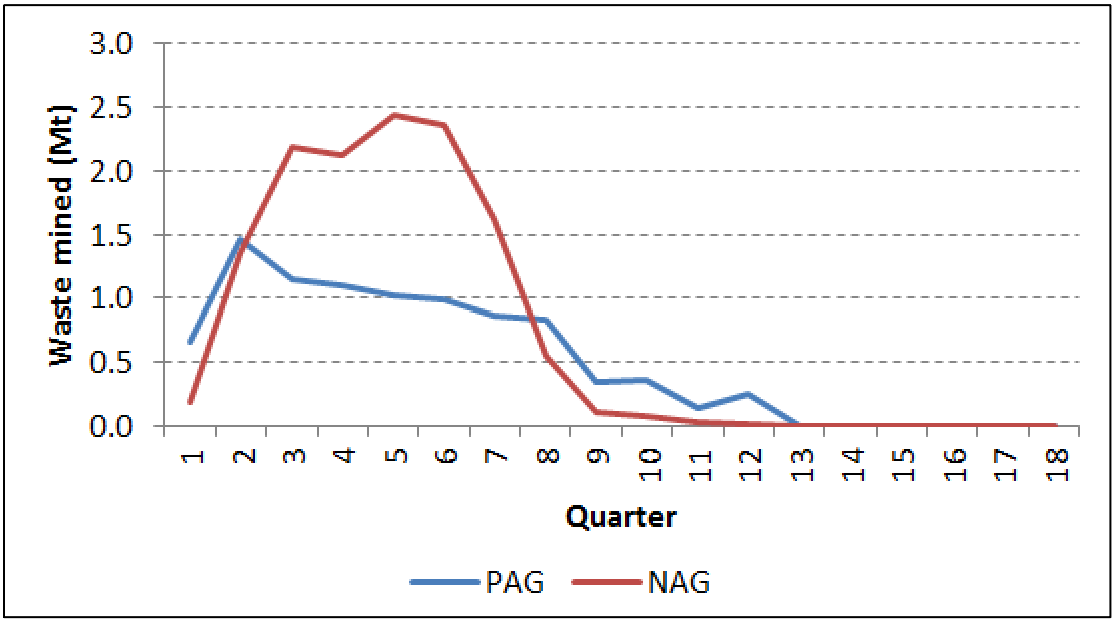
Debarwa is mined in three stages over a 3 year period (Figure 16.36). Both production stages start mining from the outset, but the starter stage (DW1) is advanced a slightly faster rate. The river diversion (DW3) is mined in the first six months to enable mining of the second stage (DW2).

Figure 16.36: Debarwa Mining Schedule[18]



Debarwa waste rock contains both PAG and NAG material. A schedule of the extraction of this material is shown in Figure 16.37. There is a fairly consistent ratio of NAG to PAG material in the first two years which allows facilities to be constructed and capping to be placed. The final year produces mostly PAG waste and there is therefore some need to stockpile some NAG material in the PAG dump for later use as capping.

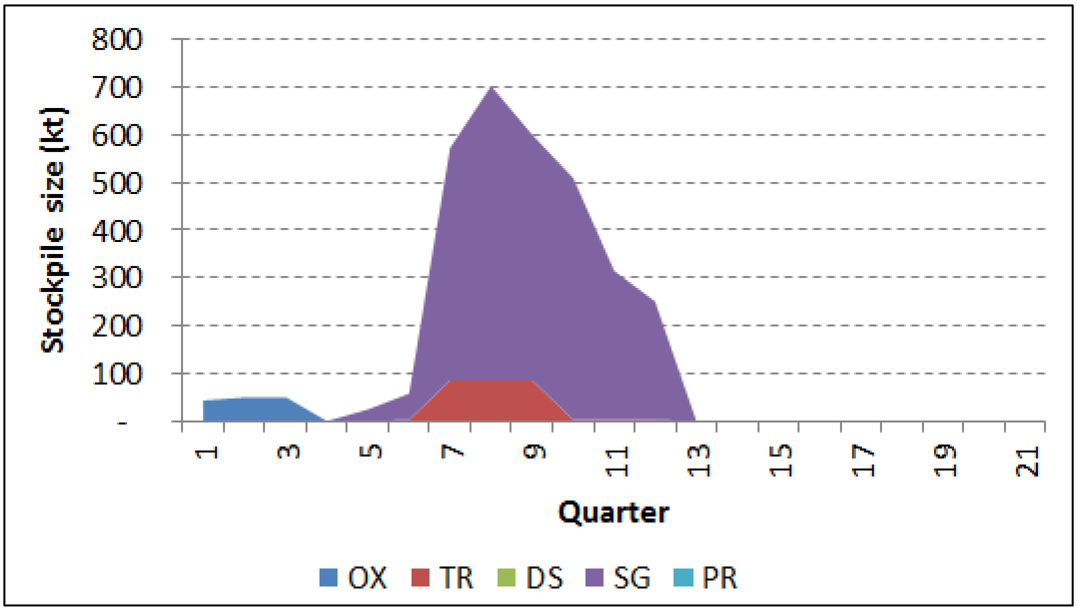
Figure 16.37: Debarwa Waste Schedule



[18] DW3 refers to the mining of the river diversion.

The Debarwa stockpile contains only material that is mined at Debarwa (Figure 16.38). Some Debarwa material is stockpiled at Emba Derho to smooth the long haulage production demand. The depletion of the Debarwa stockpile in quarter 12 ceases the Debarwa project.

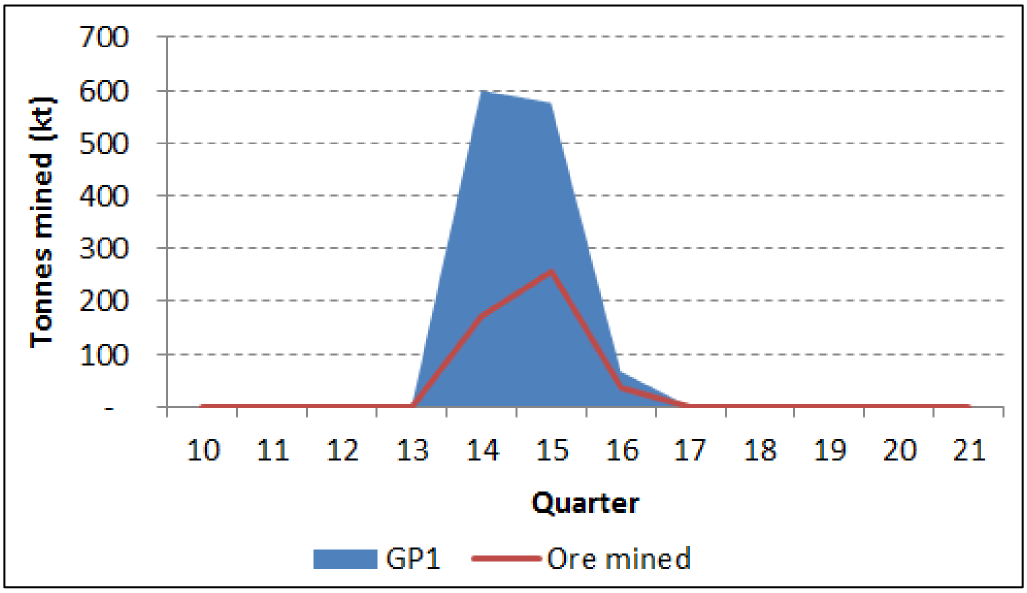
Figure 16.38: Debarwa stockpile balance



16.7.5 Gupo Schedule

Mining of Gupo is completed in approximately seven months (Figure 16.39) and is mined from a single pit.

Figure 16.39: Gupo Mining Schedule



All Gupo waste is considered to be NAG and is transported to Adi Nefas to be used as backfill for the underground operation.

16.7.6 Adi Nefas Schedule

The mining schedule targets an ore production rate of 400 ktpa ore from Adi Nefas.

Due to the high cost of maintaining both development and production fleets onsite, the schedule was based on developing the entire mine in advance of the planned ore production, allowing for the demobilizing of the development fleet from site as soon as possible. Total lateral development was limited to 300 m/month (a maximum fleet of two development jumbos). The annual Adi Nefas mining physical schedule is summarised inTable 16.37.

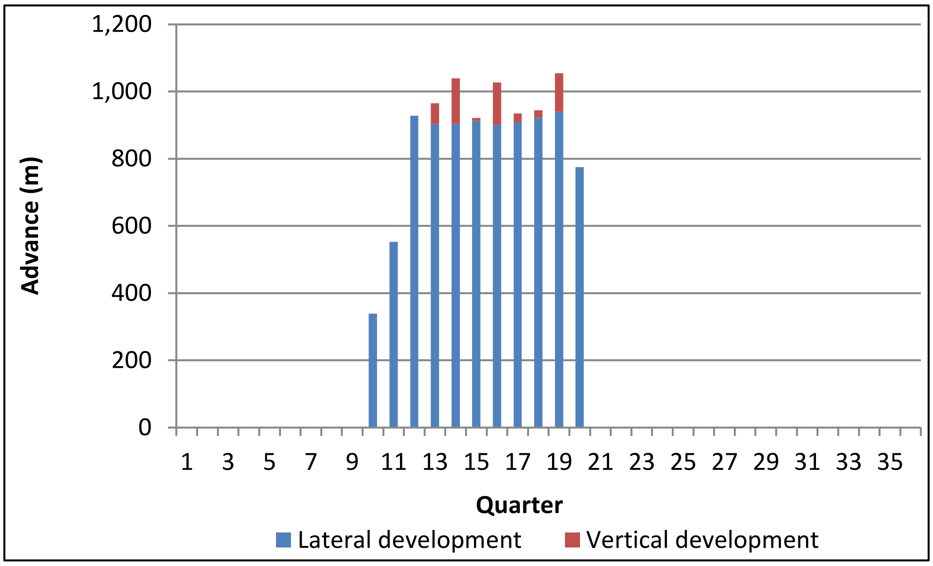
Table 16.37: Adi Nefas Annual Mining Schedule

	Units	Total LOM	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Capital development									
Lateral advance	(m)	4,794	2,518	1,615	661				
Vertical advance	(m)	496	61	296	139				
Waste tonnes	(t)	375,539	194,602	129,866	51,071				
Ore tonnes	(t)	7,392	2,868	1,764	2,761				
Au grade	(g/t)	2.35	3.3	2.3	1.4				
Ag grade	(g/t)	77.69	92.2	89.8	54.8				
Cu grade	(%)	1.22	1.5	1.7	0.6				
Zn grade	(%)	7.32	8.7	7.3	5.9				
Operating development									
Lateral advance	(m)	4,188	205	2,009	1,974				
Vertical advance	(m)	0	0	0	0				
Waste tonnes	(t)	50,689	1,469	17,118	32,102				
Ore tonnes	(t)	309,342	16,646	157,424	135,272				
Au grade	(g/t)	2.66	2.4	2.7	2.6				
Ag grade	(g/t)	91.97	73.5	94.7	91.0				
Cu grade	(%)	1.39	1.0	1.5	1.3				
Zn grade	(%)	8.12	6.8	8.1	8.4				
Stoping									
Ore tonnes	(t)	1,364,735	0	140,668	359,837	394,900	245,132	137,818	86,380
Au grade	(g/t)	2.86	0.0	3.3	2.7	2.6	3.2	3.2	2.8
Ag grade	(g/t)	97.67	0.0	104.8	100.1	89.2	104.3	103.2	87.2
Cu grade	(%)	1.60	0.0	1.5	1.5	1.3	1.9	2.3	1.4
Zn grade	(%)	8.22	0.0	9.8	8.4	8.4	7.5	7.6	7.2
Backfill									
Backfill required	(t)	1,017,038	0	73,679	312,520	308,300	219,513	103,027	
Direct-sourced U/G	(t)	132,211	0	49,038	83,173	0	0	0	
Sourced from Adi Nefas waste dump	(t)	268,359	0	24,641	229,347	14,371	0	0	
Sourced from other waste dumps	(t)	616,468	0	0	0	293,929	219,513	103,027	

16.7.6.1 Mine Development

Mine development commences in quarter 1, and is completed by quarter 11(Figure 16.40).

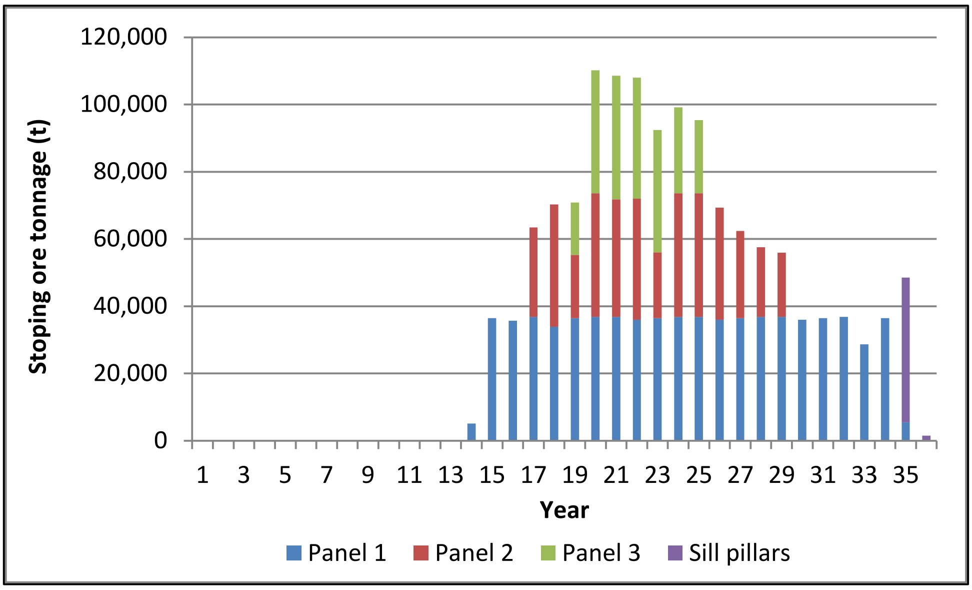
Figure 16.40: Adi Nefas Quarterly Mine Development by Quarter



16.7.6.2 Stoping

The first production ore is mined in quarter 15, 14 months after decline development commences. While the target ore production rate is 400 ktpa, the mine only achieves close to this rate for two years, when three panels are mined concurrently. Prior to this period, the mine development is not sufficiently advanced to provide the required number of stoping fronts. Once the third stoping panel (which has the smallest inventory) is exhausted, production decreases again. This is shown in Figure 16.41 below.

Figure 16.41: Adi Nefas Quarterly Stoping Ore Production By Panel



16.7.6.3 Total Material Movement

Ore production and underground waste generation are shown in Figure 16.42 and Figure 16.43. Total material movement underground (excluding backfill) is shown in Figure 16.44.

Figure 16.42: Adi Nefas Annual Ore Production

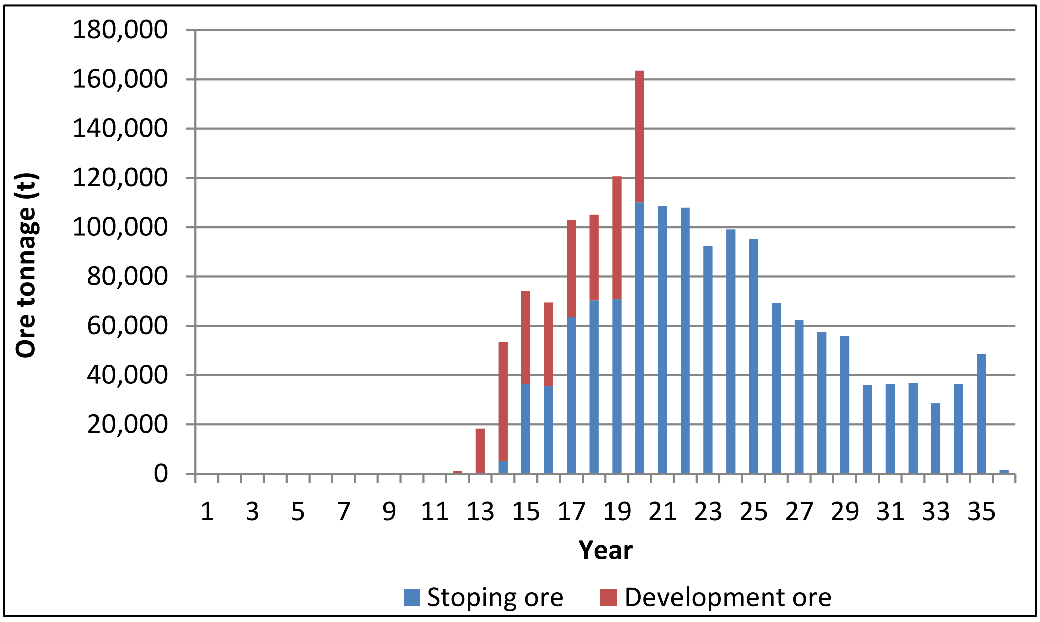


Figure 16.43: Adi Nefas Annual Waste Generation

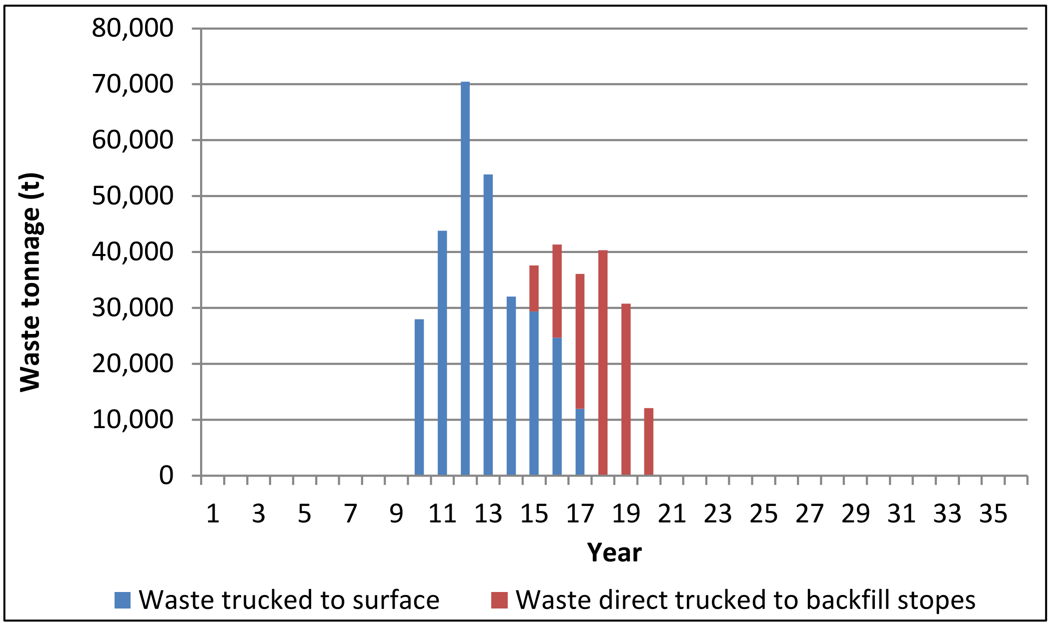
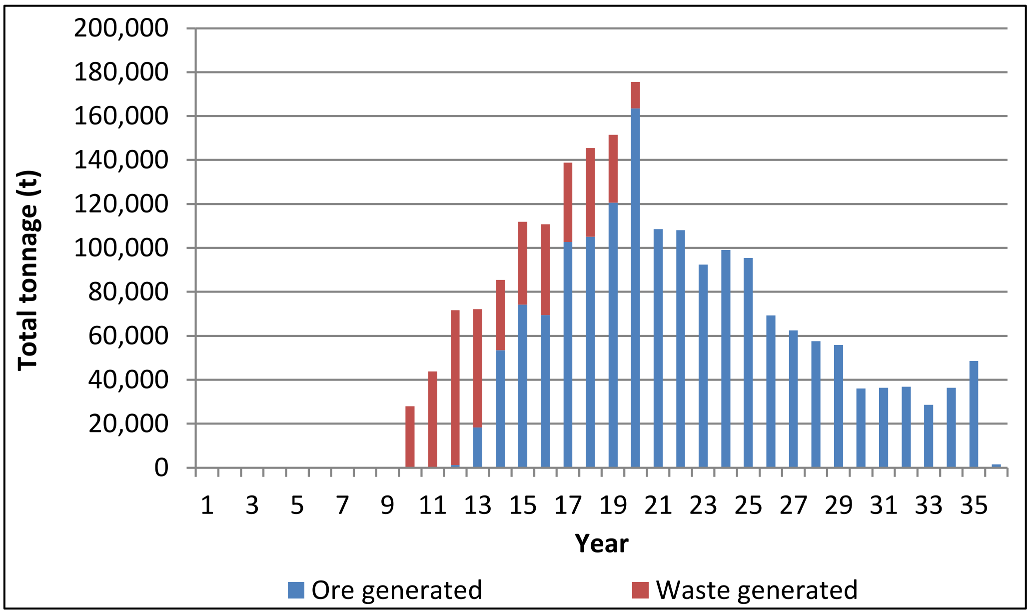


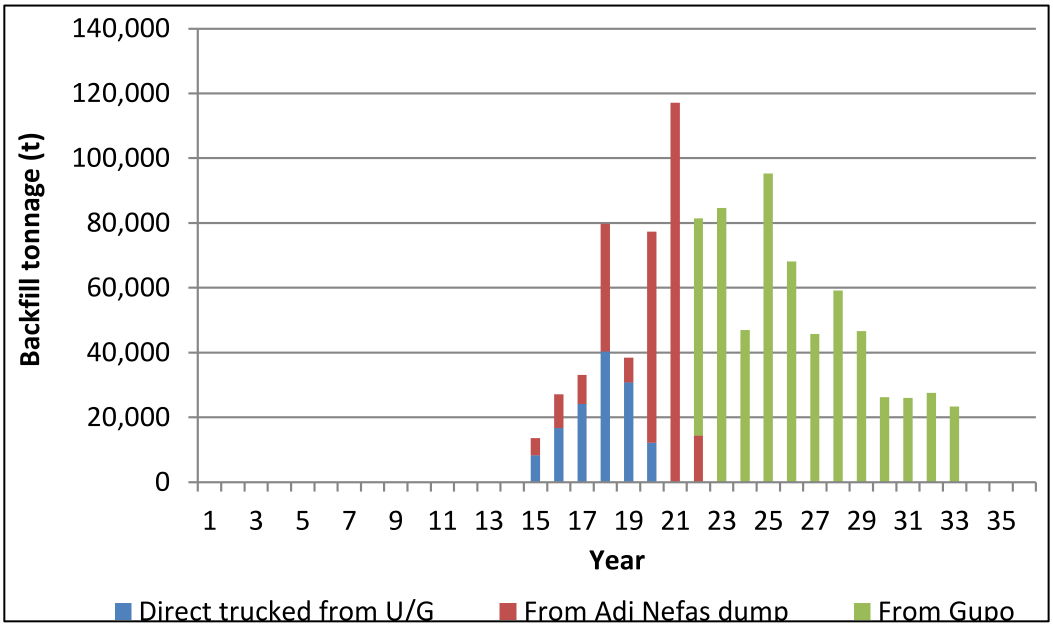
Figure 16.44: Adi Nefas Annual Total Material Movement [19]



16.7.6.4 Backfilling

Where possible, backfill material will be sourced directly from underground development. However, due to the campaigned nature of the development schedule, most of the development waste generated will be trucked to surface. This material will later be reclaimed and backhauled underground as required. From year 4 onwards, the Adi Nefas waste stockpile will be exhausted and backfill waste will instead be sourced from the Gupo waste that was stockpiled at Adi Nefas during mining at Gupo. The Adi Nefas backfilling schedule is shown in Figure 16.45.

Figure 16.45: Adi Nefas Backfill Schedule



[19] Excluding backfill

16.8 Processing Schedule

16.8.1 Overall

The overall processing schedule is shown in Figure 16.46 and Table 16.38. Processing commences in quarter 4 with heap leach loading. The heap leach continues for three years. DSO ore processing commences in quarter 5 and is completed within six months. Supergene ore processing commences in quarter 9 and continues for five quarters. The floatation plant is then modified to enable processing of primary ore in quarter 14 which is processed for a further 13 years.

Figure 16.46: Overall Processing Schedule

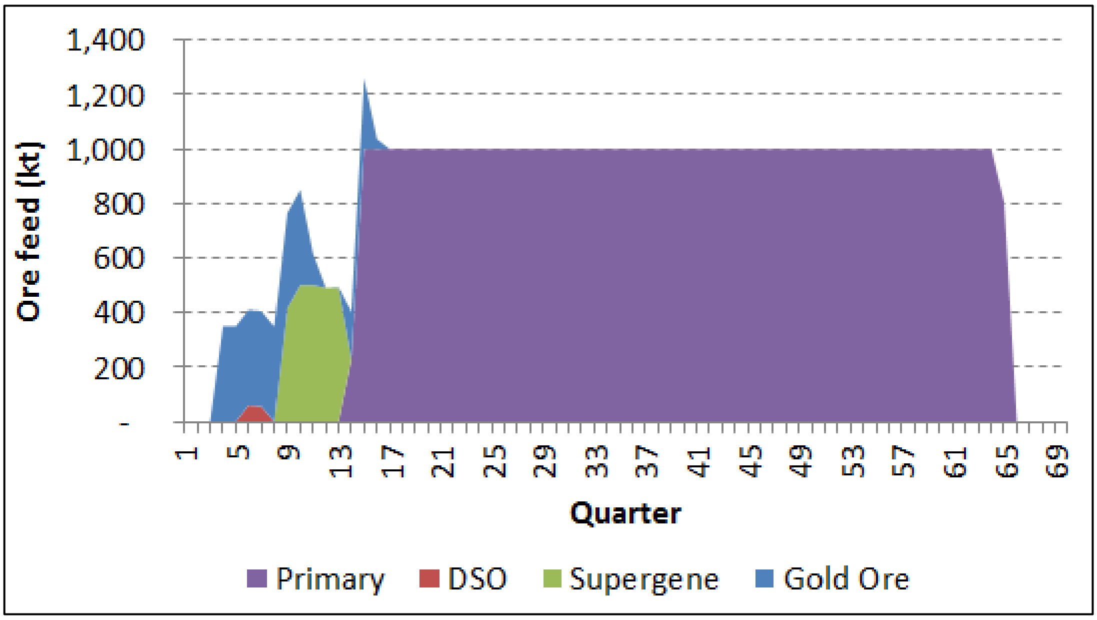


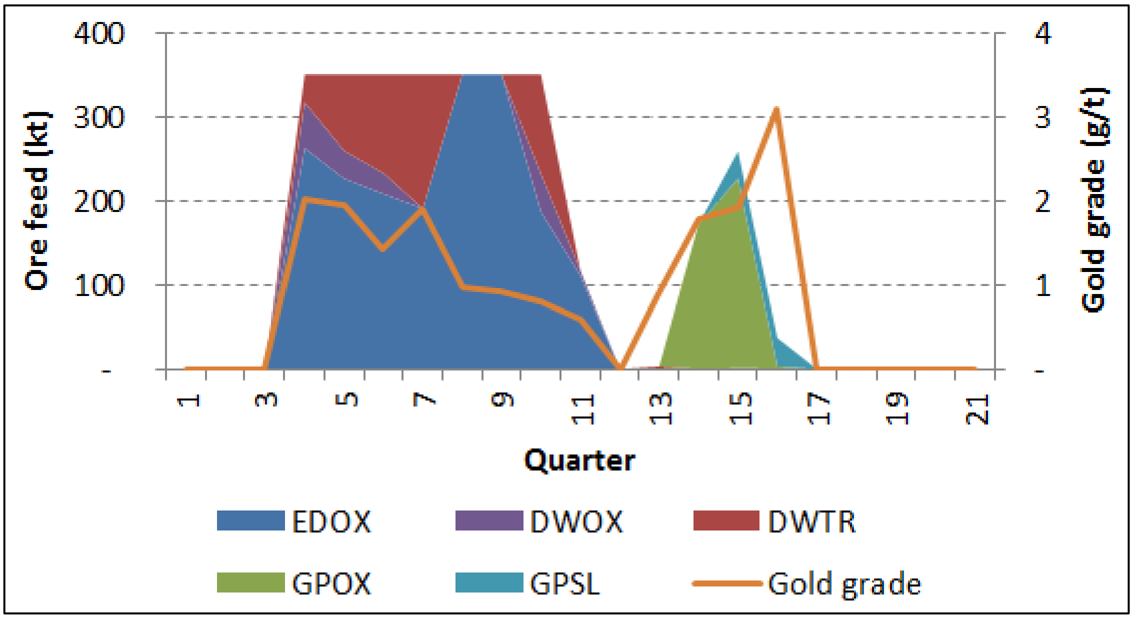
Table 16.38: Processing Schedule by Stream

	Gold Ore			DSO Ore				Supergene Ore				Primary Ore				
Year	Mass (kt)	Au (g/t)	Ag (g/t)	Mass (kt)	Cu (%)	Au (g/t)	Ag g/t	Mass (kt)	Cu (%)	Au (g/t)	Ag (g/t)	Mass (kt)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)
1	350	2.0	12.3													
2	1,400	1.6	10.8	116	15.6	3.0	76.8									
3	817	0.8	6.9					1,905	2.6	0.9	24.9					
4	471	2.0	-					490	0.8	0.4	8.8	2,230	0.9	3.7	0.8	25.0
5												4,000	0.6	3.6	0.7	25.3
6												4,000	1.0	2.8	0.5	20.0
7												4,000	1.4	1.6	0.4	16.2
8												4,000	1.1	1.6	0.4	13.1
9												4,000	0.5	2.4	0.4	15.0
10												4,000	0.5	2.5	0.2	9.2
11												4,000	0.6	1.9	0.4	9.1
12												4,000	0.7	1.5	0.2	7.0
13												4,000	1.2	0.6	0.1	7.0
14												4,000	0.5	1.2	0.2	7.7
15												4,000	0.3	1.1	0.2	7.4
16												4,000	0.3	0.8	0.2	6.1
17												805	0.2	0.8	0.2	6.8

16.8.2 Gold Ore

Gold ore processing commences in quarter 4 (Figure 16.47). In the first year of loading onto the heap leach, the ore feed is evenly split between Emba Derho and Debarwa ore. The second year of loading is dominated by Emba Derho feed, resulting in a lower average grade. There is a break in processing before Gupo ore becomes available in quarter 14. Processing of Gupo ore lasts for six months.

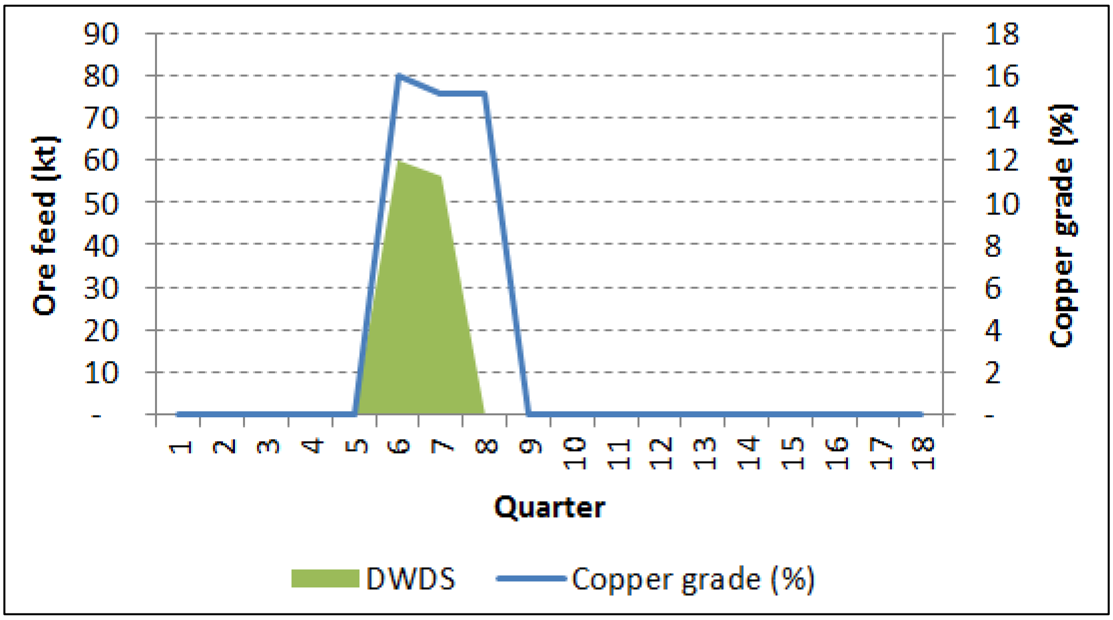
Figure 16.47: Gold Ore Processing Schedule



16.8.3 Direct Shipping Ore

DSO ore is processed over six months in quarter 5 and quarter 6 (Figure 16.48). The grade is constant over the duration at between 15% and 16% copper.

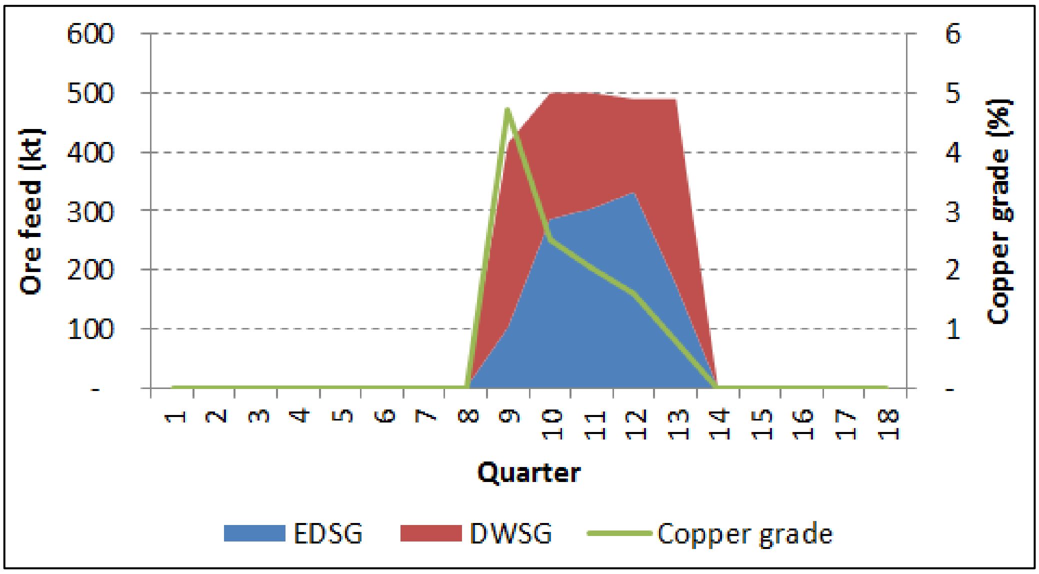
Figure 16.48: DSO Ore Processing Schedule



16.8.4 Supergene Ore

Supergene ore is processed over five quarters starting from quarter 9 (Figure 16.49). Initial production focuses on higher grade Debarwa ore with an average grade of approximately 4% copper. After the first six months, the average feed grade drops as more of the lower grade Emba Derho supergene ore is added to the production mix.

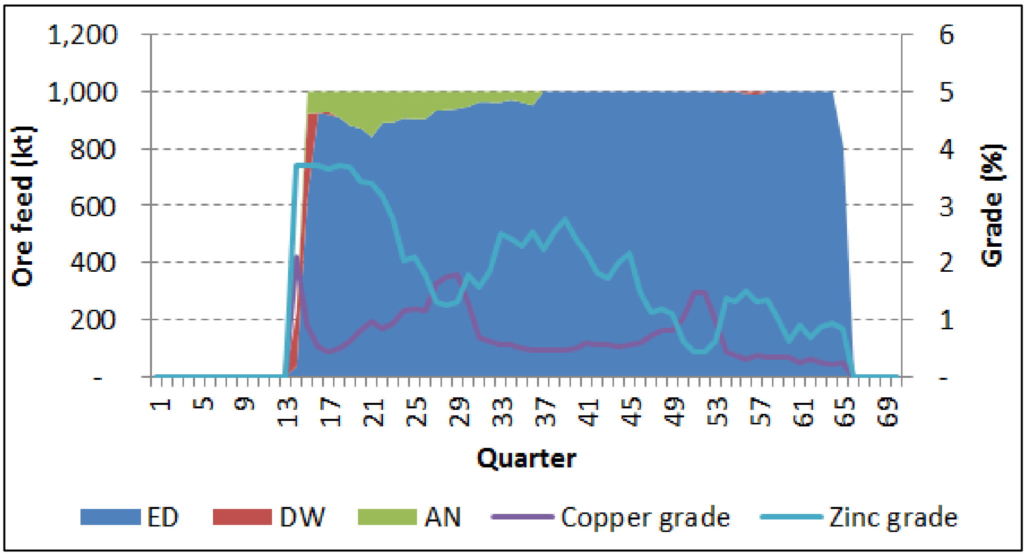
Figure 16.49: Supergene Ore Processing Schedule



16.8.5 Primary Ore

Primary ore processing commences in quarter 14 and continues for 13 years (Figure 16.50). In the first six months of processing, the feed is split between Emba Derho (75%), Debarwa (20%) and Adi Nefas ores (5%). The presence of Debarwa and Adi Nefas ores in the feed increases the average zinc grade during this time. In the following four years production is split between Emba Derho (90%) and Adi Nefas (10%). Over this period, the copper grade remains constant at about 1% copper (due to the head grade constraint) and zinc grade decreases. During the next four years Emba Derho is mined in isolation. The copper drops initially as the second Emba Derho stage is exhausted and then increases as the final stages are mined to depth. The zinc grade decreases over this time. The final three years of primary processing depletes the stockpile and accordingly the the average head grade is at its lowest.

Figure 16.50: Primary Processing Schedule



16.9 Product Schedule

A schedule of recovered product (doré, copper concentrate and zinc concentrate) is shown in Table 16.39.

Table 16.39: Product Schedule

	Doré		Copper Concentrate				Zinc Concentrate	
Year	**Gold (koz)**	**Silver (koz)**	**Conc dmt (x 1000)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Conc dmt (x 1000)**	**Zn (%)**
1	16	60						
2	48	169	116	15.6	3.0	77		
3	14	66	160	25.5	5.8	189		
4	18	-	65	25.2	13.8	449	130	54.8
5			82	24.2	20.7	695	241	55.0
6			140	24.6	9.5	320	184	56.4
7			193	27.0	5.5	180	91	59.5
8			144	27.3	5.6	178	94	61.0
9			74	25.0	13.3	449	141	62.0
10			66	25.0	7.6	242	153	60.0
11			80	25.0	7.1	172	118	60.0
12			94	25.0	3.3	108	88	60.0
13			157	27.8	0.9	40	33	57.0
14			70	25.5	4.6	138	69	59.5
15			47	24.0	7.0	220	61	59.5
16			34	24.0	8.1	249	46	59.0
17			7	24.0	8.8	290	9	59.0
Total	**97**	**295**	**1,527**	**25.0**	**6.9**	**223**	**1,457**	**58.3**

17 RECOVERY METHODS

17.1 Process Plant Summary

The Asmara Process Plant has been designed for the beneficiation of the following:

- 200,000 tonnes per annum (tpa) of copper (Cu) rich direct shipping ore (DSO)
- 1,400,000 tonnes per annum of fresh oxide/transition ore for the recovery & extraction of gold (Au) and silver (Ag)
- 2,000,000 tonnes per annum of fresh supergene ore for the recovery of copper concentrate
- 4,000,000 tonnes per annum of fresh primary ore for the recovery of copper & zinc (Zn) concentrates

Proven heap leach & CIS technology will be employed for the recovery and extraction of gold and silver from the oxide and transition ore, and sulphide copper and zinc concentrates will be recovered via sulphide flotation process technology. The DSO will be treated through the heap leach plant utilising the comminution circuit to produce a product size of ≤9.5 mm.

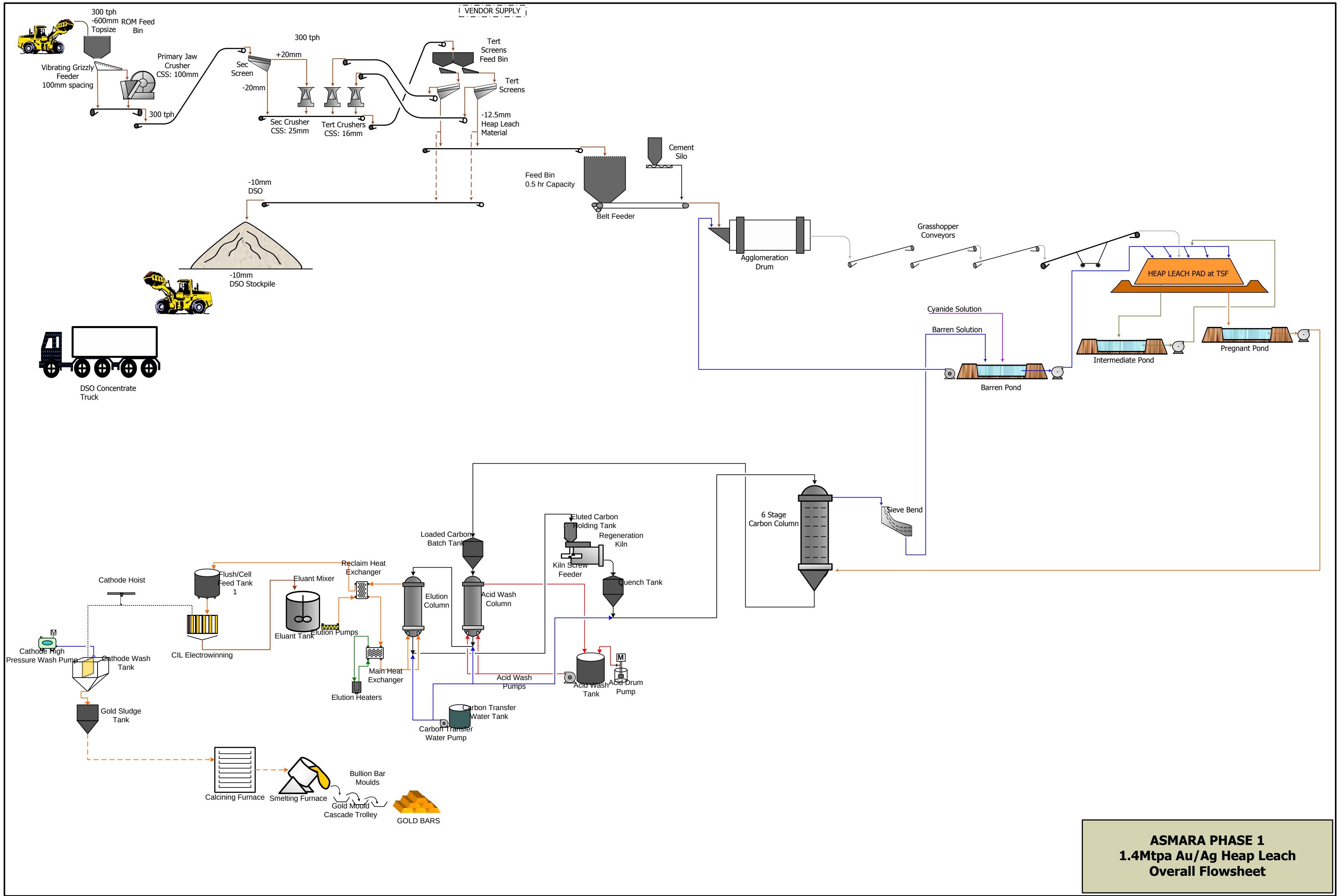
It is anticipated that the Asmara plant will be built in phases, with the proposed first phase being the heap leach and DSO plant. The second and third phases will include the flotation process plant at an initial 2 Mtpa throughput to treat the supergene ore, and will later be upgraded to 4 Mtpa to treat the primary ore. The two processes for these ores are summarised in the sections following.

17.1.1 Heap Leach & DSO

ROM ore (DSO or the oxide/transition ore) will go through a three stage crushing process to reduce the ore from ≤600 mm to a final tertiary product size of ≤12.5 & 9.5 mm. ROM ore will be scalped through a grizzly feeder to ensure that only the oversize of ≥100 mm reports to the primary jaw crusher. The crushed product will combine with the grizzly feeder undersize en route to the secondary crushing section. This material will be screened via a secondary double deck screen, with oversize from both decks reporting to the secondary cone crusher and undersize to the tertiary crushing section. The secondary crusher product will combine with secondary screen undersize and tertiary crusher product and feed the triple deck tertiary screens. The screen oversize from all decks will report to the tertiary cone crushers. The tertiary screen undersize of ≤12.5 mm or 9.5, will be the final crushing circuit product.

For the DSO this will be the end of the process. The DSO crushed product will be stockpiled to be transported in containers. The oxide/transition ore crushed product will report to the agglomeration feed bin which will feed the agglomeration drum, where it is mixed with cement/lime and barren solution in order to bind fine particles in the crushed product to produce competent agglomerates. The agglomerated ore will then be transferred and stacked on the heap leach pad via a series of grasshopper, slewing and stacking conveyors. After the ore is stacked to the required height of 10m lifts, irrigation of the ore with intermediate solution will commence. The pad will comprise two active cells, and barren solution will be pumped to the oldest cell to produce an intermediate solution which will be used for the fresh stacked ore. Provision will be made for a leaching period of 90 days for the fast leaching ores and 150 days for the slow leaching ores. Pregnant solution will be transferred to the CIS adsorption column containing activated carbon. Barren solution will be exiting at the top of the column and will be returned to the leach pads for irrigation. The process flowsheet diagram for Heap Leach and DSO is shown in Figure 17.1 below.

Figure 17.1: Process Flowsheet Diagram for Heap Leach and DSO Ores



The loaded carbon will be transferred once a day and will be hydraulically transferred via the bottom of the column, en route to acid washing. In the acid wash column, the carbon will be washed with a 3% solution of hydrochloric acid (HCl) to remove any carbonates that might otherwise foul carbon. The acid washed loaded carbon will then be transferred to the elution column, where the carbon is stripped of the gold/silver. The barren carbon will be transferred to the regeneration kiln to reactivate it for reuse in the adsorption column. The gold rich solution will be directed to the electrowinning cells via a header tank, and the gold will be deposited onto cathodes as sludge. The barren solution will be reused in the elution process. The gold sludge will be transferred to the gold room where calcining/drying of the sludge and finally smelting will take place in order to produce Au & Ag rich bullion.

The plant will be fully equipped with support services such as reagents, water and air services to provide these where required in the process.

17.1.2 Supergene & Primary Ore Flotation

ROM ore will be crushed in a single stage crushing circuit from ≤600 mm to ≤260 mm. ROM ore will be scalped through a grizzly screen to ensure only the oversize of ≥150 mm reports to the primary jaw crusher. The crushed product will combine with the grizzly screen undersize and will be conveyed to the mill feed stockpile. The crushed product will be withdrawn from the stockpile and will be conveyed to the grinding circuit where it will be milled through a SAG and ball mill in closed circuit with classification cyclones, to produce a mill product of P_{80} 70 µm.

The cyclone overflow slurry, at target solids of 35% m/m, will gravitate into the copper rougher feed tank passing through a linear trash screen. From this tank the slurry will be pumped to the copper Rougher flotation cells, where two copper concentrates will be recovered, a high (HG) and low grade (LG) copper concentrate. Depending on the copper grade in the HG concentrate, it will either be pumped to the cleaner or recleaner flotation or to the copper regrind mill. The copper rougher tails will be pumped to either the final tails thickener when treating the supergene ore, or to the zinc rougher feed tank when treating the primary ore.

The regrind mill will be utilised to improve copper liberation, which will improve the overall metallurgical performance. The copper HG & LG rougher concentrate will be pumped from the cyclone feed tank to the copper regrind mill in closed circuit with a classification cyclone. The underflow from the cyclone will gravitate to the regrind mill while the overflow reports to the cleaner flotation section. The regrind mill discharge will combine with the rougher concentrates in the cyclone feed tank.

The regrind cyclone overflow will feed the copper cleaner flotation cells, and the copper cleaner concentrate will be pumped to the copper recleaner flotation. The tails will report to final tails when treating supergene, and will report to Zn feed when treating primary ore. The copper recleaner flotation concentrate will be pumped to the head of the copper final cleaner flotation cells, the tails will gravitate to the first cell of the cleaner flotation cells. The copper final cleaner flotation concentrate will be pumped to the copper concentrate thickener. The tails will gravitate to the first cell of the copper recleaner flotation cells.

The copper concentrate thickener will be employed to increase density of this slurry to an acceptable ≥60% solids m/m, which will be suitable for the copper filtration process. The water recovered in the thickener overflow will be reused in various sections of the plant, including as spray water in the copper flotation section. The thickener underflow will report to the filtration feed storage tank, where it will be pumped to the copper concentrate filter. The filter will be employed to reduce the moisture content of the copper concentrate to ≤10% m/m.

A zinc flotation section will be employed to recover zinc form the copper rougher tails when treating the primary ore. The copper rougher tails will be pumped from the Zn rougher feed tank to the first cell of the zinc rougher flotation cells. Two zinc concentrates will be recovered, a high and low grade zinc concentrate. Depending on the zinc grade in the HG concentrate, it will either be pumped to the cleaner or recleaner flotation or to the zinc regrind mill. The zinc rougher tails will be pumped to the final tails thickener.

As in the case of copper, the regrind mill will be employed to improve zinc liberation to improve the overall metallurgical performance. The zinc HG & LG rougher concentrate will be pumped from the cyclone feed tank to the zinc regrind mill via a classification cyclone. The underflow from the cyclone will gravitate to the regrind mill while the overflow reports to the cleaner flotation section. The regrind mill discharge will combine with the rougher concentrates in the cyclone feed tank.

The regrind cyclone overflow will feed the zinc cleaner flotation cells and the zinc cleaner concentrate will be pumped to the zinc recleaner flotation. The tails will report to final tails. The zinc recleaner flotation concentrate will be pumped to the head of the zinc final cleaner flotation cells, the tails will gravitate to the first cell of the cleaner flotation cells. The zinc final cleaner flotation concentrate will be pumped to the zinc concentrate thickener. The tails will gravitate to the first cell of the zinc recleaner flotation cells.

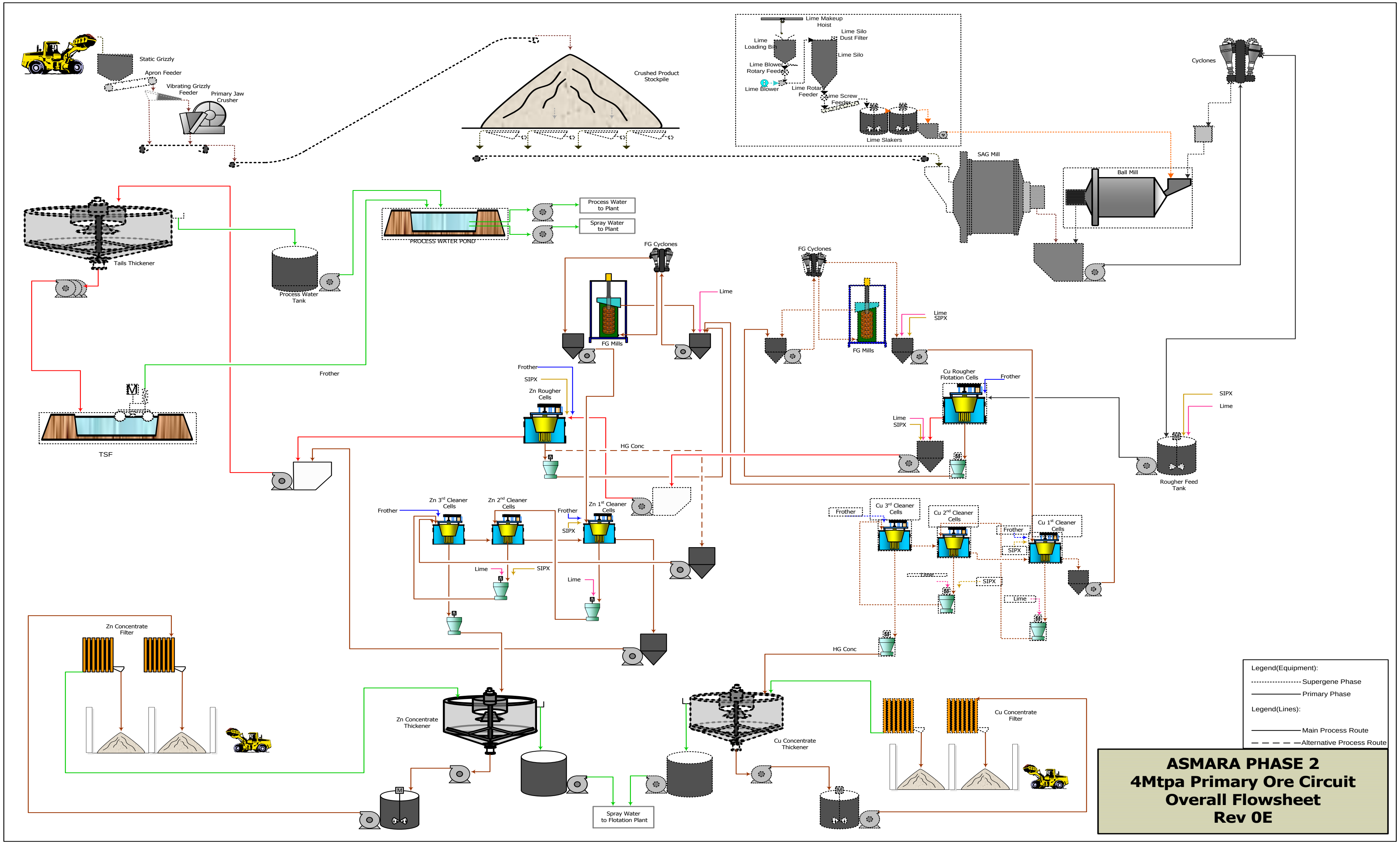
The zinc concentrate thickener will be employed to increase density of the slurry to an acceptable ≥60% solids m/m which will be suitable for the zinc filtration process. The water recovered in the thickener overflow will be reused in various sections of the plant, including as spray water in the zinc flotation section. The thickener underflow will report to the filtration feed storage tank, where it will be pumped to the zinc concentrate filter. The filter will be employed to reduce the moisture content of the zinc concentrate to ≤10% m/m.

The copper or zinc rougher tails will be pumped to the final tails thickener to recover water and also to ensure economical pumping to the final tails storage facility. The slurry will be densified to a solids content of ≥60%m/m. The thickener underflow (the final tails) will be pumped via a series of pumps to the tailings storage facility, while the thickener overflow will be reused in various sections of the plant including milling dilution water and screen spray water.

The plant will be fully equipped with support services such as reagents, water and air services to provide these where required in the process.

The supergene and primary flotation process flowsheet diagram is shown in Figure 17.2 below.

Figure 17.2: Process Flowsheet Diagram for the Flotation of Supergene and Primary Ores



17.2 Process Flowsheet Development

17.2.1 Heap Leach & Direct Shipping Ore

The development of the heap leach flowsheet followed from the testwork conducted at Kappes, Cassiday & Associates(KCA) under the supervision of Blue Coast Metallurgy (BCM). The general philosophy of the design was to ensure that the plant will be able to handle both the direct shipping ore through the crushing circuit, as well as the different ore types to be treated through the heap leach process.

17.2.2 Supergene & Primary Ore Flotation

The development of the flowsheet for the beneficiation of 2 Mtpa of supergene ore and later 4 Mtpa of primary ore followed extensive comminution, flotation and solid/liquid separation testwork. The general philosophy of the design was to ensure that the plant will be able to handle both ore types and throughputs, while minimising the changes that would be required when changing from the supergene to the primary circuit.

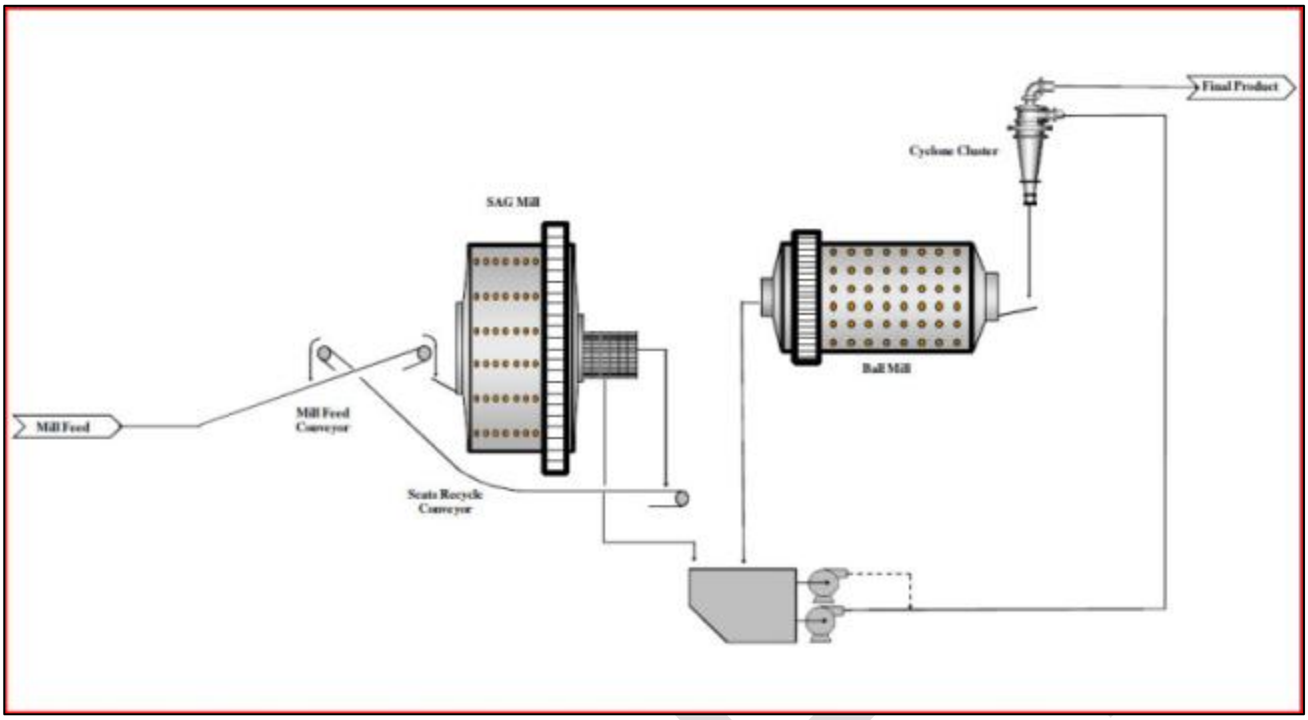
17.2.3 Comminution

A trade off study was conducted to assess the best option for the comminution circuit. Three possible options were examined, and Orway Mineral Consultants (OMC) was tasked to do a detailed comparison of these options. The three options are:

- Primary Crush, SAG and Ball Mill (SAB)
- Tertiary Crush and Ball Mill
- HGPR and Ball Mill

The trade-off study, utilising LOM as a basis, showed that the SAB circuit would be the preferred option with which to proceed in the design of the comminution circuit for the Asmara Project. The trade-off study took into account the capital and operating costs, as well as the point that the SAB comminution circuit is the least complex of the three comparative options. The selected process flowsheet diagram is depicted in Figure 17.3 below.

Figure 17.3: Process Flowsheet Diagram for SAB Comminution Circuit



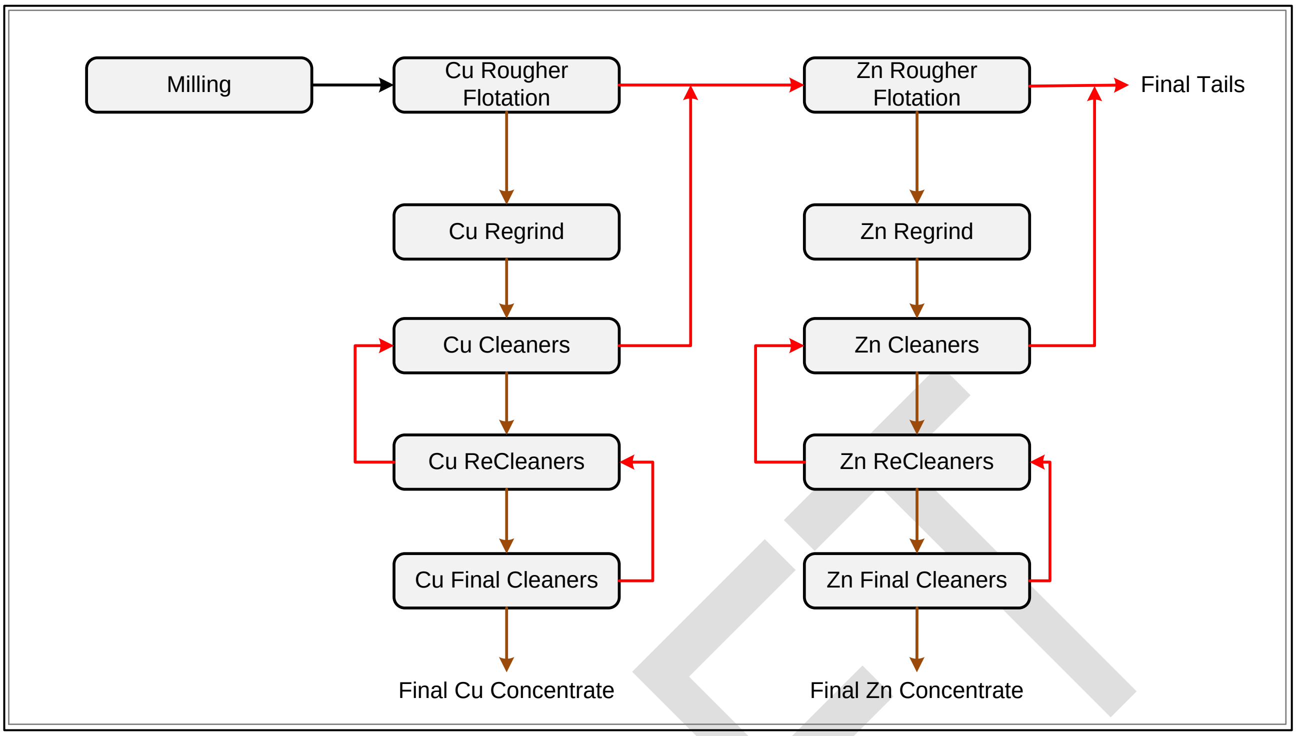
17.2.4 Flotation

Simulations were conducted by Eurus Mineral Consultants cc (EMC) and together with Blue Coast Metallurgy and SENET an optimal flowsheet design was developed for the recovery of a copper concentrate from 2 Mtpa supergene ore which will be upgraded to 4 Mtpa to recover copper and zinc concentrate.

The simulations were based on the lock cycle and flotation kinetics metallurgical testwork conducted on the various ore sources by SGS Vancouver under the direction of Blue Coast Metallurgy.

The selected process flowsheet diagram for the flotation circuit is shown in Figure 17.4 below.

Figure 17.4: Process Flowsheet Diagram for Flotation Circuit



17.3 Process Design Basis

Metallurgical testwork results were obtained from Blue Coast Metallurgy. The results from the testwork and historical data were used as the basis for the design of the Asmara feasibility study process plants for treatment of the following:

- 200,000 tonnes per annum of direct shipping ore (DSO)
- 1,400,000 tonnes per annum of oxide/transition ore
- 2,000,000 tonnes per annum of supergene ore
- 4,000,000 tonnes per annum of primary ore

Two process plants are envisaged. The heap leach plant will be designed to process the oxide/transition ore and the DSO, while the flotation plant will be designed to treat the supergene and primary ores.

Data used in the process design criteria has been obtained from various sources, including:

- Sunridge Gold Corporation (SGC)
- Metallurgical testwork
- Calculations based on collected data
- Vendor data or recommendation
- SENET database information
- Industry standard or practice
- Reference material
- Assumptions based on experience
- External consultants

17.3.1 Site Conditions

The two plants will be situated in the same area within a few kilometres of each other; therefore the same site conditions will apply to both. A summary of the site conditions for the proposed location of the Asmara plants is shown in Table 17.1 below.

Table 17.1: Summary of Site Conditions

	UNITS	SITE DATA	SOURCE
Project Location		Eritrea	SGC
Elevation / Altitude	mASL	2300	SGC
Temperature:			
Ave. Max Summer	ºC	26	SGC
Ave. Min Winter	ºC	7	SGC
Mean	ºC	18	SGC
Rainfall: Annual Ave	mm	534	SGC
Evaporation: Annual Ave	mm	1167	SGC / KP
Wind Velocity:			
Mean Velocity	km/h	29	SGC
Max Velocity	km/h	57	SGC
Wind Direction: Typical	km/h	ENE	SGC
Relative Humidity	%	40 - 70%	SGC

17.4 Process Design Basis - Heap Leach & DSO

Ore characteristics for the heap leach and DSO are described in Table 17.2 below.

Table 17.2: Ore Characteristics

	UNITS	DEB OXIDE	DEB TRANSITION	ED OXIDE	GUPO	DSO	SOURCE
Specific Gravity of ROM Ore	SG	3.23	2.97	2.16	2.775	4.94	Testwork - BCM
• Bulk Density of Crushed ore	t/m^3	1.86	1.61	1.50	1.59	2.96	Testwork - KCA
• Bulk Density of Stacked ore	t/m^3	1.89	1.62	1.50	1.61	-	Testwork - KCA
• Bulk Density of Stacked ore - Design	t/m^3	1.98	1.70	1.58	1.69	-	Calculated
Angle of Repose	º	37	37	37	37	37	
Max Lump Size	mm	600	600	600	600	600	Snowden
Required Final Product Size	mm	12.5	12.5	12.5	9.5	10	SGC / Testwork - KCA
Average Moisture Content of ROM Ore	%m/m	5.0%	5.0%	5.0%	5.0%	5.0%	SGC / Snowden
Ore Head Grades:							
Ore Head Grade (Au)	g/t	1.54	2.77	0.98	1.97	-	SGC / Snowden
Ore Head Grade (Ag)	g/t	7.89	31.13	4.46		-	SGC / Snowden
Ore Head Grade (Cu)	%						SGC / Snowden
Ore Head Grade (Zn)	%						SGC / Snowden
Lab Leach Dissolution :							
Au (as % of Heap Leach Feed)	%	59.0%	56.0%	69.0%	60.0%		Testwork - KCA
Ag(as % of Heap Leach Feed)	%	21.0%	38.0%	53.0%	44.0%		Testwork - KCA
Bond Crushing Work Index - Operating	kWh/t			8.7	11.6	10.5	Testwork - BCM
Abrasion Index - Operating	#	0.400		0.023	0.203	0.237	Testwork - BCM

17.4.1 Operating Schedule

Table 17.3: Operating Schedule

	UNITS	DEB OXIDE	DEB TRANSITION	ED OXIDE	GUPO	DSO	SOURCE
Ore reserves	kt	164	522	1 886	399	116	SGC / Snowden
Annual Tonnage Treated	Mtpa	1.40	1.40	1.40	1.40	0.20	SGC
Crushing Agglomeration and Stacking:							
Days per Year	days	365	365	365	365	365	SGC
Possible hours per annum	hrs	8760	8760	8760	8760	8760	Calculated
Operating Days per week	days	7	7	7	7	7	SGC
Operating Hours per day	hrs	14	14	14	14	2	SGC / SENET
Selected operating hours per annum	hrs	5110	5110	5110	5110	730	Calculated
Availability	%	95%	95%	95%	95%	95%	Industry Practice
Overall Utilization	%	55%	55%	55%	55%	63%	Calculated
Selected Plant Solids Throughput	tph	300	300	300	300	300	SENET
Heap Leach:							
Stacking schedule	months / year	12	12	12	12	-	SGC / SENET
Solution Application For CIS	days / year	365	365	365	365	-	SGC / SENET

17.4.3 Circuit Configuration

Table 17.4: Circuit Configuration

	UNITS	DEB OXIDE	DEB TRANSITION	ED OXIDE	GUPO	DSO	SOURCE
Crushing Circuit		Three Stage Crushing					SENET
Agglomerator Type		Rotating Drum				N/A	SENET
Agglomeration Agent	#	Cement				N/A	Industry Practice
Heap Leach Period	days	150	150	60	150	N/A	Calculated - KCA
Plant Leach Dissolution – Au (%Heap Leach Feed Grade)	%	56.0%	53.0%	66.0%	57.0%	N/A	Calculated
Plant Leach Dissolution – Au (%Heap Leach Feed Grade)	%	56.0%	53.0%	66.0%	57.0%	N/A	Calculated
Recovery circuit (ADR plant)		Adsorption, Acid wash, Elution, Electrowinning and Smelting					SGC / SENET

17.5 Process Design Basis – Supergene & Primary Ore Flotation

17.5.1 Ore Characteristics

Table 17.5: Ore Characteristics

	UNITS	SUPERGENE ORE	PRIMARY ORE	SOURCE
Specific Gravity of ROM Ore:				
Emba Derho	#	4.14	4.26	Testwork – BCM
Debarwa	#	3.99	4.05	Testwork - BCM
Max Lump Size	mm	600	600	Snowden
Average Moisture Content of ROM Ore	%m/m	5.0%	5.0%	SGC / Snowden
Average Angle of Repose of ROM Ore	º	38º	38º	SGC / Snowden
Ore Head Grades, Emba Derho:				
Ore Head Grade (Au)	g/t	0.20	0.20	SGC / Snowden
Ore Head Grade (Ag)	g/t	12.60	9.30	SGC / Snowden
Ore Head Grade (Cu)	%	1.00	0.60	SGC / Snowden
Ore Head Grade (Zn)	%	0.20	1.60	SGC / Snowden
Ore Head Grades, Debarwa:				
Ore Head Grade (Au)	g/t	1.30	1.20	SGC / Snowden
Ore Head Grade (Ag)	g/t	31.20	25.90	SGC / Snowden
Ore Head Grade (Cu)	%	4.50	2.20	SGC / Snowden
Ore Head Grade (Zn)	%	0.10	3.90	SGC / Snowden
Ore Head Grades, Adi Nefas:				
Ore Head Grade (Au)	g/t	N/A	2.90	SGC / Snowden
Ore Head Grade (Ag)	g/t	N/A	99.80	SGC / Snowden
Ore Head Grade (Cu)	%	N/A	1.60	SGC / Snowden
Ore Head Grade (Zn)	%	N/A	8.60	SGC / Snowden
Bond Crushing Work Index	kWh/t	7.3	11.2	OMC
Rod Mill Work Index	kWh/t	16.7	14.8	OMC
Ball Mill Work Index	kWh/t	14.9	12.3	OMC
Abrasion Index	#	0.218	0.242	OMC
JK Tech Parameters:				
a	#	73.9	100	OMC
b	#	0.90	0.45	OMC

17.5.2 Operating Schedule

Table 17.6: Operating Schedule

	UNITS	SUPERGENE ORE	PRIMARY ORE	SOURCE
General:				
Ore reserves	kt	2 395	51 035	SGC / Snowden
Annual Tonnage Treated	t/a	2 000 000	4 000 000	SGC
Crushing:				
Days per Year	days	365	365	SGC
Overall Utilization	%	33%	67%	Calculated
Selected plant solids throughput	t/h	685	685	Calculated
Milling:				
Days per Year	days	365	365	SGC
Selected Operating Hours per annum	hrs	8000	8000	SGC / SENET
Availability	%	95%	95%	Industry Practice
Overall Utilization	%	91%	91%	Calculated
Selected plant solids throughput	t/h	250	500	Calculated

17.5.3 Circuit Configuration & Recovery

Table 17.7: Crushing Circuit

	UNITS	SUPERGENE ORE	PRIMARY ORE	SOURCE
Crushing Circuit		Single Stage Crush, SAG and Ball Mill	Single Stage Crush, SAG and Ball Mill	SENET / SGC
Flotation Circuit		Copper Rougher, Regrind and 3 Stage Cleaning Flotation.	Copper Rougher, Regrind and 3 Stage Cleaning Flotation Zinc Rougher, Regrind and 3 Stage Cleaning Flotation.	Testwork – BCM
Overall Cu Recovery	%	79	86	Testwork – BCM
Overall Cu Grade	%	25	25	Testwork – BCM
Overall Zn Recovery	%	-	86	Testwork – BCM
Overall Zn Grade	%	-	56	Testwork – BCM

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The Asmara Project is spread over an extensive area consisting of four mine sites and a single tailings storage site. It is a greenfields project without existing infrastructure. The proposed infrastructure is to fully support the overall mining, process plant and construction operations for the various sites.

The three sites of Debarwa, Adi Nefas and Gupo will be utilised solely for mining purposes. The Emba Derho mine site contains the largest portion of on-site infrastructure for the Asmara Project, which is required for the open pit mine and two processing plants including:

- Emba Derho open pit
- NAG and PAG waste dumps
- Mine access road
- Haulage roads
- Mine water ponds
- Mine water dam
- Flotation plant
- Heap Leach plant
- Tailings storage dam
- Mai Bela extraction weir

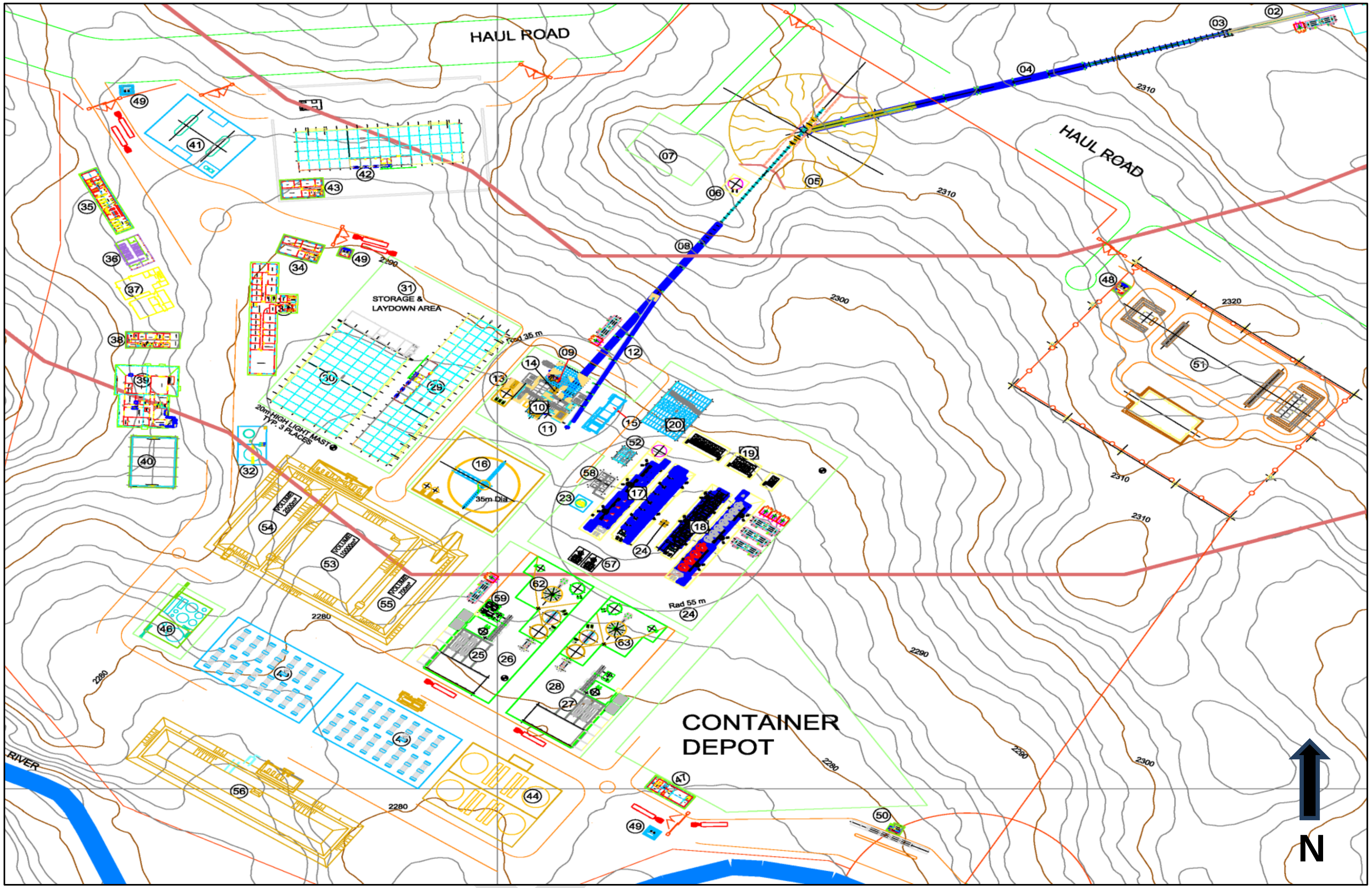
18.2 Emba Derho Mine Site

The main infrastructure required for the development of the Emba Derho mine site and flotation plant is listed below:

- Mining facilities
    - Mining building
    - Mine workshop
    - Mine refueling facility
    - Explosive magazine
- Process plant and administrative facilities
    - Administration building
    - Plant warehousing
    - Plant workshop
    - Mine laboratory and sample storage
    - Plant kitchen and messing facility
    - Changehouse
    - Mine clinic
    - Reagent storage
    - Water treatment facility
    - Sewage treatment
    - Plant control room
    - Fuel storage
    - Weighbridge

The layout of the Emba Derho mine site infrastructure is provided in Figure 18.1.

Figure 18.1: Emba Derho Mine Site Infrastructure



Legend

No	Description
01	Prim. Crush, Bin & Feed Rom Pad
05	Stockpile
10	Milling Area
15	Control Room (Over M.C.C.)
16	Tailings Thickening
17	Copper Floatation Area
18	Zinc Floatation Area
19	Reagents Make Up Area
25	Copper Concentrate Filtration Area
27	Zn Concentrate Filtration Area
29	Plant Workshop
30	Plant Warehouse
33	Administration Block
34	Metallurgical Office
35	Change House
36	Dining Room
37	Kitchen & Food Storage
38	First Aid Clinic
39	Laboratory
40	Sample Storage
42	Mine Workshop
43	Mine Office
44	Fuel Storage
45	Power Plant
47	Security Office
50	Weighbridge
51	Explosives Store
53	Raw Water Pond
54	Process Water Pond
57	Vertimills
62	Cu Concentrate Thickening
63	Zn Concentrate Thickening

Source: SENET

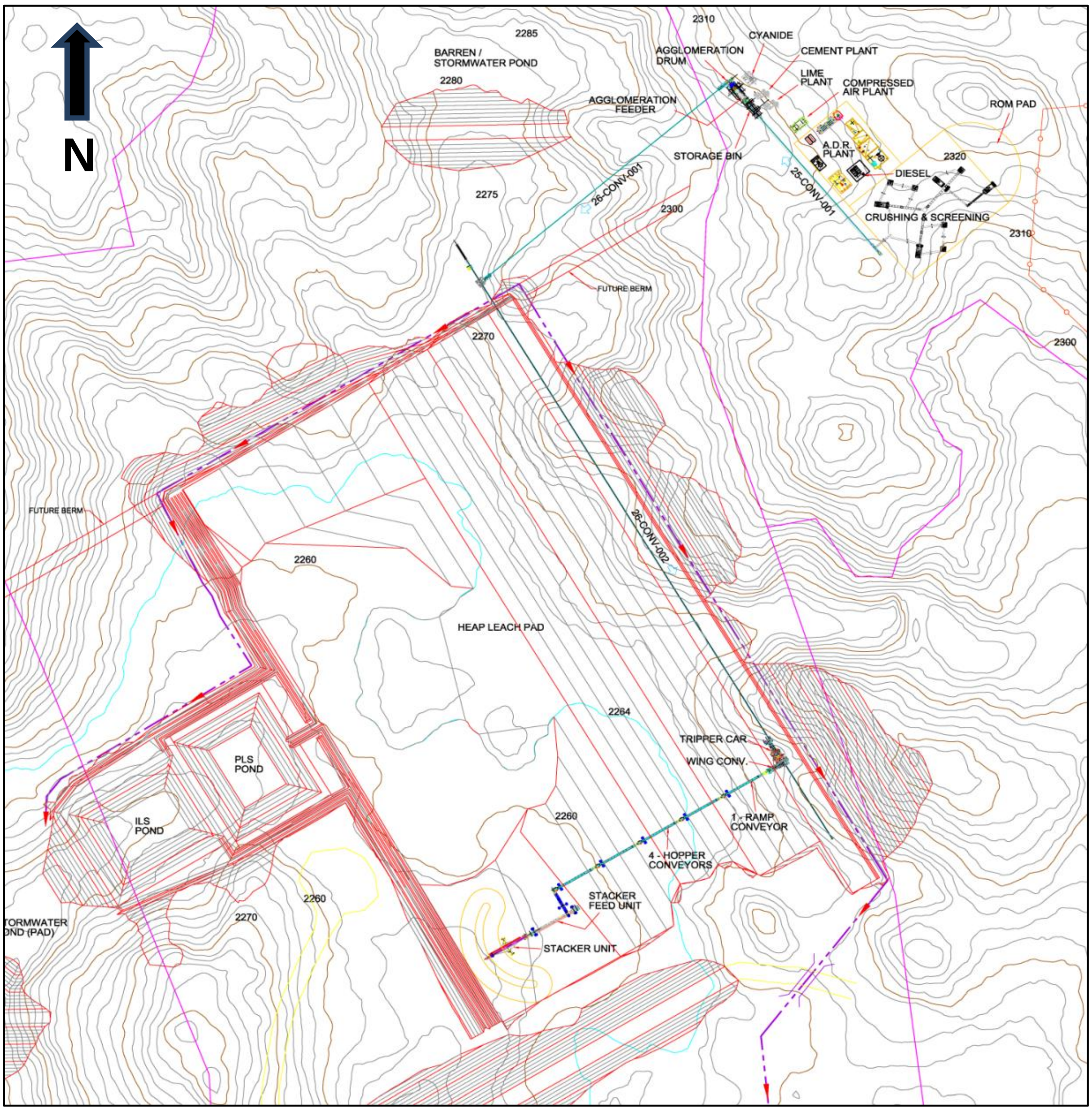
18.3 Emba Dehro Heap Leach Plant

The main infrastructure required for the development of the Emba Dehro heap leach plant will be as follows:

- In-plant roads
- Plant and administration offices
- Plant warehouse
- Plant workshop
- Plant messing facility
- Reagent storage
- Plant control room
- Ablutions
- Water supply
- Fuel storage

No mining facilities are present at the heap leach plant

Figure 18.2: Overall View of the Heap Leach Pads and Plant



Source: SENET

18.3.1.1 Mine Site Infrastructure

The three sites of Debarwa, Adi Nefas and Gupo will be utilised solely for mining purposes. The extent of site infrastructure will be kept to a minimum as the life of these mines ranges between one year and five years. Each site will be provided with temporary infrastructure including workshop, offices and ablutions which will be moved on from site to site as each mine site is mined out.

18.4 Water Management and Supply

Protection of the regional surface water and groundwater resources in Eritrea is fundamental to the successful development of the Asmara Project. The feasibility site water management design has been developed to meet the following key objectives:

- Ensuring mining operations and site access are not impeded except during extreme storm events
- Providing a continuous supply of fresh water and recycled process water to the HL facilities and the process plant
- Containing contact water and surface run-off water within the project area and using it preferentially within the mine site water balance
- Preventing contact water discharge to the surrounding surface water and groundwater resources

18.4.1 Emba Derho

The Emba Derho deposit area is located in the Mesheala River catchment area, a tributary of the Tokor reservoir approximately 3.5 km downstream. Maximizing environmental protection is particularly important in this watershed.

The TSF and integrated HL pad will be located in the Mai Bela River catchment area, the adjacent drainage to the south of the Mesheala River. Positioning this facility outside of the Tokor reservoir watershed was a strategic decision taken largely to limit disturbance and environmental risks to the domestic water supply. The Mai Bela River receives raw sewage from Asmara and the watershed is less environmentally sensitive and more suitable for tailings and heap leach operations and storage.

Project components such as WRSAs, ore stockpiles, the process plant and other associated infrastructure cannot practically or economically be located outside of the Mesheala River watershed. The Emba Derho water management concept has been developed to include seven collection ponds (excluding those associated with the process plant), ditches as appropriate to collect all mine contact water from the various disturbed areas, and pumps and pipelines to transfer the water to the EMWP.

The EMWP serves as the primary site water management facility for containment and recycle of contact water within the Tokor reservoir watershed. The EMWP will be used to manage inflows from all surface water management ponds as well as the Emba Derho pit dewatering system.

The project layout also includes the Mai Bela abstraction reservoir (MBAR) for make-up water supply. The MBAR will be located on the Mai Bela River, just west and downstream of the TSF, and will provide water to the HL facilities, TSF, or directly to the process plant as required, meeting demands under various climatic and operating conditions.

18.4.1.1 Collection Ponds, Pumps and Pipelines

Seven Collection Ponds, excluding those associated with the Process Plant, will be required at the Emba Derho project area to collect runoff from the saste rock storage areas, Rom pad, ore stockpiles and roads.

Collection Ponds have been sized according to environmental and operational design requirements. In general, ponds with low environmental risk in the event of discharge are designed to contain and transfer the volumes associated with a 1/10 year 24 hour storm event.

18.4.1.2 Emba Derho Pit Dewatering System

The Emba Derho pit dewatering system will remove the seasonal surface water inflows and the groundwater inflows to the Open Pit on an on-going basis through the mining period, and will discharge to the EMWP. The pit dewatering system will be installed in three stages to meet the increasing dewatering flow rates and pit depth.

18.4.1.3 Emba Derho Mine Water Pond

The EMWP will be located approximately 400 m to the southeast of the Emba Derho open pit, and will be the main control point for contact water at the Emba Derho site. Along with TSF recycle water, it will be used as a primary supply source for the Process Plant to sustain mill operations. The EMWP will also provide water for dust suppression until the project is closed, and after closure, will become a water supply reservoir for local agriculture and farming use.

The EMWP impoundment incorporates 1.5 Mm^3 of operating water storage capacity, with an additional 0.5 Mm^3 of stormwater capacity and a spillway to handle extreme inflows exceeding the 1/100 year 24 hour storm event.

The embankment will be constructed entirely with NAG waste rock and saprolite generated in the Open Pit pre-stripping. An HDPE liner will be placed on the upstream face of the embankment and across 20 m of basin floor at its toe, where it will be keyed in to form a low permeability seepage cut-off.

A floating pump station (pontoon) and water supply pipeline will connect the EMWP to the process water pond, and an offtake from this pipeline will be used to fill the tanks on dust control trucks.

18.4.1.4 Tokor Pipeline Offtakes

The Tokor pipeline is the domestic water supply pipeline that transports water from the Tokor reservoir to its domestic and industrial end users in Asmara and its surrounding communities. The pipeline passes in close proximity to the TSF, the HL facilities and the process plant and provides a valuable source of water for both the initial construction/commissioning period and the on-going plant fresh water requirements. Additionally, the Tokor pipeline will provide a good contingency supply in the event that there is a water shortage that cannot be met by the primary make-up water supply source on the Mai Bela River.

Two taps will be constructed to extract water from the pipeline, one near the gold plant and HL facilities, and the other near the process plant. Water will be purchased from the Eritrean government at the going rate (currently \$1.25/m^3) as required to meet the needs of the Project.

18.4.1.5 Mai Bela Abstraction Reservoir

The MBAR will be a make-up water supply reservoir located on the Mai Bela River to the west of the TSF. The MBAR will provide make-up water for use in the HL facilities and the process plant in periods of shortfall that cannot be met by the TSF and EMWP. The facility has been designed with a capacity of 1.5 Mm^3 which has been calculated to exceed the estimated maximum make-up requirements for both average and dry climatic conditions. The structure will become a long term asset after mine closure for the local people requiring water for agriculture and farming use.

18.4.2 Debarwa

The Debarwa deposit is located along the Gual Mereb River Valley at its confluence with the Mereb River. The Gual Mereb River has a sizeable upstream catchment of approximately 6 km^2, and the Mereb River is a major river with an estimated upstream catchment area of 180 km^2. Due to the location of the Open Pit across both of these river systems, water management provisions on both rivers will be required to facilitate mining.

The Debarwa open pit will be mined over a period of three years. The Gual Mereb River will be blocked and contained in the Debarwa mine water pond (DMWP), and the Mereb River will be permanently diverted around the southern extent of the Open Pit using what is summarily called the Mereb River diversion works. Other water management components of Debarwa include the pit dewatering system and the east collection pond, both of which transfer water to the DMWP.

18.4.2.1 Debarwa Mine Water Pond

The DMWP will be located across the Gual Mereb River approximately 300 m north of the open pit. The DMWP will serve as the primary site water management facility for the Debarwa mine during operations, containing catchment runoff as well as inflows that are transferred from the pit dewatering system and the east collection pond

The DMWP impoundment incorporates 1.0 Mm^3 of operating water storage capacity, with an additional 0.5 Mm^3 of stormwater capacity and a spillway to handle inflows exceeding the 1/5 year 24 hour storm event.

18.4.2.2 Mereb River Diversion Works

The Mereb River will be diverted around the southern end of the Debarwa open pit. The Mereb River Diversion Channel will be cut through the hillside on the west side of the existing river bed. The base of the channel will be excavated below the river bed through sand and gravel channel deposits to ensure surface and subsurface flows are intercepted and prevented from inflow to the pit excavation.

18.4.3 Adi Nefas

At Adi Nefas, a single collection pond will be constructed to store mine contact water from the above ground facilities as well as water pumped out of the underground workings. The pond will be located upstream of the Adi Nefas water supply reservoir and will be built with a capacity of 28,000 m^3 to store the 1 in 10 year 24 hour storm inflow event plus ten days of underground mine dewatering without any transfers out of the pond.

Table 18.1: Water Balance Summary

	PROCESS PLANT BALANCE	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Ref	**Inputs**																	
1a	TSF Reclaim																	
1b	MBAR via TSF Reclaim	0	0	275,000	723,000	1,016,000	989,000	776,000	412,000	204,000	0	0	0	0	0	0	0	0
2	EMWP Supply																	
3	Fresh Water Make-up (Tokor Pipeline)	0	0	130,000	185,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	55,000
	Sum of Inputs																	
	Outputs																	
4	Water in Tailings Slurry																	
	GOLD PLANT CONSUMPTION																	
	Inputs																	
5	MBAR Supply	0	0	430,000	259,000	0	0	0	0	0	0	0	0	0	0	0	0	0
6	Fresh Water Supply (Tokor Pipeline)	0	473,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	TSF INITIAL FILL & POND MGMT																	
9	initial fill of the TSF (from Tokor Pipeline)	0	500,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
10	MBAR to TSF	0	0	60,000	120,000	120,000	120,000	110,000	60,000	30,000	0	0	0	0	0	0	0	0
	OPEN PIT DEWATERING																	
11	Open Pit to EMWP																	
	DUST SUPPRESSION																	
7	Tokor Pipeline Supply	300,000	300,000	300,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0
8	EMWP Supply																	
	WATER SUPPLY TOTALS																	
1b, 5, 10	MBAR Supply	0	0	765,000	1,102,000	1,136,000	1,109,000	886,000	472,000	234,000	0	0	0	0	0	0	0	0
2, 8	EMWP Supply																	
3, 6, 7, 9	Fresh Water Supply (Tokor Pipeline)	300,000	1,273,000	430,000	185,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	272,000	55,000

18.5 Tailings Storage Facility

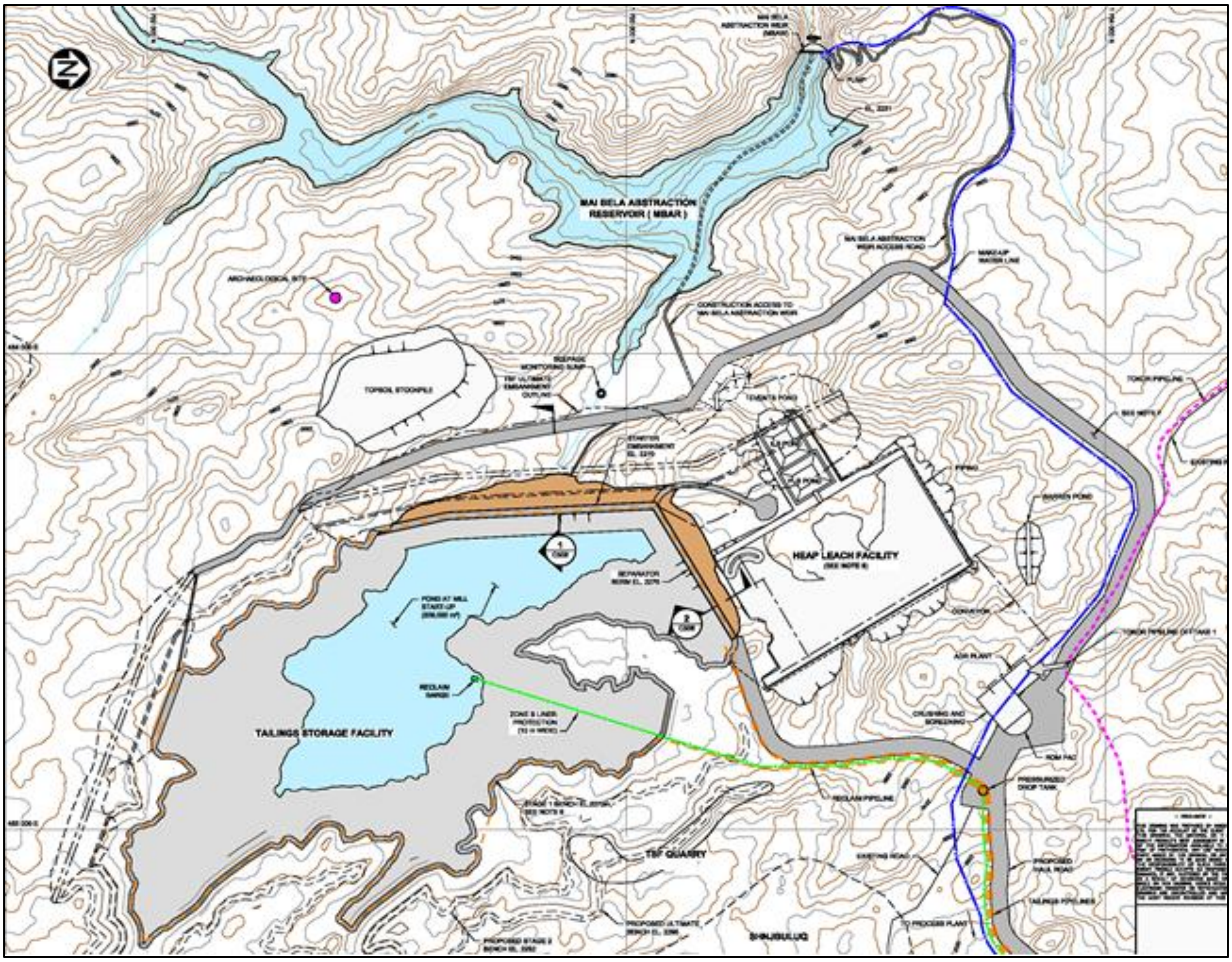
The tailings storage facility (TSF) is situated in the Mai Bela River drainage area, a watershed that receives sewage effluent from the City of Asmara. The Mesheala River drainage, located to the north of the TSF, is a tributary catchment of the Tokor reservoir which provides roughly one third of the fresh water supplied to the people of Asmara and surrounding communities. One of the fundamental factors considered in TSF site selection studies was ensuring that surface water and groundwater protection in the Tokor reservoir is maximized.

The natural ground elevation in the TSF basin ranges between 2 230 m and 2 295 m. The TSF will occupy a surface footprint of roughly 170 ha. The Abune Buruk Springs (a small spring) a church and community, the Shnjbuluq drinking water well, an irrigation reservoir, rain-fed agricultural land, and a rock quarry (Kodadu Quarry) are all located within the footprint area.

The TSF design includes expansion of the existing Kodadu Quarry to provide roughly 400 000 tonnes (t) of rock that will be crushed and used as aggregate for various construction components. The TSF will also incorporate and fully integrate the heap leach (HL) facility that will be operated in the initial few years of the project for gold recovery. The TSF has been sized to permanently store 53.4 million tonnes (Mt) of tailings (30 900 000 m^3) and 4 Mt (2 700 000 m^3) of transition and oxide ore as HL material, plus operating and storm water storage within the fully lined impoundment.

Starter and ultimate TSF layouts are shown in Figure 18.3 and Figure 18.4.

Figure 18.3: TSF General Arrangement Drawing (Startup)



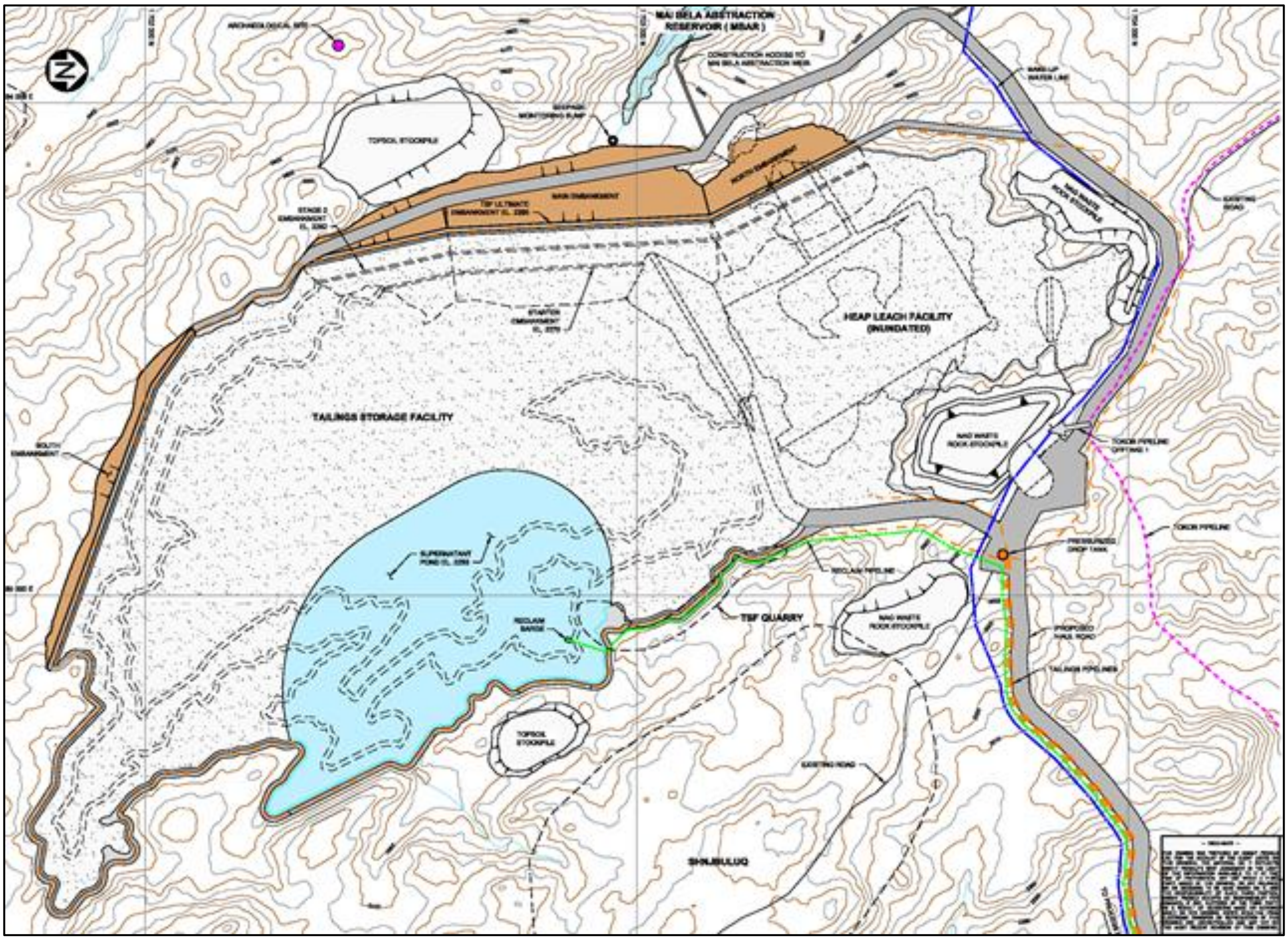
Source: KP

18.5.1 Design Objectives

The feasibility design of the TSF incorporates the following requirements:

- Permanent and secure confinement of all tailings and gold ore within a fully lined basin
- Sufficient water storage capacity to contain all mine site contact and process waters in the supernatant pond to support mill operations throughout the project duration, as well as additional freeboard capacity for containment of storm water inflows
- Staged development of the embankments and impoundment liner system over the life of the project to defer capital expenditures
- An integrated HL facility in the north portion of the basin in operations, which becomes inundated by tailings after leaching is completed
- A separator berm designed to prevent tailings from encroaching on the HL operations until the end of Year 6
- The inclusion of monitoring features for all aspects of the facility to ensure performance goals are achieved

Figure 18.4: TSF General Arrangement Drawing (Ultimate)



Source: KP

18.5.1.1 Embankments

Tailings will be contained between the natural valley slopes and constructed embankments at the west and southern extents of the facility. The embankments are defined as the main and south embankments in the main TSF area. The north embankment will be required to facilitate tailings disposal and encapsulate the HL area within the TSF basin, and is a northern extension to the main embankment.

The maximum embankment height will be 60 m above the natural ground level at the main embankment, and the TSF will be constructed with a basin liner of high density poly-ethylene (HDPE) for maximum environmental protection. TSF construction is planned in three stages to defer capital costs and to integrate construction with material availability from the open pit.

Construction of the main and south embankments to an elevation of 2,270 m and 2,282 m, and construction of the separator berm to an elevation of 2,270 m and 2,276 m respectively, will be required to contain tailings in Stage 1 and Stage 2A. Construction of the north embankment to an elevation of 2,282 m in Stage 2B will provide containment within the HL area and will merge the HL area with the rest of the TSF basin. Construction of Stage 3 will involve raising all embankments to the final crest level of 2,295 m. However, the TSF basin provides ample additional capacity for further expansion should more ore be discovered or become economically viable. Downstream construction is required for staged expansion of the tailings embankments to facilitate liner installation. A 5 m permanent bench will be built at the Stage 1 and Stage 2 crest levels to facilitate liner installation for all raises.

Stability analyses confirm the stability of the embankment under static and seismic loading conditions. The calculated factors of safety (FOS) confirm the stability of the embankment.

18.5.1.2 Leakage and Seepage Control

The entire impoundment area and upstream face of the tailings embankment will be lined with a continuous 1.5 mm thick HDPE liner overlying a low permeability subgrade bedding layer. The liner subgrade will consist of compacted low permeability saprolite or soil that is prepared into a smooth surface onto which the liner will be installed. Permeability values of the subgrade material will be 10^{-7} m/s and lower. This low permeability subgrade will inhibit any seepage water that passes through small pinholes or localized defects in the HDPE liner. Similarly, the fine grained tailings particles also have low permeability characteristics and will plug any defects to further prevent seepage losses from the impoundment. The basin liner will be constructed in three stages that correspond to the three stages of embankment construction. Liner for Stage 1, 2A, 2B and 3 will be completed by the end of Year 2, 4, 6 and 8 respectively in order to stay ahead of the growing tailings beaches and supernatant pond. The rate of tailings expansion will be monitored and the timing of staged construction may be adjusted to better match actual conditions.

18.5.1.3 Tailings Delivery

Slurry tailings will be delivered at approximately 60 percent solids content by mass from the process plant to the TSF. Tailings will be discharged into the TSF from valved off-takes (spigots) located along the distribution pipelines. The coarse fraction of the tailings will tend to settle more rapidly and will accumulate close to the discharge points, forming a gentle beach with a slope of about 0.5 to 1%. Finer tailings particles will travel further in the slurry before settling, and they will settle beneath the supernatant pond. Tailings discharge will commence along the main tailings embankment and south side of the TSF. The location of active tailings deposition will be changed routinely throughout the operating life of the mine in order to control the tailings beaches and supernatant pond location. The locations of tailings discharge will be controlled to ensure water recovery at the reclaim barge to maximize the potential for the recovery of water of acceptable clarity for plant processing.

18.5.1.4 Reclaim Water

Ponded water will be reclaimed from the tailings impoundment by a barge mounted (pontoon) pump station. The pond water will consist of process water recovered from the settled tailings slurries, make-up water supplied from the Mai Bela Abstraction Reservoir (MBAR), direct precipitation, catchment runoff, and, in wet periods, surface water collected and transferred from various mine areas as required to achieve an overall mine site water balance with no discharges Reclaimed water will be pumped from the reclaim barge to the process water pond at the process plant.

18.6 Communications and Control Network

Conceptual satellite and IT communications architectures were developed for the Project for both a satellite communications system as well as an IT and telecommunication system.

A common plant-wide fibre optic backbone will be installed to cater for the requirements of the networks.

18.6.1 Satellite Communications

A satellite network will be installed with sufficient bandwidth to provide the necessary voice and data communications for the following areas:

- Emba Derho flotation plant
- Heap leach plant
- Accommodation camp
- Debarwa mine site
- Adi Nefas mine site

The satellite network will be installed at the start of the construction phase and retained for the operations phase.

18.6.2 IT and Telecommunications

A permanent IT and telecommunications system will be installed at the main plant with sufficient capability for the operations phase of the project. The telecoms and IT network will be connected to the public switched network via a microwave or satellite link.

18.7 Mine Security

Security personnel will be employed by the mine. They will secure access at gates located at strategic points on all four mine sites. There will be automated boom gates at the flotation plant and at the two vehicle entrances to the utilities and administration areas, with security guards on duty 24 hours a day. Security access points to the plant will be monitored by CCTV. Turnstiles at the main access gate and flotation plant entrance will be controlled by a card reader system, which will allow access to authorized employees.

The security control room will be located within the mine's administration building. This office will contain the security system server, and two view nodes for CCTV monitoring. Visitors will be signed in and allowed onto site only in the presence of a senior member of the management staff.

18.8 Power Supply

18.8.1 Generating Power Plants

The design, procurement and erection of the gold heap leach plant and the copper flotation plant will run concurrently. A single power plant of approximately 30 MW capacity (MFO and diesel fuel) will be installed to meet both process plants requirements.

A 300 kVA twin generator set will be utilised at Debarwa until shifted to Adi Nefas as emergency back-up generators.

18.8.2 Power Distribution

Provision has been made in the main MV switchboard on the flotation plant for four medium voltage starters for each of a ball mill, SAG mill and two regrind mills. The starting of the ball mill and the two regrind mills will require automatic liquid resistance controllers, while the SAG mill will use a variable speed drive (VSD) to start and provide speed control.

The heap leach plant and the heap leach pads will be located at the tailings storage facility which is located approximately 4 km from the flotation plant at Emba Derho. Overhead power lines will supply power from the power plant at Emba Derho to heap leach facilities.

For the Adi Nefas mine site power will be obtained from the main power plant, located at the Emba Derho flotation plant, taken off the main 11 kV bus and transferred by means of an overhead power line of approximately 8 km in length.

18.9 Fuel Storage

Design of the main fuel storage facility depot will be based on a two-month diesel and MFO supply for the power plant and mining fleet, plant use, transport vehicles and all mobile equipment and vehicles. The storage facility will include:

- 2 x 2000 m^3 diesel storage tanks
- 2 x 2000 m^3 MFO storage tanks
- Bulk lubes storage
- Diesel day and transfer tanks
- MFO day and transfer tanks
- Diesel loading, metering and dispensing equipment
- Fire protection water and foam system

The transport of diesel and MFO to site will be carried out by the chosen fuel supplier delivering from Massawa which is approximately 130 km from site.

Mulitiple standalone satellite diesel fuel storage tanks of 15,000l each will be placed at all of the heap leach plant, mine workshop, Debarwa mine site and Adi Nefas mine site.

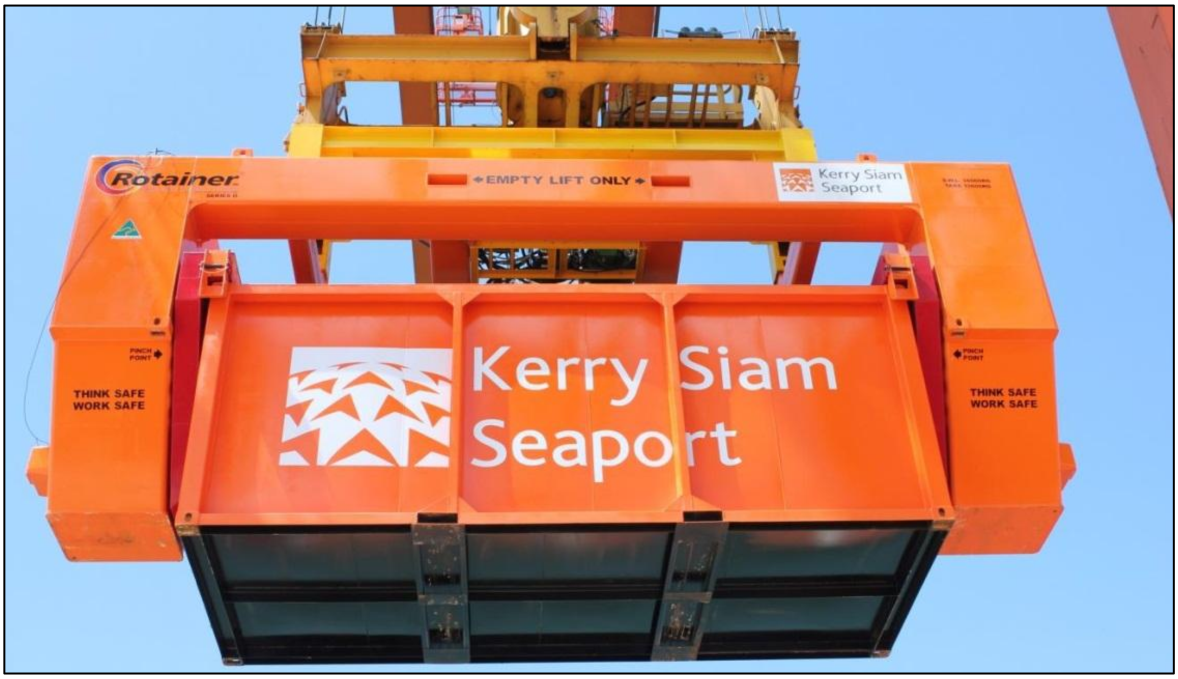
18.10 Project Construction Logistcs

All construction equipment, tools and loads will be brought through the port of Massawa and transported by road the 130 km to site. The larger portion of the logistics will be transported from country of origin by means of standard containers. Large equipment, steelwork, mobile equipment and similar will be handled by break bulk. Abnormal loads will require special trailers for the transporting on the main roads to site.

18.11 Project Concentrate Logistcs

The Emba Derho flotation plant will produce concentrates which will be containerised and transported by road to the port. Specially designed containers and an associated container handling system (rotainer system) will be purchased and utilised for the Project, shown in Figure 18.5. This will enable repeated utilisation of the containers within Eritrea, as well enable the offloading of concentrate directly into the holds of cargo vessels. Through the rotainer mechanism, the concentrate will be transferred to the cargo vessel's hold as break bulk.

Figure 18.5: Rotainer Mechanism



Source: Rotainer

The fully loaded containers transported from site will be stored for two to three weeks at the port before being emptied into the ship's hold by the retainer mechanism. The emptied containers will be transported back to site for the next cycle of loading. The cargo vessels will be specifically chartered for the shipping of the concentrate to markets in Asia, Europe or elsewhere.

19 MARKET STUDIES AND CONTRACTS

SGC has not entered into concentrate sales contracts. Contracts will be negotiated and calculated according to the products specifications, consistent with standard industry practice, and will be similar to such contracts elsewhere in the world.

The concentrate selling terms and conditions applied to economic parameters for project evaluation were supplied by an independent industry consultant. The terms and conditions applied are shown in Section 16, within normal industry operating conditions considering longer term price ranges.

SGC received two budgetary quotations from international gold refiners, based on product specifications and volumes to the refinery, provided by SGC. The costs in the quotations formed the basis of costs applied in economic evaluation for gold and silver production. The principal commodities of gold and silver are freely traded at prices that are widely known, so that prospects for sale of any production are virtually assured.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental and Social Summary

The environmental and socio-economic baseline characterization program is summarized in Table 20.1.

Table 20.1: Summary of Environmental and Socio-Economic Baseline Characterization

DISCIPLINE	PRIMARY DATA COLLECTED OR BEING COLLECTED
Meteorology	Wind speed and direction; temperature; relative humidity; solar radiation, and precipitation.
Air quality	PM10 fractions less than 10 µm (PM10) and less than 2.5 µm (PM2.5) aerodynamic diameter, dust fall, nitrous oxides, nitrogen dioxide (NOX, NO2), and sulphur dioxide (SO2), and carbon monoxide (CO).
Noise	Baseline noise measurement will follow procedures set forth in ISO/DIS 1996-1, 1996-2.2 and ANSI S12.18-1994 (R2009). Summary equivalent noise level (LAeq) profile and statistical sound data (L10, L50, L90, Lpeak, Lmax, and Lmin) will be assessed for daytime and night-time for each monitoring site and reported as required for the study area.
Vibration	A blasting assessment will be conducted to review the current blast design as part of on-going development for the SGC Mine project
Geochemistry	Static and kinetic testwork was carried out in accordance with the methodologies described in the MEND "Prediction manual for drainage chemistry from sulphidic geological materials" (2009). The completed geochemical testwork includes acid-base accounting (ABA), mineralogy, solid-phase metal analysis, and analysis of soluble constituents.
Soils	Classify soil and test for existing contaminants (particularly associated with the existing waste rock stockpiles on site) and nutrient levels.
Surface Hydrology	Hydrographs and corresponding rainfall hydrographs for the major rivers in the project area have been installed and include the Mereb, Gual Mereb, Mesheala and Mai Bela Rivers.
Water Quality	Surface and groundwater quality data is being collected prior to the construction of any project to provide baseline information for further assessment of impacts. Water quality results will be compared to guidelines and standards produced by the WHO (WHO, 2006) for drinking water, agriculture, and livestock consumption
Hydrogeology	Examination of seasonal fluctuations and spatial variations of monthly measured groundwater levels. Characterization of the regional and local study areas geology, including interpretation of aquifer and aquitard locations in the study area. Characterization of the bulk hydraulic conductivity for overburden and bedrock materials and testing protocols. Estimation of the rate and direction of groundwater flow and expected interaction of groundwater with surface water
Aquatics	The study area consists of the Mereb, Gual Mereb, Mesheala and Mai Bela Rivers. Sediment samples were collected at each site and analysed for the following variables: Total Organic Carbon, dry matter, metals. Benthic aquatic macro-invertebrates were sampled in the dominant benthic habitats present at each site, however, due to the very low abundance of individuals caught, it will be not possible to assess for biodiversity and/or abundance. Fish were sampled from various habitats at each site using an AC shocking apparatus.
Vegetation	• Species composition and abundance across the study area • General trends in distribution (if possible) • Species-habitat associations
Wildlife	• Species composition and abundance across the study area • General trends in distribution (if possible) • Species-habitat associations
Socio-Economics	The socio-economic baseline study was started in 2006 through the completion of a household survey and focus group discussions in several local communities. Separate detailed household level socio-economic censuses will be conducted within affected communities.

DISCIPLINE	PRIMARY DATA COLLECTED OR BEING COLLECTED
Worker health and safety	Workforce health conditions will be monitored throughout the lifecycle of the project as part of the health and safety program developed for the mine. SRG intends to conform with the Occupational Health and Safety standards identified in the section 16 of IFC Performance Standard 2 – Labor and Working Conditions (IFC, 2006). Similarly, General International Industry Practices (GIIP) identified in the IFC Environmental, Health, and Safety General Guidelines will be implemented.GIIP included in the project`s Health and Safety Plan will address, communications and training, physical hazards, chemical hazards, biological hazards, radiological hazards, personal protective equipment, special hazard environments and monitoring.
Land use and livelihoods	Produce present land use/land cover maps and to compare with existing land use and land cover maps currently available (e.g. from the Food and Agriculture Association of the United Nations);
Detailed land and asset survey	• Field verification, delineation and official confirmation of village boundaries • Land classification (capability) by image interpretation and then field verification of the land classification maps with working groups • Delineation and inventory of arable land with village farmers/community, including all crops and other productive assets • Delineation and inventory of all other productive land units/resources with community, including eucalyptus trees, rangeland areas, woodlots, etc. • Delineation of residential areas
Livestock	Determine the pattern of land use by livestock (migration corridors) and people (agricultural practices), baseline conditions of livestock management systems, and animal health conditions (parasitology analyses) within the project area and adjacent areas.
Cultural resources and heritage	Three consecutive phase of work, all conducted by the National Museum of Eritrea (NME). Phase I started in 2006 under the direction of GREDMCO through the completion of initial literature reviews, consultation with local communities, and transect walks in the project area. A Phase II study will be subsequently conducted, to conduct follow up assessments of the sites identified during Phase I, by way of test excavations/shovel tests and laboratory analysis. In 2011 and 2012 a Phase III study will be completed by the NME to conduct further excavations at potentially significant archaeological sites identified, prepare an updated archaeological map, and conduct further analysis of artefacts/sites in order to develop appropriate conservation plans. The main objective of the study is to ensure that any sites or artefacts of archaeological and/or cultural heritage significance are identified so that preventive mechanisms can be adopted to guard against any future archaeological site destruction due to the proposed mining activity

20.2 Expected Material Environmental and Social Impacts

A preliminary assessment has been made of the key social and environmental issues of concern utilizing existing baseline information and the mining plan. In addition, general mitigation measures are provided in this section in an effort to preliminarily address these impacts and provide enhancement measures to realize additional benefits from the Project. Key social and environmental issues for the Project area where potential adverse impacts may occur are:

- Water supply and water quality
- Changes to land use (including physical and economic displacement)
- Community health and safety
- Closure and reclamation (end land use)

Potential positive impacts (benefits) resulting from the Project will likely include:

- Employment and business opportunities
- Improved and additional infrastructure & services
- Increased water availability in water storage reservoirs
- Training and educational services related to mining and artisanal trades
- increase in national taxation from royalties, income and corporate taxes

20.2.1 Other Benefits

Other broader benefits resulting from the Project could include:

- Capacity building and institutional strengthening opportunities
- Significant investment in Zoba Maekal, Zoba Debub, and Eritrea with the potential for reinvestment into physical infrastructure and social services
- Increased scientific knowledge and data for the project area
- Provision of goods and services to the Project that will generate new jobs and economic growth in support of industries and spin-off businesses
- Revenue to national government from royalty and income taxes
- Eritrea will be internationally recognized as having a world class copper/zinc mine and as a supplier to external markets

The communities in the areas of all of the project components are likely to have elevated expectations about the benefits they may receive from this development. It is important that these expectations are managed carefully, requiring a comprehensive stakeholder engagement process.

20.3 Administrative, Policy and Legal Requirements

20.3.1 Applicable Eritrean Laws and Regulations

SGC will comply with Eritrean regulatory requirements and continue to monitor the implementation of other laws, regulations and decrees that are currently in draft form. The Eritrean Government has the following key regulatory requirements for mining projects that relate to environmental and social issues:

- National Environmental Assessment Procedures & Guidelines (NEAPG 1999)
- Mineral Resources Proclamation (Proclamation No. 68/1995) and Mining Operations Regulation (Legal Notice 19/1995)
- Eritrean National Mining Corporation (ENAMCO) Establishment Proclamation (Proclamation No. 157/2006)
- Land Proclamation (Proclamation No. 58/1994) and associated legislation
- Eritrean Port Regulations (Legal Notice 46/2000)
- Road Traffic Proclamation (Proclamation No. 154/2006)
- Eritrean Water Proclamation (Proclamation No. 162/2010)
- Forestry and Wildlife Conservation and Development Proclamation (Proclamation No. 155/2006)
- Labour Proclamation of Eritrea (Proclamation No. 118/2001)
- Proclamation to Provide for the Registration of Foreigners Who Reside, Work or Engage in Business in Eritrea (Proclamation No. 127/2002)
- Regulation to Issue Work Permit to Non-Nationals (Legal Notice No. 80/2003)

Although there is no cultural heritage proclamation passed into law, a draft National Heritage Law exists and efforts are being made to pass it into legislation. In Appendix B, Section C of the NEAPG, Areas of Potential Cultural Heritage are included as part of a proposed Environmental Sensitive Area (ESA) list. Project screening includes an assessment on whether the Project will affect one of these ESAs, as well as consulting with the National Museum of Eritrea.

20.3.2 International Guidelines and Standards

In order for development projects, to meet the World Bank and International Finance Corporation (IFC) guidelines they must address sustainability, biodiversity, and the Precautionary Principle in the project design and operation. The IFC, the private sector arm of the World Bank Group, adopted new policies on disclosure and sustainability, and eight Performance Standards first issued in 2006, which were revised on January 1, 2012. The standards that apply to the Project include:

- PS1: Social and Environmental Assessment and Management System
- PS2: Labour and Working Conditions
- PS3: Pollution Prevention and Abatement
- PS4: Community Health, Safety and Security
- PS5: Land Acquisition and Involuntary Resettlement
- PS6: Biodiversity, Conservation and Sustainable Natural Resource Management
- PS7: Indigenous Peoples
- PS8: Cultural Heritage

In addition to the Performance Standards, The IFC Environmental Health and Safety Guidelines will be used as a technical reference to identify and incorporate Good International Industry Practices. Specifically the mining sector guidelines will be referred to for environmental, occupational health and safety, community health and safety, construction and decommissioning considerations.

The Equator Principles were established in 2003 as a set of guidelines developed by a group of international lenders for managing social and environmental issues related to the financing of development projects in the extractive industries sector. The Principles are based on the policies and performance standards of the WBG, both of which are evolving to continually improve the environmental and social expectations and performance of its participants. The Equator Principles categorize the risk of a project in accordance with internal guidelines based upon the environmental and social screening criteria of the WBG. Specifically, the IFC criteria are referenced. For all Category “A” and Category “B” projects, the Project sponsor or “borrower” is required to complete an Environmental Assessment (EA).

The EA is the process of assessing all aspects of the Project’s relationship to the environment (human, biophysical, physical) and developing plans to avoid/minimize or manage adverse impacts. Each assessment as well as the final EA document package requires a variety of disclosure and consultation activities. The Equator Principles state that the EA should address “participation of affected parties in the design, review and implementation of the Project.” For Category “A” projects, the EA report is typically an SEIA. The EA is also required to address compliance with applicable host country laws, regulations, and permits required by the Project. The Project would be considered a Category “A” project under the Equator Principles. It is the intention of SGC to meet the Equator Principles.

20.3.3 Permitting Status

SGC submitted a draft Terms of Reference (TOR) to the Department of Environment in quarter 1 2013. Once the Impact Review Committee has been formed, it will work with SGC to finalize the TOR. SGC will continue to conduct baseline characterization and impact assessment in order to submit the draft SEIA in quarter 3 of 2013.

20.3.4 Stakeholder and Community Engagement

In conformance with IFC Performance Standards, public/community consultation will be conducted and documented through public meetings, workshops, focus group discussions, and through a Project Information Centre at relevant locations in the Project areas. One PIC is already in place at Debarwa and others will be located near the northern properties Project area. The communities that will participate in the consultation process include those that will be in the Project area of influence, as well as regional and national stakeholders as relevant.

The respective Sub-Zobas and Zobas will also be included in the consultation process. The consultation activities are being coordinated and conducted through the SGC community relations team operating out of the Asmara office and the Project Information Centres, in conjunction with the Zoba, Sub-Zoba and National Government officials and organizations.

International best practice also requires the preparation of a Stakeholder Engagement Plan (SEP) for large industrial projects, which are expected to have multiple and far-reaching effects.

SGC has been consulting with the local, regional and national stakeholders formally since 2008 and informally since they acquired the properties in 2006. More recently, extensive consultation has taken place for the Debarwa deposit area. In the upcoming months, SGC plans to meet with all levels of government and local communities to discuss the results of the FS and consult them directly on issues that will affect them. The results of the consultation conducted for the SEIA will be fully documented, and the proposed future consultation will be described in a final SEP.

20.3.5 Worker Health and Safety

It is anticipated that the workforce will be sourced mostly from Eritrea, including those from the immediate project areas, the region and nationally, as well expatriates where specialized skills are required that are not present in Eritrea. Workforce health conditions will be monitored throughout the lifecycle of the Project as part of the health and safety program developed for the mine. SGC intends to conform to the Occupational Health and Safety standards identified in Section 23 of IFC Performance Standard 2 – Labor and Working Conditions (IFC, 2012). Similarly, General International Industry Practices (GIIP) identified in the IFC Environmental, Health, and Safety General Guidelines will be implemented.

20.4 SGC Corporate Policies

SGC has the following corporate policies in place:

- Safety
- Environmental
- Social

20.5 Conceptual Closure Plans

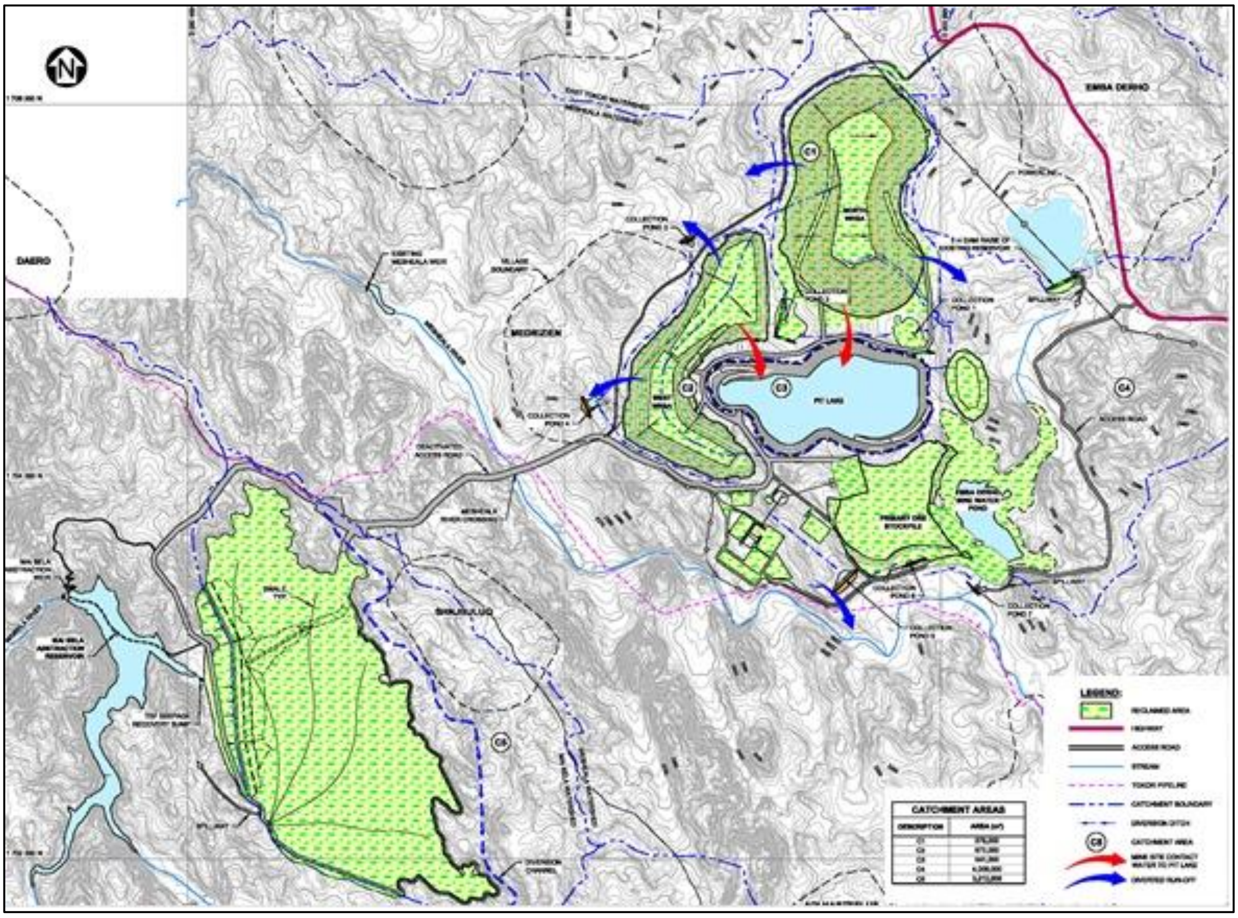
A conceptual closure plan has been developed for the Project. The general sequence for reclamation and closure of the various sites is summarized in **Error! Reference source not ound.**.

Table 20.2: Conceptual Closure and Reclamation

Mine Area	Progressive Reclamation	Final Reclamation and Closure
Debarwa	Debarwa Mine Water Pond dam embankment, West WRSA and East WRSA in Years 3 and 4	Year 4
Gupo	Progressive reclamation not required due to short mine life	Year 5
Adi Nefas	Progressive reclamation consisting of backfilling the underground works on-going through Years 7-10	Year 11
Emba Derho	Emba Derho Mine Water Pond embankment, West WRSA and North WRSA in Years 11 to 17	Year 18 and 19
TSF	TSF embankments in Years 8 to 17	Year 18 and 19

Certain post-closure monitoring activities, such as water quality monitoring and social and environmental studies will continue for approximately five years post-closure. Closure layouts for Emba Derho and the TSF and Debarwa are illustrated in Figure 20.1 and Figure 20.2 respectively.

Figure 20.1: Emba Derho Conceptual Closure



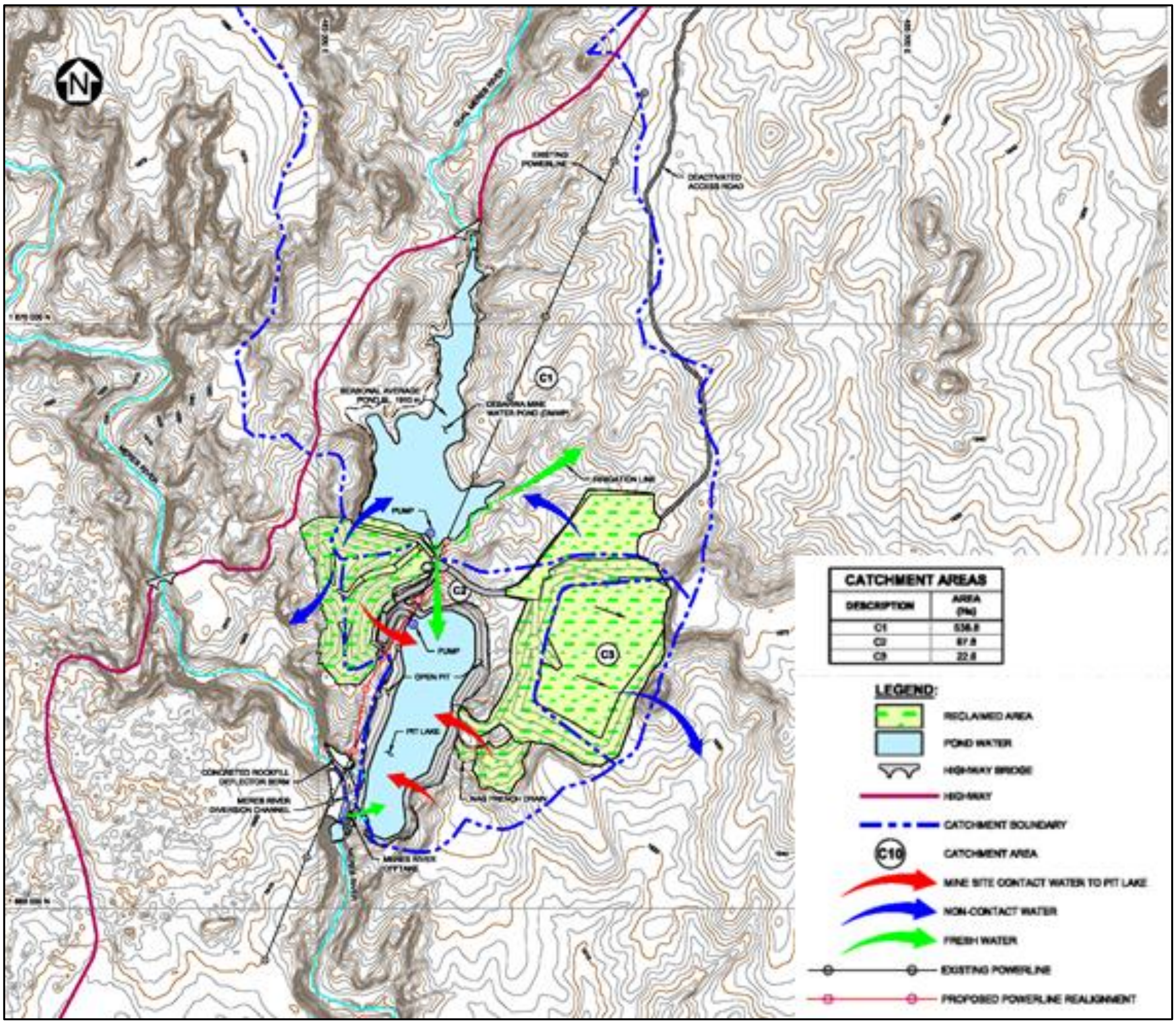
Source: KP

20.5.1 Closure Design Objectives

The conceptual closure plan is a key component of the FS and the overall project development strategy. The main closure objectives for the Asmara Project are to:

- Facilitate progressive reclamation
- Provide a sound environmental closure design
- Adequately address public safety concerns and environmental risks
- Preserve groundwater and surface water quality downstream of the decommissioned mines
- Ensure the long term stability and integrity of engineered structures
- Decommission, remove and/or properly store mine equipment not required after closure
- Grade, scarify and re-vegetate disturbed surfaces and return drainages to their natural
- Integrate disturbed areas into the surrounding landscape and restore the natural appearance of the site
- Return as much of the land as possible to the pre-mining level of productivity or higher

Figure 20.2: Debarwa Conceptual Closure



Source: KP

21 CAPITAL AND OPERATING COSTS

21.1 Background

21.1.1 Introduction

The purpose of the capital and operating cost estimate is to provide costs to an accuracy level of ±15% to be utilized to assess the economics of the Asmara Project at a nominal production rate of 1.4 Mtpa for a gold heap leach plant and 2.0 Mtpa expanded to 4.0 Mtpa for a flotation plant for copper and zinc.

21.1.2 Scope of the Estimate

The capital and operating cost estimate was developed for conventional open pit mining for the Debarwa, Emba Derho and Gupo pit, and underground mining for Adi Nefas. The first phase of operation is a heap leach process plant with supporting infrastructure, capable of treating 1.4 Mtpa oxide ore as well as 0.116 Mtpa direct shipping ore. The first phase will operate from year 1 to 4. The second phase of operation is a copper flotation process plant, treating 2.0 Mtpa of supergene ore from year 3 to 4. The third phase of operation is an expansion of the 2.0 Mtpa, adding a zinc flotation process plant, to treat 4.0 Mtpa primary ore from year 4 to 16.

21.1.3 Responsibilities

The responsibilities for the estimate are listed below, but in broad terms the following applied:

- Mining – SNOWDEN
- Plant and Infrastructure – SENET
- Water Management and Water Supply Infrastructure – KP & SENET
- Tailings Storage Facility – KP & SENET
- Closure – KP

21.1.4 Estimate Accuracy

The level of accuracy of the operating cost estimate is within ±15% of the overall Project costs as of the second quarter 2013 and does not include any escalation factors.

21.1.5 Escalation

No provision has been made for escalation in the process plant and infrastructure portion of the estimate. EPCM contractor’s rates reflect rates expected in the second quarter of 2013.

Escalation has been included for the mining scope of work costs that was sourced from database/in-house information older than 3 months.

21.1.6 Exchange Rates

The cost estimates are in United State dollars (US$). The exchange rates utilized are shown in Table 21.1.

Table 21.1: Exchange Rates

Exchange rates	ROE
US$ per EUR	1.25
ZAR per GBP	13.60
ZAR per EUR	10.63
ZAR per US$	8.50
ZAR per CA$	8.50
ZAR per AU$	8.93
ZAR per ERN	1.76

21.1.7 Exclusions

The following were not included in this estimate:

- Infrastructure costs associated with upgrades to any existing harbour, rail or road facilities
- Escalation beyond the 2nd quarter of 2013
- Scope additions or changes
- Financing costs
- Sunk costs
- Schedule delays such as those caused by:
    - Scope changes
    - Unidentified ground conditions
    - Labour disputes
- Receipt of information beyond the control of the EPCM contractors
- Environmental permitting activities
- Currency fluctuations
- Taxes and duties
- Force majeure
- Permits

21.2 Life of Mine Capital and Operating Cost Summaries

21.2.1 Life of Mine Capital Costs

The LOM capital costs for all aspects of the Project, including mining, process plant, infrastructure, tailings storage, water ponds and dams, and closure are shown in Table 21.2.

Table 21.2: Life of Mine Capital Costs

	Phase I ($M)	Phase II ($M)	Phase III ($M)	Total ($M)
Pre-strip mining and mining equipment [20]	0	116.0	0	116.0
Phase I Plant and Equipment	49.5	0	0	49.5
Copper circuit facility	0	113.8	0	113.8
Zinc circuit facility	0	0	22.8	22.8
Site development, utilities and facilities	3.8	55.5	5.5	64.8
Water management and supply facilities	0.04	19.4	0	19.44
Tailings facilities	11.2	18.3	0.2	29.7
Debarwa facilities	0	9.8	0	9.8
Adi Nefas facilities	0	3.2	0	3.2
Gupo facilities	1.1	0	0	1.1
Adi Nefas development	0	17.0	17.1	34.1
EPCM costs	4.1	29.8	5.2	39.1
First fills (fuel, reagents)	0.03	1.7	0	1.73
Owner's costs	1.0	22.7	0	23.7
Contingency	5.5	21	3.6	30.1
SUBTOTALS	**76.3**	**428.2**	**54.4**	**558.9**
Sustaining Costs				**56.0**
Social Costs				**14.8**
Closure Costs				**36.6**
TOTAL				**666.3**

21.2.2 Life of Mine Operating Costs

The Asmara Project's annual operating costs of US$29.19/t for the life of the mine were determined for mining, processing, general & administration (G&A) and assaying. These are described in detail in the sections that follow, and a summary is shown in Table 21.3.

[20] Includes all mining costs incurred until copper ore is mined (quarter 5). This excludes HL & DSO operating costs.

Table 21.3: Overall Costs Life of Mine

	Unit	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	LOM
Mining Operating Costs																			
Total Mining Operating costs	$M	37.57	53.58	61.96	71.95	78.31	76.61	65.96	59.38	55.22	42.48	41.21	31.70	23.81	12.40	6.25	6.31	1.62	**726**
Total Mining Operating costs	$/t	107.36	35.34	22.77	22.55	19.58	19.15	16.49	14.85	13.81	10.62	10.30	7.92	5.95	3.10	1.56	1.58	2.01	**12.8**
Process Plant Operating Costs																			
Consumable Costs	$/t	3.04	2.88	3.86	5.62	5.10	4.27	4.14	3.76	3.63	3.63	3.63	3.63	3.63	3.63	3.63	3.63	3.63	**3.92**
Power Costs	$/t	2.13	2.04	8.48	7.15	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	**7.11**
Maintenance Parts & Spares Costs	$/t	0.55	0.58	1.06	0.86	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	**0.83**
Plant Labour Costs	$/t	2.10	1.94	3.77	3.21	1.88	1.36	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.48	**1.52**
Total Process Plant Costs	$/t	7.81	7.44	17.16	16.84	15.02	13.67	13.38	12.99	12.86	12.86	12.86	12.86	12.86	12.86	12.86	12.86	13.15	**13.4**
G&A Costs																			
G & A Costs	$M	2.64	8.08	9.86	9.77	9.12	7.83	7.83	7.83	7.83	7.75	7.73	7.70	7.53	7.30	7.10	7.10	1.78	**125**
G & A Costs	$/t	7.55	5.33	3.62	3.06	2.28	1.96	1.96	1.96	1.96	1.94	1.93	1.93	1.88	1.83	1.78	1.78	2.20	**2.21**
Assay Costs																			
Assay Costs	$M	0.32	1.29	1.59	1.76	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	0.45	**27.1**
Total Assay Costs	$/t	0.92	0.85	0.59	0.55	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.56	**0.48**
TSF & Water Management Costs																			
TSF & Water Operating Costs	$/t	-	0.11	0.18	0.18	0.12	0.11	0.10	0.14	0.13	0.17	0.21	0.17	0.25	0.17	0.17	0.17	0.64	**0.17**
Tokor Water Purchase	$/t	1.07	1.05	0.20	0.07	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	**0.12**
Total TSF & Water Operating Costs	$/t	1.07	1.16	0.38	0.25	0.20	0.20	0.19	0.22	0.22	0.26	0.30	0.26	0.34	0.25	0.26	0.26	0.73	**0.29**
Total Operating Costs	**$/t**	**117.16**	**44.78**	**44.52**	**43.26**	**37.54**	**35.43**	**32.46**	**30.47**	**29.30**	**26.13**	**25.85**	**23.42**	**21.49**	**18.49**	**16.91**	**16.92**	**18.65**	**29.19**

21.3 Capital Cost Estimate

21.3.1 Mining Costs

21.3.1.1 Equipment Selection Basis

There are two main fleets for mining, a bulk mining short haul fleet and a selective mining / long haul fleet. The assigment of fleets is shown in Table 21.4.

Table 21.4: Fleet Assignments for Mining Activities

Activity	Fleet
Emba Derho mining (ore and waste)	Bulk
Debarwa mining (ore and waste)	Bulk/Selective
Gupo mining (ore and waste)	Selective
Debarwa ore transport	Selective
Gupo ore transport	Selective
Adi Nefas ore transport	Selective
Emba Derho stockpile reclaim	Bulk
Top soil removal and placement	Bulk/Ancillary
ROM loading	Ancillary

The primary pieces of equipment are provided in Table 21.5.

Table 21.5: Primary Mining Equipment

Function	Bulk fleet	Selective fleet	ROM loading
Loader	CAT 6018/CAT 990	CAT 6015/CAT 988	CAT 988
Haulage	Cat 777G	Renault Kerax	
Drill	Atlas Copco ROC F9	Atlas Copco ROC F9	

In addition, ancillary equipment is allocated as shown in Table 21.6.

Table 21.6: Ancillary Equipment

Function	Model	Tasks
Excavator	CAT 6015	Clean up batter, back up loader
Rock breaker	Cat 319L	Rock breaking
Track Dozer	Cat D10R	
Wheel dozer	Cat 834H	
Wheel loader	Cat 990H	
Wheel loader	Cat 988H	Back up loader
Water truck	Cat 777G WT	Dust suppression
Grader	Cat 16M	Road maintenance
Roller	Cat CS74	Road maintenance
Refueller	Renault Kerax	Equipment refuelling
Forklift	Cat DP 45N1	

21.3.1.2 Fleet Equipment Requirements

The minimum equipment requirements (loading, hauling and drilling) were determined on the basis of calculated productivities and operating hours. Requirements were then altered to smooth the fleet, with a subsequently adjusted utilization, summarised in Table 21.7.

Table 21.7: Primary Equipment Requirements

Year	CAT 6018 shovel	CAT 6015 excavator	Cat 988 FEL	Cat 990H FEL	Cat 777G truck	Renault Kerax tipper	AC DM30 drill
1	3	2	1	2	12	9	2
2	3	2	1	2	14	13	3
3	3	2	2	2	14	13	3
4	3	2	2	2	15	11	3
5	3	2	2	2	15	3	3
6	3	2	2	2	15	3	3
7	3	1	2	2	15	2	2
8	3	1	2	2	15	2	2
9	3	1	2	2	15	1	1
10	3	1	2	2	15		
11	3	1	2	2	15		
12	2	1	2	2	12		
13	1	1	2	2	10		
14	1	1	1	2	6		
15				1	2		
16				1	2		
17				1	2		

21.3.1.3 Ancillary

Requirments for ancillary equipment were developed on the basis of a ratio to the load, haul and drill fleet size, summarised in Table 21.8. Provision was made to be able to mine from two deposits simultaneously (separated by a distance).

Table 21.8: Ancillary Equipment Requirements

Year	Cat D10T track dozer	Cat 834 wheel dozer	Cat 777G water truck	Renault Kerax refueller	Cat CS74 roller	Cat 16M grader	Cat 319L rock breaker	Cat 966H	Cat DP 45N1 forklift
1	2	2	2	2	1	2	-	1	2
2	2	2	2	2	1	2	-	1	2
3	2	2	2	2	1	2	1	1	2
4	2	2	2	2	1	2	1	1	2
5	2	1	2	2	1	2	1	1	2
6	2	1	2	2	1	2	1	1	2
7	2	1	2	1	1	2	1	1	2
8	2	1	2	1	1	2	1	1	2
9	2	1	2	1	1	2	1	1	1
10	2	1	2	1	1	2	1	1	1
11	2	1	2	1	1	2	1	1	1
12	2	1	2	1	1	2	1	1	1
13	2	1	2	1	1	1	1	1	1
14	2	1	2	1	1	1	1	1	1
15	2	1	2	1	1	1	1	1	1
16	2	1	2	1	1	1	1	1	1
17	2	1	1	1	1	1	1	1	1

21.3.1.4 Equipment Purchase Schedule

Equipment is purchased on an “as needs” basis, occurring at the commencement of mining as well as when the fleet expands. A schedule of the purchase of equipment is shown in Table 21.9, and ancillary equipment purchases are shown in Table 21.10.

Table 21.9: Equipment Purchase (load, haul, drill)

Quarter	Cat 6018	Cat 6015	Cat 777G	DM30	Cat 990	Cat 988
1	2	1	4	2	2	1
2	1	1	6	1	-	-
3	-	-	2	-	-	-
4	-	-		-	-	-
5	-	-		-	-	-
6	-	-	1	-	-	-
7	-	-	1	-	-	-
8	-	-	-	-	-	-
9	-	-	-	-	-	-
15	-	-	1	-	-	-

Table 21.10: Equipment Purchases – Ancillary Equipment

Quarter	Cat D10T	Cat 834	Cat 319	Cat CS74	Cat 16M	Cat 777G WT	Cat 966H TH	Renault Kerax refueler	Cat DP 45N1
1	1	1		1	1	1	1	1	1
2	1	-	-	-	-	-	-	1	1
3	-	-	-	-	1	1			
4	-	-	-	-	-	-	-	-	-
5	-	-	-	-	-	-	-	-	-
6	-	-	-	-	-	-	-	-	-
7	-	-	-	-	-	-	-	-	-
8	-	1	-	-	-	-	-	-	-
9	-	-	1	-	-	-	-	-	-

21.3.1.5 Equipment Capital Costs

Unit capital costs were sourced from vendor quotations. In addition to these quotes the following costs were added in proportion to the purchase price:

- 4.0% for shipping
- 0.2% for land transport from Massawa to site
- 5.0% for optional accessories and upgrades
- 0.5% for training operators
- 0.5% for local taxes and duties
- 8.0% for first fills and spares
- 10.0% for contingency

Total capital costs are shown in Table 21.11.

Table 21.11: Equipment Capital Costs

Equipment	Quantity	Unit price ($M)	Total cost ($M)
CAT 6018 face shovel	3	3.4	10.2
CAT 6015 excavator	2	2.1	4.2
Cat 777G truck	15	2.2	33.0
AC DM30 drill	3	1.5	4.6
Renault Kerax refueller	2	0.2	0.4
Cat 988 wheel loader	2	1.3	2.6
Cat 990H wheel loader	2	1.9	3.7
Cat D10T track dozer	2	1.9	3.7
Cat 777G water truck	2	2.4	4.8
Cat 834 wheel dozer	2	1.4	2.9
Cat 16M grader	2	1.1	2.2
Cat CS74 roller	1	0.2	0.2
Cat 966H tire handler	1	0.6	0.6
Cat 319L rock breaker	1	0.4	0.4
Cat DP 45N1 forklift	2	0.1	0.2
Total	**42**		**73.9**

21.3.1.7 Equipment Replacement Schedule

No equipment replacement is required over the life of the operation, as the mining rate decreases after Year 8. However, key equipment items will be rebuilt beyond Year 10. In total, $20.3 M has been budgeted for this between Year 11 and Year 12.

21.3.1.8 Non-Equipment Capital Costs

Certain one-off or replacement costs are considered to be capital costs even though they do not relate to mobile or specific equipment. These items include the Emba Derho wireless fleet management system for installation on trucks, lighting plant purchases, and mining technical software and survey equipment purchases. These costs are detailed in Table 21.12 below.

Table 21.12: Mining General Capital for LOM

Item	Life of mine cost ($M)
Software and survey gear	0.4
Lighting plants	0.2
Fleet management system	1.9
Total	**2.6**

21.3.1.9 Sustaining Capital

An amount of $3.6 M is forecast for sustaining capital over the life of the mine.

21.3.1.10 Capital Cost Schedule

A schedule of how the capital costs are distributed across the project is shown in Table 21.13. There are no capital purchases after quarter 20. Since the delivery time on most equipment is less than three months, capital costs have been applied in the quarter in which the machine starts work.

Table 21.13 Initial Capital Cost Schedule

Quarter	Equipment capital ($M)	Non-equipment capital ($M)	Total| ($M)
1	33.8	1.6	35.5
2	22.6	0.2	22.8
3	8.0	0.2	8.2
4	-	0.2	0.2
5	-	0.2	0.2
6	2.2	0.1	2.3
7	2.2	0.1	2.3
8	1.4	0.2	1.6
9	1.6	0.1	1.7
15	2.2	0.1	2.3
Total	**73.9**	**2.6**	**77.0**

21.3.2 Process Plant and Infrastructure

21.3.2.1 Process Plant Costs

A summary of the capital costs for the heap leach and flotation plants for all three phases is shown in Table 21.14.

Table 21.14: Process Plant Capital Costs

Costs	Phase I ($M)	Phase II ($M)	Phase III ($M)	Total ($M)
Mechanicals	12.30	32.50	11.14	55.94
Earthworks	3.68	16.32	-	19.99
Heap Leach Pads	21.36	-	-	21.36
Civils	2.06	8.28	1.21	11.56
Platework	0.46	0.65	0.38	1.50
Structural	1.36	4.54	1.70	7.60
Piping	1.02	3.56	1.16	5.74
Instrumentation	0.61	5.08	0.79	6.48
Electricals	1.94	14.45	2.08	18.48
Vendor Services	0.39	1.05	0.24	1.68
First Fill	0.03	1.74	-	1.77
Transport	2.52	10.73	1.14	14.39
Plant Infrastructure	0.27	4.11	-	4.38
Direct Costs	**48.00**	**103.01**	**19.84**	**170.86**
Spares	-	11.70	-	11.70
Buildings	-	2.02	-	2.02
Fencing	-	0.15	-	0.15
Mobile Equipment	0.78	3.16	-	3.94
Power Plant	2.04	17.31	4.62	23.97
Fuel Farm	0.14	6.69	-	6.83
Access Roads	-	8.62	-	8.62
Transport	-	4.38	-	4.38
Indirect Costs	**2.96**	**54.02**	**4.62**	**61.60**
Installation & EPCM	9.66	55.60	8.67	73.92
Installation & EPCM	**9.66**	**55.60**	**8.67**	**73.92**
Total	**60.62**	**212.63**	**33.12**	**306.38**

21.4 Operating Cost Estimate

21.4.1 Mining Operating Costs

21.4.1.1 Mining Equipment Supplier Preference

No specific equipment manufacturer is recommended. Where particular models of equipment have been nominated, these should be considered to be representative of an equipment class only (size and productivity).

21.4.1.2 Open Pit Mining Operating Cost Schedule

The total open pit mining operating cost schedule is shown in Table 21.15.

Table 21.15: Open Pit Operating Cost Schedule

Year	Operator cost ($M)	Maintenance cost ($M)	Fuel cost ($M)	Lease cost ($M)	Explosives cost ($M)	Contractor margin ($M)	Other ($M)	Total cost ($M)
1	0.6	8.8	6.6	0.0	3.9	0.0	5.8	25.8
2	1.4	18.3	15.1	0.4	7.9	0.4	8.1	51.6
3	1.6	20.9	16.9	0.5	5.7	0.4	8.4	54.4
4	1.4	19.0	15.6	0.4	4.7	0.3	7.9	49.3
5	1.1	17.9	15.6	0.1	4.1	0.1	7.5	46.4
6	1.2	20.0	18.9	0.1	4.8	0.1	7.1	52.1
7	1.0	16.9	16.6	0.1	3.8	0.0	7.0	45.4
8	1.0	16.8	15.5	0.0	3.5	0.0	7.0	43.9
9	0.9	15.2	13.9	0.0	2.1	0.0	7.1	39.2
10	0.9	15.2	14.7	0.0	3.2	0.0	6.6	40.7
11	0.9	16.1	16.1	-	3.7	-	6.4	43.2
12	0.7	11.9	12.4	-	2.4	-	6.1	33.5
13	0.5	8.8	9.3	-	1.4	-	5.3	25.3
14	0.3	5.9	5.5	-	0.6	-	4.2	16.5
15	0.1	2.4	1.5	-	-	-	2.2	6.2
16	0.1	2.4	1.5	-	-	-	2.2	6.2
17	0.1	1.3	0.8	-	-	-	1.0	3.2
Total	**13.8**	**217.9**	**196.5**	**1.5**	**51.7**	**1.4**	**99.8**	**582.7**

21.4.1.3 Underground Mining Cost

The underground mine at Adi-Nefas is planned to be a contractor operation. Underground mining capital and operating costs were estimated, and a contractor margin was then added to the costs. Underground costs also include some surface works and infrastructure put in place by SGC.

Underground General Assumptions

Some general assumptions drive the underground cost model as detailed in Table 21.16.

Table 21.16: Underground Mining Cost Model General Assumptions

Item	Value	Unit
Equipment capital cost interest	12.0%	%
Contractor margin - labour	21%	%
Contractor margin - other	21%	%
Other ownership costs	3.0%	%
Fuel cost	1.10	$/L
Oils/lubes cost	6.50	$/L
Electrical power cost	250	$/MWhr

Underground operating costs are derived from the physicals of the underground mine and assumptions with respect to the number and cost of running and manning the equipment as well as assumptions about the consumption rate of and the cost of the various consumables. The operating costs are also built up from estimates of fleet requirements (based on the schedules) and assumptions with respect to the productivity of the fleet.

The total underground mining operating cost schedule is shown in Table 21.17.

Table 21.17: Underground Operating Cost Schedule

	m	dmt Ore	$/t	dmt (other)	$/t or $/m	Value ($M)
Lateral development	8 982	-	-	-	2 207	19.82
Vertical development	496	-	-	-	13 595	6.75
Stoping	-	1 364 735	12.58	-	-	17.17
Backfilling	-	1 364 735	4.11	899 688	6.24	5.61
Load and haul	-	1 681 468	2.14	2 107 696	1.71	3.60
Mine services	-	1 681 468	-	-	-	12.87
Labour	-	1 681 468	-	-	-	92.30
Contractor corporate overhead	-	1 681 468	-	-	-	8.22
Contractor mobilization and demobilization	-	1 681 468	-	-	-	8.22
Ancillary (light vehcles)	-	1 681 468	-	-	-	4.07
Operating Costs	**-**	**1 681 468**	**104.75**	**-**	**-**	**176.10**
Capital costs	**-**	**1 681 468**	**0.91**	**-**	**-**	**1.54**
Total	**-**	**1 681 468**	**105.64**	**-**	**-**	**177.64**

21.4.2 Processing Plant Operating Costs

21.4.2.1 Summary

The average annual process plant operating costs for the life of the mine are US$13.38/t of ore processed. These are summarized in Table 21.19.

21.4.2.2 Basis of Estimate

The plant operating costs were compiled from a variety of sources:

- First principles, where applicable
- Supplier quotations on reagents and consumables
- SENET's experience on similar operational plants.

The major cost elements of the plant whose estimates were made were:

- Reagents and consumables
- Power
- Maintenance supplies
- Process plant operating & maintenance labour

Operating costs for the different ores; heap leach, DSO ore, supergene ore and primary ore are summarized in Table 21.18.

Table 21.18: Individual Ore Operating Costs

Cost Summary	Oxide Debarwa ($/t)	Transition Debarwa ($/t)	Oxide Gupo ($/t)	Oxide Emba Derho ($/t)	DSO Ore ($/t)	Supergene Ore ($/t)	Primary Ore ($/t)
Consumable Costs	2.91	3.37	3.02	2.32	0.09	4.19	4.04
Power Costs	2.13	2.13	2.13	2.13	0.94	11.16	7.22
Maintenance Costs	0.55	0.55	0.55	0.55	1.04	1.28	0.82
Plant Labour Costs	2.10	2.10	2.10	2.10	2.10	3.82	1.32
Process Costs	**7.68**	**8.14**	**7.80**	**7.09**	**4.17**	**20.45**	**13.40**

21.4.3 Other Operating Costs

Over and above the main operating costs of mining and plant operations are the G&A, Assay and TSF and water management costs. The LOM operating costs for these are $2.2/t, $0.48/t and $0.29/t respectively, totaling $2.98/t. The combined LOM operating costs of mining and plant operations together with G&A, Assay and TSF and water management costs totals $29.19/t.

Table 21.19: Overall Process Costs LOM

	Unit	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	LOM
Tonnage Processed																			
Oxide - Emba Derho	kt	262	975	647	2	-	-	-	-	-	-	-	-	-	-	-	-	-	**1 886**
Oxide - Debarwa	kt	55	58	51	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**164**
Oxide - Gupo	kt	-	-	-	399	-	-	-	-	-	-	-	-	-	-	-	-	-	**399**
Transition - Debarwa	kt	33	368	118	3	-	-	-	-	-	-	-	-	-	-	-	-	-	**522**
DSO ore	kt	-	116	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**116**
Supergene ore	kt	-	-	1 905	490	-	-	-	-	-	-	-	-	-	-	-	-	-	**2 395**
Primary ore	kt	-	-	-	2 296	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	805	**51 101**
Combined Tonnage Processed	**ktpa**	**350**	**1 516**	**2 722**	**3 191**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**4 000**	**805**	**56 584**
Process Plant Operating Costs																			
Consumable Costs	$M	1.06	4.36	10.5	17.9	20.4	17.1	16.6	15.0	14.5	14.5	14.5	14.5	14.5	14.5	14.5	14.5	2.92	**222**
Power Costs	$M	0.74	3.09	23.1	22.8	28.9	28.9	28.9	28.9	28.9	28.9	28.9	28.9	28.9	28.9	28.9	28.9	5.81	**402**
Maintenance Parts & Spares Costs	$M	0.19	0.89	2.88	2.74	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	0.66	**46.9**
Plant Labour Costs	$M	0.73	2.94	10.25	10.25	7.53	5.45	4.76	4.76	4.76	4.76	4.76	4.76	4.76	4.76	4.76	4.76	1.19	**85.9**
Total Plant Costs	$M	**2.73**	**11.3**	**46.7**	**53.7**	**60.1**	**54.7**	**53.5**	**52.0**	**51.4**	**51.4**	**51.4**	**51.4**	**51.4**	**51.4**	**51.4**	**51.4**	**10.6**	**757**
Consumable Costs	$/t	3.04	2.88	3.86	5.62	5.10	4.27	4.14	3.76	3.63	3.63	3.63	3.63	3.63	3.63	3.63	3.63	3.63	**3.92**
Power Costs	$/t	2.13	2.04	8.48	7.15	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	7.22	**7.11**
Maintenance Parts & Spares Costs	$/t	0.55	0.58	1.06	0.86	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	**0.83**
Plant Labour Costs	$/t	2.10	1.94	3.77	3.21	1.88	1.36	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.48	**1.52**
Total Plant Costs	**$/t**	**7.81**	**7.44**	**17.16**	**16.84**	**15.02**	**13.67**	**13.38**	**12.99**	**12.86**	**12.86**	**12.86**	**12.86**	**12.86**	**12.86**	**12.86**	**12.86**	**13.15**	**13.38**

22 ECONOMIC ANALYSIS

A cashflow model for the Asmara Project feasibility study was prepared and included mining and processing from four sources (Emba Derho, Adi Nefas, Gupo and Debarwa). Heap leaching producing gold and silver doré commences first and continues while DSO is processed and sold and subsequently flotation recovery of copper and zinc concentrates commences. Two flotation processing plant configurations are included, producing copper and then copper and zinc concentrates until the end of the life of mine (LOM).

The pre-production period for the copper/zinc flotation plant construction is two years and includes 0.75 year pre-production for the heap leach and 1.25 years' preproduction for the commencement of DSO mining. Waste pre-stripping will take place and some mill feed will be mined and stockpiled during this period. Ore will be mined for 14.3 years, with stockpiled ore processed for an additional 2 years. The overall project life is 16.3 years.

The financial analysis was completed using four metal price scenarios shown in Table 22.1.

Table 22.1: Metal Prices for Four Scenarios

	Base Case Prices	Low Copper Metal Price	Low Metal Prices	Current Metal Prices (May 10 2013)
Copper ($/lb)	3.25	3.00	2.75	3.35
Zinc ($/lb)	1.00	1.00	0.80	0.83
Gold ($/oz)	1,400	1,400	1,250	1,449
Silver ($/oz)	25.00	25.00	21.00	24.00

22.1 Mining Reserves

The Mineral Reserves are for 3 open pit mines and one underground mine. The mill feed ore and waste ore will be scheduled quarterly for the life of the project. These are detailed in Table 22.2.

Table 22.2: Mineral Reserves

Rock Type	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Tonnes (kt)
Proven					
Emba Derho Primary	0.9	1.7	0.2	11.6	4,337
Debarwa Oxide	-	-	1.0	6.7	1.0
Debarwa Transition	-	-	4.3	84.1	94
Debarwa Supergene	8.9	0.2	2.2	53.2	423
Debarwa Primary	1.6	2.8	0.6	15.6	6.0
Total Proven Reserves					**4,861**
Probable					
Emba Derho Supergene	1.0	0.4	0.3	14.9	1,200
Emba Derho Primary	0.7	1.6	0.3	9.2	44,497
Debarwa Oxide	-	-	1.6	8.2	163
Debarwa Transition	-	-	2.5	17.0	428
Debarwa Supergene	2.5	0.2	1.0	22.9	888
Debarwa Primary	1.9	4.0	1.1	25.4	514
Adi Nefas Primary	1.6	8.2	2.8	96.5	1,682
Gupo Oxide	-	-	1.9	-	399
Gupo Sulfide	-	-	2.4	-	66
Total Probable Reserves					**51,723**
Total Proven and Probable Reserves					**56,584**

22.2 Processing

The processing of ores is planned to be completed in three separate phases, detailed below.

22.2.1 Phase 1A – DSO (Year 1 – Year 2)

- Mining of 116,000 tonnes of high-grade DSO with an average grade of 15.6% copper, 2.96 g/t gold, and 76.8 g/t silver from Debarwa
- Crushing at Emba Derho and shipping to smelter
- Mine and ship in 6 months

22.2.2 Phase 1B – Heap Leach Gold production – (Year 1 – Year 5)

- Mine and process 3.037 million tonnes near-surface gold "caps" at Debarwa and Emba Derho followed by Gupo
- Process at a gold heap-leaching operation near the Emba Derho deposit at a rate of 1.4 million tonnes per year
- Average grades 1.48 g/t gold , 8.2 g/t silver
- Average recoveries 67% gold, 37% silver

22.2.3 Phase II – Supergene Copper Production (Year 2 – Year 3.25)

- Mine and process 2.4 million tonnes of high-grade copper supergene ore from Debarwa and Emba Derho at a rate of 2 million tonnes per year for 1.25 years
- Average grades 2.25% copper, 0.76 g/t gold, 21.6 g/t silver
- Average recoveries 79% copper, 51% gold, 58% silver
- Copper concentrate – 24.5% copper, 5.6 g/t gold, 183 g/t silver

22.2.4 Phase III - Full Production (Year 3.25 – Year 16.3)

- Mine and process primary copper and zinc ore from Emba Derho, Debarwa, and Adi Nefas at a rate of 4 million tonnes per year for 13 years
- Final waste:ore ratios - 2.5:1 at Emba Derho, 9.8:1 at Debarwa and Gupo at 1.7:1
- Adi Nefas to be mined using underground long hole bench retreat ranging between 36,000 and 160,000 tonnes per year for a total of 1.682 million tonnes mined over 6 years and blended with ore from Emba Derho
- Adi Nefas average mining cost - $86/tonne
- Average grades 0.71% copper, 1.88 zinc, 0.35 g/t gold, 12.4 g/t silver
- Average recoveries 87% copper, 87% zinc, 48% gold, 43% silver
- Copper concentrate – 25% copper, 8.3 g/t gold, 268 g/t silver
- Zinc concentrate – 59% zinc

22.2.5 Operating Costs

The key operating costs are shown in Table 22.3.

Table 22.3: Operating Costs

Item	Unit	Amount
Phase IB mining	$/t ore	2.46
Phase IA,II and III mining	$/t ore	12.82
Phase IB processing, G&A	$/t ore processed	8.68
Phase IA,II and III processing G&A	$/t ore processed	17.64
Copper & Zinc concentrate transport, shipping and port fees	$/t Concentrate	61.5
Total operating cost	$M	1,868
Total operating costs	**$/t ore processed**	**33.01**

22.2.6 Total Metal Production (Life of Mine)

The total LOM metal recovered during the Project is shown in Table 22.4.

Table 22.4: LOM Total Metal Production

Item	Unit	Amount
Copper in concentrate	Millions of pounds	841
Zinc in concentrate	Millions of pounds	1,874
Gold in concentrate	Thousands of ounces	339
Silver in concentrate	Thousands of ounces	10.927
Gold doré from heap leach	Thousands of ounces	97
Silver doré from heap leach	Thousands of ounces	295

22.3 Income

22.3.1 Revenue

22.3.1.1 DSO Sale Terms

The DSO is sold to the smelter which will deduct 1% unit of copper from the grade received, and also levy a Treatment Charge (TC) of \$80.00 per tonne of DSO ore and a copper refining charge (RC) of \$0.08 per pound of copper. The smelter will deduct 1 gram of gold per tonne of DSO and levy a RC of \$15.00 per ounce of gold. The smelter will deduct 30 grams of silver per tonne of DSO and levy a RC of \$2.00 per ounce of silver.

The project provides for the production of a copper concentrate and zinc concentrate which are to be shipped to separate smelters for smelting and refining.

22.3.1.2 Copper Smelter Terms

The smelter will determine payable copper subject to the following calculation. Payment will be made for 97% of the copper, subject to a minimum deduction of 1% unit of copper. For the copper concentrate from Asmara, where the maximum grade of the concentrate is 30%, the smelter will always take the 1% copper unit deduction in order to determine the payable metal. The standard TC and RC charges will be levied to pay for the smelting process.

For the Asmara analysis, a TC of \$70 per tonne of concentrate was applied and \$0.07 per pound of copper for refining. These total deductions from the copper submitted to the smelter, divided by the value of the copper in the concentrate, determines the Net Smelter Return (NSR). For Asmara the NSR ranges from a low of 84.4% to a high of 90.7% for Phase II and III.

22.3.1.3 Zinc Smelter Terms

The smelter will determine payable zinc subject to a calculation. The smelter pays for 85% of the zinc subject to a minimum deduction of 8% units of zinc. For the zinc concentrate from Asmara where the average grade of the concentrate is 58.9% the smelter will always take a 15% overall deduction in order to determine the payable metal. From the value of the payable metal per tonne of concentrate, the smelter will deduct the TC per tonne of concentrate to pay for the smelting process. The smelter will also enter into a price participation agreement where they will reduce the TC if the price of zinc falls below a threshold and increase the TC if the price of zinc increases above a threshold. In the case of Asmara, the threshold was determined to be \$2000 per tonne of zinc (\$0.907 per pound). The escalator was set as 5% above \$2000 and -2% below \$2000.

For the Asmara analysis a TC of \$190 per tonne of concentrate was applied. These total deductions from the zinc submitted to the smelter divided by the value of the zinc in the concentrate determines the NSR. For Asmara the NSR averaged 69.4% for Phase III.

22.3.1.4 Gold in the Copper Concentrate

Gold is generally a precious metal credit in copper concentrates and the value of the gold paid for by the smelter is calculated in a similar method to the copper concentrate. The smelter will determine payable gold subject to a calculation. The smelter pays for 97% of the gold subject to a minimum deduction of 1 gram of gold per tonne of concentrate. For the copper concentrate from Asmara where the grade of gold in the concentrate is variable from 0.9 g/t to 26.4 g/t, the smelter will take the most advantageous method of calculation to determine the payable metal. From the value of the payable metal per tonne of concentrate the smelter will deduct the RC for gold.

For the Asmara analysis a RC of $15 per ounce of gold will be charged by the smelter. These total deductions from the gold submitted to the smelter divided by the value of the gold in the concentrate determines the NSR. For Asmara the NSR ranged from a low of 0% to a high of 94.1% for Phase II and III.

22.3.1.5 Silver in the Copper Concentrate

Silver is generally a precious metal credit in copper concentrates and the value of the silver paid for by the smelter is calculated in a similar method to the copper concentrate. The smelter will determine payable silver subject to a calculation. The smelter pays for 90% of the silver subject to a minimum deduction of 30 grams of silver per tonne of concentrate. For the copper concentrate from Asmara where the grade of silver in the concentrate is variable from 39 g/t to 695 g/t the smelter will take the most advantageous method of calculation to determine the payable metal. From the value of the payable metal per tonne of concentrate the smelter will deduct the RC for silver.

For the Asmara analysis a RC of $2.00 per ounce of silver will be charged by the smelter. These total deductions from the silver submitted to the smelter divided by the value of the silver in the concentrate determines the NSR. For Asmara the NSR ranged from a low of 22.3% to a high of 82.8% for Phase II and III.

The LOM net revenue after NSR for each metal and the total for the project are detailed in Table 22.5.

Table 22.5: LOM Net Revenue

Metal	Base Case Prices ($M)	Low Copper Metal Price ($M)	Low Metal Prices ($M)	Current Metal Prices (May 10, 2013) ($M)
Copper	2,459	2,257	2,055	2,539
Zinc	1,301	1,301	1,005	1,051
Gold	537	537	479	555
Silver	221	221	182	211
Total	**4,517**	**4,315**	**3,721**	**4,356**

22.4 Expenditure

22.4.1 Capital costs

The infrastructure and plant construction costs for each of the phases are presented in Table 22.6.

Table 22.6: Life of Mine Capital Costs

	Phase I $ million	Phase II $ million	Phase III $ million	Total $ million
Pre-strip mining and mining equipment [21]	0	116.0	0	116.0
Phase I Plant and Equipment	49.5	0	0	49.5
Copper circuit facility	0	113.8	0	113.8
Zinc circuit facility	0	0	22.8	22.8
Site development, utilities and facilities	3.8	55.5	5.5	64.8
Water facilities	0.04	19.4	0	19.44
Tailings facilities	11.2	18.3	0.2	29.7
Debarwa facilities	0	9.8	0	9.8
Adi Nefas facilities	0	3.2	0	3.2
Gupo facilities	1.1	0	0	1.1
Adi Nefas development	0	17.0	17.1	34.1
EPCM costs	4.1	29.8	5.2	39.1
First fills (fuel, reagents)	0.03	1.7	0	1.73
Owner's costs	1.0	22.7	0	23.7
Contingency	5.5	21	3.6	30.1
SUBTOTALS	**76.3**	**428.2**	**54.4**	**558.9**
Sustaining Costs				**56.0**
Social Costs				**14.8**
Closure Costs				**36.6**
TOTAL				**666.3**

22.4.2 Royalties

Royalties are paid to the Eritrean government on the basis of 3.5% and 5.0% of the gross value of all base metals and precious metals sold, respectively. The total value of the royalties payable for the Base Case will be $205.8 M.

[21] Includes all mining costs incurred until copper ore is mined (quarter 5). This excludes HL & DSO operating costs.

22.4.3 Operating Costs

On site operating costs average $29.42 per tonne throughout the life of mine. The operating costs for each phase are provided in Table 22.7.

Table 22.7: Average Operating Costs

	Heap-Leach Phase IB	Flotation Phase II & III
Mining $/t ore mined	2.46	13.35
Process $/t ore processed	8.68	17.64
TOTALS	**11.14**	**30.99**

22.5 Financial Analysis

The base case uses constant metal prices of $3.25/lb copper, $1.00/lb zinc, $1,400/oz gold and $25.00/oz silver for the LOM. The pre and post-tax financial results are provided for four different cases as detailed in Table 22.8. All prices are reflected as $ million ($M).

Table 22.8: Financial Results

	Base Case Prices	Low Copper Metal Price	Low Metal Prices	Current Metal Prices May 10 2013
Pre-tax				
NPV @ 10% discount ($M pre-tax)[22]	692	595	309	623
NPV @ 8% discount ($M pre-tax)	837	728	404	758
NPV @ 0% discount ($M pre-tax)	1,791	1,596	1,026	1,638
IRR %[23]	34%	31%	22%	33%
Payback (years)[24]	4.1	4.3	5.1	4.2
Post-tax				
NPV @ 10% discount ($M post- tax)	345	275	69	296
NPV @ 8% discount ($M post- tax)	443	364	131	386
NPV @ 0% discount ($M post-tax)	1,276	1,136	727	1,166
IRR %	27%	24%	17%	26%
Payback (years)	4.6	4.8	5.6	4.7
Metal Prices				
Copper	3.25	3.00	2.75	3.35
Zinc	1.00	1.00	0.80	0.83
Gold	1,400	1,400	1,250	1,449
Silver	25.00	25.00	21.00	24.00

[22] The NPV (Net Present Value) is the total of all the period net cashflows discounted at the nominated rate to the start of the pre-production period

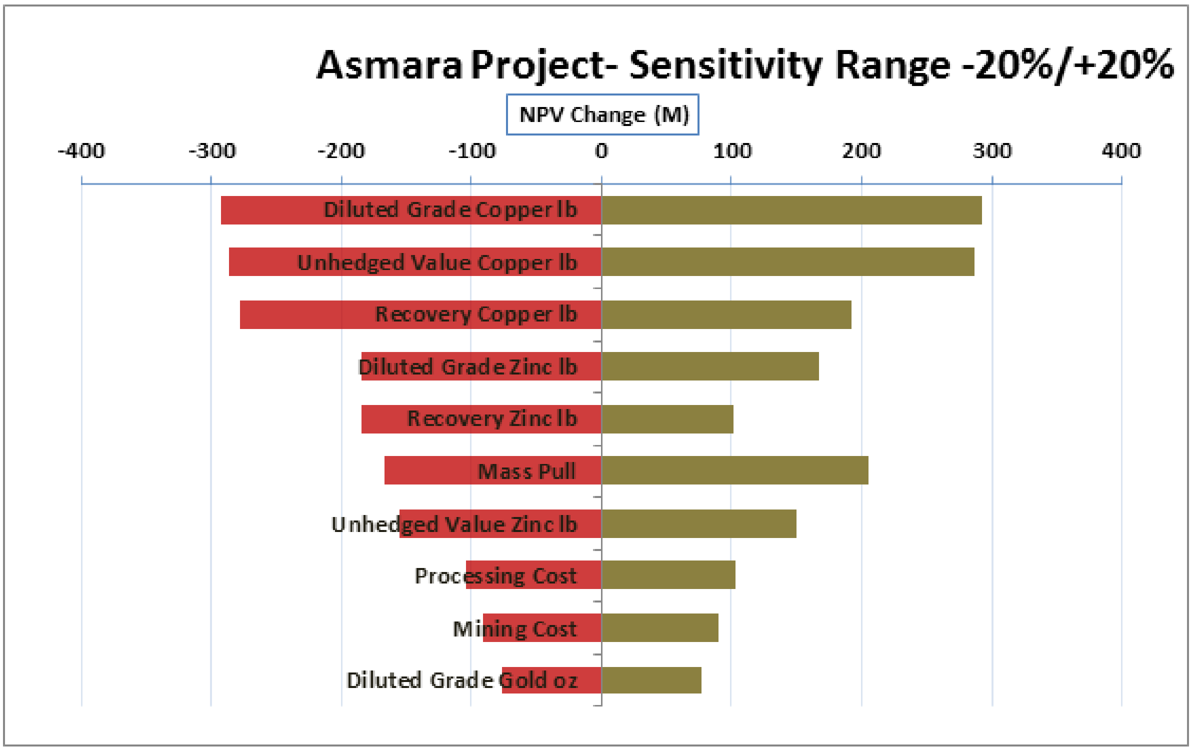
[23] The IRR (Internal Rate of Return) is the discount rate when applied to the NPV calculation brings the NPV of all the periodic net cashflows to zero.

[24] The Payback period is at the time in the production profile that the cumulative negative net cashflow begins to turn positive. From this time forward there will generally be periodic positive cashflows and the total cost of bring the Project into production will be paid back

22.5.1 Sensitivity Analysis

The sensitivity analysis was conducted on the base case to determine how changes to particular selected parameters will impact the NPV. Sensitivity analysis is a process where all of the other parameters are held constant while changing one parameter. This is shown in a tornado graph in Figure 22.1.

Figure 22.1: Sensitivity Range



Source: Snowden

Figure 22.1 shows the range of NPV values for a ±20% change to each of the variables and also ranks the effect to the project that a change of the variable will cause. The variable with the greatest impact is at the top and that with the least impact is at the bottom. Note that the positive side of the bar for recovery may be shorter as the recovery cannot be higher than 100%.

This sensitivity analysis allows for the calculation of the change to the variable that will induce a breakeven NPV while holding the other variables constant. In addition, an elasticity measure is calculated which determines what percentage change in the NPV is induced by a 1% change of the variable. A value above 1% indicates that a change in the variable induces a change greater than the change of the variable in the NPV. Elasticities and breakeven points are shown in Table 22.9 below.

Table 22.9: Sensitivity Analysis

Variable	Breakeven Change	Elasticity
Diluted grade and recovery of copper	-58%	1.7
Diluted grade and recovery of zinc	-79%	1.0
Price of copper	-59%	1.7
Price of zinc		1.2
Pre-production capital		0.9
Production capital		0.2
Mining cost		0.4
Processing cost		0.6
Onsite overheads and concentrate transport		0.1

22.6 Key Performance indicators (KPIs)

22.6.1 KPIs Average Life of Mine

KPIs have been calculated as the average value for the LOM and are presented for the base case in Table 22.10.

Table 22.10: KPIs Average for LOM

Descriptor	Unit	KPI
Pro-rata cash cost copper[25]	$/lb	1.48
Pro-rata cash cost zinc	$/lb	0.46
Pro-rata cash cost gold	$/oz	637
C1 cash cost copper[26]	$/lb	-0.25
Breakeven grade of copper	%	0.33
Breakeven grade of zinc	%	Always positive
Breakeven price of copper[3]	$/lb	1.34
Breakeven price of zinc	$/lb	0.04

[25] The pro-rata cash cost is a KPI used for polymetallic projects such as the Asmara Project as it allocates the operating costs pro-rata to the value of the metal to the project.
[26] C1 cash cost is calculated using the Brooke Hunt methodology. As the Project is a polymetallic project with the bi-product metals having a significant value the C1 cost for copper is negative as the value of bi-products is netted off.

23 ADJACENT PROPERTIES

There is no information from adjacent properties applicable to the Asmara Property for disclosure in this report.

The nearest producing mine is the Bisha mine, operated by Nevsun Resources (TSX:NSU / NYSE MKT NSU). Bisha is located approximately 320 km west of Asmara, and will treat supergene ore and primary mill feed ore to produce copper and zinc concentrates as well as gold and silver.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Implementation

24.1.1 Introduction

The schedule showing the implementation of the Asmara Project, which SGC intends to follow, is shown at the end of this section. The schedule reflects the work required from project initiation to detailed engineering, construction and through to commissioning. The schedule assumes that there is a seamless advancement of the Project between the various phases of its' evolution. It is recognized that this is a very aggressive schedule and will require diligent progress and co-ordination of parties involved, including the Eritrean Government personnel.

It is envisaged that SGC will appoint an independent engineering company to execute the Project on an EPCM basis according to the milestones listed in section 24.1.3.

24.1.2 Project Execution Strategy

The strategy for the execution of the Asmara Project includes the following major roles and responsibilities:

- Project manager
- Owner's team
- EPCM consultant
- Construction team
- Commissioning team
- Operation team

24.1.3 Project Schedule

24.1.3.1 Schedule Overview

The projected development schedule in this report is predicated on the following assumptions:

- Completion of the feasibility study in June 2013
- Commencement of detailed engineering by January 2014
- Placing orders for long lead delivery items from March 2014
- Mobilization for construction in September 2014
- Commencement of mining pre-production in October 2014
- Completion of detailed engineering in March 2015
- First gold production in September 2015
- Completion of the heap leach processing facility ramp-up to 100% in December 2015
- First copper production in October 2016
- Completion of the copper flotation plant ramp-up to 100% in December 2016
- First zinc production in February 2018
- Completion of the zinc flotation plant ramp-up to 100% in April 2017

During the above time the civil and earthworks contract should be awarded, in time for a successful contractor to establish site to best utilise the dry season conditions.

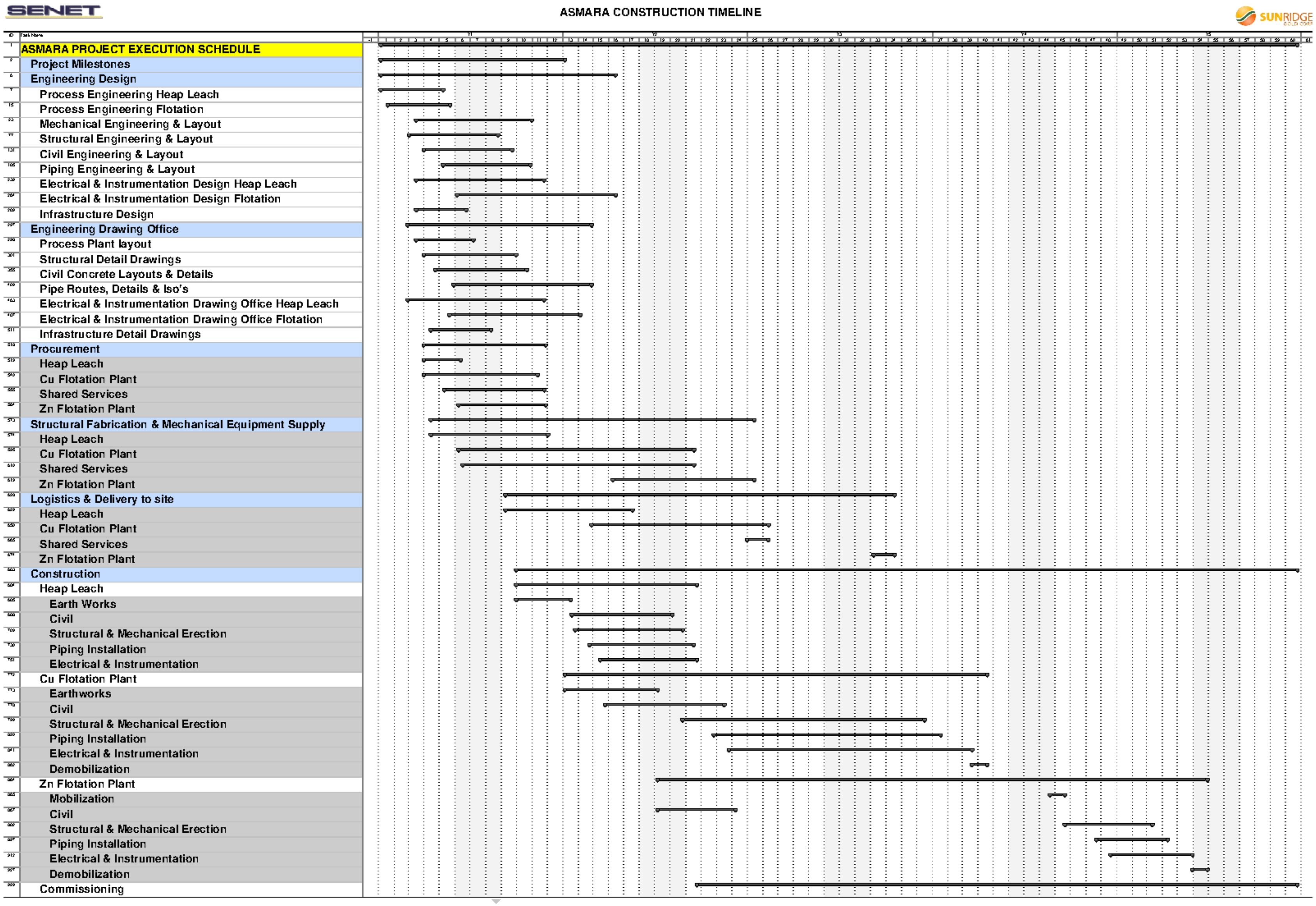
A roads sub-contractor or the civil and earthworks sub-contractor will be utilised to perform the necessary road upgrades at each of the site of Debarwa, Gupo, Adi Nefas and Emba Derho and build the bridge for the TSF haul road.

Detailed engineering should commence as soon as possible after the FS completion and would require up to 15 months for the full scope of the project to be completed.

A maximum of 12 weeks have been allowed for shipment of items to site. Due largely to the long delivery time on the SAG and Ball mills (15 months) as well as flotation cells (12 months) the copper project schedule will be completed within 24 months, by September 2016, ready for hot commissioning.

The summarised project schedule is shown in Figure 24.1.

Figure 24.1: Summarised Project Schedule



25 INTERPRETATION AND CONCLUSIONS

25.1 Estimation of Mineral Resource

Mineral Resource estimates for the Asmara Project deposits (Emba Derho. Adi Nefas, Gupo and Debarwa) have been generated using accepted industry practices and reported in conformity with CIM Definition Standards (2010) as required by NI 43-101. Only those parts of the Mineral Resource estimates that are categorised as Measured and Indicated may be considered for reporting of Mineral Reserves in Feasibility Studies.

25.1.1 Emba Derho

The updated resource estimate for the Emba Derho deposit was completed by Snowden Mining Industry Consultants Inc., as of 6 February 2012 and is based on geological interpretations and a drill database (current as at 9 September 2011) provided by SGC.

The resource reporting was constrained by a conceptual pit shell and a conceptual assessment of underground mining extractability to identify those regions of the model that have reasonable prospects for eventual economic extraction.

Mineral Resource Estimates are reported in Table 25.1, Table 25.2, Table 25.3 and Table 25.4.

Table 25.1: Measured Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper| (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	-	-	-	-	-
Cu Supergene	0.5% Cu	-	-	-	-	-
Copper-rich Primary	0.3% Cu	0.97	1.50	0.23	11.3	3.64
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.19	2.68	0.32	12.5	0.78
TOTAL						**4.42**

Table 25.2: Indicated Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	0.07	0.04	1.06	4.3	1.74
Cu Supergene	0.5% Cu	0.94	0.38	0.17	12.2	1.64
Copper-rich Primary	0.3% Cu	0.81	0.89	0.16	7.44	46.19
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.14	2.81	0.31	9.82	15.97
TOTAL						**65.55**

Table 25.3: Measured & Indicated Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	0.07	0.04	1.06	4.3	1.74
Copper Supergene	0.5% Cu	0.94	0.38	0.17	12.2	1.64
Copper-rich Primary	0.3% Cu	0.83	0.93	0.17	7.7	49.8
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.14	2.80	0.31	9.9	16.8
TOTAL						**70.0**

Table 25.4: Inferred Mineral Resource Estimate – Emba Derho

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Gold Oxide	0.5 g/t Au	-	-	-	-	-
Copper Supergene	0.5% Cu	-	-	-	-	-
Copper-rich Primary	0.3% Cu	0.87	0.89	0.25	10	13.28
Zinc-rich Primary	<0.3% Cu >1.0% Zn	0.20	1.94	0.39	11	1.77
TOTAL						**15.05**

25.1.2 Adi Nefas

The updated resource estimate for the Adi Nefas deposit was completed by Snowden Mining Industry Consultants Inc., is as of 20 February 2012 and is based on geological interpretations and a drill database (current as at 19 September 2011) provided by SGC.

The resource reporting was considered in the context of underground mining extractability and reasonable prospects for eventual economic extraction.

Mineral Resource Estimates are reported in Table 25.5 below:

Table 25.5: Indicated Mineral Resource Estimate – Adi Nefas

Zone	Cut-off grade	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (Mt)
Primary	2.0 % Zn	1.78	10.05	3.31	115	1.841

25.1.3 Gupo

The resource estimate for the Gupo deposit was completed by Snowden Mining Industry Consultants Inc., is as of 3 April 2012 and is based on geological interpretations and a drill database (current as at 12 March 2012) provided by SGC.

Mineral Resource estimates reported for Gupo are constrained by a conceptual pit shell in order to determine the potential quantity for eventual economic extraction. These resources are reported in Table 25.6 and Table 25.7 below.

Table 25.6: Indicated Mineral Resource Estimate- Gupo

Cut-off grade	Gold (g/t)	Gold (oz)	Mass (t)
0.50 g/t Au	1.53	46,780	951,800

Table 25.7: Inferred Mineral Resource Estimate- Gupo

Cut-off grade	Gold (g/t)	Gold (oz)	Mass (t)
0.50 g/t Au	1.83	106,340	1,808,550

25.1.4 Debarwa

The updated resource estimate for the Debarwa Project was completed by AMC Consultants (UK) Ltd, is as of 11 August 2011, reviewed by Snowden, and is based on geological interpretations and a drill database (current as at 20 April 2011) provided by SGC.

The preliminary classified block model was then subjected to two levels of constraint to ensure that only those portions which demonstrated potential economic viability were retained. Firstly an optimised pit shell derived using metal price parameters at a premium above long term prices (copper \$3.00 per pound, gold \$1,200 per ounce, zinc \$1.00 per pound and silver \$20.00 per ounce) was used to identify potential open pit material, after which optimised stope shapes, based on the same prices, were used to incorporate further material considered to be potentially mineable by underground methods.

Mineral Resource estimates for Debarwa are reported in Table 25.8, Table 25.9, Table 25.10 and Table 25.11 below.

Table 25.8: Measured Resources - Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5 g/t	0.01	0.01	1.03	4	3
Transition	Au 0. 5g/t	0.07	0.03	4.59	90	103
Supergene	Cu 0.5%	11.63	0.07	2.58	65	321
Primary (Cu)	Cu 0.5%	2.39	5.97	1.32	26	7
Primary (Zn)	ZN 2.0% (Cu<0.5%)					
Total						**434**

Table 25.9: Indicated Resources – Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5g/t	0.06	0.05	1.47	6	368
Transition	Au 0.5g/t	0.08	0.06	2.55	17	617
Supergene	Cu 0.5%	3.21	0.08	1.04	23	1,068
Primary (Cu)	Cu 0.5%	2.34	3.90	1.30	29	767
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.36	3.05	1.24	22	58
Total						**2 878**

Table 25.10: Measured and Indicated Resources – Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5g/t	0.06	0.04	1.47	6	371
Transition	Au 0.5g/t	0.08	0.05	2.85	27	720
Supergene	Cu 0.5%	5.15	0.07	1.40	33	1,389
Primary (Cu)	Cu 0.5%	2.34	3.92	1.30	29	774
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.36	3.05	1.24	22	58
Total						**3 312**

Table 25.11: Inferred Resources – Debarwa

Material Type	Cut-off	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Mass (kt)
Oxide	Au 0.5g/t	0.1	0.1	1.1	5	239
Transition	Au 0.5g/t	0.1	0.0	1.4	22	138
Supergene	Cu 0.5%	2.7	0.1	0.6	31	144
Primary (Cu)	Cu 0.5%	1.2	3.6	2.6	41	154
Primary (Zn)	Zn 2.0% (Cu<0.5%)	0.4	3.3	1.1	21	6
Total						**681**

25.2 Mineral Reserves

The mining methods selected consist of open pit truck shovel mining for Emba Derho, Debarwa and Gupo. At Adi Nefas underground long hole bench retreat was selected as the most appropriate method. The methods selected are in common practice around the world and in Africa. They utilize existing technology and equipment.

The study used the 2012 estimate of Measured and Indicated Resources for the Asmara Project as reported in previous technical reports. Table 25.12 summarizes the Mineral Reserves included in the study reported by both ore type and classification.

Table 25.12: Mineral Reserves

Rock Type	Copper (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Tonnes (kt)
Proven					
Emba Derho Primary	0.9	1.7	0.2	11.6	4,337
Debarwa Oxide	-	-	1.0	6.7	1
Debarwa Transition	-	-	4.3	84.1	94
Debarwa Supergene	8.9	0.2	2.2	53.2	423
Debarwa Primary	1.6	2.8	0.6	15.6	6
Total proven					**4,861**
Probable					
Emba Derho Supergene	1.0	0.4	0.3	14.9	1,200
Emba Derho Primary	0.7	1.6	0.3	9.2	44,497
Debarwa Oxide	-	-	1.6	8.2	163
Debarwa Transition	-	-	2.5	17.0	428
Debarwa Supergene	2.5	0.2	1.0	22.9	888
Debarwa Primary	1.9	4.0	1.1	25.4	514
Adi Nefas Primary	1.6	8.2	2.8	96.5	1,682
Gupo Oxide	-	-	1.9	-	399
Gupo Sulfide	-	-	2.4	-	66
Total Probable					**51,723**
Total Proven and Probable					**56,584**

25.3 Testwork

Gold and silver-bearing oxide composites representing the Debarwa transition and oxide zones have been subjected to grindability, gravity, cyanide leaching and flotation testing. Heap leach cyanidation was the chosen process resulting from economic evaluation of the candidate processes. Gold extractions from seven column tests ranged from 42% to 76%. The Gupo and Emba Derho Oxide composites showed the best and most consistent gold extractions ranging from 62% to 73%. The Debarwa Transition Composite #1 exhibited a relatively low gold extraction of 51%. Silver extractions ranged from 13% to 70%. Cyanide consumptions were relatively consistent and ranged from 0.82 kg/tonne to 1.35 kg/tonne.

Following mineralogical characterisation and grindability testing, composites representing supergene ores from the Debarwa and Emba Derho deposits were tested by flotation. Debarwa supergene ores are typical high grade, fine-grained VMS secondary copper ores. Emba Derho supergene ores are lower grade but coarser grained, hence they are easier to process. They are also more transition in nature, with typically a 50:50 mix of primary and secondary copper sulphides, and the intermittent presence of zinc as sphalerite. They are moderately soft, with a mean Bond Ball Mill Work Index (BBMWI) of 9.9 kWh/mt.

The process adopted is highly conventional, including grinding to a k80 product size of roughly 65 microns, rougher flotation at pH11 with xanthate. The rougher concentrate is reground to roughly 17 microns, and then cleaned in three stages at pH 11.5 using xanthate. The first cleaner includes a scavenger stage, the tails from which report to final tails. Locked cycle testing of blended Debarwa/Emba Derho supergene ores yielded concentrate grades of 24-25% copper at 85-86% percent recovery. Mean gold and silver recoveries of 59% and 65% percent respectively left payable grades of gold (4 g/t) and silver (96 g/t) in the concentrate.

Locked cycle test composites representing the projected mine production schedule as developed in the pre-feasibility study were created, together with samples representing life of mine and end-member copper and zinc contents. Composites representing early years of production (Years 0-1, 1-2 and 2-4) included, as needed, representative components from Debarwa, Adi Nefas and Emba Derho. The BBMWI averaged 11.2, 10.5 and 8.5 kWh/mt for Emba Derho, Adi Nefas and Debarwa respectively.

Emba Derho primary sulphide mineralogy is straightforward with copper present as moderately fine chalcopyrite and zinc as slightly coarser sphalerite in a pyrite-dominant rock. Adi Nefas and Debarwa both contain a broader suite of copper minerals, some of which are more reactive so more depressants are needed when treating these ores. The flowsheet developed, and ultimately tested in locked cycle and batch variability mode included:

Copper flotation followed primary grinding to a k_{80} product size of 80 microns, with a pre-mix of zinc sulphate and sodium cyanide added as zinc depressants to the mill, and was run at pH 9-10 using xanthate. Copper rougher concentrate was reground to a k_{80} product size of 25-30 microns, again with the zinc depressants, and then cleaned in two or three stages of copper cleaning again using xanthate.

Copper sulphate activation of the copper tails (zinc feed) used roughly 120 g/t copper sulphate per percent zinc in the feed, following adjustment to pH 11.6. Ensuing zinc rougher flotation used isopropyl xanthate. The zinc rougher concentrate was reground to a k_{80} of about 35 microns, and then cleaned in two stages of cleaning using xanthate at pH 11.8 with a first cleaner scavenger, the scavenger tail being open circuited to final tails.

All composites were successfully tested in locked cycle mode. Locked cycle copper recoveries were lower in the early year composites (81-85%), but rose to 93% for the Years 5-11 composite.

Precious metal recoveries from the chronology samples ranged from 35 to 62% with substantial payable precious metals in the early years' copper concentrates. Zinc recoveries ranged from 88% in the Year 0-1 composite to 93% later in the life of the mine.

Variability batch cleaner testing on 18 samples from Emba Derho demonstrated good consistency in metallurgical performance. Recoveries were linked to head grades, but consistently in line with those from the cycle test composites for head grades above 0.4% copper and 1% zinc, but dropping significantly at head grades below 0.3% copper and 0.5% zinc.

25.4 Environmental Studies, Permitting and Social or Community Impact

The next phase of the project will require the development and submission of the SEIA and SEMP to enable the granting of a mining license and relevant permits to construct, service, and operate the mine sites, processing facility and port operations.

To date no environmental, social, community or permitting issues have arisen that are seen as a threat to the project. The focus of the environmental, social and permitting studies continues to be to identify the requirements of all relevant statutory authorities and ensure that suitable strategies and management plans are put in place to address statutory requirements, as well as those of the community, and other relevant stakeholders.

25.5 General

The study shows that within an overall accuracy the of +-15%, using the assumption stated, that the project is robust economically and uses well established recovery and mining technologies to deliver a reliable product stream over a 16 year life.

The project utilises standard equipment and infrastructure and the centralization of the processing facility at Emba Derho minimises duplication of capital infrastructure and tailings facilities.

The provision of basic services and site access are straightforward and the climate and location are amenable to year round mining, construction and maintenance.

The designs for the TSF, EMWP, MBAR, DMWP, and all other water management infrastructure may be further optimized in detailed design, depending on specific water management objectives that may change as the project evolves. The availability and supply of make-up water for the mining operation, particularly surrounding the use of water from the Tokor pipeline, may be a project component that is subject to socio-political risks. Water supply assumes that all process water is recycled, all contact water is collected (and prevented from discharging into the Tokor reservoir), and that additional make-up water will be sourced from the MBAR and/or purchased from the Tokor reservoir water supply pipeline.

It is possible that additional design and operational provisions will be needed to ensure water quality concerns are alleviated. It is also possible that the amount of water available from the Tokor reservoir may be reduced or restricted during prolonged dry conditions or due to government restrictions.

26 RECOMMENDATIONS AND OPPORTUNITIES

26.1 Estimation of Mineral Resource

The Inferred Resource at Emba Derho NW extension is recommended to be targeted for in-fill drilling since this mineralised zone lies immediately adjacent to the planned FS pit. In the event that in-fill drilling is successful in improving the resource confidence, it is likely that the planned pit could be enlarged or, alternatively, a portion of the NW extension could be considered for underground mining.

In-fill drilling is recommended for the Gupo gold deposit in order to upgrade the large component of Inferred to at least the Indicated category (pending the outcome of further metallurgical testing and assessment of its value to the project). Due to the large reliance on reverse-circulation drilling, a study of this mode of drill sampling with core drill sampling is recommended to understand whether sampling bias exists.

26.2 Mining

The FS Mineral Reserves estimate serves to validate the FS result. It is therefore recommended that the project progress to detail design and execution and that fixed firm quotations be obtained for the supply of mining equipment and mining services.

Mining engineering on the four mines is well advanced. Opportunities exist in the following areas:

- The significant support costs at Adi Nefas could be reduced after more detailed geotechnical analysis
- Further development of the resource via drilling at Gupo may lead to a larger gold reserve

26.3 Testwork

A limited program of focused metallurgical testwork should be completed prior to progressing to final detailed design and before plant start-up, focusing on samples of mill feed in the very early stages of flotation processing. This could include:

- Testwork to enhance gold and silver recovery in the early years of primary ore processing
- Assessment of the value of gravity recovery to improving gold metallurgy
- Grindability testing on early primary (years 1 & 2) material by quarter to better predict mill capacity in the early sulphide milling years
- Flotation on blends representing start-up primary ores (years 1 & 2) by quarter using the feasibility mine plan – but only once the mine plan has been finally frozen
- More Debarwa/Emba Derho supergene mixes, spanning the blend ratios as established in the mine plan, again initiated once the mine plan has been finally frozen
- Flotation of low grade primary stockpile material
- Establish tolerance for copper in the heap leach and associated firm limit for feed material into the heap

26.4 Environmental Studies, Permitting and Social or Community Impact

Close-out of ongoing studies will be required to support project permitting. These include, but are not limited to, the following:

- Completion of the stakeholder consultation process
- Completion of baseline studies and impact assessments for the disciplines identified in this feasibility study, as well as any additional studies identified through the stakeholder consultation process
- Confirmation of compensation and other assistance measures required for displacement impacts on land assets and livelihoods
- Definition of the roles and responsibilities between SGC and Government related to land access/acquisition and resettlement

26.5 General

This study supports moving the project through to detailed design and the gathering of fixed firm quotations for the supply of goods and services. After this is complete, final economic evaluation can then be completed, and if favourable, project financing should be sought so that construction can commence

It is recommended that the following further activities occur to support project development:

- Confirm that the extraction volumes that are required from the Tokor pipeline for the first five years of the project are formally agreed and secured.
- Confirm that the quality of water obtained from the MBAR is acceptable as a make-up water source for the HL facilities and the Process Plant.
- Confirm that the water quality in the EMWP is acceptable for use in dust suppression around the Emba Derho mine site.
- Further study surrounding the integration of the heap-leach facility within the TSF should occur in detailed design to ensure the most economic and technically viable concept is developed.
- Further study surrounding the integration of the quarry within the TSF should occur in detailed design to ensure that the final basin shaping activities will facilitate subsequent liner installation over the finished quarry area.
- Confirm that the mine waste production schedule is capable of providing all of the required construction materials in the needed time periods through the initial construction phase, by updating the execution schedule in the next project phase (detailed design).
- Further geotechnical site investigations are required at some key infrastructure locations, including the Mereb River diversion works at Debarwa and the Mai Bela abstraction weir, the process plant and associated facilities and at the bridge location to the Mesheala River crossing.

27 REFERENCES

The following documents have been cited in this document.

Barrie, C.T. (2004). Report on Geology and Geochemistry for the Asmara VMS Base Metal Properties, Eritrea. An Internal Report for Sub-Sahara Resources N.L. and Sunridge Gold Corp. by C.T. Barrie and Associates, Inc., 31p.

Blackburn, G. and Chisholm, R.E. (2004). Independent Consulting Geologist's Report: Asmara Project, Hamasien Goldfield, Debarwa Base Metal Camp, Eritrea. Unpublished Report for Sunridge Gold Corp., 93p.

CIM (2010) CIM Definition Standards – on Mineral Resources and Mineral Reserves (2010). Prepared by the CIM Standing Committee on Reserve Definitions. Adopted by CIM Council on 27 November, 2010.

Coumoul, A. (1998) Evaluation of the Adi Nefas Doop gold deposit, Eritrea. BRGM Report No. N2571 – La Source 10.065. (Note. Appendices 1, 2 pt.1 &3, and Appendix 4 missing) Report to the Department of Mines – Eritrea.

Daoud, D.K. and Greig, C.J. (2007). Geology and mineralization of the Emba Derho deposit. Internal Sunridge report, 7p.

Drury, S.A. and De Souza Filho, C.R. (1998). Neoproterozoic terrane assemblages in Eritrea: review and prospects. Journal of African Earth Sciences, v27, p331-348.

Galley, A. Hannington, M. and Jonasson, I. (2004). Volcanogenic sulphide deposits. Geological Survey of Canada, 19p.

Gribble, P., Webster, A., Cowie, S. (2009). Emba Derho Project, Eritrea, Preliminary Economic Assessment. NI 43-101 Technical Report undertaken by Wardrop Engineering International for Sunridge Gold Inc., 196p.

Hopley, M.J., Arnold, C.G., Martin, C.J. (2011). Debarwa Copper Gold Deposit Eritrea Technical Report on Additional Drilling and Revised Mineral Resource Estimates. NI 43-101 Technical Report prepared by and for Sunridge Gold Corp., with contributions by AMC Consultants (UK) Limited and Blue Coast Metallurgy Ltd. 188 pages. Effective date 18 August 2011.

Hall, M. Robertson, M. and Greenway, G. (2008). 43-101 Technical Report: Adi Nefas Project Eritrea Mineral Resource Estimate. Undertaken by MSA Geoservices (Pty) Ltd. for Sunridge Gold Corp, 126p.

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Emba Derho Deposit, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc for Sunridge Gold Inc with contributions by Blue Coast Metallurgy Ltd. 110 pages. Effective date 6 February 2012.

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc for Sunridge Gold Corp with contributions by Blue Coast Metallurgy Ltd. 82 pages. Effective date 20 February 2012.

Ross, A.F. and Martin C.J. (2012). Sunridge Gold Corp: Gupo Gold Mineral Resource Estimate Update, Adi Nefas Property, Eritrea. NI 43-101 Technical Report prepared by Snowden Mining Industry Consultants Inc for Sunridge Gold Corp with contributions by Blue Coast Metallurgy Ltd. 97 pages. Effective date 3 April 2012.

Singer, D.A. (1995). World-class base and precious metal deposits – a quantitative analysis. Economic Geology, v90, p88-104.

Stokes, W.P., Kirkham, G., Greig, C.J. and Smee, B. (2007). Mineral Resources Estimate on the Adi Nefas Property, Adi Nefas, Eritrea. Independent Technical Report for Sunridge Gold Corp. prepared by Beacon Hill Consultants (1988) Ltd., March 15, 2007.

Thomas, D. And Marais, C. (2009). Gupo Drill Hole Database Data Verification and Data Quality Assessment. Undertaken by AMEC for Sunridge Gold Corp. 88p.

Van der Merwe, A., Goldschmidt, A., Nesbitt, A. and McDonald, A. (2006). Independent Valuation of the Asmara Project Mineral Properties, Eritrea. Valuation Date: 13 January 2006. Prepared for Sunridge Gold Corp., Sub-Sahara Resources NL, and Africa Wide Resources Ltd. by SRK Consulting, August, 2006.

Zeremariam, O. (2005). Report on detailed geological mapping of the Adi Nefas gossan. Unpublished internal map and report for Sunridge Gold Corp. 25pGan, J. K-M., Fredlund, D.G., (1996), "Shear Strength Characteristics of Two Saprolitic Soils", Soil Vision, 1993

Giroud, J.P. and Bonaparte, R., (1988), "Leakage Through Liners Constructed with GeomEmbanes", GeoServices Inc. Consulting Engineers.

ICOLD - International Commission on Large Dams, (1995), "Tailings Dams and Seismicity: Review and Recommendations," Bulletin 98.

Knight Piésold, "2010 Geotechnical and Hydrogeological Site Investigation Report", VA402-9/8-1 Rev 0 September 7 2011 for Sunridge Gold Corp., Debarwa Project.

Knight Piésold, "Debarwa Hydrometeorology Report", VA402-9/7-1 Rev 0 June 9 2011 for Sunridge Gold Corp., Debarwa Project.

Knight Piésold, "Hydrogeological Factual Data Report", VA401-27/1-2 Rev 0 March 8 2008.

Knight Piésold, "Preliminary Assessment of Mine Waste Management Facilities", VA401-27/1-1 Rev A March 15 2007.

Leps, T.M., (1970), "Review of Shearing Strength of Rockfill", Journal of the Soil Mechanics and Foundations Division.

Makdisi, F.I., Bolton Seed H., (1977), "A Simplified Procedure for Estimating Earthquake Induced Deformations in Dams and Embankments", Report No. UBC/EERC-77/19.

Newmark, N.M., (1965), "Effect of Earthquake on Dams and Embankments", Geotechnique, 15:139-160.

Swaisgood, J.R., (2003), "Embankment Dam Deformations Caused by Earthquakes", 2003 Pacific Conference on Earthquake Engineering

British Columbia Ministry of the Environment. 1992. Framework for a Hydrogeologic Study. Available at: http://www.env.gov.bc.ca/wsd/plan_protect_sustain/groundwater/library/framework.html. Accessed May 2012.

28 CERTIFICATES OF QUALIFIED PERSONS

28.1 Neil Senior

CERTIFICATE OF QUALIFIED PERSON

Neil Senior Pr.Eng. FSAIMM

Building 12, Greenstone Hill Office Park

Emerald Boulevard

Modderfontein, Johannesburg

Gauteng 1609 Republic of South Africa

I, Neil Senior am a Professional Engineer, employed as an Executive Director of SENET.

- This certificate applies to the technical report entitled “Asmara Project Feasibility Study NI 43-101 Technical Report” (“The Report”) dated 16th May 2013.
- I am a fellow of the South African Institute of Mining and Metallurgy (SAIMM) and a registered Professional Engineer (registration number 800284). I graduated from Cranfield University, United Kingdom with an MSc. Engineering (mechanical) in 1972.
- I have practiced my profession for 35 years. I have been directly involved as the Sponsor of the Debarwa Project and reviewer of the report on behalf of SENET.
- As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).
- I visited the Asmara Project Site in September 2012.
- I am responsible for Sections 1, 2, 3, 4 and 17 to 27 of the Asmara Project Feasibility Study NI 43-101 Technical Report.
- I am independent of Sunridge Gold Corporation as independence is described by Section 1.4 of NI 43-101.
- I have had no previous involvement with the Asmara Project.
- 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.
- I consent to the filing of the Technical Report with any Canadian stock exchange and other Canadian regulatory authorities and publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated the 26th day of June, 2013

[SIGNED]

Mr Neil Senior, P.Eng

MSc Mech.Eng FSAIMM

28.2 Anthony Finch

CERTIFICATE of QUALIFIED PERSON

I, Mr Anthony Finch, P.Eng., MAusIMM (CP Mining), B.Eng., BEcon., Divisional Manager of Mining with Snowden Mining Industry Consultants Pty Ltd., Level 3, The Magdalen Centre, Robert Robinson Avenue, The Oxford Science Park, Oxford 0X4 4GA, United Kingdom, do hereby certify that:

- I am one of the authors of the technical report for the Asmara Project entitled "Asmara Project Feasibility Study Technical Report" with an effective date of 16 May 2013 (the "Technical Report").
- I graduated with a degree in Mining Engineering from the University of Queensland in Australia in 1986. I am a Professional Engineer in the Province of British Columbia, number 164687, and a Member of the Australasian Institute of Mining and Metallurgy, number 103583 and a Certified Professional (Mining) of that Institute. Since graduation I have had 25 years continuous experience in the mining industry, in both operations and consulting, in various roles of increasing seniority. I have worked in hard rock underground mining, including precious metals, for over ten years, and in open pit mining for over ten years. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be considered a "qualified person", as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).
- I visited the Asmara Project Area in February 2011, February 2012, June 2012, & March 2013
- I am responsible for relevant components in sub-sections 1: Summary; 2: Introduction; 3: Reliance on Other Experts; 15: Mineral Reserve Estimates; 16: Mining Methods; 21: Capital and Operating Costs; 22: Economic Analysis; 24: Other relevant data and information; 25: Interpretation and Conclusions; 26: Recommendations; 27: References of the Technical Report.
- I am independent of Sunridge Gold Corp. as described in Section 1.5 of NI43-101.
- I have had no involvement with the properties that are the subject of this Technical Report prior to execution of the work recorded in the Technical Report.
- I have read NI43-101 and the portions of the technical report for which I am responsible have been prepared in accordance with NI43-101.
- As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 26th day of June, 2013

[SIGNED]

Mr. Anthony Finch, P.Eng,

MAusIMM (CP Min), B.Eng.(Min), BEcon.

28.3 Andrew Ross

I, Andrew F. Ross, Senior Principal Consultant of Snowden Mining Industry Consultants Pty Ltd., 87 Colin St., West Perth, Australia, do hereby certify that:

- I am one of the authors of the technical report for the Asmara Project entitled “Asmara Project Feasibility Study Technical Report” with an effective date of 16 May 2013 (the “Technical Report”).
- I graduated with an Honours Degree in Bachelor of Science in Geology from the University of Adelaide in 1972. In 1985 I graduated with a Master of Science degree in Geology from James Cook University of North Queensland. I am: a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy; a member of the Australian Institute of Geoscientists; licensed as a Professional Geoscientist with APEG (British Columbia). I have worked as a geologist continuously for a total of 40 years since graduation. I have been involved in resource evaluation consulting for 17 years, including resource estimation of base metal and gold deposits for at least 5 years. I have been involved in base metal and gold exploration and mining operations for at least 5 years. I have read the definition of ‘qualified person’ set out in NI 43-101 (‘the Instrument’) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a ‘qualified person’ for the purposes of the Instrument.
- I visited the Sunridge Asmara property from 14 February to 19 February, 2011.
- I am responsible for the preparation of sections 4,5,6,7,8,9,10,11,12 and 14..
- I am independent of the issuer as defined in section 1.4 of the Instrument.
- I have had no prior involvement with the property that is the subject of the Report apart from authoring previously filed Technical Reports for Emba Derho, Adi Nefas and Gupo, as referenced in the Report.
- I have read the Instrument and Form 43-101F1, and the Report has been prepared in compliance with that instrument and form.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Report contains all the scientific and technical information that is required to be disclosed to make the Report not misleading.
- I consent to the filing of the Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Report.

Dated at Perth, Western Australia this 27 June 2013.

[SIGNED]

Andrew F. Ross

FAusIMM (CP Geo)

28.4 Scott Rees

CERTIFICATE of QUALIFIED PERSON

I, Scott Rees, P.Eng., of Knight Piésold Ltd., 1400-750 W. Pender St., Vancouver, BC., V6C 2T8, do hereby certify that:

- I am one of the authors of the technical report for the Asmara Project entitled “Asmara Project Feasibility Study Technical Report” with an effective date of 16 May 2013 (the “Technical Report”).
- I graduated with degrees in Geological Engineering (Bachelor) and Civil Engineering (Masters) from the University of British Columbia in Vancouver in 1982 and 1985. I am registered as a Professional Engineer with the Association of Professional Engineers and Geoscientists in the Province of British Columbia, member number 15117. I have gained 30 years of continuous experience, primarily as a consultant in the mining industry, in various roles of increasing seniority. I have worked extensively in the areas of geotechnical design and site investigations, tailings and water management infrastructure design, construction and operations supervision and management, as well as social and environmental studies, stakeholder engagement, and permitting. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be considered a “qualified person”, as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).
- I have visited the Asmara Project areas multiple times from 2007 to 2012. The most recent site visit was from February 27 to March 3, 2012.
- I am responsible for the social and environmental baseline and permitting, tailings and water management in the following sub-sections 1: Summary; 2: Introduction; 18: Project Infrastructure; 20: Environmental studies, permitting and social or community impacts; 21: Capital and Operating Costs; 25: Interpretation and Conclusions; 26: Recommendations
- I am independent of Sunridge Gold Corp. as described in Section 1.5 of NI43-101.
- I have had on-going involvement in various engineering and social/environmental studies completed by Knight Piésold Ltd. for Asmara Project sites since 2007.
- I have read NI43-101 and the portions of the technical report for which I am responsible have been prepared in accordance with NI43-101.
- As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 26th day of June, 2013

[SIGNED]

Scott Rees P.Eng

BEng (Geo), C.Eng (Mast),

Chris Martin

I, Christopher John Martin of Blue Coast Metallurgy, 1020 Herring Gull Way, Parksville, British Columbia hereby certify:

- I am one of the authors of the technical report for the Asmara Project entitled "Asmara Project Feasibility Study Technical Report" with an effective date of 16 May 2013 (the "Technical Report").
- I am a graduate of Camborne School of Mines and hold a B.Sc (Honours) Degree in Mineral Processing Technology (1984)
- I am a graduate of McGill University and hold a M.Eng Degree in Metallurgical Engineering (1988)
- I am presently President and Principal Metallurgist at Blue Coast Metallurgy Ltd, 1020 Herring Gull Way, Parksville, British Columbia.
- I have ten years' plant operations experience as Metallurgist, Senior Metallurgist and Plant Superintendent with Rustenburg Platinum Mines in South Africa, as Operations Engineer at Nerco Con Mine in the Northwest Territories and Chief Corporate Metallurgist at Sunshine Mining in Idaho.
- I have fifteen years' flowsheet development and plant optimisation consulting experience as manager SGS Lakefield Mineral Processing group, General Manager of SGS Vancouver Metallurgy, Manager Metallurgy at AMTEL and most recently as President of Blue Coast Metallurgy.
- I have been a licenced Chartered Engineer in good standing with the IMMM since 1990. I am also a member of the Canadian Institute of Mining, Metallurgy and Petroleum.
- I have read the definitions of "Qualified Person" set out in NI43-101 and certify that, by reason of my education, affiliation to a professional institution and past work experience, I fulfill the requirements to be a Qualified Person" for the purposes of NI43-101.
- I have visited the Asmara Project site during 2012.
- I have no direct involvement with Sunridge Gold Corp, and am independent of Sunridge Gold Corp applying all the tests of Section 1.5 of NI43-101.
- I have read NI43-101 and NI43-101F1 and the technical report is been compiled in compliance with the instrument and form.
- As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 26th day of June, 2013

[SIGNED]

Christopher J. Martin, C.Eng

BSc (Hons) ACSM, M.Eng, MIMMM,