# Preliminary Economic Assessment Technical Report for the Halilağa Project, Turkey

**Report Prepared for**

## Truva Bakir Maden İşletmeleri A.Ş. and Pilot Gold Inc.

**Report Prepared by**

SRK Consulting (Canada) Inc.
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## For:

## Truva Bakir Maden İşletmeleri A.Ş.,

Turan Güneş Bulvari No. 8614,
Çankaya, Ankara,
Turkey 06550

And

## Pilot Gold Inc.

Suite 1650, 1055 West Hastings St.
Vancouver, BC Canada V6E 2E9

**Compiled by:**
**SRK Consulting (Canada) Inc.**
Suite 2200 -1066 West Hastings Street
Vancouver, B.C., Canada V6E 3X2

e-mail: vancouver@srk.com
website: www.srk.com

Tel: +1.604.681.4196
Fax: +1.604.687.5532

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**Authored by:**

Kevin Scott, P. Eng.
James Gray, P.Geo.
Garth Kirkham, P.Geo.
Gordon Doerksen, P.Eng.
Dino Pilotto, P.Eng.
Maritz Rykaart, P.Eng.

**Peer Reviewed by:**

Neil Winklemann

# Important Notice

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# Executive Summary

This Preliminary Economic Assessment Technical Report ("PEA" or "report") summarizes the findings of a preliminary economic assessment and was prepared by the following qualified persons ("QP"):

- Gordon Doerksen, P.Eng., Project Manager, JDS Energy and Mining Inc. ("JDS");
- Dino Pilotto, P.Eng. and Maritz Rykaart, P.Eng., SRK Consulting (Canada) Inc. ("SRK");
- Kevin Scott, P.Eng., Ausenco Solutions Canada Inc. ("Ausenco");
- Garth Kirkham, P.Geo., Kirkham Geosystems Ltd. ("Kirkham") and
- James Gray, P.Geo., Advantage Geoservices Ltd. ("Advantage").

The report was commissioned by Truva Bakir Maden İşletmeleri A.Ş. ("Truva Bakir"), the Joint Venture (Joint Stock) holding company for the interests of Teck Madencilik Sanayi Ticaret A.S.and Pilot Gold Inc. ("Pilot" or "Pilot Gold"). The principal asset of Truva Bakir is the Halilağa Project. This technical report was written in compliance with disclosure and reporting requirements set forth in the Canadian Securities Administrators National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1 (collectively, NI 43-101).

This study was overseen by Mr. Jim Lincoln of Pilot Gold.

The purpose of this technical report is to present the "first time" findings of a PEA of the project. The report is based upon the mineral resource estimate first presented by Gray and Kirkham in the project's March 23, 2012 Technical Report titled "Resource Estimate for the Halilağa Copper-Gold Property NI 43-101 Technical Report ("Gray and Kirkham 2012"). The contents of this report reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

The reader is cautioned that the PEA summarized in this technical report is preliminary in nature and is only intended to provide an initial, high-level review of the project. Further studies are required with regards to infrastructure and operational methodologies. The PEA mine plan and economic model include the use of a significant portion of inferred resources which are considered to be too speculative geologically to have the economic considerations applied to them that would enable them to be characterized as mineral reserves. There is no guarantee that inferred resources can be converted to indicated or measured resources. There is no guarantee that the project economics described herein would be achieved.

## Project Concept

The proposed Halilağa Project concept is to develop a low-grade, green-fields copper-gold deposit with open pit ("OP") mining and conventional milling and flotation concentration methods. The production rate was assumed to be 50,000 tonnes per day ("t/d" or "tpd") with about 243 million tonnes ("Mt") of mineralized material mined and processed during the project life. The overall strip ratio (the ratio of waste rock to economic mineralized rock) of the mine is approximately 1:1 and the average grade of the plant feed would be an estimated 0.28% copper and 0.3 gram/tonne of gold.

It is proposed that the copper-gold concentrate would be shipped to an existing smelter in Europe or the Far East.

The project site is completely undeveloped but has road access and electrical grid power supply. Labour supply and industrial service providers are available in the region.

The proposed mine production life is 14 years. In addition, it is anticipated that there would be a five year feasibility, permitting and pre-production construction period as well as a reclamation period.

## Project Ownership

A joint venture company called Truva Bakir Maden İşletmeleri A.Ş. (“Truva Bakir”) owns, or has beneficial interest in, the licenses that comprise the Halilağa Project. Pilot Gold Inc. has a 40% interest and Teck Madencilik Sanayi Ticaret A.S.has a 60% interest in Truva Bakir.

## Property Description

The Halilağa Project is located about 40 km southeast of Çanakkale between the villages of Halilağa and Muratlar on the Biga Peninsula in Northwestern Turkey. The main area of interest is the Kestane porphyry copper-gold zone located at 483200E, 4419200N UTM Central meridian 27 (ED50 datum).

The Halilağa Project consists of 19 licenses covering 9,299.79 ha. Eighteen licenses are directly held by Truva Bakir, and one license is held by Teck Madencilik for the benefit of Truva Bakir. Three of the licenses are “Exploration-Type”, four are ‘Operation-Type”, and a further 12 are operation pending licenses.

The main license hosting the Central Zone at the Kestane porphyry has been converted to an operation license.

## Geology and Mineralization

The Halilağa Project is located in the south-central part of the Biga Peninsula in Western Turkey. The geology of the peninsula is complex and characterized by various lithological associations made up of: (i) basement metamorphic rocks, (ii) Permian and Mesozoic rock units, (iii) ophiolitic rocks, (iv) tertiary (Eocene) volcanogenic units, (v) Neogene sediments, and (vi) collisional to extensional Tertiary granitoids.

The property is located in a district with major potential for both porphyry copper-gold and high-sulphidation style gold deposits. Investigation of geological relationships in the field along with geochemical and geochronological data suggest that magmatic activity began with the intrusion of granitoids, coeval with an early phase of volcanic activity, and was superseded by a second phase of volcanism. The mineralized intrusives are thought to be part of a suite of 26 million year old (“Ma”) granites and granodiorites.

The Halilağa district is mainly underlain by volcano-sedimentary basement cover sequences of the Oligo-Miocene age. The basement consists of schists and carbonates, which outcrop to the southeast of the Bakirlik area. The granodioritic batholith intrudes the basement rocks, which includes carbonates, and generates metasomatism and skarnification. The Kestane porphyry was emplaced into the volcano-sedimentary cover sequence and produced hornfelsed halos adjacent to the stock.

The Halilağa Project is interpreted to host a single widespread mineralized system containing porphyry copper-gold mineralization, skarn, and related high-sulphidation gold mineralization. The Halilağa alteration system covers an area of more than 4 km x 2 km and displays all porphyry-related

alteration assemblages, including epithermal and skarn systems. The Kestane porphyry outcrops show potassic alteration overprinted by phyllic alteration. The highest gold and copper grades in the drill core are associated with early biotite + magnetite + chalcopyrite associated with A- and B-type quartz veins. At the Kestane porphyry, most quartz veins are B-type that average 5% of the rock by volume, but locally up to 20% are A-type veinlets that are rare or difficult to recognize in the outcrop. Chalcopyrite and pyrite are the dominant sulfides in potassic-altered rocks. Magnetite is abundant (5-8%) in zones of high copper and gold grades. At higher elevations to the south of Kestane, the Kunk-Kumlugedik hilltops are characterized by silicification surrounded by argillic and further propylitic alteration. Skarn-related alteration is located around the Bakirlik and Bostanlikbasi areas.

The geology of the Halilağa area is highly influenced by structures which relate to, or which are part of, the North Anatolian Fault System. The general architecture of the faults appears controlled by major ENE-WSW strike-slip structures which could be part of a transtensional tectonic regime. Local transpression has been interpreted in the Kestane area producing the exhumation of the porphyry and basement units.

## Exploration

In 2006-2007, a total of 23 holes (including five abandoned holes) totalling 6,346 m were completed. Most of the holes targeted the Kestane porphyry and intersected porphyry-style copper-gold mineralization with economic grades, as shown by discovery drill hole HD-01, which intersected 1.03 g/t gold and 1.03% copper over 105.4 m. A 25 metre-thick chalcocite blanket averaging approximately 2% copper was also intersected close to the surface in holes HD-01, HD-02, HD-04, and HD-14.

In 2008, the Bakirlik skarn zone (4 km ESE of Kestane) was the major focus of the drilling program. A total of 20 diamond holes totalling 4,051 m were completed during that period. Holes HD-21 and HD-25 intersected narrow zones of skarn mineralization with high grade copper + gold ± silver values. Detailed prospecting and systematic rock chip sampling of altered volcanic rocks at Kumlugedik Hill, the northern limb of Kunk Hill, and Madenderesi targets returned with anomalous gold values that ranged from 0.2 g/t to 2.0 g/t.

In 2009, a total of 18 holes (including four abandoned holes), totalling 5,670 m (excluding the 247 m of hole HD-42D, a deviated hole) were completed at Kestane.

In 2010, the program was designed to continue grid-drilling the Central Zone (the main area of porphyry copper-gold mineralization). A total of 25 holes (20 diamond and five RC) totalling 9,076.6 m (including 14 abandoned holes) were completed.

In addition to drilling, Induced Polarization (“IP”) geophysical surveys were carried out in 2009 and 2010 that highlighted deep chargeability targets 1 km west of Kestane, and also a chargeability target at Madeneresi.

The 2011 program focused on extending the mineralization and acquiring data sufficient for producing an NI 43-101 resource estimate. Significant intersections were encountered, including a zone of significant grade corresponding to the near-surface chalcocite blanket encountered in 2007 drilling. A total of 44 holes (including four abandoned holes) totalling 19,599 m were completed. A series of north-south geological sections were constructed every 100 m through the deposit. Sectional interpretations now show two E-W-trending normal faults bounding the porphyry

mineralization to the north and south, creating a mineralized horst at the centre; another NW-SE-trending, east-dipping normal fault bounds the top of the horst.

## Mineral Resource and Mineral Reserve Estimates

The Halilağa Mineral Resource is presented in Table i. For the main sulphide resource, a cut-off of 0.2 percent copper equivalent ("CuEq (%)") was felt to be reasonable based on a production rate of 50,000 to 70,000 t/d from a pit feeding a mill and flotation plant where total operating costs ("OPEX") would be in the range of $10-12 /t. Due to the differing metallurgical characteristics and anticipated metal extraction methods, the oxide resource is tabulated separately. Although the gold resource is extractable, it is not expected that the base metals within the oxide zone would be recoverable. The cut-off of 0.2 g/t gold is judged as reasonable based on other heap leach gold projects including Alamos Gold's nearby Ağı Dağı project.

**Table i: Halilağa Estimated Mineral Resources**

| Resources | Tonnes (1,000s) | In Situ Grade | | | |
|---|---|---|---|---|---|
| | | Cu (%) | Au (g/t) | Mo (%) | CuEq (%) |
| **Sulphide Resource[(1)]** | | | | | |
| Indicated | 168,167 | 0.30 | 0.31 | 0.006 | 0.45 |
| Inferred | 198,662 | 0.23 | 0.26 | 0.007 | 0.36 |
| **Oxide Resource[(2)]** | | | | | |
| Inferred | 4,914 | | 0.60 | | |

Strip Ratio: 2.5 : 1
(1) at a 0.2% copper equivalent cut-off grade.
(2) at a 0.2g/t gold cut-off grade.

Inferred resources were used in the life-of-mine ("LOM") plan with inferred resources representing 44% of the material planned for processing. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study ("PFS") or a feasibility study ("FS") of a mineral project. Inferred resources are considered to be too speculative geologically to have the economic considerations applied to them that would enable them to be characterized as mineral reserves. Accordingly, at the present level of development, there are no mineral reserves at the Halilağa Project.

## Mine Development and Operations

It is proposed that the Halilağa deposit is amenable to be developed as an OP mine. Mining of the deposit is planned to produce a total of 243 Mt of processing plant feed and 234 Mt of waste (0.96:1 overall strip ratio) over a fourteen year mine production life. The current LOM plan focuses on achieving consistent plant feed production rates, and mining of higher grade material early in schedule, as well as balancing grade and strip ratios. Figure i illustrates the proposed overall site layout for the Halilağa Project, including the OP and proposed plant site locations.

The mine design process for the deposit commenced with the development of CAE Mining NPV Scheduler ("NPVS") OP optimization input parameters. These parameters included estimates of

metal price, mining dilution, process recovery, offsite costs, geotechnical constraints (slope angles) and royalties (see Table ii). The resource model was based on a 20 m x 20 m x 10 m block size.

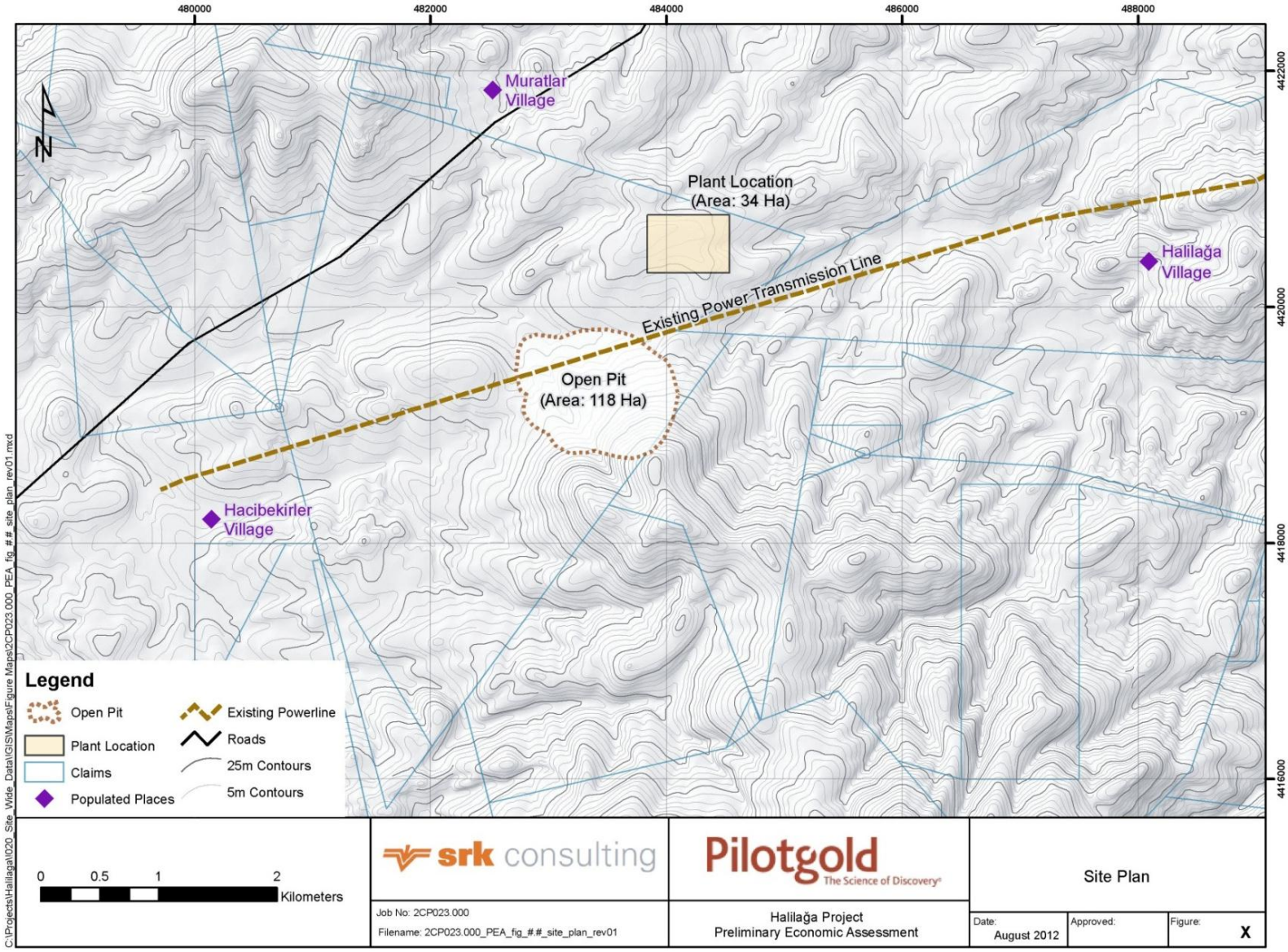


**Figure i: Halilağa Site Plan (SRK, 2012)**

**Table ii: Mine Planning Optimization Input Parameters**

| Item | Unit | Values |
|---|---|---|
| *Metal Prices* | | |
| Gold | $/oz | 1,200 |
| Copper | $/lb | 2.90 |
| *Recovery to Cu Concentrate* | | |
| Gold | % | 60.8 |
| Copper | % | 85.6 |
| *Cu Concentrate Grade ("conc.")* | | |
| Gold | g/t | var.w/Au and Cu grade |
| Copper | % | 30.0 |
| Moisture content | | 8.0 |
| *Smelter Payables* | | |
| Gold in Cu conc | % | 96 |
| Gold deduction in Cu conc. | g/t in conc | 1 |
| Copper in Cu conc | % | 96 |
| *TCRC* | | |
| Cu conc treatment | $/dmt conc | 75.00 |
| Cu refining charge | $/lb pay Cu | 0.075 |
| Au refining charge | $/oz pay Au | 7.00 |
| Transport, marketing, ins., etc. | | |
| Ocean freight to Europe | $/wmt conc | 30.00 |
| Truck freight to Port | $/wmt conc | 20.00 |
| Port charges | $/wmt conc | 6.00 |
| Marine/transportation insurance | % | 0.054 |
| Transport, marketing, ins, etc. | $/wmt conc | 56.00 |
| Transport, marketing, ins, etc. | $/dmt conc | 60.87 |
| *Other Parameters* | | |
| Grade factor (variable) | % | 95 |
| Royalties | % | 2.00 |
| *Operating Costs* | | |
| O/P Waste mining Cost | $/waste tonne | 1.86 |
| OP Mineralized material Mining Cost | $/ore tonne | 1.69 |
| OP Average Mining Cost | $/total tonne | 1.77 |
| OP Processing and G&A Cost | $/milled tonne | 5.11 |
| Pit Slope Angles | overall degrees | 38 to 48 |
| Dilution | % | 5.0 |
| Mining recovery | % | 100 |
| Strip ratio (est.) | t:t | 0.90 |
| Internal NSR cut-off | $/t | 5.19 |
| External NSR cut-off (est.) | $/t | 9.08 |
| Processing rate | t/day milled | 50,000 |
| Processing rate | t/yr milled | 18,250,000 |

The OP mineable resources are reported at an internal cut-off value of $5.19/t based on input parameters above.

NPVS software was used to determine the optimal mining shells with the assumed overall slope angles above. Preliminary phases were selected and preliminary mine planning and scheduling was then conducted on these selected optimal shells. The mineable resources for the Halilağa deposit are presented in Table iii.

Both Indicated and Inferred resources were used in the LOM plan of which Indicated resources represent 56% (137 Mt) of the material planned to be processed. Inferred resources are considered to be too speculative geologically to have the economic considerations applied to them that would enable them to be characterized as mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a PFS or a FS of a mineral project. Accordingly, at the present level of development there are no mineral reserves at the Halilağa Project.

**Table iii: PEA Proposed Mining Plan**

| Description | Unit | Value |
|---|---|---|
| Mine Production Life | yr | 14 |
| Process Feed Material | Mt | 243 |
| Diluted Copper grade (mill head grade) | % | 0.28 |
| Contained copper | Mlbs | 1,516 |
| Diluted Gold grade (mill head grade) | g/t | 0.30 |
| Contained gold | koz | 2,317 |
| Waste | Mt | 234 |
| **Total material** | **Mt** | **477** |
| Strip ratio | t:t | 0.96 |

The mining sequence was divided into a number of stages designed to maximize grade, reduce pre-stripping requirements in the early years and, maintain the plant at full production capacity. The LOM production schedule is shown in Table iv.

A total of 243 Mt of mineralized material is proposed to be processed.

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**Table iv: LOM Production Schedule**

| Description | Unit | Year | | | | | | | | | | | | | | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | |
| Mineralized material mined | Mt | 14.0 | 18.3 | 18.3 | 18.3 | 18.3 | 18.3 | 18.2 | 18.2 | 18.3 | 18.2 | 18.3 | 18.3 | 18.2 | 10.3 | 243.3 |
| Gold feed grade | g/t | 0.49 | 0.38 | 0.41 | 0.28 | 0.35 | 0.19 | 0.23 | 0.27 | 0.24 | 0.21 | 0.29 | 0.34 | 0.27 | 0.19 | 0.30 |
| Contained gold | koz | 220 | 222 | 243 | 167 | 203 | 112 | 137 | 157 | 140 | 124 | 169 | 200 | 161 | 64 | 2,317 |
| Copper feed grade | % | 0.83 | 0.37 | 0.34 | 0.29 | 0.29 | 0.17 | 0.20 | 0.24 | 0.23 | 0.21 | 0.19 | 0.24 | 0.25 | 0.21 | 0.28 |
| Contained copper | Mlbs | 256 | 149 | 139 | 115 | 115 | 68 | 80 | 96 | 92 | 85 | 76 | 97 | 99 | 48 | 1,516 |
| Waste mined | Mt | 10.1 | 9.6 | 21.4 | 24.0 | 22.8 | 22.4 | 22.8 | 23.3 | 23.7 | 24.0 | 23.5 | 6.4 | 0.0 | 0.0 | 234.0 |
| Strip ratio | t:t | 0.72 | 0.53 | 1.17 | 1.31 | 1.25 | 1.23 | 1.25 | 1.28 | 1.30 | 1.32 | 1.29 | 0.35 | 0.00 | 0.00 | 0.96 |
| Total material mined | Mt | 24.1 | 27.9 | 39.6 | 42.2 | 41.0 | 40.7 | 41.0 | 41.6 | 41.9 | 42.2 | 41.8 | 25.5 | 17.7 | 10.0 | 477.3 |
| Oxide material mined* | Mt | 4.96 | 0.00 | 0.09 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 5.04 |

*included in waste total

## Waste Management

Waste rock from mining operations would be deposited in engineered waste rock facilities (“WRF”) adjacent to the deposit. The two waste rock facilities planned are adjacent to one another and would be designed to hold a total of 234 Mt of material

A tailings management facility (“TMF”) is planned to be located within 10 km from the mill. The 60% solids tailings is planned to be sub-aerially deposited, initially behind an earth fill starter dam, and then would be raised using cyclone tailings technology.

Several options were investigated for the TMF in the vicinity of the mill. For the purposes of cost estimation a preferred location was selected. The preferred TMF spans a total footprint of 475 ha. The 565 m long, 55.5 m high (El. 255 m) tailings starter dam would have a design capacity of 20.4 $Mm^3$ which is equivalent to one year of tailings deposition at a processing rate of 50,000 tpd, assuming 20% cyclone underflow.

## Mineral Processing and Metallurgical Testing

Preliminary metallurgical test work was conducted in 2007 and 2011, which focused on developing a preliminary understanding of plant feed sample hardness and flotation response. The results show that the Halilağa mineralized material is of moderate competency and hardness, and amenable to grinding in a conventional SAG-ball milling circuit with pebble crushing. In addition, locked-cycle flotation tests showed that 85% of the feed copper reported to the final concentrate, which results in a grade of 30% copper. Approximately, 61% of the feed gold reported to the final concentrate with a gold grade of 21 g/t. The mineralogy is fine grained and test work indicated a requirement to re-grind to a fine particle size of approximately 20 µm to achieve adequate liberation for flotation. A copper recovery versus copper feed grade relationship has also been estimated based on preliminary variability testing.

The Halilağa process plant and associated service facilities are envisioned to process 50,000 t/d of Run of Mine (“ROM”) material and to produce copper concentrate and tailings. The proposed process includes crushing and grinding of the ROM material, rougher and cleaner flotation, regrinding, and dewatering of copper sulphide minerals. The copper concentrate would be trucked from the site to the Port of Bandirma, while the flotation tailings would be thickened before placement in the TMF.

## Project Cost Estimates

Project costs were estimated from a combination of sources including first principles, reference projects, vendor’s quotes, cost service publications and SRK and Ausenco experience. OPEX for the project are summarized in Table v. The OP mining OPEX assume owner-operated mining including technical/supervisory support staff. Diesel fuel was estimated to cost $1.86/litre (exclusive of VAT) and power was estimated to cost $0.14/kWh.

**Table v: Operating Costs**

| Item | Unit Cost Estimate | | Total LOM Cost (M$) |
|---|---|---|---|
| | ($/t mined) | ($/t milled) | |
| Mining | 1.74 | 3.40 | 828 |
| Processing | | 4.69 | 1,141 |
| Tailings Management | | 0.23 | 56 |
| Environment and water treatment | | 0.05 | 12 |
| G&A | | 0.42 | 102 |
| **Total** | **1.74** | **8.79** | **2,139** |

The capital cost ("CAPEX") estimate for the project is shown in Table vi. A contingency of 25% was used for all items except mining equipment, which was assigned a contingency of 5%. Property acquisition costs are not included in the capital estimate. Working CAPEX were included in the year -1 processing costs.

**Table vi: Capital Cost Estimate**

| Description | Estimate (M$) |
|---|---|
| Primary mining equipment fleet | 171.8 |
| Support equipment | 8.3 |
| Misc. | 7.1 |
| Spare parts | 9.4 |
| Topsoil stripping and mining preparation | 11.7 |
| Crushing | 50.2 |
| Process Plant | 199.9 |
| Concentrate Loadout | 1.7 |
| Tailings Management Facility | 105.2 |
| Utilities | 38.8 |
| On-site Infrastructure | 33.3 |
| Off-site Infrastructure | 13.8 |
| Indirect Costs | 148.7 |
| Owner's Costs | 41.9 |
| Sustaining | 51.6 |
| Closure | 75.0 |
| CAPEX w/o Contingency | **968.3** |
| Contingency | 200.4 |
| **TOTAL CAPITAL COST** | **1,168.7** |

## Economic Analysis and Results

A simplified economic model was developed to determine the pre-tax and after tax net present value ("NPV"), Payback Period and internal rate of return ("IRR") for the project.

Three metal price scenarios were used and for each case, the copper and gold prices were held constant through the project life. The metal prices used were as shown in Table vii.

**Table vii: Metal Price Assumptions for the Three Economic Cases**

| Case | Copper Price ($/lb) | Gold Price ($/oz) | Comments |
|---|---|---|---|
| Case A | 2.90 | 1,200 | Approximate metal prices as per recent SEDAR postings. This is the base case used for OP optimization. |
| Case B | 3.30 | 1,350 | Approximate London Metal Exchange fixed 3-year average to the end of April, 2012. SRK considers this to be an optimistic case. |
| Case C | 3.70 | 1,500 | Approximate London Metal Exchange fixed 2-year average to the end of April, 2012. SRK considers this to be a very optimistic case. |

Common assumptions to all three cases include:

- 7% discount rate for net present value calculation;
- 100% equity financing;
- VAT recovery;
- 20% corporate income tax;
- No tax incentives for large projects from the Turkish government;
- No withholding tax;
- 1.3% royalty on payable copper and 2.6% royalty on payable gold; and
- Mineable tonnes and grade in the LOM mine schedule based on OP optimization conducted using Case A metal prices.

Using the pit design developed for the PEA LOM plan, and a discount rate of 7%, the after-tax break-even metal prices for the project are 19% lower than those used in Case A. I.e. if metal prices fall by 19% from the base case to approximately $972/oz Au and $2.35/lb Cu, the project shows an after-tax $NPV_{7\%}$ of $0 and an IRR of 7%.

Table viii presents a summary of the key economic results.

It must be noted that the economic analysis in this report provides only a preliminary overview of the project economics based on broad, factored assumptions. The mineral resources used in the LOM plan and economic analysis include 44% Inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the inferred resources will be upgraded to a higher resource category. Based on this, there is no certainty that the results of this preliminary assessment will be realized.

**Table viii: Key Economic Results**

| Parameter | Unit | Pre-tax Results | After-tax Results |
|---|---|---|---|
| **Case A ($2.90/lb Cu and $1,200/oz Au)** | | | |
| $NPV_{0\%}$ | M$ | 1,418 | 1,115 |
| $NPV_{7\%}$ | M$ | 675 | 474 |
| IRR | % | 26 | 20 |
| Payback period | Production years | 2.1 | 2.7 |
| **Case B ($3.30/lb Cu and $1,350/oz Au)** | | | |
| $NPV_{0\%}$ | M$ | 2,099 | 1,660 |
| $NPV_{7\%}$ | M$ | 1,081 | 799 |
| IRR | % | 35 | 27 |
| Payback period | Production years | 1.5 | 2.1 |
| **Case C ($3.70/lb Cu and $1,500/oz Au)** | | | |
| $NPV_{0\%}$ | M$ | 2,779 | 2,204 |
| $NPV_{7\%}$ | M$ | 1,487 | 1,124 |
| IRR | % | 43 | 34 |
| Payback period | Production years | 1.2 | 1.6 |

# Conclusions

Industry standard mining, processing, construction methods and economic evaluation practices were used to assess the Halilağa Project. There was adequate geological and other pertinent data available to generate a PEA.

Based on current knowledge and assumptions, the results of this study show that the project has positive economics (within the very preliminary parameters of a PEA) and could be advanced to the next level of study by conducting the work indicated in the recommendations section of this report.

While a significant amount of information is still required for a complete assessment of the project, at this point, there do not appear to be any fatal flaws.

The study has achieved its original objective of providing a preliminary review of the potential economic viability of the Halilağa Project.

As the project develops, it is SRK's recommendation and Truva Bakir's goal to use appropriate environmental standards and safeguards for all aspects of drill testing, engineering testing, construction, operation and closure. If the project is determined to be economic through further studies, Halilaga would be constructed, operated and closed under the Turkish mine permitting regulations. The operation would also conform to best practices for mining, processing (milling and flotation) and closure including tailings management, waste rock management and appropriate water, air and social responsibility standards.

## Risks and Opportunities

As with almost all mining ventures, there are a large number of risks and opportunities that can affect the outcome of the project. Most of these risks and opportunities are based on uncertainty, such as lack of scientific information (test results, drill results, etc.) or the lack of control over external factors (metal price, exchange rates, etc.).

Subsequent higher-level engineering studies would be required to further refine these risks and opportunities, identify new risks and opportunities, and define strategies for risk mitigation or opportunity implementation.

The main preliminary risks identified for the Halilağa Project are, summarized as follows:

- Permit acquisition and stakeholder support;
- Reduced metal prices;
- Geological interpretation and mineral resource classification (44% of the resources used in the mine plan are Inferred);
- Increased OPEX and/or CAPEX;
- Geotechnical and hydrogeological considerations;
- Metal recovery and mineral processing assumptions, including deleterious elements; and
- Water supply and the right to use it.

The following opportunities may improve the project economics:

- The potential to increase mineral resources;
- The potential to upgrade the mineral resource classification of the deposit;
- Contractor mining to reduce initial capital cost (“CAPEX”);
- Staged development to reduce initial CAPEX;
- The potential to acquire less expensive power to reduce OPEX;
- Processing of the oxide material (currently treated as waste);
- Improved metal prices;
- Tax and investment incentivespotentially available to the project.

## Recommendations

SRK believes the project should be taken to the next level of engineering study and economic assessment, typically a Pre-Feasibility Study (“PFS”). It is estimated that a PFS, along with all of the accompanying engineering and field work would cost approximately $5M (exclusive of the recommended additional geology and drilling program outlined below).

To enable a comprehensive and professional PFS study the mineral resource estimate would ideally have a much higher percentage of Indicated resources in relation to total resources and mineralization. The recommendations to attempt to achieve this are:

- Geology/Mineral Resources Program Outline (Cost estimate: $4.3M, Prior to the start of the PFS);

- To increase confidence and upgrade resource classifications, the present drilling density should increase to approximately 50 m spacing with 12,000 m of diamond drilling in 30 holes;
- Conduct additional drilling oriented on an azimuth of 000 (due north) to extend the south-dipping mineralization (approximately 20 drill holes);
- Perform geostatistical evaluation in support of future resource estimates;
- Initiate a geotechnical study to determine potential pit slope parameters which includes five PQ drill holes to a depth of 400 m for a total of 2,000 m;
- Collect advanced metallurgical test samples which would include four PQ drill holes to a depth of 300 m for a total of 1,200 m;
- Social baseline and stakeholder engagement;
- Continue environmental baseline studies; and
- Perform initial condemnation drilling at potential plant site, waste storage, and tailings impoundment areas (900 m in three drill holes).

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# Appendices

**Appendix A**: End of Period Maps

**Appendix B:** Process Flow Sheet Designs

# 1 Introduction

This Preliminary Economic Assessment Technical Report ("PEA" or "report") summarizes the findings of a preliminary economic assessment. This report was prepared by the following qualified persons ("QP"):

- Gordon Doerksen, P.Eng, Project Manager, JDS;
- Dino Pilotto, P.Eng. and Maritz Rykaart, P.Eng., SRK Consulting (Canada) Inc. ("SRK");
- Kevin Scott, P.Eng., Ausenco Solutions Canada Inc. ("Ausenco");
- Garth Kirkham, P.Geo., Kirkham Geosystems Ltd. ("Kirkham") and
- James Gray, P.Geo., Advantage Geoservices Ltd. ("Advantage").

The report was commissioned by Truva Bakir Maden İşletmeleri A.Ş. ("Truva Bakir"), the Joint Venture company for the interests of Teck Madencilik and Pilot Gold in the Halilağa Project. This technical report was written in compliance with disclosure and reporting requirements set forth in the Canadian Securities Administrators National Instrument ("NI") 43-101, Companion Policy 43-101CP, and Form 43-101F1 (collectively, NI 43-101).

The purpose of this technical report is to present the "first time" findings of a PEA of the project. The report is based upon the mineral resource estimate first presented by Gray and Kirkham in the project's March 23, 2012 Technical Report titled "Resource Estimate for the Halilağa Copper-Gold Property NI 43-101 Technical Report"("Gray and Kirkham 2012"). The contents of this report reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Truva Bakir, Teck or Pilot. SRK has and is still providing environmental and permitting consulting services to Truva Bakir for the project. The results of the PEA are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

The economic analysis described in this report provides only a preliminary overview of the project economics based on broad, factored assumptions. No reserves can be declared for the project as insufficient detailed engineering and costing work was done. As per CIM guidelines, reserves can only be declared with a preliminary feasibility-level study.

Garth Kirkham, P. Geo., visited the property between August 13 and August 16, 2011. The site visit entailed inspection of the camp, accommodations, core logging facilities, offices, active drill sites, outcrops, historic drill collars, core storage facilities, core receiving area, core sawing stations and tours of major centres and surrounding villages most likely to be affected by any potential mining operation.

Gordon Doerksen, P.Eng., Maritz Rykaart, P.Eng., Dino Pilotto, P.Eng., and Kevin Scott, P.Eng. visited the project site on February 29 and March 1, 2012. The site visit encompassed an inspection

of the core storage area, a review of selected core and a tour of major centres, local villages and potential open pit ("OP"), plant site, water reservoir and waste management facility locations.

The reader is cautioned that the PEA summarized in this technical report is only intended to provide an initial, high-level review of the project. The PEA mine plan and economic model include the use of a significant portion of inferred resources which are considered to be too speculative to be used in an economic analysis except as permitted by NI 43 101 for use in PEA's. There is no guarantee that inferred resources can be converted to indicated or measured resources and, as such, there is no guarantee that the project economics described herein will be achieved.

The units of measure used in this report are as per International System of Units ("SI") except where otherwise noted. All dollar figures quoted in this report refer to United States dollars ("USD", "US$" or "$") unless otherwise noted. Frequently used abbreviations and acronyms can be found in Section 26.

# 2 Reliance on Other Experts

To prepare this report, the authors relied on Halilağa's 2009-2010 exploration reports and data from previous exploration programs, internal reports, and consultants' reports, including the NI 43-101 Technical Report written by Ian Cunningham-Dunlop, P.Eng., on behalf of Pilot Gold as amended June 7, 2011. It is believed that the information, conclusions, and recommendations are accurate and reliable. It is also believed that the drilling, geological, geochemical, and geophysical data reported by other companies and government agencies regarding the prospect and its environment is accurate and reliable and has been performed by competent professionals operating to industry standards and best practices.

This report was prepared using public and private information provided by the Joint Venture partners and information from previous technical reports listed in Section 19 of this report. The current report also relies on the work and opinions of non-QP experts and non-independent QPs. However, the authors believe that the information provided and relied on for preparation of this report was accurate at the time of reporting, and that the interpretations and opinions expressed by these individuals are reasonable and based on their current understanding of the deposits. Each contributing QP has made a reasonable effort to verify the accuracy of the data used to develop this report and takes full responsibility for the information contained in this report.

The results and opinions expressed in this report are conditional on the aforementioned information being current, accurate, and complete as of the date of this report, and with the understanding that no information has been withheld that could affect the conclusions made in this report. The authors reserve the right to, but are not obliged to, revise this report and its conclusions if and when additional information becomes available, subsequent to the date of this report.

# 3 Property Description and Location

## 3.1 Property Description

The Halilağa Project is located about 40 km southeast of Çanakkale between the villages of Halilağa and Muratlar on the Biga Peninsula in Northwestern Turkey (Figure 3.1). The main area of interest is the Kestane porphyry copper ("Cu")-gold zone located at 483200E, 4419200N UTM Central meridian 27 (ED50 datum) in the central part of the tenement group.

**Figure 3.1: Location Map (Gray and Kirkham 2012)**

## 3.2 Mineral Tenure

The Halilağa Project consists of 19 licenses covering 9,299.79 ha (see Table 3.1 and Figure 3.2). Eighteen licenses are directly held by Truva Bakir, and one license is held by Teck Madencilik for the benefit of Truva Bakir. Three of the licenses are "Exploration-Type", four are 'Operation-Type", and a further 12 licenses are operation pending. The main license hosting the Central Zone at the Kestane porphyry has been converted to an operation license.

According to Turkish mining law, the property boundaries are defined by the coordinate descriptions on the original license application and awarded to the applicant by the government. The licenses that define the Halilağa Project are expressed according to the UTM northern Zone 35 coordinate system and European Datum 1950.

The author is not aware that the properties are subject to any environmental liabilities other than those attached to drill site permits that have been, or may be, issued.

**Table 3.1: Halilağa Tenure Summary, Çanakkale Province**

| TO | ACQ DATE | DUE DATE | AREA (ha) | REGISTRATION NO | LICENSE NO | LICENCE TYPE | OWNER |
|---|---|---|---|---|---|---|---|
| Bayramic | 10/7/2003 | 10/7/2013 | 428.99 | 2399031 | IR-7468 | Operation | Teck |
| Bayramiç | 5/21/2009 | 5/21/2019 | 1,994.47 | 3290089 | 81802 | Operation | Truva Bakir |
| Bayramic | 4/8/2005 | 4/8/2010 | 605.32 | 3052748 | 20050053 | Operation Pending | Truva Bakir |
| Bayramic | 3/2/2012 | 3/2/2022 | 829.10 | 1098491 | 20054260 | Operation | Truva Bakir |
| Bayramic | 3/8/2006 | 3/9/2011 | 131.39 | 2389904 | 20061699 | Operation Pending | Truva Bakir |
| Bayramic | 3/8/2006 | 3/9/2011 | 433.61 | 2428284 | 20061700 | Operation Pending | Truva Bakir |
| Can | 3/8/2006 | 3/8/2011 | 823.24 | 3074271 | 20061704 | Operation Pending | Truva Bakir |
| Can | 3/21/2012 | 3/21/2022 | 1,328.73 | 3080113 | 20064172 | Operation | Truva Bakir |
| Can | 7/13/2007 | 7/13/2012 | 250.00 | 3129124 | 200707000 | Operation Pending | Truva Bakir |
| Bayramic | 11/13/2007 | 11/13/2012 | 75.15 | 3146203 | 200710080 | Operation Pending | Truva Bakir |
| Bayramic | 11/13/2007 | 11/13/2012 | 686.20 | 3146197 | 200710081 | Operation Pending | Truva Bakir |
| Bayramic | 11/13/2007 | 11/13/2012 | 404.10 | 3146206 | 200710082 | Operation Pending | Truva Bakir |
| Bayramic | 11/13/2007 | 11/13/2012 | 128.09 | 3146212 | 200710083 | Operation Pending | Truva Bakir |
| Bayramic | 11/13/2007 | 11/13/2012 | 58.05 | 3146215 | 200710084 | Operation Pending | Truva Bakir |
| Bayramic | 11/13/2007 | 11/13/2012 | 146.64 | 3146209 | 200710087 | Operation Pending | Truva Bakir |
| Can | 3/7/2008 | 3/7/2013 | 8.80 | 3167539 | 200801694 | Operation Pending | Truva Bakir |
| Bayramic | 3/7/2008 | 3/7/2013 | 18.80 | 3167537 | 200801695 | Exploration | Truva Bakir |
| Can | 5/3/2012 | 5/3/2019 | 769.94 | 3270271 | 201200524 | Exploration | Truva Bakir |
| Bayramic | 5/3/2012 | 5/3/2019 | 179.17 | 3255378 | 201200525 | Exploration | Truva Bakir |

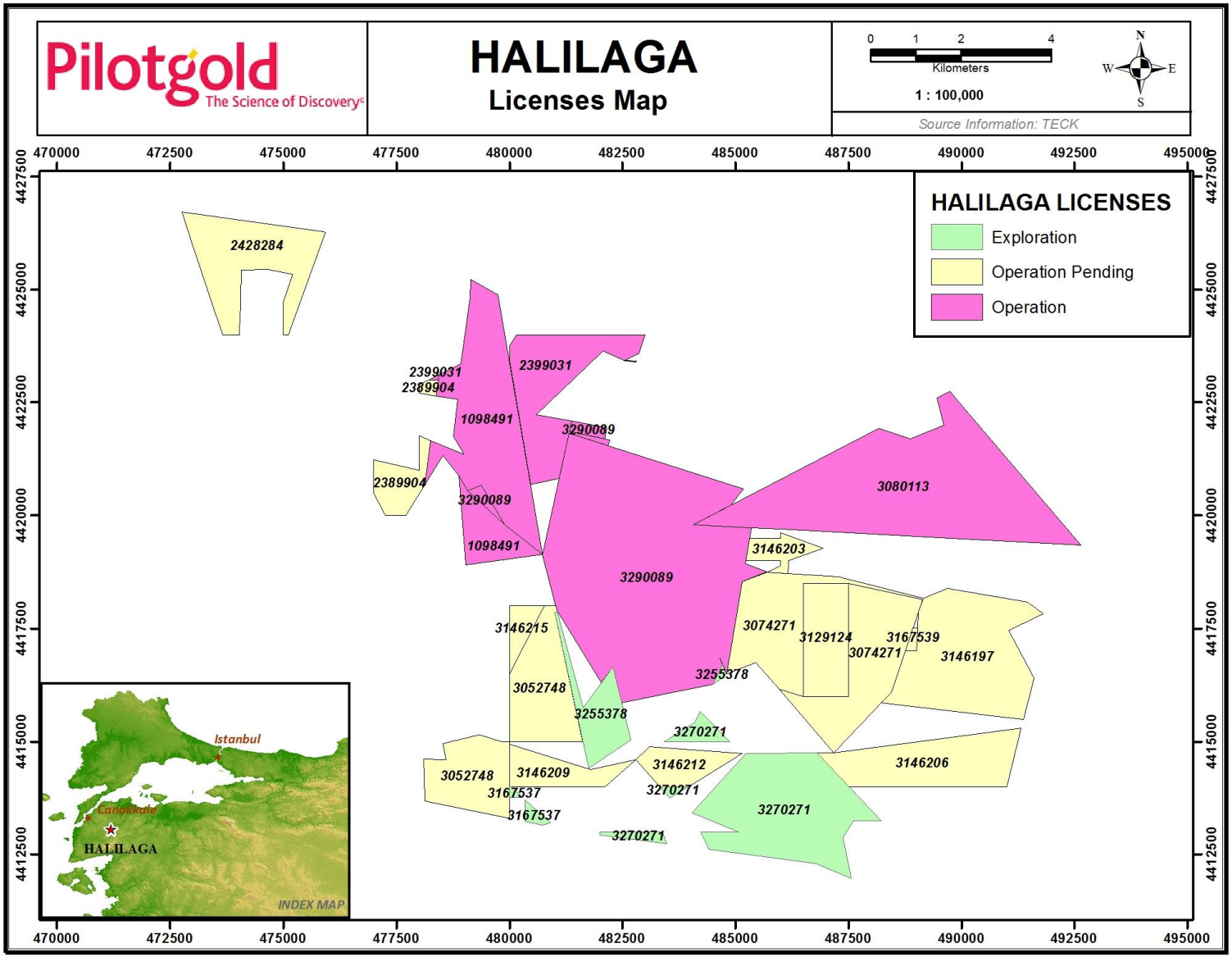



**Figure 3.2: Halilağa License Map**

## 3.3 State Royalties

The Government of Turkey would receive a 2% of Net Smelter Royalty (known as the State's rights) for precious metals and copper. Because the project uses copper flotation, the copper net smelter royalty is reduced to 1%.

The State's rights, paid by the license holder, would be distributed to; the Special provincial Administration of Canakkale (25%), Turkish Treasury (50%), and Sub-provincial Administration (in this case Bayramiç and Can as it is in between those sub-provinces) to be used for infrastructure (25%). The Council of Ministers can apply a maximum 25% discount in the State's rights rates depending on the type of mineral, the region of production, and other criteria.

The project is located on State-owned land, therefore an additional 30% of the royalty payment is required to be paid, increasing the gold royalty to 2.6% and the copper royalty to 1.3%.

Each year the license holder pays the royalty on the last day of June.

## 3.4 Joint Venture Agreements

In 2002, the Halilağa Project was acquired at auction by Teck Cominco Arama ve Madencilik Sanayi Ticaret A. Ş. (now Teck Madencilik Sanayi Ticaret A.Ş ("Teck Madencilik"). In 2004, Teck Madencilik and Fronteer entered into an option agreement that covered several properties in the Biga area (including the Halilağa Project) that enabled Fronteer to acquire 100% interest in the properties subject to certain earn-back rights by Teck Madencilik; Fronteer was required to spend $2.0 million on exploration over four years, with a first-year firm commitment of $200,000.

After the initial program by Fronteer, Teck Madencilik decided to earn-back into Halilağa on November 30, 2006. Subsequently, Teck Madencilik earned a 60% interest in the property by investing $2.5 million during 2007. On December 31, 2009, Teck Madencilik declined to earn an additional 10% interest in the Halilağa Project.

Teck Madencilik (60%) and Fronteer Gold Inc. (Fronteer) (40%) formed a Joint Venture company called Truva Bakir Maden İşletmeleri A.Ş. ("Truva Bakir") which owned, or had beneficial interest in, the licenses that comprise the Halilağa Project. Fronteer's interest in the Halilağa Project was subsequently transferred to Pilot Gold Inc. in April 2011.

## 3.5 Mining Rights and Title in Turkey

Truva Bakir's project activities are required to follow the mining codes as set out within Turkey's state and local environmental regulations. Truva Bakir must protect the environment from spills; capture and dispose of hazardous material including aviation fuel; reclaim disturbed ground; cap and plug drill holes; and remove all refuse. All of the necessary forest and environmental permits were obtained for 2011, including permission for timbering, road construction, drill site construction, and drilling.

The Law Regarding Amendments on Mining Law, and other Certain Laws, No. 5995, was approved by the Grand National Assembly of Republic of Turkey on June 10, 2010, but has not been put into force yet. The effective date of such law has been stated as its publication date on the Official Gazette (Kayıkçı, 2010).

On May 16, 2012 Truva Bakir received a renewed Group IV minerals license, renewing the exploration permit at the Halilağa Project. Shortly thereafter, a variety of the necessary operating permits were also been received to keep the operation licenses covering the resource area in good standing. At the date of this report, Truva Bakir awaits receipt of a business operating permit relating to workplace safety and sanitation (a "GSM permit") from the Office of the Governor of Çanakkale (the "Governor"). This permit is the final permit required to renew the copper-gold Operations Permit, allowing copper-gold related operations (beyond exploration) at Halilağa to continue.

Mining rights and minerals are exclusively owned by the State. The ownership of the minerals in Turkey is not subject to the ownership of the relevant land. The State, under the mining legislation, delegates its rights to explore and operate to individuals or legal entities by issuing licences for a determined period of time in return for a royalty payment. Mining rights, with respect to certain types of mines, belong to State or State enterprises.

The licenses for mining rights are granted to Turkish citizens, legal entities established under Turkish laws, and some authorized public bodies. Companies established under Turkish law, according to the provisions of the Turkish Commercial Code, are Turkish companies even if they are established by foreign persons with 100% foreign capital. Consequently, there is no distinction between the mining rights that may be acquired by local investors and those that may be acquired by foreign investors, provided that the foreign investors establish a company in Turkey under Turkish law (Ozkan, 2007).

Under the Turkish Mining Law, mines have been divided into five groups which are subject to different terms and conditions on licensing principals and procedures.

According to Article 6 of the Mining Law, mining rights can be defined as the licenses and permits for prospecting and operating mines, and can only be granted to the following real or legal persons:

- Turkish citizens;
- Legal entities incorporated under the laws of the Republic of Turkey, including legal entities having foreign shareholders, provided that the articles of association of such legal entities shall contain a mining operation clause; and
- Authorized public entities and administrative bodies.

The three types of licenses granted for prospecting and operating the mines stated under the laws of the Republic of Turkey are as follows: a prospecting license, enabling its holder to carry out prospecting activities in a specific area; an operation license, enabling its holder to carry out operational activities within the same area as stated in the prospecting license; and an operation permit, enabling its holder to operate a specific mine as specified in the operation license.

Prospecting activities are defined as all mining activities, other than those carried out for production. As an exception, the prospecting licensee shall have the right to carry out production and sale activities in respect to a maximum 10% of the proven mine reserves within the prospecting license period, as long as the licensee applies to the General Directorate of Mining Affairs (Maden İsleri Genel Mudurlugu) and provides the prospecting activity report.

Terms and Procedures for Prospecting License: The application for a prospecting license shall be made to the General Directorate of Mining Affairs for the mining groups other than group I (a). For the group I (a) mines the application shall be made to the relevant Provincial Special Administration

(Il Ozel Idaresi). Prospecting licensees are obliged to submit a prospecting activity report to the General Directorate of Mining Affairs within two years of obtaining such a license, as provided in Annex 5 of the Regulation on the Implementation of Mining Law. In case of non-compliance with such provision, the prospecting license guarantee provided by the prospecting licensee would be recorded as revenue. The term of the prospecting license is three years and may be extended for a period of two years upon the demand and submission of the second prospecting activity report by the prospecting licensee.

An operation license may only be obtained if the prospecting activities are carried out initially. The operation license would be granted to the prospecting licensee for the proved, potential, and feasible mine reserve area determined during the prospecting period and would be valid for a period of at least 10 years upon the evaluation of the aforementioned documents by the General Directorate of Mining Affairs. The term of the operation license may be extended for at least three years upon the application of the holder of the prospecting license and operation license with a new operation project; however, such a term cannot exceed 60 years. The Council of Ministers is authorized to grant an extension more than sixty years.

The operation licensee must apply to the General Directorate of Mining Affairs to obtain necessary permits stipulated under Article 7 of the Mining Law within three months.In case the operation licensee fails to apply to the General Directorate of Mining Affairs within the stated period, the guarantee shall be recorded as revenue and a further period of three months shall be provided to the operation licensee for application. If non-compliance continues within the additional three months period, the operation license shall be revoked. Thereafter, within 15 days as of the issuance date of the permits required, the operation licensee is granted an operation permit. Such permit is granted only for the proved mine reserves area which is determined during the prospecting period. Operation activities must be started within one year commencing as of the date of the operation permit. In case the operation activities do not commence within such a one year period, 10% of the production amount shall be paid as the state right for each year of inactivity. The terms of the aforementioned necessary permits and an operation permit shall be the same as the term of an operation license and in case an extension is granted to an operation license, the term of such permits too shall be extended accordingly.

The Law Regarding Amendments on Mining Law, and other Certain Laws, No. 5995, was approved by the Grand National Assembly of Republic of Turkey on June 10, 2010, but has not been put into force yet. The effective date of such law has been stated as its publication date on the Official Gazette (Kayıkçı, 2010).

## 3.6 Environmental Studies, Permitting and Social or Community Impact

In December 2011 Truva Bakir submitted an Environmental Impact Assessment (“EIA”) report to the Turkish Ministry of Environment and Urban Planning (the “Ministry”) in connection with an application to renew the principal licenses that comprise the Halilağa Project. The license and EIA contemplate a small-scale copper-gold test mining scenario as part of an ongoing effort to advance understanding of the Halilağa Project.

SRK became aware, through Pilot Gold, that in August 2012, Truva Bakir was informed that the Ministry had been served a legal petition by certain claimants in Turkey to annul the Ministry’s approval of the EIA report.

The petition filed with the Canakkale Administrative Court (the "Court") names the Ministry as the respondent and does not name Truva Bakir or its shareholders. The petition also requests suspension of any activities contemplated in the EIA by way of an interim decision to be granted by the Court. Truva Bakir was advised that the Ministry was expected to respond to the petition in the 4th quarter of 2012.

Pilot Gold believes the petition is without merit, and even if the petition is successful, and the EIA is annulled, the ability to continue the planned 2012 exploration program at the Halilağa Project is unaffected. Should the EIA be annulled, Truva Bakir would revisit the EIA process, and resubmit an updated EIA to address identified issues, and update for the results of ongoing economic and environmental studies, including those contemplated in this report. Truva Bakir has also been advised that the petition does not challenge or impact the underlying tenure on the licenses that comprise the Halilağa Project.

Truva Bakir is working to identify and facilitate the development of an additional and alternate water supply for the Muratlar village located several km west of the Kestane Zone. Golder and Associates has been contracted by Truva Bakir to assist.

Environmental Law No. 2872 and Environmental Impact Assessment Regulation and Labour Law No. 4857 are the two principle regulations governing mining activities; there are additional regulations, based on those principle regulations, which are applicable to mining industry.

Further to the amendments made in the Mining Law, most of the mining activities are excluded from the EIA, but for those mining activities that require an EIA report, the necessary procedures must be completed within three months.

Specific environmental regulations apply to sensitive areas, such as forest areas, hunting areas, special protection areas, national parks, agricultural areas, including culturally-protected areas, coastal areas, tourism regions, and military forbidden zones. In order to reduce the bureaucracy in the granting of mining rights, the General Directorate is given the authority to designate and delineate such areas. The new regulations do not require any further permission for exploration activities to be conducted in such areas. A notification to the relevant authorities (such as relevant forest directorate) shall suffice. However, mining operational activities in such areas shall be subject to additional permissions and EIA requirements.

Truva Bakir is building long-term relationships within surrounding communities by:

- Providing information and regular updates to the public regarding exploration activities in the region;
- Providing a safe environment for exploration;
- Providing a regular flow of information between project leaders and community groups;
- Ensuring all concerns, questions, and feedback from community groups are efficiently and effectively addressed;
- Assessing the environmental and social risks associated with each phase of the project;
- Implementing mitigation measures when necessary;
- Introducing a Grievance/Complaint Resolution procedure to give all stakeholders an opportunity to present a dispute or concern related to exploration activities;
- Supporting transparent and fair employment strategies at the local level;

- Allocating resources to study and procurement social, economic, and educational infrastructure within those settlements directly affected by the project; and
- Supporting community investment focused on education and health to promote local development, decrease poverty and enhance quality of life.

The goal for all community investment ("CI") projects is to have the initiative benefit the community as a whole, rather than benefitting an individual or specific group within the community. In addition to the immediately-required infrastructure and educational support, CI activities would include long-term, sustainable goals for local communities that generate employment opportunities; improve long-term health services; and increase the capacity for growth as defined within the business objectives of the company.

## 3.7 Water Supply Considerations

Truva Bakır has been carrying out exploration activities in the Halilağa region since 2006. The exploration program has concentrated around the valley between the hills of Camtasi and Kunk where the three springs that supply drinking and utilization water for the village of Muratlar are located.

All three springs reside in the mine exploration area and the quantity and the quality of the water obtained from the springs would potentially be affected once any proposed pit development is initiated. Given this valid concern, an alternative resource for the drinking and utilization water for the village of Muratlar needs to be identified.

In general, drinking water in the region is supplied from springs flowing out of the hills at higher elevations by using gravity to transport water. The spring water discharge is collected in water depots close to springs. From these small depots, the water flows downhill through small diameter PVC pipes to the large storage depots which hold approximately 70 $m^3$ of water and are located near the villages. From these storage depots, the water is distributed to the individual houses within the village. The distance from the springs to villages typically varies from 3 km to 11 km.

For its water resource study, SRK focused on four villages namely, Muratlar, Hacıkasım, Halilağa and Keçiağılı. They all use the water resources from the same region, which encompass the hills on the South East of the potential mining area. SRK considered the usage of the resources in this region to avoid future problems, such as:

- Muratlar is using the three springs located on the eastern edge of the potentially planned pit area;
- Hacıkasım is using the water resource at a distance of 8 km in the south;
- Halilağa is using three springs to the South. Two of them are mixed in a depot and from there the mixed water is shared with Keçiağılı; and
- Keçiağılı is using the two springs on the south that are shared with Halilağa.

There are approximately 600 people living in the village of Muratlar. With the assumption that each occupant consumes about 150 L water per day, the approximate water demand for the village of Muratlar has been estimated to be 90 $m^3$/day, or approximately 1 L/s.

Three springs that currently supply water for the village of Muratlar are mixing in a collection depot downstream. Previous flow measurements conducted at this point are 0.78 L/s and 0.58 L/s as

reported in December 2010 (Golder, 2010) and June 2009 (SRK), respectively. Measurements show that current discharge is significantly below the estimated demand.

The results of a water quality analysis reported by Golder Associates (2010) shows that the quality of the samples taken from the village of Muratlar main depot on June 1, 2010 satisfied the accepted maximum concentrations of major ions (primary criteria) in the Ministry of Health's "Regulation on quality of water intended for human consumption". However, the low pH level and the elevated aluminum content do not comply with the secondary criteria and is therefore not recommended for human consumption. Additionally, according to the Ministry of Environment and Forestry's "Turkish Water Pollution Control Regulation" the water is classified as "Low Quality Groundwater (Class III)" which means that the use of such water is possible after a purification process but its use is determined by the degree of purification attainable economically and technologically and with respect to health.

SRK conducted a systematic screening of all possible supply options. For this purpose, the following actions were performed to collect the missing information about the water resources within the region:

- Action 1: Extend the hydrocensus to find water resources at higher altitudes;
- Action 2: Evaluate the limestone aquifer;
- Action 3: Assess the water quality of the two existing wells: MWS1 and MWS2; and
- Action 4: Locate alternative groundwater wells.

In Action 1, two main springs and one small developed spring can supply the required quality and quantities necessary on the hills in the Southwest of the village of Söğütalan. Because of the elevation, (390 m a.s.l.) the water can also be transported to the village of Muratlar by gravity. According to wet season measurements, approximately 3 L/s of good quality (above drinking standards) water was identified. It is recommended that this action be taken.

In Action 2, the marble aquifer was further appraised in terms of quantity and quality. In terms of quantity, there is an excess amount of water which is not used by the villages of Halilağa or Keçialan and this amount can easily satisfy the demands of Muratlar. In terms of quality, the arsenic concentrations were found to be higher than the regulatory standards and this would need to be addressed if it is to be used as drinking water. In order to achieve this, a "blending" of this water with the two small springs found within the scope of the Action I study was proposed. Chemical modeling has also shown that blending is an adequate and reliable option for reducing elevated levels of arsenic. With this blending, approximately 2 L/s of good quality water can be obtained; this is typically the recorded flow during the wet season.

In Action 3; the water quality of the two existing wells: MWS1 and MWS2 (drilled for the village of Muratlar) was assessed with standard sampling and analytical procedures. Arsenic concentrations in both wells were found to be high and this issue would need to be addressed.

Among the actions outlined here, Action 1 was found to be the most feasible and sustainable solution, and Action 2 was considered to be a backup solution for the water supply.

It is proposed and discussed in detail late in Section 15.5 report that a water reservoir be built to collect water for the use in the process plant. The reservoir would be down stream of the proposed tailings facility and would be filled with surface run-off water and would mitigate the need for the project to use well water for mineral processing.

# 4 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Halilağa Project is located 40 km southeast of the city of Çanakkale between the villages of Halilağa and Muratlar on the Biga Peninsula, Northwestern Turkey (Figure 4.1). Well maintained, paved roads provide excellent access from the main highways to the facilities and property. The deposit area is accessed by a series of good forestry roads. Population and agricultural activity is concentrated in the valleys, while most areas of active exploration are located in highlands which are predominantly forested and owned by the Turkish government (see Figure 4.2). The labour force for the project is planned to be sourced largely, from nearby villages.

**Figure 4.1: Location of Halilağa, Western Turkey**

## 4.1 Accessibility

Year-round access to the properties for field exploration is possible, although snow during winter may temporarily restrict vehicle movement.

## 4.2 Local Resources and Infrastructure

The region is well-serviced with electricity transmission lines and generating facilities with the most significant being a large coal-fired power plant outside the town of Çan. (This power station currently has insufficient capacity to power the project, however. Further discussion of power supply options is contained in Section 16.4)

There is no exploration infrastructure (for example, buildings, core storage, core logging facilities, and camp accommodations) located within the property limits

## 4.3 Climate

The Biga Peninsula has fertile soil and a Mediterranean climate with mild, wet winters and hot, dry summers. Temperatures range from 15° to 35° Celsius in the summer season and -20° to -10° Celsius in the winter months. The annual rainfall is approximately 30 cm, generally falling as mixed rain and snow in late fall and winter.

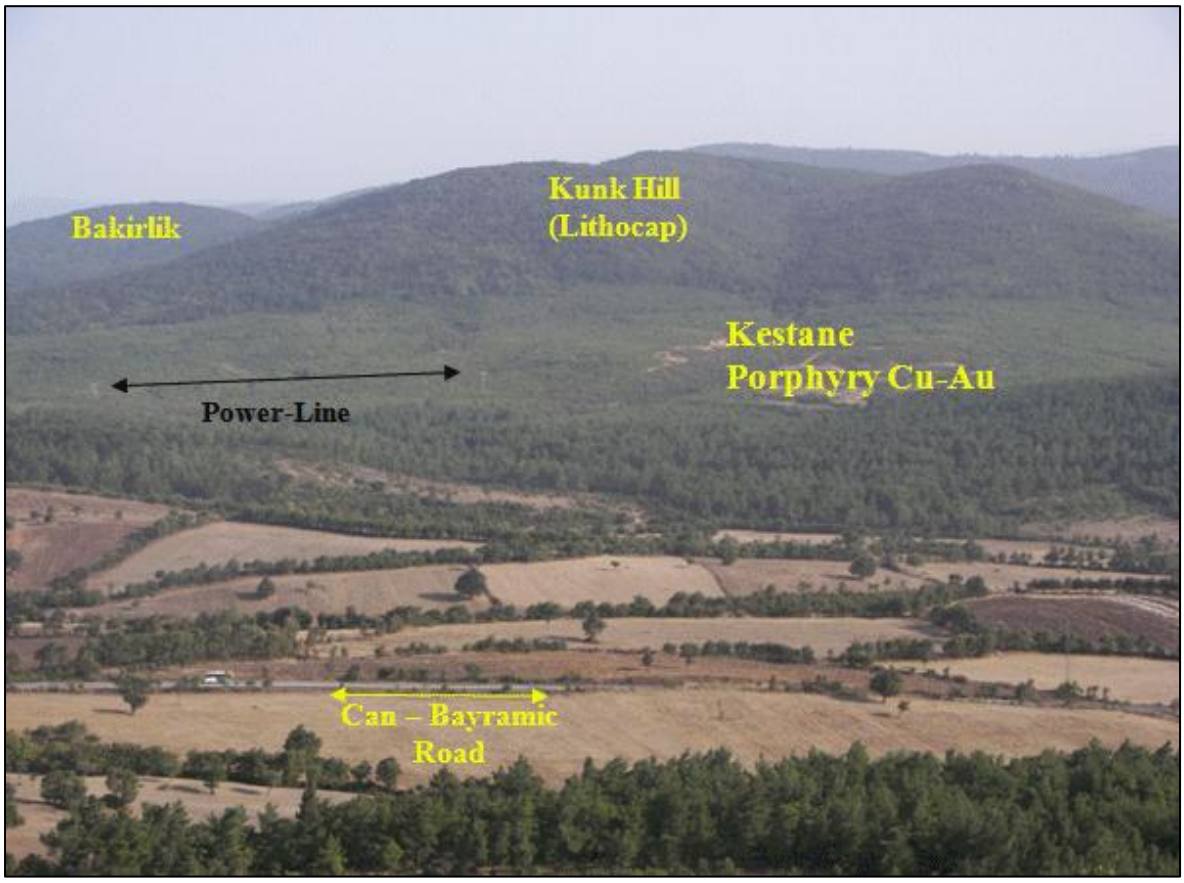
Vegetation in higher evelavation is predominantly coniferus, trees; and, various crops and grasses predominate in areas developed for farming.

## 4.4 Vegetation

Vegetation in the area is made up of plants able to withstand a relatively dry climate including scrub oak and various low-lying shribs as well as pine trees. Various grasses also grow in the area and provide grazing for livestock.

## 4.5 Physiography

The central Halilağa Project contains a 4-km-long, E-W-trending topographic high, with the Kestane porphyry located on the northern flank of the hill (see Figure 4.2). The highest elevations on the property are approximately 600 m with the Kestane Zone occurring at an elevation of approximately 350 m




Note: Kestane area and drill roads are in the middle ground crossed by a power line.

**Figure 4.2: General View of Halilağa (view to the southeast)**

# 5 History to 2000

The following details outline the exploration activities on the Halilağa Project that occurred prior to involvement of Truva Bakir or related parties. Historic exploration activities were focused primarily on the Halilağa and Pirentepe targets. These targets, along with the Halilağa North target, are now all within the Halilağa Project and are owned by Truva Bakir.

## 1988-1991

The government's General Directorate of Mineral Research and Exploration of Turkey (MTA) conducted a regional scale exploration program over the Biga Peninsula between 1988 and 1991. In the vicinity of the Halilağa village, MTA located zones of silicification and argillic alteration at Halilağa North covering approximately 700 m by 150 m and trending NE-SW at Sağluk Tepe. Samples with significant gold values (> 0.5 g/t Au in rock) originated from the western part of Sağluk Tepe in silica-argillic altered zones. Southeast of Halilağa at Taskesilen-Kocatas Tepe, a NE-trending silicification zone, measuring 1,000 m long and 300 m wide, was also mapped.

MTA drilled two diamond drill holes totalling 302 m to test a geochemical anomaly identified by rock chip sampling at Halilağa North. MJTC-16 intersected narrow intervals of gold mineralization and returned 0.58 g/t Au over 13.85 m. MJTC-17 did not intersect any significant mineralization. Core samples were selectively analyzed for gold, silver, copper, lead, zinc, antimony, mercury, and molybdenum.

## 1997

Cominco collected several rock chip samples from silicified outcrops at Halilağa North and at Kumlugedik Hill area, where numerous gold anomalies have been detected. The highest- grade sample from Halilağa North contained 1.17 g/t Au and the highest grade sample from Kumlugedik contained 2.2 g/t Au.

## 1998

A total of 293 soil samples were collected from the Kunk-Kumlugedik lithocap by Cominco. The most anomalous gold in these soil samples highlights the area east of Kumlugedik and Guvemtasi Hills.

## 2000

Prior to ownership, Cominco conducted reconnaissance soil sampling and rock chip sampling. A total of 107 samples were collected over five N-S soil lines. The assay results returned anomalous gold (> 10 ppb) from the East and Southwest Limbs of Celdiren Hill.

There are historical adits and a small pit on the property; however, the background and production history on these workings are undocumented and unknown, and would not significantly affect future development.

The authors are not aware of any previous mineral resource estimates, reserve estimates, or mineral production from the property.

# 6 Geological Setting and Mineralization

## 6.1 Regional Geology

The Halilağa Project is located in the south-central part of the Biga Peninsula in Western Turkey. The geology of the peninsula is complex and characterized by various lithological associations made up of the following and as shown in Figure 6.1:

- Collisional to extensional Tertiary granitoids and associated volcanic rocks;
- Neogene sediments;
- Tertiary (Eocene) units;
- Ophiolitic rocks;
- Permian and Mesozoic rock units; and
- Basement metamorphic rocks.

Metamorphic rocks occur in three distinct associations:

- The Çamlıca metamorphics: These occur as a NE–SW-trending strip of quartz-micaschists (Kemer unit) with calc-schist, quartzite, and amphibolite horizons;
- Kazdağ Massif: The second group of metamorphic rocks is exposed in the Kazdağ mountain range and consists of high-grade metamorphic rocks (amphibole-bearing gneisses with marble intercalations, meta-ophiolite, marble, and gneiss); and
- Karakaya Complex: This group comprises two distinct lithologic associations: a strongly deformed greenschist-facies metamorphic sequence of metabasites intercalated with phyllite and marble accompanied by minor amounts of metachert, metagabbro, and serpentinite, and a thick series of low grade metamorphic rocks.

These rocks are, in turn, unconformably overlain by weakly-deformed, Jurassic–Lower Cretaceous sandstones and limestones. The sequence includes the following:

- Triassic terrigeneous to shallow marine clastic sedimentary rocks;
- Middle to Upper Jurassic platform-type neritic limestones;
- Lower Cretaceous pelagic limestones; and
- Upper Cretaceous-Palaeocene volcanic and sedimentary rocks.

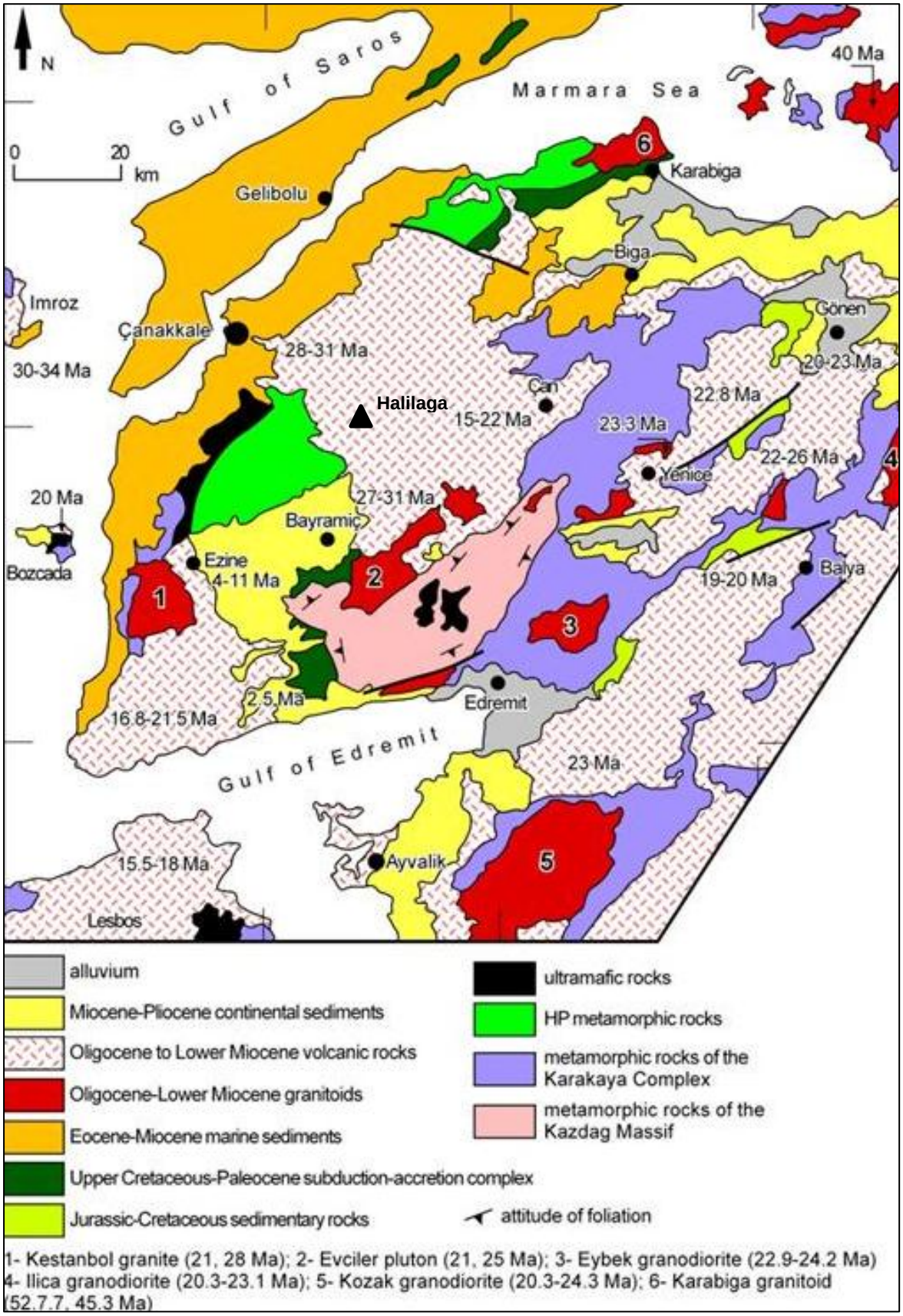
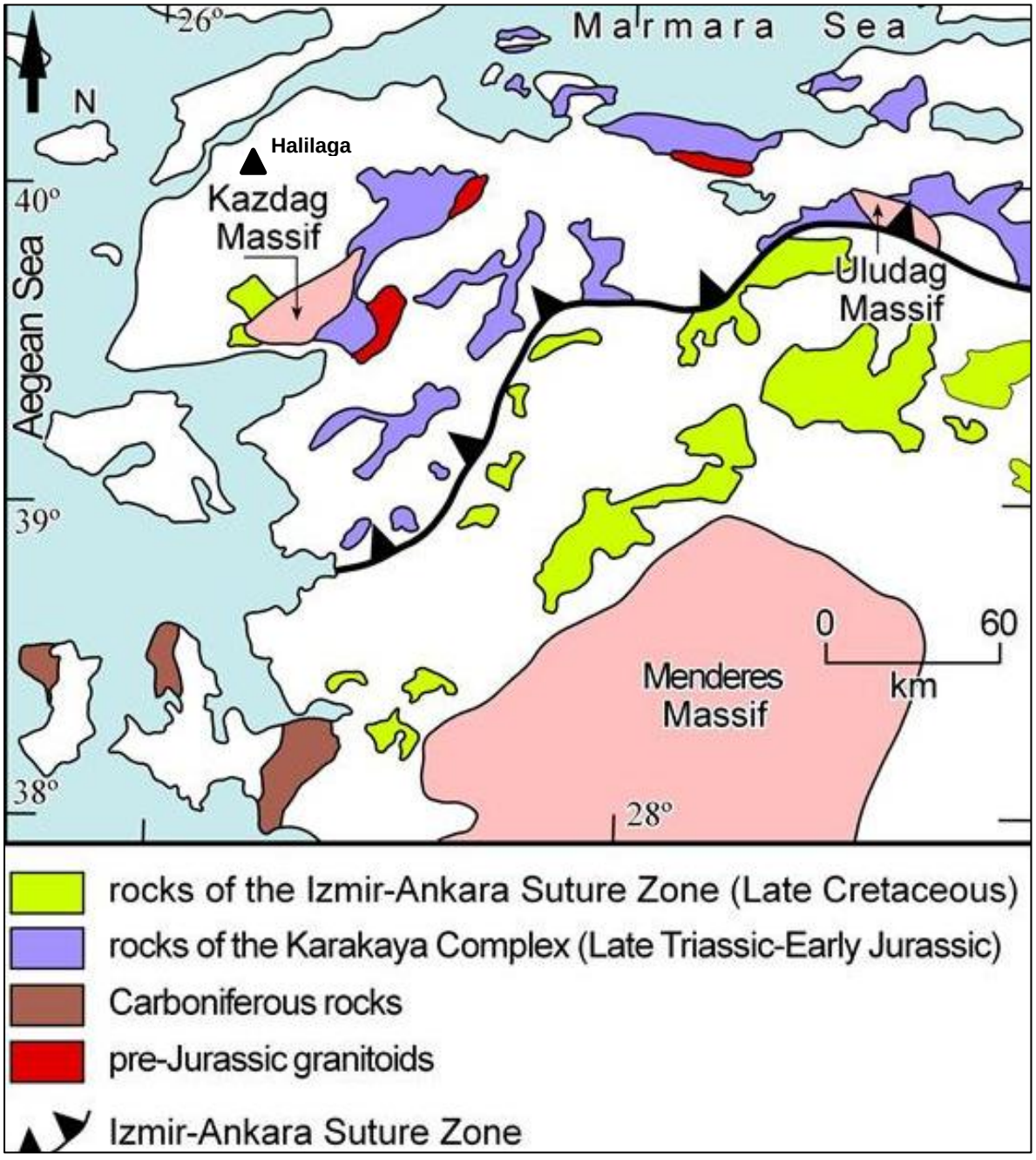



**Figure 6.1: Simplified Geological Map of Biga Peninsula (Grieve, 2009)**

West of the Kazdağ Massif, a Late Cretaceous to Palaeocene oceanic accretionary melange tectonically overlies the Kazdağ metamorphic rocks along a low-angle detachment fault/shear zone. The Kazdağ Massif, the shear zone, and the accretionary melange are intruded by a late Oligocene pluton. The Çetmi Melange, the Karakaya Complex, the Kazdağ Massif, and the Jurassic–Lower Cretaceous are unconformably overlain by Miocene lacustrine sediments and volcanic rocks (see Figure 6.2). All these rocks are, in turn, unconformably overlain by the fluviatile and alluvial sediments of Plio–Quaternary age.




Note: This shows the distribution of the Karakaya Complex and the ophiolitic rocks of the İzmir-Ankara suture zone

**Figure 6.2: Simplified Geological Map of Western Turkey (Grieve, 2009)**

Neogene calc-alkaline plutonic and associated volcanic rocks form the most widespread rock units in the Biga Peninsula. The field relations, geochemical data, and geochronological data suggest that magmatic activity commenced with intrusions of granitoids, coeval with initial phase of volcanic activity, and continued with a second phase of volcanism. Apart from large plutonic bodies, there are several small-scale granitoid bodies in the region. They were emplaced mostly into the basement metamorphic rocks and associated volcanics and generated well-developed high temperature metamorphic aureoles. Geochronological data (K-Ar and Rb-Sr ages on mica) suggests that these plutonic rocks range in age from 20.3 Ma to 45.3 Ma, but with a tight cluster between 20.3 Ma and 25.0 Ma.

The volcanic rocks appear to be associated with plutonic rocks both in space and time. K-Ar and Rb-Sr ages of the volcanic rocks range from 15.0 Ma to 34 Ma. They display two distinct associations which are commonly considered as lower and upper volcanic associations. The lower volcanic association is composed mainly of andesite and dacite lavas and associated pyroclastic rocks, intercalated with sedimentary rocks. The upper volcanic association is represented by a sequence of andesite, latite, rhyolite, basaltic andesite, basaltic trachyandesite lavas, and associated pyroclastic rocks that alternate with coal-bearing, lacustrine sediments. The rhyolite occurs as small domes and lava flows.

The upper sequence of volcanic rocks is unconformably overlain by coarse- to fine-grained clastic rocks, and lacustrine limestones and mudstones that were deposited within E–W-trending fault-bounded basins.

### 6.1.1 Regional Structural Geology

The structural geology of the Biga Peninsula is complex and poorly understood. One of the main difficulties arises from the fact that the region is subject to active dextral strike-slip faulting and N–S continental extension. The region forms a transition between two active tectonic structural styles and many of the active structures reflect the interaction between strike-slip and normal faulting. Nevertheless, the active geologic structures of the region fall, based on their orientation, into three distinct groups:

- NW–SE faults;
- NE–SW faults; and
- E–W faults.

Among these, the most prominent faults are a number of NE–SW-trending dextral strike-slip fault systems that represent the branches of the North Anatolian Fault Zone in the Biga peninsula (for example, Yenice-Gönen and Etili faults seen in Figure 6.3 and Figure 6.4). The second group of prominent structures comprise E-W-trending normal faults. They occur along the northern margin of the Gulf of Edremit and along the southern margin of Lake Manyas. There are less prominent NW-SE-trending faults which are interpreted as synthetic Riedel shears of the North Anatolian Fault Zone.

NE–SW-trending faults controlled both the sedimentation and volcanic activity during the Oligocene–Early Miocene period when the so-called lower volcanic association and coeval sediments were deposited. It is suggested that Late Miocene Ezine alkaline basalts are common along E–W-trending normal faults, thus suggesting a possible feeder relationship.

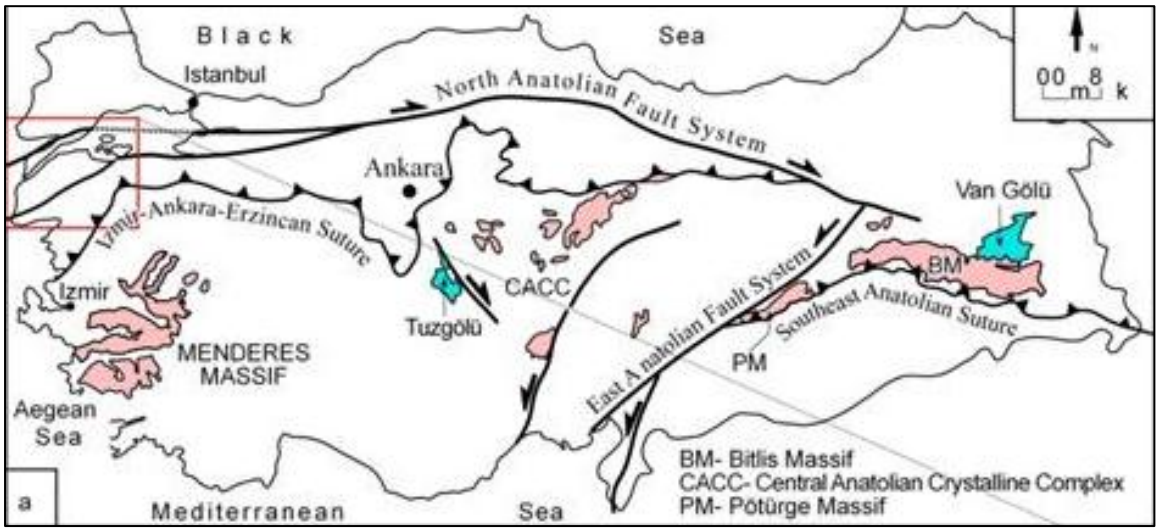



**Figure 6.3: Simplified Neotectonic Map of Turkey Showing Major Neotectonic Structures**

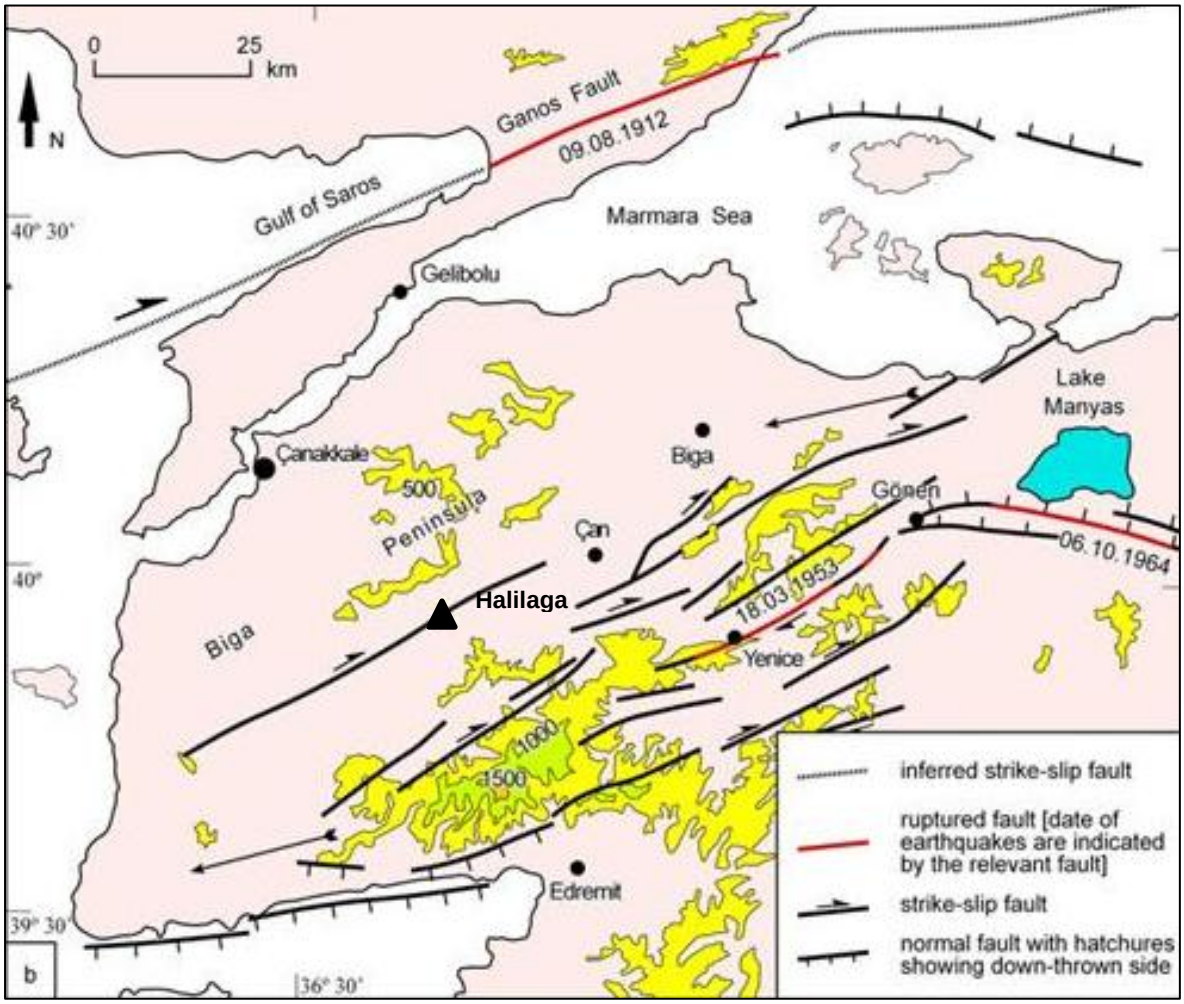



**Figure 6.4: Major Neotectonic Structures of the Biga Peninsula (Grieve, 2009)**

## Property Geology

The Halilağa area is mainly underlain by volcano-sedimentary basement cover sequences of Oligo-Miocene age. The basement consists of schists and carbonates that outcrop to the southeast of the Bakirlik area. The granodioritic batholith intrudes into the basement rocks including carbonates and generates metasomatism and skarnification.

The Kestane porphyry was emplaced into a volcano-sedimentary sequence and produced hornfels halos around the Kestane area. The geology and alteration of the area with units listed from youngest to oldest is as follows:

## Colluvium

The prospect area is extensively covered by colluvium, particularly on the steeper slopes of the Kunk Tepe, Guvemtasi Tepe, Tasyatak Tepe, and Kumlugedik Tepe. Note that a tepe is defined as a hill. In road cuts, this colluvial cover can be up to 2 m–3 m thick, limiting the total exposure of bedrock outcrop across the property. The geology and alteration maps do not show this cover and are an interpretation of the underlying lithologies.

## Volcanic Cover Unit

Loose, un-mineralized andesitic/basaltic volcanics ± brownish sediments are distributed to the N and NE of the Kestane porphyry. The fault contact (110/65N) between the cover unit and quartz feldspar porphyry (QFP) was observed in some holes and in a trench wall located at HD-38 and HD-47.

## Polymictic Conglomerate

Polymictic conglomerates are widely distributed, particularly in the NW of the area. The conglomerate consists of rounded to sub-rounded clasts of silicified tuff, porphyritic volcanics, and vein quartz in a chlorite/epidote altered matrix. The timing relationships are not clear. The quartz clasts are possibly sourced from granites to the SE of Halilağa.

## Quartz Feldspar Porphyry (QFP)

Quartz-feldspar porphyry intrusive rocks are well-exposed in the Kestane Zone (Central Zone) as shown in Figure 6.5. Previous soil sampling in the area returned up to 1,265 ppb Au. Quartz phenocrysts are commonly preserved. To the west, the porphyry appears to be more phyllically altered (+/- argillic and silica) and the style of veining is restricted to E-W striking veins of coarse saccharoidal quartz up to about 30 cm wide. Skarn subcrop and float are spatially related to the porphyry and occur as large float blocks and smaller fragments associated with an old pit. This occurrence may indicate the presence of a melange zone at the northern margin of the porphyry. The porphyry also appears to have hornfelsed clastic sediments to the west, which are strongly pyritized.

## Volcanic/Subvolcanic Rocks

The dominant units in the mapped area are andesitic/dacitic volcanic and sub volcanic rock and a feldspar-hornblende intermediate felsic subvolcanic unit with significant quartz. Part of the andesite unit includes tuffs implying, at least in part, an extrusive origin. Where the tuffs can be mapped over larger areas they have been identified on the geological map as a separate unit.

Inferred discontinuities between the tuff units and the andesite/tuff sequence, and the feldspar-hornblende-quartz volcanic unit as shown on the maps define a NW-SE-trending structural lineament, along which the Kestane quartz-porphyry was emplaced.

## Andesitic Tuffs

This unit forms part of the same volcanic pile as the andesite flows. However, where enough outcrops are present they are depicted on the interpreted geology map as andesitic tuff.

## Quartzites and Carbonates

Basement clastic and carbonate rocks in the vicinity of historic mining activity at Kumlugedik Tepe have been propylitically altered (chlorite, epidote, actinolite, and quartz) as determined by thin section analysis.

## Schistose Basic Volcanics and Sediments

Scattered float on Bakirlik Tepe of un-mineralized schistose basic volcanic and sedimentary rocks likely represent the Palaeozoic basement rocks of the Biga Peninsula, although no outcrops have been observed.

Figure 6.5 shows the current understanding of the local geology.

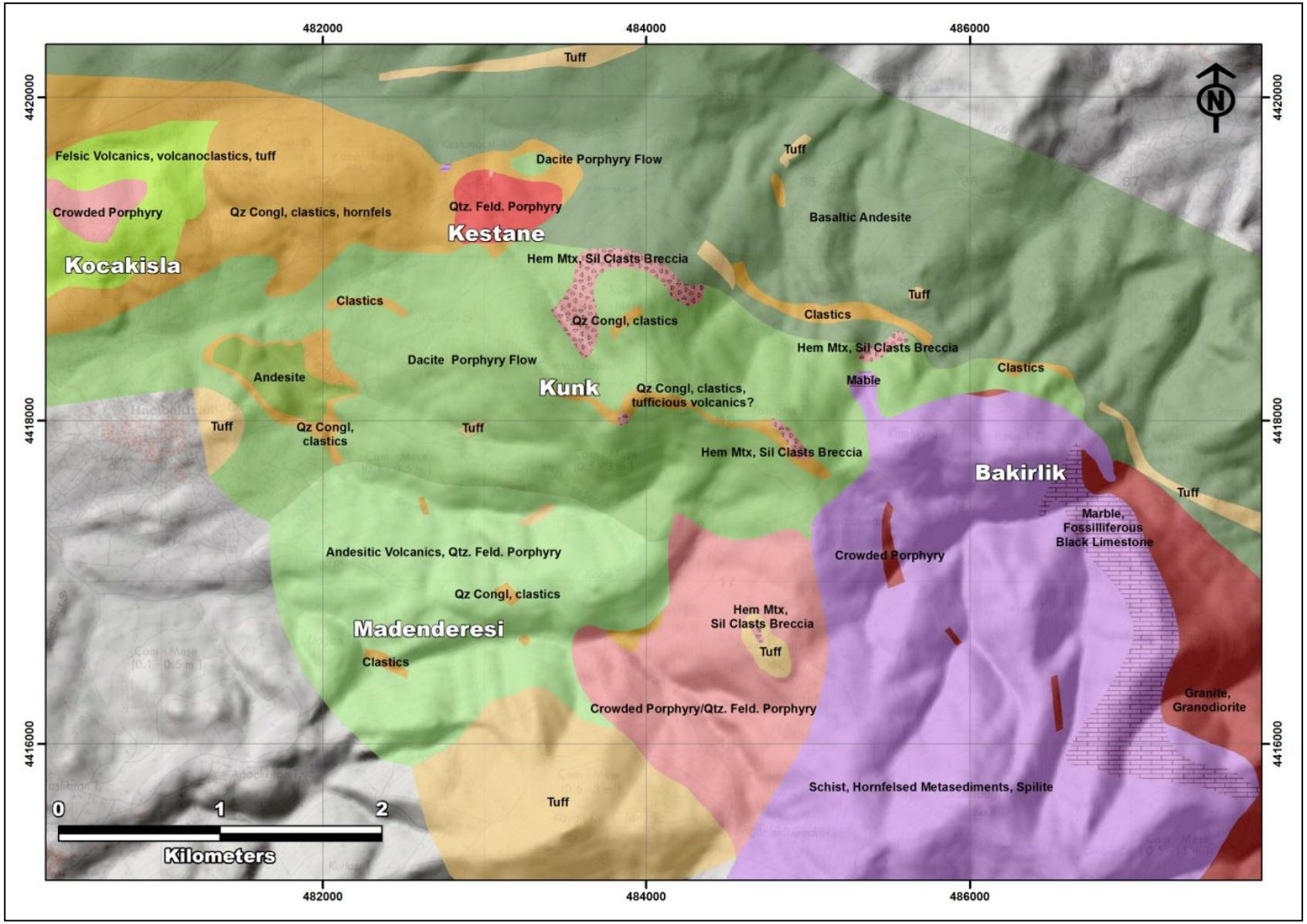



**Figure 6.5: Halilağa Local Geology**

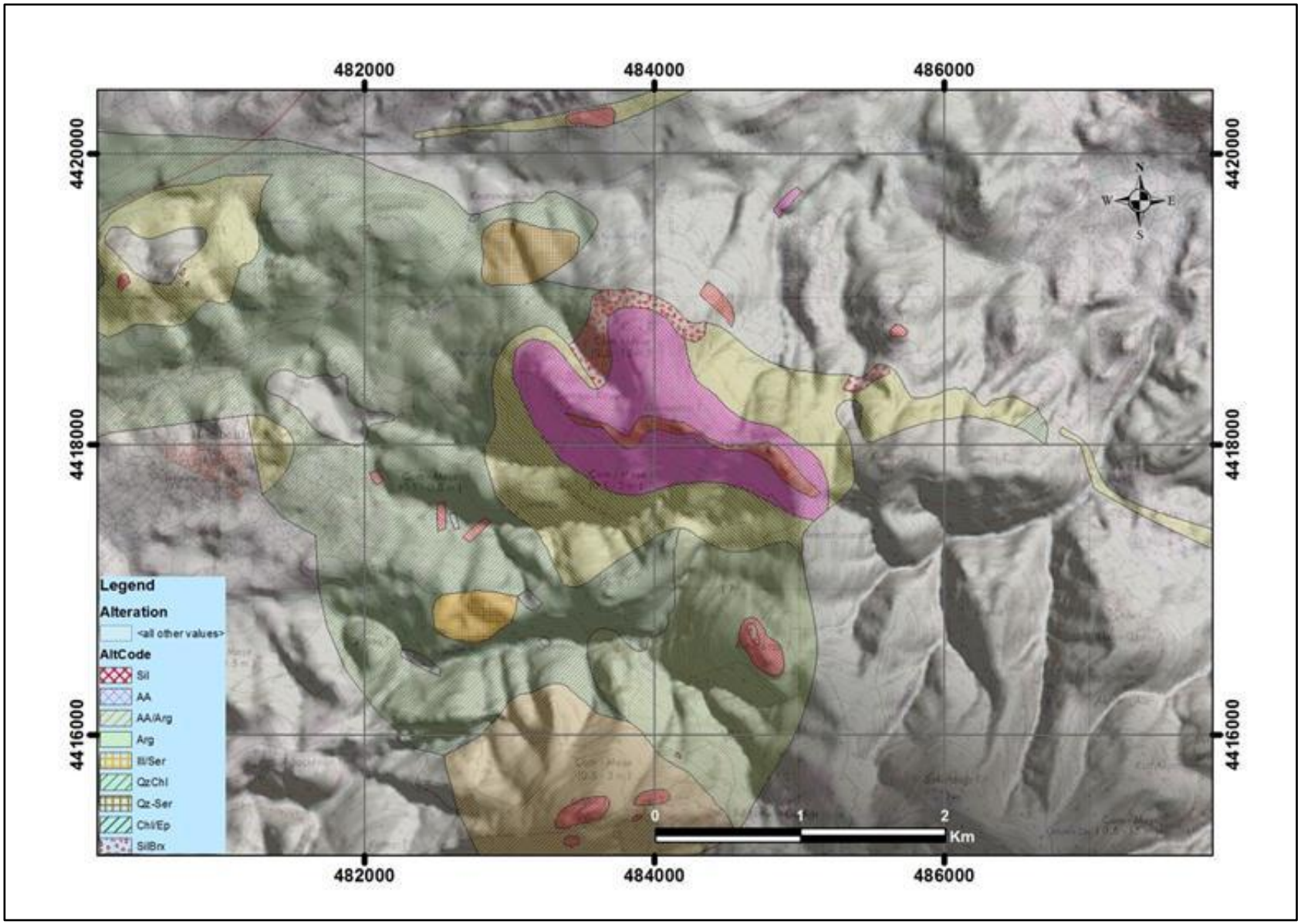



**Figure 6.6: Halilağa Alteration Map**

### 6.1.2 Alteration

The Halilağa alteration system covers a large area of more than 4 km x 2 km and displays all porphyry alteration types as well as related epithermal and skarn systems. The Kestane porphyry outcrops are characterized by potassic overprinted by phyllic alteration whereas Kunk-Kumlugedik Hill-tops are characterized by silicification surrounded by advanced argillic to argillic and distal propylitic alteration (see Figure 6.6). Skarn-related alteration is located around the Bakirlik and Bostanlikbasi areas.

The details of the alteration styles are described here:

#### Propylitic/Sub propylitic

Basement units in the vicinity of historic mining activity at Kumlugedik Tepe have been propylitically altered (chlorite, epidote, actinolite, and quartz) as determined by thin section analysis. Sub-propylitic alteration of the volcanic and sub-volcanic rocks is typically weak to moderate in intensity. The occurrence of chlorite, smectite, and varying degrees of carbonate and pyrite define this alteration zone at Halilağa, which grades out into unaltered rocks to the NE and SW.

#### Argillic

A broad zone of argillic-altered andesitic volcanic rocks and tuffs are developed around the advanced argillic alteration zones developed along the Kunk Tepe-Guvemtasi Tepe-Tasyatak Tepe ridge line. Typically the alteration preserves texture with feldspars altering to kaolin and smectites, with the groundmass replaced by varying degrees of silica. The best exposures of this style of alteration are found in excavations for spring water on the NE flanks of Guvemtasi Tepe.

#### Advanced Argillic (Quartz-Alunite)

Advanced argillic alteration along the NW-trending ridge lines of Guvemtasi Tepe and Tasyatak Tepe ranges from quartz-alunite altered andesites at Guvemtasi, to near complete silica replacement of tuffs and volcanics at Tasyatak Tepe. This zone is inferred to represent the distal outflow of acidic fluids from the porphyry source. Previous rock chip sampling along this zone returned anomalous (< 1 ppm) gold. Soil sampling returned localized anomalies with respect to gold, bismuth, and mercury.

Commonly, the quartz-alunite alteration is cut by mm-scale goethite-hematite veinlets generally trending NNE which are well-exposed in saw-cut channels.

#### Silica-pyrite

Texture preservative and pervasive silica-pyrite alteration is mapped in the SW portion of the property within andesites and a felsic porphyritic volcanics, probably of dacite composition. A surficial argillic alteration zone has developed on weathered outcrops due to the oxidation of pyrite. The silica-pyrite alteration is inferred to be an early stage of alteration resulting from moderate temperature sulphur-rich and metal-poor fluids derived from a porphyry intrusion. Rock sampling has been limited in this area; however, soil sampling shows that this area is not anomalous with respect to gold. Later higher-temperatures and more saline fluids resulted in gold deposition associated with the advanced argillic alteration developed along NW-trending zones as described above. In this model, the advanced argillic alteration post-dates and cuts the earlier silica-pyrite alteration.

### Silicic

Texturally-destructive silica replacement of tuffs and volcanics occurs on Tasyatak Tepe. The zone here dips steeply to the NE and is approximately 25 m thick in outcrop.

### Phyllic

The quartz porphyry located at Kestane is phyllically altered (sericite and pyrite); although near-surface it is commonly overprinted by supergene argillic alteration. Phyllic alteration is both pervasive and fracture-controlled.

### Potassic

The Kestane porphyry is dominated by potassic alteration (biotite, K feldspar, and magnetite). The highest gold and copper grades in the core are associated with early biotite + abundant magnetite (5-8%) + chalcopyrite associated with A- and B-type quartz veins. Chalcopyrite, in some cases accompanied by trace amounts of pyrite, is the dominant sulphide in the potassic alteration. Bornite was not found.

## 6.1.3 Structural Geology

The Halilağa area is highly influenced and transected by the presence of structures which relate to, or are part of, the North Anatolian Fault System. The general architecture of the fault appears highly controlled by major ENE-WSW transtensional structures. Local transpression has been interpreted for the Kestane area producing the exhumation of the porphyry and older stratigraphy.

The ongoing transtensive kinematics has reactivated pre-existing extensional structures within zones of ~N60E to EW orientation. Within this regional transtension regime, local restraining and releasing bends and oblique extensional relay ramps could represent key elements which increase structural permeability controlling the emplacement of porphyry, meso-, and epithermal mineral deposits.

A first approach on a regional interpretation suggests that the north-tilting of the pre-quaternary geology could be due to Late Tertiary to recent kinematic of the major transtensive ENE-WSW structures. These structures could also have constitutive boundaries for Tertiary volcano sedimentary depocentres by accommodating several asymmetrical and sub-parallel grabens. All post basement volcano-sedimentary sequences show a clear tilting to the NW, N, and NE by 20º-35º which increases at the vicinity of a major structure.

Significant faults act as bounding structures for the economic mineralization including the North Fault and South Fault. These faults show normal displacement and together form a horst with the porphyry mineralized Kestane stock in the centre.

The South Fault separates propyllitized to phyllic altered volcanic cover to the south, from the mineralized Kestane stock to the north. It is E-W trending and dips -60º to the south. This structural contact was intersected in holes HD-41 (between 123.70 m and 173 m), HD-97 (between 124.5 m and 130.9 m), and HD-94A (between 126.7 m and 132.7 m).

The North Fault separates propyllitized volcanic cover and hornfels to the north, from the mineralized Kestane stock to the south. It is WNW trending and dips approximately 65º to the north. This structural contact was intersected in holes HD-49 (at 14.6 m), HD-67A (at 229 m), HD-42 (between 238 m and 246 m), HD-82 (at 313.6 m), and HD-87 (at 378.6 m). In addition, there is a low angle structure between mineralized material stock and the volcanic cover unit that starts at 483250 E line and dips at -20º to the east.

# 7 Deposit Types

Halilağa is classified as a copper-gold porphyry system. Advanced argillic alteration and gold mineralization at Kunk Hill and Pirentepe are classified as high sulphidation epithermal mineralization. Copper mineralization in the Bakirlik Tepe area is classified as proximal copper skarn. All three types are related to magmatic-hydrothermal activity associated with intrusion of the Kestane porphyry stock and other intrusions in the area.

## 7.1 General Background

Porphyry copper deposits are widely distributed at convergent plate margins, in association with arc-related volcanic and intrusive rocks of intermediate composition. They typically occur in association with skarn and intermediate- to high-sulphidation epithermal base and precious metal deposits. They range in size from tens of millions of tonnes to billions of tonnes of mineralized material. Copper and copper-gold porphyry systems supply a large proportion of the world's copper, molybdenum, and gold. Some of the largest Cu-Au porphyry systems include Grasberg (Indonesia), Oyu Tolgoi (Mongolia) and Bajo de Alumbrera (Argentina). Associated high sulphidation epithermal deposits can also be quite large, with the best example being Yanacocha in Peru. Typical hypogene grades of porphyry deposits range from 0.2% to 0.8% copper. Copper to gold ratios vary widely in Cu-Au porphyry deposits, but a Cu%:Au ppm ratio of 1:1 is not uncommon.

A very large volume of literature exists on porphyry deposits because of their large size and economic importance. The following description of a porphyry deposit comes from a summary by Sillitoe (2010).

Porphyry deposits are typically centred on polyphase stocks and porphyry dyke swarms, with skarn deposits formed adjacent to and epithermal deposits above the porphyry mineralization (Figure 7.1). The metal endowment of a porphyry system is related to the geochemistry of the oxidized magmas that contribute to the formation of the stocks and dykes, with gold and/or molybdenum commonly found in association with copper. Porphyry deposits typically occur in association with Mesozoic and Tertiary intrusions, probably as a result of poor preservation of older rocks.

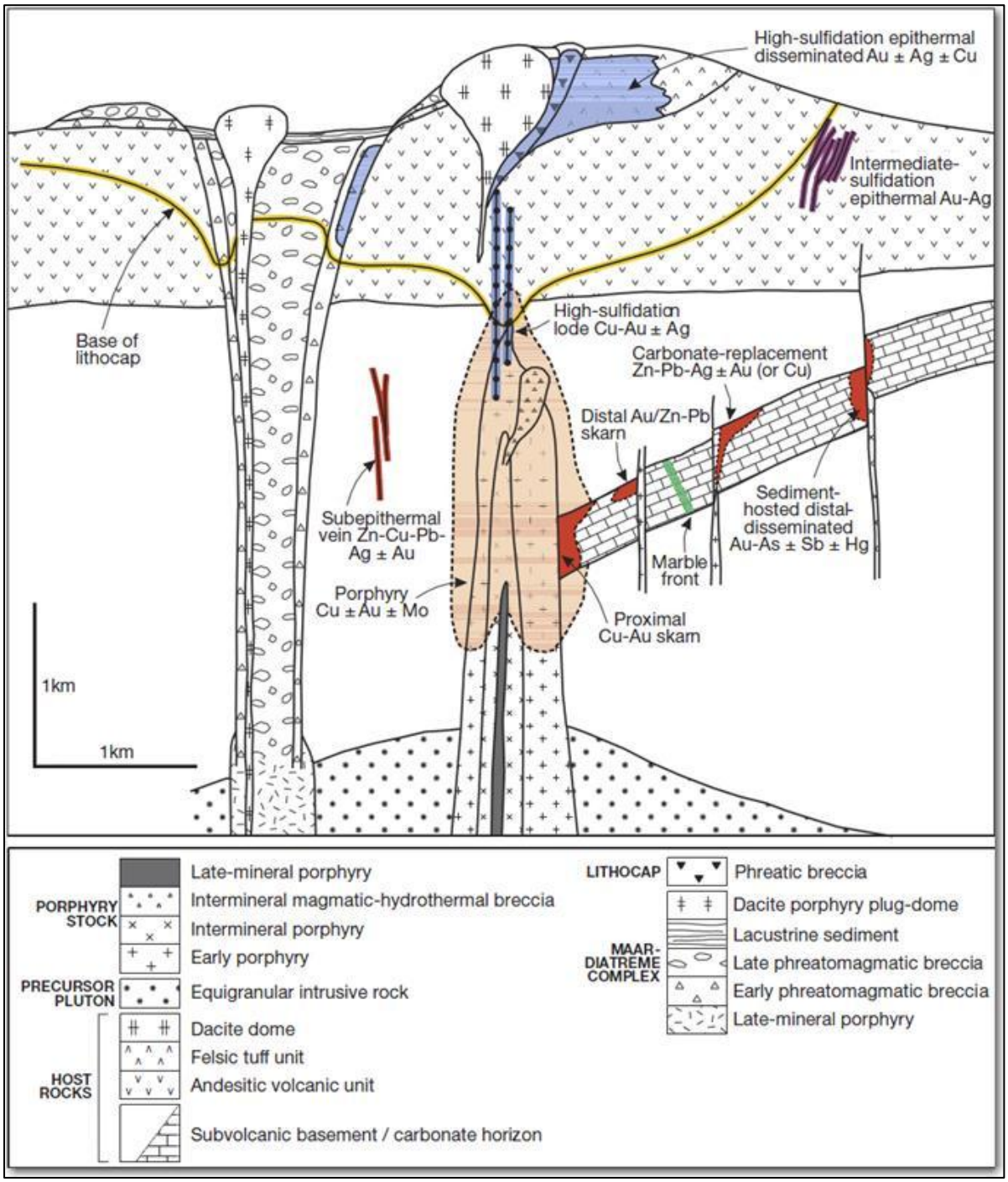



**Figure 7.1: Anatomy of a Telescoped Porphyry System, Sillitoe (2010)**

Porphyry systems are typically zoned from a potassic altered (biotite-potassium feldspar) core overlying barren, calcic-sodic altered rock, upward through phyllic altered (sericite or chlorite-sericite) margins to propylitic altered (chlorite-epidote) rocks (Figure 7.2). Porphyry systems also grade upward into advanced argillic, argillic and silicic alteration related to epithermal mineralization. Alteration zoning may be complex and overlapping due to successive injections of magma into country rocks. The vertical distance between porphyry mineralization and overlying epithermal mineralization may range from one telescoped kilometre to several un-telescoped kilometres.

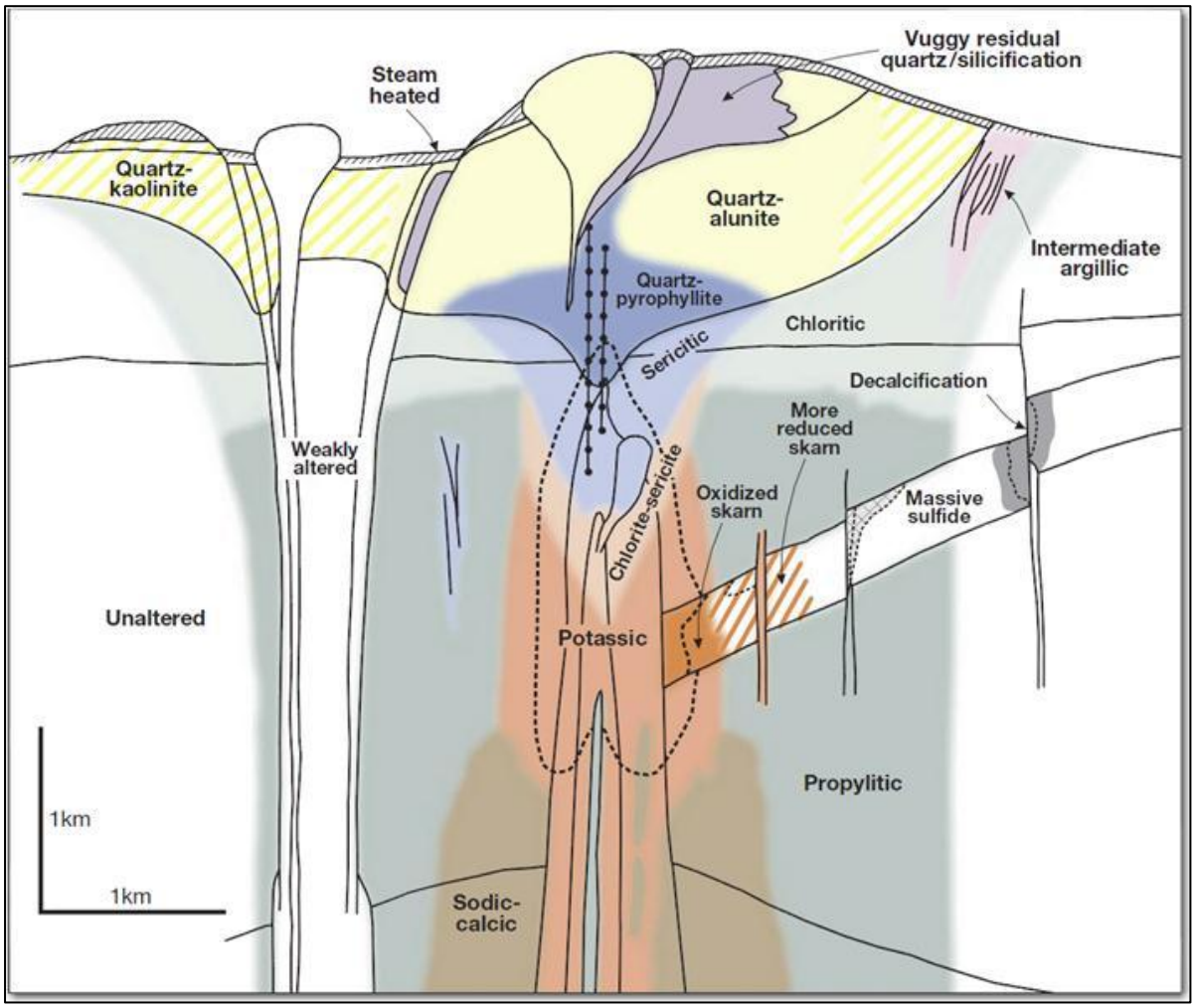



**Figure 7.2: Generalized Alteration-Mineralization Zoning Pattern for Telescoped Porphyry Copper Deposits, Sillitoe (2010)**

Hypogene copper mineralization is disseminated and veinlet-hosted, and zoned from bornite-rich in the core through chalcopyrite to pyrite in distal areas. Magnetite (in Cu-Au porphyries) and molybdenite (in Cu-Mo porphyries) are common accessory minerals.

Quartz veins and veinlets as stockworks and sheeted arrays are ubiquitous in these systems, and typically occur in a sequence from early quartz-feldspar A-veins, through quartz-sulphide (mainly chalcopyrite-molybdenite) B-veins with potassic-altered margins to late, sulphide-dominant (primarily pyrite) D-veins with phyllic altered margins (Gustafson and Hunt 1975). Veining in Cu-Au deposits may differ slightly, with quartz-magnetite-chalcopyrite and magnetite-dominant M-veins present or dominant (Arancibia and Clark 1996).

Due to the large amount of disseminated pyrite in most porphyry systems, these systems are susceptible to supergene weathering and leaching. Copper is oxidized and leached from areas above the water table and deposited as chalcocite and other supergene copper minerals at or near the water table, leading to enrichment in copper grades. Supergene chalcocite enrichment can increase grades locally by 200% to 300% or more, with a significant impact on the overall economics of these deposits.

Alteration and mineralization associated with high sulphidation epithermal deposits in the upper portions of porphyry systems consist of pyrite, enargite, and covellite hosted in silicified and often brecciated silicified volcanic rocks, accompanied by advanced argillic alteration minerals, including pyrophyllite, alunite, dickite, and kaolinite (Hedenquist et al 2000). Alteration and mineralization at this elevation in the system comprise a lithocap and may be far more laterally extensive than the porphyry deposit itself.

Proximal skarn deposits are located laterally from porphyry deposits (Meinert, 2000). They consist of replacement bodies within the causative intrusion (endoskarn) or marginal to it (exoskarn). Skarn may be particularly well-developed in limestones and other calcium or carbonate-rich rocks. Skarn alteration assemblages include garnet, pyroxene, wollastonite, magnetite, actinolite, pyrite, magnetite, and chalcopyrite.

## 7.2 Halilağa Porphyry Mineralization

Cu-Au porphyry, skarn, and high sulphidation epithermal alteration and mineralization are all found in close proximity in the Halilağa area. Recognizing that the high sulphidation deposits underlying many of the hills in the area could be overlying or concealing porphyry deposits at depth, led to the discovery of Halilağa in the valley bottom adjacent to the Kunk high sulphidation epithermal system; this occurred long after the epithermal potential of the hilltops was explored. The Kestane Cu-Au porphyry system exhibits alteration and mineralization zoning typically seen in deposits of this type. This includes a low-grade, potassic altered core and relatively high copper and gold grades, often associated with a high density of quartz-magnetite-sulphide veins in areas flanking the core. Mineralization is also associated with an overlap of phyllic and potassic alteration, a small supergene chalcocite blanket, and adjacent areas of hornfelsing and skarn alteration. One departure from the standard porphyry model is a lack of bornite mineralization in the core of the deposit. Either this represents a departure from the model, or possibly the core of the deposit has not yet been located

Magmatic-hydrothermal processes related to hypabyssal quartz diorite to quartz monzonite intrusions have resulted in porphyry copper-gold (proximal, Kestane); epithermal gold (distal, Kunk Hill); and skarn gold-silver +/-base metal mineralization (Bakirlik + Kestane W) in and near Halilağa.

## 7.3 Kestane Porphyry Mineralization

At the Kestane porphyry, most quartz veins are ‘B-type’, averaging 5% of the rock by volume, but locally up to 20% (shown in Figure 7.3), and ‘A-type’ veinlets are rare or difficult to recognize on outcrops. The fact that B-veins, shreddy biotite, and D-veins can be recognized in an outcrop is significant because these indicate the presence of moderately intense potassic alteration with a moderate sericitic overprint (Einaudi 2007). Given the tendency for the best grades in porphyry Cu-Au deposits to be associated with potassic alteration associated with abundant quartz veins, the possibility of high primary grades in chalcopyrite or chalcopyrite (± magnetite) assemblages can be inferred from these outcrops. Additionally, the moderate degree of sericitic alteration suggests that chalcocite enrichment below the leached cap might be present because acidic conditions at the water table favour chalcocite rather than copper-oxides, silicates, and carbonates (Einaudi 2007).

**Figure 7.3: B-Veining in Kastane Porphyry**

The geometry of gold mineralization at the Central Zone at Kestane is approximately 800 m x 1600m in the E-W direction and porphyry-style copper-gold mineralization has been intersected in drilling to a depth of approximately 500 m.

### 7.3.1 Epithermal – High Sulphidation

At Kunk Hill, ENE- and ESE- trending ridges are capped by extensive areas of silicified volcanic rocks. These lithocaps are formed by massive to vuggy silica (quartz alunite), extensive areas of strongly limonitized breccias, and argillic to advanced argillic alteration which are the host for high sulphidation gold mineralization. Limited drilling in this area has returned anomalous gold.

### 7.3.2 Skarn Mineralization

At Bakirlik, (Four km ESE of Kestane) copper-bearing garnet skarn is present. It occurs in carbonaceous limestone near the contact with a quartz monzonite intrusion. Whether the Bakirlik occurrence is the same age as Kestane remains to be determined. West of the Central Zone, another copper-bearing garnet skarn is present which requires further investigation

# 8 Exploration Since 2002

Since acquisition in 2002, exploration activities performed by Teck Madencilik and Truva Bakir have primarily centred on the Halilağa and Pirentepe areas.

## 2002

The property was acquired by Teck Madencilik via auction.

## 2002-2004

No activity on the property.

## 2004

Teck Madencilik and Fronteer entered into an option agreement that covered several properties in the Biga area, including the Halilağa Project.

## 2005-2006

Fronteer/Teck Madencilik conducted an exploration program consisting of geological mapping, surface geochemical sampling, a pole-dipole IP survey, and a ground magnetics survey. A drill program was initiated at Halilağa on November 15, 2006.

The soil and rock chip sample results highlighted the porphyry-related mineralization of the Central Zone at Kestane. Rock chip sampling of oxidized and leached outcrops returned 19 samples (out of 40 collected) with gold values greater than 1.0 g/t.

Forty-three line km of IP Chargeability/Resistivity and 44 line km of ground magnetic surveying were completed. The most significant feature generated by the surveys was a coincident high chargeability and high magnetic anomaly associated with the Central Zone.

## 2007

Geological mapping of the Central Zone, to the northwest and to the southeast which includes the Bakirlik Hill area, was completed by Teck Madencilik at a scale of 1/10,000. Extensive soil, rock chip, and silt orientation sampling programs were also carried out. A total of 3,650 soil, 172 rock, and 58 silt orientation samples were collected from the Halilağa area. The soil results showed that strong surface geochemical anomalies were not only restricted to the Kestane area, but also occurred to the southeast at Bakirlik and in the central and southern parts of the property.

Sixty-four line km of IP Chargeability/Resisitivity and 263 line km of ground magnetics were completed. Subsequent 3-D magnetic modelling of the data by Truva Bakir (see Figure 8.1) showed a large magnetic body at depth that comes to surface.

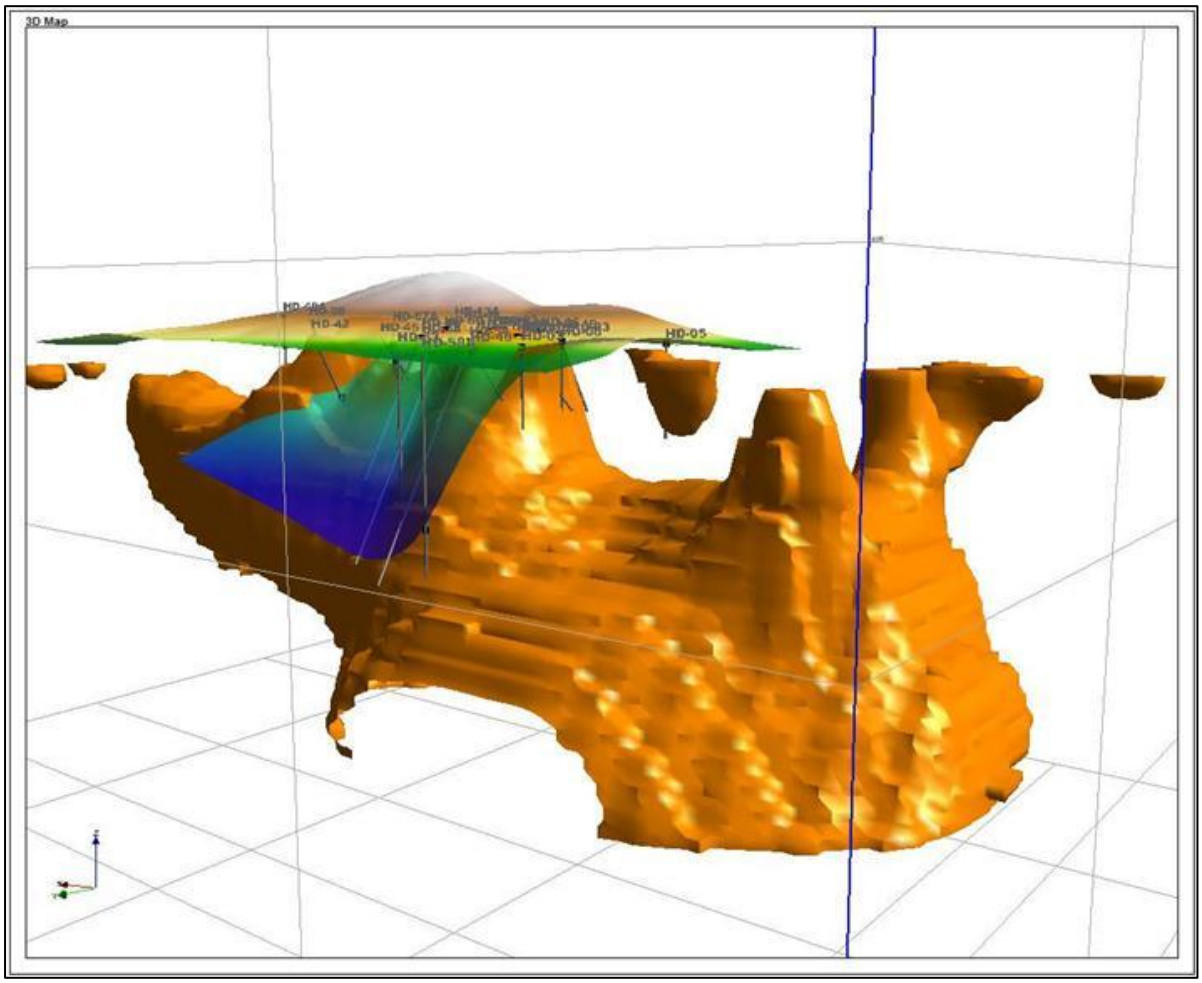



**Figure 8.1: 3-D Magnetic Inversion of Halilağa Data**

This anomaly is the coincident high chargeability and high magnetic anomalies associated with the Central Zone porphyry copper-gold mineralization. High resistivity maps the silica rocks along the top of ridge line

A total of 1,164 soil and 409 rock samples were also collected from the Pirentepe area. A total of 16.3 line km of IP Chargeability/Resisitivity and 58.8 line km of ground magnetics were also completed. The most significant anomaly generated was a coincident high chargeability and low magnetic anomaly associated with the NE-trending structures and epithermal gold mineralization. High magnetic anomalies map the basaltic volcanics SE of Pirentepe, as well as the Kocakısla area, suggesting potential for porphyry mineralization.

## 2008

Matias Sanchez (Sanchez 2008) regionally mapped the Halilağa Project as part of his Ph.D. thesis (see Figure 8.2). The map includes lithologies, alteration, and detailed structural information.

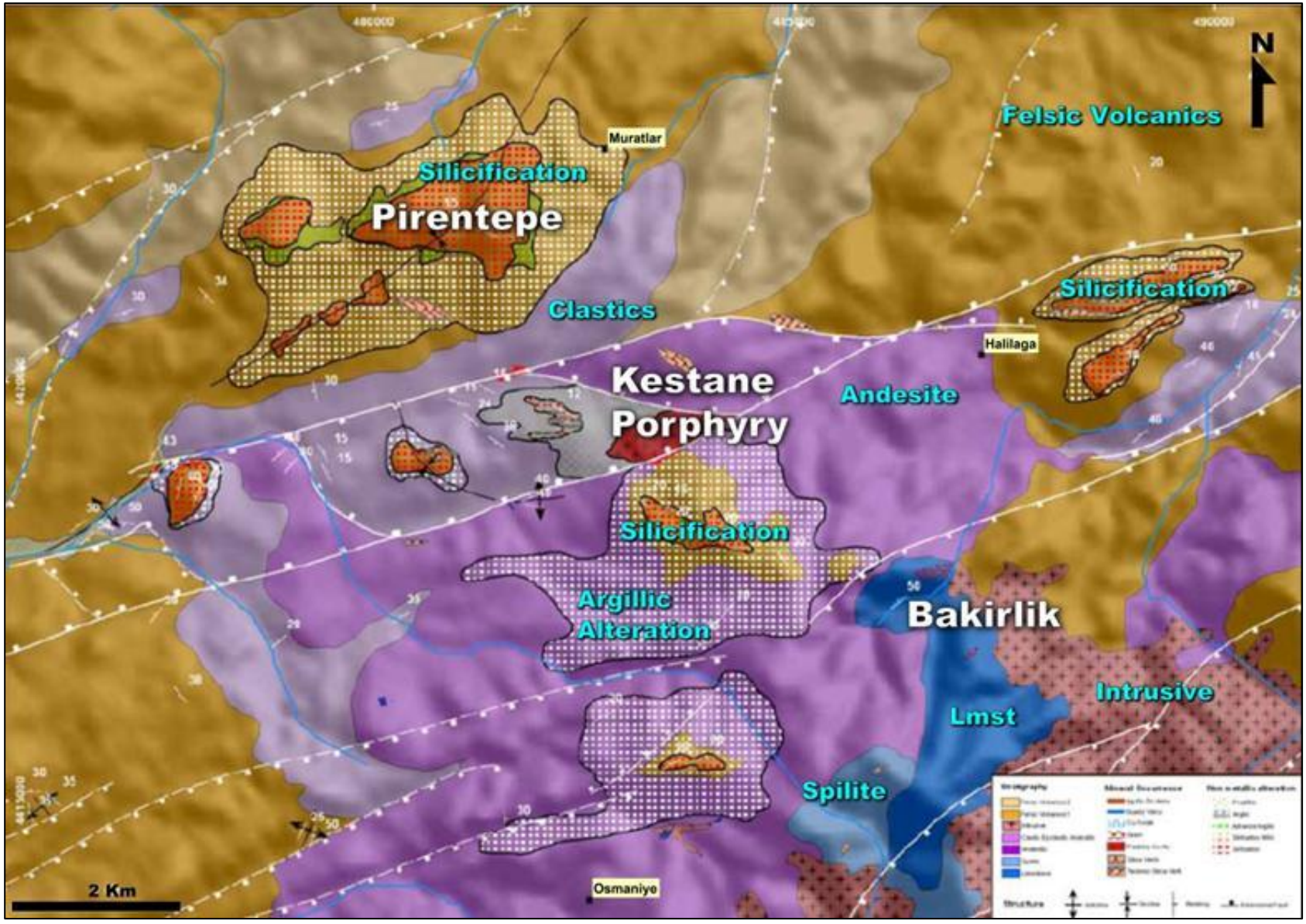



**Figure 8.2: Geological Map of Halilağa Property (Sanchez, 2008)**

A total of 566 rock samples were collected from selected areas (see Figure 8.3). The 2008 rock geochem highlighted three new targets:

1. **Kunk North** (> 0.5 g/t Au) is part of the Kunk Hill lithocap with strongly silicified, locally vuggy and brecciated volcanics.
2. **Kumlugedik Hill** (> 1.0 g/t Au) has widespread float of silicified and quartz-veined (more epithermal-low temperature) meta-sediments, and volcano sediments.
3. **Madendere** (> 0.2 g/t Au) has more porphyry style alteration characterized by strong quartz sericitic volcanic-sub volcanics with a few millimetre-thick quartz veins (possibly B-type). The size of the selectively sampled area extends more than 2 km long by 200 m wide.

In addition to these areas, additional copper-gold rock geochemistry at Kizilciktasi, Osmaniye, and Yaniklar were identified (Ceyhan et al 2009).

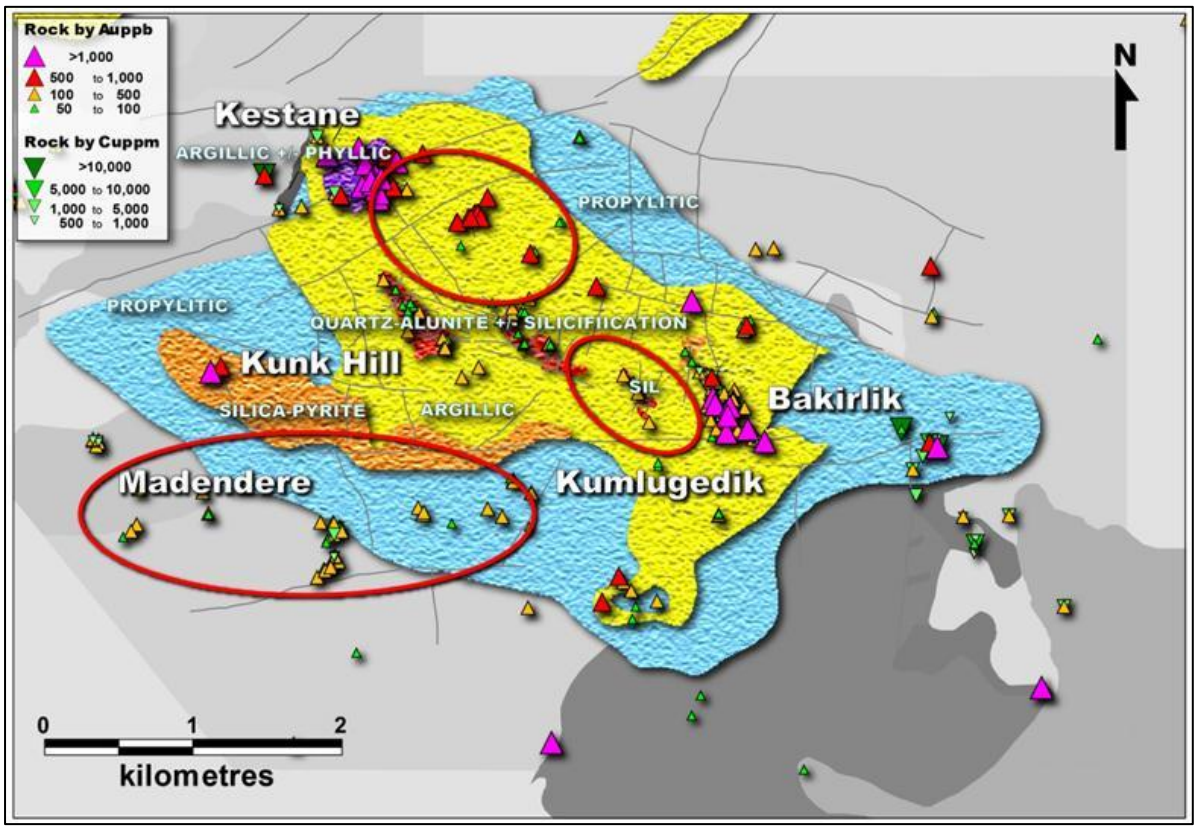



**Figure 8.3: Rock Geochemistry after 2008 Sampling (Grieve, 2009)**

A total of 163 rock-saw samples were collected and returned several gold anomalies up to 7.5 g/t. The 2008 rock sampling program confirmed a greater than 3 km long NE-trending structural corridor including a possible feeder zone. The program also highlighted the Kocakisla area with > 0.1 g/t Au assay results. The results were also conformable with soil anomalies that detected surface geochemical anomalies (> 10 ppb Au) along a NE-trending structure as three separate zones, as well as in the Kocakısla area.

## 2009 and 2010

A total of 36 rock chip samples were collected during the 2009 field season. Significant highlights of this program include:

- Rock chip sampling results from Kizilciktasi (0.1–0.5 g/t Au); and
- Confirmation of anomalous gold in rock-saw samples from north of Kunk Hill (> 0.5 g/t Au).

IP Chargeability/Resisitivity surveys were carried out in 2009 and 2010. The 2009 IP survey focused on deep targets in the area of Kestane Porphyry by using six 200 m dipoles in the pole-dipole array. This data was acquired on lines 19300N (EW), 82000E, 83300E, 83800E, 84300E, and 84800E (all N-S). In May 2010, a large IP program was carried out in the southern portion of the Halilağa licenses in order to extend the original IP coverage from previous years. The data was acquired with a conventional n=6, a=100 m pole-dipole setup and this data was subsequently merged into the older data sets to create complete chargeability and resistivity plan maps for the Halilağa area (see Figure 8.4 and Figure 8.5). The entire program totaled 54 line km of data.

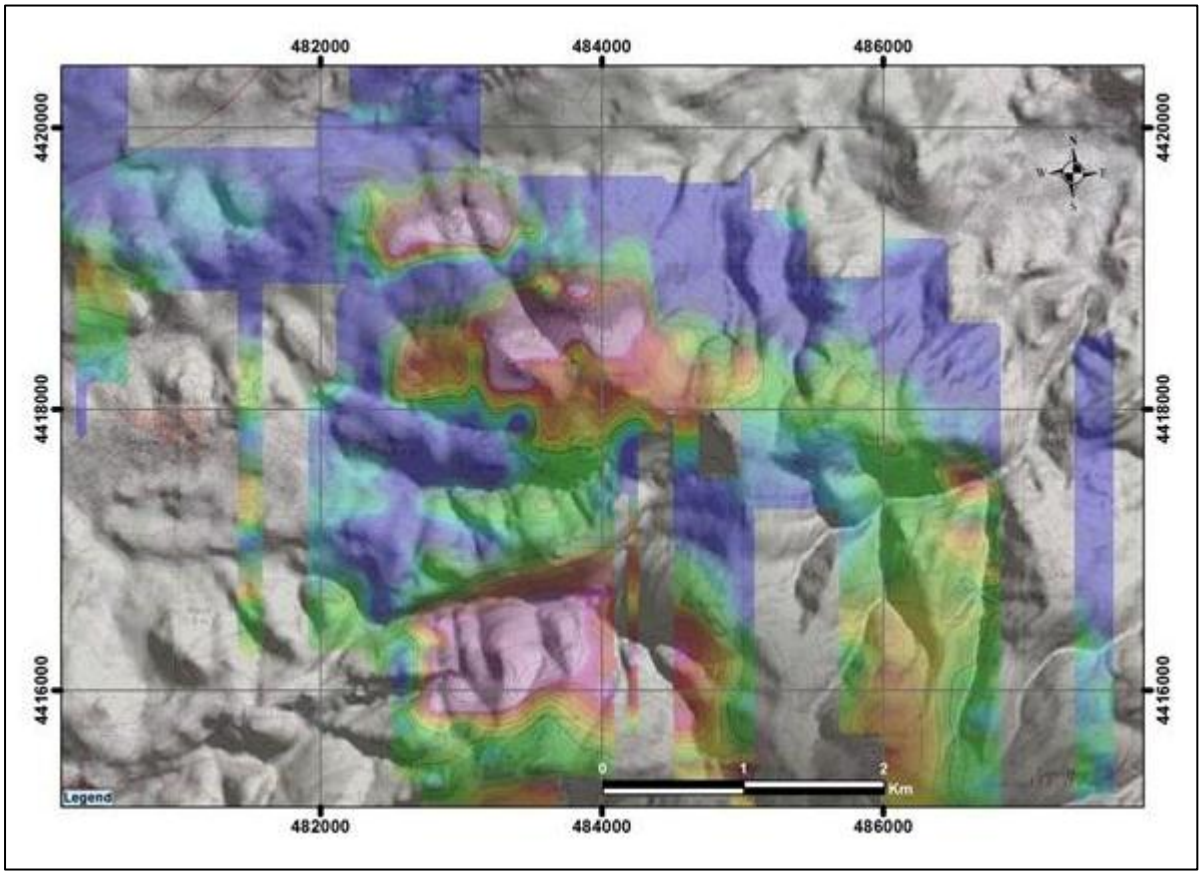



**Figure 8.4: Compiled IP Chargeability Plan for 100 m Depth**

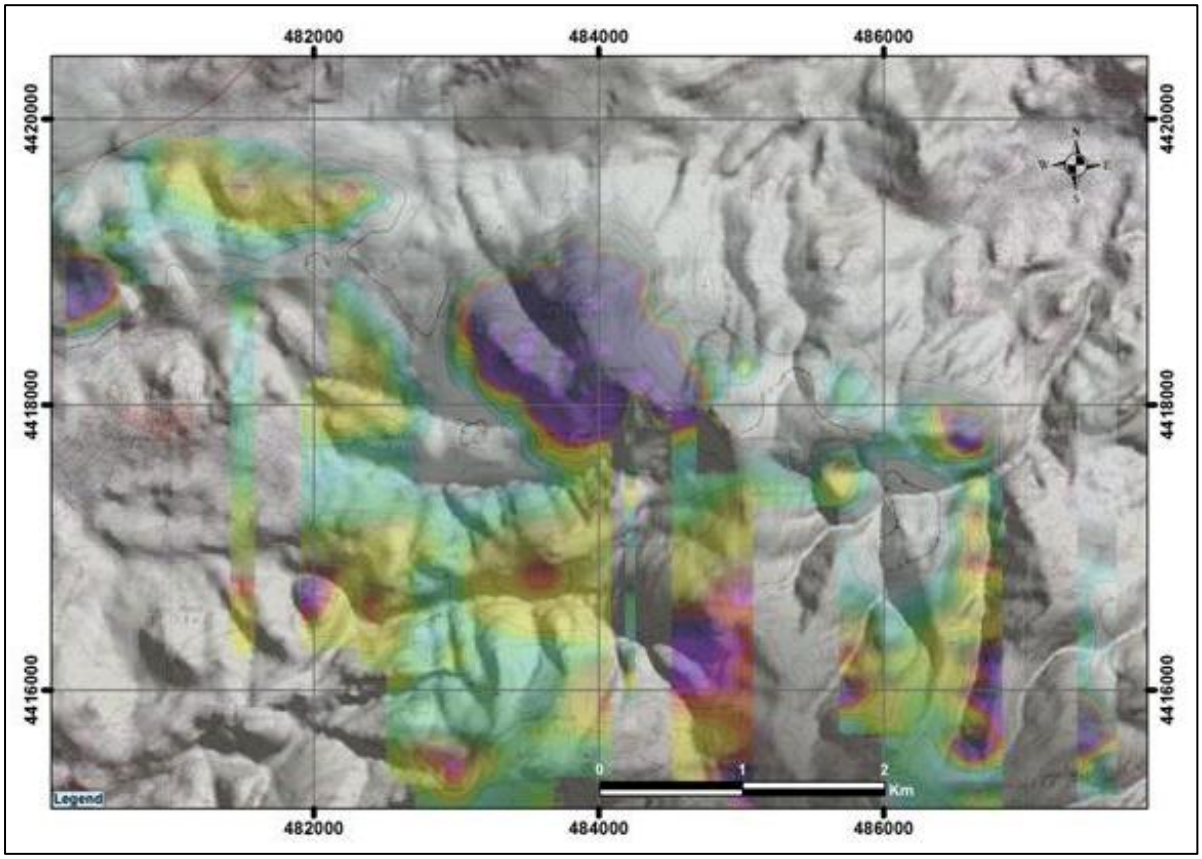



**Figure 8.5: Compiled IP Resistivity Plan for 100 m Depth.**

On-going metallurgical sampling tested the hornfels unit of holes HD-40 and HD-49. Acid Rock Drainage ("ARD") tests were performed on the pulps from holes HD-05, HD-09, and HD-48A. Further petrographic samples were taken from holes HD-40, HD-48A, HD-49, and HD-54. An environmental base-line study was also initiated over the Halilağa Project.

Table 8.1 shows the program summerized, by number and line-km, for geochemical sampling and by year for geophysics.

**Table 8.1: Details of the Halilağa Project by Year**

| YEAR | 2005 | 2006 | 2007 | 2008 | 2009 | 2010 |
|---|---|---|---|---|---|---|
| Geochemical Sampling ( #) | 383 | 1197 | 3880 | 729 | 36 | 0 |
| IP/Resistivity (line-km) | 8 | 35 | 64 | 0 | 21 | 33 |
| Ground Magnetics (line-km) | 0 | 44 | 263 | 0 | 0 | 0 |

# 9 Drilling

The drilling between 2007 and 2011 was performed by Spectra Jeotek of Ankara, Turkey, and was conducted using two to five, contractor-manufactured drill rigs. The model numbers are D150 and D220 with depth capacities of 1,000 m and 1,500 m of HQ, respectively. In 2011, the drilling was done with tri-cone bit (Q=120 mm) until the contact with the QFP/hornfels was made and then continued with HQ core after casing the hole. Between 2007 and 2010 the drilling was mostly HQ drilling which was then reduced to NQ when ground conditions became difficult. The HRC series drill holes are reverse circulation ("RC") type holes.

The drill hole collars for holes HD-01 through HD-35 were surveyed using Total Station methods. The subsequent holes HD-36 and above were surveyed using a differential global positioning system ("DGPS") with a horizontal and vertical accuracy of generally ± 20 cm. Drill hole deviation was measured using Reflex Survey tests taken between 50 m and 100 m intervals down hole to provide control.

## 9.1 Halilağa Drilling

Subsequent to property acquisition, all drilling was performed by Fronteer or Teck Madencilik or Truva Bakir and is listed in Table 9.1.

**Table 9.1: Halilağa Drill Summary**

| Target | Year | # of drilled holes | Total (m) | # of Abandoned holes | Total Abandoned (m) | Gross # of holes | Gross Total (m) |
|---|---|---|---|---|---|---|---|
| Kestane-Kunk | 2006-2007 | 18 | 5,827 | 5 | 519 | 23 | 6,346 |
| Kestane+Kunk | 2009 | 14 | 5,243 | 4 | 427 | 18 | 5,670 |
| Kestane | 2010 | 11 | 6,098 | 14 | 2,978 | 25 | 9,076 |
| Kestane | 2011 | 39 | 17,973 | 4 | 1,159 | 43 | 19,132 |
| **Kestane Total** | **2006-2011** | **82** | **35,141** | **27** | **5,083** | **109** | **40,224** |
| Pirentepe | 2006-2007 | 28 | 4,970 | 1 | 37 | 29 | 5,007 |
| Bakirlik | 2008 | 18 | 4,019 | 2 | 32 | 20 | 4,051 |
| Madendere | 2011 | 1 | 467 | | | 1 | 467 |
| **Grand Total** | **2006-2011** | **129** | **44,597** | **30** | **5,152** | **159** | **49,749** |

In the 2006-2007 period, a total of 20 diamond holes totalling 5,950 m (including five abandoned holes) and three reverse circulation holes totalling 396 m were completed. The drilling focused in two main zones: the Kestane porphyry and the Kunk-Kumlugedik lithocap areas. 17 holes (15 diamond and 2 RC) totalling 4,756.1 m were drilled at Kestane. Most of the 2007 drill holes at the Kestane Zone intersected porphyry-style Cu-Au mineralization with economic grades (for example, HD-01 returned 1.03 g/t Au and 1.03% Cu over 105.4 m). A 25 metre-thick chalcocite blanket (approximately 2% Cu) was intersected close to the surface in holes HD-01, HD-02, HD-04, and HD-14. A total of six holes (five diamond and one RC) totalling 1,589.9 m were drilled at the Kunk-Kumlugedik Zone. The holes intersected either narrow low-grade gold mineralization (HD-09 and HD-10) or no significant mineralization (HD-11, HD-12, HD-12A, and HRC-01).

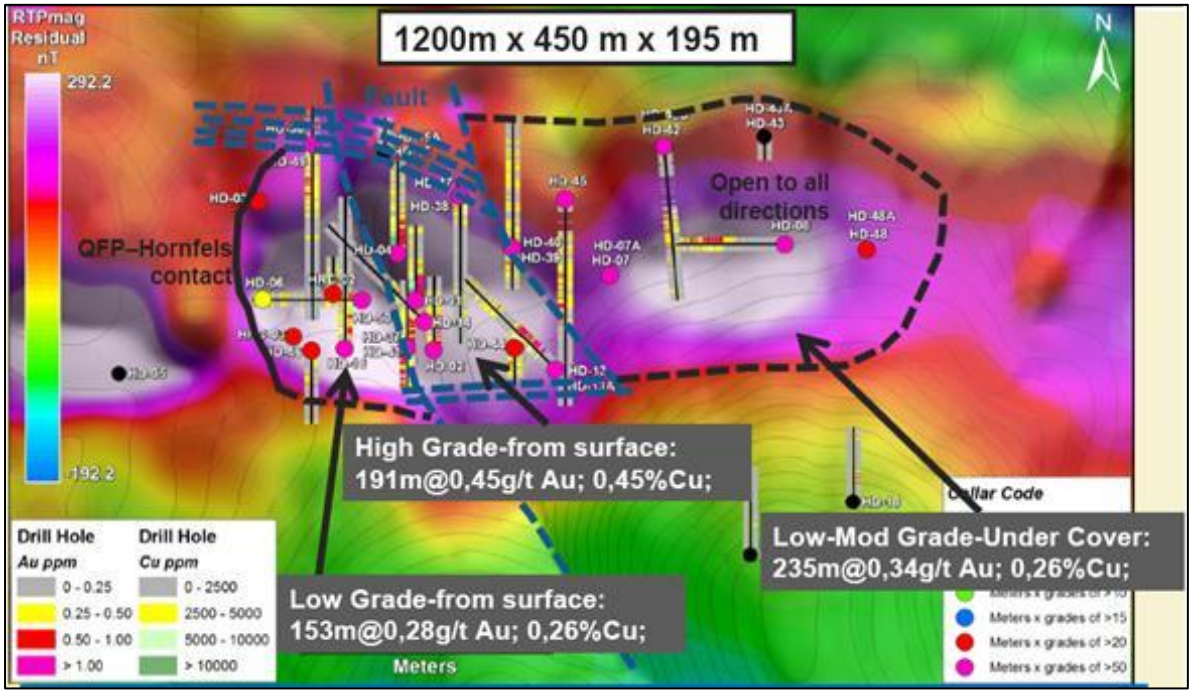
In 2008, a total of 20 diamond holes (including one abandoned hole) totalling 4,051 m were completed. The main objective of the program was to test the coincidence of magnetic high

anomalies and anomalous Cu and gold values in the soil at the Bakirlik area which is located 4 km SE of Kestane porphyry Cu-Au mineralization.

A total of nine samples from the holes drilled at Kestane and Bakirlik zones were petrographically and mineralogically analyzed (Kuşcu 2008).

In 2009, a total of 18 diamond holes (including four abandoned holes) totalling 5,670 m, including deviated hole HD-42D, were completed. All holes were located at Kestane (central) Zone and tested porphyry Cu-Au mineralization. Drilling focused on further defining the limits of the Kestane target which steps out to the north, south, and east. The 2009 drilling footprint increased the extent of the known Kestane mineralization to 1,200 x 450 x 195 m. Significant intersections were encountered in HD-41: 159.5 m @ 0.59 g/t Au and 0.50% Cu; HD-45: 308.1 m @ 0.34 g/t Au and 0.30% Cu; and HD-38: 103.0 m @ 0.40 g/t Au and 0.70% Cu; all other significant drill intervals are shown in Table 9.2. A series of consistently eastward, down dropping fault bound blocks are shown in Figure 9.1 and

Figure 9.2.




Note: MAG in the background.

**Figure 9.1: Kestane Target with Drill Collars and Traces**

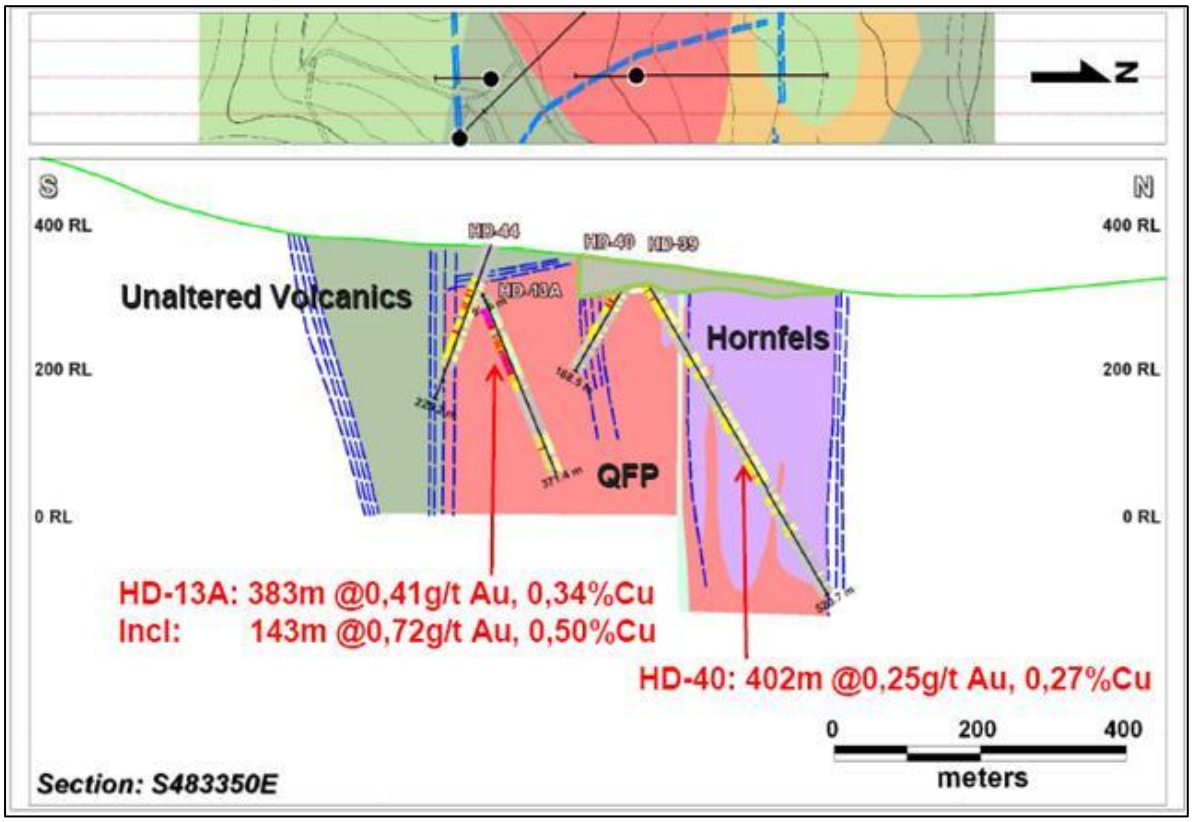



**Figure 9.2: Kestane Target Showing the Down Dropping Fault Bound Blocks of QFP Section 483350**

## 9.2 Drilling

In 2010, 25 holes (20 diamond and 5 RC) totalling 9,076.6 m (including 14 abandoned holes) were completed. With the exception of holes HD-64 and HD-69, which tested the Kestane-West target, all holes were located at Kestane (central) Zone. Drill hole HD-54 returned an intercept of 646.5 m @ 0.26 g/t Au and 0.33% Cu, representing the longest interval of copper-gold mineralization to date at Halilağa. The hole collared in a high grade, near surface blanket of chalcocite, then intersected moderate to high-grade copper zones and then passed into low to moderate-grade copper and gold mineralization for the remainder of its length (see Table 9.2). The drill hole tested an area where the QFP intrusive was 20.9 m below surface and was interpreted to represent an up-thrust block related to the general regional, ENE-WSW transtensional structures of the North Anatolian Fault System. HD-54 successfully tested the northern margin of the intrusive and passed into a down-dropped block of QFP intrusive at depth. The southern margin of this same up-thrusted block was tested with HD-57 which returned 348.80 m @ 0.28 g/t Au and 0.31% Cu (see Figure 9.3). Holes HD-56, HD-60, and HD-61 drilled into the hornfels aureole on the north and west side of the Kestane Zone and also intersected anomalous molybdenum over long widths.

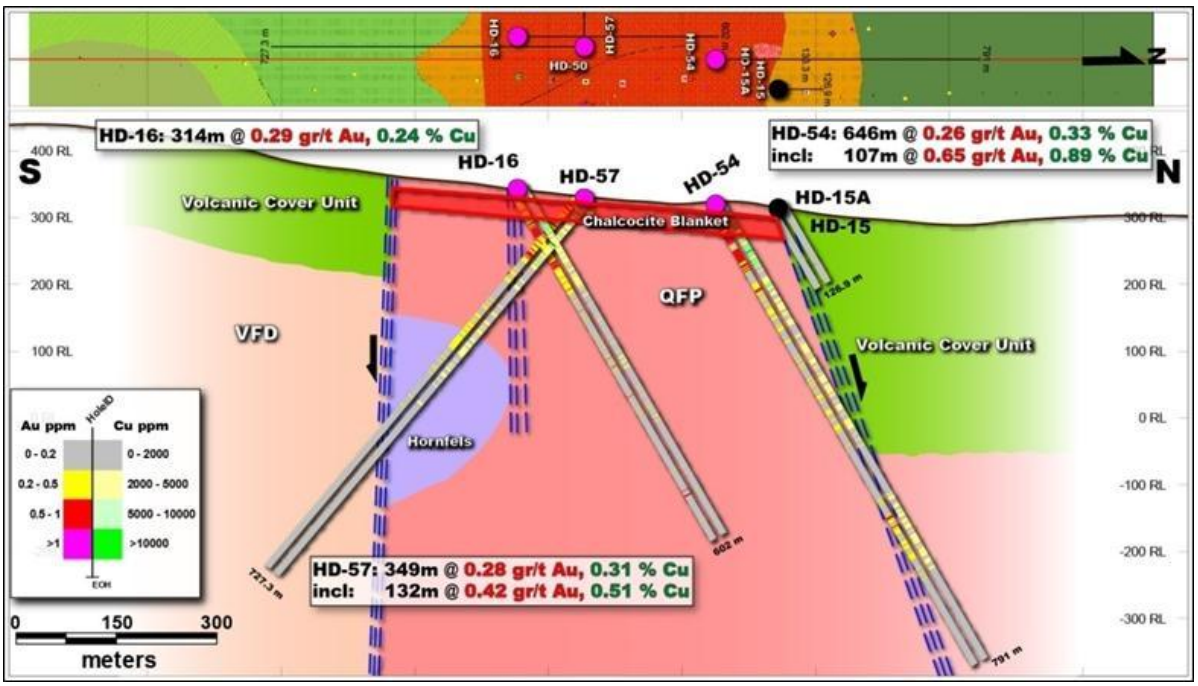



**Figure 9.3: Kestane Horst at the Center Bounded by North and South Normal Faults**

The 2011 program included 44 drill holes (including four abandoned holes) for a total 19,599 m of drilling. One of those drill holes was directed toward testing the Madendere area which extended 467 m. The focus of the 2011 drilling was to extend and in-fill the Kestane Zone to demonstrate continuity and increase confidence in order to calculate a resource estimate for the deposit. Significant intersections included HD-76 with 448.7 m @ 0.35 g/t Au and 0.22% Cu, and HD-80 with 323.3 m @ 0.27 g/t Au and 0.32% Cu. HD-86 returned an intercept of 185.6 m @ 0.46 g/t Au and 0.61% Cu. However, the hole collared in the high grade, near surface blanket of chalcocite, as previously identified, included 71.3 m @ 0.71 g/t Au and 1.1% Cu. The 2011 drilling has now shown the Kestane porphyry to be mineralized over a strike length of 1,200 metres, a width of 850 m, and a thickness of up to 600 m.

It should be noted that assays for up to HD-116 with the exception of 10 assays from HD-113A that are still pending. Assays are pending for holes HD-117, HD-118 and HD119A as of the effective date of this report. Holes and assays up to HD-98 were utized in the calculation of this resource estimate. Drillholes HD-120, HD-121 and HD-122 are in progress.

Table 9.2 lists all of the Halilağa drill holes and Table 9.3 summarizes the assay results.

**Table 9.2: Halilağa Drill Hole Collars**

| Hole ID | East (UTM m) | North (UTM m) | Elevation (masl) | Depth (m) | Azimuth (degrees) | Dip (degrees) |
|---|---|---|---|---|---|---|
| HD-02 | 483192.54 | 4419151 | 349.0 | 292.8 | 0 | -60 |
| HD-03 | 482843.91 | 4419446 | 308.0 | 242.9 | 0 | -90 |
| HD-04 | 483120.28 | 4419342 | 337.9 | 320.4 | 0 | -55 |
| HD-05 | 482568.48 | 4419104 | 313.5 | 285.2 | 0 | -90 |
| HD-06 | 482852.82 | 4419252 | 318.0 | 279.1 | 0 | -90 |
| HD-07 | 483539.88 | 4419296 | 360.1 | 23.3 | 0 | -90 |
| HD-07A | 483539.92 | 4419299 | 360.0 | 330.7 | 0 | -90 |
| HD-08 | 483886.85 | 4419361 | 376.3 | 434.5 | 270 | -60 |
| HD-09 | 483817.64 | 4418745 | 530.1 | 355.4 | 0 | -60 |
| HD-10 | 484022.06 | 4418850 | 497.9 | 297.5 | 0 | -60 |
| HD-100 | 483440.87 | 4419450 | 326.8 | 564.6 | 90 | -60 |
| HD-101 | 482804.15 | 4416723 | 383.5 | 467.3 | 0 | -90 |
| HD-102 | 482947.49 | 4419151 | 337.8 | 325 | 0 | -75 |
| HD-103 | 483741.34 | 4419553 | 347.2 | 505.5 | 180 | -75 |
| HD-104 | 484043.37 | 4419167 | 415.1 | 275 | 315 | -60 |
| HD-105 | 482873.74 | 4416036 | 403.6 | 43.6 | 90 | -60 |
| HD-105A | 482872.83 | 4416036 | 403.8 | 163.5 | 90 | -60 |
| HD-106 | 483350.5 | 4419048 | 370.5 | 205 | 0 | -90 |
| HD-107 | 483650.11 | 4419446 | 353.3 | 435.9 | 180 | -55 |
| HD-108 | 482853 | 4419253 | 318.1 | 200 | 0 | -60 |
| HD-108A | 482852 | 4419251 | 318.2 | 45.9 | 0 | -60 |
| HD-109 | 483046 | 4419363 | 318.2 | 401.2 | 345 | -65 |
| HD-109A | 483045 | 4419364 | 318.2 | 30.8 | 345 | -65 |
| HD-11 | 484201.34 | 4418248 | 556.1 | 296 | 0 | -60 |
| HD-110 | 483349 | 4419051 | 370.6 | 300 | 0 | -65 |

| Hole ID | East (UTM m) | North (UTM m) | Elevation (masl) | Depth (m) | Azimuth (degrees) | Dip (degrees) |
|---|---|---|---|---|---|---|
| HD-111 | 483537 | 4419295 | 359.6 | 378.5 | 180 | -50 |
| HD-112 | 483740 | 4419552 | 347.3 | 497.4 | 180 | -63 |
| HD-113 | 483544 | 4419302 | 359.7 | 383.2 | 140 | -50 |
| HD-113A | 483547 | 4419305 | 359.6 | 491.1 | 140 | -50 |
| HD-114 | 484042 | 4419177 | 421.2 | 401.5 | 0 | -90 |
| HD-115 | 483444 | 4419251 | 356.7 | 413.4 | 180 | -50 |
| HD-116 | 484042 | 4419166 | 413.4 | 407 | 250 | -60 |
| HD-117 | 483925 | 4419359 | 377.5 | 497 | 180 | -50 |
| HD-118 | 483437 | 4419355 | 342.3 | 410.5 | 0 | -50 |
| HD-119 | 483541 | 4419457 | 335.4 | | | |
| HD-119A | 483541 | 4419456 | 335.4 | 472.6 | 0 | -50 |
| HD-12 | 483405.77 | 4418310 | 531.9 | 138 | 0 | -60 |
| HD-120 | 483925 | 4419360 | 377.4 | 500 | 180 | -75 |
| HD-121 | 483450 | 4419450 | 325.0 | 450 | 0 | -90 |
| HD-122 | 483250 | 4419050 | 360.0 | 450 | 180 | -80 |
| HD-12A | 483405.83 | 4418312 | 531.9 | 353 | 0 | -60 |
| HD-13 | 483431.97 | 4419112 | 376.3 | 97.6 | 315 | -60 |
| HD-13A | 483431.3 | 4419113 | 376.3 | 505.2 | 315 | -60 |
| HD-14 | 483173.69 | 4419207 | 348.8 | 539.7 | 315 | -60 |
| HD-15 | 483094.81 | 4419544 | 314.1 | 126.9 | 0 | -60 |
| HD-15A | 483094.87 | 4419544 | 314.2 | 133.3 | 0 | -60 |
| HD-16 | 483015.93 | 4419154 | 343.5 | 602 | 0 | -60 |
| HD-17 | 485636.11 | 4418257 | 440.6 | 285.5 | 330 | -60 |
| HD-18 | 485672.97 | 4418687 | 394.7 | 212 | 0 | -50 |
| HD-19 | 487400.53 | 4416747 | 492.4 | 189.6 | 325 | -50 |
| HD-20 | 486608.09 | 4417847 | 444.1 | 26 | 135 | -50 |

| Hole ID | East (UTM m) | North (UTM m) | Elevation (masl) | Depth (m) | Azimuth (degrees) | Dip (degrees) |
|---|---|---|---|---|---|---|
| HD-20A | 486608.89 | 4417846 | 444.1 | 311.4 | 135 | -50 |
| HD-21 | 486429.68 | 4417817 | 486.4 | 418.4 | 100 | -50 |
| HD-22 | 485486.2 | 4417834 | 536.2 | 230.6 | 300 | -60 |
| HD-23 | 485470.78 | 4417745 | 527.1 | 247.2 | 115 | -50 |
| HD-24 | 486603.5 | 4417709 | 428.9 | 201.4 | 90 | -50 |
| HD-25 | 486793.99 | 4417441 | 361.4 | 265.1 | 120 | -50 |
| HD-26 | 484938.87 | 4416621 | 561.1 | 291.5 | 245 | -50 |
| HD-27 | 487208.92 | 4416746 | 546.5 | 6 | 335 | -50 |
| HD-28 | 486788.57 | 4417441 | 361.4 | 157.7 | 300 | -50 |
| HD-29 | 486781.04 | 4417735 | 401.9 | 438.3 | 180 | -60 |
| HD-30 | 486604.19 | 4417599 | 392.2 | 56 | 170 | -60 |
| HD-31 | 486134.85 | 4417905 | 476.1 | 196.6 | 360 | -60 |
| HD-32 | 486805.39 | 4417270 | 384.3 | 256 | 135 | -60 |
| HD-33 | 486581.39 | 4417909 | 442.7 | 71.5 | 0 | -60 |
| HD-34 | 486739.11 | 4416913 | 400.1 | 83.4 | 270 | -70 |
| HD-35 | 486846 | 4417609 | 343.0 | 106.8 | 225 | -70 |
| HD-36 | 482951.8 | 4419558 | 306.2 | 149 | 0 | -60 |
| HD-37 | 483139.2 | 4419154 | 336.5 | 268.5 | 0 | -60 |
| HD-38 | 483244.55 | 4419456 | 326.9 | 586.1 | 180 | -60 |
| HD-39 | 483348.49 | 4419354 | 341.4 | 168.5 | 180 | -60 |
| HD-40 | 483347.28 | 4419354 | 341.4 | 520.7 | 0 | -60 |
| HD-41 | 483138.7 | 4419156 | 336.4 | 180.6 | 180 | -60 |
| HD-42 | 483646.47 | 4419555 | 337.6 | 393.5 | 175 | -60 |
| HD-42D | 483646.47 | 4419555 | 337.6 | 614 | 175 | -60 |
| HD-43 | 483845.67 | 4419575 | 345.0 | 71.9 | 180 | -60 |
| HD-43A | 483845.62 | 4419574 | 345.1 | 95.8 | 180 | -60 |

| Hole ID | East (UTM m) | North (UTM m) | Elevation (masl) | Depth (m) | Azimuth (degrees) | Dip (degrees) |
|---|---|---|---|---|---|---|
| HD-44 | 483351.77 | 4419155 | 364.6 | 225.2 | 180 | -70 |
| HD-45 | 483438.01 | 4419448 | 327.3 | 638.4 | 180 | -50 |
| HD-46 | 482946.18 | 4419154 | 337.6 | 292.5 | 180 | -60 |
| HD-47 | 483243.44 | 4419456 | 327.1 | 53 | 345 | -50 |
| HD-48 | 484040.27 | 4419348 | 392.0 | 212 | 0 | -90 |
| HD-48A | 484039.92 | 4419350 | 392.0 | 603.9 | 0 | -90 |
| HD-49 | 482950.78 | 4419551 | 306.5 | 541.4 | 180 | -50 |
| HD-50 | 483031.15 | 4419254 | 329.0 | 308 | 270 | -45 |
| HD-51 | 483841.28 | 4419577 | 345.0 | 76 | 180 | -60 |
| HD-52 | 483148.93 | 4419638 | 307.0 | 287 | 0 | -60 |
| HD-53 | 483742.69 | 4419554 | 347.0 | 108 | 160 | -60 |
| HD-54 | 483060.6 | 4419454 | 325.6 | 791 | 0 | -60 |
| HD-55 | 483249.69 | 4419044 | 352.0 | 98.2 | 200 | -60 |
| HD-56 | 482560.78 | 4419153 | 316.4 | 611.1 | 315 | -75 |
| HD-57 | 483036.78 | 4419252 | 329.1 | 727.3 | 180 | -50 |
| HD-58 | 482947.9 | 4419748 | 296.9 | 314 | 0 | -90 |
| HD-58A | 482944.5 | 4419746 | 297.1 | 431.8 | 0 | -90 |
| HD-60 | 483250.9 | 4419349 | 344.5 | 798.3 | 0 | -60 |
| HD-61 | 482854.9 | 4419353 | 310.8 | 502.2 | 0 | -60 |
| HD-62 | 482854.1 | 4419255 | 317.7 | 625.1 | 180 | -50 |
| HD-63B | 483149.9 | 4419635 | 306.9 | 350 | 0 | -90 |
| HD-64 | 482098 | 4419434 | 340.8 | 683 | 225 | -70 |
| HD-65 | 483548.5 | 4419554 | 325.0 | 265 | 0 | -90 |
| HD-66 | 483248.6 | 4419047 | 356.7 | 161.4 | 0 | -90 |
| HD-66A | 483249.5 | 4419048 | 356.6 | 183 | 0 | -90 |
| HD-67 | 483548.7 | 4419552 | 324.8 | 194.5 | 180 | -60 |

| Hole ID | East (UTM m) | North (UTM m) | Elevation (masl) | Depth (m) | Azimuth (degrees) | Dip (degrees) |
|---|---|---|---|---|---|---|
| HD-67A | 483548.7 | 4419549 | 325.2 | 667.2 | 180 | -60 |
| HD-69 | 482206.3 | 4418781 | 283.3 | 550 | 270 | -50 |
| HD-70 | 481944.7 | 4418506 | 285.2 | 595.8 | 170 | -70 |
| HD-71 | 483246.6 | 4419043 | 356.7 | 71 | 0 | -60 |
| HD-72 | 484251.4 | 4419347 | 379.0 | 710.6 | 0 | -90 |
| HD-73 | 484360.6 | 4417957 | 748.0 | 632.1 | 0 | -60 |
| HD-74 | 482742.52 | 4419539 | 299.3 | 515.6 | 180 | -60 |
| HD-75 | 482848.13 | 4419047 | 331.8 | 46.1 | 180 | -60 |
| HD-75A | 482848.13 | 4419046 | 331.8 | 151.4 | 180 | -60 |
| HD-75B | 482849.72 | 4419046 | 332.0 | 300.7 | 180 | -60 |
| HD-75C | 482849.91 | 4419040 | 331.9 | 153.6 | 180 | -60 |
| HD-76 | 483647.91 | 4419445 | 353.1 | 869 | 135 | -60 |
| HD-77 | 484040.69 | 4419166 | 415.8 | 853.8 | 0 | -65 |
| HD-78 | 482750.25 | 4419348 | 310.8 | 510 | 180 | -60 |
| HD-79 | 482744.84 | 4419162 | 320.8 | 500 | 180 | -60 |
| HD-80 | 483140.42 | 4419453 | 331.6 | 857 | 0 | -65 |
| HD-81 | 482944.05 | 4418969 | 344.8 | 512.5 | 180 | -60 |
| HD-82 | 483841.71 | 4419571 | 344.5 | 437 | 180 | -60 |
| HD-82D | 483841.71 | 4419571 | 344.5 | 635.8 | 180 | -60 |
| HD-83 | 484837.09 | 4419155 | 317.7 | 247 | 0 | -90 |
| HD-83A | 484837.59 | 4419157 | 317.2 | 167.8 | 0 | -90 |
| HD-84 | 482447.02 | 4419457 | 333.5 | 709.9 | 90 | -60 |
| HD-85 | 484540.08 | 4419262 | 358.4 | 473.7 | 0 | -90 |
| HD-86 | 483047.76 | 4419358 | 317.2 | 504.9 | 0 | -90 |
| HD-87 | 484049.38 | 4419561 | 353.1 | 884.5 | 180 | -60 |
| HD-88 | 484737.73 | 4419057 | 337.9 | 427 | 0 | -90 |

| Hole ID | East (UTM m) | North (UTM m) | Elevation (masl) | Depth (m) | Azimuth (degrees) | Dip (degrees) |
|---|---|---|---|---|---|---|
| HD-89 | 482444.97 | 4419555 | 328.8 | 326.6 | 90 | -50 |
| HD-90 | 482950.15 | 4419360 | 314.7 | 57.1 | 0 | -90 |
| HD-90A | 482950.31 | 4419363 | 314.6 | 462.6 | 0 | -90 |
| HD-91 | 483345.29 | 4419252 | 353.0 | 449.4 | 0 | -90 |
| HD-92 | 483346.61 | 4419555 | 318.1 | 60 | 180 | -60 |
| HD-92A | 483346.51 | 4419552 | 318.1 | 592.4 | 180 | -60 |
| HD-93 | 484751.09 | 4419254 | 340.9 | 436 | 0 | -90 |
| HD-94 | 483246.58 | 4419052 | 355.5 | 453 | 0 | -70 |
| HD-94A | 483246.29 | 4419051 | 356.2 | 250 | 0 | -70 |
| HD-95 | 484440.08 | 4419157 | 380.3 | 518 | 0 | -75 |
| HD-96 | 483022.43 | 4419765 | 292.7 | 351.6 | 0 | -90 |
| HD-96A | 483023.63 | 4419762 | 292.7 | 489.5 | 0 | -90 |
| HD-97 | 483041.48 | 4419050 | 349.4 | 400.2 | 0 | -80 |
| HD-98 | 483541.56 | 4419297 | 359.8 | 254 | 90 | -60 |
| HD-98A | 483540.29 | 4419296 | 359.5 | 503 | 90 | -60 |
| HD-99 | 484036.09 | 4419353 | 383.0 | 368 | 270 | -60 |
| HD-99A | 484032.95 | 4419353 | 383.0 | 503.6 | 270 | -60 |
| HRC-01 | 485580.97 | 4417853 | 549.8 | 150 | 265 | -60 |
| HRC-02 | 482993.39 | 4419263 | 328.1 | 145.5 | 0 | -60 |
| HRC-03 | 482914.05 | 4419177 | 329.7 | 100.5 | 0 | -90 |
| HRC-59 | 483345.3 | 4419665 | 304.0 | 192 | 0 | -90 |
| HRC-63 | 483153.6 | 4419640 | 307.2 | 54 | 0 | -90 |
| HRC-63A | 483146.7 | 4419625 | 307.6 | 192 | 0 | -90 |
| HRC-68 | 483137.8 | 4419153 | 336.1 | 143.5 | 190 | -60 |

**Table 9.3: Assay Summary**

| Hole No. | Zone | From | To | Interval | Au | Cu | Ag | Mo |
|---|---|---|---|---|---|---|---|---|
| | | (m) | (m) | (m) | (g/t) | (%) | (g/t) | (%) |
| HD-100 | Kestane | 229 | 437.8 | 208.8 | 0.28 | 0.29 | _ | 0.011 |
| HD-101 | Kestane | | | | | | | |
| HD-102 | Kestane | 0 | 192.4 | 192.4 | 0.22 | 0.22 | _ | 0.004 |
| HD-103 | Kestane | 360 | 424.5 | 64.5 | 0.32 | 0.16 | _ | 0.002 |
| HD-107 | Kestane | 151 | 434 | 283 | 0.35 | 0.19 | _ | 0.003 |
| HD-108 | Kestane | 0 | 200 | 200 | 0.25 | 0.23 | _ | 0.008 |
| HD-108A | Kestane | 0 | 45.9 | 45.9 | 0.6 | 0.48 | | 0.002 |
| HD-109 | Kestane | 0 | 401.2 | 401.2 | 0.23 | 0.35 | _ | 0.005 |
| HD-110 | Kestane | 122.7 | 300 | 17.3 | 0.34 | 0.25 | _ | 0.003 |
| HD-111 | Kestane | 72 | 252.8 | 180.8 | 0.22 | 0.18 | _ | 0.005 |
| HD-112 | Kestane | 266.4 | 497.4 | 231 | 0.33 | 0.2 | _ | 0.001 |
| HD-113 | Kestane | 103 | 268 | 165 | 0.28 | 0.17 | _ | 0.003 |
| HD-113A | Kestane | 69.3 | 106 | 36.7 | 0.26 | 0.17 | _ | 0.003 |
| HD-114 | Kestane | 18.1 | 51.6 | 33.5 | 0.34 | 0.12 | _ | 0.001 |
| HD-115 | Kestane | 93 | 229.8 | 136.8 | 0.64 | 0.52 | _ | 0.003 |
| HD-116 | Kestane | 4.7 | 65.6 | 60.9 | 0.32 | 0.02 | _ | 0.001 |

## 9.3 Pirentepe Drilling

In 2006, a total of 890.1 m of drilling was completed at the Pirentepe area. Ten holes were drilled with the first two holes in the program intersecting gold mineralization (PD-1: 1.83 g/t Au over 46.90 m, starting at 17 m depth, and PD-2: 1.82 g/t Au over 38.0 m starting at 24 m). In 2007, thirteen diamond holes totalling 3,031.9 m and six reverse circulation holes totalling 1,084.7 m were completed.

The objectives of the drilling program included:

- Drill test the main NE-trending structural corridor where the highest gold in soil coincides with IP chargeability; and
- Drill test the Kocakısla area for any hidden porphyry Cu-Au and/or epithermal gold potential.

## 9.4 West Kestane Drilling

HD-56, HD-64, and HD-69 were drilled to the west of Central Zone at Kestane to test bulls-eye high magnetics anomalies and deep high chargeability targets.

HD-56 intersected disseminated pyrite in hornfels. HD-64 was located 1 km west of Kestane and intersected narrow high grade Zn (1.90 m @ 16.76% Zn) mineralization with anomalous gold and Cu (see Table 9.4). The skarn mineralization occurs at the contact between hornfels and recrystallized limestone. Another garnet skarn zone was intersected by historic hole HD-03 at the NW edge of Kestane. It was recommended to drill test the area between HD-03 and HD-64 for possible economic

skarn mineralization. HD-69 intersected chlorite-altered porphyritic volcanics and returned no significant mineralization.

**Table 9.4: HD -64 Skarn Zone Assays**

| Hole ID | Sample From (m) | Sample To (m) | Sample Interval (m) | Au (ppm) | Cu (%) | Zn (%) |
|---|---|---|---|---|---|---|
| HD-64 | 501.70 | 502.60 | 0.90 | 1.002 | 1.464 | 1.01 |
| HD-64 | 502.60 | 504.50 | 1.90 | 0.099 | 0.192 | 16.76 |
| HD-64 | 504.50 | 506.00 | 1.50 | 0.013 | 0.045 | 2.48 |

# 10 Sample Preparation, Analyses, and Security

## 10.1 Sample Preparation and Analyses

Collars were set up under the direct supervision of Truva Bakir staff and were drilled with HQ and PQ diameter core. The holes were reduced to NQ when problems were encountered due to difficult ground conditions and/or thick fault zones. Core was placed in plastic boxes with depth markers for every drill run of up to 3 m.

At the Halilağa Project, core recoveries are considered by the authors to be good and within tolerance to include in a resource estimate. Figure 10.1 and Figure 10.2 show the scatter plots for Recovery% versus Cu and gold grade, respectively. There does not appear to be any bias toward high grade versus low grade recoveries. Drill holes HD-41, HD-44, HD-46, and HD-94A returned relatively low recoveries in relation to other drill holes. These holes are within the central volume used for resource estimation. The grades were visually checked and compared against adjacent data and no issues were identified. Other holes with low recoveries were identified but were outside the resource area. Recoveries in drillholes HD-101, HD-105 and HD-105A were relatively low in comparison to other holes however they were drilled after the effective date of the resource but should be reviewed.

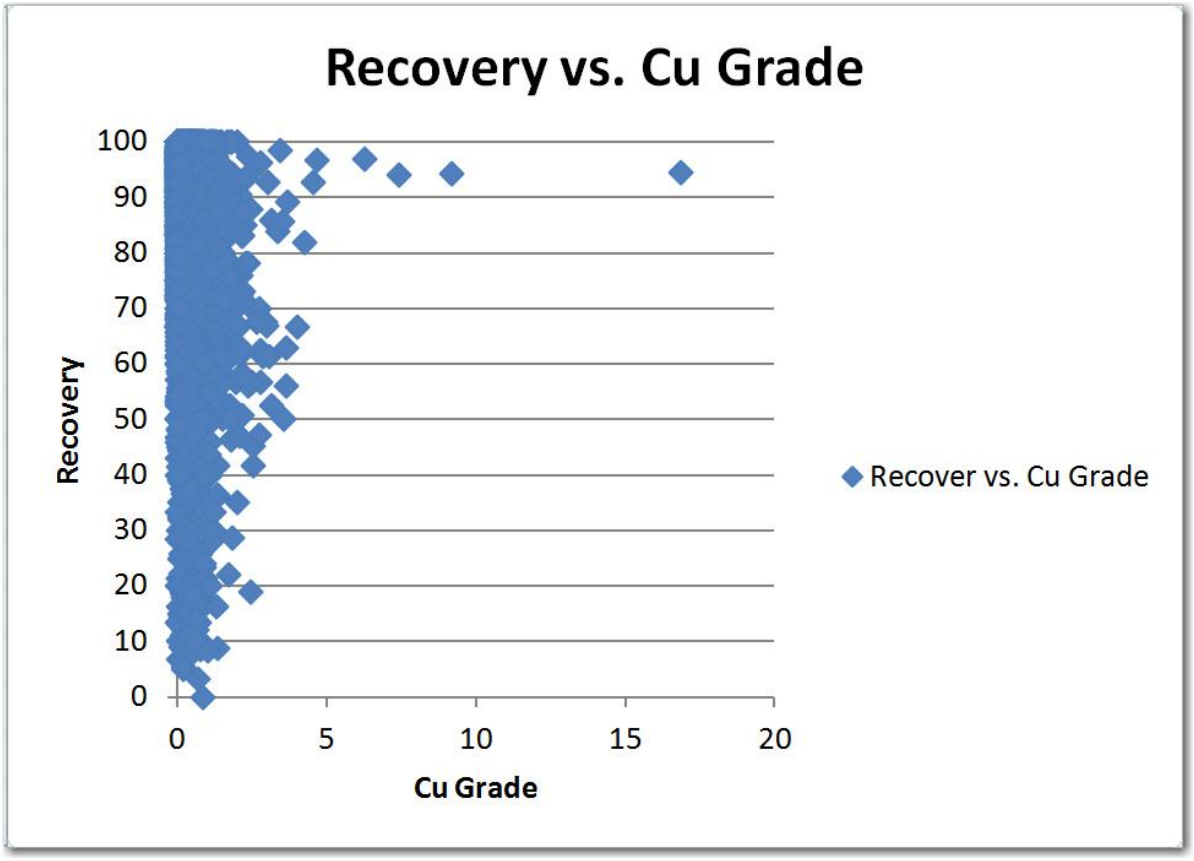



**Figure 10.1: Core Recovery vs. Copper Grade (%)**

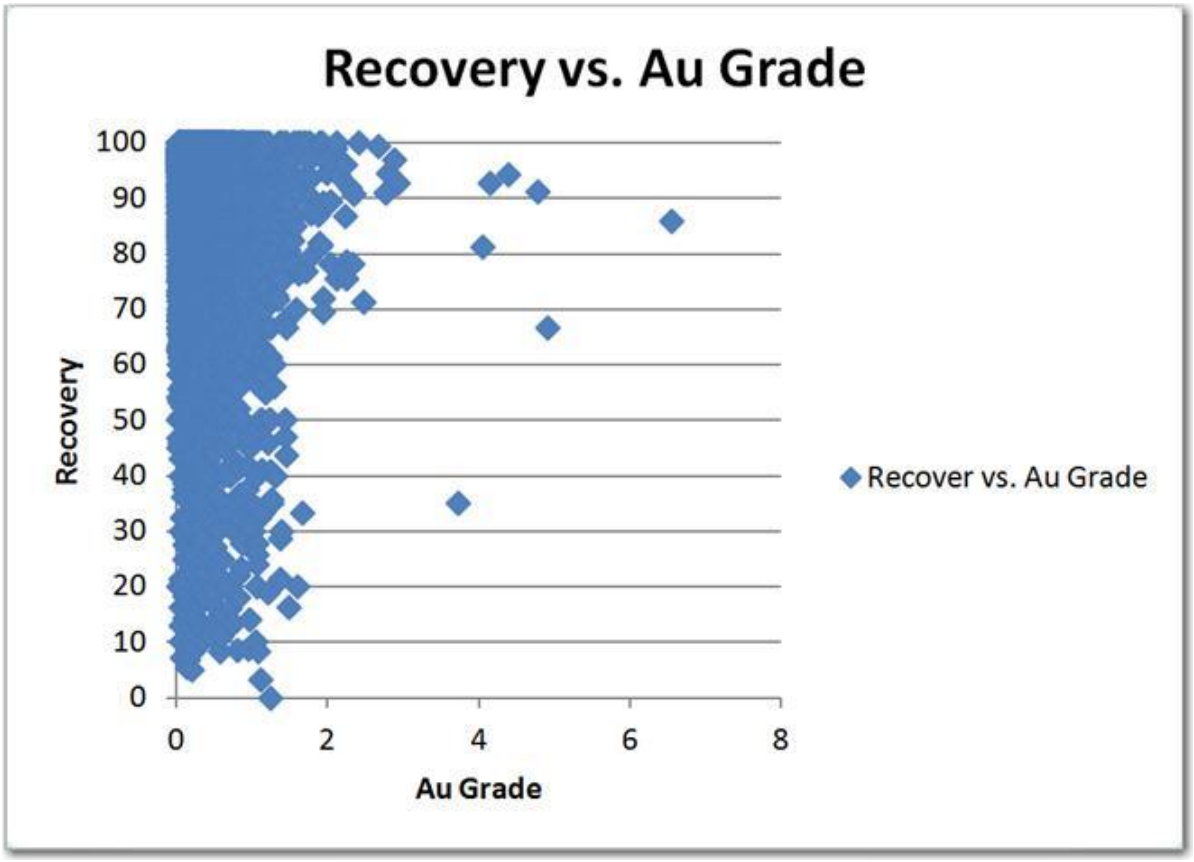



**Figure 10.2: Core Recovery vs. Gold Grade (g/t)**

Most of the diamond drill holes were completed using HQ size core and recovery was not an issue, except for fault zones.

Boxes were securely sealed and brought by truck to the core facility at the Etili camp once a day by the drilling company or Truva Bakir staff. Reflex survey tests were taken at 50 m intervals down-hole to provide control on drill hole orientation.

At the core handling facility, drill holes were logged by Truva Bakir geologists recording observations using the Anaconda method (Einaudi 1997) and then entered into the database using Acquire® software. Prior to logging, the geologist and the field technicians performed the following tasks:

- Inspected core boxes;
- Recorded missing boxes and footage errors;
- Replaced footage markers with clean, clear markers;
- Digitally photographed all boxes;
- Recorded rock quality designation (RQD) and core loss; and
- Logged core; information included engineering comments regarding the competency of core and a fracture analyses that included quantitative measurements of primary fractures, gouge material, veins, and dominant fracture patterns.

Specific gravity (SG) measurements from drill cores were routinely carried out for both oxide and sulphide mineralization. Pieces (10 cm-20 cm) of solid core were used for SG measurements. Both mineralized and un-mineralized zones were measured. The solid core was cleaned and washed and

dried in an oven. Samples were dried at 105°C for 8-12 hours and then weighed in air (dry weight). They were then coated with paraffin, which was allowed to dry, and then the samples were reweighed in water.

The SG values were identified by the following formula:

$$SG = \frac{(\text{dry weight})}{(\text{dry weight}) - (\text{weight in water})}$$

## 10.2 Sample Preparation

Sample preparation and analysis of core from the first 35 drill holes at Halilağa was conducted by ALS Minerals ("ALS"). In addition, samples from three early reverse circulation holes were similarly prepared and analyzed by ALS. Since October 18, 2009, or from drill hole HD-36 onward, the drill core samples were prepared and analyzed by Acme Analytical Laboratories ("Acme").

### 10.2.1 Preparation at ALS Minerals

Before October 2009, all core samples were prepared at the on-site preparation lab at Truva Bakir's Etili camp which was set up, managed, and run by personnel from the ALS Minerals sample preparation facility in Izmir, Turkey (Cook, 2007). Samples were weighed on arrival, dried, crushed in their entirety to at least 70% passing 2 mm using a TM Terminator crusher, and riffle splitting to 1,000 grams. This amount was pulverized to 85% passing 200 mesh in a TM Max2 pulverizer and disk mill. The master pulp remained in Turkey while an approximately 100g pulp packet was forwarded to the ALS laboratory in North Vancouver, Canada for analysis and assay.

### 10.2.2 Preparation at Acme Labs

Core samples were prepared at Acme Lab's sample preparation facility in Ankara, Turkey, from October 2009 onward. The R200-1000 sample preparation method were used, which includes barcoding and weighing of the sample on arrival, drying, coarse crushing of the entire core sample to at least 80% passing 10 mesh (2 mm) in a TM Terminator, riffle splitting to approximately 1,000 grams, and pulverizing this material in a LM-2 carbon steel disk mill to 85% passing 200 mesh. An approximately 100g pulp packet was forwarded by independent transport to Acme's main laboratory in Vancouver, Canada for analysis and assay, with the remaining master pulp material for each sample staying in Ankara, Turkey. Rejects and pulps were stored either on-site at the Etili core shack or in Ostim, Ankara, Turkey. Documentation of preparation, analysis, and QA/QC protocols for the Halilağa Project was provided by Cook and Houle (2009).

In mid-2011, Acme Labs opened a new laboratory in Ankara, and since April 18, 2011 all sample preparation and subsequently gold fire assay work, has been carried out at this location.

## 10.3 Assay and Analysis

### 10.3.1 Assay and Analysis at ALS Minerals

As previously mentioned, before October 2009, core samples from the first 35 drill holes at Halilağa (up to HD-35) were analyzed at the ALS laboratory in North Vancouver, Canada. Core samples were analyzed by a general aqua regia-digestion geochemical suite, a gold fire assay technique, a

copper assay technique and, where warranted, a sequential copper procedure to determine supergene copper. The following is a synopsis of the specific methods used:

- Geochemical Analysis – ME ICP41 suite (aqua regia digestion/ICP-ES). This method was originally employed to screen samples for copper assay, prior to assaying all samples within the deposit.
- Gold fire assay – Au-AA24 method. This lead-collection fire assay technique uses a 50 g sample split size with AAS finish, and has a stated detection limit of 0.005 ppm.
- Copper Assay – Cu-AA62 method. This four-acid digestion/AAS method is a total copper assay method. The initial base metal assays of the first 35 drill holes by ALS were for copper assay only. No molybdenum, iron, or silver assays have been conducted on these samples.
- Sequential Copper Assay - These were conducted, where warranted, on all core samples with visible supergene copper mineralization. The method includes three sequential leaches, including the total copper assay of the residual component, and a total copper assay on a separate sample split. Sequential copper assays were also applied to the full length of the first eight Halilağa drill holes. The process includes the following procedures, the first three of which are conducted on a single sample split:
  - AA06 sulphuric acid leach;
  - AA16s cyanide leach;
  - AA62s total copper on the residual (4-acid digestion/AAS); and
  - AA62 total copper on a separate sample split (4-acid digestion/AAS).

### 10.3.2 Assay and Analysis at Acme Labs

As previously mentioned, since October 2009 the Halilağa core samples were analyzed at Acme Analytical laboratories in Vancouver, Canada, beginning with HD-36. Core samples were analyzed for a general aqua regia-digestion geochemical suite to provide information on deleterious and pathfinder elements, a gold fire assay technique, a base metal assay technique and, where warranted, a sequential copper procedure for to determine supergene copper. The following is a synopsis of the specific analytical methods used:

- Geochemical Analysis – Group 1DX 01 multi-element suite (aqua regia digestion/ICP-MS). This method provides low-level concentrations of aqua regia-digestible commodity, deleterious and pathfinder elements such as As, Sb, Hg, Bi, and S on a 0.5 g sample weight;
- Gold Fire Assay – Group 3B01 method. This lead-collection fire assay technique uses a 30 g sample split with ICP-ES instrumental finish. The method, which has a stated detection limit of 2 ppb, is suitable for gold concentrations < 10 g/t. As noted, since mid-2011, all gold fire assay work has been carried out at Acme's new laboratory in Ankara;
- Base Metal Assay – Group 7TD02 method. This base metal total assay method uses a strong four-acid digestion followed by an ICP-ES instrumental finish to provide data for 23 elements including Cu, Mo, Ag, and Fe; and
- Sequential Copper Assay – Group 9 04 method. These were conducted, where warranted, on all core samples with visible supergene copper mineralization. The method includes three sequential leaches, including the total copper assay of the residual component, and a total

copper assay on a separate sample split. All sequential copper methods were conducted at the Acme Laboratory in Santiago, Chile.

In Acme-Santiago, the Group 8TD method (4-acid digestion/AAS) was used for total copper assay in place of the usual Group 7TD method. The Acme Labs sequential leach process includes the following procedures, the first three of which are done on a single sample split:

- Group 9 04 sulphuric acid leach;
- Group 9 04 cyanide leach;
- Group 9 04 total copper on the residual (Group 8TD 4-acid digestion/AAS); and
- Group 8TD total copper on a separate sample split (4-acid digestion/AAS).

Acme laboratories operate according to the guidelines set out in ISO/IEC Guide 25 – “General requirements for the competence of calibration and testing laboratories”.

When sample shipments are received by the laboratory, an emailed notification is sent. Throughout the program, no problems were encountered during transport.

Note that Total Sulphur by Leco analyses have not been routinely carried out on Halilağa core at either ALS or Acme Labs. Leco Sulphur has been determined on only a single drill hole namely, HD-02. Also a small number of low-concentration core samples from earlier drill holes (i.e. 19 samples from ALS, and 1 from Acme Labs), originally analyzed for Cu and other elements, were done by aqua regia/ICP-ES or ICP-MS analysis only and have since been re-analysed for Cu and Mo.

Reverse Circulation Drilling Sampling

All drilling samples were subjected to quality control procedures that ensured best practice in the handling, sampling, analysis, and storage of the drill core. All drill holes (except abandoned holes in cover rock) were sampled and assayed continuously. Sample intervals were selected on a geological basis and were typically 2.0 m in length. Core was cut length-wise with half the core placed in individually sealed cloth bags and submitted for assaying, and the other half was kept in the core box.

Reverse circulation (“RC”) samples were collected and split using a 24-slot rotary splitter at the drill site and then sealed in plastic bags. Samples were collected continuously at 1.0 m-1.5 m intervals. The splitter was cleaned between each sample with a compressed air hose. The RC drill samples were taken and kept under constant supervision by Truva Bakir personnel.

There are no known factors related to drilling and sampling that would materially impact the accuracy and reliability of the results. Recoveries are generally high although issues in some holes have been identified and the authors are satisfied that the holes with low recoveries have satisfactory results and can be included in the database.

## 10.4 Quality Assurance and Quality Control Programs

Core boxes were securely sealed and brought, by truck, to the core facility at the Etili camps once a day by the drilling company or Truva Bakir. Here they were logged, cut, bagged, tagged and stored prior to being shipped to the analytical laboratory. Prior to 2009, all core samples were prepared at the ALS sample preparation laboratory located at Etili camp and processed within the secure confines of the camp prior to the pulp packets being transported by commercial air carrier to the ALS laboratory in North Vancouver, Canada. Following the switch to Acme Labs in late 2009, all Halilağa

core samples were first trucked from Etili to the Acme preparation facility in Ankara by independent transport. Pulp packets were subsequently transported by commercial air carrier to Acme Labs in Vancouver, Canada, for assay and analysis.

Quality assurance and quality control measures used on the Halilağa Projects were employed at all stages of work in the core shed, the sample preparation facility, and in the analytical laboratory. Evaluation of QA/QC results was done systematically and promptly to ensure that only the best quality data was entered into the project database. Umpire, or external check, assays have been carried out as a further means of data verification. At all times this work, whether in the field, the lab, or the exploration office, was consistent with best practices currently in use in the mineral exploration industry.

### 10.4.1 Core Shed Procedures

Quality control measures and data verification procedures were applied to the acquisition of drill data and to the sampling of core. The following procedures were employed to ensure quality assurance and quality control:

- Standardized sample booklets were used at all times. All booklets were marked up, prior to use, with the standards, field blanks, and core duplicates locations clearly defined;
- Standards and field blanks were entered every 20th sample. Core duplicate samples (1/4 core) are entered into the sample flow, at the discretion of the geologist, approximately every 20 samples;
- All holes were sampled from top to bottom of the hole, with most samples being taken every 2 m;
- For each sample interval, all required parts (from-to) of the standard sample card were filled in and half of the sample number tag was placed at the starting point of the sample interval in the core box;
- The second half of the tag was put into the sample bag (labelled on both sides with the sample number) by the splitters when they were taking the sample;
- The beginning of a sample is clearly marked with a black marker, and a line perpendicular to the core; and
- It was the geologists'responsibility to double-check the samples once they were cut, and to verify that all of the collected samples were properly labelled, with the sample tags inside of the sample bags.

At the Halilağa Project, inserting quality control samples takes place in the core shack before samples are shipped to the lab. These samples are routinely inserted and are used to check for accuracy, precision and cleanliness in the analytical laboratory. At the beginning of sampling, sample tags are pre-marked before logging with locations for standards, core duplicates, and field blanks. The process is as follows:

- Core duplicate samples are taken every 20 samples within the sample series (5%). Core duplicate samples are used to evaluate combined field, preparation and analytical precision. The core duplicate samples are quarter-spilt cores sampled on-site before the samples leave camp;
- Field blanks are non-mineralized limestone material collected from a local source, broken with a hammer, and inserted into the sample series every 20 samples (5%). Field blanks are inserted to

test for any potential carry-over contamination which might occur in the crushing phase of sample preparation, as a result of poor cleaning practices;

- Standards are used to test the accuracy of the assays and to monitor the consistency of the laboratory over time. Commercially-available Cu-Au assay standards were purchased from CDN Resource Laboratories Ltd., Vancouver, Canada. These standards are inserted into the sample sequences approximately once every 20 samples (5%). The identities of the standards are blind to the laboratory; and
- Pulp blanks, purchased commercially from CDN Resource Laboratories Ltd., are also inserted in the analytical suites in order to assess any potential carry-over contamination in the analytical lab. The pulp blanks are not involved in sample preparation, and do not test the preparation stage of the sample stream as the field blanks do.

The author has relied on data and information related to quality assurance and quality control that has been prepared by a QP as defined by the requirements of NI 43-101. The QA/QC data was prepared and evaluated by Truva Bakir personnel.

Commercial standards sourced from CDN Resource Laboratories Ltd., Vancouver, Canada, were used to test the accuracy of the assays and to monitor the consistency of the laboratory over time. All standards listed here are Cu-Au standards with recommended values (between-lab mean ± 2 standard deviations) for copper, gold, and in some cases, molybdenum. These standards were randomly inserted into the sample sequences approximately once every 20 samples. Table 10.1 and Table 10.2 show the standards used for the Halilağa Project in 2009 and 2010, along with their recommended mean gold and copper concentrations.

**Table 10.1: Recommended Gold Concentrations of Standards Used at Halilaga (Teck Medencilik, 2010)**

| Standard ID | Au (ppb) |
|---|---|
| CDN-CGS-7 | 950 |
| CDN-CGS-16 | 140 |
| CDN-CGS-15 | 570 |
| CDN-CGS-12 | 290 |
| CDN-CGS-11 | 730 |
| CDN-CGS-10 | 1730 |

**Table 10.2: Recommended Copper Concentrations of Standards Used at Halilaga (Teck Medencilik, 2010)**

| Standard ID | Cu (%) |
|---|---|
| CDN-CGS-10 | 1.55 |
| CDN-CGS-11 | 0.683 |
| CDN-CGS-12 | 0.265 |
| CDN-CGS-15 | 0.451 |
| CDN-CGS-16 | 0.112 |
| CDN-CGS-7 | 1.01 |

## Sample Preparation Lab Procedures

Crusher duplicate samples were prepared in the sample preparation labs by laboratory personnel, at Truva Bakir's request, and inserted into the laboratory sample batches for subsequent assay and analysis. Crusher duplicates were prepared after the crushing stage, but prior to the pulverization stage of sample preparation. Their purpose was to test the combined preparation and analytical precision in the laboratory, and was designed to measure the quality of sample preparation.

## Analytical Laboratory Procedures

In addition to standards, core duplicates, field blanks, and pulp blanks which were inserted in the core shed, and the crusher duplicate samples which was prepared in the prep lab, laboratory pulp duplicate samples were also prepared during final sample analysis. In this case the pulp duplicates were part of the laboratory's (ALS and Acme Labs) own internal QA/QC procedures; these results were reported by the labs and subsequently captured as a measure of analytical precision at the final digestion and/or fusion and instrumental stages of analysis.

## Umpire (External Check) Assays

After initial analysis at ALS, five percent of all assayed sample pulps were sent to a second laboratory for analysis as a further means of data verification. Note that this was employed during the time when ALS was the source laboratory, was discontinued in 2009 and re-implimented in 2012. This procedure helped to identify any substantial biases between laboratories that may have been introduced during the project.

Only reputable laboratories have been used as the secondary laboratory for umpire assays. After Acme Labs took over the role as primary laboratory for the Halilağa Project in late 2009, ALS -North Vancouver was identified as the secondary laboratory and all external check samples were to be assayed there. The digestion/decomposition method used at the secondary laboratory was to be identical to the original assay method, and the same standards were to be used in both analytical suites.

## Evaluation of QA/QC Results

Results for standards, field blanks, pulp blanks, and various types of duplicate samples were reviewed and evaluated quickly after the batch results were received. Several types of control charts are used to plot and monitor the data. Decisions were made immediately, based on simple criteria, as to whether or not the quality control results were acceptable.

## Standards

Failure of a standard implied that all routine samples within its sphere of influence were also considered to have failed, and must be re-analyzed at the same primary laboratory. Two criteria for the passing and failing of standards results have been used on the Halilağa Project:

- Standards were considered to have failed if the reported copper or gold assay concentration was greater or less than three standard deviations from the recommended mean value for that standard; and
- Standards were also considered to have failed if the reported copper or gold assay concentration was greater or less than two standard deviations from the recommended mean value for that standard, in two or more successive insertions.

In the case of failure of any standard, the procedure was to re-assay the block of samples within its sphere of influence. In practice, this meant that all consecutively listed samples, down list from the failing standard to the next passing standard, and up list from the failing standard to the next prior passing standard, were considered to have failed, and were to be re-assayed.

Upon receipt of re-assay data, the standards were re-evaluated, and if they passed the data is similarly deemed to have passed. At this point, the failed data was ejected from the project database and replaced with the new, passing data.

During the significant drilling campaigns of 2009-2011, the copper standards had a failure rate of between 1% – 3%, and molybdenum standards had a failure rate of 1%. However, the gold standards exhibited a failure rate of between 10% –11% for 2009, 2010, and 2011 which is very high; however, these are a result of two consistently failing single gold standards: CDN-CGS-15 and CDN-CGS-16 and due to the adoption of a low failure threshold. It is important to note that the majority of failures were attributed to these failing standards. In addition, note that when a standard fails, the complete assay run, before and after the fail,was re-run until the standard passes.

The overall failure rate at ALS was 19 in 733 analyses, or 2.6% as shown in Table 10.3. This compares to the Acme rate of 126 in 1,864 analyses, or 6.8% as shown Table 10.4. The gold by fire assay failure rate was 3.2% and 6.3% for ALS and Acme, respectively.

**Table 10.3: Standards Performance for the Halilağa Project (2004 - 2009)**

| STD-ID | N | No of Cu% by ME41 (ICP) | No of Cu% By AA62b (AA) | No. of Au ppm by AA24 (FA) | No. of Mo % by ME41 (ICP) | Cu % Limits | Mo Limits % | Au ppm Limits | Failures (No, %) |
|---|---|---|---|---|---|---|---|---|---|
| CDN-CGS-10 | 80 | 30 (0) | 19 (0) | 31 (1) | ***30*** | 1.505-1.955 | NA | 1.505-1.955 | 1 (1.3%) |
| CDN-CGS-11 | 95 | 37 (2) | 19 (0) | 39 (0) | ***37*** | 0.644-0.722 | NA | 0.628-0.832 | 2 (2.1%) |
| CDN-CGS-12 | 104 | 42 (1) | 16 (0) | 46 (6) | ***42*** | 0.243-0.288 | NA | 0.230-0.350 | 7 (6.7%) |
| CDN-CGS-7 | 87 | 34 (0) | 18 (0) | 35 (2) | ***34*** | 0.905-1.115 | NA | 0.830-1.070 | 2 (2.2%) |
| CDN-CGS-8 | 82 | 30 (0) | 19 (0) | 33 (3) | ***30*** | 0.093-0.117 | NA | 0.062-0.098 | 3 (3.7%) |
| CDN-CM-3 | 32 | 1(0) | 0(0) | 30 (0) | 1(1) | 0.517-0.580 | 0.025-0.034 | 0.370-0.550 | 1 (3.1%) |
| CDN-GS-1P5 | 25 | ***24*** | ***14*** | 25 (0) | ***24*** | NA | NA | 1.349-1.816 | 0 (0%) |
| CDN-GS-2A | 30 | ***29*** | ***28*** | 30 (0) | ***29*** | NA | NA | 1.758-2.325 | 0(0%) |
| CDN-GS-2B | 6 | ***6*** | ***4*** | 6 (0) | ***7*** | NA | NA | 1.850-2.210 | 0(0%) |
| CDN-GS-2C | 23 | ***23*** | ***0*** | 23 (0) | ***0*** | NA | NA | 1.835-2.285 | 0(0%) |
| CDN-GS-5B | 57 | ***54*** | ***48*** | 57 (0) | ***54*** | NA | NA | 4.260-5.397 | 0(0%) |
| CDN-GS-P3 | 58 | ***57*** | ***51*** | 58 (1) | ***57*** | NA | NA | 0.240-0.360 | 1(1.72%) |
| CDN-GS-P5 | 50 | ***50*** | ***41*** | 50(0) | ***50*** | NA | NA | 0.461-0.589 | 0(0%) |
| CDN-GS-5C | 4 | ***3*** | ***0*** | 4(2) | ***3*** | NA | NA | 4.320-5.160 | 2 (50%) |
| | 733 | 174 (3) | 91 (0) | 467(15) | 1 (1) | | | | 19 in 733 |
| | | 1.80% | 0% | 3.20% | 100% | | | | 2.60% |

**Table 10.4: Standards Performance for the Halilağa Project (2009 – 2012)**

| STD-ID | N | No of Cu by 1DX | No of Au by 3B | No of Cu by 7TD | No of Mo by 7TD | No of Mo By 1DX | Cu% Limits | Mo% Limits | Au ppm Limits | Failures (No, %) |
|---|---|---|---|---|---|---|---|---|---|---|
| CDN-CGS-10 | 113 | 3 (3) | 56 (2) | 54(1) | ***53*** | ***3*** | 1.505-1.955 | NA | 1.505-1.955 | 6 (2.6 %) |
| CDN-CGS-11 | 120 | 11 (7) | 56 (3) | 53 (1) | ***3*** | ***11*** | 0.644-0.722 | NA | 0.628-0.832 | 11 (9.2%) |
| CDN-CGS-12 | 133 | 48 (5) | 69 (6) | 16 (0) | ***16*** | ***48*** | 0.243-0.288 | NA | 0.230-0.350 | 11 (8.2%) |
| CDN-CGS-15 | 221 | 47 (25) | 96 (8) | 78 (1) | ***76*** | ***47*** | 0.421-0.481 | NA | 0.480-0.660 | 34 (15.3%) |
| CDN-CGS-16 | 465 | 42 (18) | 233 (20) | 190 (3) | ***184*** | ***42*** | 0.104-0.112 | NA | 0.071-0.209 | 41 (8.8%) |
| CDN-CGS-7 | 12 | 4 (0) | 4 (0) | 4 (0) | ***4*** | ***4*** | 0.905-1.115 | NA | 0.830-1.070 | 0 (0%) |
| CDN-CGS-21 | 166 | 0 (0) | 84 (2) | 82 (2) | ***80*** | ***0*** | 1.174-1.426 | NA | 0.855-1.125 | 4 (2.4%) |
| CDN-CM-2 | 209 | 0 (0) | 73 (6) | 70 (4) | 66(1) | 0 | 0.949-1.078 | 0.026-0.032 | 1.225-1.615 | 11(5.2%) |
| CDN-CM-3 | 335 | 0(0) | 117 (5) | 110 (3) | 108(0) | 0 | 0.517-0.580 | 0.025-0.034 | 0.370-0.550 | 8(2.4%) |
| CDN-CM-5 | 90 | 0(0) | 30(0) | 30(0) | 30(0) | 0 | 0.289-0.349 | 0.043-0.058 | 0.225-0.363 | 0(0) |
| | 1,864 | 155(58) | 818(52) | 687(15) | 204(1) | | | | | 126(6.8%) |
| | | 37.40% | 6.30% | 2.20% | 0.50% | | | | | |

There continues to be a relatively high failure rate of the gold standards which appears to be attributed to two particular standards. However, any failures were recorded and the lab requested the complete job be re-run. Additional check assays were also sent to the ALS Chemex Lab in North Vancouver as a further means of data verification. An internal review of the QA/QC procedures of the lab and Truva Bakir's protocols are on-going. The author considers that the adequacy of sampling, security, and analytical procedures is satisfactory with the exception of the high failure rate of standards.

## Field Blanks

Field blanks are used to check the level of cleanliness at a laboratory, and to more specifically check for the presence of any carry-over contamination during the crushing phase of sample preparation. Proper cleaning of the coarse crushers between samples, and between sample batches, should ensure that there is no carry-over of material between samples, producing negligible gold and Cu results on a consistent basis. Field blanks are typically created from barren rock material, preferably of hardness similar to the target lithologies. At Halilağa, non-mineralized limestone material collected from a local source, broken with a hammer, and inserted into the sample series every 20 samples (5%). In addition to the field blanks, commercial CDN Resource Laboratories Ltd. pulp blanks were randomly inserted into the sample series every 20 samples as a check on any possible carry-over contamination in the analytical laboratory.

In general, field blanks exhibited a failure rate of 1% for each of the years, which indicated that carry-over contamination at the crusher stage has not been a problem on this project.

## Duplicate Samples

Both core-duplicate and crushed-duplicate samples are created for insertion in Halilağa core assay batches. The core duplicate samples are 1/2 split of core which are cut on-site before the sample leaves camp. Acme Labs has also prepared coarse-crusher duplicates and inserted these at a rate of one every 20 samples. Core-duplicate samples are taken every 20 samples within the sample series.

## Umpire (External Check) Assays

A protocol has been initiated to send 5% of all assayed sample pulps to a second laboratory for analysis. Prior to 2009, ALS was the main lab at the project with Acme Labs providing external check assays. In 2009, Acme became the main lab with ALS stated to provide external check assays. Umpire, or external check, assays are an additional means of data verification and are useful in identifying any substantial biases between laboratories which may have been introduced during the course of the project. Although 5% external checks are part of the QA/QC protocol, in reality much less than that has been completed.

Correlation plots are shown for Hole HD-02 between the two laboratories for Cu (Figure 10.3), gold (Figure 10.4) and molybdenum (Figure 10.5).

Figure 10.3 shows the umpire, external checks, for Cu geochemistry (ALS;ME-ICP41 and Acme;Group 1DX) and Cu assay (ALS;Cu-OG-62 and Acme;Group 7AR) which illustrates an excellent correlation between the two laboratories.

Figure 10.4 shows the umpire, external checks for gold assay (ALS;Au-AA24 and Acme;Group 3B). With the exception of a small number of outliers, the data between the two laboratories shows good agreement.

Figure 10.5 shows the umpire, external checks, for Mo geochemistry (ALS;ME-ICP41 and Acme;Group 1DX) which illustrates a good correlation between the two laboratories.

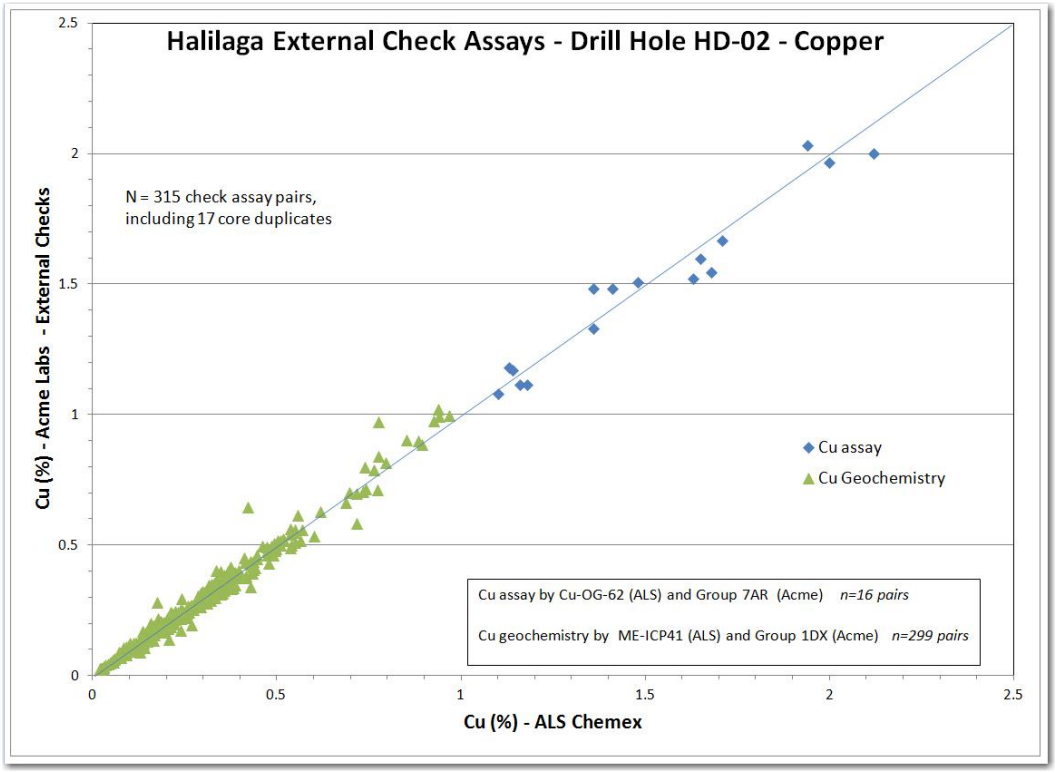



**Figure 10.3: Umpire Check Assays for Drill Hole HD-02; Cu (%)**

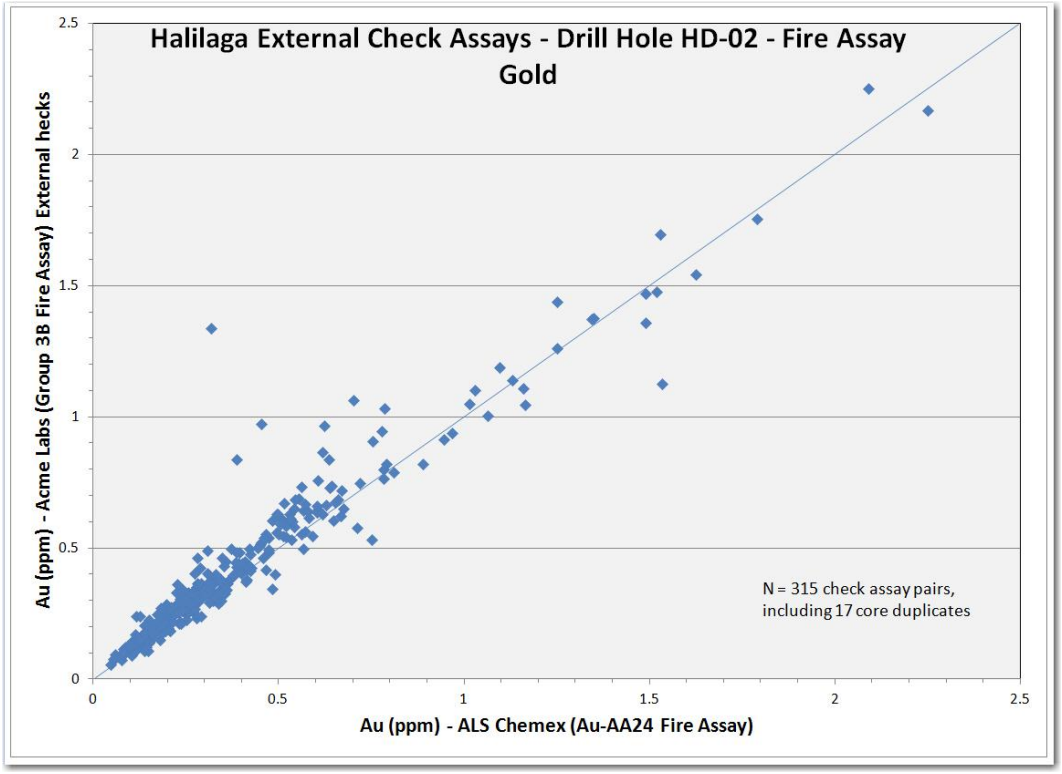



**Figure 10.4: Umpire Check Assays for Drill Hole ID HD-02; Au (ppm)**

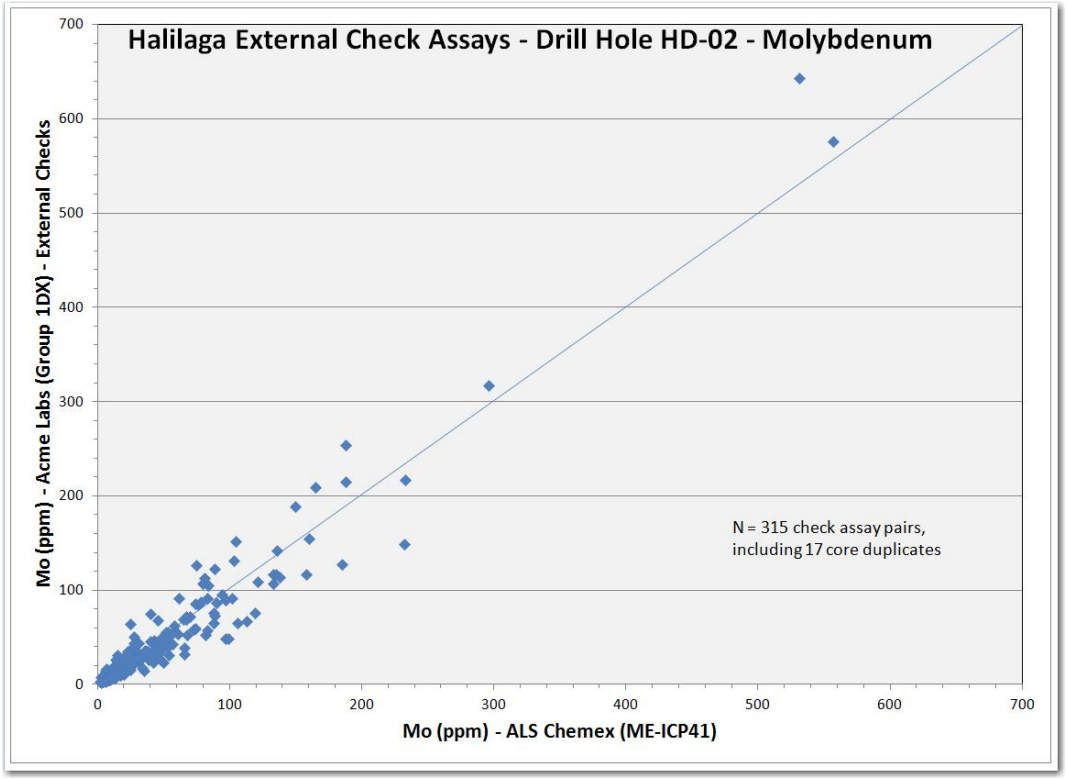



**Figure 10.5: Umpire Check Assays for Drill Hole HD-02; Mo (%)**

Heberlien Geosconsulting was contracted to audit the quality control procedures and data for the Halilağa Project which resulted in the report “Halilağa Project; An Audit of Quality Control Procedures and Data for Teck Resources Inc.” dated July 20, 2012. This audit has shown that the QC procedures employed at Halilaga are for the most part consistent with industry “Best Practices” and compliant with NI 43-101 requirements.

Key findings and recommendations from the audit, areas stated in the report (Heberlien, 2012), are:

1. Data quality was found to be acceptable. The precision for copper and gold are on the whole very good for porphyry-style mineralization. Nevertheless precision analysis has highlighted some potential problems with the quality of the sample preparation specifically with the crusher product, where duplicate samples indicate poor homogeneity. This issue needs to be discussed with the prep lab as soon as possible and steps taken to improve the homogeneity of the crushed material. Increasing the crusher specification to 90% passing -2mm should help to achieve this. Adding a homogenization step before sub-sampling would also help to improve precision.
2. 2. Routine Thompson and Howarth and CVav precision analysis should be implemented to monitor precision over time. Pay attention to precision at cut-off grade. Precision should be estimated on at least a quarterly basis for the entire duplicate dataset (field, crusher and pulp).
3. Accuracy is very good as shown by the failure rates of the standards and interlaboratory check assays. For standards with high failure rates resulting from bias, it is recommended that the accepted value and standard deviation limits be recalculated using the actual results for those standards. This would significantly improve the failure rate. Standards that would benefit from recalculated limits are: CDN-CGS-11, CDN-CM-1 and CDN-CGS-12.

4. Maintain copies of standard certificates wherever the standards are used. This would normally be in the core shed and database area.
5. Monitoring of gold accuracy using CDN-CGS-16 should be discontinued. This standard has an unacceptably high RSD for gold. It can still be used to monitor copper accuracy.
6. The use of Rock Labs oxide standards should be discontinued because their matrices are incompatible with the sulphide mineralization being drilled. Use of non-matrix matched standards can result in biases, particularly for gold.
7. Standard results should be interpreted on control charts using appropriate failure rules (see above). In addition to the AcQuire snap shot plots, standard results should be plotted on control graphs with the entire dataset for each standard. By doing this abrupt changes in standard behaviour, systematic trends and biases can more readily be identified and acted on.
8. If not already present in AcQuire, add a failure/action table to document failures and actions taken to correct them. Such a table provides a digital paper trail for the database.
9. Develop procedures to investigate standard failures. The possibility of self inflicted failures such as standard mix-ups should be eliminated before requesting re-analysis of a batch at the lab.
10. Obtain a suitable molybdenum standard with values certified in ppm or percent to four significant figures.
11. Replace the limestone field blank with one with a silicate matrix. An ideal blank should have a similar or greater hardness than the rocks being analyzed so that it can scour material from previous samples from the crusher jaws. Field blanks should be certified as blank by repeated analyses for the elements of interest.
12. Request monthly granulometry reports for crusher and pulverizer products from the laboratory.
13. Carry out periodic audits and visits to the Ankara prep lab.

# 11 Data Verification

## 11.1 Site Visit

Garth Kirkham, P. Geo., visited the property between August 13 and August 16, 2011. The site visit entailed inspection of the camp, accommodations, core logging facilities, offices, active drill sites, outcrops, historic drill collars, core storage facilities, core receiving area, core sawing stations, and tours of major centres and surrounding villages most likely to be affected by any potential mining operation.

The tour of the offices, core logging and storage facilities showed a clean, well-organized, professional environment. On-site staff led Kirkham through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are to North American, industry standards and no issues were identified with the exception of the suspension of the measurement of SGs at the designated density measurement stations. This oversight has since been rectified.

Kirkham also visited two active drill sites. One of the drills was down due to mechanical issues. However, the second drill was observed and the drill site processes and procedures were demonstrated. The drill sites are clean and well-maintained. The process of extracting the core and core boxing is standard practice. The boxes are sealed in plastic core boxes, labelled with permanent marker, and sealed securely with bailing wire. The boxes are loaded into the drillers' truck or into the geologist's vehicle and taken to the camp.

A visit to collar locations showed the collars were well marked and labelled, therefore easily identified. The earlier drill holes were cased and permanently cemented.

Four complete drill holes: HD-01, HD-08, HD-92A, and HD-94A were selected at random from the database by Kirkham and laid out at the core storage area. Site staff supplied the logs and assay sheets for verification against the core and the logged intervals. The data correlated with the physical core and no issues were identified. In addition, Kirkham toured the complete core storage facilities pulling and reviewing core throughout. No issues were identified and recoveries appeared to be very good to excellent.

Kirkham is confident that the data and results are valid based on the site visit and inspection of all aspects of the project, including methods and procedures used. It is the opinion of the independent Kirkham that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101. No duplicate samples were taken to verify assay results, but Kirkham is of the opinion that the work is being performed by a well-respected, large, multi-national company that employs competent professionals that adhere to industry best practices and standards.

# 12 Mineral Processing and Metallurgical Testing

## 12.1 General

Ausenco has reviewed the Halilağa metallurgical test reports and data to develop process design criteria for the scoping study. The existing test work is considered suitable for this level of study and the comminution data are considered adequate for a conceptual milling circuit design. The design of the processing circuits is based on test work data in conjunction with reasonable assumptions based on typical industry values when insufficient data were available.

The Halilağa mineralized material is of moderate competency and hardness, and amenable to grinding in a conventional SAG-ball milling circuit with pebble crushing ("SABC"). The mineralogy is fine grained and test work indicates a requirement to re-grind to a fine particle size to achieve adequate liberation for flotation.

The Halilağa process plant and associated service facilities would process 50,000 t/d of Run of Mine ("ROM") material, to produce copper concentrate and tailings. The proposed process includes crushing and grinding of the ROM material, rougher and cleaner flotation, regrinding, and dewatering of copper sulphide minerals. The copper concentrate would be trucked from site to the Port of Bandirma while the flotation tailings would be thickened before placement in the TMF.

## 12.2 Metallurgical Testing

### 12.2.1 Test work Programs

Ausenco has reviewed the following metallurgical and mineralogy test reports supplied by Pilot Gold:

- G&T Metallurgical Services Ltd., Preliminary Metallurgical Assessment of Samples from the Halilağa Project KM2020, 11th July 2007;
- G&T Metallurgical Services Ltd., Mineralogy Testing on Samples from the Halilağa Project KM2157, 1st April 2008; and
- G&T Metallurgical Services Ltd., Metallurgical Testing KM2098, 15th April 2011.

The first test program (G&T 2007) tested composites made from two drill holes HD-01 and HD-04 for preliminary flotation to produce copper concentrates and 25 variability rougher flotation tests.

For the second test program (G&T 2011), 15 composite samples were formed from drill holes HD-40 and HD-49. A master composite called "Typical Ore" was then formed from the 15 composites. *(The use of the term "Ore" represents the common name for plant feed material used to identify a sample but in the case of this project, does not refer to mineral reserve material).* Comminution testing (Bond ball mill work index) was performed on six of the composite samples and all 15 composite samples were tested for preliminary flotation to produce copper cleaner concentrates. Two locked cycle tests were performed on the "Typical Ore" composite.

Ausenco has reviewed the location of the samples and drill holes and concludes the samples are reasonably representative of the deposit for a scoping study stage given the nature and continuity of rock types and mineralization of the deposit.

Where no test work data are available, reasonable assumptions, based on operating data or test work from other projects has been used to develop the process design criteria used for the plant design.

## G&T July 2007 Test Program

The first test program conducted in 2007 was based on sample intervals from two drill holes. Bench scale flotation testing was conducted on two master composites as well as 25 discrete variability composites.

The composites were made up of the reject samples from two drill holes designated HD-01 and HD-04. The location of these two drill holes in the economic mineralized zone relative to the proposed pit outline are shown in Figure 12.1 and Figure 12.2. HD-01 comprised intervals from 23.85 m in depth through 200.6 m while HD-04 comprised intervals from 22.3 m in depth through 200.8 m.

The 25 discrete composite samples were prepared by combining individual intercepts of the two drill holes with a target total composite weight of 20 kg, thirteen composite from drill hole HD-01 and twelve composites from drill hole HD-04.

The two master composites were made by taking 2 kg from each of the 25 composites (Master HD-01 and Master HD-04)

The two main phases of G&T 2007metallurgical testing undertaken on the Halilağa samples were:

- Bench scale flotation testing, rougher and cleaner testing on master composites; and
- Bench scale flotation testing consisting of rougher flotation to map the variability on rougher metallurgical performance, supplemented with mineralogical examination.

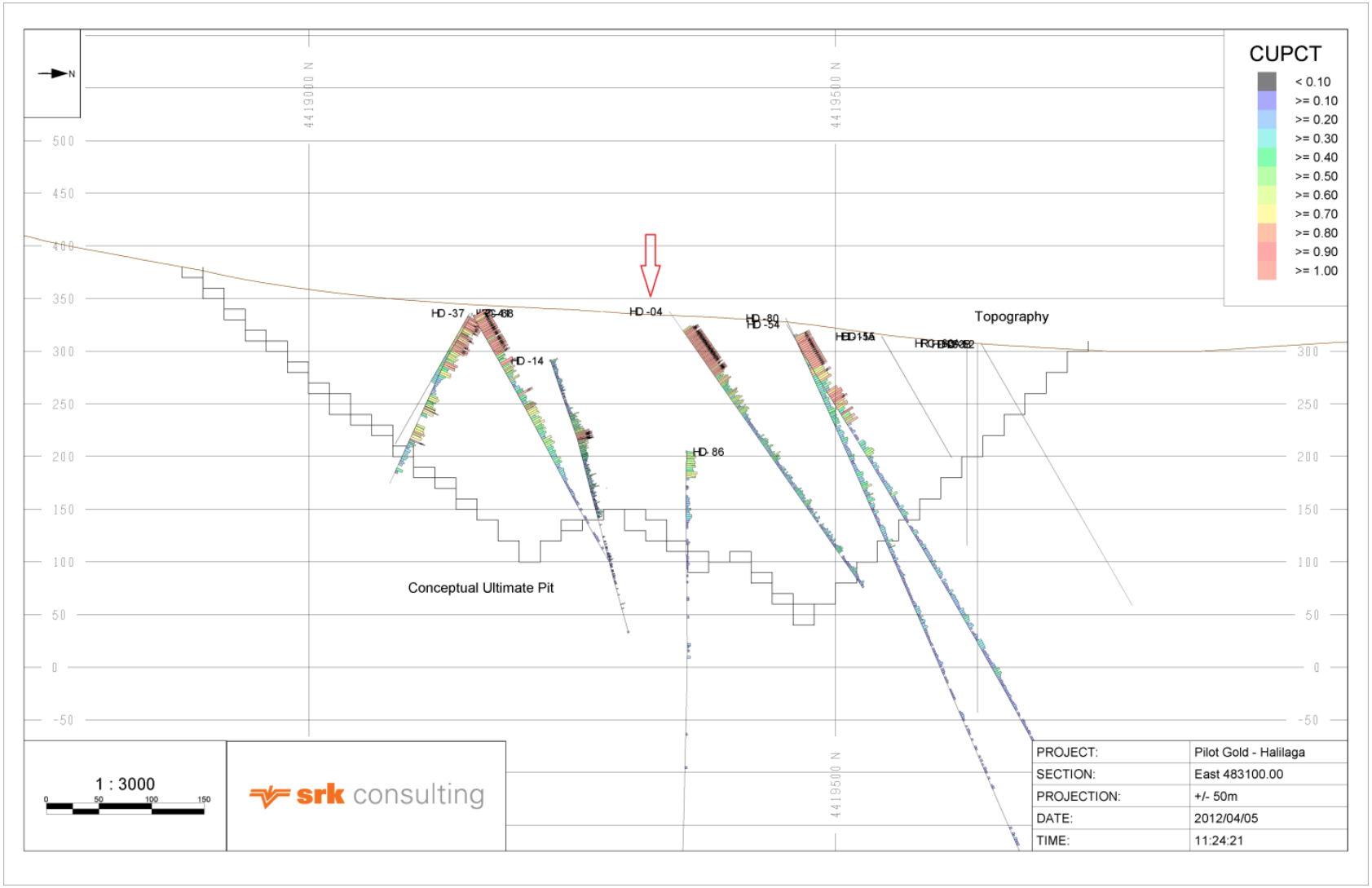



**Figure 12.1: Location of Drill Hole HD-04**

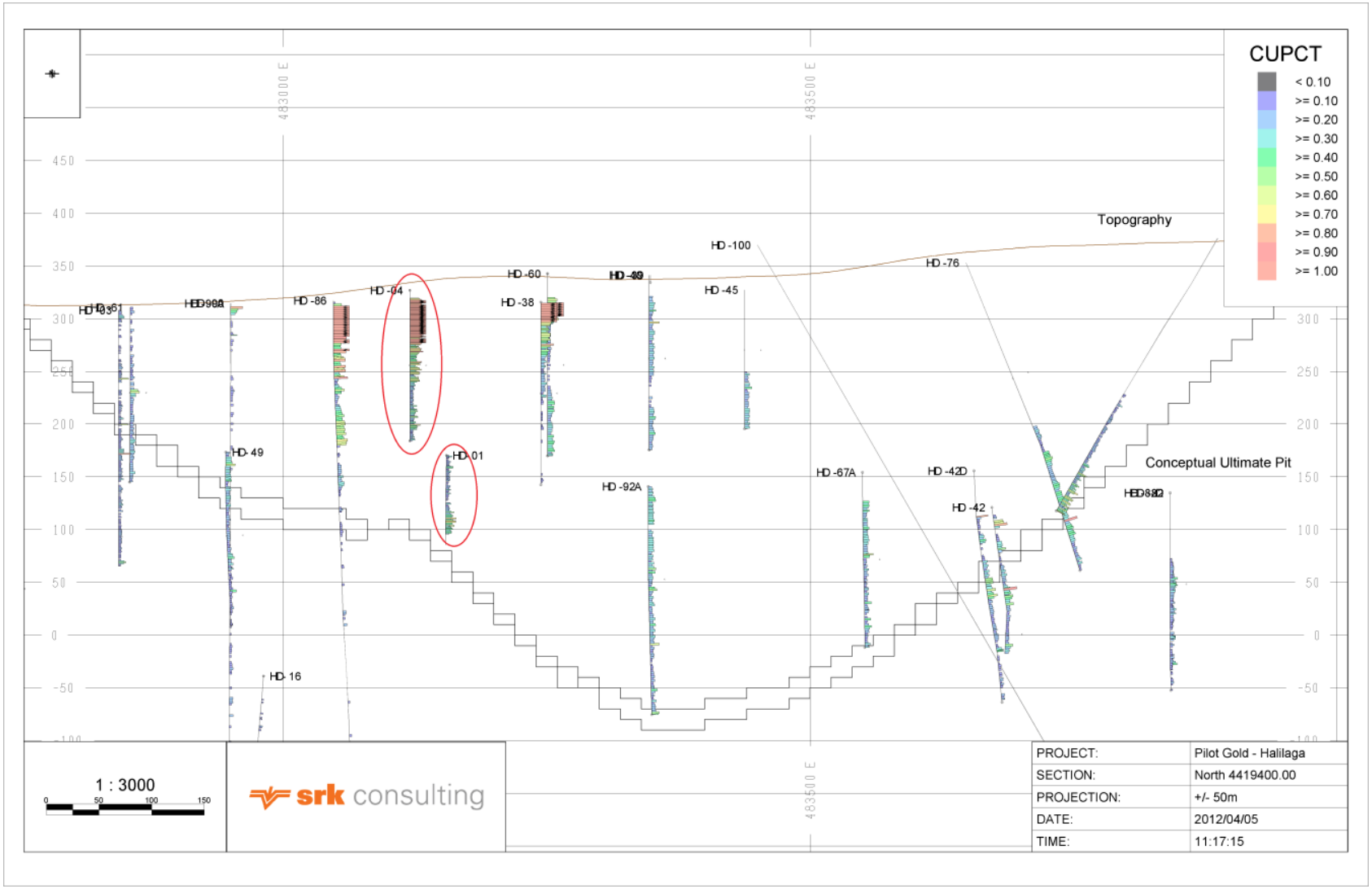



**Figure 12.2: Location of Drill Holes HD-01 and HD-04**

## Flotation Test Work Results

Flotation test work commenced with rougher and cleaner open circuit bench tests on the two major composites and rougher flotation on the 25 discrete composites. Copper sulphide flotation was undertaken to produce a copper concentrate. Analysis of the test work forms the basis of the plant process design criteria and estimates of concentrate copper grade, and copper and gold recovery.

The 2007 flotation test work program was divided into three phases:

1. Rougher flotation on the two master composites.
2. Open circuit cleaner flotation on the two major composites.
3. Rougher flotation on the twenty-five discrete composites.

## Rougher Flotation

Initially, a series of rougher kinetic flotation tests were conducted on the master composites to determine the sensitivity of the mineralized material types to grind size and reagent scheme. These tests were supplemented with mineralogical examination on the two master composites by Modal analysis to determine fundamental mineral liberation and mineral speciation.

These tests indicated the following:

- The sulphide minerals occur in order of abundance as pyrite, chalcopyrite and chalcocite/covellite;
- Chalcopyrite liberation appears to be relatively insensitive to grind size in the primary grind sizes between 100 µm and 200 µm;
- Overall copper sulphide liberation should be adequate to achieve good metallurgical performance in the rougher flotation circuit. A target primary grind size of 150 µm was established through rougher kinetic testing at primary grind size of 100 µm, 150 µm, and 200 µm
- Ore hardness increases with depth in the drill hole; and

The results from the rougher tests are summarized in Table 12.1.

These tests indicate that the mineralized material is amenable to flotation, resulting in good recovery of target mineral species into a low mass concentrate stream using potassium amyl xanthate (PAX), lime, and methyl isobutyl carbinol (MIBC).

**Table 12.1: Rougher Flotation results for Halilağa Master Composite Samples**

| Hole ID | Grind Size | Mass Recovery | Feed | | Concentrate Grade | | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | µm | % | % Cu | g/t Au | % Cu | g/t Au | % S | % Cu | % Au | % S |
| HD01 | 152 | 14.2 | 0.75 | 0.62 | 4.91 | 3.26 | 26.3 | 93.8 | 75.2 | 97.3 |
| HD01 | 152 | 18.5 | 0.73 | 0.66 | 3.83 | 2.91 | 19.9 | 96.6 | 81.6 | 98.5 |
| HD01 | 152 | 15.8 | 0.76 | 0.59 | 4.67 | 3.18 | 24.4 | 96.6 | 84.4 | 98.7 |
| HD01 | 94 | 14.2 | 0.76 | 0.64 | 5.18 | 3.82 | 28.5 | 96.5 | 84.0 | 98.1 |
| HD01 | 192 | 17.2 | 0.74 | 0.68 | 10.30 | 6.83 | 24.8 | 95.9 | 89.1 | 99.4 |
| HD04 | 151 | 11.2 | 0.65 | 0.46 | 4.19 | 2.65 | 15.2 | 71.5 | 64.4 | 54.3 |
| HD04 | 151 | 22.3 | 0.65 | 0.51 | 2.79 | 1.98 | 16.5 | 95.7 | 86.7 | 98.7 |
| HD04 | 151 | 15.5 | 0.64 | 0.53 | 4.01 | 2.87 | 0.11 | 96.2 | 84.0 | 97.5 |
| HD04 | 93 | 15.4 | 0.63 | 0.50 | 3.95 | 2.62 | 22.1 | 96.1 | 79.9 | 97.8 |
| HD04 | 186 | 14.9 | 0.63 | 0.46 | 3.98 | 2.58 | 22.7 | 94.3 | 83.4 | 96.4 |

## Cleaner Flotation

Cleaner flotation was conducted on selective master composite samples at a target primary grind size of 80% passing 150 µm and incorporated a rougher regrind stage to increase mineral liberation. Varying re-grind times, reagent dosages, and pH in cleaners were tested to determine optimum flotation conditions. The cleaner flotation reagent scheme was not changed from that tested in the rougher tests.

The fine grained structure of the mineralized material identified by the Modal Analysis and the increase in rougher grade and recovery with finer re-grind indicated that re-grinding of rougher concentrates would be required to achieve adequate concentrate grades. Concentrate regrinding was therefore incorporated in all subsequent cleaner testing. Multiple stages of cleaning were incorporated to target high concentrate grades, typically with an elevated pH level in the cleaning.

The results from optimization tests are summarized in Table 12.2 show the relationship of copper recovery versus grade and copper recovery versus gold recovery for both master composites.

The tests where rougher concentrate was reground to approximately 20 µm achieved the best copper grades at 31.1% Cu to 39.2% Cu and the best recoveries of 89.7% and 91.1% from master composite samples from HD-01 and HD-04 respectively. This is clearly illustrated in Figure 13.3 and Figure 13.4. Gold recoveries ranged from 53.5% to 71.3%.

**Table 12.2: Cleaner Flotation results for Halilağa Master Composite Samples**

| Hole ID | Grind Size | Mass Recovery | Feed | | Concentrate Grade | | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | µm | % | % Cu | g/t Au | % Cu | g/t Au | % S | % Cu | % Au | % S |
| HD01 | 55 | 3.0 | 0.73 | 0.60 | 17.5 | 11.3 | 39.9 | 72.5 | 57.2 | 27.0 |
| HD01 | 47 | 2.4 | 0.74 | 0.52 | 26.5 | 12.9 | 39.3 | 87.0 | 59.7 | 24.9 |
| HD01 | 22 | 1.8 | 0.77 | 0.52 | 39.2 | 18.84 | 32.2 | 91.1 | 65.0 | 14.8 |
| HD04 | 46 | 2.7 | 0.63 | 0.46 | 17.5 | 10.2 | 42.4 | 74.4 | 58.8 | 33.0 |
| HD04 | 40 | 1.7 | 0.63 | 0.46 | 26.4 | 13.54 | 41.2 | 69.3 | 53.5 | 19.3 |
| HD04 | 23 | 1.8 | 0.63 | 0.45 | 31.2 | 17.53 | 28.4 | 89.7 | 71.3 | 14.4 |

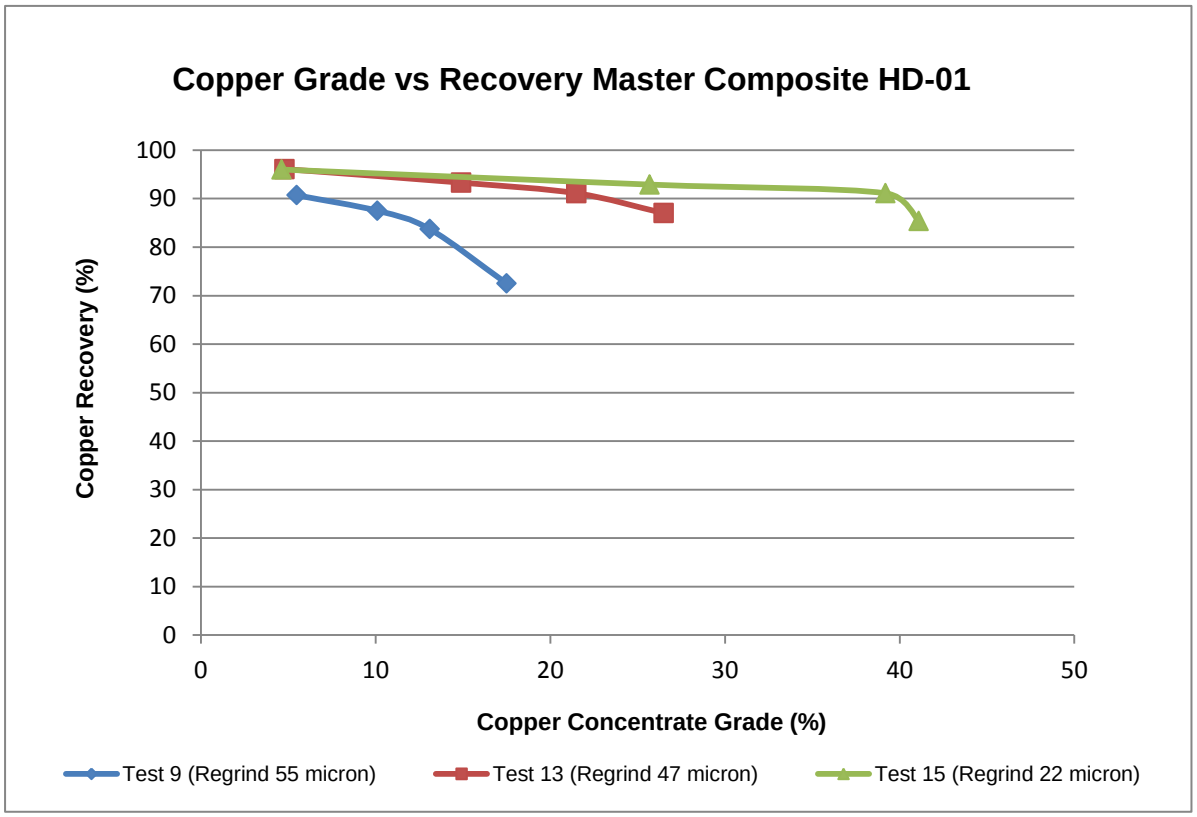



**Figure 12.3: Master Composite HD01 Copper Rougher Recovery versus Copper Grade**

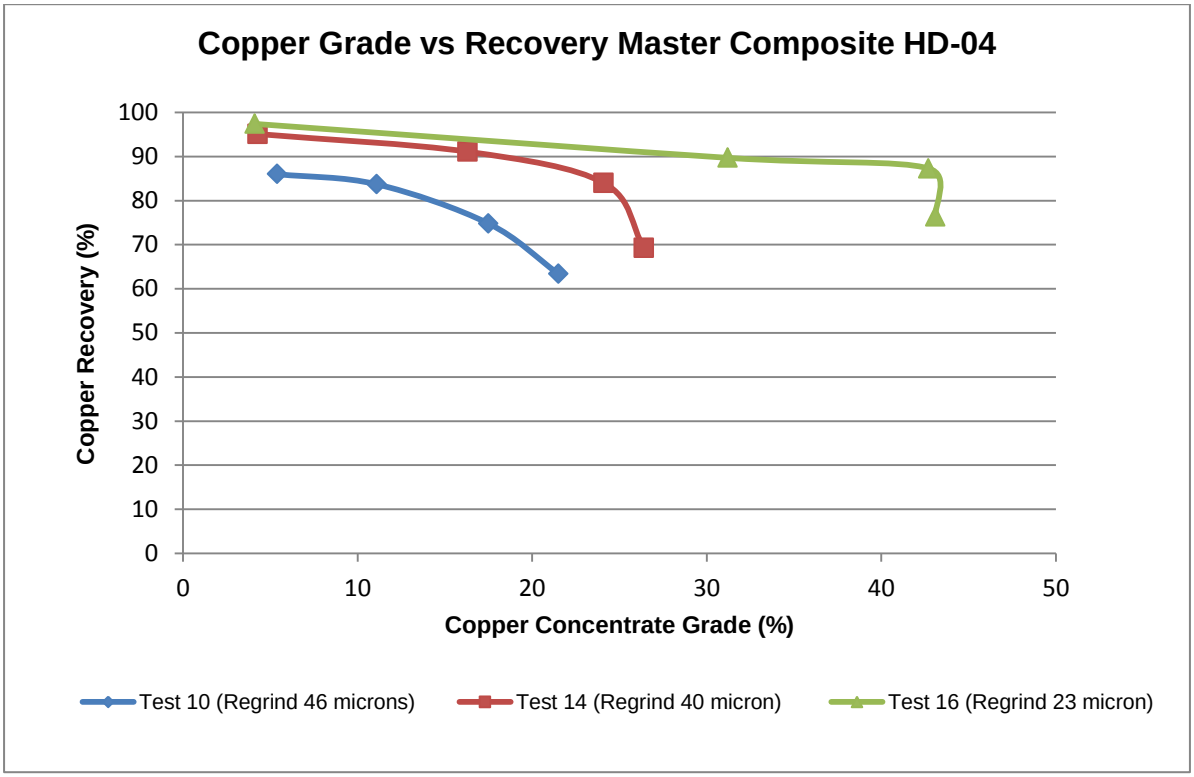



**Figure 12.4: Master Composite HD04 Cleaner Copper Recovery versus Cleaner Gold Recovery**

Twelve of the thirteen samples from drill hole HD-01 produced rougher concentrate recoveries between 93.8% and 97.6%.

Gold recovery to rougher concentrate for samples C2 to C13 ranged from 78.0% to 91.8%. Recovery and Grade for sample C1 were low; this is likely due to the location of the sample near the surface which may be oxidized.

Figure 12.5 shows the copper and gold recovery for each of the thirteen discrete composite samples from drill hole HD-01.

The twelve samples from drill hole HD-04 produced rougher concentrate recoveries between 92.8% and 97.1%. Gold recovery to rougher concentrate for samples C1 to C12 ranged from 76.5% to 92.6%.

Figure 12.6 shows the copper and gold recovery for each of the twelve discrete composite samples from drill hole HD-04.

## Variability Rougher Flotation Testing

Twenty five samples were subject to rougher flotation variability testing. The results are summarized in Table 12.3 and Table 12.4.

**Table 12.3: Variability Rougher Flotation Testing Discrete Composites Drill Hole HD-01**

| Comp | Grind Size | Mass Rec | Feed | | Concentrate Grade | | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | µm | % | % Cu | g/t Au | % Cu | g/t Au | % S | % Cu | % Au | % S |
| HD01C1 | 124 | 10.7 | 1.86 | 0.76 | 11.1 | 3.92 | 15.0 | 64.1 | 55.6 | 40.6 |
| HD01C2 | 134 | 7.6 | 2.40 | 0.90 | 9.06 | 2.90 | 24.8 | 97.4 | 83.5 | 99.4 |
| HD01C3 | 121 | 9.3 | 0.68 | 0.56 | 2.68 | 1.98 | 25.3 | 96.1 | 85.7 | 99.5 |
| HD01C4 | 132 | 6.8 | 1.03 | 1.08 | 5.34 | 4.92 | 22.7 | 97.6 | 85.7 | 99.3 |
| HD01C5 | 140 | 5.2 | 1.00 | 1.14 | 6.02 | 6.23 | 24.9 | 97.4 | 88.2 | 99.4 |
| HD01C6 | 152 | 4.1 | 0.97 | 1.01 | 4.65 | 4.35 | 28.5 | 94.6 | 85.3 | 98.6 |
| HD01C7 | 143 | 12.9 | 0.40 | 0.39 | 1.94 | 1.53 | 27.3 | 95.4 | 78.0 | 98.7 |
| HD01C8 | 136 | 4.1 | 0.25 | 0.24 | 20.9 | 1.36 | 24.8 | 93.8 | 78.9 | 97.6 |
| HD01C9 | 155 | 9.7 | 0.36 | 0.45 | 1.93 | 2.05 | 18.7 | 96.4 | 83.5 | 99.0 |
| HD01C10 | 131 | 7.8 | 0.38 | 0.38 | 2.09 | 1.89 | 17.8 | 96.7 | 86.9 | 100.0 |
| HD01C11 | 136 | 7.6 | 0.21 | 0.19 | 0.97 | 0.86 | 18.9 | 95.1 | 91.8 | 99.3 |
| HD01C12 | 136 | 6.12 | 0.20 | 0.21 | 1.20 | 1.03 | 14.6 | 95.8 | 79.5 | 99.1 |
| HD01C13 | 116 | 7.3 | 0.16 | 0.13 | 0.84 | 0.63 | 16.3 | 94.9 | 87.4 | 99.2 |

**Table 12.4: Variability Rougher Flotation Testing Discrete Composites Drill Hole HD-04**

| Comp | Grind Size | Mass Rec | Feed | | Concentrate Grade | | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | µm | % | % Cu | g/t Au | % Cu | g/t Au | % S | % Cu | % Au | % S |
| HD04C1 | 146 | 30.1 | 1.15 | 0.63 | 3.73 | 1.87 | 15.6 | 97.6 | 88.9 | 99.9 |
| HD04C2 | 149 | 23.9 | 1.85 | 1.44 | 7.50 | 5.59 | 22.3 | 97.2 | 92.6 | 99.5 |
| HD04C3 | 136 | 29.7 | 1.15 | 0.86 | 3.73 | 2.47 | 19.2 | 96.4 | 85.3 | 99.1 |
| HD04C4 | 133 | 19.7 | 0.72 | 0.81 | 3.49 | 3.50 | 24.1 | 95.9 | 85.1 | 98.7 |
| HD04C5 | 146 | 20.5 | 0.51 | 0.49 | 2.42 | 2.06 | 23.4 | 96.9 | 85.5 | 99.3 |
| HD04C6 | 150 | 19.6 | 0.51 | 0.45 | 2.52 | 1.91 | 16.0 | 97.1 | 83.8 | 99.2 |
| HD04C7 | 144 | 16.6 | 0.37 | 0.37 | 2.16 | 1.76 | 16.1 | 96.1 | 79.6 | 97.4 |
| HD04C8 | 139 | 13.4 | 0.19 | 0.22 | 1.30 | 1.26 | 17.2 | 94.4 | 76.5 | 96.9 |
| HD04C9 | 145 | 14.0 | 0.34 | 0.27 | 2.35 | 1.59 | 23.9 | 96.0 | 81.2 | 97.8 |
| HD04C10 | 146 | 16.5 | 0.35 | 0.21 | 2.00 | 1.10 | 14.5 | 94.5 | 87.9 | 98.4 |
| HD04C11 | 110 | 18.2 | 0.21 | 0.13 | 1.10 | 0.56 | 10.4 | 94.2 | 80.6 | 98.5 |
| HD04C12 | 111 | 15.6 | 0.25 | 0.16 | 1.46 | 0.82 | 11.3 | 92.8 | 79.1 | 97.1 |

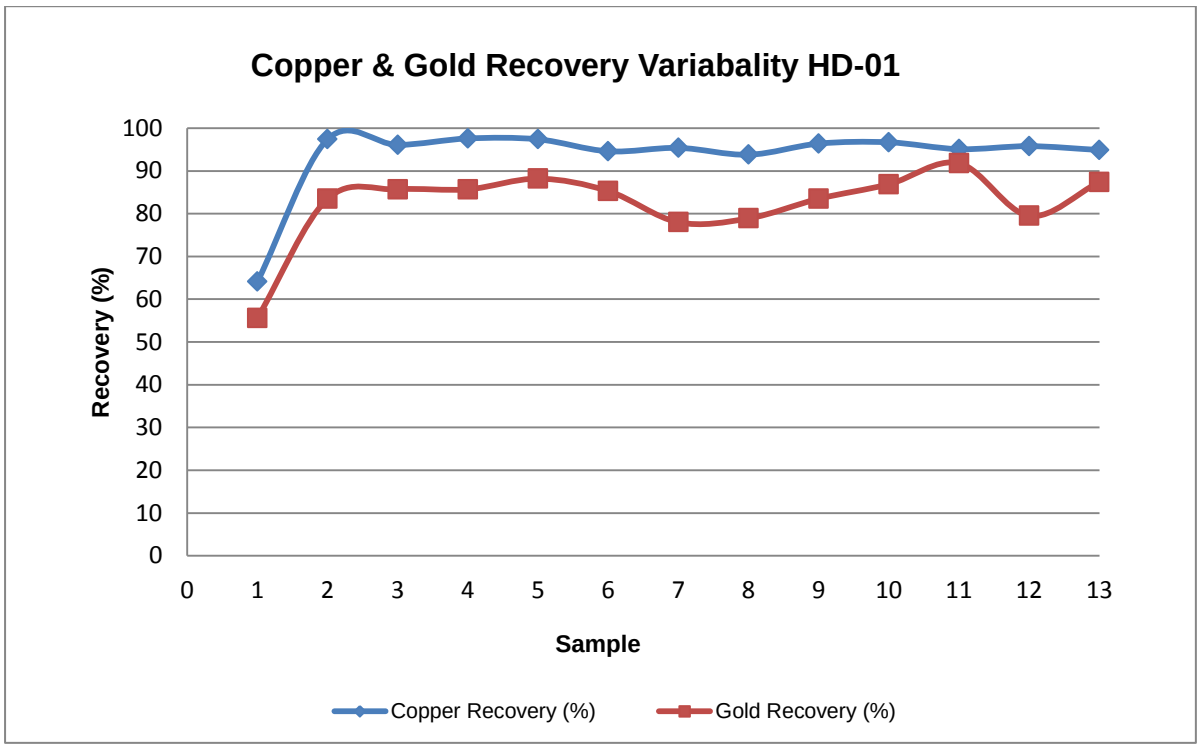



**Figure 12.5: Copper and Gold Rougher Recovery Variability HD-01**

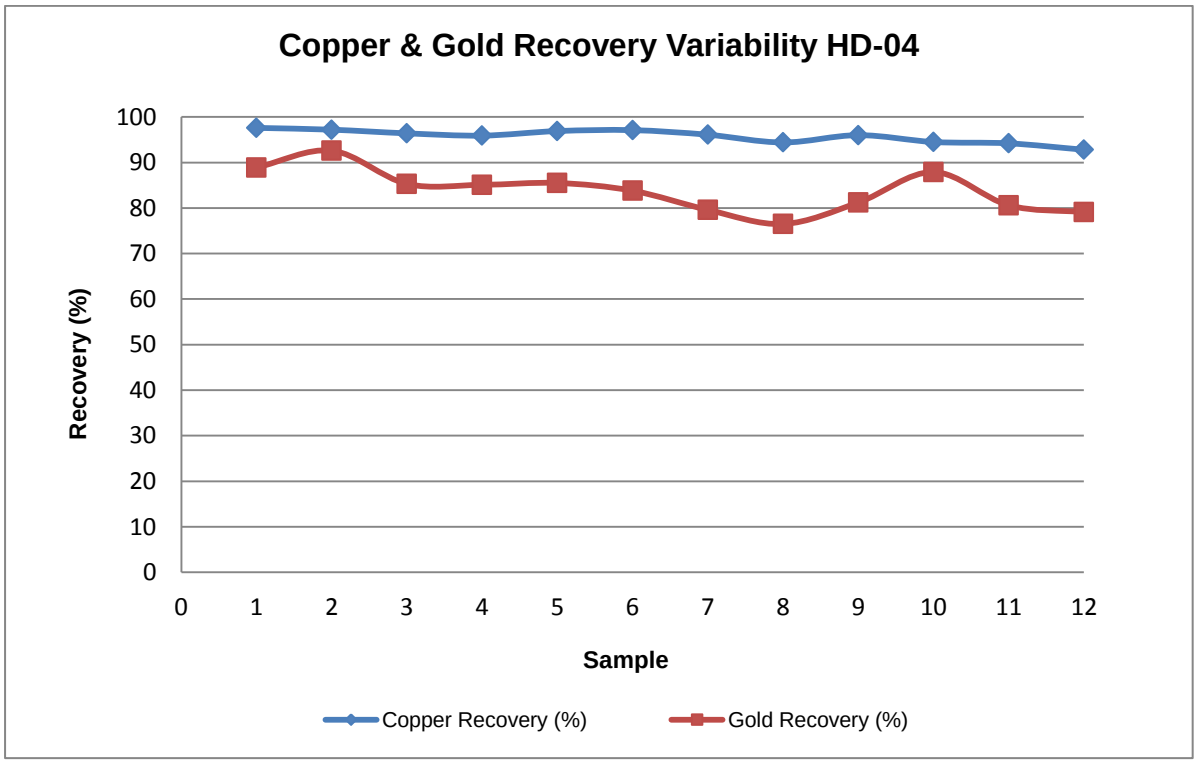



**Figure 12.6: Copper and Gold Rougher Recovery Variability HD-04**

## July 2011 Test Program

The second test program conducted in 2011 was based on two shipments of samples. The first shipment contained 203 samples weighing a total of 1.2 tonnes. The second shipment contained 115 samples weighing a total of 672 kg. A total of 15 composites were made using the received samples. The samples were collected from drill holes HD-40 and HD-49.

Figure 12.7 and

Figure 12.8 show the location of these drill holes in the pit zone.

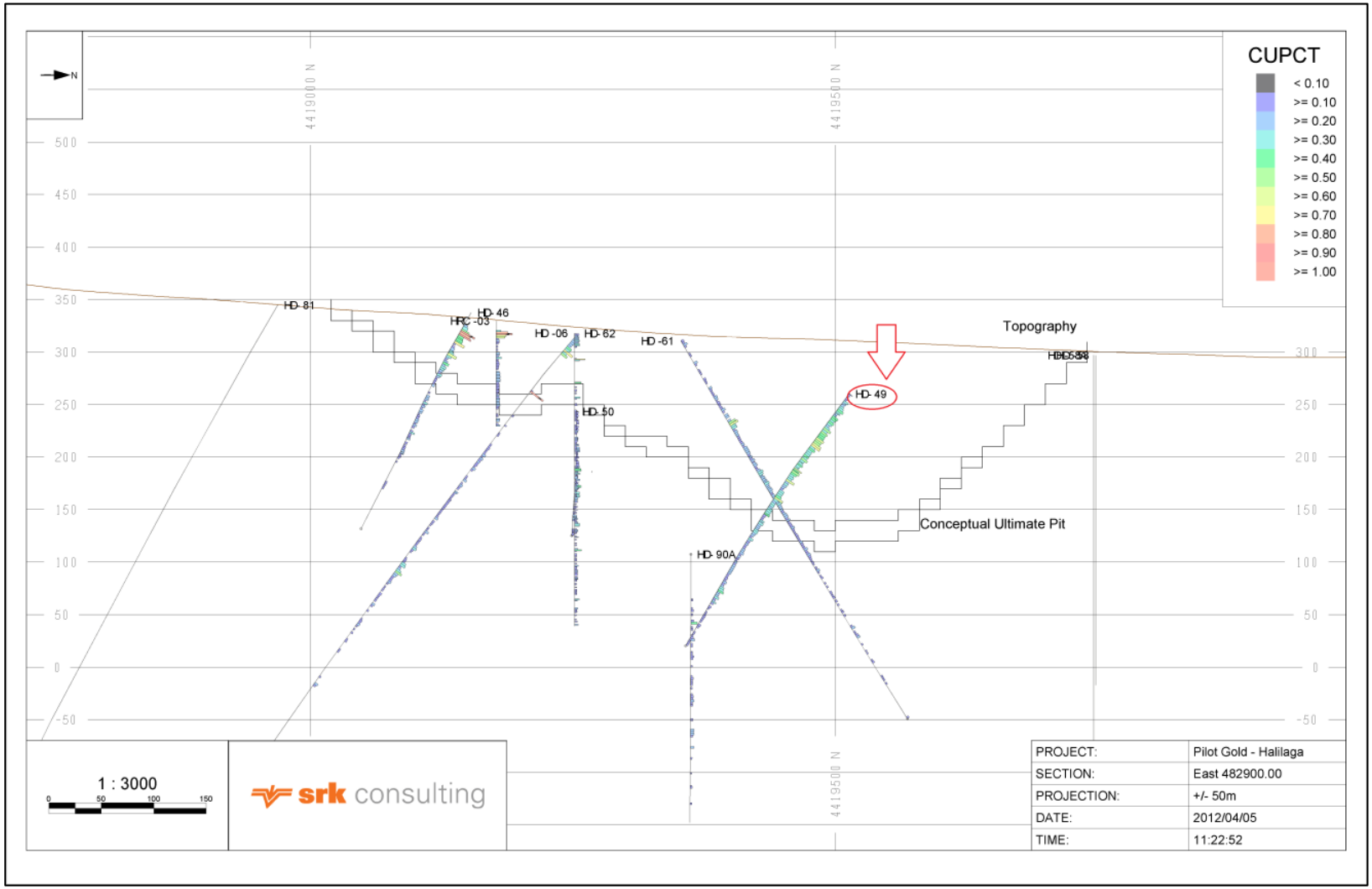



**Figure 12.7: Location of Drill Hole HD-49**

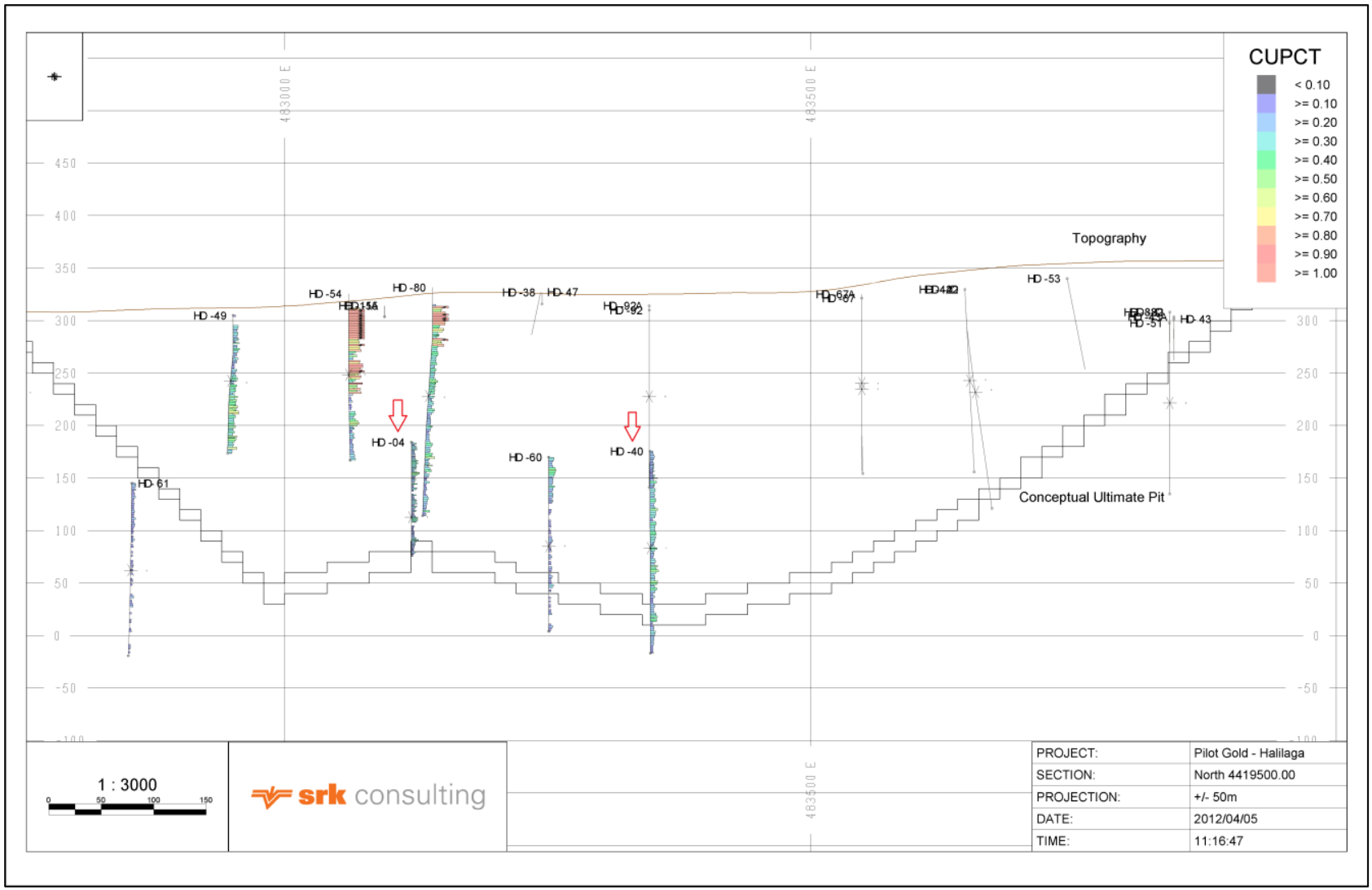



**Figure 12.8: Location of Drill Hole HD-40**

A global composite designated “Typical Ore” was made by taking 4 kg from each Composite 1, 3, 8, and 9 for a total 16 kg sample.

Two main phases of metallurgical testing were undertaken on the Halilağa mineralized samples:

1. Open circuit cleaner flotation testing.
2. Determine the mineralized material hardness of 6 of the 15 variability composites.

### 12.2.2 Comminution Test Work

The current comminution dataset consists of six Bond ball mill work index tests, one test each for six of the 15 plant feed composites.

Table 12.5 summarizes the outcomes of the comminution laboratory test work undertaken for this study, as well as the selected design case. To date no Drop Weight Index Testing (either by the JK Drop Weight Test or SAG Media Competency Test), Bond Crushing Index, Bond Rod Mill Index or Bond Abrasion Index testing has been undertaken. Values for these metrics have been estimated from the available data or from typical values for similar mineralized material.

**Table 12.5: Parameters for Comminution Design**

| Characteristic | Units | Value |
|---|---|---|
| Rod mill work index | kWh/t | 12.3 |
| Crusher work index | kWh/t | 12.3 |
| DWi | kWh/t | 4.7 |
| Axb | | 60 |

**Table 12.6: Summary of Comminution Test Work Data**

| Characteristic | Units | Comp 1 | Comp 2 | Comp 3 | Comp 4 | Comp 5 | Comp 6 | Design |
|---|---|---|---|---|---|---|---|---|
| Ball mill work index | kWh/t | 12.4 | 9.2 | 12.2 | 11.7 | 11.3 | 12.7 | 12.3 |

The design comminution characteristics are selected to ensure that the majority of the mineralized material can be treated at the nominated design rate. (Table 12.6)This is typically achieved by selecting the 75th percentile comminution characteristics as the basis for comminution circuit design. This is based on the assumption that the hardest 25% of mineralized rock samples tested can be blended with softer material during normal mining and stockpile operations, allowing the plant to achieve the nameplate capacity at all times.

The 75th percentile has been used as the design basis for Halilağa. This premise would need to be tested in the next phase of study as more detailed mine schedule information and mineralized rock comminution characteristics become available.

### 12.2.3 Flotation Test Work Results

Flotation test work included open circuit cleaner flotation testing and culminated in two locked cycle tests of the “Typical Ore” composite to produce a copper concentrate. The test work was used to develop the plant process design criteria and estimates of concentrate grade, copper, and gold recovery. Work carried out to date is sufficient to support the conceptual level design and costing.

The Conceptual Study flotation test work program was divided into two phases:

1. Open circuit cleaner flotation; and
2. Locked cycle flotation.

## Open Circuit Cleaner Flotation

Cleaner flotation was conducted at a target primary grind size of 80% passing 150 µm and incorporated a rougher regrind stage to a target grind size of 80% passing 20 µm to increase mineral liberation.

Multiple stages of cleaning were incorporated to target high concentrate grades, typically with an elevated pH level in the cleaning. The results from selected optimization tests are summarized in Table 12.7.

**Table 12.7: Cleaner Flotation results for Halilağa Composite Samples**

| Ore Type | Re-Grind Size | Mass Rec | Feed | | Concentrate Grade | | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | µm | % | % Cu | g/t Au | % Cu | g/t Au | % S | % Cu | % Au | % S |
| Comp 1 | 21 | 0.7 | 0.30 | 0.18 | 33.2 | 11.4 | 33.8 | 80.4 | 44.3 | 13.5 |
| Comp 2 | 21 | 0.6 | 0.21 | 0.20 | 33.4 | 15.8 | 32.2 | 76.6 | 38.5 | 13.8 |
| Comp 3 | 28 | 0.8 | 0.27 | 0.13 | 27.7 | 10.7 | 32.0 | 79.5 | 62.5 | 15.5 |
| Comp 3 | 17 | 0.8 | 0.29 | 0.15 | 32.0 | 13.4 | 30.9 | 83.5 | 68.0 | 14.3 |
| Comp 4 | 23 | 0.7 | 0.27 | 0.24 | 33.4 | 17.6 | 33.4 | 80.8 | 47.8 | 13.9 |
| Comp 5 | 24 | 0.8 | 0.26 | 0.13 | 32.0 | 11.3 | 32.2 | 80.6 | 58.0 | 15.1 |
| Comp 6 | 40 | 2.2 | 0.16 | 0.06 | 9.90 | 2.94 | 12.0 | 69.2 | 52.5 | 4.8 |
| Comp 7 | 29 | 1.0 | 0.18 | 0.06 | 21.9 | 5.75 | 22.3 | 69.3 | 50.6 | 3.5 |
| Comp 8 | 17 | 1.6 | 0.27 | 0.19 | 32.9 | 18.9 | 33.2 | 75.5 | 64.2 | 14.9 |
| Comp 9 | 24 | 0.9 | 0.36 | 0.33 | 30.0 | 19.6 | 36.7 | 77.2 | 55.1 | 21.9 |
| Comp 10 | 19 | 1.4 | 0.47 | 0.39 | 30.9 | 19.9 | 34.0 | 78.0 | 60.8 | 25.9 |
| Comp 11 | 18 | 1.2 | 0.40 | 0.30 | 32.5 | 22.6 | 32.7 | 80.2 | 72.9 | 31.5 |
| Comp 12 | 18 | 1.1 | 0.44 | 0.47 | 33.4 | 22.2 | 33.6 | 79.9 | 50.2 | 28.4 |
| Comp 13 | 20 | 0.9 | 0.35 | 0.20 | 32.8 | 16.2 | 34.3 | 74.6 | 65.8 | 19.3 |
| Comp 14 | 20 | 0.7 | 0.21 | 0.12 | 27.7 | 13.5 | 36.6 | 76.6 | 64.5 | 14.8 |
| Comp 15 | 24 | 0.8 | 0.20 | 0.13 | 26.3 | 13.2 | 35.9 | 75.1 | 59.4 | 13.0 |

Copper recovery ranged from 74.6% to 83.5% (excluding composites 6 and 7) with a copper grade ranging from 26.3% Cu to 33.4% Cu. The low values for composites 6 and 7 are mainly attributed to the coarser re-grind of 29 µm and 40 µm.

Figure 12.9 shows the copper and gold recovery for the composite samples.

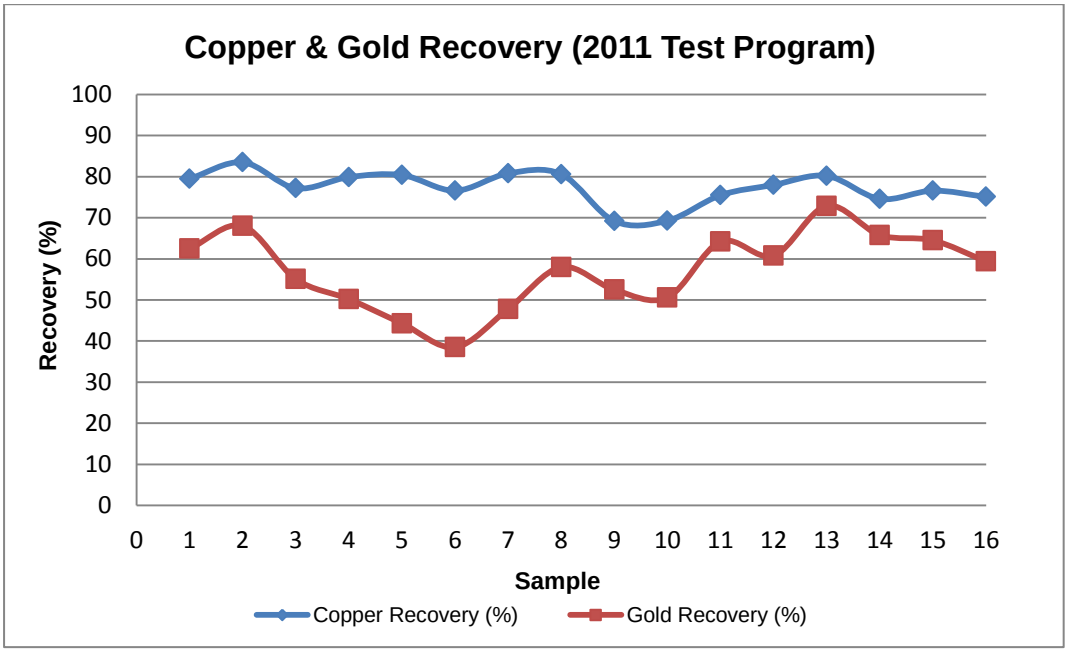



**Figure 12.9: Cleaner Copper and Gold Recovery for Composite Samples**

## Locked Cycle Test Work

Following the completion of the open circuit cleaner flotation test work phase, two locked cycle tests were conducted on the “Typical Ore” composite. This phase looked at testing the optimized flow sheet conditions in a locked cycle test to determine the closed circuit grade recovery performance of each of the mineralized material types for project evaluation.

Flotation results from the optimization test work highlighted the benefit of fine regrinding and multiple stages of concentrate cleaning on improving concentrate grade. A flow sheet incorporating rougher concentrate re-grinding and multiple stages of cleaning, similar to that from the open circuit cleaner testing was selected. To test the flow sheet performance two locked cycle tests were conducted.

Locked cycle tests are used to determine the effects of recycling intermediate streams, such as the cleaner tailings, on the overall grade recovery performance of the mineralization type. By retaining these streams and combining them with concentrates from a subsequent flotation test, an assessment can be made of the overall performance of a full scale plant operation.

Two locked cycle tests were undertaken for the “Typical Ore” composite, the results are summarized in Table 12.8.

**Table 12.8: Locked Cycle Test Results**

| Test No. | Product | Re-Grind Size | Mass Recovery | Concentrate Grade | | | Recovery | | |
|---|---|---|---|---|---|---|---|---|---|
| | | µm | % | % Cu | g/t Au | % S | % Cu | % Au | % S |
| KM2908-17 | Flotation Feed | 20 | 100.0 | 0.30 | 0.30 | 1.64 | 100 | 100 | 100 |
| | Concentrate | | 0.9 | 25.4 | 15.5 | 40.1 | 80.7 | 49.2 | 23.2 |
| | 1st Clean Tails | | 8.8 | 0.41 | 0.84 | 13.3 | 12.0 | 25.0 | 72.0 |
| | Rougher Tails | | 90.2 | 0.02 | 0.08 | 0.09 | 7.3 | 25.8 | 4.9 |
| KM2908-18 | Flotation Feed | 16 | 100.0 | 0.31 | 0.30 | 1.61 | 100 | 100 | 100 |
| | Concentrate | | 0.9 | 29.6 | 20.6 | 31.7 | 85.4 | 60.81 | 17.6 |
| | 1st Clean Tails | | 9.3 | 0.27 | 0.46 | 13.4 | 8.1 | 14.0 | 77.6 |
| | Rougher Tails | | 89.8 | 0.02 | 0.09 | 0.09 | 6.5 | 25.2 | 4.8 |

Analysis of these results indicate that recoveries of target minerals are acceptable and are generally in line with those achieved in the open circuit cleaner testing.

## 12.3 Grade and Recovery Predictions

Analysis of the locked cycle tests have been undertaken to determine flotation performance predictions. Locked cycle test KM2908-18 was selected as the basis for the design recoveries of the target metals copper and gold, the results are summarized in

Figure 12.10.

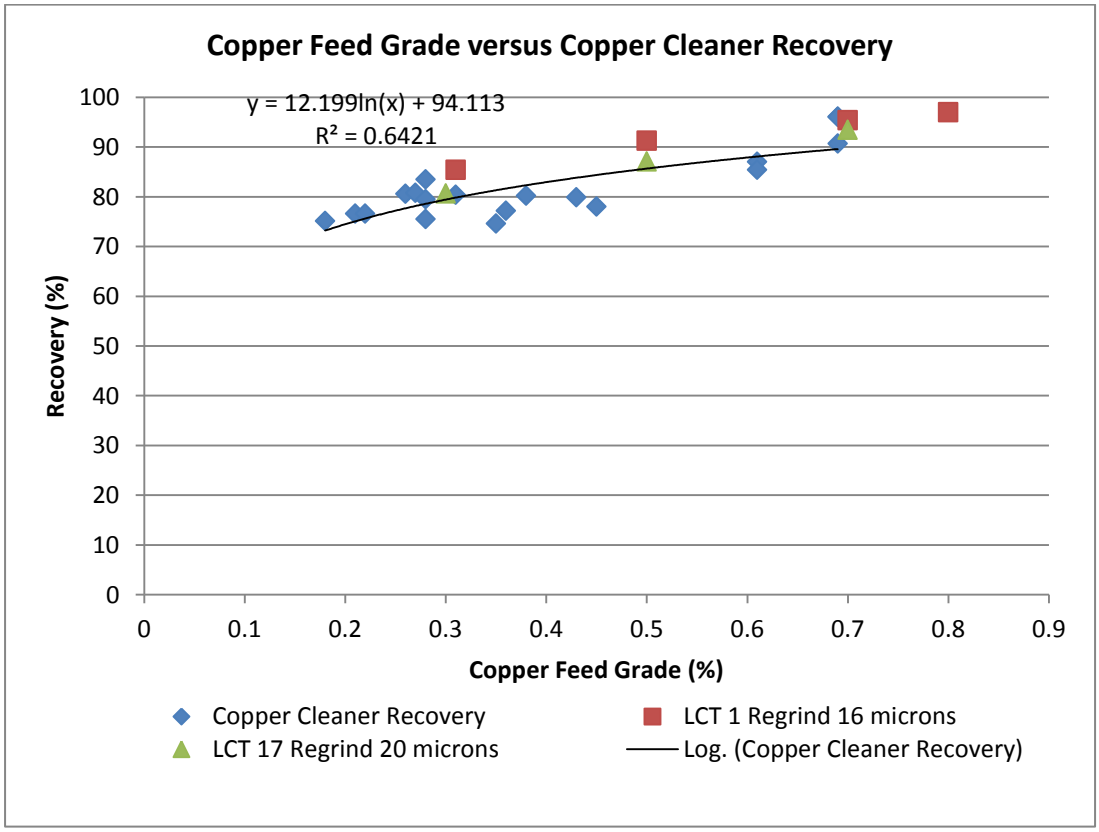


**Figure 12.10: Copper Feed Grade versus Copper Cleaner Recovery**

Copper recovery versus copper feed grade has been plotted for all the cleaner tests in the two test programs in

Figure 12.10. An equation for copper recovery based on feed grade has been established for the purposes of modeling the recovery to copper feed grade in the life of mine plan as higher grades are expected in the first few years of production. The equation established for the open circuit cleaner tests has then been modified to account for the improvement seen with recycling streams in locked cycle testing. As a result of recycling streams in the locked cycle tests the recovery increased by approximately 6% compared to the equivalent batch cleaner test. The equation has been modified by increasing the intercept value of the equation from 94.1 to 99.7:

Cu Recovery % = 12.2 Ln (%Cu in Feed) + 99.7.

Ausenco has reviewed the specified recoveries and believes that they are reasonable for a copper concentrate from the Halilaga mineralization types.

In the next phase of the project more locked cycled tests are required on samples from different zones of the mineralized material to confirm the design recoveries.

**Table 12.9: Grade/Recovery Predictions for Typical Halilağa Mineralized Material**

| Ore Type | Feed Grade | | Concentrate Grade | | Concentrate Recovery | | |
|---|---|---|---|---|---|---|---|
| | % Cu | g/t Au | % Cu | g/t Au | % Cu | % Au | % S |
| “Typical Ore” Composite | 0.29 | 0.30 | 29.6 | 20.6 | 85.6 | 60.8 | 17.6 |

## 12.3.1 Concentrate Quality

The copper concentrate from locked cycle test 18 was subjected to a detailed chemical analysis, the results are presented in Table 12.10. The copper concentrate was free of elements that might incur smelter penalties.

**Table 12.10: Quality of Concentrate**

| Element | Units | Locked Cycle Test 18 Concentrate |
|---|---|---|
| Aluminum | % | 0.27 |
| Antimony | % | 0.04 |
| Arsenic | g/t | 163 |
| Bismuth | g/t | 26 |
| Cadmium | g/t | 20 |
| Calcium | % | 0.21 |
| Carbon | % | 0.13 |
| Cobalt | g/t | 72 |
| Copper | % | 30.6 |
| Fluorine | g/t | 51.0 |
| Gold | g/t | 19.7 |
| Iron | % | 29.2 |

| Element | Units | Locked Cycle Test 18 Concentrate |
|---|---|---|
| Lead | % | 0.06 |
| Magnesium | % | 0.08 |
| Manganese | % | <0.001 |
| Mercury | g/t | 3 |
| Molybdenum | % | 0.17 |
| Nickel | g/t | 92 |
| Phosphorus | g/t | 50 |
| Selenium | g/t | 80 |
| Silicon | % | 1.13 |
| Sulphur | % | 32.1 |
| Silver | g/t | 50.6 |
| Zinc | % | 0.10 |

## 12.4 Further Test Work Required

For a PFS, Ausenco would recommend conducting flotation variability test work across the mineralized zones and use this to develop preliminary grade/recovery models that take into account variations in head grade.

Further work will be required for a PFS in order to provide more confidence in the design and generate more detailed process design criteria.

Recommended additional work identified as part of the Scoping Study includes the following:

- Test samples that provide greater spatial representivity of the mine and which represent early years of the mine plan;
- Comminution test work, including JK drop weight, SMC, Bond rod mill work index, Bond Crusher work index, Bond ball mill work index, Bond abrasion index;
- Comminution variability testing to determine the validity of the assumptions used for the comminution design;
- Review of comminution circuit selection and design by an external consultant;
- Additional reagent and flotation optimization flow sheet testing to confirm process design criteria and potentially improve metallurgical performance;
- Flotation variability and lock cycle testing to confirm metallurgical modeling and regrind size;
- Test work to confirm concentrate thickening rates for concentrate thickener selection;
- Test work to confirm tailings thickening rates for tailings thickener selection;
- Test work to confirm concentrate filtration rates and optimise filter selection;
- Rheology test work to confirm tailings pumping, pipeline and distribution design;
- Bulk materials handling test work to optimize design of the ROM bin, crushed plant feed material stockpile and reclaim facility; and

- Confirmation of geotechnical conditions for engineering design purposes in the plant and TMF locations, particularly in the locations of heavy structures.

# 13 Mineral Resource Estimates

## 13.1 Introduction

This resource is the first reported for the Halilağa Project. Porphyry deposits are generally well suited to geostatistical estimation (ordinary kriging); however, current drill spacing at Halilağa is not sufficiently dense to allow robust variogram modeling and a geometric estimation approach has been chosen. The estimation approach, described here, is suited to this early stage project.

Improved future resource estimates will require additional drill data.

## 13.2 Available Data

This initial resource estimate for the Halilağa Project is based on assay data available as of October 17, 2011. Results from 72 holes have been used for this estimate, of these 69 were core holes and three were RC. Figure 13.1 shows drill hole locations as well as the limits of the resource model. The block model geometry is listed in Table 13.1.

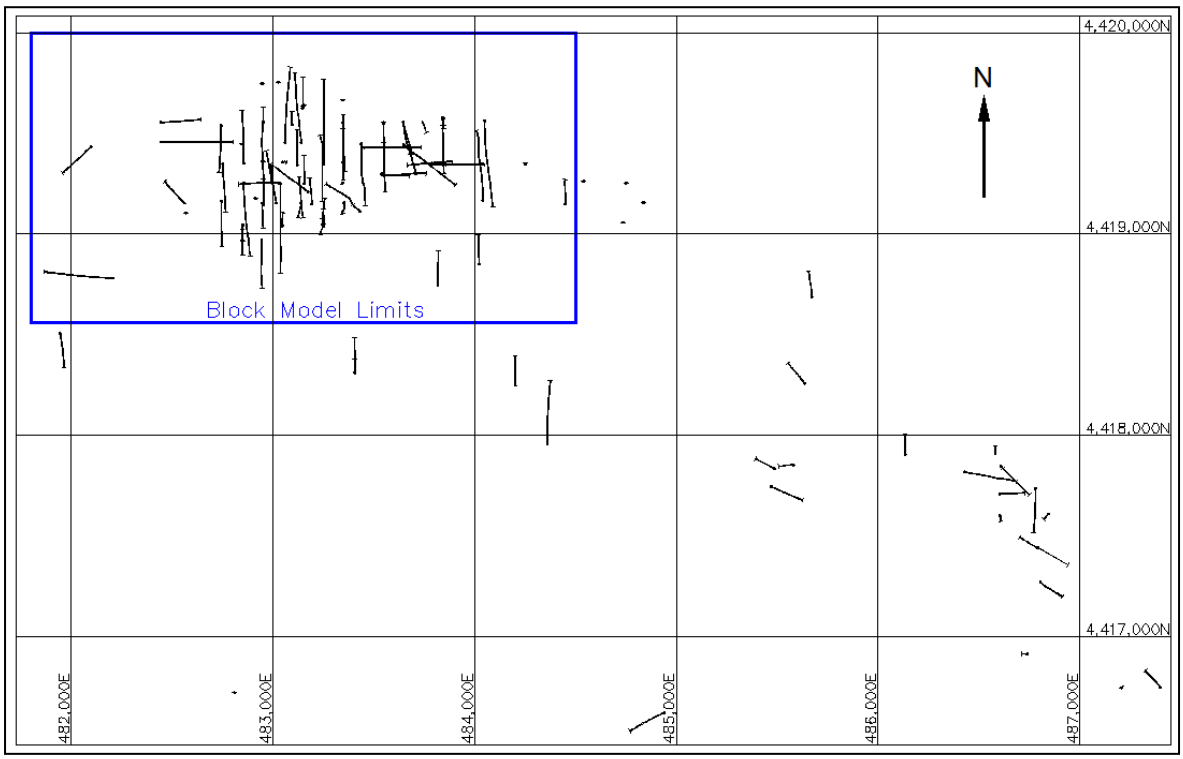



**Figure 13.1 Halilağa Exploration Drilling and Resource Model Outline**

**Table 13.1: Resource Block Model Setup**

| Block: | X | Y | Z |
|---|---|---|---|
| **origin**[(1)] | 481,800 | 4,418,560 | 600 |
| **size (m)** | 20 | 20 | 10 |
| ***n*blk** | 135 | 72 | 105 |

Non-rotated
1,020,600 blocks
(1) SW model top, block edge

## 13.3 Geologic Model

Truva Bakir project geologists have prepared a series of N-S sections generally spaced at 100 m intervals across the deposit. Four lithologic units were interpreted and constrained within 3-D solids honouring drill hole contacts. These reflect the porphyry intrusion into felsic volcanic rock to the south, a barren cover unit consisting of a mix of loose un-mineralized andesitic/basaltic volcanics and sediments to the north, and a hornfels unit that variably surrounds the porphyry. The northern side of the porphyry body is interpreted to be in faulted contact with overlying cover material.

Contacts of the oxide and supergene volumes were based primarily on soluble copper assays; nineteen holes penetrate these units. The top of the supergene zone (bottom of oxide zone) was marked down-hole by an abrupt increase in total and cyanide soluble Cu grade. A pronounced reduction in cyanide soluble to total copper assay was used to delineate the bottom of the supergene enriched zone.

Overburden was not modeled separately. Its thickness is generally low (< 5 m) and overburden is often not differentiated from the cover unit in logs. Rock type assignment to individual blocks was on a majority rule basis as this was determined to adequately reproduce solid volumes. An example section is shown in Figure 13.2.

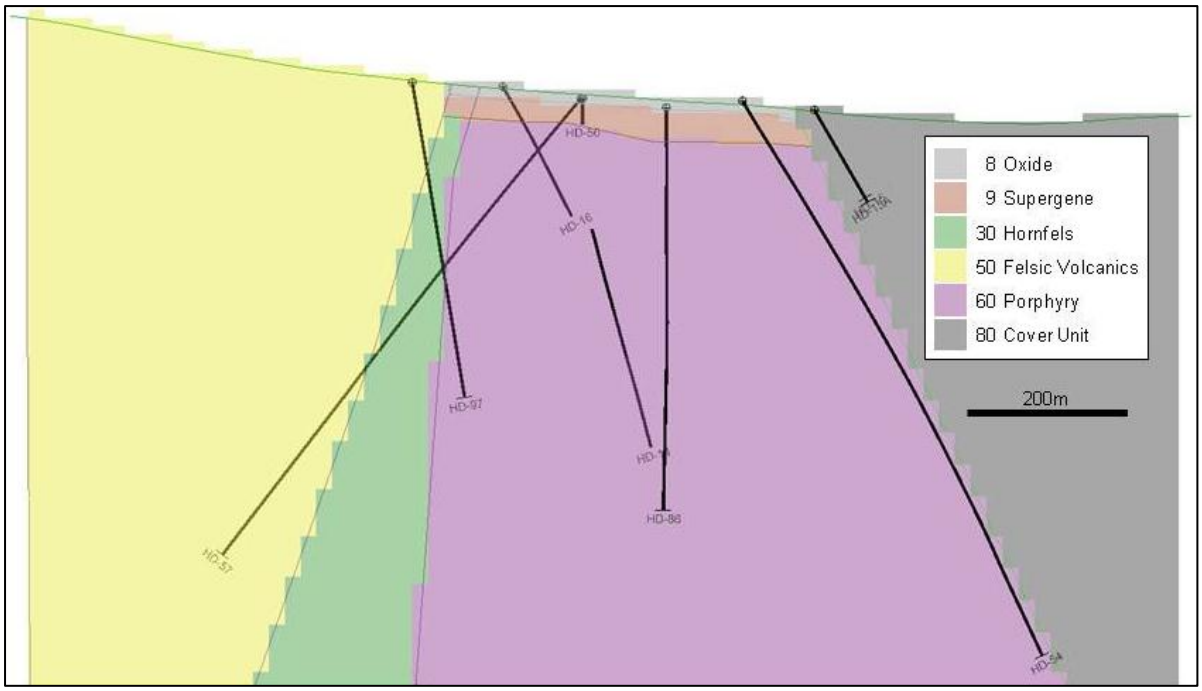



**Figure 13.2: Sectional Interpretation 483,050E (Westward)**

## 13.4 Assay Compositing

Sample data was composited to a down-hole length of 2.0 m. The choice of composite length was based primarily on its relation to the length of samples assayed. Sixty-seven percent of samples were either one (17%) or two (50%) m in length.

A total of 27 composites of less than half length (1.0 m) were removed from the dataset used for grade estimation, only after it was determined that this did not fundamentally affect the grade

statistics by rock type. Table 13.2 compares assay and composite statistics for the modeled rock units. The length-weighted mean grades of assays compare closely to the mean composite grades.

**Table 13.2: Sample and Composite Grades**

| **Assays** | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **Rock Type** | | **Length** | **Cu (%)** | | **Au (g/t)** | | **Mo (%)** | |
| | | **(metres)** | **Mean** | **lw-Mean[(1)]** | **Mean** | **lw-Mean** | **Mean** | **lw-Mean** |
| 8 | Oxide | 227 | 0.09 | 0.09 | 0.61 | 0.57 | 0.0047 | 0.0044 |
| 9 | Supergene | 461 | 1.26 | 1.20 | 0.66 | 0.62 | 0.0030 | 0.0028 |
| 30 | Hornfels | 6,313 | 0.08 | 0.07 | 0.10 | 0.08 | 0.0059 | 0.0047 |
| 50 | Felsic Volcanics | 2,328 | 0.02 | 0.01 | 0.05 | 0.03 | 0.0014 | 0.0011 |
| 60 | Porphyry | 14,432 | 0.21 | 0.20 | 0.23 | 0.21 | 0.0047 | 0.0048 |
| 80 | Cover | 741 | 0.01 | 0.00 | 0.01 | 0.00 | 0.0003 | 0.0001 |
| **Total** | | **24,502** | **0.17** | **0.14** | **0.19** | **0.14** | **0.0045** | **0.0038** |
| [(1)] length weighted and including unsampled intervals (4,874m in total) at zero grade | | | | | | | | |

| **2 m Composites** | | | | | |
|---|---|---|---|---|---|
| **Rock Type** | | **Length** | **Cu (%)** | **Au (g/t)** | **Mo (%)** |
| | | **(metres)** | **Mean** | **Mean** | **Mean** |
| 8 | Oxide | 232 | 0.10 | 0.58 | 0.0046 |
| 9 | Supergene | 466 | 1.19 | 0.61 | 0.0027 |
| 30 | Hornfels | 7,750 | 0.07 | 0.08 | 0.0047 |
| 50 | Felsic Volcanics | 3,282 | 0.01 | 0.03 | 0.0011 |
| 60 | Porphyry | 14,550 | 0.20 | 0.21 | 0.0048 |
| 80 | Cover | 3,096 | 0.00 | 0.00 | 0.0001 |
| **Total** | | **29,376** | **0.14** | **0.14** | **0.0038** |

## 13.5 Grade Capping

Grade capping is used to control the impact of extreme, outlier high-grade samples on the overall resource estimate. The levels selected as being outliers to the general population were determined by examining histograms and probability plots by rock type and by metal. Copper, gold, and molybdenum grades in the mineralized units generally display a log-normal distribution with very little scatter at the highest portion of the distribution. For copper and gold, the majority of composites that required grade capping were outside the main mineralized porphyry unit.

Capping levels are listed in Table 13.3. Capped versus uncapped composite statistics are presented in Table 13.4.

**Table 13.3: Grade Capping Levels**

| Rock Type | | Cu (%) | Au (g/t) | Mo (%) |
|---|---|---|---|---|
| 8 | Oxide | 0.33 | 1.40 | 0.016 |
| 9 | Supergene | 3.40 | 1.40 | 0.014 |
| 30 | Hornfels | 0.48 | 1.20 | 0.090 |
| 50 | Felsic Volcanics | 0.12 | 0.41 | 0.021 |
| 60 | Porphyry | 1.40 | 2.00 | 0.090 |
| 80 | Cover | 0.01 | 0.09 | 0.002 |

The amount of metal removed through the top-cutting process was calculated by comparing capped and uncapped nearest neighbour models. In total, metal removed by capping composite grades amounted to 0.8% copper, 1.2% gold, and 1.0% molybdenum. The relatively low impact of grade capping is typical of porphyry-style mineralization.

**Table 13.4: Composite Statistics**

| Cap Impact | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Rock Type | | Cu (%) | | | | CuCap (%) | | | |
| | | count | mean | max | CV | count | mean | max | CV |
| 8 | Oxide | 116 | 0.10 | 0.93 | 1.5 | 6 | 0.08 | 0.33 | 1.0 |
| 9 | Supergene | 233 | 1.19 | 5.87 | 0.6 | 1 | 1.18 | 3.40 | 0.6 |
| 30 | Hornfels | 3,875 | 0.07 | 1.41 | 1.5 | 13 | 0.07 | 0.48 | 1.5 |
| 50 | Felsic Volcanics | 1,641 | 0.01 | 0.39 | 2.5 | 24 | 0.01 | 0.12 | 2.2 |
| 60 | Porphyry | 7,275 | 0.20 | 1.80 | 0.9 | 8 | 0.20 | 1.40 | 0.9 |
| 80 | Cover | 1,548 | 0.00 | 0.20 | 4.1 | 8 | 0.00 | 0.01 | 1.2 |
| **Total** | | **14,688** | **0.14** | **5.87** | **1.7** | **60** | **0.14** | **3.40** | **1.6** |
| **Rock Type** | | **Au (g/t)** | | | | **AuCap (g/t)** | | | |
| | | **count** | **mean** | **max** | **CV** | **count** | **mean** | **max** | **CV** |
| 8 | Oxide | 116 | 0.58 | 1.78 | 0.7 | 5 | 0.57 | 1.40 | 0.6 |
| 9 | Supergene | 233 | 0.61 | 2.18 | 0.6 | 4 | 0.60 | 1.40 | 0.5 |
| 30 | Hornfels | 3,875 | 0.08 | 3.67 | 1.9 | 10 | 0.08 | 1.20 | 1.7 |
| 50 | Felsic Volcanics | 1,641 | 0.03 | 1.14 | 2.2 | 4 | 0.03 | 0.41 | 1.8 |
| 60 | Porphyry | 7,275 | 0.21 | 3.94 | 1.1 | 3 | 0.21 | 2.00 | 1.1 |
| 80 | Cover | 1,548 | 0.00 | 0.17 | 2.8 | 6 | 0.00 | 0.09 | 2.5 |
| **Total** | | **14,688** | **0.14** | **3.94** | **1.5** | **32** | **0.14** | **2.00** | **1.5** |
| **Rock Type** | | **Mo (%)** | | | | **MoCap (%)** | | | |
| | | **count** | **mean** | **max** | **CV** | **count** | **mean** | **max** | **CV** |
| 8 | Oxide | 116 | 0.0046 | 0.0275 | 1.1 | 4 | 0.0044 | 0.0160 | 0.9 |
| 9 | Supergene | 233 | 0.0027 | 0.0267 | 1.3 | 4 | 0.0026 | 0.0140 | 1.1 |
| 30 | Hornfels | 3,875 | 0.0047 | 0.1660 | 1.9 | 4 | 0.0047 | 0.0900 | 1.8 |

| Cap Impact | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| 50 | Felsic Volcanics | 1,641 | 0.0011 | 0.0856 | 2.6 | 2 | 0.0011 | 0.0210 | 1.9 |
| 60 | Porphyry | 7,275 | 0.0048 | 0.2030 | 1.6 | 6 | 0.0047 | 0.0900 | 1.5 |
| 80 | Cover | 1,548 | 0.0001 | 0.0057 | 1.6 | 3 | 0.0001 | 0.0017 | 1.3 |
| **Total** | | **14,688** | **0.0038** | **0.2030** | **1.9** | **23** | **0.0038** | **0.0900** | **1.8** |

## 13.6 Grade Interpolation

There proved to be insufficient drilling to complete a satisfactory geostatistical analysis. Directional variography was examined, but at the current drill spacing of 80-120 m, grade variability in all but a down-hole direction was poorly correlated. More closely-spaced drilling is required to adequately define the short range of variograms. However, grade continuity is visually apparent through the examination of plans and sections and, as discussed here, grade statistics by rock type are not highly variable as shown by low coefficients of variation.

At this stage of project exploration, geometric interpolation by inverse distance weighting was felt to be most appropriate. The choice of inverse distance squared (2nd power) is based on the relatively low variability of grades as is typical with this type of deposit.

The estimation search strategy was established based mainly on visual directions of grade continuity. Most notably, copper and gold grades trend roughly parallel to the margins of the porphyry unit. Plots of grades versus distance from the porphyry contact led to the creation of north-dipping and south-dipping domains that extended 20 m into the low grade units to the north and south (cover and felsic volcanics), and up to 100 m into the hornfels or into the core of the porphyry. This process adequately constrained most of the anomalously high grades in the cover and felsic volcanic units.

Search distances in the dipping zones were established based on a visual assessment of sections as well as by looking at the range of correlation of down-hole variograms of groups of holes drilled approximately down the dipping zones. Holes in the north-dipping and south-dipping zones were grouped as those approximately down-dip, and those intersecting the zones more obliquely. Down-hole variograms were used to arrive at search ranges (radii) of 150 m x 150 m x 50 m in the plane of the north-dipping zone and 100 m x 100 m x 50 m in the south-dipping zone.

Blocks outside the dipping corridors were interpolated with a 150 m spherical search except in the core of the porphyry where an oblate E-W, vertically-oriented search was deemed most appropriate. Search ellipse details as well as the proportions of the various rock types are listed in Table 13.5.

A two pass search approach was used; units for which an anisotropic search was used in Pass 1 had a second 150 m spherical search applied in Pass 2 to adequately fill blocks.

Based on contact plot analysis of assay data, contacts among the rock types were treated as hard for all metals. Copper, gold, and molybdenum grades were estimated according to the above search criteria using a minimum of five samples, a maximum of 24 samples, and a maximum of seven samples per hole. This approach resulted in 50% of blocks being estimated by two or more holes.

**Table 13.5: Interpolation Parameter and Rock Type Proportions**

| Rock Type or Directional Domain | | Blocks | | Search Pass 1 | | | | | Search Pass 2 | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Search (m) | dip direction/dip | | | | Search (m) | Dip direction/Dip | | | |
| | | Model | Est'd | X/Y/Z | X | Y | Z | Est'd | X/Y/Z | X | Y | Z | Est'd |
| 8 | Oxide | 757 | 100% | 150/150/35 | 90/0 | 0/0 | 0/-90 | 100% | | | | | |
| 9 | Supergene | 765 | 100% | 150/150/35 | 90/0 | 0/0 | 0/-90 | 100% | | | | | |
| 30 | Hornfels | 201,293 | 30% | 150/150/150 | -- isotropic -- | | | 30% | | | | | |
| | *north dipping* | 19,876 | 66% | 150/150/50 | 102/0 | 12/-66 | 12/+24 | 49% | 150/150/150 | Isotropic | | | 16% |
| | *south dipping* | 7,697 | 62% | 100/100/50 | 92/0 | 2/+70 | 2/-20 | 34% | 150/150/150 | isotropic | | | 27% |
| 50 | Felsic Volcanics | 223,467 | 21% | 150/150/150 | -- isotropic -- | | | 21% | | | | | |
| | *south dipping* | 4,481 | 14% | 100/100/50 | 92/0 | 2/+70 | 2/-20 | 5% | 150/150/150 | Isotropic | | | 9% |
| 60 | Porphyry | 89,746 | 58% | 150/65/100 | 90/0 | 0/0 | 0/-90 | 34% | 150/150/150 | isotropic | | | 24% |
| | *north dipping* | 33,847 | 77% | 150/150/50 | 102/0 | 12/-66 | 12/+24 | 64% | 150/150/150 | Isotropic | | | 13% |
| | *south dipping* | 27,283 | 25% | 100/100/50 | 92/0 | 2/+70 | 2/-20 | 14% | 150/150/150 | Isotropic | | | 11% |
| 80 | Cover | 174,370 | 16% | 150/150/150 | -- isotropic -- | | | 16% | | | | | |
| | *north dipping* | 2,426 | 34% | 150/150/50 | 102/0 | 12/-66 | 12/+24 | 32% | 150/150/150 | Isotropic | | | 2% |

## 13.7 Density Assignment

In total, 2,466 dry bulk density measurements were available for this resource estimate. These samples were coded for rock type based on drill holes intersections with the geologic solids. Probability plots of density by rock type were examined for any obviously spurious values; none were detected and all values were used. Table 13.6 lists details of available density measurements and the mean values assigned to the six rock type domains.

**Table 13.6: Average Rock Type Density**

| Rock Type | | Density (t/m$^3$) | | | |
|---|---|---|---|---|---|
| | | Count | Mean | Min | Max |
| 8 | Oxide | 56 | 2.36 | 2.01 | 2.78 |
| 9 | Supergene | 99 | 2.48 | 2.04 | 2.81 |
| 30 | Hornfels | 604 | 2.63 | 1.93 | 3.33 |
| 50 | Felsic Volcanics | 199 | 2.59 | 1.75 | 3.22 |
| 60 | Porphyry | 1,417 | 2.61 | 2.02 | 3.11 |
| 80 | Cover | 91 | 2.44 | 2.10 | 2.78 |
| **Total** | | **2,466** | **2.60** | **1.75** | **3.33** |

## 13.8 Model Validation

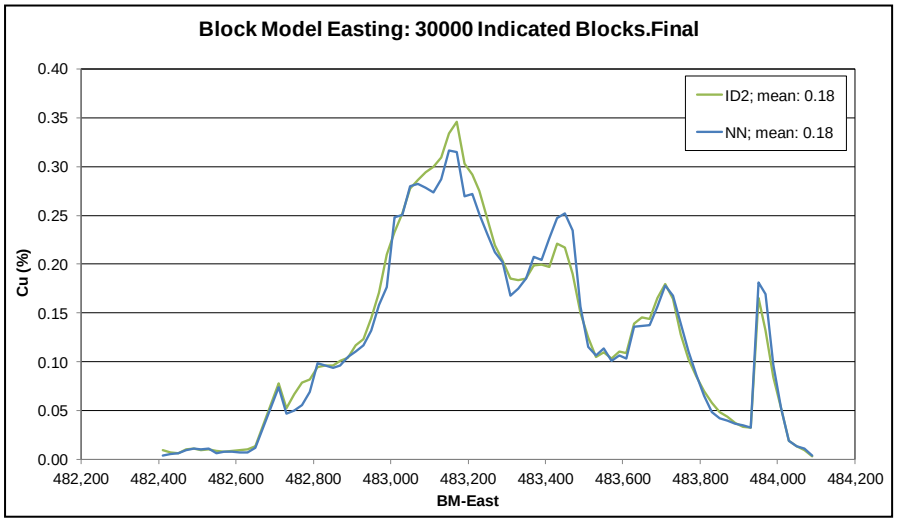
A nearest neighbour (NN) model was estimated using rock type matching as in the inverse distance estimate. This model used a 2 m block height reflecting the composite length. The NN model was re-blocked to the resource model grid (5:1) and used to check various aspects of the estimation process.

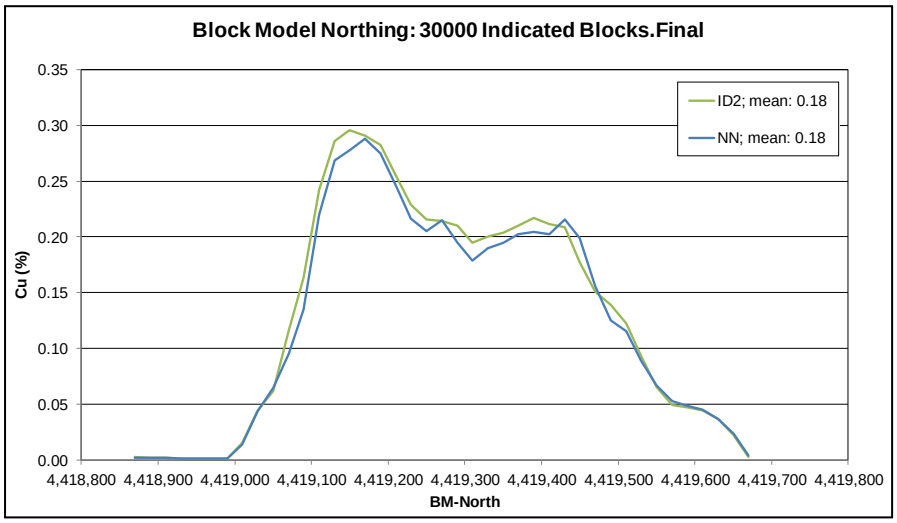
Individual metal as well as copper equivalent grades were validated to ensure consistency with supporting composite data. Visual checks, comparing sample points and block grades on plans and sections, showed good correlation. The copper equivalent grade variable was calculated based on block estimated grades using parameters listed in Table 13.7.

**Table 13.7: Copper Equivalence Parameters**

| Metal | Metal Price | Recovery |
|---|---|---|
| Cu | $ 2.3/lb | 90% |
| Au | $ 850/oz | 70% |
| Mo | $ 12.5/lb | 50% |

A more quantitative validation was made by generating swath plots by block model northings, eastings, and elevations to spatially compare the resource model against NN results. Plots were generated by resource class, globally and within an optimized pit shell. Plots of all Indicated blocks are presented in Figure 13.3 for copper grade and in Figure 13.4 for gold grade; swath plots were also examined for molybdenum as well as for copper equivalent blocks. All plots show good spatial correlation between estimated blocks and the underlying composite data.



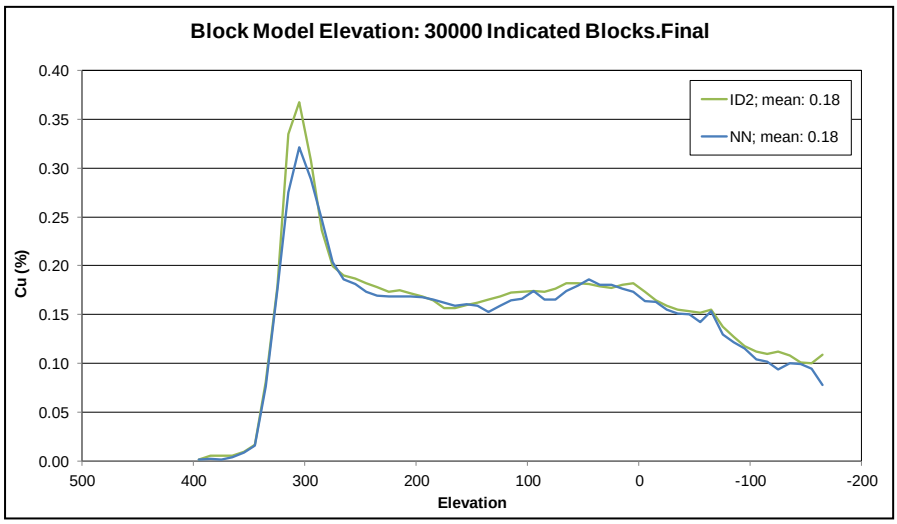
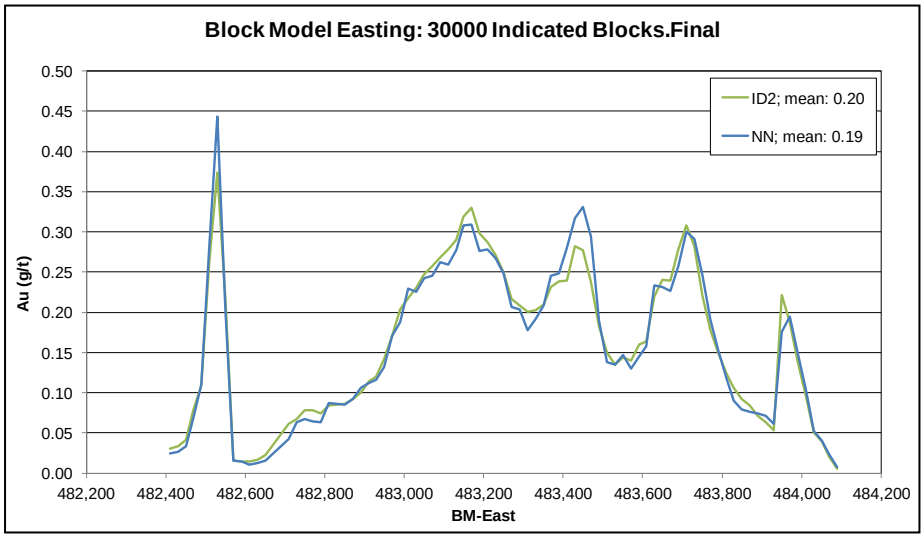
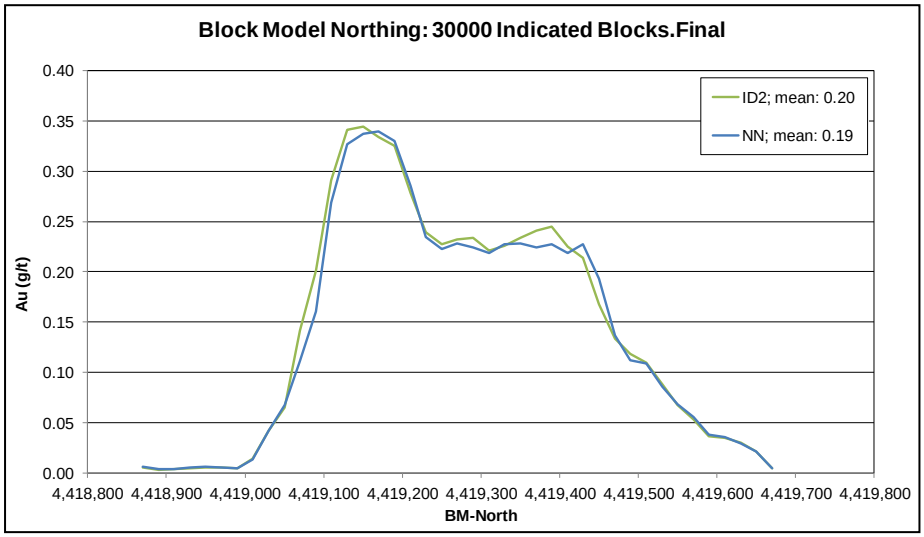




**Figure 13.3: Copper Grade Swath Plots Comparing $ID^2$ and NN Estimates**



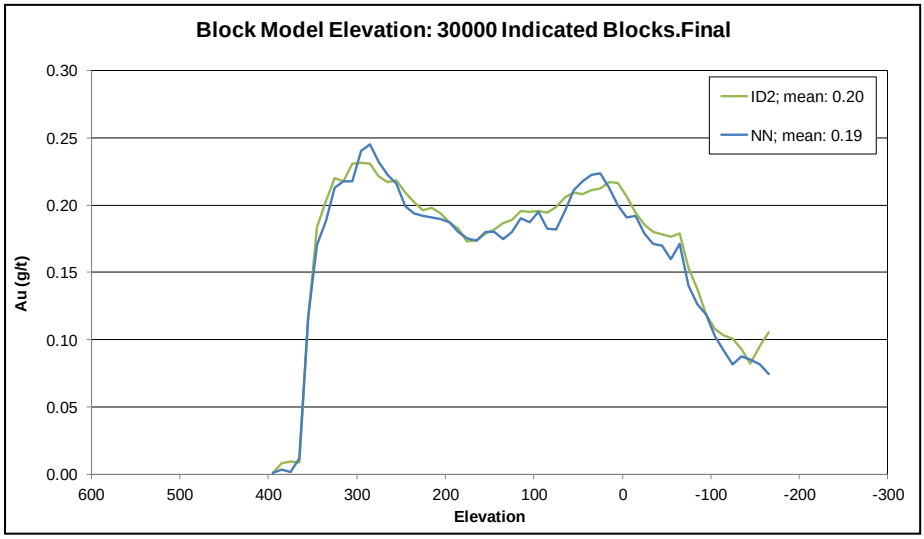




**Figure 13.4: Gold Grade Swath Plots Comparing $ID^2$ and NN Estimates**

A check of grade distribution between the resource estimate and the NN model was made by way of the grade-tonnage curve. Figure 13.5 shows that in the range of likely cut-off grades, the inverse distance model predicts a slightly higher tonnage, and at a similar grade, compared to the NN model. This is felt to be reasonable and to reflect the grade anisotropy built into the model along porphyry contacts. Increased drill density would allow the selectivity of the model to be assessed in future resource estimates.

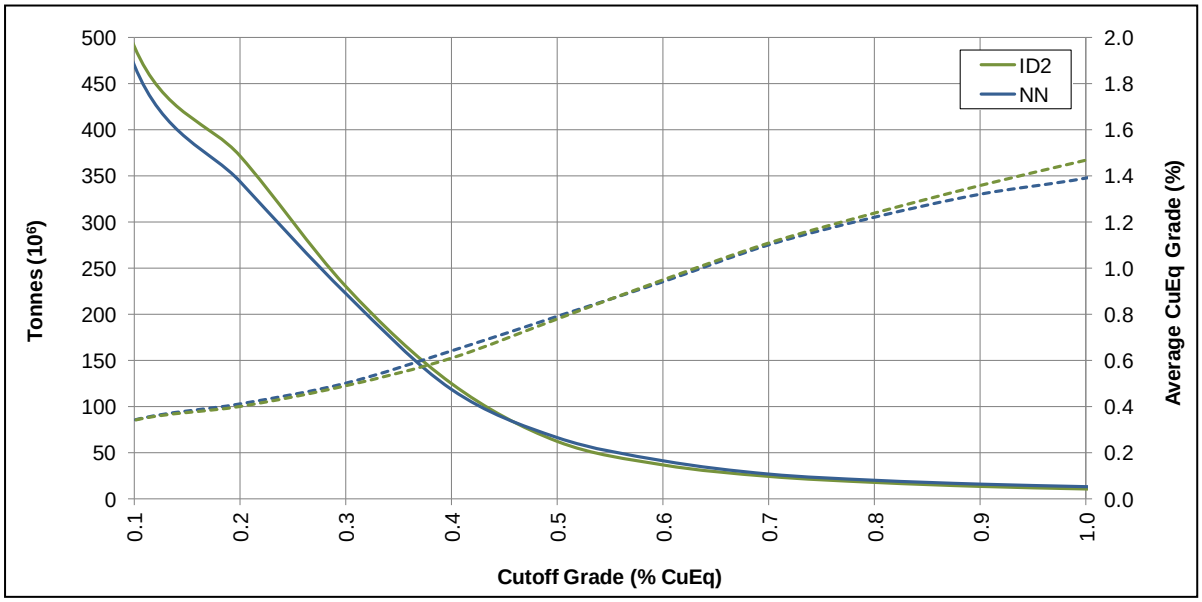



**Figure 13.5: Grade and Tonnage Comparison of $ID^2$ and NN Estimates**

## 13.9 Resource Classification and Tabulation

This estimate was classified based on spatial parameters related to available composite data. These parameters are: (i) the minimum number of holes used to estimate grade, (ii) the average distance to samples used to estimate grade, (iii) the distance to the closest sample used in the estimate, (iv) the distance to the second closest sample, and (v) the distance of the third closest sample.

Classification criteria were established iteratively by visually assessing the impact of parameter adjustment on resultant maps of classified blocks. The goal was to have reasonably cohesive volumes rather than a scattered patchwork of Indicated and Inferred blocks, while assigning the Indicated category in a justified pattern based on sampled locations.

The application of these parameters is listed in Table 13.8. Blocks were initially coded as Indicated if they were:

- Estimated by two holes and within 25 m of a drill hole, or
- Estimated by two holes and within 50 m of the second closest hole, or
- Estimated by three holes and within 25 m of a drill hole, or
- Estimated by three holes and within 80 m of the second closest hole, or
- Estimated by three holes and within 100 m of the third closest hole, or
- Estimated by four holes if the average distance to samples is ≤ 100 m.

Measures were taken to ensure the resource meets the condition of "reasonable prospects of economic extraction" as suggested under NI 43-101. A Lerchs-Grossman pit shell was generated for the purpose of resource tabulation. This pit volume was generated using MineSight® software using a copper price of $4.00/lb applied to copper equivalent grade and an overall pit slope of 45°. Only blocks within the pit volume are included in this resource estimate.

**Table 13.8: Resource Classification Criteria**

| Category | No. Holes min. | Max. Distance to (metres) | | | Avg. Distance max. (metres) |
|---|---|---|---|---|---|
| | | closest | 2nd closest | 3rd closest | |
| **Indicated** | 2 | 25 | | | |
| | 2 | | 50 | | |
| | 3 | 25 | | | |
| | 3 | | 80 | | |
| | 3 | | | 100 | |
| | 4 | | | | 100 |
| **Inferred** | remainder estimated and within pit shell | | | | |

The Halilağa Sulphide Mineral Resource is presented in Table 13.9. A cut-off of 0.2 g/t is felt to be reasonable based on a production rate of 50,000-70,000 tonnes per day from a pit feeding a mill and flotation plant where total operating costs ("OPEX") would be in the range of $10-12 per tonne. At the 0.2% CuEq cut-off, the strip ratio is 2.5:1 within the resource shell.

**Table 13.9: Halilağa Estimated Sulphide Mineral resource by Copper Equivalent Cut-off**

| COG (% CuEq) | Indicated | | | | | Inferred | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes (1,000s) | Cu (%) | Au (g/t) | Mo (%) | CuEq (%) | Tonnes (1,000s) | Cu (%) | Au (g/t) | Mo (%) | CuEq (%) |
| 0.1 | 212,583 | 0.26 | 0.27 | 0.0053 | 0.39 | 272,955 | 0.19 | 0.22 | 0.0063 | 0.30 |
| **0.2** | **168,167** | **0.30** | **0.31** | **0.0055** | **0.45** | **198,668** | **0.23** | **0.26** | **0.0068** | **0.36** |
| 0.3 | 110,162 | 0.37 | 0.38 | 0.0052 | 0.55 | 117,250 | 0.28 | 0.33 | 0.0069 | 0.44 |
| 0.4 | 64,820 | 0.48 | 0.48 | 0.0046 | 0.69 | 58,566 | 0.34 | 0.40 | 0.0071 | 0.53 |
| 0.5 | 40,143 | 0.60 | 0.56 | 0.0036 | 0.85 | 22,020 | 0.43 | 0.53 | 0.0053 | 0.67 |
| 0.6 | 26,605 | 0.73 | 0.63 | 0.0032 | 1.00 | 10,190 | 0.54 | 0.66 | 0.0033 | 0.82 |
| 0.7 | 18,461 | 0.86 | 0.68 | 0.0030 | 1.15 | 5,884 | 0.63 | 0.76 | 0.0026 | 0.96 |
| 0.8 | 14,193 | 0.97 | 0.71 | 0.0030 | 1.28 | 3,639 | 0.73 | 0.85 | 0.0025 | 1.09 |

Due to the differing metallurgical characteristics and anticipated metal extraction methods, the oxide resource is shown below separately in Table 13.10. Although the gold resource is extractable, it is not expected that the base metals within the oxide zone would be recoverable. The cut-off of 0.2 g/t gold is judged as reasonable based on other heap leach gold projects including Alamos Gold's nearby Ağı Dağı project.

**Table 13.10: Halilağa Estimated Oxide Mineral Resource by Gold Cut-off**

| COG (g/t Au) | Inferred | |
|---|---|---|
| | Tonnes (1,000s) | Contained Au (g/t) |
| 0.1 | 5,026 | 0.59 |
| 0.2 | 4,914 | 0.60 |
| 0.3 | 4,604 | 0.63 |
| 0.4 | 4,032 | 0.67 |
| 0.5 | 3,148 | 0.73 |
| 0.6 | 2,200 | 0.81 |
| 0.7 | 1,476 | 0.88 |
| 0.8 | 1,037 | 0.94 |

# 14 Mining Methods

## 14.1 Mining Context

### 14.1.1 Geotechnical Considerations

This study estimates suitable overall pit slope angles and identifies geotechnical issues that need further investigation at pre-feasibility stage. Data used for this PEA level study includes:

- Geological block model;
- Drillhole database containing RQD values;
- Core box photographs; and
- 2012 Halilağa NI 43-101 Report by Kirkham and Gray.

Lithological units across the pit are divided into Kestane Porphyry (quartz porphyry), Hornfels Halo, with felsic volcanics either side. The pit walls will likely comprise felsic volcanics with some porphyry in the slope toes (Figure 14.1).

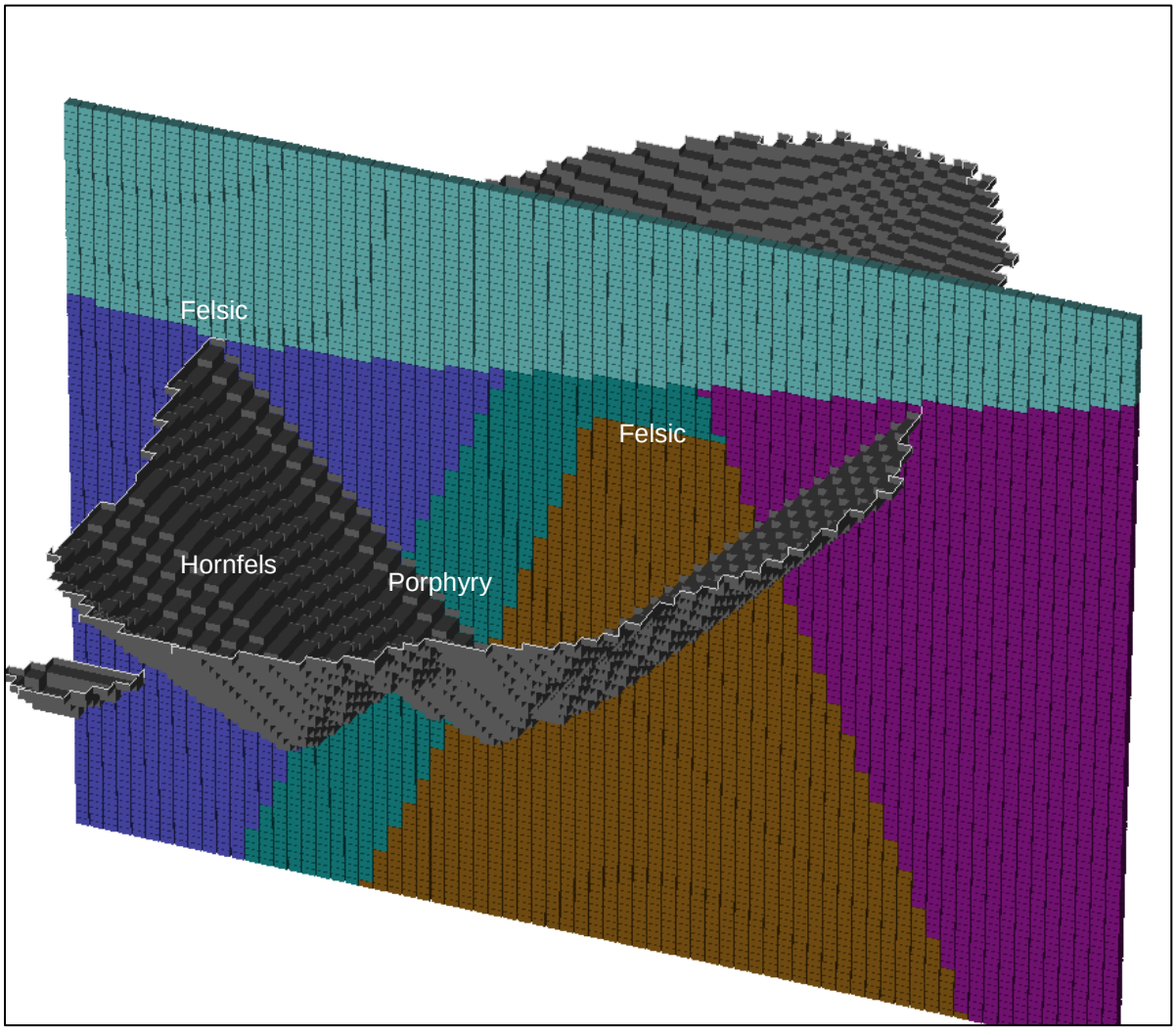



**Figure 14.1 Geological block model through pit showing the main lithological units.**

Data suggests that RQD is largely independent of lithology; although a slight improvement in the rock mass is seen in the porphyry relative to the volcanics. Histograms of RQD per lithology does highlight the large proportion of low quality rock mass (RQD<20%) encountered on site.

Examination of RQD down drill hole in 3D space suggests RQD values in the upper 150-200m are generally less than 20% and rock mass quality appears very low except in the western area of the pit. This is likely a function of alteration and may also be related to the intersection of minor structures (Figure 14.2). Investigation of potential structures is recommended at a pre-feasibility level to develop a structural model. Note that faults have been logged down drill holes but a structural model has not yet been generated.

At depths greater than around 200m rock quality appears to significantly increase. RQD values here are typically 60%-80%. On the western side of the pit RQD values are generally 60%-80% irrespective of depth.

**Figure 14.2: Low RQD in drill hole HD-9 due to relatively high alteration of the rock mass.**

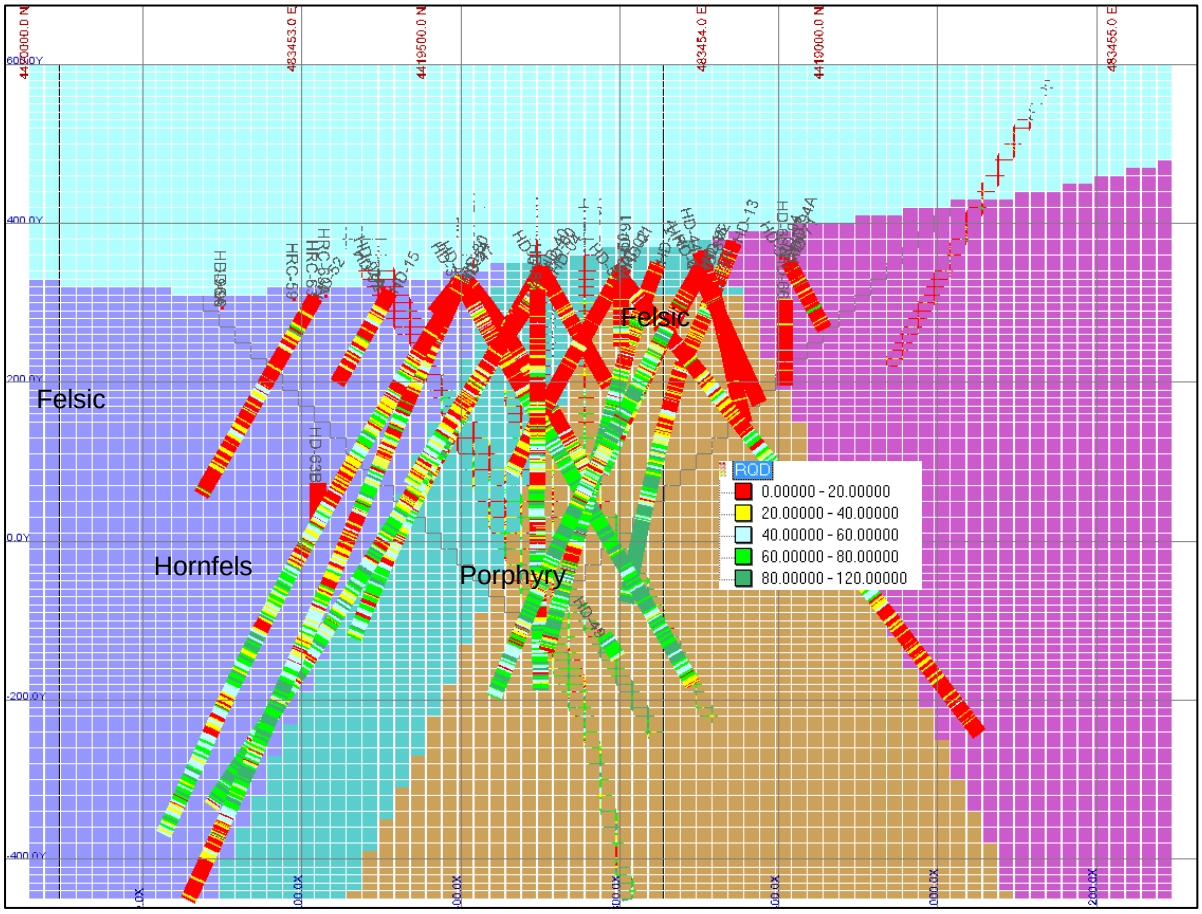



**Figure 14.3: Cross section looking east through the pit with the geological block model superimposed. Drill hole traces are coloured according to RQD value. An improvement in rock mass quality is inferred at approximately 200m depth (SRK).**

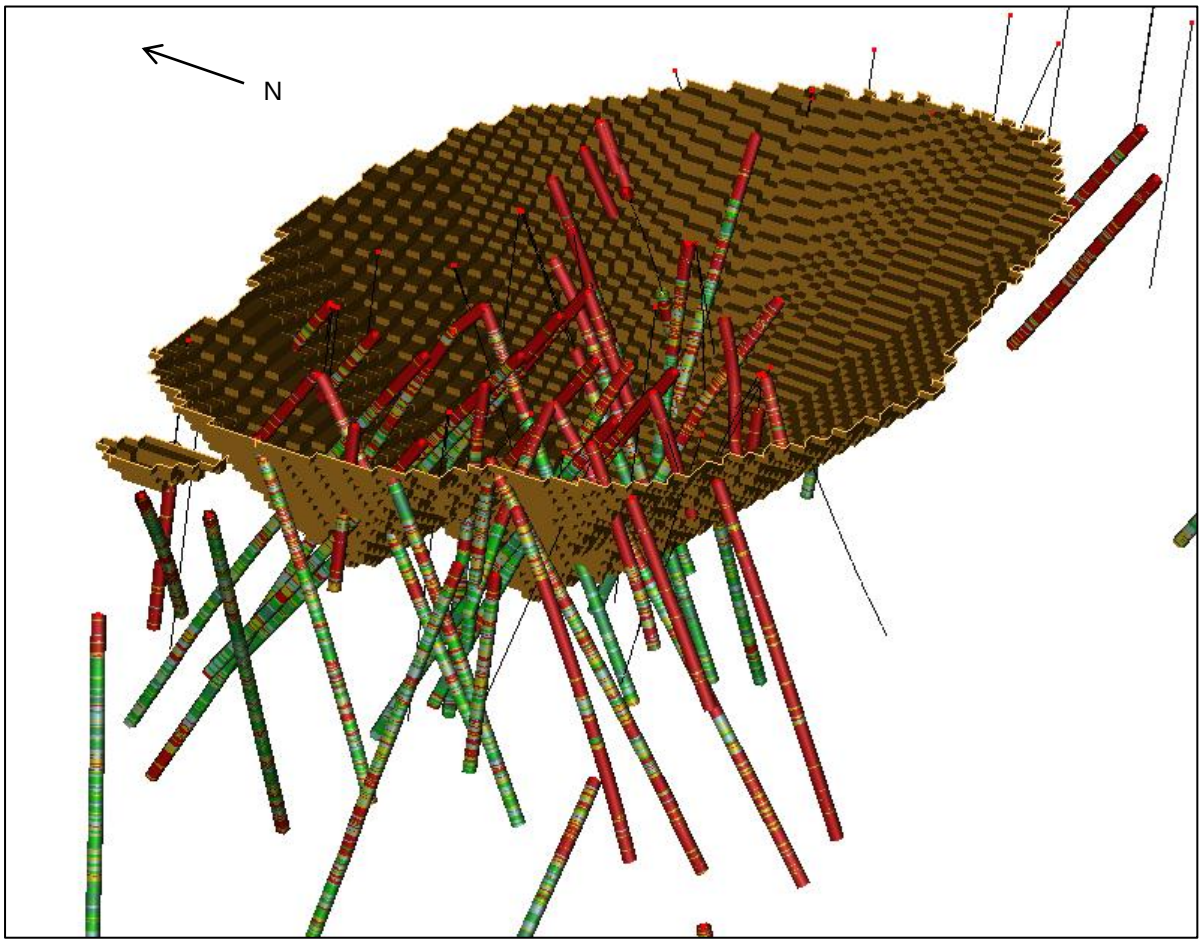



**Figure 14.4: RQD plotted down drillhole for the Halilağa pit. An improvement in rockmass is inferred along the western wall (SRK)**

### 14.1.2 Pit Design Sectors

Three pit design sectors have been generated (Figure 14.5) based on the geotechnical conditions identified. Overall pit slope angles are given for each design sector and discussed in Table 14.1.

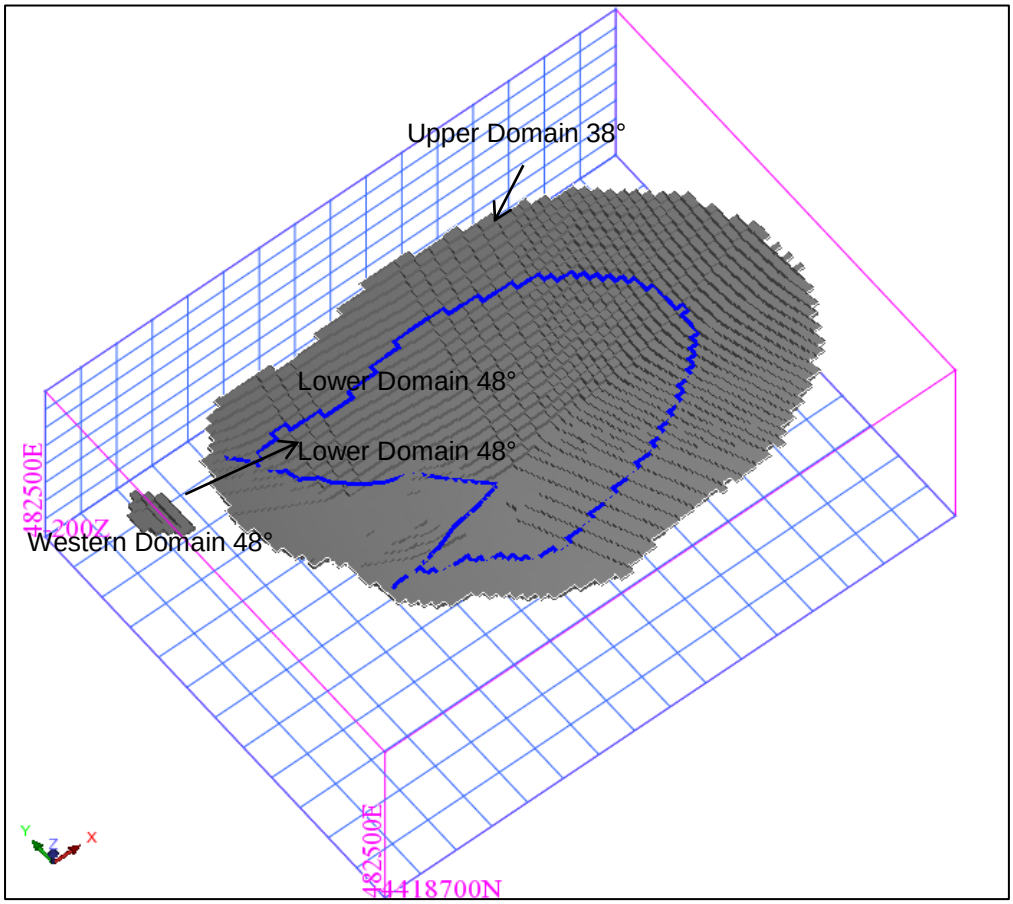



**Figure 14.5: PEA level pit design sectors and overall slope angles (SRK)**

**Table 14.1: Overall slope angles and geotechnical considerations for the Halilağa pit.**

| Design Sector | Maximum overall slope angle | Geotechnical Considerations |
|---|---|---|
| Upper Domain | 38 | Low RQD values and core photographs suggest poor quality rock mass; heavily broken and fractured. May affect overall and interramp stability.<br>Develop model of alteration intensity to assess the impact this has on rock mass quality. |
| Lower Domain | 48 | Rock mass quality improves in this domain. $RMR_{89}$ approximately 50-65. Local zones of poor quality rock mass do exist, however these are often likely to be intersections with major and minor structures. |
| Western Domain | 48 | |

## 14.1.3 Recommendations

To facilitate a mine design process at the pre-feasibility level and develop pit slope bench and interramp configurations, the following work is recommended:

- Develop structural model to enable differentiation between poor rock mass and structures;
- Develop 3D alteration model. Alteration intensity must be recorded using standard classification; and

- Undertake geotechnical drilling into, and behind, the proposed pit walls. Data collected during drilling should typically include, but not be limited to: RQD; rock strength; degree and type of weathering and alteration; joint orientation; joint roughness; joint infilling; aperture; water content.

### 14.1.4 Seismicity

Turkey is situated in a seismically active region. This was considerd in the facilities designs in a broad sense although much more detail will be needed at the next level of study.

## 14.2 Open Pit Optimization

### 14.2.1 Input Parameters

The 3D mineral resource block model, as developed by Gray/Kirkham, was used as the basis for deriving the economic shell limits for the Halilağa Project. The block model dimensions were 20 m x 20 m x 10m.

Estimates were made for gold and copper price, mining dilution, heap leach recovery, offsite costs and royalties. Mining, processing, and general administration OPEX were also calculated based on calculated processing throughput and, along with geotechnical parameters, formed the basis for OP optimization (see Table 14.2). The OP mining costs were estimated for both plant feed material and waste mining, where variations in haulage profiles and equipment selection were taken into account in the cost estimate.

**Table 14.2: Input parameters Used in the LOM Open Pit Optimization**

| Item | Unit | Values |
|---|---|---|
| *Metal Prices* | | |
| Gold | $/oz | 1,200 |
| Copper | $/lb | 2.90 |
| *Recovery to Cu Concentrate* | | |
| Gold | max | 60.8% |
| Copper | max | 85.6% |
| *Cu Concentrate Grade* | | |
| Gold | g/t | var.w/Au and Cu grade |
| Copper | % | 30.0 |
| Moisture content | | 8.0% |
| *Smelter Payables* | | |
| Gold in Cu conc | % | 96% |
| Gold deduction in Cu cons | g/t in conc | 1 |
| Copper in Cu conc | % | 96% |
| *TCRC* | | |
| Cu conc treatment | $/dmt conc | 75.00 |
| Cu refining charge | $/lb pay Cu | 0.075 |
| Au refining charge | $/oz pay Au | 7.00 |
| Transport, marketing, ins., etc. | | |
| Ocean freight to Europe | $/wmt conc | 30.00 |
| Truck freight to Port | $/wmt conc | 20.00 |
| Port charges | $/wmt conc | 6.00 |
| Marine/transportation insurance | % | 0.054% |
| Transport, marketing, ins, etc. | $/wmt conc | 56.00 |
| Transport, marketing, ins, etc. | $/dmt conc | 60.87 |
| *Other Parameters* | | |
| Grade factor (variable) | % | 95 |
| Royalties | % | 2.00% |
| *Operating Costs* | | |
| OP Waste mining Cost | $/waste tonne | 1.86 |
| OP Mineralized material Mining Cost | $/ore tonne | 1.69 |
| OP Average Mining Cost | $/total tonne | 1.77 |
| OP Processing and G&A Cost | $/milled tonne | 5.11 |
| Pit Slope Angles | overall degrees | 38 to 48 |
| Dilution | % | 5.0% |
| Mining recovery | % | 100% |
| Strip ratio (est.) | t:t | 0.90 |
| Internal NSR cut-off | $/t | 5.19 |
| External NSR cut-off (est.) | $/t | 9.08 |
| Processing rate | t/day milled | 50,000 |
| Processing rate | t/yr milled | 18,250,000 |

The mineral inventory block model for the Halilağa deposit was then used with CAE Mining NPV Scheduler (“NPVS”) OP optimization software to determine optimal mining shells. This evaluation included the aforementioned parameters. The economic shell limits included indicated and inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have

the economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the inferred resources would be upgraded to a higher resource category.

### 14.2.2 Cut-off Grade

Table 14.3 summarizes the parameters used, along with incremental (or mill) COG calculations (based on NSR) and mining dilution. The incremental (or mill) COG incorporates all OPEX except mining. This incremental cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the NPVS optimization. This mill cut-off was applied to all of the estimates that follow.

**Table 14.3: Cut-off Grade Calculations used in Pit Optimization**

| Item | Unit | Parameters | |
|---|---|---|---|
| | | Ext. COG (includes mining cost) | Incr. COG (excludes mining cost) |
| **Revenue, smelting & refining** | | | |
| Copper Price | $/lb Cu | 2.90 | 2.90 |
| Payable metal | % Cu | 96% | 96% |
| TC/RC/Transport | $/lb Cu payable | 0.28 | 0.28 |
| Royalties @2% | $/lb | 0.05 | 0.05 |
| Net Cu price | $/lb Cu | 2.45 | 2.45 |
| **On Site Costs** | | | |
| OP Waste Mining Cost | $/t waste mined | 1.86 | 1.86 |
| OP Plant Feed Material Mining Cost | $/t Plant feed material mined | 1.69 | 1.69 |
| Strip ratio (estimated) | Wt:Ot | 0.96 | - |
| OP Mining Cost | $/t milled | 3.65 | - |
| Milling Cost | $/t milled | 4.36 | 4.36 |
| G&A/Sustaining Capital | $/t milled | 0.75 | 0.75 |
| Total Site Costs | $/t milled | 8.76 | 4.94 |
| **Process and Mining Losses** | | | |
| Process Recovery | % Cu | 85.6% | 85.6% |
| Dilution | % | 5% | 5% |
| **Cut-off grade** | | | |
| Insitu Cut-off Cu grade (Cu only)* | % Cu | 0.20 | 0.11 |
| By-product contribution (est) | % of Cu value | 33% | 33% |
| Insitu Cut-off Cu grade (equivalent)* | % Cu | 0.15 | 0.08 |
| Insitu Cut-off NSR (includes dilution) | $/t | 9.19 | 5.19 |
| *estimates only as NSR cut-off used in Whittle | | | |

### 14.2.3 Optimization Results

A series of optimized shells were generated for the Halilağa deposit based on varying revenue factors. The results were analyzed with shells chosen as the basis for ultimate limits and preliminary phase selection.

The results of the pit optimization evaluation on the Halilağa deposit for varying revenue factors values are summarized in Table 14.4 and Figure 14.6 to Figure 14.8, for inferred and indicated resources. Note that the NPV in this optimization summary does not take into account Capital expenditures and is used only as a guide in shell selection. The actual NPV of the project is summarized in the Economics section of this report.

NPVS produces both “best case” (i.e. mine out shell 1, the smallest shell, and then mine out each subsequent shell from the top down, before starting the next shell) and “worst case” (mine each bench completely to final limits before starting next bench) scenarios. These two scenarios provide a bracket for the range possible outcomes. The shells were produced based on varying revenue factors (0.3 through to 1.4 of base case) to produce a series of nested shells with the NPV results shown in Figure 14.6 through Figure 14.7. The NPV values noted here are based on a gold price of $1,200/oz and a copper price of $2.90/lb, do not include CAPEX and were used only to determine the mining shapes.

**Table 14.4: Overall Optimization Results**

| Final Pit | Revenue Factor | Mine Life | Total Mineralized Material Diluted | | | Cu (%) | Cu (Mlbs) | Waste (Mt) | Strip Ratio | Total (Mt) | Total CF ($M) | NPV Best $M disc | NPV Worst $M disc |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | (Mt) | Au (g/t) | Au (koz) | | | | | | | | |
| 1 | 0.30 | 1.4 | 23.3 | 0.53 | 401 | 0.73 | 376 | 7.6 | 0.32 | 30.9 | 826 | 802 | 798 |
| 2 | 0.32 | 1.6 | 26.6 | 0.52 | 441 | 0.68 | 401 | 8.3 | 0.31 | 34.9 | 878 | 849 | 844 |
| 3 | 0.34 | 1.8 | 30.7 | 0.50 | 490 | 0.64 | 432 | 9.7 | 0.32 | 40.4 | 939 | 904 | 897 |
| 4 | 0.36 | 2.1 | 36.0 | 0.48 | 556 | 0.59 | 472 | 13.7 | 0.38 | 49.7 | 1,015 | 971 | 963 |
| 5 | 0.38 | 2.5 | 42.9 | 0.47 | 648 | 0.55 | 524 | 20.9 | 0.49 | 63.9 | 1,117 | 1,059 | 1,048 |
| 6 | 0.40 | 2.9 | 50.6 | 0.46 | 749 | 0.52 | 580 | 30.0 | 0.59 | 80.6 | 1,223 | 1,148 | 1,132 |
| 7 | 0.42 | 3.1 | 55.5 | 0.45 | 808 | 0.50 | 615 | 35.9 | 0.65 | 91.4 | 1,284 | 1,198 | 1,179 |
| 8 | 0.44 | 3.3 | 59.2 | 0.44 | 845 | 0.49 | 637 | 38.4 | 0.65 | 97.6 | 1,322 | 1,229 | 1,207 |
| 9 | 0.46 | 3.5 | 62.1 | 0.44 | 873 | 0.48 | 654 | 40.3 | 0.65 | 102.3 | 1,349 | 1,251 | 1,227 |
| 10 | 0.48 | 3.8 | 68.9 | 0.42 | 936 | 0.46 | 694 | 46.0 | 0.67 | 114.8 | 1,412 | 1,300 | 1,271 |
| 11 | 0.50 | 4.1 | 74.5 | 0.41 | 988 | 0.44 | 726 | 51.1 | 0.69 | 125.5 | 1,459 | 1,337 | 1,303 |
| 12 | 0.52 | 4.5 | 81.5 | 0.40 | 1,039 | 0.42 | 761 | 54.4 | 0.67 | 136.0 | 1,508 | 1,373 | 1,333 |
| 13 | 0.54 | 4.8 | 88.4 | 0.38 | 1,094 | 0.41 | 797 | 60.3 | 0.68 | 148.7 | 1,556 | 1,408 | 1,362 |
| 14 | 0.56 | 6.3 | 117.1 | 0.35 | 1,329 | 0.36 | 941 | 85.0 | 0.73 | 202.1 | 1,747 | 1,537 | 1,470 |
| 15 | 0.58 | 7.0 | 129.6 | 0.34 | 1,417 | 0.35 | 997 | 91.5 | 0.71 | 221.1 | 1,817 | 1,581 | 1,507 |
| 16 | 0.60 | 7.9 | 146.5 | 0.33 | 1,552 | 0.33 | 1,077 | 108.3 | 0.74 | 254.8 | 1,914 | 1,639 | 1,551 |
| 17 | 0.62 | 8.7 | 163.1 | 0.32 | 1,681 | 0.32 | 1,157 | 127.3 | 0.78 | 290.4 | 2,006 | 1,691 | 1,590 |
| 18 | 0.64 | 9.0 | 168.5 | 0.32 | 1,724 | 0.32 | 1,181 | 132.6 | 0.79 | 301.1 | 2,033 | 1,705 | 1,598 |
| 19 | 0.66 | 10.7 | 201.3 | 0.31 | 1,981 | 0.30 | 1,334 | 176.1 | 0.88 | 377.4 | 2,195 | 1,785 | 1,648 |
| 20 | 0.68 | 11.9 | 223.9 | 0.30 | 2,161 | 0.29 | 1,434 | 205.9 | 0.92 | 429.7 | 2,296 | 1,831 | 1,666 |
| 21 | 0.70 | 12.4 | 233.4 | 0.30 | 2,251 | 0.29 | 1,475 | 222.5 | 0.95 | 455.9 | 2,339 | 1,849 | 1,668 |
| **22** | **0.72** | **12.9** | **243.3** | **0.30** | **2,317** | **0.28** | **1,516** | **234.0** | **0.96** | **477.3** | **2,373** | **1,863** | **1,668** |
| 23 | 0.74 | 13.3 | 251.3 | 0.30 | 2,385 | 0.28 | 1,550 | 248.3 | 0.99 | 499.5 | 2,403 | 1,875 | 1,667 |

| | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 24 | 0.76 | 13.8 | 259.6 | 0.29 | 2,454 | 0.28 | 1,587 | 264.7 | 1.02 | 524.3 | 2,432 | 1,887 | 1,665 |
| 25 | 0.78 | 14.5 | 273.1 | 0.29 | 2,548 | 0.27 | 1,640 | 282.9 | 1.04 | 556.0 | 2,469 | 1,900 | 1,654 |
| 26 | 0.80 | 15.1 | 284.2 | 0.29 | 2,611 | 0.27 | 1,683 | 295.5 | 1.04 | 579.7 | 2,496 | 1,910 | 1,649 |
| 27 | 0.82 | 15.8 | 298.4 | 0.28 | 2,715 | 0.26 | 1,738 | 320.1 | 1.07 | 618.6 | 2,528 | 1,921 | 1,631 |
| 28 | 0.84 | 16.5 | 310.8 | 0.28 | 2,774 | 0.26 | 1,779 | 327.8 | 1.05 | 638.6 | 2,548 | 1,927 | 1,619 |
| 29 | 0.86 | 17.1 | 322.4 | 0.28 | 2,856 | 0.26 | 1,823 | 348.9 | 1.08 | 671.3 | 2,568 | 1,933 | 1,603 |
| 30 | 0.88 | 17.3 | 327.6 | 0.27 | 2,880 | 0.25 | 1,840 | 353.1 | 1.08 | 680.7 | 2,575 | 1,935 | 1,597 |
| 31 | 0.90 | 17.8 | 336.0 | 0.27 | 2,939 | 0.25 | 1,873 | 371.1 | 1.10 | 707.1 | 2,585 | 1,938 | 1,583 |
| 32 | 0.92 | 18.1 | 342.9 | 0.27 | 2,979 | 0.25 | 1,897 | 381.5 | 1.11 | 724.5 | 2,592 | 1,940 | 1,571 |
| 33 | 0.94 | 18.3 | 346.0 | 0.27 | 2,996 | 0.25 | 1,907 | 385.3 | 1.11 | 731.3 | 2,594 | 1,940 | 1,566 |
| 34 | 0.96 | 18.6 | 351.3 | 0.27 | 3,018 | 0.25 | 1,925 | 391.0 | 1.11 | 742.3 | 2,596 | 1,941 | 1,560 |
| 35 | 0.98 | 19.1 | 360.9 | 0.27 | 3,079 | 0.25 | 1,959 | 411.6 | 1.14 | 772.5 | 2,600 | 1,942 | 1,539 |
| **36** | **1.00** | **19.4** | **367.8** | **0.26** | **3,109** | **0.24** | **1,982** | **420.3** | **1.14** | **788.1** | **2,600** | **1,942** | **1,531** |
| 37 | 1.02 | 19.6 | 371.9 | 0.26 | 3,134 | 0.24 | 1,995 | 427.9 | 1.15 | 799.8 | 2,600 | 1,942 | 1,521 |
| 38 | 1.04 | 19.9 | 377.7 | 0.26 | 3,164 | 0.24 | 2,013 | 437.3 | 1.16 | 815.0 | 2,598 | 1,941 | 1,509 |
| 39 | 1.06 | 20.5 | 389.0 | 0.26 | 3,216 | 0.24 | 2,050 | 455.3 | 1.17 | 844.4 | 2,593 | 1,940 | 1,485 |
| 40 | 1.08 | 20.8 | 393.3 | 0.26 | 3,240 | 0.24 | 2,062 | 462.6 | 1.18 | 855.9 | 2,590 | 1,939 | 1,474 |
| 41 | 1.10 | 21.0 | 397.0 | 0.26 | 3,266 | 0.24 | 2,075 | 473.4 | 1.19 | 870.4 | 2,587 | 1,938 | 1,462 |
| 42 | 1.12 | 21.5 | 407.1 | 0.25 | 3,310 | 0.23 | 2,106 | 489.5 | 1.20 | 896.7 | 2,577 | 1,936 | 1,435 |
| 43 | 1.14 | 21.6 | 409.4 | 0.25 | 3,318 | 0.23 | 2,113 | 492.5 | 1.20 | 902.0 | 2,575 | 1,935 | 1,432 |
| 44 | 1.16 | 22.0 | 416.8 | 0.25 | 3,364 | 0.23 | 2,136 | 511.9 | 1.23 | 928.7 | 2,563 | 1,933 | 1,407 |
| 45 | 1.18 | 22.1 | 419.9 | 0.25 | 3,377 | 0.23 | 2,145 | 516.7 | 1.23 | 936.6 | 2,559 | 1,932 | 1,400 |
| 46 | 1.20 | 22.4 | 424.9 | 0.25 | 3,397 | 0.23 | 2,160 | 524.9 | 1.24 | 949.8 | 2,551 | 1,930 | 1,387 |
| 47 | 1.22 | 22.8 | 432.1 | 0.25 | 3,443 | 0.23 | 2,182 | 546.1 | 1.26 | 978.1 | 2,536 | 1,927 | 1,361 |
| 48 | 1.24 | 23.0 | 437.1 | 0.25 | 3,462 | 0.23 | 2,197 | 555.6 | 1.27 | 992.6 | 2,526 | 1,925 | 1,347 |
| 49 | 1.26 | 23.2 | 439.2 | 0.25 | 3,473 | 0.23 | 2,202 | 559.4 | 1.27 | 998.7 | 2,522 | 1,924 | 1,340 |
| 50 | 1.28 | 23.3 | 442.7 | 0.25 | 3,489 | 0.23 | 2,212 | 567.8 | 1.28 | 1,010.5 | 2,513 | 1,922 | 1,329 |
| 51 | 1.30 | 23.7 | 450.2 | 0.24 | 3,529 | 0.23 | 2,235 | 588.3 | 1.31 | 1,038.5 | 2,493 | 1,918 | 1,300 |

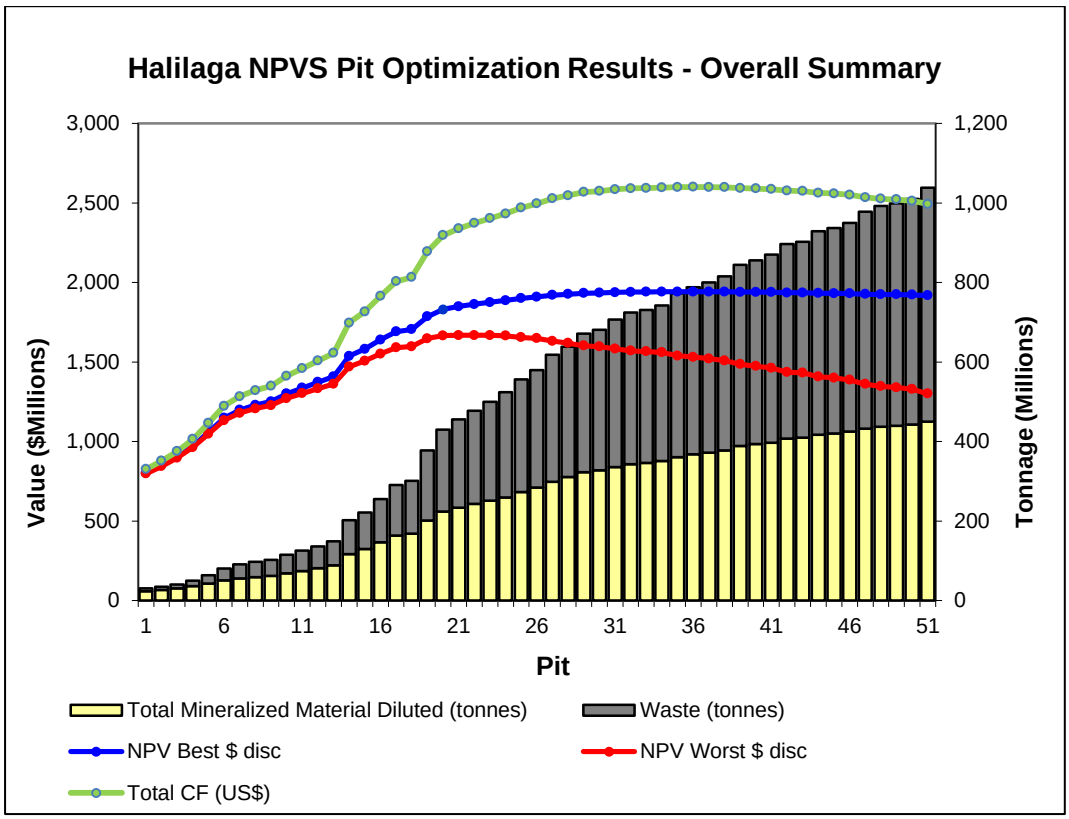



**Figure 14.6: Overall Optimization Results**

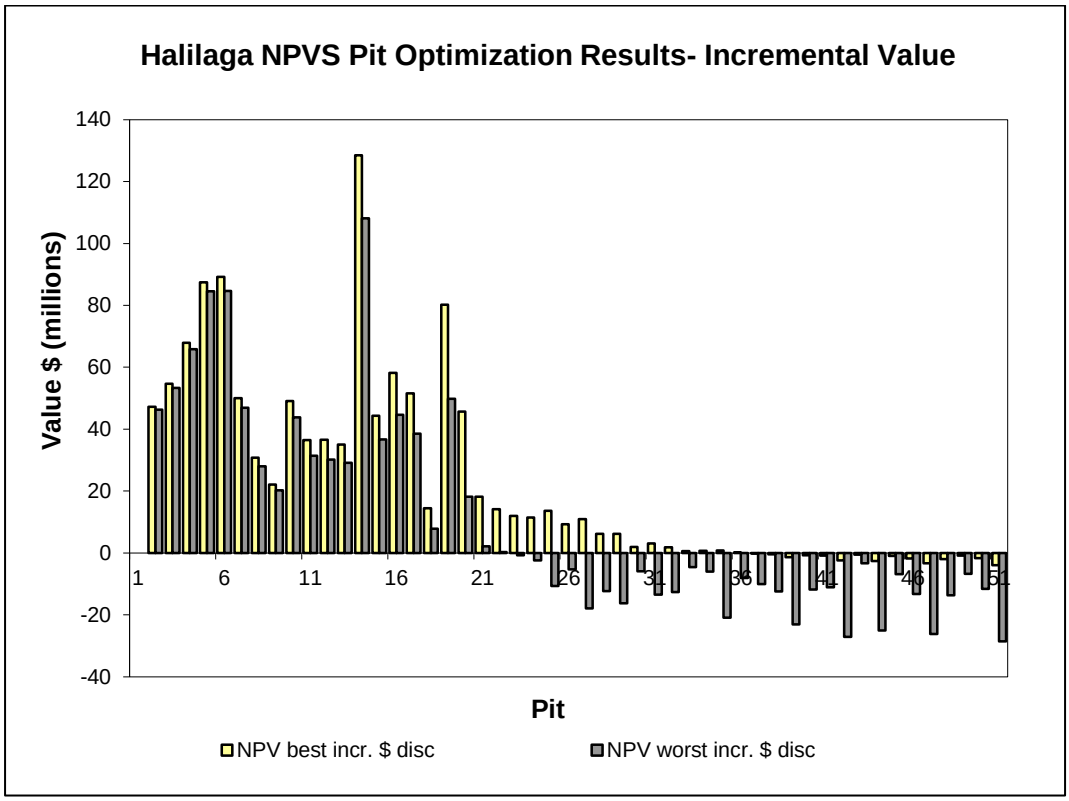



**Figure 14.7: Open Pit Optimization Incremental Value Results**

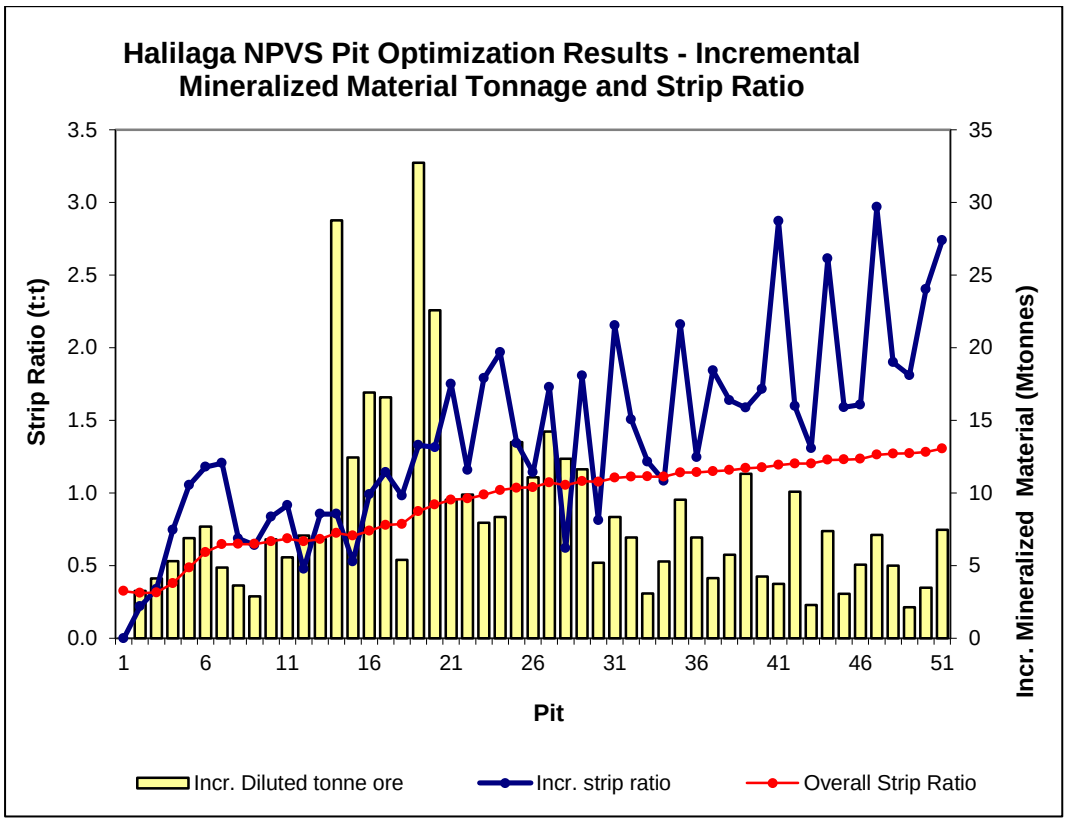



**Figure 14.8: Open Pit Optimization Incremental Tonnage Results**

For the Halilağa deposit, shells beyond Pit Shell 22 add mineralized rock and waste tonnages to the overall pit, but have higher incremental strip ratios with minimal positive impact on the NPV. To better determine the optimum shell on which to base the phasing and scheduling and to gain a better understanding of the deposit, the shells were analyzed in a preliminary schedule. The schedule assumed a maximum processing rate of 18.3 Mt/yr. No stockpiles were used in the analysis and no CAPEX were added.

Based on the analysis of the shells and preliminary schedule, Pit Shell 22 was chosen as the base case shell for further phasing and scheduling for the Halilağa deposit. This shell contains 243 Mt of mineralized material above cut-off with an average copper grade of 0.28% and 1,516 Mlbs contained copper along with a gold grade of 0.30 g/t and 2,317 koz of contained gold. The total waste tonnage in the shell is 234 Mt for a strip ratio of 0.96:1.

Table 14.5 summarizes the tonnages and grades contained within the shell limits (using the incremental cut-off value of $5.19/t). There is a total of 5 Mt of oxide material that is included in the waste tonnages above. This material would be segregated in order to provide ability to process in the future.

**Table 14.5: Resources to be Extracted in LOM Plan**

| Description | Unit | Value |
|---|---|---|
| Mine Production Life | yr | 14 |
| Process Feed Material | Mt | 243 |
| Diluted Copper grade | % | 0.28 |
| Contained Cu | Mlbs | 1,516 |
| Diluted Gold grade | g/t | 0.30 |
| Contained gold | koz | 2,317 |
| Waste | Mt | 234 |
| Total material | Mt | 477 |
| Strip ratio | t:t | 0.96 |

## 14.3 OP Mine Design

Mine planning for the Halilağa deposit was conducted using a combination of software packages, including MINTEC Inc. MineSight™, GEMS™, and NPVS. The base 3D block model was analyzed using GEMS™. The phase selection and production scheduling was undertaken with the use of MineSight™ and NPVS software.

For the Halilağa deposit, the ultimate shell limits, along with the associated phasing, were based on the shell analysis described in this report. Preliminary waste dumps were then designed to account for the material produced in each mining phase and shell.

Shell 22 was chosen as the mining shape limit for the Halilağa deposit. Figure 14.9 and Figure 14.10 represent plan and section views of the ultimate pit shape

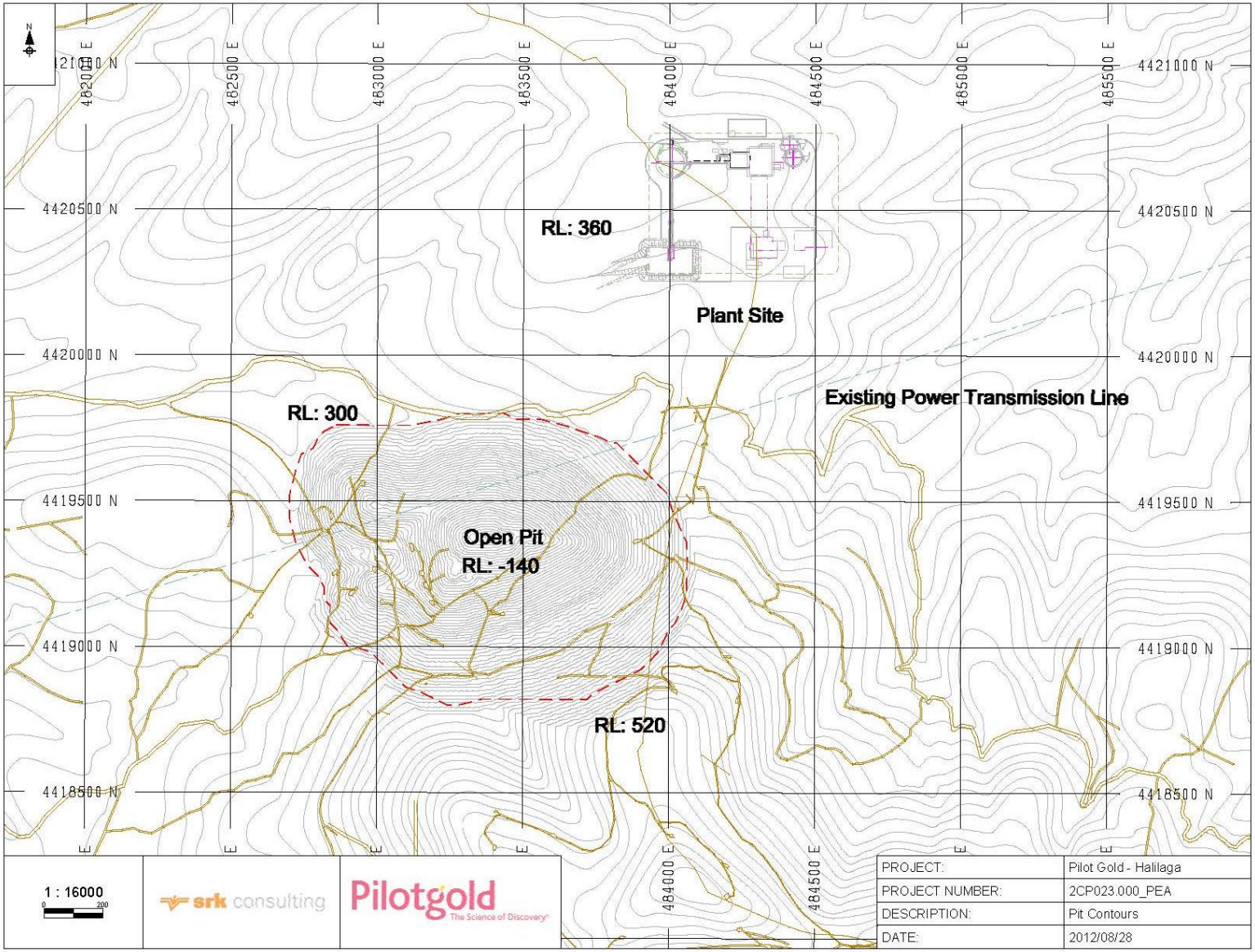



**Figure 14.9: Plan View Halilağa Pit Shell and Waste Rock Management Facilities**

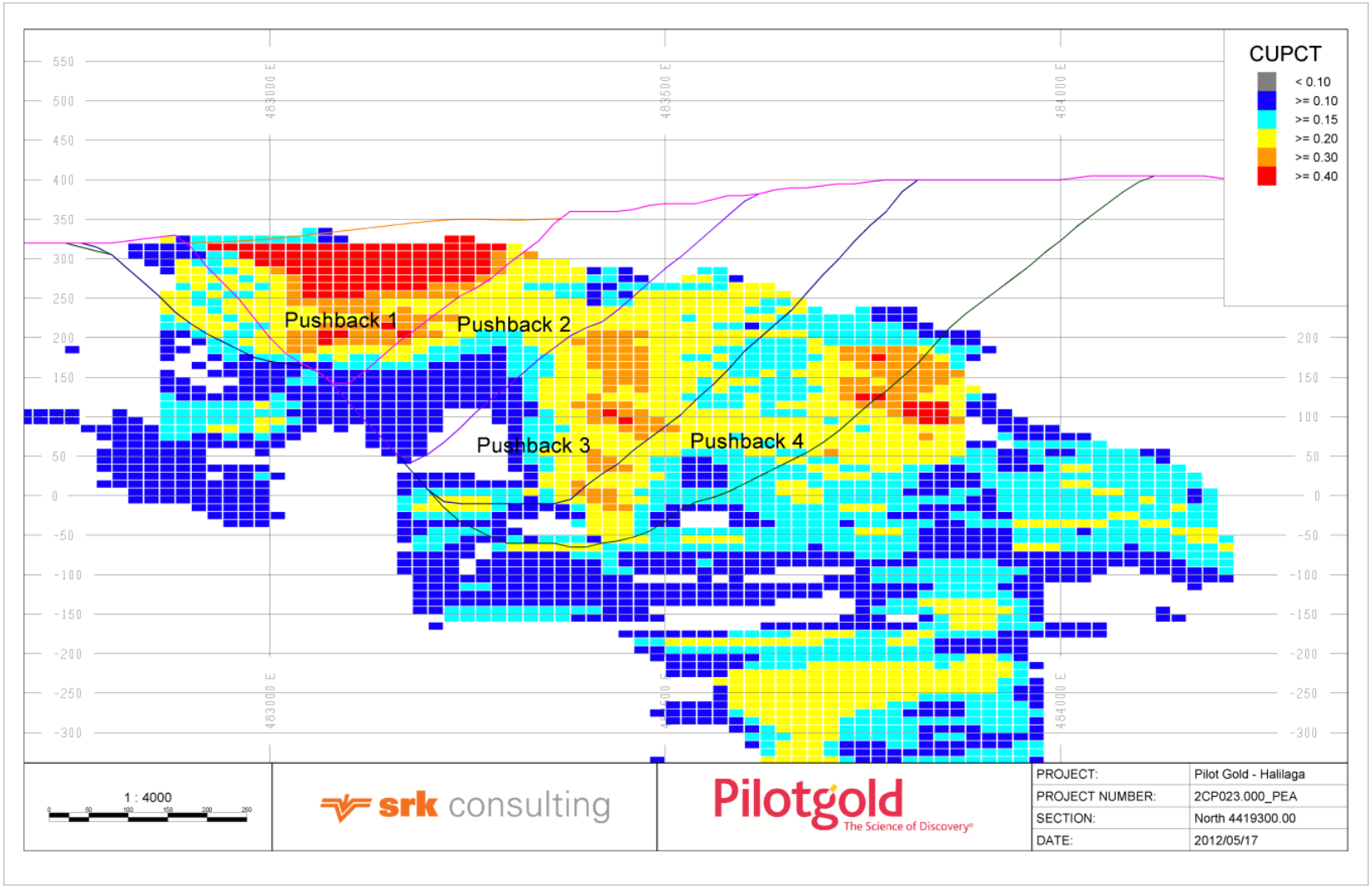



**Figure 14.10: Section View of Halilağa Pit Shell Showing Phase (Stage) Pushbacks**

## 14.4 Mine Sequence/Phasing

The preliminary shells for Halilağa were further analyzed and optimizations were conducted in order to better define the possible stage designs within the ultimate shell limit. The Halilağa pit was further divided into four phases for the mine plan development to maximize the grade in the early years, reduce the pre-stripping requirements in the early years, and to maintain the process facility at full production capacity. The shell tonnages, grades, and contained metal of the preliminary phases (stages) are summarized in Table 14.6.

**Table 14.6: Halilağa Pit/Phase Tonnages and Grades**

| Stage | Total Diluted Millfeed | | | | | Waste (Mt) | Strip Ratio | Total (Mt) |
|---|---|---|---|---|---|---|---|---|
| | (Mt) | Au (g/t) | Au (koz) | Cu (%) | Cu (Mlb) | | | |
| Stage 1 | 52.3 | 0.36 | 612 | 0.46 | 531 | 11.9 | 0.23 | 64.2 |
| Stage 2 | 43.6 | 0.35 | 494 | 0.29 | 279 | 27.9 | 0.64 | 71.5 |
| Stage 3 | 86.9 | 0.23 | 635 | 0.21 | 407 | 83.1 | 0.96 | 170.0 |
| Stage 4 | 60.5 | 0.30 | 576 | 0.22 | 299 | 111.1 | 1.84 | 171.6 |
| **Total** | **243.3** | **0.30** | **2,317** | **0.28** | **1,516** | **234.0** | **0.96** | **477.3** |

Figure 14.11 and Figure 14.12 further illustrate the phase designs for Halilağa, with tonnes, grades, and contained metal shown.

The phases were based on the optimized shells summarized above. Shells selected provide reasonable pushback widths with mining starting in the high grade zone and progressing to the east.

During the active mining and processing of the Halilağa deposit, the waste would be placed into WRFs adjacent to the final shell limits. All mineralized material would be hauled to the process facility to the north of the deposit.

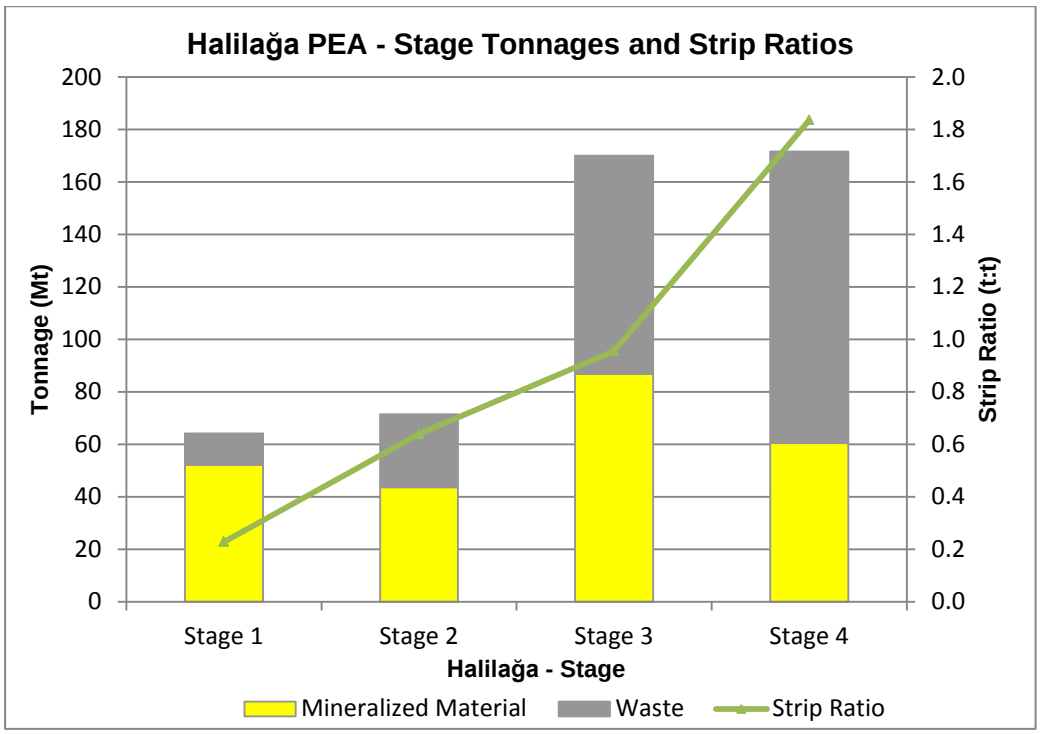



**Figure 14.11: Halilağa Project - Phase Summary**

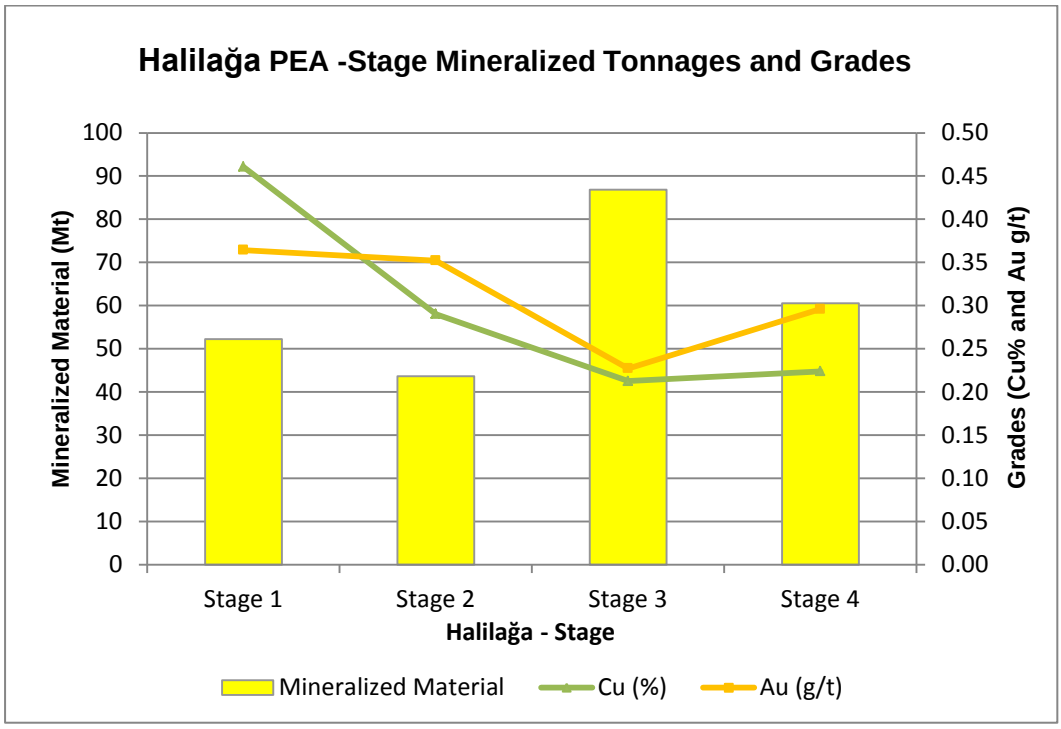



**Figure 14.12: Phase Process Feed Summary**

## 14.5 OP Mine Operation

The OP mining activities for the Halilağa pit were assumed to be primarily undertaken by the owner as the basis for this preliminary economic assessment. The average unit mining costs used in the project economics was $1.66/t of plant feed material mined and $1.86/t waste mined, for pit and dump operations, road maintenance, mine supervision and technical services. The cost estimate was built from first principles and based on experience of similar sized OP operations and local conditions. The OP mining costs for both mineralized material and waste mining take into account variations in haulage profiles and equipment selection.

Labour rates were estimated using local information.

### 14.5.1 Equipment

The major owner mining equipment requirements are indicated in Table 14.7 and are based on similar sized OP operations. The proposed processing rate of 18.2 Mt per annum ("Mtpa") was used, along with deposit and pit geometry constraints, to estimate the mining equipment fleet needed. The fleet has an estimated maximum capacity of 120,000 t/d total material, which would be sufficient for the LOM plan.

**Table 14.7: Major OP Equipment Requirements**

| Equipment Type | No. of units |
|---|---|
| D11-class Dozer | 2 |
| D10-class Dozer | 1 |
| 26 $m^3$ Cable Shovel (electric) | 2 |
| 20 $m^3$ Wheel Loader (diesel) | 1 |
| 220 t Haul Truck | 12 |
| 16H-class Grader | 2 |
| 311 mm dia. Rotary, Crawler Drill (electric) | 2 |
| 250 mm dia. Rotary, Crawler Drill (electric) | 1 |
| 115 mm dia. Rotary, Crawler Drill (diesel) | 1 |
| 150 t Water Truck | 2 |

### Unit Operations

311 mm diameter blast hole drills are planned to perform the bulk of the production drilling in the mine (both mineralized and waste rock). The smaller 250 mm diameter drills would be used in the mineralized zones to allow for better definition drilling. The hydraulic drill with a 115 mm diameter bit is to be used for secondary blasting requirements and may be used on the tighter spaced patterns required for pit development blasts. The main loading and haulage fleet is planned to consist of 220 t haul trucks, loaded primarily with the electric powered 26 cubic metre ("$m^3$") cable shovels or the 20 $m^3$ wheel loader, depending on pit conditions.

As pit conditions dictate, the D11-class dozers are planned to rip and push material to the excavators with the D10-class dozer maintaining the waste dumps.

The additional equipment listed in Table 14.7 is planned to be used to maintain and build access roads and to meet various site facility requirements, including stockpile maintenance and further exploration development.

## 14.6 Mine Schedule

The production schedule for the Halilağa deposit was developed with the aid of NPVS and MineSight™ software, and incorporated the various pits and phases mentioned above.

With the higher grade mineralized material near surface at Halilağa, Year 1 represents the commencement of pre-stripping as well as the processing of mineralized material. The LOM maximum planned amount of total material to be moved is approximately 115,000 t/d. The average total mining rate was planned to be 95,000 t/d.

Indicated and Inferred resources were used in the LOM plan, with Indicated resources making up 56% of the total LOM tonnage processed. The resources calculated included an estimated external dilution factor of 5%. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the inferred resources would be upgraded to a higher resource category.

Table 14.8 is a summary of total material movement by year for the LOM production schedule (both as totals, as well as by each phase).

**Table 14.8: Proposed LOM Open Pit Production Schedule**

| Item | Unit | Year | | | | | | | | | | | | | | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | |
| Mineralized material mined | Mt | 14.0 | 18.3 | 18.3 | 18.3 | 18.3 | 18.3 | 18.2 | 18.2 | 18.3 | 18.2 | 18.3 | 18.3 | 18.2 | 10.3 | **243.3** |
| Gold feed grade | g/t | 0.49 | 0.38 | 0.41 | 0.28 | 0.35 | 0.19 | 0.23 | 0.27 | 0.24 | 0.21 | 0.29 | 0.34 | 0.27 | 0.19 | **0.30** |
| Contained gold | koz | 220 | 222 | 243 | 167 | 203 | 112 | 137 | 157 | 140 | 124 | 169 | 200 | 161 | 64 | **2,317** |
| Copper feed grade | % | 0.83 | 0.37 | 0.34 | 0.29 | 0.29 | 0.17 | 0.20 | 0.24 | 0.23 | 0.21 | 0.19 | 0.24 | 0.25 | 0.21 | **0.28** |
| Contained Cu | Mlbs | 256 | 149 | 139 | 115 | 115 | 68 | 80 | 96 | 92 | 85 | 76 | 97 | 99 | 48 | **1,516** |
| Waste mined | Mt | 10.1 | 9.6 | 21.4 | 24.0 | 22.8 | 22.4 | 22.8 | 23.3 | 23.7 | 24.0 | 23.5 | 6.4 | 0.0 | 0.0 | **234.0** |
| Strip ratio | t:t | 0.72 | 0.53 | 1.17 | 1.31 | 1.25 | 1.23 | 1.25 | 1.28 | 1.30 | 1.32 | 1.29 | 0.35 | 0.00 | 0.00 | **0.96** |
| Total material mined | Mt | 24.1 | 27.9 | 39.6 | 42.2 | 41.0 | 40.7 | 41.0 | 41.6 | 41.9 | 42.2 | 41.8 | 25.5 | 17.7 | 10.0 | **477.3** |
| Oxide material mined* | Mt | 4.96 | 0.00 | 0.09 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | **5.04** |

*included in waste total

The Halilağa deposit is planned to produce a total of 243 Mt of plant process feed and 234 Mt of waste (0.96:1 overall strip ratio) over a fourteen year mine operating life. The current LOM plan focuses on achieving consistent processing feed production rates, mining of higher grade material early in the schedule, balancing grade and strip ratios, while trying to maximize NPV. No blending of stockpiled material was included in this preliminary schedule. In order to achieve targets, up to three phases are active in any one year.

Figure 14.13 summarizes process tonnage, waste tonnages, and strip ratio by period. Figure 14.14 illustrates the feed tonnage by phase and period, as well as overall gold grades, while Figure 14.15 illustrates annual bench advance and material movement. During full production, the mine, on average, is estimated to produce total recovered copper of 97 Mlbs per year and recovered gold of approximately 100 koz per year.

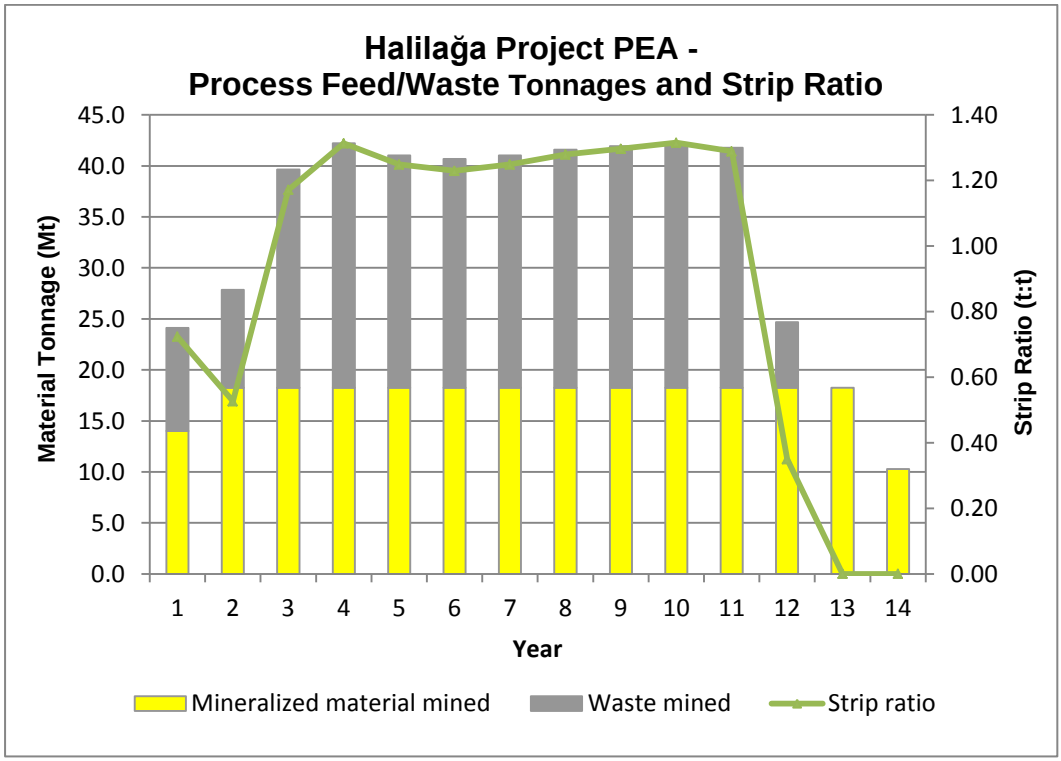



**Figure 14.13: Process Tonnes, Waste Tonnes and Strip Ratio by Period**

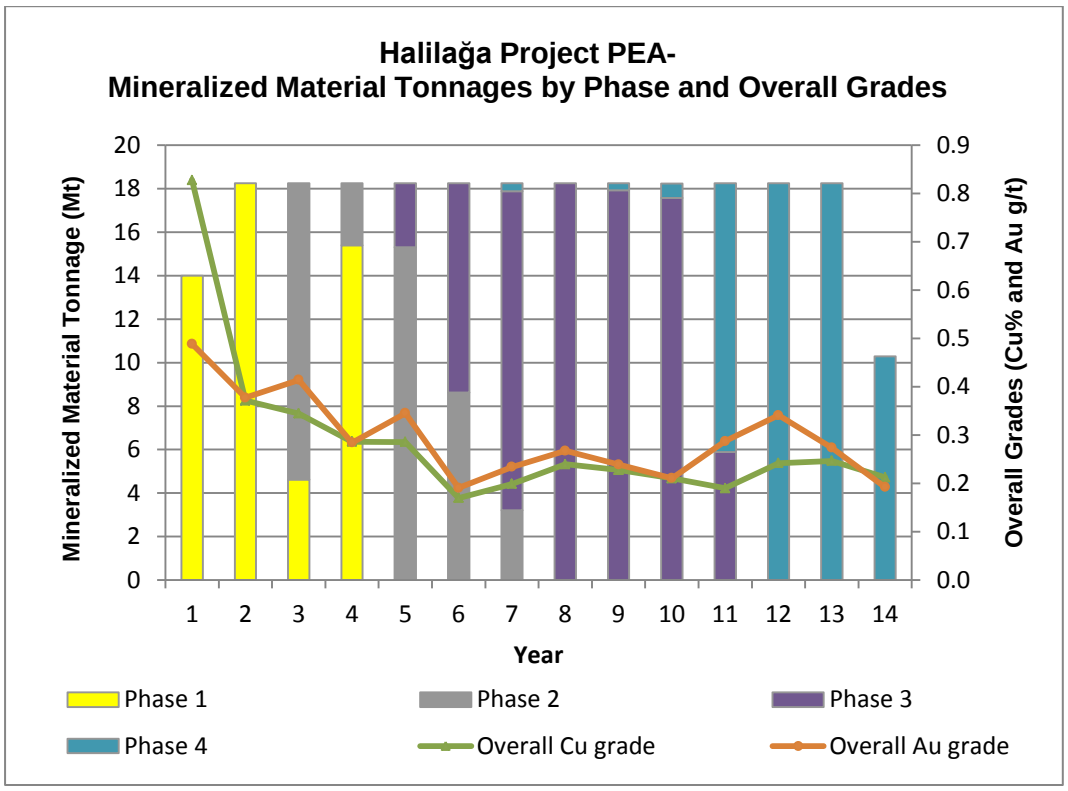



**Figure 14.14: Mineralized Tonnes and Grade by Phase and Period**

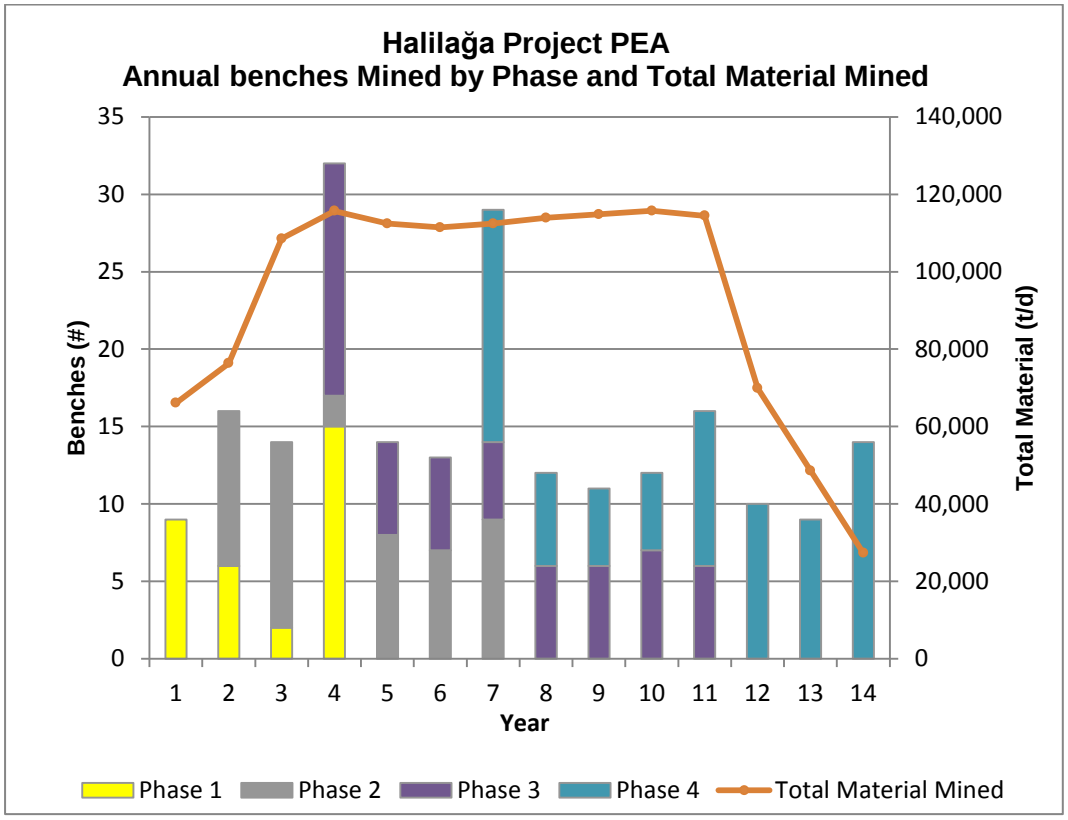



**Figure 14.15: Annual Benches and Total Material Mined by Phase and Period**

To further illustrate the progression of mining of the Halilağa deposit, Appendix A provides snapshots of the pit and waste rock facilities at the end of various periods.

The Halilağa deposit provides maximum returns when the various phases are mined concurrently. The pit is mined out in a series of push-backs in order to achieve the required process feed, while trying to maximize the NPV of the project.

## Mine Development Schedule

**Year 1:** Development of the Halilağa Project commences with pre-stripping and mine production of the pit. Plant processing is also scheduled to begin in Year 1 as a result of the location of high grade mineralized material being near surface leading to minimal pre-stripping requirements. A total of 14.0 Mt of mineralized plant feed (~75% of full production target) and 10 Mt of waste are scheduled. The average copper grade is estimated to be 0.83% and the gold grade is estimated to be 0.49 g/t with an overall strip ratio of 0.72:1 (waste:plant feed).

**Year 2**: The 18.3 Mtpa target plant feed is envisioned to be attained with mining continuing in Stages 1 and 2. The average mine grade in Year 2 is estimated to 0.37% Cu and 0.38 g/t Au. A total of 9.6 Mt of waste rock is scheduled at a mined strip ratio of 0.5:1.

**Year 3**: Process plant feed production is scheduled to be maintained at the target of 18.3 Mtpa (or 50 ktpd). Mining continues in Stages 1 and 2. Total waste planned to be mined from the two active phases is 21 Mt. The average gold grade is estimated to be 0.41 g/t Au, with a copper feed grade of 0.34 %Cu. Stripping of push backs is planned to increase overall strip ratio to 1.2:1. Production rates are envisioned to increase to an average of 109,000 t/d total material for the year.

**Year 4**: Mining of Stage 1 is completed with Stage 2 still active. Process plant feed production is planned to be maintained at steady state of 18.3 Mt total delivered. Given the stripping of pushbacks required, 24 Mt of waste is planned to be mined at an overall strip ratio of 1.3:1. Production rates for the year are scheduled to peak at 116,000 t/d total material.

**Year 5 to 9**: Mining in Stage 2 is completed over this time frame, while Stages 3 and 4 see some activity, with an average of 41 Mt of total material mined with an overall strip ratio of 1.2:1. The average gold grade is expected to be 0.26 g/t Au with an average copper grade of 0.22 % Cu.

**Year 10 to 14**: Mining is concentrated in Stages 3 and 4 over the final years of the LOM plan. The average annual strip ratio is expected to decrease to 0.6:1with an average of 12 Mt of waste and 16.7 Mt of process plant feed planned to be mined in each period. Average mining rate is scheduled to be 75,000 t/d.

# 15 Mineral Recovery Methods

## 15.1 Process Plant Design Considerations

The Halilağa process plant and associated service facilities would process 50,000 t/d of ROM material, to produce copper concentrate and tailings. The proposed process includes crushing and grinding of the ROM material, rougher and cleaner flotation, regrinding, and dewatering of copper sulphides. The copper concentrate is trucked from site to the Port of Bandirma for transport for downstream processing at another facility outside the scope of this report. The flotation tailings would be thickened before placement in the TMF.

### 15.1.1 Process Plant Design Criteria

The overall goal was to design a robust process plant delivering good value for capital. The key process criteria for the plant design and OPEX are provided in Table 15.1.

**Table 15.1: Summary of the Process Design Criteria**

| Criteria | Description | Units | Design | Source |
|---|---|---|---|---|
| Crusher Feed | | kt/d | 50 | Pilot Gold Inc. |
| | | Mt/a | 18.25 | Pilot Gold Inc. |
| Crusher Availability | | % | 70 | Ausenco |
| Crusher Throughput | | t/h | 2 976 | Calculation |
| Crusher Selection | | Size | 60 x 89 | Ausenco |
| | | Number | 1.0 | Ausenco |
| Mill Throughput | | Mt/y | 18.25 | Pilot Gold Inc. |
| Mill/Flotation Availability | | % | 92 | Ausenco |
| Mill Throughput | | t/h | 2 264 | Ausenco |
| Physical Characteristics | BWI | kWh/t | 12.3 | Test work |
| Primary Grind Size | $P_{80}$ | µm | 150 | Test work |
| Concentrate Grind Size | $P_{80}$ | µm | 20 | Test work |
| Head Grade (Design) | | % Cu | 0.25 | SRK |
| | | % S | 1.61 | Test Work |
| | | g/t Au | 0.28 | SRK |
| Flotation Recovery | Copper | % | 85.6 | Ausenco/Test Work |
| (Average) | Gold | % | 60.8 | Ausenco/Test Work |
| Cu Circuit Residence time | Roughers | mins | 20 | Test work |
| | Cleaner 1 | mins | 15 | Ausenco/Test work |
| | Cleaner Scav. | mins | 6.3 | Ausenco |
| | Cleaner 2 | mins | 10 | Ausenco/Test work |
| | Cleaner 3 | mins | 7.55 | Ausenco/Test work |
| Cu Concentrate Filtration Rate | | $kg/m^2/h$ | 229 | Ausenco |
| Concentrates Thickening Flux | | $t/m^2/h$ | 0.250 | Ausenco |

| Criteria | Description | Units | Design | Source |
|---|---|---|---|---|
| Tailings Thickening Flux | | $t/m^2/h$ | 0.85 | Ausenco |
| Tailings Thickener Underflow Density | | % w/w | 60 | Ausenco |
| Collector Consumption (PAX) | | g/t | 20 | Ausenco/Test work |
| Collector Consumption (W34) | | g/t | 40 | Ausenco/Test work |
| Frother Consumption (MIBC) | | g/t | 25 | Ausenco/Test work |
| Lime Consumption | | kg/t | 0.45 | Ausenco/Test work |
| Flocculant Consumption (Concentrate and tailings) | | g/t | 20 | Ausenco |
| SAG Mill Media Consumption | | kg/t | 0.27 | Calculation |
| Ball Mill Media Consumption | | kg/t | 0.25 | Calculation |
| Regrind Mill Media Consumption | | kg/t | 0.04 | Ausenco |

Detailed Process Design Criteria incorporating the process mass balance, engineering design criteria and key design, are derived from the results of the metallurgical test work program.

Sections 15.1.3 and 15.1.7 outline the flow sheet development and describe plant processes and plant layout.

These sections should be read in conjunction with the:

- Process Flow Diagram (Appendix B)
- General Arrangement Drawing (Appendix B)

### 15.1.2 Plant Design Basis

The key criteria selected for the plant design are:

- Treatment of 50kt/d;
- Design availability of 92% (after ramp-up), resulting in 8,059 operating hours per year, with standby equipment in critical areas;
- Sufficient plant design flexibility for treatment of all mineralized material types at design throughput; and
- The selection of these parameters is discussed in detail below.

#### Throughput and Availability

A throughput of 50,000 t/d was selected for the scoping study. Ausenco has designated an overall plant availability of 92% or 8,059 h/a. This is an industry standard for a large, multi-train, flotation plant with moderately abrasive plant feed material. Benchmarking indicates that similar plants have consistently achieved this level or above after ramp-up.

#### Head Grade

The plant is designed to treat a tonnage of primary feed material with a nominal average head grade of 0.29% Cu and 0.30 g/t Au, with a design maximum of 0.80% Cu.

### 15.1.3 Flow Sheet Development and Equipment Sizing

The process plant flow sheet design for the Halilağa plant was based on the process design criteria and metallurgical test results. A process plant schematic for the Halilağa plant is shown in Figure 15.1.

Details of the flow sheet design and selection of major equipment for the various options are discussed in the sections on following pages.

#### Unit Process Selection

- The process plant design is based on a flow sheet with unit operations that are well proven in the sulphide flotation industry, incorporating the following unit process operations:
- Ore from the OP is crushed using a primary gyratory crusher to a crushed product size of nominally 80% passing (P80) 150 mm and fed onto the stockpile feed conveyor;
- Conical stockpile of crushed plant feed material with a live capacity of 24 h, with two apron feeders, each capable of feeding 120% of the full mill throughput;
- A 15 MW SAG mill, 11.58 m diameter x 5.79 m EGL, in closed circuit with pebble crushing based on the comminution design parameters defined in Section 11.2.2;
- Pebble crushing would be comprised of one MP1000, crushing to a product size of nominally 80% passing (P80) 12 mm;
- Two 7 MW ball mills, 6.10 m diameter x 9.14 m EGL, in closed circuit with hydro-cyclones, grinding to a product size of nominally 80% passing (P80) 150 µm;
- Rougher flotation consisting of six 300 m3 forced air tank flotation cells to provide a total of 20 minutes of retention time;
- Rougher concentrate regrinding in one 3.0 MW horizontal fine grinding Isamill to a P80 of 20 µm;
- Cleaner 1 and cleaner scavenger flotation consisting of ten 50 m3 forced air tank flotation cells to provide a total of 21 minutes of retention time;
- Cleaner 2 flotation cells consisting of five 38 m3 trough shaped flotation cells to provide a total of 10 minutes of retention time;
- Cleaner 3 flotation cells consisting of three 38 m3 trough shaped flotation cells to provide a total of 7.5 minutes retention time;
- Copper concentrate thickening in a high rate thickener and filtration in a horizontal plate and frame pressure filter;
- Tailings thickening in a 60 m high rate thickener to an underflow density of 60% solids;
- Tailings Storage Facility (TSF) for process tailings in a conventional dam;
- Process plant water supply reticulated throughout the plant as required;
- Process water dam storage facility and distribution system. Process water is supplied from water reclaimed from the TMF, from process operations and site run-off with raw water used as make-up water as required;
- Potable water is generated by treatment of raw water in a reverse osmosis (RO) unit at the process plant. Potable water is distributed to the plant, and for miscellaneous purposes around the site, and
- Plant, instrument and flotation air services and associated infrastructure.

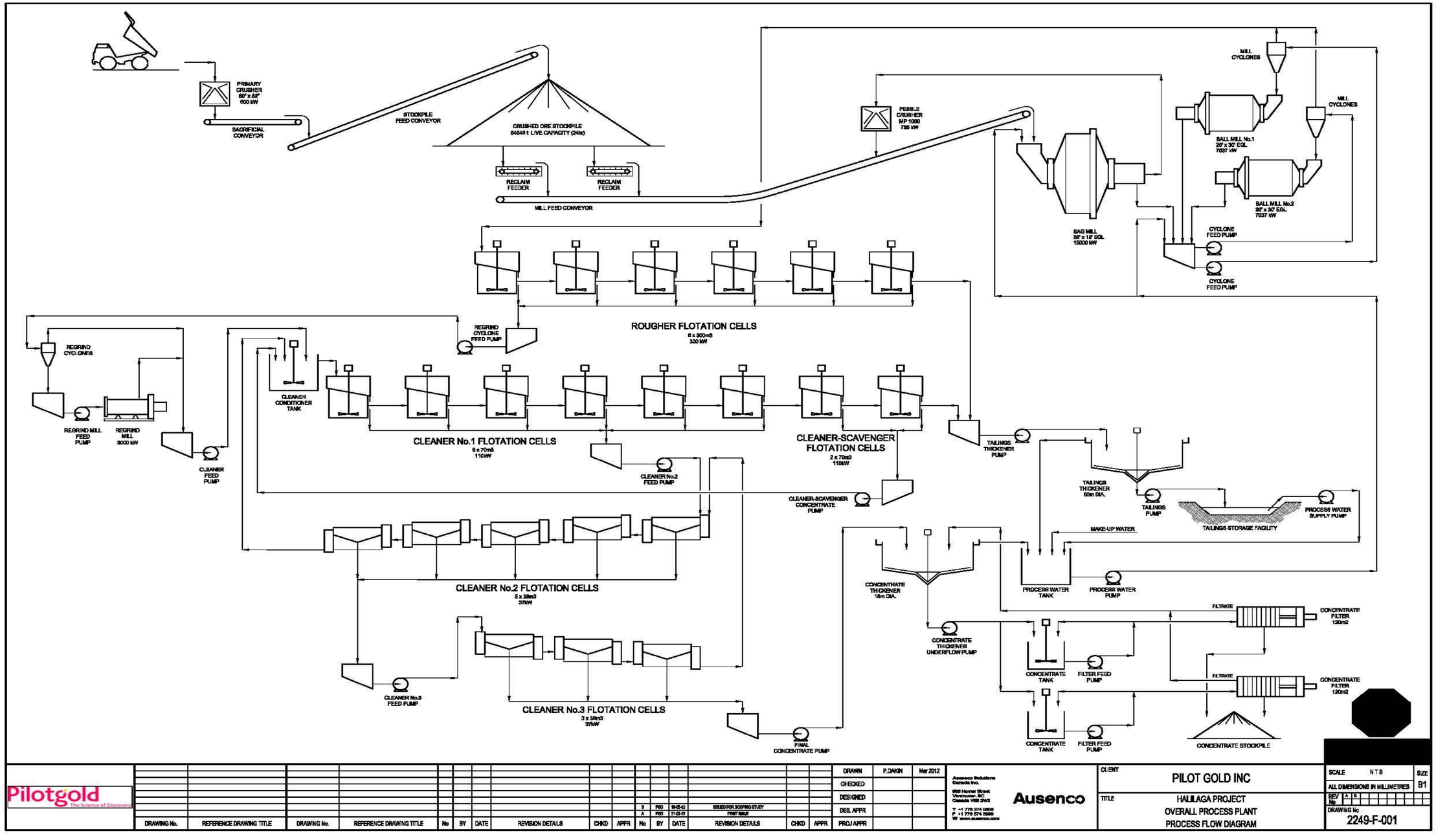



**Figure 15.1: Halilağa Process Plant Schematic**

## 15.1.4 Comminution Circuit Design

### Comminution Circuit Selection

The test work data and evaluation of bench scale flotation results indicate that the majority of the mineralized material is of relatively soft to moderate competency as indicated by Bond ball mill data. Competency testing for design of the primary semi-autogenous mill (SAG) was not conducted.

Discharge from the SAG mill would gravity flow through a trommel screen and into the hydro-cyclone feed pumpbox where it would combine with the two ball mill discharges. The combined SAG and ball mill discharge would be pumped to two hydro-cyclone clusters (one dedicated cluster per ball mill) to split the slurry; underflow would return to the ball mill, overflow would gravity flow to rougher flotation.

Ausenco's analysis of this milling configuration indicates that this circuit would be able to achieve the design throughput of 50 kt/d for mineralized material with a competency (as measured by the A x b) of 50 or greater. More competent mineralized material would only be able to be processed at lower throughput rates.

### Primary Crusher Sizing

Based on the design throughput and mineralized rock characteristics, a gyratory crusher is considered the most suitable primary crusher for the duty. A crusher size of 60" X 89" has been selected.

### Crushed Mineralized Rock Surge Capacity

A crushed mineralized rock stockpile with 24 h live mill feed capacity (50,000 tonnes) has been selected to provide surge between the crusher operation and the continuous milling circuit operation. Two apron feeders have been selected to reclaim plant feed material from the stockpile, each able to deliver the design mill feed rate.

### Comminution Design Criteria

The major comminution design parameters used for this study are:

- Bond Ball mill Work Index (BWI) of 12.3 kWh/t which is the 75th percentile of the six composite samples tested at G&T; and
- Target grind size P80 of 150 µm, based on the G&T rougher flotation test work.

Flotation test work and mineralogy have indicated that Halilağa mineralized material are fine-grained. In order to achieve the specified design metal recoveries, a regrind size of 20 µm has been selected for this study.

The milling power needed to achieve the target grind size, at the nominated design rates in an SABC circuit, requires 27.5 MW of power at the pinions.

### Comminution Circuit Power Efficiency

There are several techniques used by various consultants to determine comminution power requirements. These methods are typically power-based (empirical modification of Bond power) or model-based (e.g. JKSimMet modeling).

Ausenco uses a power approach, based on empirically derived models developed from a database of actual plant operating data and associated bench-scale test work. Critical input parameters to the model are material competency (measured by either JK drop weight Axb or SMCC DWI values) and

Bond work indices (CWI, RWI and BWI). Ausenco's power-based model predicts the milling efficiency of the various circuits based on the JK drop weight/SMCC data, which is a measure of rock competency.

The specific energy and mill sizing determined using Ausenco's in-house method is shown in Table 15.2.

**Table 15.2: Mill Design Criteria**

| Criteria | | Units | Design |
|---|---|---|---|
| Throughput | | t/h | 2 264 |
| Mill Type | | | SAG Grate D/C |
| Pinion Power required | | kW | 12,481 |
| Mill Speed | | % Nc | 75 |
| Ball Charge Volume | Nominal, operating | % vol | 9 |
| | Maximum for design | % vol | 15 |
| Total Charge Volume | Nominal, operating | % vol | 26 |
| | Maximum for design | % vol | 35 |
| Mill Diameter | Inside shell | m | 11.58 |
| Mill Length | EGL | m | 5.79 |
| Installed Motor power | | kW | 15,000 |
| Mill Type | | | Ball |
| No. | | | 2 |
| Grind Size | $P_{80}$ | µm | 150 |
| Pinion Power required | | kW | 6,222 |
| Mill Speed | | % Nc | 75 |
| Ball Charge Volume | Nominal, operating | % vol | 35 |
| | Maximum for design | % vol | 40 |
| Mill Diameter | Inside shell | m | 6.10 |
| Mill Length | EGL | m | 9.14 |
| Installed Motor Power | | KW | 7,000 |

Installed ball mill power of 7,000 kW incorporates the allowances for drive train losses to determine the motor power installed from the pinion power required.

The installed motor power for the SAG mill incorporates similar allowances as well as an additional contingency to allow adjustment in the mill operating conditions to handle plant feed material variability. These allowances and contingencies resulted in the selection of a 15,000 kW SAG mill.

## Pebble Crushing

The upfront circuit design would incorporate pebble crushing, with conveyors returning the crushed pebbles to the SAG mill feed. The pebble crushing circuit would comprise of a single feed bin, with a reclaim feeder feeding a 900 kW pebble crusher.

A preliminary pebble circulating load of 15% of the new feed rate was selected at this stage from comparable benchmarking for the design of the pebble crusher, based on typical industry experience

with mineralized material of similar competency. The conveyors are designed to handle peak loads of up to 25% of new feed.

### Mill Circuit Classification

The classification circuit has been designed for a maximum circulating load of 300%. This is a typical industry design value, widely used for an SABC circuit. The SAG mill discharge slurry passes through a trommel screen with 12.5 mm apertures to remove pebbles, the screen undersize flows into the hydro-cyclone feed hopper and combines with ball mill discharge.

The ball mills operate in closed circuit with hydro-cyclones to reduce the particle size to the desired 150 µm. Combined SAG and ball mill discharge is pumped to two clusters of 760 mm diameter hydro-cyclones (15 operating and two spare per cluster) to split the flow to the two secondary mills. Underflow from the hydro-cyclone clusters would each a ball mill operating in parallel for further grinding.

Overflow from the hydro-cyclone clusters report as flotation feed.

## 15.1.5 Flotation Circuit Design

Mineralogical examination has highlighted that the copper mineralogy is fine grained. The plant feed material would require a fine grind to achieve adequate recovery, but should be amenable to conventional flotation separation to produce saleable concentrate grade.

### Circuit Type and Size

The design flow sheet selected consists of rougher flotation, rougher concentrate regrind, cleaner 1, cleaner scavenger, cleaner 2 and cleaner 3 flotation stages. The residence times and flotation parameters for the flotation circuit have generally been based on the test work parameters obtained on the various Halilağa samples. The test work flotation and design residence times are summarized in Table 15.3.

**Table 15.3: Summary of Flotation Residence Times**

| Circuit | Lab | Scale Factor | Specified Design |
|---|---|---|---|
| Rougher, mins | 8.0 | 2.5 | 20 |
| Cleaner 1, mins | 6.0 | 2.5 | 15.0 |
| Cleaner Scavenger, mins | n/a | | 6.0 |
| Cleaner 2, mins | 4.0 | 2.5 | 10 |
| Cleaner 3, mins | 3.0 | 2.5 | 7.5 |

### Flotation Circuit Configuration

The rougher flotation circuit consists of six 300 $m^3$ flotation cells in a single train.

The flotation circuit consists of three pairs of twinned cells. The twinning of rougher cells is a common practice used to reduce both CAPEX as well as installed cost by reducing the number of steps down the length of the rougher train. Single cells have metallurgical advantages over twinned cells in terms of reduced short circuiting and better level control.

In some installations, these benefits outweigh the capital savings and single cells are a more economic approach e.g. if the advantage of running the rougher train down an existing hill is possible the limited capital savings from twinning the cells are outweighed by the metallurgical advantages.

- Cleaner 1 flotation circuit including cleaner scavengets consists of eighteen 70 $m^3$ tank flotation cells.
- Cleaner 2 flotation circuit consists of five 38 $m^3$ trough flotation cells.
- Cleaner 3 flotation circuit consists of three 38 $m^3$ trough flotation cells.

For smaller cells the higher lip length and cell open area to volume ratios of trough cells, as well as the lower CAPEX per cubic meter, make this style of cell more suited for this application in the Halilağa cleaner flotation circuit than tank cells. Subsequent optimization work may identify better alternate configurations. However, given the information available for this study, this design represents the most economic option. The decision to use the same size cells for the 2nd and 3rd cleaners was based on layout considerations and spare part optimization.

## Regrind Circuit Configuration

Rougher and cleaner-scavenger concentrate report to the regrind hydro-cyclone feed pumpbox. The regrind hydro-cyclones target a cut size of 80% passing 16 µm; this is achieved with a cluster of 18 operating 250 mm hydro-cyclones. The cluster includes 2 spare hydro-cyclones and 3 blank nozzles for future hydro-cyclone addition. The hydro-cyclones operate in open circuit, with the underflow reporting to the M10 000 Isamill. The Isamill discharge is combined with the overflow and reports to the cleaner 1 feed pumpbox.

## Concentrate Thickening and Filtration

Due to the conceptual nature of this study, no concentrate handling test work has been undertaken. Ausenco has based the design on experience with similar style concentrators. These assumptions would require test work and confirmation during subsequent study phases.

Copper flotation concentrate would be thickened to approximately 60% w/w solids in an 18 m diameter above ground high-rate thickener. The thickener is planned to be designed based on a typical settling rate of 0.25 t/m2/h for copper concentrates.

Two copper concentrate storage tanks with live total volumes of 1200 $m^3$ would provide surge capacity ahead of each copper concentrate filter. The specification of 48 h residence time is typical for this size and style of concentrator and would provide surge capacity between the filter and the rest of the flotation plant to account for the lower availability typically achieved by these machines.

The copper concentrate filters have been sized depending on the throughput in the plant, a head grade of 0.8% Cu and a final concentrate grade of 29% Cu. The filters have been sized assuming an availability of 80% while the flotation plant is operating.

The filter size is based on filtration rates determined from previous experience with similar style concentrators, 229 kg/$m^2$/h. For the study, two Larox plate and frame filter press with a 45 mm chamber depth have been selected. The press is supplied with one hydraulic power unit and one control panel. Prior to final selection, it is recommended that test work is conducted by Metso and Larox to accurately evaluate filter sizing and costs.

### 15.1.6 Tailings Disposal

The design basis chosen for this level of study includes a tailings thickener, with disposal of thickened tailings in a nearby TMFand recovery of water from the TMF surface. Due to the low levels of pyrite in the feed, it is assumed that overall the tailings woud be non-acid-generating ("NAG") and separation of pyrite from the tailings stream for disposal in a separate TMF would not be required.

The tailings thickener design is envisioned to be based on settling criterion of 0.85 $t/m^2/h$. This has resulted in the requirement for a 60 m diameter tailings thickener. Thickener underflow would be pumped to the TMF, while thickener overflow would gravity flow to the process water tank.

### 15.1.7 Plant Layout & Description

A layout has been prepared based placing the plant as reasonably close to the pit as possible. The layout for the 50 kt/d plant is shown below in Figure 14.2. The plant layout has taken site topography and permit boundaries into account and works within the bounds imposed by preliminary locations of the pit, stockpiles and waste dumps.

The process includes a gyratory crusher, stockpile conveyor, coarse plant feed rock stockpile, SAG and ball mill grinding circuit, pebble crusher, flotation circuit including regrind, concentrate dewatering, concentrate load-out and tailings thickening facilities.

The concentrator would use a conventional processing flow sheet and industry standard equipment. Concentrator operation would be monitored using a control system from a centrally located control room. Sampling and stream assay monitoring would be via an automated system linked to the control system.

Refer to Appendix B for process flow sheet and general arrangement drawings.

## 15.2 Processing Plant Configuration

The Halilağa process plant and associated service facilities would process 50,000 t/d of ROM material, to produce copper concentrate and tailings. The proposed process includes crushing and grinding of the ROM material, rougher and cleaner flotation, regrinding, and dewatering of copper sulphides. The copper concentrate would be trucked from site to the Port of Bandirma for transport for downstream processing at another facility outside the scope of this report. The flotation tailings would be thickened before placement in the TMF.

### 15.2.1 Primary Crushing

The crushing circuit would be situated next to ROM pad. Mining trucks would haul plant feed material from the OP to the ROM pad. The primary crusher would be located at the edge of the ROM pad. Trucks would dump from both sides to a 400 t capacity live hopper above the crusher; alternately ROM can be re-handled and fed with a front end loader ("FEL") for blending of operational needs. The maximum feed size to the crusher will be 1 m.

### 15.2.2 Stockpile Feed Conveyor and Coarse Plant Feed Material Stockpile

Crusher product would be reclaimed from the discharge hopper by the variable speed primary crusher discharge apron feeder, which would discharge to the sacrificial conveyor, followed by a short conveyor to the mill stockpile. In event of the crushing circuit being out of operation for

extended periods, a bulldozer can be used to reclaim the ‘dead’ material in the stockpile to provide emergency feed to the milling circuit.

### 15.2.3 Stockpile Reclaim and Grinding

The crushed rock would be reclaimed from the plant feed material stockpile by two apron feeders to the SAG mill feed conveyor. Both feeders would be equipped with variable speed drives and would be capable of providing 120% of the design SAG mill feed rate.

The SAG mill feed weightometer will be installed on the SAG mill feed conveyor to provide feed rate for control of the reclaim feeders. The reclaimed crushed plant feed material would be fed at a controlled rate to a 15 MW SAG mill. The SAG mill would be equipped with a twin pinion motor variable speed drive system.

Discharge from the SAG mill would gravity flow through a trommel screen into a pumpbox where it would combine with the two ball mill discharges. Oversized pebbles from the trommel screen would be recycled back onto the mill feed conveyor via dedicated pebble crushing circuit and reintroduced into the mill after having metal magnetically separated and the pebbles crushed to minus 12 mm.

The ball mills (6.10 m diameter by 9.14 m EGL) would be supplied with cast steel liners, 7 MW wound rotor induction motors, liquid resistance starter, trommel screen and retractable feed spout/chute. The combined mills discharge slurry would be pumped to the two mill hydro-cyclone clusters operating in closed circuit configuration with each of the ball mills.

Mill hydro-cyclone underflow would gravity flow to feed each of the two ball mills and hydro-cyclone overflow would gravity flow to a vibrating trash screen ahead of the flotation circuit. Trash from the screen would be removed and trash screen underflow would flow via gravity to the flotation circuit.

Two vertical spindle sump pumps would be provided in the grinding area to facilitate clean up.

### 15.2.4 Flotation and Regrind

#### Rougher Conditioning

Flotation feed would be conditioned in a rougher conditioner tank where flotation reagents wouldbe added. Frother would be added to the conditioner tanks overflow pipes feeding the first rougher flotation cell in parallel rougher trains.

#### Rougher Flotation

The rougher conditioner tank would overflow to the rougher flotation cells connected in series. Six 300 $m^3$ forced air tank cells have been selected to provide the required residence time for the roughing flotation duty. The cells would be arranged with a step in level between each pair of cells.

Concentrate from the rougher cells would gravity flow to a rougher concentrate launder and pumped to the regrind circuit.

Vertical spindle sump pumps would be provided in the rougher flotation area to facilitate clean up.

#### Regrind

Concentrate from the rougher cells would be pumped to a regrind mill to achieve the fine regrind size of 20 µm. Regrinding will be achieved in a 3 MW M10 000 Isamill to a target grind size of 80% passing 20 µm.

Regrind mill media would be 12 mm ceramic balls. The media would be loaded into the regrind media storage bin from 20 t trucks and fed to the regrind mill feed hopper via overhead crane.

A vertical spindle sump pump would be provided in the regrind area to facilitate clean up.

### Cleaner Flotation

Reground concentrate would be mixed with flotation reagents: collector and frother, before being pumped to the cleaner 1 flotation cells. Concentrate from the cleaner 1 flotation cells will flow via gravity to the cleaner 2 feed pump. Tailings from the cleaner 1 flotation cells will gravity flow to the cleaner scavenger flotation cells.

Concentrate from the cleaner scavenger cells will flow via gravity to the cleaner scavenger concentrate hopper and will be pumped to cleaner feed box. Tailings from the cleaner scavenger cells will flow via gravity to the tailings pumpbox.

Concentrate from the cleaner 2 cells will be pumped by the cleaner 3 feed pump to the cleaner 3 flotation cells. Tailings from the cleaner 2 cells would gravity flow to the cleaner 1 distributor box.

Concentrate from the cleaner 3 cells would be pumped to the concentrate thickener feed box. Tailings from the cleaner 3 cells will report to the cleaner 2 flotation circuit.

## 15.2.5 On Stream Analysis & Particle Size Analysis

The On Stream Analysis (“OSA”) system would provide online copper assay analysis.

In addition, on-line Particle Size Analysis (“PSA”) would be performed on the rougher feed slurry sample. Analysed samples, exiting the OSA and PSA systems, would be discharged into sample return hoppers and pumped back to the relevant part of the process.

The OSA and PSA systems would be located at an optimal position which would permit the maximum use of gravity flow of sample feed and sample reject slurries and minimize requirements for sample pumps and hoppers.

## 15.2.6 Thickening and Concentrate Filtration

Copper concentrate from cleaner 3 would be pumped to a high-rate concentrate thickener via the concentrate thickener feed box. Thickener overflow would be pumped to the process water tank for storage and re-use in the circuit, while thickener underflow would be pumped to the copper concentrate filter feed tank. The copper concentrate filter feed tank would have approximately 48 hours storage capacity.

Thickened copper concentrate slurry would be delivered to the copper concentrate filter. The filter press would reduce the moisture content of the concentrate prior to transport. Filter cake would discharge through the floor of the filter building into a concrete area underneath the press. Filter cake would be removed from the bunker by front-end loader (“FEL”) and stored in the covered concentrate storage shed.

The concentrate filter press and concentrate storage stockpile wouldbe housed in a building, which would be fully sheeted on all sides for protection from wind and rain. The concentrate would be stockpiled in the shed by FEL to provide covered on-site storage capacity. Concentrate would be reclaimed by FEL from the stockpile and loaded into 20 t tipper trucks for transport off-site.

The moisture limit has been assumed to be 12% w/w in the filter cake for the filter duty.

### 15.2.7 Raw and Process Water

The water systems will consist of:

- A high pressure/low volume plant raw water system for the mill and rougher flotation areas (mainly reagent mixing and spray water); and
- A low pressure/high volume plant process water system for the mill and rougher flotation areas.

The majority of the plant water requirements would be met from the plant process water system, which would be composed of recycled water streams i.e. thickener overflows and TMF return water, supplemented with raw water as required. Raw water would be sourced from the water storage facility and used for unit processes requiring clean water such as reagent mixing and gland water.

Plant process water would be contained in the plant process water pond. The pond would provide approximately 1.0 hours storage capacity for plant process water requirements at nominal flow rates.

Raw water would be harvested from site runoff, pit dewatering and possibly well sites. Water would be extracted from this Water Storage Facility as required and pumped to the plant. The Water Storage Facility would provide water storage capacity to ensure sufficient water for continued plant operations throughout the entire year.

### 15.2.8 Flotation Tailings

Flotation tailings slurry, consisting of rougher and cleaner scavenger tailings, would flow via gravity to a high rate tailings thickener. Thickener underflow, expected to be in the range of 60%-65% w/w solids would be pumped to the TMF. Process water from the thickener overflow would flow by gravity to the plant process water tank.

A vertical spindle sump pump would be provided within the tailings thickener bunded area to facilitate clean up.

### 15.2.9 Reagents

#### Collector - PAX

Collector would be supplied in pellet form, in bags, by road and offloaded by forklift. Collector would be stored in the reagent storage shed and transferred to the collector mixing tank as required. Batches of collector solution would be made-up on site in the PAX mixing tank. The tank would be filled with raw water and the bulk bag would be lifted onto the bag breaker. The mixing tank agitator would be started to dissolve the pellets to make a 10% w/v solution. On completion of mixing, the solution would be pumped to the PAX storage tank using the transfer pump (one duty).

The PAX solution would be reticulated in a ring main system to the various dosing points using the PAX ring main pump (one duty and one standby). Line pressure would be maintained by an automated diaphragm valve. Dosing to each point would be by flow control valve and flow meter.

#### Frother – MIBC

Frother would be supplied in 1000 L isotainers by road and offloaded by forklift. Frother would be stored in the reagent storage shed and transferred to the frother storage tank by feed pump (one duty) as required. The tank will have capacity for approximately 7 days consumption at design flow rates. The frother would be distributed to the flotation circuit dosing points via a ring main to an elevated header tank with tank overflow returning via gravity to the main distribution tank. A manifold

with multiple dosing pumps (each dedicated to a single dosing point) will meter the frother addition to each dosing point.

### Collector – W34

Collector would be supplied in 1000 L isotainers by road and offloaded by forklift. Collector would be stored in the reagent storage shed and transferred to the collector storage tank by feed pump (one duty) as required. The tank will have capacity for approximately 7 days consumption at design flow rates. The collector would be distributed to the flotation circuit dosing points via a ring main to an elevated header tank with tank overflow returning via gravity to the main distribution tank. A manifold with multiple dosing pumps (each dedicated to a single dosing point) will meter the collector addition to each dosing point.

### Flocculant

One flocculent mixing, storage and dosing system would be provided. Flocculant powder would be delivered in bulk bags and loaded into hoppers. Dry powder flocculant would be mixed with raw water to make a 0.25% w/v solution in a package flocculant mixing system. The mixed flocculant solution would be pumped to a storage tank with 12 h capacity at design flow rates. Flocculant solution would be dosed to the each thickener by dedicated variable speed helical rotor pumps. Process water would be mixed into the flocculant lines to dilute solution to 0.025% w/v before addition to the thickener feed slurry.

### Hydrated Lime

Powdered hydrated lime would be delivered to site in 20 t bulk tankers. The lime would be pneumatically offloaded into a 200 m3 lime silo.

The lime powder would be fed by a variable speed rotary feeder to the mixing tank where water is added. Mixed “milk-of-lime” slurry would be transferred to the lime slurry storage tank with 24 h capacity. The lime slurry would be reticulated in a ring main system to various user points using the lime ring main pump (one duty and one standby). Line pressure would be maintained by a manually operated diaphragm valve. Dosing to each point would be by timer operated pulsing diaphragm valve.

### Grinding Media

Forged carbon steel grinding media are delivered to site in 20 t containers. The balls are unloaded into a storage hopper via a vendor-supplied hydraulically operated container unloader.

Overhead cranes in the primary milling and regrind areas would be used to charge steel and ceramic balls into the SAG, ball and regrind mills. Balls would be transported from the SAG, ball or regrind mill grinding media storage bunker in the storage yard by FEL to the grinding media hoppers located near the mill feed end. From the hoppers, balls would be added to a bottom discharging kibble and hoisted using the overhead crane to a position above the mill feed chute and the contents emptied into the mill.

## 15.2.10 Ancillary Services

### Low Pressure Air

Low pressure air at a suitable supply pressure would be supplied to the flotation cells from the flotation blowers. There would be two blowers per grinding train (one duty and one standby) installed

to meet the flotation air requirements. Pressure control valves would be installed in the air distribution lines to cater for different air pressure requirements of different flotation cells. The blowers would be multiple stage centrifugal type equipped with variable speed drives to cater for fluctuations in flotation air demand.

The blowers would be housed inside an acoustic enclosure to reduce noise to an acceptable level. The enclosure will have ventilation for cooling.

### High Pressure Air

High pressure air for plant and instrument air requirements would be provided by four rotary screw air compressors. There would be three duty compressors and one standby compressor operating in lead-lag mode. Plant air would be stored in the plant air receiver prior to being reticulated throughout the plant.

### Plant Buildings

A number of plant buildings would be required for operation and maintenance of the plant. These include administration office, laboratory, plant workshop, warehouse, reagent stores, guard house, and plant control room and ablutions.

The administration building will consist of a number of internal offices and meeting rooms. A first aid facility would be provided near to this building. The warehouse will house mechanical, electrical, instrumentation and general items in discrete areas. The warehouse structure would be adjacent to the plant maintenance workshop. Internal offices would be supplied for warehouse and maintenance staff.

The plant control room would be located at an elevated position adjacent to the hydro-cyclone clusters and would be provided with aluminum framed windows for viewing out onto the process plant. In particular, stockpile, grinding, flotation and thickening areas can be easily viewed from the control room. External windows shall be double glazed with tinted glass.

## 15.2.11 Process Control Philosophy

The control philosophy to be implemented for the Halilağa Project is typical of those used in modern mineral processing operations.

Field instruments provide inputs to a set of Programmable Logic Controllers (“PLCs”). Process control cubicles are located in the Motor Control Centres (“MCCs”) and contain the PLC hardware, power supplies and I/O cards for instrument monitoring and loop control.

The PLCs perform the control functions by:

- Collecting status information of drives, instruments and packaged equipment;
- Providing drive control and process interlocking; and
- Providing PID (proportional-integral-derivative) control for process control loops.

Standard Personal Computers (“SPC”) located in the Main Control Room (“MCR”) and the Crusher Control Room (“CCR”). The PCs are networked to the PLCs and operate a Supervisory Control and Data Acquisition (“SCADA”) program that provides an interface to the PLCs for control and monitoring of the plant.

The SCADA is configured to provide outputs to alarms, control the function of process.

# 16 Project Infrastructure

## 16.1 Access and Existing Infrastructure

The Halilağa Project is located 40 km southeast of the city of Çanakkale between the villages of Halilağa and Muratlar in Çanakkale Province in Northwestern Turkey (Figure 4.1). Çanakkale Province comprises an area approximately 9,700 km$^2$ around the Dardanelles straits, which connect the Sea of Marmara and the Aegean Sea and has a population of approximately 500,000. The city of Çanakkale is the provincial capital and has a district population of 137,000. Çanakkale has an airport with daily flights to and from Istanbul and Ankara. Well maintained, paved roads provide excellent access from main regional highways to the project site.

The town of Çan is a main industrial centre in the project region with a producing coal mine, a nearby 50 MW power station, and a population of 30,000. It is located approximately 28 km by road to the northeast of the project. A main regional highway, D210, connects the city of Çanakkale with the town of Çan, a distance of 72 km. The project site is located approximately 15 km from this main highway corridor, and is just off of a main regional paved road.

Other major centres in Çanakkale Province are Biga with a population of 37,000 to the northeast and Ezine, Ayvacik and Bayramiç to the southwest with an area population of approximately 100,000. To the northeast from Çan the regional and state highway network connects through Biga to the city of Bandirma, a major regional port facility with railway, highway and ferry service to Istanbul.

## 16.2 Road Access

The project plant site is located approximately 1.5 km from the main regional road connecting the town of Çan to the northeast with the Çanakkale-Izmir highway to the southwest. An existing forestry road provides access to the project site from the paved regional road. It is anticipated that this general routing would be used for site access. The upgrading of this access road from the regional road is included in allowances for site civil infrastructure. Further road accesses would be upgraded or built to provide access from the mine to the plant and to the waste dumps, TMF and water reservoir.

## 16.3 Port Facilities

Three different port facilities were considered for this study for the export of copper concentrate:

- Çanakkale, located 66 km by road to the Northwest of the project site;
- Bandirma, located 135 km by road to the Northeast of the project site; and
- Izmir, located approximately 300 km by road to the South of the project site.

The pier at Çanakkale is the closest port facility to the project and is located in the Kepez district at the southern end of the city of Çanakkale. Çanakkale Kepez is a new 214 m long pier (built in 2005) with two berths and capacity for bulk materials, general cargo and passengers. It has open storage facilities and 3,000 m² of enclosed storage space. It is connected to the main state highway by a 2-km private road. Significant new facilities would need to be built for concentrate storage and ship loading.

The port of Bandirma is a large multi-use port facility with 20 berths. The port is situated on the south coast of the Sea of Marmara in the city of Bandirma with a population of 135,000. It is a major hub

for exports/imports of dry bulks in the Marmara Region. The port has modern infrastructure and is connected to the Turkish railway and highway network. It has a bulk storage area for 165,000 t and a bulk warehouse storage area for 35,000 t. Bulk materials shipped out from the port include various concentrates, acid and marble. Four berths are dedicated to dry bulk materials handling and it has a stated capacity of 10 Mt/a for dry bulk materials and has averaged 3 Mt/a in past years. Although new facilities may be required for concentrate storage and handling, it is expected that Bandirma is reasonably well equipped to handle the export of copper concentrate without significant expansion.

The port of Izmir is situated at the east end of Izmir Bay directly adjacent to the Aegean Sea. It is Turkey's third largest port and is located in the city of Izmir with a metropolitan population of 3.95 million. Although Izmir has some concentrate handling facilities and is considered a possible port for exporting concentrate from the project, it is mainly a container and passenger port.

For the purposes of this study, Bandirma was considered the most reasonable choice as a port for exporting copper concentrate, as well as for importing supplies from outside of Turkey. Further study is recommended for the pre-feasibility stage to determine the best port facility and to better assess the associated costs with using the port.

## 16.4 Electrical Power Supply

### 16.4.1 Existing Power Line

There is an existing 154 kV power transmission line owned by the Turkish Electricity Transmission Corporation (“TEIAS”) that runs through the project site, and would cross the planned OP. This high voltage line transmits electrical power from the Çan area to the towns of Ezine, Ayvacik and Bayramiç which are 30-60 km to the west-southwest of the project site.

It is possible that this 154 kV line could be used to provide power to the plant, but this is complicated by a number of factors. Part of the issue is that a private company, Uludağ Elektrik Dağıtım A.Ş (“UEDAS”) is the sole authorized provider allowed to sell power in Çanakkale Province and it is understood that the project would have to secure power from them. UEDAS owns the 50 MW power plant near Çan, but 39 MW is already being used. Pilot Gold is currently investigating the best solution for tying in to the regional electrical grid through discussion with TEIAS, UEDAS and local contractors.

Currently it is believed that this existing 154 kV line will have to be re-routed around the site and it has been determined by SRK-Ausneco and a local electrical contractor that is better to move this line to the south into Forestry land than to move it to the north into farm land which would require individual land owner agreement. It is considered more efficient from a time and cost perspective to move it to the south. A preliminary re-routing was determined by SRK-Ausenco which requires 10.5 km of new line to avoid the OP and waste management facilities.

### 16.4.2 New Power Line

Ausenco estimates that the site power demands would be approximately 50 MW. It is currently understood that a new power line would be required to bring power to the project site from either Çan or Etili, about 10 km from the project site. There is currently not enough spare electrical power generated in the region, however there is a possibility that the existing 50 MW power plant at Çan could be expanded to 100 MW.

Based on the information available at this time, Ausenco has assumed that a new 154 kV transmission line from the Çan area would be required, and it would be approximately 25 km in length and will include step down transformers. Further study is recommended for the pre-feasibility stage to determine the best strategy for tie-in to the regional electrical power grid.

## 16.5 Water Supply Reservoir

### 16.5.1 Preliminary Design Assumptions

A preliminary site wide water balance suggest that there would be a shortfall of fresh water to support milling operations of about 400,000 $m^3$ per month during the dry season (April through October). This translates into an annual demand of 2.8 $Mm^3$. SRK completed a preliminary water balance and confirmed that a surface catchment of at least 1,750 ha (17.5 $km^2$) is required to supply this amount of water on a year-round basis.

### 16.5.2 Reservoir Alternatives Assessment

#### Alternate Reservoir Dam Technology

Alternate dam construction technologies considered for the water reservoir include:

- Conventional low permeability core earth fill dam;
- Lined or concrete faced rock fill dam; and
- Concrete gravity or gravity arch dam.

Concrete gravity or gravity arch dams are typically the most cost intensive dams and are not normally considered for relatively short-lived mining projects. For this reason this technology was not considered further.

Just as described for the tailings dams, the rugged terrain necessitates low efficiency impoundments (i.e. ratio of dam volume relative to the amount of water retained is low). As a result sourcing of suitable construction materials would be a key factor in determining a suitable dam construction technology for the reservoir.

A conventional low permeability core earth fill dam require availability of multiple types of construction materials, each of which with tight tolerance on their allowable specification. No borrow source investigation has been carried out to date and therefore the preferred reservoir dam is a lined rock fill dam. The type of liner is a geosynthetic product rather than concrete for simplicity.

#### Alternate Reservoir Sites

The search for a suitable water reservoir site was arbitrarily limited to an area within a 10 km radius from the proposed mill location and was focussed as far as practical to find a location on land for which Truva Bakir holds either mining or exploration rights.

Two approaches to finding a suitable reservoir site were adopted:

- Identifying sites that have sufficient upstream catchment to ensure supply of the required water volume, and
- Identifying sites which had smaller catchments, where a number of smaller dams could be used to supply the required volume.

Three candidate reservoir sites, each capable of supplying the entire design volume was identified as summarized in Table 16.1. The location of potential multiple smaller water collection points were also identified. It was however decided not to consider this alternative as the cost of multiple pumps and pipelines would be high and would pose many operational challenges.

**Table 16.1: Water Reservoir Alternative Site Selection**

| Alt. | Dam Height (m) | Dam Volume (m3) | Footprint (Ha) | Catchment Area (Ha) | Comments |
|---|---|---|---|---|---|
| 1 | 30 | 345,761 | 25 | 1,770 | This alternative will barely meet the capture requirements to sustain enough water for the mill through the summer months<br>The footprint area overlaps with tailings areas 4 and 5+6 |
| 2 | 18 | 113,905 | 36 | 4,487 | This alternative has the smallest dam volume, but covers the largest area<br>Most of this area is outside current property boundaries |
| 3 | 22 | 188,561 | 25 | 5,550 | This alternative is primarily within the current property boundaries<br>Roughly 80% of the footprint covers current farmland |

## Preferred Water Reservoir

The preliminary water reservoir alternatives assessment presented in this report confirmed the following:

- Factors that contribute towards selection of the preferred water reservoir location include the fact that the site lies predominantly on land controlled by the JV, has sufficient catchment yield to supply the design volume of water, and is moderately effective in terms of the size of the dam required; and
- The preferred dam structure design is a bulk rock fill dam with an upstream geosynthetic liner, and a permanent side-hill spillway.

The preferred water reservoir spans a total footprint of 21.9 ha. 100% of this surface area lies within existing property boundaries under control of the JV. Farmland currently covers about 80% of the footprint area.

Environmental and socio-economic criteria have only been considered in broad terms in this alternatives analysis, and it is therefore biased towards technical and economic criteria. As the project advances and more information become available about baseline conditions and potential environmental effects, the analysis should be revisited to confirm that the identified preferred options remain valid.

Environmental and social information along with stakeholder engagement will be considered in all related decisions moving forward.

## 16.5.3 Water Reservoir Design

### Dam

The water reservoir dam is a 20.5 m high (El. 171 m), 255 m long rock fill structure with upstream and downstream slopes of 2.5H:1V and 3H:1V respectively. The dam has a crest width of 10 m and would be about 123 m wide at its widest point along its base. The total freeboard allowance is 2 m (note no hydraulic calculations have been done in support of this assumption).

An upstream geosynthetic liner (assumed to be a high grade bituminous liner or HDPE) provides primary water containment. This liner is keyed to a 3 m wide grout curtain extending about 21 m below ground. Note that no geotechnical investigations have been carried out to date to confirm foundations conditions and as a result assumptions on the key trench excavation and grout curtain requirements are simply allowances.

The bulk rock fill would be geochemically suitable run-of-quarry material from a locally developed rock quarry (assumed to be within 5 km haul distance from the structure). The liner would be placed on a 30 cm thick layer of bedding material assumed to be crushed (20 mm minus) rock or other suitable borrow material. A 2 m thick layer of transition material (150 mm minus material acting as a filter media) separates the bedding and run-of-quarry material. The liner would be covered with a 0.6 m thick layer of bedding material followed by a 1 m thick layer of rip-rap to protect the dam against wave run-up.

The dam has a key trench assumed to be 10% of the dam height and 1/3 of the dam section in length and would be backfilled with transition material or run-of-quarry material as access permits. The key-trench would be backfilled with transition material. A typical cross section of the dam is presented in Figure 16.1.

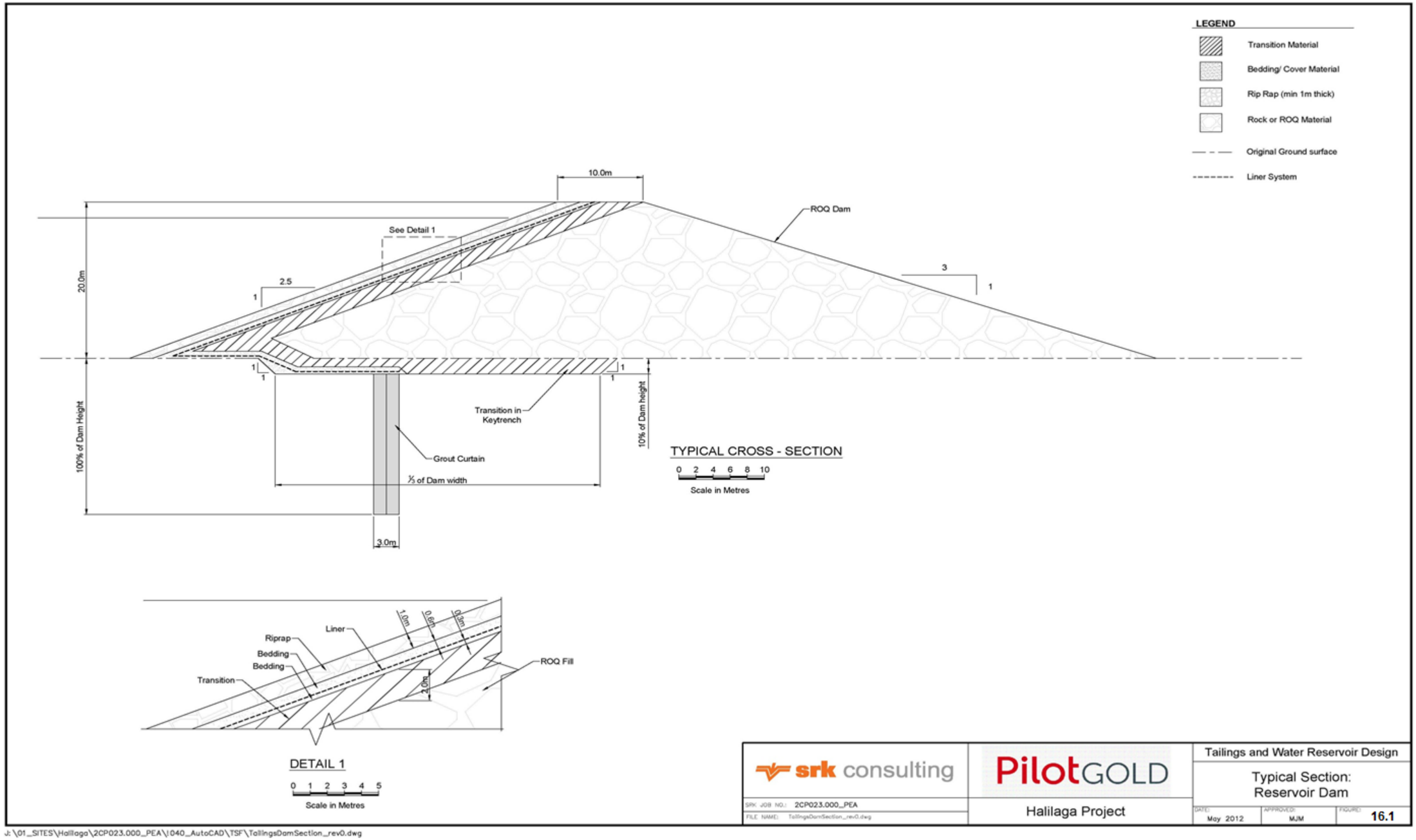



**Figure 16.1: Typical Section: Water Reservoir Dam**

### Spillway

A permanent armoured spillway capable of at least the 1:100 year, 24-hour duration storm event would be constructed, preferably as a side-hill structure. Details of this have not been developed at this time.

It should be noted that a hazard classification for the dam has not been carried out. The flood magnitude for which the spillway must be sized would be determined by this classification and may increase the proposed spillway substantially.

### Water Management

All overland surface water from the upstream catchment (excluding that of the TMF) will flow unhindered into the reservoir. A pump station would be constructed near the dam wall from where water would be pumped to the mill via a 8.4 km long 350 mm diameter HDPE pipeline. The pipeline will follow the same route at the access road from the mill to the reservoir.

The total static head against which this water must be pumped is about 200 m, and the pipeline has been sized to supply the design volume to the mill at a rate of 13,600 $m^3$/day.

Water quality within the reservoir is not expected to exceed any licence or environmental criteria and therefore the pump station and pipeline will have no special environmental containment considerations.

### Closure and Reclamation

When mining ceases the water reservoir would be drained and the dam breached and ultimately completely removed. The area would be reclaimed and would be returned for use as farmland.

Alternatively, should there be interest, there may be an alternate opportunity for local government authorities to take over the reservoir and retain its use for the region.

## 16.6 Tailings Management Facility

### 16.6.1 Preliminary Design Assumptions

About 240 Mt of tailings (50,000 tpd) would be produced as part of the 14 year (LOM) plan from a single OP. For volumetric calculations it was decided to use the most conservative estimates for tailings density (i.e. requiring the largest footprint) which would be attained when using conventional low solids content slurry. Physical testing has not been carried out on any of the tailings but based on experience with similar tailings an in place density of 1.4 t/$m^3$ was used. This translates into a design volumes of 171 $Mm^3$ of tailings.

Important facts considered in the analysis to date include:

- The site is in a seismically active area and therefore long-term integrity of any retaining structures would be critical;
- No geotechnical and/or hydrogeological testing have been carried out to confirm foundation conditions at candidate sites;
- The site has a net negative water balance and therefore maximizing water recirculation at the mill is critical;
- The site has significant relief ranging from 100 to 750 m;

- Truva Bakir holds a number of disjointed mining and exploration leases/permits with private and government land in between. Potential TMF sites located outside of Truva Bakir control is not desirable.
- The project site is surrounded with active farmland and villages. Placement of TMF facilities in relation to these elements poses significant challenges;
- The rugged terrain, land use and land ownership issues limit to a large extent where a TMF can be constructed;
- There is competing interest for available surface area from deposition of waste rock. Since project economics is more sensitive to waste rock location than tailings, waste rock gets preference in siting decisions; and
- Tailings geochemical testing has not been carried out. The relatively low sulphide content of the mineralized rock suggests that acid generation and/or metal leaching may not be a concern. This must be confirmed by further testwork.

### 16.6.2 Tailings Manage Facility Alternative Assessment

#### Alternative Tailings Deposition Methods

Alternate deposition methods considered for the project included:

- Conventional low solids content (typically less than 30%) slurry;
- Thickened tailings slurry (solids content typically between 30% and 60%), but still pumpable with centrifugal pumps;
- Paste tailings (solids content typically in excess of 60%) requiring positive displacement pumps; and
- Filtered (i.e. dry-stack tailings) tailings.

Paste and filtered tailings were not considered for further evaluation at this time, primarily due concerns with capital and OPEX associated with the large tonnage operation. It should however be noted that given many of the uncertainties listed in this analysis, these deposition strategies may have to be revisited at a later stage.

The preferred tailings deposition method is therefore thickened slurry tailings and the preliminary processing flow sheet recommends a solid content of 60%. Rheological testing has not yet been carried out but it is expected that this solids content will allow maximum water recovery at the mill, while still allowing the slurry to be pumped using conventional centrifugal pumps.

Two alternate placement strategies considered (irrespective of the tailing stream) for the slurry tailings include:

- Sub-aqueous deposition; and
- Sub-aerial deposition.

Given the seismic activity of the site and the net negative water balance the preferred tailings placement strategy is sub-aerial deposition.

## Alternative Tailings Management Facility Retaining Structure Technology

Hydraulically placed sub-aerial slurry tailings require construction of retaining structures. The alternative retaining structure technologies considered included:

- Conventional low permeability core earth fill dam;
- Lined rock fill dam;
- Cycloned tailings following earth fill starter dam (using either of the above two designs); and
- In-pit deposition (mined out pen pit).

In-pit disposal in a disused open pit was considered but dismissed due to the current mine development sequence which does not lend itself to in-pit disposal.

The rugged terrain necessitates low efficiency impoundments (i.e. ratio of retaining structure volume to retained tailings capacity is low). The required retaining structures are therefore large and without using cyclone dams multiple tailings deposition sites would be required for the project. Although the unknown geochemistry, seismicity and rapid Rate-of-Rise (ROR) of the facility are concerns, the preferred retaining structure technology at this stage has been set as a starter dam followed by cycloning.

A conventional low permeability core earth fill starter dam require availability of multiple types of construction materials, each of which with tight tolerance on their allowable specification. No borrow source investigation has been carried out to date and therefore the preferred starter dam is a lined rock fill dam.

In order to minimize pre-production capital the tailings retaining structures would be continuously raised over the LOM. Alternate dam raising strategies considered included:

- Upstream raising;
- Centerline raising; and
- Downstream raising.

An upstream raising approach was not considered due to stability concerns given the area is seismically active, and the use of cyclone tailings. Downstream raising is the most capital intensive option, and therefore a centerline raising strategy has been adopted.

## Alternative Tailings Deposition Sites

The search for a suitable tailings site was limited to an area within a 10 km radius from the proposed mill location. Considering the significant topographical relief associated with the site this distance was considered to be the maximum practical distance before pumping tailings may become cost prohibitive.

The site selection was carried out in three stages:

- Primary site screening assessment undertaken during the site visit;
- Secondary site screening considering earth fill retaining structures; and
- Tertiary site selection considering earth fill starter dams followed by cyclone tailings raises.

Site selection for all stages was done using topographical and cadastral mapping supplied by Pilot and imported into Global Mapper 12. Secondary site screening volumetric take-offs was done assuming dams with upstream and downstream slopes of 2H:1V and a 10 m wide crest. Tertiary

volumetric take-offs was generated assuming starter dams with an upstream slope of 2.5H:1V, and a downstream slope of 3H:1V. The downstream cyclone underflow slope is 5H:1V.

A total of five sites were evaluated during the primary screening assessment and a further nine during the secondary screening stage.

Pertinent information associated with the sites that were evaluated during the secondary screening stage is listed in Table 16.2.

**Table 16.2: Secondary Tailings Management Facility Site Screening Alternatives**

| Alternative | Dam Height (m) | Storage Capacity ($Mm^3$) | Storage Efficiency | Land Use/Tenure Considerations |
|---|---|---|---|---|
| 1 | 95 | 25.17 | 5.2 | Partially outside current tenure<br>About half of footprint currently used for farming<br>Osmaniye village under the dam footprint |
| 2a | 90 | 16.5 | 3.2 | Forested area with no farmland or villages<br>Footprint almost entirely within current tenure<br>Closest to deposit |
| 2b | 100 | 24.7 | 3.1 | |
| 3 | 155 | 68.7 | 2.5 | Forested area with no farmland or villages<br>Footprint entirely within current tenure<br>4.5 km from proposed mill |
| 4a | 90 | 16.9 | 3.1 | Forested area with no villages and minimal farmland<br>7.5 km from proposed mill |
| 4b | 130 | 46.1 | 2.9 | |
| 4c | 120 | 36.3 | 3.1 | |
| 5 | 65 | 8.8 | 4.6 | Mostly in forested area, with few small sections of farmland<br>Footprint extends beyond current tenure |
| 6 | 95 | 45.8 | 7.5 | Roughly 1/3 covers existing farmland<br>Some roads will requiring re-routing<br>8.2 km from proposed mill |

The primary outcome of the secondary site screening confirmed that unless cyclone dams were to be used multiple tailings disposal sites would be required for the project. The sites considered in this analysis have all been optimized to store 10% above the design capacity 171 $Mm^3$, i.e. 190 $Mm^3$ (264 Mt) and are summarized in Table 16.3.

**Table 16.3: Tertiary Tailings Management Facility Site Assessment**

| Alternative | Final Cyclone Dam Height (m) | Starter Dam Volume ($Mm^3$) | Underflow Ratio | Land Use/Tenure Considerations |
|---|---|---|---|---|
| 3 | 160 | 7.8 | 0.53 | Toe of dam extends onto farmland<br>Underflow ratio is very high and likely unachievable |
| 4 | 145 | 3.8 | 0.45 | Toe of dam extends onto farmland<br>Underflow ratio is very high and likely unachievable |
| 5+6 | 110 | 2.4 | 0.15 | Covers some farmland and transportation infrastructure<br>Low dam volume and underflow ratio make this alternative viable |

### Preferred Tailings Management Facility

The preliminary tailings alternatives assessment presented in this memo confirmed the following:

- The preferred tailings disposal facility location is Site 5 and 6 combined. Factors that contribute towards this decision include the fact that this site lies predominantly on land controlled by Truva Bakir.
- Truva Bakir has sufficient capacity to contain the LOM tailings, requires the smallest starter dam, and requires the least amount of underflow tailings;
- The preferred tailings disposal method is thickened (60% solids) slurry tailings, which allows for maximum water recovery at the mill while remaining pumpable with conventional centrifugal pumps;
- Sub-aerial deposition would be the preferred tailings placement technology;
- Centerline dam raises are the preferred dam raising technology; and
- The preferred retaining structure design is a bulk rock fill starter dam with an upstream geosynthetic liner. The remainder of the structure would be a cycloned impoundment.

The preferred TMF spans a total footprint of 475 ha. 89% of this surface area lies within existing property boundaries under the control of Truva Bakir. The farmland, both on and off of the property is under the control of Truva Bakir and covers 20% of the proposed tailings footprint area.

It should be noted that environmental and socio-economic criteria have only been considered in broad terms in this alternatives analysis, and it is therefore biased towards technical and economic criteria. As the project advances and more information become available about baseline conditions and potential environmental effects, the analysis should be revisited to confirm that the identified preferred options remain valid.

## 16.6.3 Tailings Management Facility Design

### Starter Dam

The 565 m long, 55.5 m high (El. 255 m) tailings starter dam has a design capacity of 20.4 Mm3 which is equivalent to one year of tailings deposition at a processing rate of 50,000 tpd, assuming 20% cyclone underflow.

The dam is a rock fill structure with upstream and downstream slopes of 2.5H:1V and 3H:1V respectively. The dam has a crest width of 10 m and would be about 314 m wide at its widest point along its base. There is no freeboard allowance (that is catered for in subsequent raises).

An upstream geosynthetic liner (assumed to be a high grade bituminous liner or HDPE) provides primary water containment. This liner is keyed to a 3 m wide grout curtain extending about 21 m below ground. Note that no geotechnical investigations have been carried out to date to confirm foundations conditions and as a result assumptions on the key trench excavation and grout curtain requirements are simply allowances.

The bulk rock fill would be geochemically suitable run-of-quarry material from a locally developed rock quarry (assumed to be within 5 km haul distance from the structure). The liner would be placed on a 30 cm thick layer of bedding material assumed to be crushed (20 mm minus) rock or other suitable borrow material. A 2m thick layer of transition material (150 mm minus material acting as a filter media) separates the bedding and run-of-quarry material. The liner will not be covered. The dam has a key trench assumed to be 10% of the dam height and 1/3 of the dam section in length and would be backfilled with transition material or run-of-quarry material as access permits.

The key-trench would be backfilled with transition material. A typical cross section of the dam is presented in Figure 16.2.

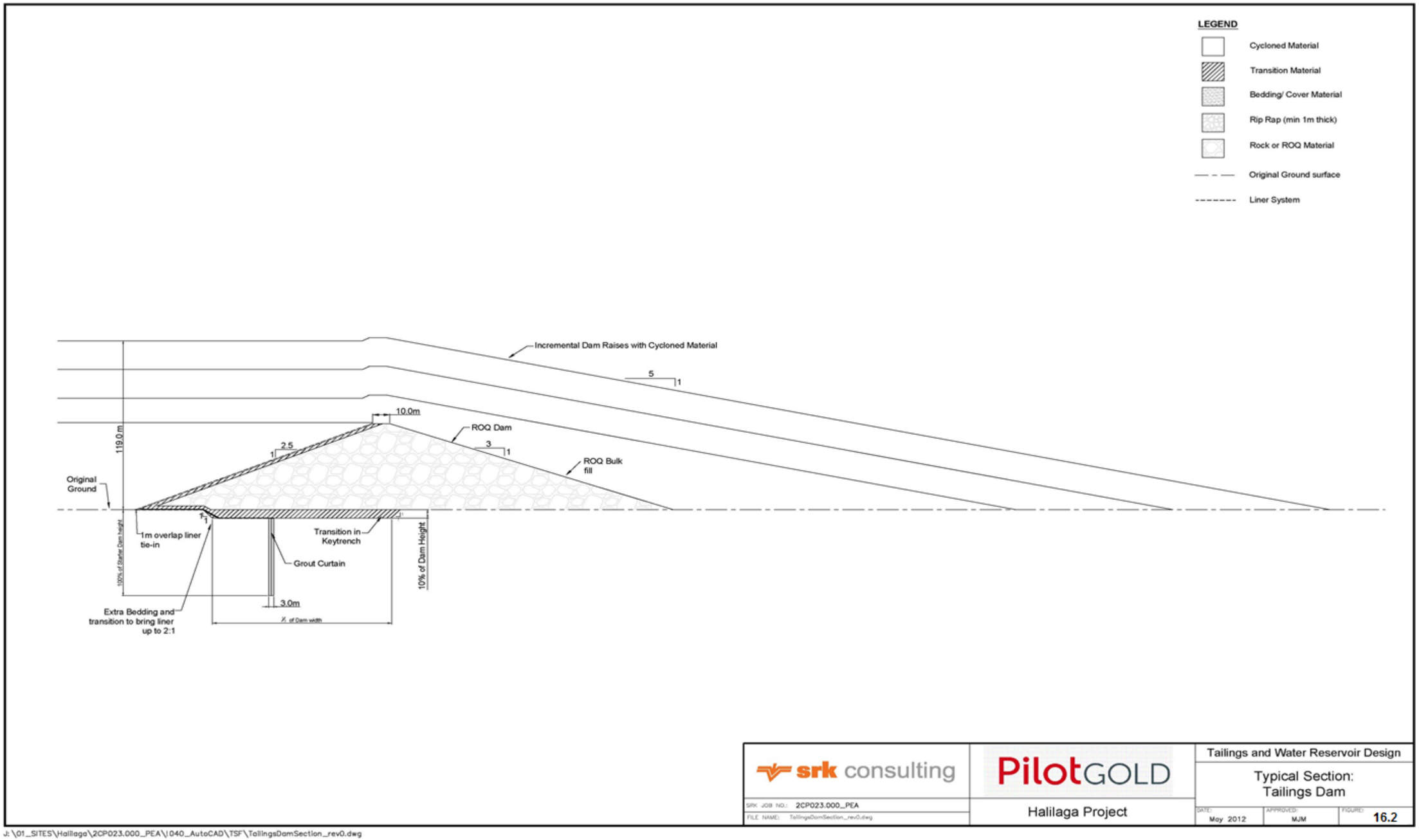



**Figure 16.2: Typical Section: Tailings Dam**

## Secondary Containment Berm (Starter Berm)

The TMF consists of two parallel valleys, but at different elevations. A secondary starter dam may be required contingent on geotechnical investigations; however, at this time it has been assumed that a basic starter berm could be used to initiate cyclone deposition.

The starter berm would be about 3 m high, 355 m long. The berm will generally follow the ground topography as it is simply intended to stage the cyclones from. The starter berm will however have a lined key trench and a liner similar to the starter dam to ensure environmental containment.

## Cyclone Raises

The dam would be continuously raised using cyclones as illustrated in Figure 16.2. About 20% of underflow tailings is required to construct the compacted 5H:1V downstream slope of the embankment with an expected loose density of about 1.6 t/m$^3$. Compaction would be attained by using a dozer to spread the tailings into uniform lifts before passing over it with a vibratory drum compactor. The final height of the cyclone dam would be about 110 m which includes a final freeboard height of 2 m. The total crest length would be about 2,000 m. This translates into a rate-of-rise (ROR) for the impoundment of 9.5 m/year. Ideally the ROR for cyclone dams should be less than 5 m/year rate to reduce risks associated with buildup of pore water pressure. As detailed engineering advances and tailings physical property testing becomes available this aspect will have to be carefully evaluated to determine whether cyclone dam construction would remain feasible.

## Deposition Plan

Tailings with a solid content of 60% would be pumped to the TMF via a 8.2 km long, 700 mm diameter steel pipeline. The pipeline will follow the same route as the proposed access road to the TMF. The TMF is down gradient from the mill.

The slurry would be directed to 10 cyclones from where the assumed 20% underflow volume gets directed towards the downstream shell of the dam for use in construction and the remaining overflow is impounded behind the dam. The assumed density of the overflow is about 1.2 kg/m$^3$.

## Water Management

The TMF is constructed close to the headwater of the catchment and therefore no clean water diversion ditches would be constructed. A permanent pond would be developed at least 100 m upstream of the dam wall. A reclaim system would be put in place and recovered water would be pumped back to the mill for reuse.

## Closure & Reclamation

On closure a permanent spillway would be constructed to redirect water through the facility. Depending on environmental considerations the tailings surface will either be directly re-vegetated or alternately covered and re-vegetated as required. A partial pond may persist in perpetuity to allow attenuation of flows.

The downstream slope of the dam would be stabilized against erosion which may include a cover and vegetation. Specific details pertaining to the treatment would be based on environmental considerations.

## 16.7 Waste Rock Facilities

The WRFs are planned to be located adjacent to the final pit limits for the Halilağa deposit. The East WRF is the larger of the two facilities planned, which also include the smaller West WRF. Given the deposit configuration and extraction sequence, no backfilling into previously mined out areas is planned for Halilağa.

The two WRF's would be built in a series of lifts in a "bottom-up" approach in order to maximize stability. The WRFs would be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with safety berms spaced at regular intervals giving an overall operational slope of 2:1.

Table 16.4 summarizes the waste emplacement volumes during the mining operations in the various WRFs.

**Table 16.4: Design capacities of the various WRFs**

| Waste Rock Facility | Design capacity (Mt) |
|---|---|
| West WRF | 31 |
| East WRF | 203 |
| **Total** | 234 |

Table 16.5 provides the annual sequential development of the various WRFs during the mining operation.

**Table 16.5: Open Pit Production Waste Schedule**

| Waste Rock Facility | Unit | Year | | | | | | | | | | | | | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | |
| West WRF | Mt | 10.1 | 9.6 | 11.0 | | | | | | | | | | | | **30.7** |
| East WRF | Mt | | | 9.0 | 24.0 | 22.8 | 22.4 | 22.8 | 23.3 | 23.7 | 24.0 | 23.5 | 6.4 | | | **201.9** |
| **Total** | **Mt** | **10.1** | **9.6** | **21.4** | **24.0** | **22.8** | **22.4** | **22.8** | **23.3** | **23.7** | **24.0** | **23.5** | **6.4** | **0.0** | **0.0** | **234.0** |

# 17 Market Studies and Contracts

## 17.1 Market Studies

Halilağa concentrate is projected to contain an average of 30% copper and 22 g/t gold. Concentrate production is envisioned to average 140,000 dmt per year over the life of the project but annual variances from 377,000 dmt in the first year to 80,000 dmt in later years is planned. Increased capital was planned to deal with higher mill head grades and concentrate production in years 1 and 2. Gold production is planned to average about 90,000 ounces per annum.

It was assumed that Halilağa concentrate would be marketed to international smelters in Asia, Europe as there is limited smelter capacity in Turkey. It was assumed that the concentrate would not contain deleterious elements at a concentration that would trigger penalties.

A market study was not conducted for the Halilağa concentrate, instead, general industry instead smelter terms were selected based upon general industry experience and by guidance for CostMine 2011 information. The smelter term assumptions used in the economic analysis are shown in Table 17.1.

**Table 17.1: Assumed Concentrate Treatment and Refining Charges**

| Parameter | Unit | Estimated Value |
|---|---|---|
| Treatment charge | $/dmt Cu concentrate | 75 |
| Copper refining charge | $/lb payable Cu | 0.075 |
| Copper payable | % | 96 |
| Copper deduction | % (minimum only – not applied as covered by payable above)) | 1 |
| Gold refining charge | $/oz payable Au | 7.00 |
| Gold payable | % of payable Au | 90 |
| Gold deduction | g/t | 1 |

## 17.2 Metal Price Assumptions

The authors of this report are not specialists in the prediction of metal prices. Due to the high volatility of metal prices in the past years it was decided to use three metal price scenarios in the economic analyses to show the varying impact of different metal prices. The prices selected and their justifications are shown in Table 17.2.

**Table 17.2: Metal Price Assumptions by Case**

| Case | Copper Price ($/lb) | Gold Price ($/oz) | Comments |
|---|---|---|---|
| Case A | 2.90 | 1,200 | Approximate metal prices as per recent SEDAR postings. This is the base case used for OP optimization. |
| Case B | 3.30 | 1,350 | Approximate London Metal Exchange fixed 3-year average to the end of April, 2012. SRK considers this to be an optimistic case. |
| Case C | 3.70 | 1,500 | Approximate London Metal Exchange fixed 2-year average to the end of April, 2012. SRK considers this to be a very optimistic case. |

## 17.3 Contracts

There are currently no established contracts relating to mining, concentrating, smelting, refining, transportation, handling, forward sales or hedging.

# 18 Environmental Studies, Permitting and Social or Community Impact

Ongoing environmental studies are being conducted on the project with the assistance of SRK Danışmanlık ve Mühendislik A.Ş.

Environmental Law No. 2872 and Environmental Impact Assessment Regulation and Labour Law No. 4857 are the two principle regulations governing mining activities; there are additional regulations, based on those principle regulations, which are applicable to mining industry.

Further to the amendments made in the Mining Law, most of the mining activities are excluded from the EIA, but for those mining activities that require an EIA report, the necessary procedures must be completed within three months.

The issue of the allocation of environmentally sensitive areas such as forest areas, hunting areas, special protection areas, national parks, agricultural areas, including culturally-protected areas, coastal areas, tourism regions, and military forbidden zones the granting of licenses for activities to be carried out in such areas are specifically regulated. In order to reduce the bureaucracy in the granting of mining rights, the General Directorate is given the authority to designate and delineate such areas. The new regulations do not require any further permission for exploration activities to be conducted in such areas. A notification to the relevant authorities (such as relevant forest directorate) shall suffice. However, mining operational activities in such areas shall be subject to additional permissions and EIA requirements.

The EIA process in Turkey now covers all activities from drill campaigns in excess of 5,000 metres to small scale excavations through to full-blown mining activity. Truva Bakir submitted the Halilağa EIA report, for a stockpiling / bulk sampling operation, on February 14th, 2012. On February 24th, 2012 the EIA report was approved for adequacy with regards to the Terms of Reference and was passed to the EIA Review Committee at the Ministry of Mines for further consideration on February 27, 2012. As at the date of this report, Truva Bakir awaits details of this review. A public consultation process occurs following receipt of approval. Truva Bakir, under direction of TMST is also in the process of applying for the related permits, including those from the Ministry of Forest and Environment, and the Governor of Çanakkale.

Truva Bakır has been carrying out exploration activities in the Halilağa region since 2006. The exploration program has concentrated around the valley between the hills of Camtasi and Kunk where the three springs that supply drinking and utilization water for the village of Muratlar are located.

All three springs reside in the mine exploration area and the quantity and the quality of the water obtained from the springs will potentially be affected once any proposed pit development is initiated. Given this valid concern, an alternative resource for the drinking and utilization water for the village of Muratlar needed to be identified.

In general, drinking water in the region is supplied from springs flowing out of the hills at higher elevations by using gravity to transport water. The spring water discharge is collected in water depots close to springs. From these small depots, the water flows downhill through small diameter PVC pipes to the large storage depots which hold approximately 70 $m^3$ of water and are located near the villages. From these storage depots, the water is distributed to the individual houses within the village. In general the distance from the springs to villages varies from 3 to 11 km.

For this water resource study, SRK focused on four villages namely, Muratlar, Hacıkasım, Halilağa and Keçiağılı. They all use the water resources from the same region, which encompass the hills on the South East of the potential mining area. Additional to the fact that this region may also supply water for the Muratlar village, SRK had considered the usage of the resources in this region to avoid future problems, such as:

- Muratlar is using the three springs located on the eastern edge of the potentially planned pit area
- Hacıkasım is using the water resource at a distance of 8 km in the south
- Halilağa is using three springs to the South. Two of them are mixed in a depot and from there the mixed water is shared with Keçiağılı
- Keçiağılı is using the two springs on the south that are shared with Halilağa

There are 600 people living in the village of Muratlar. With the assumption that each occupant consumes about 150 L water per day, the approximate water demand for the village of Muratlar has been estimated to be 90 m3/day, or approximately 1 L/s.

Three springs that currently supply water for the village of Muratlar are mixing in a collection depot downstream. Previous flow measurements conducted at this point are 0.78 L/s and 0.58 L/s as reported in December 2010 (Golder, 2010) and June 2009 (SRK), respectively. Measurements show that current discharge is significantly below the estimated demand.

The results of a water quality analysis reported by Golder Associates (2010) shows that the quality of the samples taken from the village of Muratlar main depot on June 1, 2010 satisfied the accepted maximum concentrations of major ions (primary criteria) in the Ministry of Health's "Regulation on quality of water intended for human consumption". However, the low pH level and the elevated aluminum content don't comply with the secondary criteria and is therefore not recommended for human consumption. Additionally, according to the Ministry of Environment and Forestry's "Turkish Water Pollution Control Regulation" the water is classified as "Low Quality Groundwater (Class III)" which means that the use of such water is possible after a purification process but its use is determined by the degree of purification attainable economically and technologically and with respect to health.

SRK conducted a systematic screening of all possible supply options. For this purpose, the following actions were performed to collect the missing information about the water resources within the region:

- Action 1: Extend the hydrocensus to find water resources at higher altitudes
- Action 2: Evaluate the limestone aquifer
- Action 3: Assess the water quality of the wells: MWS1 and MWS2
- Action 4: Locate alternative groundwater wells

In Action 1, two main springs and one small developed spring can supply the required quality and quantities necessary on the hills in the Southwest of the village or Söğütalan. Because of the elevation, (390 m a.s.l.) the water can also be transported to the village of Muratlar by gravity. According to wet season measurements, approximately 3 L/s of good quality (above drinking standards) water was identified. It is recommended that this action be taken.

In Action 2, the marble aquifer was further appraised in terms of quantity and quality. In terms of quantity, there is an excess amount of water which is not used by the villages of Halilağa or Keçialan and this amount can easily satisfy the demands of Muratlar. In terms of quality, the arsenic concentrations were found to be higher than the regulatory standards and needs to be addressed if it is to be used as drinking water. In order to achieve this, a "blending" of this water with the two small springs found within the scope of the Action I study was proposed. Chemical modeling has also shown that blending is an adequate and reliable option for reducing elevated levels of arsenic. With this blending, approximately 2 L/s of good quality water can be obtained; this is typically the recorded flow during the wet season.

In Action 3; the water quality of the two wells: MWS1 and MWS2 (drilled for the village of Muratlar) were assessed with standard sampling and analytical procedures. Arsenic concentrations in both wells were found to be higher than the regulatory standards and arsenic removal is required if this is to be used for drinking water. Moreover, the energy requirement of the pump constitutes a significant disadvantage.

Among the actions outlined here, Action 1 was found to be the most feasible and sustainable solution, and Action 2 was considered to be a backup solution for the water supply.

As the project develops, it is SRK's recommendation and Truva Bakir's goal to use appropriate environmental standards and safeguards for all aspects of drill testing, engineering testing, construction, operation and closure. If the project is determined to be economic through further studies, Halilaga would be constructed, operated and closed under the Turkish mine permitting regulations. The operation would also conform to best practices for mining, processing (milling and flotation) and closure including tailings management, waste rock management and appropriate water, air and social responsibility standards. Closure costs have been allocated based on broad factored estimates for appropriate practices.

# 19 Capital and Operating Costs

## 19.1 CAPEX

The CAPEX of the project has been estimated based on the scope defined in previous sections of this report. The following parties have contributed to the preparation of the CAPEX estimate in specific areas:

- Ausenco:
  - Process plant;
  - Plant infrastructure and services, including road diversion, high-voltage transmission line, and high-voltage substation;
  - Construction accommodation camp;
  - Mine workshop;
  - EPCM costs relating to the process plant and infrastructure outlined above; and
  - Owner's Costs.
- SRK:
  - Initial mine development costs;
  - Water reservoir ;
  - Waste management facilities;
  - Mine working capital; and
  - Mine haul road.

### 19.1.1 Capital Cost Summary

All CAPEX are expressed in Q2 2012 US dollars. There are no allowances for escalation. The estimated costs include all mining, site preparation, process plant, first fills, buildings, road works and the construction camp. The estimates are considered to have an overall accuracy of +35% and assume the project would be developed on an EPCM basis.

The following parameters and qualifications are made:

- VAT (assumed to be recoverable);
- Estimate was based on Q2 2012 prices and costs; and
- No allowance has been made for exchange rate fluctuations.

Data for these estimates have been obtained from numerous sources, including:

- Scoping level engineering design;
- Budgetary equipment proposals; and
- Data from recently completed similar studies and projects.

Table 19.1 provides a summary of the CAPEX estimate.

**Table 19.1: CAPEX Estimate**

| Description | Estimate (M$) |
|---|---|
| Primary mining equipment fleet | 171.8 |
| Support equipment | 8.3 |
| Misc. | 7.1 |
| Spare parts | 9.4 |
| Topsoil stripping and mining preparation | 11.7 |
| Crushing | 50.2 |
| Process Plant | 199.9 |
| Concentrate Loadout | 1.7 |
| Tailings Facility and Water Reservoir | 105.2 |
| Utilities | 38.8 |
| On-site Infrastructure (inc. Reservoir) | 33.3 |
| Off-site Infrastructure | 13.8 |
| Indirect Costs | 148.7 |
| Owner's Costs | 41.9 |
| Sustaining | 51.6 |
| Closure | 75.0 |
| CAPEX w/o Contingency | 968.3 |
| Contingency | 200.4 |
| **TOTAL CAPITAL COST** | **1,168.7** |

A contingency of 25% for all capital was used, except for mining equipment which was assigned a contingency of 5% due to the relatively simple estimation of project needs and accurate equipment vendor quotes.

The purpose of the contingency provision is to make allowance for uncertain elements of costs to cover such factors as:

- Limited information on site conditions, especially concerning sub-surface conditions and the engineering properties of excavated materials;
- Completeness and accuracy of quantity take-offs and estimate assembly and consolidation based on the level of engineering and design undertaken at study level;
- Accuracy of materials and labour rates (excluding extreme variations that would be covered under contingency);
- Accuracy of productivity expectations; and
- Accuracy of equipment pricing.

The following assumptions were used in the CAPEX estimate:

- The detail of the design is discussed in the relevant sections of this report;
- Benchmarked plant CAPEX estimates are correct

- Suitably qualified and experienced construction labour would be available at the time of execution of the project;
- No exceptional geotechnical and drainage issues during, therefore, no allowance for special ground preparation was made;
- Borrow sources for construction are available from within the mine limits;
- A power and water supply capable of supplying the required demand of the processing plant is assumed to be available; and
- No extremes in weather would be experienced during the construction phase and as such, no allowances are included for construction-labour stand-down costs.

The following items are excluded from the estimate:

- Cost changes due to currency fluctuation;
- Force majeure issues;
- Sunk costs up to the project go-no go decision point. The costs that are excluded encompass feasibility study costs, resource definition drilling, EIA work, metallurgical testing, hydrogeological and geotechnical drilling and testwork and all other work associated with a feasibility study and EIA (these costs could be approximately $50M total over two years);
- VAT (assumed to be recoverable);
- Future scope changes;
- Project insurances;
- Project interest and financing cost;
- Land acquisition and compensation cost;
- Operational insurances such as business interruption insurance and machinery breakdown;
- Public road maintenance; and
- Relocation or preservation costs, delays and redesign work associated with any antiquities and sacred sites.

### 19.1.2 Mine Capital Cost

#### Open Pit Mobile Equipment

The CAPEX estimate for the OP operation is based on the scheduled plant throughput rates, as well as comparing to similar sized OP gold operations (throughput of 18.3 Mt/a process plant feed). The OP mining activities for the Halilağa pit was assumed to be undertaken by an owner-operated fleet as the basis for this preliminary study with the fleet having an estimated maximum capacity of 120,000 t/d total material, which would be sufficient for the proposed LOM plan.

The OP equipment CAPEX (including sustaining and replacement costs) required to achieve the target processing rate is summarized in Table 19.2. Mining cost service information, as well as factors based on experience, was taken into consideration in determining the OP CAPEX estimate. No equipment was considered as lease.

The Ancillary equipment includes light trucks and service vehicles, backhoes, and fuel trucks, along with a number of other required OP mining support equipment.

**Table 19.2: Open Pit Equipment CAPEX Summary**

| Item | Unit | Unit Cost | Initial units | Replace units | Total units | Total |
|---|---|---|---|---|---|---|
| **Primary** | | | | | | |
| Crawler-Mounted, Rotary Tri-Cone, 311mm Dia. Electric | M$ | 4.4 | 2 | 1 | 3 | **13** |
| Crawler-Mounted, Rotary Tri-Cone, 250mm Dia. Electric | M$ | 2.5 | 1 | 1 | 2 | **5** |
| Crawler-Mounted, Rotary Tri-Cone, 115mm Dia. Diesel | M$ | 0.7 | 1 | 0 | 1 | **1** |
| Electric, 26-cu-m Cable Shovel | M$ | 20.0 | 2 | 0 | 2 | **40** |
| Diesel, 20-cu-m Wheel Loader | M$ | 4.6 | 1 | 1 | 2 | **9** |
| 220 t class Haul Truck | M$ | 3.9 | 10 | 10 | 20 | **78** |
| D11-class Track Dozer 21' blade | M$ | 2.1 | 2 | 2 | 4 | **8** |
| D10-class Track Dozer 17.3' blade | M$ | 1.4 | 1 | 1 | 2 | **3** |
| 834H-class Rubber tire Dozer 15.2' blade | M$ | 1.1 | 1 | 1 | 2 | **2** |
| 16H-class Grader 16' blade | M$ | 1.0 | 3 | 2 | 5 | **5** |
| 150 ton-class Water truck | M$ | 1.8 | 2 | 2 | 4 | **7** |
| **Subtotal Primary** | **M$** | | | | | **172** |
| **Subtotal Ancillary** | **M$** | | | | | **8** |
| **Subtotal Miscellaneous** | **M$** | | | | | **7** |
| **Total Equipment & Misc.** | **M$** | | | | | **187** |
| Spares Inventory @ 5% | M$ | | | | | 9 |
| **TOTAL MINE CAPITAL** | **M$** | | | | | **197** |

## OP Development

Given that high grade mineralized material outcrops at surface, no pre-stripping requirements were allocated for the Halilağa pit. The first year of mine productions predicts that 14 Mt of mineralized material being fed to the process plant. As such, no pre-stripping CAPEX have been estimated, other than those described below in relation to clearing, grubbing and topsoil management.

The cost estimate for the clearing and grubbing of the various pit areas and waste dumps cost estimate was based on total area to be cleared and assumes clearing/grubbing to be undertaken with owner-operated fleet prior to full scale production taking place. Topsoil removal and storage (for use in reclamation) has also been included in these costs. Total CAPEX of $10.9M is estimated as summarized in Table 19.3 below.

**Table 19.3: Clearing, Grubbing and Topsoil Removal and Storage CAPEX Estimate**

| Location | Unit | CAPEX Estimate |
|---|---|---|
| Halilağa Pit | M$ | 3.4 |
| WRFs | M$ | 7.5 |
| **Total** | **M$** | **10.9** |

## Water Control

Table 19.4 summarizes the CAPEX estimated for the surface water control of the OP and WRFs. Costs were built up based on the proposed pit and WRF design configurations and existing topography and are included in mining preparation costs. Water diversion and catchment requirements were estimated.

**Table 19.4: Open Pit Surface Water Control CAPEX Estimate**

| Site | Unit | Site Clearance | Earthworks | Concrete (small works) | Total |
|---|---|---|---|---|---|
| Halilağa Pit/WRFs | M$ | 0.12 | 0.02 | 0.20 | 0.3 |

Pit dewatering CAPEX requirements for the various OP are summarized in Table 19.5These preliminary costs were based on estimates of inflows of water into the pit; well drilling and construction; pipeline supply and installation; and pumping requirements.

**Table 19.5: Open Pit Dewatering CAPEX Estimate**

| Pit | Unit | Pipeline | Pumps | Wells | Total |
|---|---|---|---|---|---|
| Halilağa Pit | M$ | 0.35 | 0.07 | 0.08 | 0.5 |

### 19.1.3 Water Reservoir and Tailings Management Facility Capital Cost

CAPEX developed for the tailings starter dam, starter berm and water reservoir was estimated. The primary material take-offs were calculated using AutoCAD Civil 3D 2011, and are summarized in Table 19.6.

**Table 19.6: Summary of Key Construction Quantities**

| Element | Units | TMF Starter Dam (including Starter Berm) | Water Reservoir Dam |
|---|---|---|---|
| Rip-Rap | $m^3$ | n/a | 8,683 |
| Bedding material | $m^3$ | 42,045 | 10,905 |
| Transition material | $m^3$ | 328,499 | 29,836 |
| Run-of-quarry material | $m^3$ | 2,052,343 | 149,369 |
| Key Trench Excavation / Fill | $m^3$ | 236,170 | 15,872 |
| **Total Fill** | $m^3$ | **2,362,886** | **190,110** |

Note: The total fill volume is the sum of the main dam fill (Rip-Rap, Bedding, Transition and ROQ material) and the key-trench volume. Volumes are reported as in-place volumes to neat lines with no account for bulking or shrinkage.

Assumptions have been made about haul distances, road grades, material properties, productivity, labour rates, fleet type and equipment and materials rates. These assumptions were based on

engineering judgement, past project experience and supplemented with conventional costing databases. Key assumptions can be summarized as follows:

- Construction would be done by a dedicated contractor using their own fleet of equipment;
- A fleet of seven 36 t (19.85$m^3$) CAT 769D haul trucks would be used;
- ACAT 960F loader is used to load the haul trucks at the rock quarry;
- The haul distance for ROQ, Transition, Rip-Rap and Bedding material is assumed to be about 5 km, with maximum road grades of 8%;
- All construction materials would be from a locally developed quarry. Development of the quarry will require drilling and blasting;
- Costs include construction of access roads from the mill and haul roads from potential quarries;
- A dozer to continuously spread the cyclone underflow has been included in the capitol costs;
- A preliminary estimate for required pumping stations has been included. For tailings, one pumping station is assumed to be housed in the mill or plant building while the second station is expected to be housed in its own building/structure; a slurry booster is also included;
- For the reservoir, a series of pumps would be installed along the waterline without any housing;
- The estimated grout curtain costs assume that three rows of 8.5” diameter drill holes at 1 m spacing are installed along the central upstream dam toes. The length of the grout curtain is assumed to equal the crest length (i.e. the side slopes or higher on the abutments would be assumed to not require grouting); and
- Indirects have been assumed to be 10% if the direct costs and allows for only EPCM costs. The water reservoir dam must be constructed at least two years before mine production starts to ensure it is at full capacity when production starts. The CAPEX required for this construction is $8.0M. The TMF starter dam and starter berm must be constructed in the year immediately preceding production. The CAPEX for these structures are $89.6M. A summary of these costs are presented in Table 19.7.

**Table 19.7: Summary of Water Reservoir and TMF CAPEX Estimate**

| Cost Type | Cost Element | Unit | Year | | |
|---|---|---|---|---|---|
| | | | Year -2 | Year -1 | Year 6 |
| Direct Costs | TMF starter dam (including starter berm) | M$ | | 68.9 | |
| | Replacement equipment (dozer & compactor) | M$ | | | 1.0 |
| | TMF feed pumps & pipeline (excludes return water line) | M$ | | 15.8 | |
| | Water Reservoir | M$ | 6.2 | | |
| | Water Reservoir pumps & pipeline | M$ | | 1.8 | |
| | SUBTOTAL Direct Costs | M$ | 6.2 | 86.5 | 1.0 |
| Indirect Costs (EPCM) | TMF starter dam (including starter berm) | M$ | | 6.9 | |
| | Replacement equipment (dozer & compactor) | M$ | | | 0.1 |
| | TMF feed pumps & pipeline (excludes return water line) | M$ | | 1.6 | |
| | Water Reservoir | M$ | 0.6 | | |
| | Water Reservoir pumps & pipeline | M$ | | 0.2 | |
| | **SUBTOTAL Indirect Costs** | **M$** | **0.6** | **8.6** | **0.1** |
| **TOTAL Cost** | | **M$** | **6.8** | **95.1** | **1.1** |

No sustaining capital is required for the water reservoir. The life of mine is sufficiently short that upgrades to the dam will not be required. Tailings dam raises would be done using cylone technology and as a result the only sustaining capital would be replacement of cyclones. This cost is however so small that it has been included in the OPEX.

### 19.1.4 Process Plant Capital Cost

The process plant design for the scoping study incorporates primary gyratory crushing, coarse plant feed material stockpiling, SAG and ball milling with pebble crushing, copper separation and conventional tailings disposal.

The estimate has been prepared on a commodity basis (i.e., separated into earthworks, concrete, structural, etc.) and reported by area (i.e., crushing, grinding, flotation, etc.). Further details of the structure of the estimate are provided by commodity below.

The project is based on the purchase of new mechanical equipment, and factors for other components, such as civil and structural. These factors have been assessed using benchmarking and developed from historical data obtained from similar projects.

These factors listed in Table 19.8 include costs for material and labour for installation.

**Table 19.8: Material and Installation Factors by Area and Discipline**

| Discipline / Area | Earth-works | Civil/ Con-crete | Struc-tural/ Steel | Plate work | Mechanical | Piping | Elec-trical | Instru-mentation |
|---|---|---|---|---|---|---|---|---|
| Crushing | 75% | 78% | 12% | 15% | 42% | 16% | 30% | 13% |
| Ore Storage | - | - | - | 5% | 3.5% | 5.7% | - | 3.2% |
| Stockpile/ Reclaim | 3.8% | 186% | 25% | 11.4 % | 22% | 3.5% | 12% | 2.5% |
| Grinding | 33% | 16.8% | 7.8% | 2.8% | 14% | 8.9% | 8.2% | 5% |
| Flotation | 73% | 18% | 7.6% | 7.5% | 35% | 39% | 28.5% | 8.6% |
| Regrind | - | - | | 3% | 15.5% | 16.4% | 43% | 4.5% |
| Thickening | 10.8% | 171% | 106% | 114% | 105% | 46.9% | 118% | 17.8% |
| Reagents | 0.3% | 17% | 53.5% | 17.5 % | - | 127.8% | 66.7% | 2.46% |
| Tails & Reclaim | 1.5% | 223% | 13.4% | 4% | 26.7% | 108% | 11% | 2.5% |
| Filtration | 2% | 52.2% | 78.2% | 102% | 82% | 31% | 16.2% | 9.2% |
| Air Blowers | 0.6% | 8.8% | 29.8% | - | - | 76.6% | 26.3% | 1.3% |
| Water System | 154.5% | 121.4% | 207.4% | 45% | - | 1500% | 37.6% | 50.3% |
| Electrical | 0.9% | 3.3% | 14.3% | 0.2% | 0.0% | - | - | - |

Earthworks and Civil WorksFactors were applied for all commodities associated with the earthworks, drainage and concrete requirements for the project on an area-by-area basis.

Factors for bulk earthworks have been developed from benchmarking other projects of similar scope and using limited topographical survey information, and thus require further review when detailed topographical data are available.

## Concrete

Concrete factors have been determined from reference to similar plants.

## Mechanical Equipment

The unit costs for major pieces of equipment have been acquired from budget quotes received on recent projects covering 60% of the total equipment cost. The remaining 40% is based on in-house data.

Recent equipment quotes from vendors include:

- Crushers;
- Ball and SAG mills;
- Flotation cells;
- Concentrate filters;
- Thickeners;
- Large slurry pumps;

- Cyclone clusters;
- Vibrating screens;
- Apron feeders;
- Agitators;
- Air compressors;
- Air blowers;
- Mill realign machines;
- Lime plant;
- Samplers and analysers; and
- Stockpile cover.

## Structural Steelwork

Structural steelwork quantities were determined by reference to similar plants.

These factors include supply, shop detailing, fabrication, surface preparation and final painting in the shop, transport to site, site erection and paint touch-up as required.

## Platework

Similarly to other bulk materials, unit rates were benchmarked from other projects of similar scope.

These factors include supply, shop detailing, fabrication, surface preparation and final painting in the shop, transport to site, site erection, and paint touch-up as required.

## Piping and Electrical and Instrumentation

Piping and E&I were factored from mechanical equipment pricing for the crushing and process areas based on actual historical factors for similar plants internationally.

## Installation

Factors included a site productivity allowance for each type of work, plus the appropriate gang rate for the commodity and the actual cost of the permanent materials.

Labour rate factors are based on a 50-hour work week, which is typical for recent projects executed internationally. The average all-in labour rate for a 50-hour work week is $30/h and productivity factors were included for each commodity, ranging from 2.1 to 2.5. The productivity factors were checked against actual historical data from recent projects. Projects used to benchmark include Collahuasi, Los Pelambres, Candelaria, Escondida and Los Brances.

The rates include the following items:

- Base rate per hour
- Sick time;
- Holiday pay;
- Insurance;
- Health and welfare;
- Small tools and consumables;
- Safety gear and clothing;

- Site supervision;
- Mobilization and demobilization;
- Transportation – turnaround;
- Site and head office overhead; and
- Profit.

The estimate is based on the majority of the work being carried out under fixed price or re-measurable unit price contracts under a normal development schedule. No allowance is included for contracts on a cost plus or fast-track accelerated schedule basis.

Local freight associated with contractor-supplied material is included in the unit rates.

The estimate for ocean freight and Turkish inland freight were based on first principles, with appropriate percentages applied depending on where the particular equipment item originated. Overall, this figure worked out to be 5.0% of the equipment costs, which falls within the expected range.

The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works would be performed by experienced contractors, using a mix of national and non-national labour; the project is allowed by the Turkish government to utilize non-nationals for construction as necessary to achieve the required quality and meet the project schedule.

A summary of the estimated CAPEX for the processing plant and on-site ancillary facilities is provided in Table 19.9.

Indirect costs, including project contingency have been provided for in the CAPEX estimates. Indirect costs have been estimated based on a factor of the total direct costs established from previous projects.

**Table 19.9: Summary of Plant CAPEX Estimate**

| Area | Estimated CAPEX (M$) |
|---|---|
| Direct Costs | |
| Process Plant (crushing and concentrator) | 251.8 |
| Concentrator utilities and services | 38.8 |
| Tailings thickener, pumping and pipeline | 11.5 |
| On-site Infrastructure | 33.3 |
| Off-site Infrastructure | 13.8 |
| **TOTAL Direct Costs** | **349.2** |
| Indirect Costs | |
| EPCM | 62.9 |
| Temporary construction facilities (incl. camp) | 39.6 |
| Cranage | 4.1 |
| Spares and first fill | 26.1 |
| Freight and Logistics | 11.0 |
| Commissioning Support | 1.3 |
| Pre-production Owner's costs | 41.9 |
| **TOTAL Indirect Costs** | **186.9** |
| **Total Directs + Indirects** | **536.1** |
| **Contingency (25%)** | **134.0** |
| **TOTAL** | **670.1** |

See body of document for scope and battery limits

### 19.1.5 On-Site Infrastructure Capital Cost

Mobile Equipment

Table 19.10 describes the cost of the mobile fleet required to support plant operations.

**Table 19.10: Plant Support Mobile Equipment CAPEX Estimate**

| Description | Number | Total Cost (M$) |
|---|---|---|
| Run-of-mine rock breaker – Excavator with rock breaker hammer | 1 | 0.50 |
| Plant front-end loader – CAT 929Dl | 1 | 0.26 |
| Skid steer loader – CAT 262C | 3 | 0.14 |
| Plant workshop forklift – S55FTS | 2 | 0.05 |
| Plant forklift truck – 18 t All terrain H450 | 1 | 0.24 |
| Plant telehandler – 4 t capacity | 1 | 0.10 |
| Telescopic boom elevated work platform – 40 ft reach | 1 | 0.09 |
| Telescopic boom elevated work platform – 60 ft reach | 1 | 0.13 |
| Yard crane – 20t YB7725 | 1 | 0.31 |
| Rough terrain crane – 80t RT890E | 1 | 0.83 |
| Tyre handler forklift – S55FTS | 2 | 0.05 |
| Elevated work platform – 7.9 m height | 2 | 0.03 |
| Light vehicle – 4x4 Dual Cab | 4 | 0.09 |
| Flatbed truck – 10 t carry capacity | 1 | 0.07 |
| Flatbed truck – 15 t carry capacity | 1 | 0.09 |
| Minibus – 21 seats | 2 | 0.12 |
| Fire response vehicle | 1 | 0.09 |
| Ambulance | 1 | 0.09 |
| **Total** | | **3.30** |

Ancillary Buildings

The following administration buildings are included in the CAPEX estimate:

- Main administration building with medical centre and training room;
- Plant workshop and warehouse;
- Security office;
- Security gatehouse;
- Metallurgical office;
- Metallurgical laboratory; and
- Plant cafeteria and training room.

The plant workshop and warehouse would be a fully-clad steel portal frame building. Offices would be attached to the warehouse for procurement, maintenance, and planning personnel.

The costs of ancillary buildings were estimated on a lump sum basis by benchmarking similar projects from Ausenco data base. In addition to the building structures, the cost includes the supply of the buildings electrics, fittings, and furnishing, but excludes earthworks. The total cost was estimated at $9.2 million.

The cost to supply power and water services to the buildings form part of the process plant cost. In addition, reagent storage facilities are included in the process plant cost estimate.

Mine Services Facilities

The mine service facilities would include:

- Heavy vehicle workshop with overhead gantry cranes;
- Light vehicle workshop;
- Mine services area bond store;
- Administration building;
- Cafeteria; and
- Mine changehouse.

The buildings would be fully-clad steel portal frame building and have been estimated on a lump sum basis by benchmark of similar projects. The total cost was estimated at $10.9 million.

### 19.1.6 Off-Site Infrastructure and Capital Cost

#### Site Power Supply

The project requires electrical transmission, sub-transmission, and distribution infrastructure as described below:

High-voltage transmission would be achieved using a 154 kV double-circuit overhead line from the existing regional grid. The transmission line is proposed to be approximately 25 km long to the main substation at site.

Medium-voltage power would be transmitted to the different motor control centres of consumption via a 13.8 kV line.

Medium-voltage power would be transmitted to the different motor control centres via 15.8 kV lines or other suitable voltage meeting typical Turkish electrical codes.

Low-voltage power distribution would be at 400 V for offices and ancillary systems.

The equipment required includes a 154 kV exit sections at the feeder substation, switchgear, and a 154/15.8 kV transformer yard at the project main substation.

#### Port Facilities

Port facilities were excluded from the CAPEX estimate as it was assumed that the Port of Bandirma has the facilities to handle both incoming supplies and out-going concentrate.

### 19.1.7 Indirect Costs plus Owners Costs

Indirect costs and Owner's costs total an estimated $180.3 M, equal to 54% of the total direct cost.

The various cost centres that comprise the indirect costs are described in the following sections.

#### EPCM

For the purpose of the scoping estimate, 18% of the direct costs was selected to cover the cost of EPCM services, which includes detailed engineering, procurement, project management and home office services as well as construction management. This was calculated on direct costs that excluded the mine equipment and mine development.

## Temporary Construction Facilities and Services

For the purpose of the PEA, 1.3% of the direct costs was selected to cover the cost of temporary construction facilities and services. It includes the project management team's site installation requirements for project execution, such as:

- Project management team office complex supply and setup;
- Toilets facilities for construction;
- Vehicles;
- General site clean-up;
- Refuse disposal;
- Site security guards;
- Computer, phone and communication system and office supply;
- Dismantling of facilities at contract's end ; and
- Commissioning support.

## First Fills / Spares

The estimate for first fills and spares (excluding mining equipment) is based on 5% of total mechanical equipment cost. This figure is based on actual factored historical data on similar projects.

## Cranage

This applies to heavy cranage only, as this would be controlled by the EPCM contractor. It applies to cranage in excess of 150 tonnes capacity.

## Freight and Logistics

The estimate for ocean freight and Turkish inland freight were based on first principles, with appropriate percentages applied depending on where the particular equipment item originated. Overall, this figure worked out to be 5.0% of the equipment costs, which falls within the range expected.

## Construction Camp

The construction camp costs include costs for the construction and operation of the camp. The camp design is based on 100 beds for management and supervision, 400 beds for staff personnel, and 1,500 beds for craft personnel, bringing the camp total to 2,000 beds. The average cost per bed is $8,750. The cost of the construction of the camp is inclusive of all facilities.

The camp OPEX (all-inclusive of meals, maintenance, etc.) is based on $25 per person camp day, for a total of 700,000 camp days.

Both these rates are based on actual historical data from other camps.

### 19.1.8 Owner's Cost

#### Owners Team Costs

The Owner's team costs cover the costs incurred to build the operations team. The organisation structure will include the following departments:

- Senior management;
- Mining;
- Process;
- Finance and administration;
- Legal and community affairs;
- Safety, heath, environment and community affairs; and
- Occupational health and safety.

The selection, training and management of personnel would be undertaken in accordance with labour and recruitment and human resources management policies consistent with other major international operators. Successful recruitment and training represent key issues in the successful development and operation of the project.

#### Insurance

One percent of direct cost was calculated to cover insurances, including mining equipment and mine development.

#### Pre-Production

The pre-production cost is based on the estimate OPEX for the process facilities at the mine and port for the first two months of operation.

#### Project Development

The pre-development cost does not include the following:

- Pre-feasibility and feasibility studies;
- High voltage power studies;
- In-fill and geotechnical drilling;
- Hydrogeology investigations;
- EIA preparation; and
- Land access/acquisition.

### 19.1.9 Contingency

The contingency reflects the potential growth in CAPEX within the same scope of work. The contingency includes variations in quantities, differences between estimated and actual equipment and material prices, labour costs and site-specific conditions. It also accounts for variation resulting from uncertainties that are clarified during detail engineering, when designs and specifications of the basic engineering scope are finalized.

Contingency is an amount of money allowed in an estimate for cost which, based on past experience, are likely to be encountered, but are difficult or impossible to identify at the time the

estimate is prepared. It is an amount expected to be expended during the course of the project. Contingency does not include scope changes, force majeure, labour disruptions or lack of labour availability.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, pipelines, indirect costs and Owner's costs) were identified and analyzed. An overall contingency of $129 M was obtained, representing 25% of the total CAPEX.

### 19.1.10 Duties and Taxes

#### Direct Costs

Local taxes on contractor-supplied materials and installation labour are not included in the estimate.

#### Indirect Costs

Taxes not included in the estimate are VAT. VAT has not been applied to spare or ocean freight. VAT has been assumed to be recoverable. The value effect of delays in recovery has been assumed to be negligible for PEA purposes.

### 19.1.11 Escalation

No escalation costs have been included. All costs and prices are expressed in 2012 dollars.

### 19.1.12 Project Deferred and Sustaining Capital

Ongoing capital requirement for the mine production period totals $516 M over the mine life. This cost covers the phased construction of the tailings management facility. The cost also covers mine and process plant equipment to sustain the ongoing operation of the project.

## 19.2 Operating Costs

### 19.2.1 Operating Cost Summary

The OPEX estimate is based on a variety of sources including cost service data, vendor quotes, first principle calculations, and reference projects. The summary OPEX costs are shown in Table 19.11.

**Table 19.11: OPEX Estimate Summary**

| | Unit Cost Estimate | | Total LOM Cost (M$) |
|---|---|---|---|
| **Item** | **($/t mined)** | **($/t milled)** | |
| Mining | 1.74 | 3.40 | 828 |
| Processing | | 4.69 | 1,141 |
| Tailings Management | | 0.23 | 56 |
| Environment and water treatment | | 0.05 | 12 |
| G&A | | 0.42 | 102 |
| **Total** | **1.74** | **8.79** | **2,139** |

### 19.2.2 Basis of Estimate

The main OPEX assumptions used in the study are shown in Table 19.12.

**Table 19.12: Main OPEX Assumptions**

| Item | Unit | Cost Estimate | Comment |
|---|---|---|---|
| Diesel Fuel | $/litre | 1.86 | Exclusive of VAT |
| Electricity | $/kWh | 0.14 | Energy and demand cost estimate |
| Labour | $/hr | 7.00 (Labourer) to 13.00 (Journeyman) | Estimate based on various sources |
| | % Overhead on Labour | 25 | Estimate for hourly labour |

### 19.2.3 Mining Operating Costs

The OP mining activities for the Halilağa Project were assumed to be undertaken by the owner as the basis for this PEA and are presented in Q2-2012 US dollars.

The cost estimate was built from first principles based on equipment required for the mining configuration of the operation, along with input from Pilot Gold, as well as SRK experience of similar sized OP operations. Equipment efficiency was estimated based on Halilağa conditions (e.g. haul routes for each phase). Local labour rates and estimates on diesel fuel pricing were taken into consideration for the mining cost estimate. The OP mining costs encompass pit and dump operations, road maintenance, mine supervision and technical services cost.

The average OP OPEX for the LOM plan were estimated to be $1.74/t material mined or $3.40/t plant feed, for pit and dump operations, road maintenance, mine supervision ,and technical services. Table 19.13 summarizes the mining OPEX by function. The OP mining costs were calculated for both mineralized material and waste mining, where variations in haulage profiles and equipment selection were taken into account in the cost estimate.

**Table 19.13: Mine Average OPEX Estimate by Function**

| OP Function | Unit Cost Estimate ($/t mined) |
|---|---|
| Drill | 0.11 |
| Blast | 0.20 |
| Load | 0.14 |
| Haul | 0.84 |
| Roads/Dumps/Support Equipment | 0.31 |
| General Mine/Maintenance | 0.06 |
| Supervision/Technical | 0.08 |
| **Total** | **$1.74** |

### 19.2.4 Processing Operating Costs

A summary of the average OPEX per tonne of feed treated for the Project is outlined in Table 19.14 and Figure 19.1. The costs have been separated into the key cost components. All costs have been based on estimates as of the 2nd quarter 2012.

**Table 19.14: Processing Average Unit OPEX Estimate**

| Summary | Estimate ($/t milled) |
|---|---|
| Labour | 0.27 |
| Power | 2.35 |
| Maintenance Materials and Services | 0.37 |
| Reagents & Consumables | 1.62 |
| Miscellaneous | 0.08 |
| **Total Unit Cost** | **4.69** |

The OPEX are considered to have accuracy in the order of ± 30 percent, based on the assumptions listed in this section of the report.

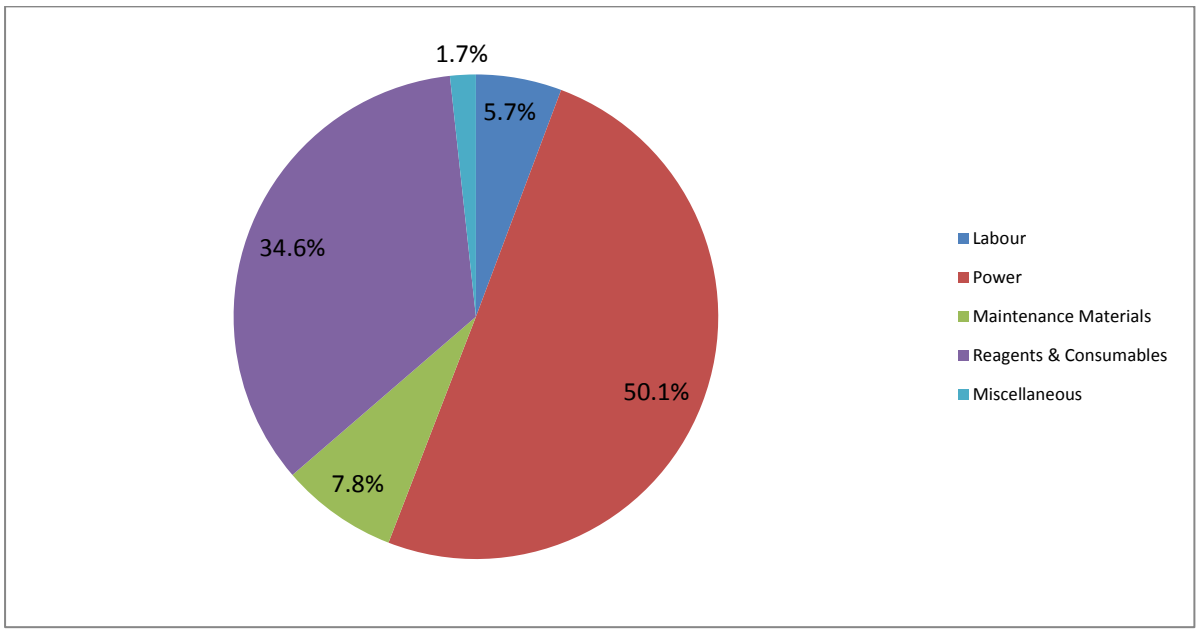



**Figure 19.1: Chart of Processing OPEX Estimate by Component**

## Processing Operating Cost Estimate Basis and Method

The OPEX estimate has been developed from a number of sources. Cost determinations have been based on fixed and variable components relating to plant throughput and plant feed material characteristics. The sources of data are detailed in Table 19.15.

**Table 19.15: Derivation of Plant OPEX Estimate**

| Cost Category | Source Of Cost Data |
|---|---|
| Power | Consumption from load estimate and power rate agreed by project team. |
| Labour | Manning schedules and rates developed by Ausenco from similar sized projects in the region. |
| Reagents | Consumptions from test work and benchmarking data; unit prices from suppliers and Ausenco database. |
| Consumables | Consumptions predicted from test work and experience and unit prices from suppliers and Ausenco database. |
| Maintenance Materials | Calculated as a percentage of direct CAPEX. |

## Exclusions

Costs that have not been included in this section (processing OPEX) of the report are:

- Concentrate handling off-site, including trucking, sea freight & insurance;
- Concentrate smelting and refining;
- Government fees and charges;
- Import duty and applicable taxes (VAT is assumed to be recoverable);
- Interest and finance charges; and
- Contingency.

## Inclusions

Specifically included in the Processing OPEX Estimate are:

- Labour for supervision, management and reporting of onsite organizational and technical activities directly associated with the processing plant;
- Labour for operating and maintaining plant mobile equipment and light vehicles, process plant and supporting infrastructure;
- Costs associated with direct operation of the processing plant, including all fuels, reagents, consumables and maintenance materials;
- Fuels, lubricants, tires and maintenance materials used in operating and maintaining the plant mobile equipment and light vehicles based on similar projects and the Ausenco database;
- Tailings discharge pumping and return water, excluding management of the TMF;
- Cost of power as supplied from the local power grid;
- Operation of raw water supply facility; and
- Allowance for assaying of production samples based on similar projects.

## Labour

Ausenco developed an overall workforce plan by position for the process plant based on typical manning schedules in similar plants. Input into unit labour costs were provided by Pilot Gold and reviewed by Ausenco for different levels of workers hired in-country based on information compiled in the regional area of the project in Turkey. Ausenco has assumed that expatriates would be employed in some of the key positions. These costs are summarized in Table 19.16.

Total employee costs have been developed by applying factors to base salaries as determined by Pilot Gold. The factors include the cost of travel, overtime and shift premiums, leave pay, bonuses, pension benefits, insurance coverage, educational assistance and supply of uniforms and personal protective equipment.

No allowance has been made in the labour estimate for additional labour requirements during plant initial operations for additional management support, and to assist in concentrator commissioning and ramp-up activities. However, the budget numbers supplied have an allowance for supervision of the ongoing training of personnel.

**Table 19.16: Site Labour Cost Summary ($)**

| Labour | No. | Base Salary |
|---|---|---|
| **MANAGEMENT** | | |
| Mill Manager | 1 | 175,000 |
| Mil Superintendent | 1 | 175,000 |
| Administrative Assistant | 1 | 18,000 |
| **MILL TECHNCIAL** | | |
| Chief Metallurgist | 1 | 175,000 |
| Senior Metallurgist | 2 | 144,000 |
| Metallurgist | 2 | 108,000 |
| Metallurgical Technician | 4 | 144,000 |
| Process Control Engineer | 1 | 72,000 |
| Process Control/Instrumentation Technician | 4 | 144,000 |
| **ASSAY LAB** | | |
| Chief Chemist/Assayer | 1 | 150,000 |
| Senior Assayer | 2 | 84,000 |
| Assayer | 2 | 60,000 |
| Assayer (Helper) | 4 | 120,000 |
| Sample Preparation | 2 | 60,000 |
| **MILL PRODUCTION** | | |
| Shift Supervisor | 4 | 168,000 |
| Control Room Operator | 4 | 168,000 |
| Reagent & Relief Operators | 4 | 144,000 |
| Trainer | 1 | 42,000 |
| Crushing Operator | 4 | 144,000 |
| Grinding Operator | 4 | 144,000 |
| Flotation Operator | 8 | 288,000 |
| Thickening Operator | 4 | 144,000 |
| Water Treatment Operator | 4 | 144,000 |
| Labourer/Clean Up | 8 | 144,000 |
| Loader Operator | 4 | 96,000 |
| Regrind Operator | 4 | 144,000 |
| **MILL MAINTENANCE** | | |
| Mill Maintenance Superintendent | 1 | 150,000 |
| Maintenance Supervisor | 2 | 84,000 |
| Maintenance Planner | 1 | 36,000 |
| Senior Mechanic \ Millwright | 2 | 72,000 |
| Mechanic \ Millwright | 8 | 240,000 |

| Labour | No. | Base Salary |
|---|---|---|
| Mechanical Fitter | 8 | 240,000 |
| Mechanical Trainee | 2 | 48,000 |
| Electrical Trainee | 2 | 48,000 |
| Electrical & Instrumentation Supervisor | 1 | 42,000 |
| Senior Electrician | 6 | 252,000 |
| Electrician | 8 | 240,000 |
| Crane Operator | 2 | 48,000 |
| **TOTAL $M/y** | | **$4,899,000** |
| **TOTAL $/T** | | **$0.27** |

## Power

Power would be supplied to the mine site from the local power grid. Unit power costs are estimated to be approximately $0.14/kWh based on initial discussions between Pilot Gold and the local electrical utilities.

The power requirements for the plant have been developed from the electrical load list generated as part of the mechanical equipment list. The calculated load on which the power costs have been based is determined by applying a specific power draw for each piece of installed equipment (excluding installed standby equipment) and a utilization factor.

A summary of power costs by area for the plant site, TMF, return water and raw water pumping is given in Table 19.17.

**Table 19.17 - Process Plant Power Cost Summary (M$)**

| Area | Annual Cost Estimate |
|---|---|
| Crushing / Stockpile | 1.7 |
| Grinding | 32.9 |
| Flotation / Bulk Cleaner | 3.0 |
| Regrind | 2.7 |
| Concentrate Thickening/Filtration | 2.5 |
| Reagents | 0.6 |
| Tailings | 0.9 |
| Utilities | 1.1 |
| **TOTAL ANNUAL COST ($M)** | **45.5** |
| **Unit Cost ($/t)** | **$2.49** |

## Maintenance Consumables and Services

Maintenance consumable costs were estimated as a percentage of the direct equipment CAPEX (percent factor). The factor is based on actual data from similar projects and takes into consideration an assumed Bond abrasion index of 0.35. The reference plant data suggests that there are two distinct groups of data for maintenance costs, with the maintenance materials for processing

averaging 5% of direct CAPEX and the services and infrastructure areas averaging 1% of direct CAPEX.

These factors were applied to the direct equipment CAPEX estimate by area to determine the estimated maintenance materials cost.

The maintenance costs that were applied to each CAPEX discipline are summarized in Table 19.18.

**Table 19.18: Process Plant Maintenance Cost Summary**

| Area | Annual Cost Estimate (M$) |
|---|---|
| Crushing / Stockpile | 0.80 |
| Grinding | 3.50 |
| Flotation / Bulk Cleaner | 0.50 |
| Regrind | 0.40 |
| Copper Con Handling | 0.30 |
| Reagents | 0.06 |
| Tailings | 0.15 |
| Utilities | 1.00 |
| **TOTAL ANNUAL COST (M$)** | **6.70** |
| **Unit Cost ( $/t)** | **0.37** |

## Reagents and Consumables

Reagent consumptions have been estimated from metallurgical test work or comparable operations. Although, reagent consumptions will vary according to metallurgical and production parameters, average predicted consumptions have been used for this exercise. Generally, these values are in line with the average consumptions from the test work to date.

In-house data from recent quotes have been used for plant reagents costs. Unit costs include an allowance for delivery to site but do not include duties, brokerage, handling charges or applicable taxes.The average LOM consumptions and the unit costs are presented in Table 19.19.

**Table 19.19: Reagent Consumptions and Unit Costs**

| Reagent | Unit Cost ($/kg) | Consumption (kg/t milled) | Annual Estimate (M$) |
|---|---|---|---|
| Primary Collector (SIBX) | $3.00 | 0.06 | 3.3 |
| Secondary Collector | $3.00 | 0.044 | 2.4 |
| Frother (MIBC) | $2.00 | 0.04 | 1.5 |
| Lime | $0.35 | 0.45 | 2.9 |
| Flocculant | $3.50 | 0.20 | 1.3 |
| **Average Unit Cost $/t milled** | **0.62** | | |
| **TOTAL M$/year** | **11.3** | | |

## Plant Consumables

Plant consumables include major items, such as crusher and mill liners and grinding media. Expected consumptions for each material type have been estimated based on experience and used to calculate LOM averages. Budget quotation costs from equipment manufacturers have been used for crusher and mill liner costs. Unit costs include an allowance for delivery to site but do not include

duties, brokerage, handling charges or applicable taxes. Consumptions and unit costs are presented in Table 19.20.

The annual liner costs have been calculated based on mineralized rock properties and specific mill power and are presented in Table 19.21.

**Table 19.20: Crusher and Mill Liner Consumption and Unit Cost**

| Item | Consumption | Unit Cost ($ per set) |
|---|---|---|
| SAG Mill Liners | 1.0 sets per year | 2,100,000 |
| Ball Mill Liners | 1.5 sets per year | 2,400,000 |
| Gyratory Crusher Shell Liners | 1.0 sets per year | 60,000 |
| Gyratory Crusher Wear Plate | 1.0 sets per year | 40,000 |
| Gyratory Crusher Concave | 2.0 sets per year | 320,000 |
| Gyratory Crusher Mantle | 4.0 sets per year | 325,000 |
| Gyratory Crusher Mantle Head | 2.0 sets per year | 45,000 |
| Pebble Crusher Liners | 2.0 sets per year | 25,000 |
| Regrind Mill Liners | 0.5 sets per year | 340,000 |

**Table 19.21: Crusher and Mill Liner Unit Cost Estimate**

| Item | Estimate ($/t milled) |
|---|---|
| SAG Mill Liners | 0.11 |
| Ball Mill Liners | 0.13 |
| Gyratory Crusher liners | 0.02 |
| Gyratory Crusher Mantle | 0.02 |
| Pebble Crusher Liners | 0.00 |
| Regrind Mill Liners | 0.02 |
| **Average Unit Cost** | **0.31** |
| **Total Annual Cost (M$)** | **5.6** |

Specifications of the grinding material for the SAG, ball and regrind mills have been detailed in Table 19.22.

**Table 19.22: Grinding Media Details Usage and Pricing**

| Mill | Detail | Spec | Type | Package | Cost $/kg |
|---|---|---|---|---|---|
| SAG Mill | 125 mm | Cr-Mo | Forged | 40' cont. (20 t) | 0.867 |
| Ball Mill | 75 mm | Cr-Mo | Forged | 40' cont. (20 t) | 0.730 |
| Regrind Mill | 5 mm | Ceramic | | 40' cont. (20 t) | 8.000 |

### Miscellaneous

A summary of Plant Miscellaneous costs is given in Table 19.23.

**Table 19.23: Process Plant Miscellaneous Cost Summary**

| Miscellaneous | Estimate (M$/year) |
|---|---|
| Vehicles Processing and Maintenance | 1.20 |
| Assay Laboratory Consumable Costs | 0.15 |
| Consultants & Metallurgical Test work | 0.10 |
| **TOTAL** | **1.45** |

## 19.2.5 Water Reservoir and Tailings Management Facility Operating Cost

OPEX for the TMF consists of the ongoing pumping and underflow tailings placement costs and was estimated to be $0.23/t milled. The cost excludes return water pumping which was captured in processing costs.

The water reservoir OPEX is limited to pumping of the water to the mill at a cost of $0.28/$m^3$.

## 19.2.6 General and Administration Operating Costs

### G&A Summary

The average G&A OPEX for the supporting facilities and administration for a typical year are estimated to be $7.6 million per annum or $0.42 /t milled. These costs are assumed to consist of both fixed costs, independent of plant throughput or mining rate and partially variable, changing in direct proportion to the plant throughput rate. The G&A costs are summarized below Table 19.24.

**Table 19.24: General and Administration Costs Summary**

| G&A Item | M$/year | $/t |
|---|---|---|
| G&A Labour | 2.97 | 0.163 |
| G&A Light Vehicle fuel & consumables | 0.06 | 0.003 |
| Telecommunications | 0.15 | 0.008 |
| Stationary | 0.01 | 0.000 |
| Postage,Courier & Light Freight | 0.02 | 0.001 |
| Computer Supplies | 0.05 | 0.003 |
| First Aid Costs | 0.10 | 0.005 |
| External Metallurgical Testing | 0.19 | 0.010 |
| Consultants & Vendors | 0.25 | 0.014 |
| Contractors | 0.25 | 0.014 |
| Safety, Clothing | 0.15 | 0.008 |
| Business Travel | 0.05 | 0.003 |
| Recruitment | 0.15 | 0.008 |
| Training | 0.15 | 0.008 |
| Community Support | 0.00 | 0.000 |
| Conferences | 0.05 | 0.003 |
| Legal | 0.00 | 0.000 |
| Insurance | 2.00 | 0.110 |
| Road Maintenance | 0.16 | 0.009 |
| Environmental | 0.15 | 0.008 |
| Audit | 0.02 | 0.001 |
| Bank Fees | 0.02 | 0.001 |
| Site Security | 0.50 | 0.027 |
| Translation/Interpreter Contact | 0.05 | 0.003 |
| Medical/Doctor Contract | 0.00 | 0.000 |
| Port Facility G&A | 0.12 | 0.007 |
| **TOTAL** | **$7.61** | **$0.42** |

Allowances for miscellaneous G&A costs were developed by Ausenco by benchmarking similar projects.

## G&A Labour Costs

Estimated G&A labour costs totalled $3.0M per annum as summarized in Table 19.16.

**Table 19.25: G&A Labour Cost Summary**

| G&A Labour Cost Summary | $M/year | $/t | Headcount |
|---|---|---|---|
| Plant Management | 0.44 | 0.02 | 3 |
| Finance | 0.28 | 0.02 | 7 |
| IT | 0.07 | 0.00 | 3 |
| Human Resources | 0.35 | 0.02 | 5 |
| Health and Safety | 0.38 | 0.03 | 8 |
| Environmental | 0.25 | 0.02 | 4 |
| Procurement | 0.48 | 0.02 | 9 |
| Miscellaneous | 0.09 | 0.16 | 21 |
| **TOTAL** | **3.76** | **0.17** | **60** |

# 20 Economic Analysis

## 20.1 Methodology

Three economic cases were analyzed using pre-tax and after-tax discounted cash flows to capture the variations of results using three sets of metal prices. The size of the OP resource was held constant for all of the cases. The OP shell, as discussed in Section 13.2, was based a Cu price of $2.90/lb and a gold price of $1,200/lb.

It must be noted that the economic analysis conducted in this report includes the use of Inferred mineral resources and only provides a preliminary overview of the project economics based on broad, factored assumptions. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the inferred resources would be upgraded to a higher resource category. There is also no certainty that the results of this preliminary economic assessment would be realized.

In the economic model the first year of analysis starts with the decision point of the project and the completion of the EIA and preliminary environmental permit approval. In the model this point is three years in advance of the first production year. No capital was considered prior to the decision point but would likely be in the order of $50M spread over two years for definition drilling, feasibility study, metallurgical testing, EIA work and permitting.

## 20.2 Assumptions

The simplified cash flow model used in this study assumes that the project is 100% equity funded, as a result, no financing charges have been taken into account.

The assumptions used in the economic analyses are listed in Table 20.1 and Table 20.2 and in the CAPEX, OPEX and metallurgical sections of the report.

There is no guarantee that any of the copper prices estimated would be representative of future metals prices. In each case the selected metal prices were held constant for the life of the mine.

**Table 20.1: Metal Price Assumptions for the Three Economic Cases**

| Case | Copper Price ($/lb) | Gold Price ($/oz) | Comments |
|---|---|---|---|
| Case A | 2.90 | 1,200 | Approximate metal prices as per recent SEDAR postings. This is the base case used for OP optimization. |
| Case B | 3.30 | 1,350 | Approximate London Metal Exchange fixed 3-year average to the end of April, 2012. SRK considers this to be an optimistic case. |
| Case C | 3.70 | 1,500 | Approximate London Metal Exchange fixed 2-year average to the end of April, 2012. SRK considers this to be a very optimistic case. |

There is no guarantee that any of the copper or gold prices estimated would be representative of future metals prices. In each case the selected metal prices were held constant for the life of the mine.

**Table 20.2: Economic Model Assumptions Common to All Cases**

| Parameter | Unit | Value |
|---|---|---|
| *Flotation Recoveries* | | |
| Gold recovery | % | 60.8 |
| Copper recovery | % | Recovery = 12.2xLN(Cu grade) + 99.7 |
| Copper in concentrate | % | 30 |
| Gold in concentrate | g/t | variable |
| *Smelter Payables - Copper* | | |
| Gold in concentrate | % | 96.0 |
| Gold in concentrate deduction | g/t in conc | 1.0 |
| Copper in concentrate | % | 96.0 |
| Copper in concentrate deduction | % | 1.0 |
| Penalties | | 0.0 |
| *TC/RCs* | | |
| Gold refining charge | $/oz pay Au | 7.00 |
| Copper concentrate treatment | $/dmt conc | 70.00 |
| Copper refining charge | $/lb pay Cu | 0.07 |
| *Concentrate Transport* | | |
| Truck freight to Port | $/wmt conc | 20.00 |
| Port charges | $/wmt conc | 6.00 |
| Marine/transportation insurance | $/wmt conc | 1.00 |
| Ocean freight | $/wmt conc | 30.00 |
| Transport, marketing, ins, etc. | $/wmt conc | 57.00 |
| Moisture content | % | 8.00 |
| Transport, marketing, ins, etc. | $/dmt conc | 61.56 |
| *Costs* | | |
| CAPEX | $/t milled | See Section 20.1 |
| OPEX | M$ | See Section 20.2 |
| *Other* | | |
| Discount Rate | % | 7.0 |
| Royalties | % | 2.6 for Au and 1.3 for Cu |
| Tonnes per day to mill | t/d | 50,000 |
| Mining Recovery | % | 100 |
| Grade Factor (head grade/in situ grade) | % | 95 |

### 20.2.1 Taxation

In Turkey, multi-tax system is applied. The Turkish tax regime can be classified under three main headings (only taxes applicable to corporations are explained);

- Income Taxes:
  - Personal Income Taxes; and
  - Corporate Income Taxes: Turkish corporate tax is levied on the income and earning derived by corporations and corporate bodies. Corporate tax is payable on net profit and is applied at a rate of 20%.
- Taxes on Expenditure:
  - Value Added Tax: The most significant indirect tax in Turkey is VAT. VAT applies to the importation or supply of goods and services. The standard rate of VAT on taxable transactions is set at 10% in the VAT Law, but this rate was increased to 18% as of 15 May 2001.
- Special Consumption Tax:
  - Banking and Insurance Transaction Taxes (“BITT”): Transactions and services of banks, bankers and insurance companies are exempt from VAT but taxed at a rate of 5% for the money which they collect as interest, commission and fees. Other transactions of bankers are subject to VAT. Foreign exchange transactions are subject to 0.1% BITT.
- Stamp Tax:
  - Applies to a wide range of documents, including but not limited to, contracts, agreements, notes payable, letters of credit and letters of guarantee, financial statements and payrolls. Stamp duty is levied as a percentage of the value stated on the document at rates ranging from 0.15% to 0.75%. The Stamp Tax Law provides that each relevant party shall be responsible for payment of the total amount of stamp tax on the agreements.
- Special Communication Tax:
- Goods in the list below are subject to Special Consumption Taxes and are taxed at different rates:
  - List I is related to petroleum products, natural gas, lubricating oil, solvents and derivatives of solvents.
  - List II is related to automobiles and other vehicles, motorcycles, planes, helicopters, yachts.
  - List III is related to tobacco and tobacco products, alcoholic beverages and cola.
  - List IV is related to luxury products.
- Tax on Customs:
  - Goods imported from abroad are subject to tax. The economic model did not account for duties.
- Various Fees:
  - There are different types of fees: Judgment Fees, Notary Fees, Tax Judgment Fees, Title Deed Fees, Consulate Fees, Ship and Harbor Fees, Permit of License and Certificate Fees, Traffic Fees, Passport, Visa and Ministry of Foreign Affairs. Certification Fees.
- Taxes on Wealth
  - Inheritance and Gift Taxes

- Property Tax:
  - Property taxes are paid on the tax values of land and buildings at rates varying from 0,1% to 0.3%. In the case of the sale of a property a 1% levy is paid on the sales value by both the buyer and the seller.
- Motor Vehicle Tax:
  - Vehicles are subject to a tax at the time of registration.

None of these taxes or fees were used in the economic model. The payment of corporate income tax is complex and involves many considerations that are well beyond the scope of this report. The Halilağa Project is assumed to be exempt from VAT as the project involves the extraction of gold.

## Tax Assumptions in Economic Model

The tax modeling was performed primarily using nominal US dollars. The long-term USD inflation rate is adjustable in the model. 2% is used in the base case and is considered a reasonable estimate. The project value is relatively insensitive to this assumption. Nominal dollars are used in order to accurately calculate the value of deferred allowances, both in estimation of depletion, depreciation and amortisation, and in tax-loss carry-forward/back calculations.

**Corporate Tax Rate**

The corporate tax rate of 20% was used.

**Depreciation Rates**

Mobile equipment was depreciated using a double-declining balance approach, allowable as per Turkish tax code. The depreciation rate is 40%.

Fixed plant was depreciated using a Unit of Production (UOP) depletion-type calculation. This is consistent with the principles in the Turkish tax code of allowing depreciation over the useful life of the asset. An alternative of using a standard 4% that usually applies to fixed plant is considered too conservative, given that the fixed plant has little value beyond the life of the project. Conversely, using 20% for this type of asset is likely too aggressive. The UOP approach is considered a reasonable compromise.

Indirect costs associated with the project development were depreciated at 20% p.a., reflecting the nature of those types of costs. In practice, some indirect costs may be expensed against other income immediately, and some would be attached to the underlying asset and attract longer depreciation times. The 20% rate is considered a reasonable compromise.

In actuality, the depreciation treatment will be more complex than is modeled here, but given the preliminary nature of the economic analysis for the project, and the high-level aggregation of the costs, the assumptions used are considered an appropriate approximation.

Note that no assumptions of depreciation and depletion were made with respect to any current book-value of the deposit. Any such depletion or depreciation is complex and beyond the scope of this model.

The depreciation treatment is informed by a review of guidance documents available in the public domain. It must be noted that no country-specific tax expertise was consulted. It is recommended that country-specific expertise be obtained for feasibility-level studies.

**Loss carry-forward**

Tax Losses were carried forward as allowed under Turkish tax code.

**Loss Carry-back**

Tax losses from beyond the end of the project (associated exclusively with closure costs) were allowed to be carried back by 5 years. Turkish tax code allows 5-year carry-back in the case of company liquidation. Even if this were not the case, it is reasonable to assume that some tax benefit would be derived from the expenditure on closure costs. This could be by a combination of many approaches, for example:

- Early investment in closure activities allowing deductions (expensed or depreciated) against operating income
- Consolidated company positioning allowing deductions against other company income
- Cash or accounting provisioning for closure costs being deemed deductible for tax

The effect of allowing the loss carry-back is not material to the project value. More detailed advice and modeling should be sought for feasibility-level studies.

**Value Added Tax**

Whilst Turkey does have a value added tax, it was not modeled for this analysis. It is possible that the project would be completely exempt due to the export nature of the business. In any case, the effect of a fully refundable VAT on the project value is not material for a study at this level.

### 20.2.2 Royalties

Total government royalties were assumed to be 2.6% for Au and 1.3% for Cu as they are defined as 4th Group minerals as per the Turkish Mining Law.

## 20.3 Economic Model

Table 20.3 presents tha Case A economic model.

## Table 20.3: Case A Economic Model

| Section | Item | Unit | LOM Total or Avg. | YEAR | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | -3 | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 |
| **MATERIAL SCHEDULE** | | | | | | | | | | | | | | | | | | | | | |
| **Total All Deposits** | Operating Days | *days* | **4,951** | | | | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 206 | |
| | Waste tonnes | Mt | **234.0** | | | | 10.1 | 9.6 | 21.4 | 24.0 | 22.8 | 22.4 | 22.8 | 23.3 | 23.7 | 24.0 | 23.5 | 6.4 | - | - | |
| | Mineralized tonnes (Plant feed) | Mt | **243.3** | | | | 14.0 | 18.3 | 18.3 | 18.3 | 18.3 | 18.3 | 18.3 | 18.3 | 18.3 | 18.2 | 18.3 | 18.3 | 18.3 | 10.3 | |
| | Total Mining tonnes | Mt | **477.3** | | | | 24.1 | 27.9 | 39.6 | 42.2 | 41.0 | 40.7 | 41.0 | 41.6 | 41.9 | 42.2 | 41.8 | 24.7 | 18.3 | 10.3 | - |
| | Strip Ratio | t waste:t Plant feed | **0.96** | | | | 0.7 | 0.5 | 1.2 | 1.3 | 1.2 | 1.2 | 1.2 | 1.3 | 1.3 | 1.3 | 1.3 | 0.4 | - | - | - |
| | Mill Head GradeGold | g/t Au | **0.30** | | | | 0.49 | 0.38 | 0.41 | 0.28 | 0.35 | 0.19 | 0.23 | 0.27 | 0.24 | 0.21 | 0.29 | 0.34 | 0.27 | 0.19 | |
| | Copper | % Cu | **0.28** | | | | 0.83 | 0.37 | 0.34 | 0.29 | 0.29 | 0.17 | 0.20 | 0.24 | 0.23 | 0.21 | 0.19 | 0.24 | 0.25 | 0.21 | |
| | Contained Metal Gold | Mg Au | **72** | | | | 6.86 | 6.94 | 7.48 | 5.11 | 6.39 | 3.47 | 4.20 | 4.93 | 4.38 | 3.83 | 5.29 | 6.21 | 4.93 | 1.96 | 0.00 |
| | | koz Au | **2,314** | | | | 221 | 223 | 241 | 164 | 205 | 111 | 135 | 158 | 141 | 123 | 170 | 200 | 158 | 63 | - |
| | Copper | t Cu | **688,938** | | | | 116,200 | 67,525 | 62,050 | 52,925 | 52,925 | 31,025 | 36,500 | 43,800 | 41,975 | 38,304 | 34,675 | 43,800 | 45,625 | 21,609 | - |
| | | Mlb Cu | **1,519** | | | | 256 | 149 | 137 | 117 | 117 | 68 | 80 | 97 | 93 | 84 | 76 | 97 | 101 | 48 | - |
| **COPPER FLOTATION CIRCUIT** | | | | | | | | | | | | | | | | | | | | | |
| **Recovery to Concentrate** | Au | % | **60.8** | | | | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | 60.8 | |
| | Cu | % | **85.5** | | | | 97.4 | 87.6 | 86.5 | 84.6 | 84.6 | 78.1 | 80.1 | 82.3 | 81.8 | 80.7 | 79.4 | 82.3 | 82.8 | 80.7 | |
| **Concentrate Grade** | Au | g/t Au | **22** | | | | 11 | 21 | 25 | 21 | 26 | 26 | 26 | 25 | 23 | 23 | 35 | 31 | 24 | 20 | - |
| | Cu | % Cu | **30** | | | | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 |
| **Pull factor (based on copper)** | Tonnes milled per t Cu con. | t milled/t Cu con. | **123.9** | | | | 37 | 93 | 102 | 122 | 122 | 226 | 187 | 152 | 160 | 177 | 199 | 152 | 145 | 177 | - |
| **Concentrate Tonnes** | Dry | dmt | **1,963,619** | | | | 377,366 | 197,106 | 178,991 | 149,245 | 149,245 | 80,750 | 97,412 | 120,142 | 114,410 | 102,987 | 91,818 | 120,142 | 125,906 | 58,099 | - |
| | Wet | wmt | **2,120,708** | | | | 407,556 | 212,874 | 193,310 | 161,184 | 161,184 | 87,210 | 105,205 | 129,754 | 123,563 | 111,226 | 99,164 | 129,754 | 135,978 | 62,747 | - |
| | Dry | t/day | 396.6 | | | | 1,034 | 540 | 490 | 409 | 409 | 221 | 267 | 329 | 313 | 282 | 252 | 329 | 345 | 282 | - |
| | Wet | t/day | 428.4 | | | | 1,117 | 583 | 530 | 442 | 442 | 239 | 288 | 355 | 339 | 305 | 272 | 355 | 373 | 305 | - |
| **Contained Metal** | Au | koz Au | **1,407** | | | | 134 | 136 | 146 | 100 | 125 | 68 | 82 | 96 | 86 | 75 | 103 | 121 | 96 | 38 | - |
| | Cu | Mlb Cu | **1,299** | | | | 250 | 130 | 118 | 99 | 99 | 53 | 64 | 79 | 76 | 68 | 61 | 79 | 83 | 38 | - |
| **NET SMELTER RETURN** | | | | | | | | | | | | | | | | | | | | | |
| **Metal Price** | Au | $/oz | 1,200 | | | | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 | 1,200 |
| | Cu | $/lb | 2.90 | | | | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 | 2.90 |
| **Payables** | Au | % | 96.0 | | | | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 |
| | Cu | % | 96.0 | | | | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 | 96.0 |
| **Copper Concentrate Payables** | Au Payables | koz Au | 1,290 | | | | 117 | 124 | 135 | 91 | 115 | 63 | 76 | 89 | 79 | 69 | 96 | 113 | 89 | 35 | - |
| | Cu Payables | Mlb Cu | 1,247 | | | | 240 | 125 | 114 | 95 | 95 | 51 | 62 | 76 | 73 | 65 | 58 | 76 | 80 | 37 | - |
| | **Gross Payable Revenue** | M$ | 5,163 | | | | 835 | 512 | 491 | 384 | 413 | 224 | 270 | 328 | 305 | 272 | 285 | 357 | 338 | 149 | - |
| **Offsite costs** | Total Conc. transport costs | M$ | 121 | | | | 23 | 12 | 11 | 9 | 9 | 5 | 6 | 7 | 7 | 6 | 6 | 7 | 8 | 4 | - |
| | Treatment Charge Cu Concentrate | M$ | 137 | | | | 26 | 14 | 13 | 10 | 10 | 6 | 7 | 8 | 8 | 7 | 6 | 8 | 9 | 4 | - |
| | Refining charge Au | M$ | 9 | | | | 1 | 1 | 1 | 1 | 1 | 0 | 1 | 1 | 1 | 0 | 1 | 1 | 1 | 0 | - |
| | Refining charge Cu | M$ | 87 | | | | 17 | 9 | 8 | 7 | 7 | 4 | 4 | 5 | 5 | 5 | 4 | 5 | 6 | 3 | - |
| | Subtotal TC/RC and transport | M$ | 355 | | | | 67 | 36 | 32 | 27 | 27 | 15 | 18 | 22 | 21 | 19 | 17 | 22 | 23 | 10 | - |
| | Payable Au | M$ | 1,539 | | | | 140 | 148 | 161 | 109 | 138 | 75 | 90 | 106 | 94 | 82 | 115 | 135 | 106 | 42 | - |
| | Payable Cu | M$ | 3,270 | | | | 628 | 328 | 298 | 249 | 249 | 134 | 162 | 200 | 191 | 172 | 153 | 200 | 210 | 97 | - |
| **Cu Con NSR** | **Total Cu Conc. NSR (ex. Roylaties)** | **M$** | 4,809 | | | | 768 | 476 | 459 | 357 | 386 | 209 | 253 | 306 | 284 | 253 | 268 | 335 | 315 | 138 | - |
| **Unit NSR** | **Unit NSR** | **$/t** | 19.20 | | | | 54.87 | 26.10 | 25.15 | 19.59 | 21.15 | 11.46 | 13.84 | 16.77 | 15.58 | 13.90 | 14.69 | 18.33 | 17.28 | | |

| | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Royalty** | **Royalty** | **M$** | 83 | | | | 12 | 8 | 8 | 6 | 7 | 4 | 4 | 5 | 5 | 4 | 5 | 6 | 5 | 2 | - |
| **Mining Revenue** | **NSR including Royalty** | **M$** | 4,726 | | | | 756 | 468 | 451 | 351 | 379 | 205 | 248 | 301 | 279 | 249 | 263 | 328 | 310 | 136 | - |
| **OPERATING COST** | | | | | | | | | | | | | | | | | | | | | |
| **Unit OPEX** | Mining | $/t mined | 1.74 | | | | 1.72 | 1.67 | 1.54 | 1.60 | 1.62 | 1.62 | 1.71 | 1.75 | 1.77 | 1.77 | 1.78 | 1.94 | 2.06 | 2.68 | |
| | | $/t milled | 3.40 | | | | 2.96 | 2.55 | 3.34 | 3.69 | 3.64 | 3.62 | 3.84 | 3.99 | 4.07 | 4.10 | 4.08 | 2.62 | 2.06 | 2.68 | |
| | Processing | $/t milled | 4.69 | | | | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | 4.69 | |
| | Tailings Management | $/t milled | 0.23 | | | | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | 0.23 | |
| | Environment and water treatment | $/t milled | 0.05 | | | | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | |
| | G&A | $/t milled | 0.42 | | | | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | 0.42 | |
| **Total OPEX** | Mining | M$ | 828.2 | | | | 41 | 47 | 61 | 67 | 66 | 66 | 70 | 73 | 74 | 75 | 75 | 48 | 38 | 28 | - |
| | Processing | M$ | 1,141.0 | | | | 66 | 86 | 86 | 86 | 86 | 86 | 86 | 86 | 86 | 86 | 86 | 86 | 86 | 48 | - |
| | Tailings Management | M$ | 56.0 | | | | 3 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 2 | |
| | Environment and water treatment | M$ | 12.2 | | | | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | |
| | G&A | M$ | 102.2 | | | | 6 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 4 | - |
| *1.00* | **Total OPEX** | **M$** | 2,139 | | | | 117 | 145 | 159 | 166 | 165 | 164 | 168 | 171 | 173 | 173 | 173 | 146 | 136 | 83 | - |
| | Unit OPEX per t processed | $/t processed | 8.79 | | | | 8.35 | 7.94 | 8.73 | 9.08 | 9.03 | 9.01 | 9.23 | 9.38 | 9.46 | 9.49 | 9.47 | 8.01 | 7.45 | 8.07 | - |
| **Cost/Payable Metal** | Unit OPEX per Au oz | $/oz pay Au | 1,659 | | | | 998 | 1,168 | 1,182 | 1,816 | 1,430 | 2,626 | 2,224 | 1,929 | 2,193 | 2,519 | 1,792 | 1,296 | 1,534 | 2,379 | - |
| | Unit OPEX per Au oz w/ Cu credit | $/oz pay Au eq. | -876 | | | | -4,368 | -1,478 | -1,028 | -906 | -726 | 478 | 83 | -325 | -228 | 23 | 207 | -478 | -833 | -393 | - |
| | Unit OPEX per Cu lb | $/lb pay Cu | 1.72 | | | | 0.49 | 1.16 | 1.40 | 1.75 | 1.74 | 3.21 | 2.72 | 2.24 | 2.38 | 2.65 | 2.97 | 1.92 | 1.70 | 2.25 | - |
| | Unit OPEX per Cu lb w/ Au credit | $/lb pay Cu eq. | 0.48 | | | | -0.10 | -0.02 | -0.01 | 0.60 | 0.29 | 1.75 | 1.26 | 0.86 | 1.08 | 1.39 | 0.99 | 0.15 | 0.38 | 1.12 | - |
| **NET PRE-TAX OPERATING INCOME** | | **M$** | 2,587 | | | | 639 | 323 | 292 | 186 | 214 | 41 | 80 | 129 | 107 | 76 | 90 | 182 | 174 | 53 | - |
| **TAXATION** | | | | | | | | | | | | | | | | | | | | | |
| **NET PRE-TAX OPERATING INCOME (Nominal)** | | **M$** | 3,000 | - | - | - | 679 | 350 | 322 | 209 | 246 | 48 | 95 | 158 | 133 | 96 | 117 | 241 | 234 | 73 | - |
| UOP Depletion, Depreciation and Amortization Allowance | | M$ | 1,263 | - | - | - | 130 | 128 | 137 | 128 | 121 | 52 | 61 | 70 | 64 | 59 | 53 | 50 | 47 | 27 | 30 |
| **Net income after D and A** | | **M$** | 1,737 | - | - | - | 549 | 222 | 185 | 81 | 125 | -4 | 34 | 88 | 69 | 38 | 63 | 191 | 188 | 45 | -30 |
| Opening Balance | | | | - | - | - | - | - | - | - | - | - | -4 | - | - | - | - | - | - | - | - |
| New tax losses | | | | - | - | - | - | - | - | - | - | -4 | - | - | - | - | - | - | - | - | -30 |
| Carried Back | | | Y | - | - | - | - | - | - | - | - | - | - | - | - | -30 | -28 | -27 | -26 | -26 | -0 |
| Claimed tax losses | | | | - | - | - | - | - | - | - | - | - | 4 | - | - | 30 | 28 | 27 | 26 | 26 | - |
| Closing Balance | | | | - | - | - | - | - | - | - | - | -4 | - | - | - | - | - | - | - | - | -30 |
| | | | | | | | | | | | | | | | | | | | | | |
| Net Taxable Income | | M$ | 1,737 | - | - | - | 549 | 222 | 185 | 81 | 125 | - | 30 | 88 | 69 | 8 | 35 | 164 | 161 | 19 | - |
| Federal Income Tax | 20% | M$ | 347 | - | - | - | 110 | 44 | 37 | 16 | 25 | - | 6 | 18 | 14 | 2 | 7 | 33 | 32 | 4 | - |
| **Operating after-tax Cash Flow (Nominal)** | | **M$** | 2,653 | - | - | - | 569 | 305 | 285 | 193 | 221 | 48 | 89 | 140 | 119 | 95 | 110 | 208 | 202 | 69 | - |
| **Operating after-tax Cash Flow(Real)** | | **M$** | 2,284 | - | - | - | 536 | 282 | 258 | 171 | 193 | 41 | 75 | 115 | 96 | 75 | 85 | 157 | 150 | 50 | - |
| **CAPITAL COST** | | | | | | | | | | | | | | | | | | | | | |
| | Primary mining equipment fleet | M$ | 171.8 | | | 65.0 | 3.9 | 36.1 | 3.9 | - | - | 24.5 | 26.9 | 6.9 | 4.6 | - | - | - | - | - | |
| | Support equipment | M$ | 8.3 | | | 8.3 | 0.0 | - | - | 0.0 | - | 0.0 | 0.0 | 0.0 | - | - | - | - | - | - | |
| | Misc. | M$ | 7.1 | | | 4.8 | - | - | - | - | - | - | 2.3 | - | - | - | - | - | - | - | |
| | Spare parts | M$ | 9.4 | | | 3.9 | 0.2 | 1.8 | 0.2 | 0.0 | - | 1.2 | 1.5 | 0.3 | 0.2 | - | - | - | - | - | |
| | Topsoil stripping and mining preparation | M$ | 11.7 | - | - | 11.7 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| | Crushing | M$ | 50.2 | - | 25.1 | 25.1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| | Process Plant | M$ | 199.9 | - | 99.9 | 99.9 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |

| | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Concentrate Loadout | M$ | 1.7 | - | - | 1.7 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| | Tailings Facility and Water Reservoir | M$ | 105.2 | 6.2 | 5.8 | 92.2 | - | - | - | - | - | 1.0 | - | - | - | - | - | - | - | - | |
| | Utilities | M$ | 38.8 | - | 19.4 | 19.4 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| | On-site Infrastructure (inc. Reservoir) | M$ | 33.3 | - | 16.7 | 16.7 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| | Off-site Infrastructure | M$ | 13.8 | - | 13.8 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| | Indirect Costs | M$ | 148.7 | 45.1 | 39.0 | 64.5 | - | - | - | - | - | 0.1 | - | - | - | - | - | - | - | - | |
| | Owner's Costs | M$ | 41.9 | - | 20.9 | 20.9 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| | Sustaining | M$ | 51.6 | - | - | - | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | 4.3 | - | - | - |
| | Closure | M$ | 75.0 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 75.0 |
| | CAPEX w/o Contingency | M$ | 968.3 | 51.3 | 240.5 | 434.2 | 8.4 | 42.2 | 8.4 | 4.3 | 4.3 | 31.1 | 35.0 | 11.5 | 9.1 | 4.3 | 4.3 | 4.3 | - | - | 75.0 |
| | Contingency | M$ | 200.4 | 12.8 | 60.1 | 89.8 | 1.3 | 3.0 | 1.3 | 1.1 | 1.1 | 2.6 | 2.6 | 1.4 | 1.3 | 1.1 | 1.1 | 1.1 | - | - | 18.8 |
| | Working CAPEX | M$ | - | | | | 39.0 | | | | | | | | | | | | | -39.0 | |
| | **TOTAL CAPITAL COST** | **M$** | 1,168.7 | 64.1 | 300.7 | 524.0 | 48.6 | 45.2 | 9.7 | 5.4 | 5.4 | 33.7 | 37.6 | 13.0 | 10.4 | 5.4 | 5.4 | 5.4 | - | -39.0 | 93.8 |
| **Cash Flow** | | | | | | | | | | | | | | | | | | | | | |
| Pre-Tax | Undiscounted Pre-tax Cash-flow | **M$** | 1,418 | (64) | (301) | (524) | 591 | 278 | 282 | 180 | 209 | 7 | 42 | 116 | 96 | 71 | 85 | 177 | 174 | 92 | (94) |
| | Undiscounted Cumulative Pre-tax Cash-flow | M$ | | (64) | (365) | (889) | (298) | (20) | 262 | 442 | 651 | 658 | 701 | 817 | 913 | 984 | 1,069 | 1,246 | 1,420 | 1,512 | 1,418 |
| 7% | Discounted Pre-tax Cash-flow | M$ | 675 | (64) | (281) | (458) | 482 | 212 | 201 | 120 | 130 | 4 | 23 | 59 | 46 | 31 | 35 | 69 | 63 | 31 | (30) |
| | Discounted Cumulative Pre-tax Cash-flow | M$ | | (64) | (345) | (803) | (321) | (108) | 93 | 213 | 343 | 347 | 370 | 429 | 475 | 506 | 542 | 610 | 673 | 704 | 675 |
| After-tax | Undiscounted After-tax Cash-flow | **M$** | 1,115 | (64) | (301) | (524) | 487 | 237 | 248 | 166 | 187 | 7 | 37 | 102 | 85 | 69 | 79 | 152 | 150 | 89 | (94) |
| | Undiscounted Cumulative After-tax Cash-flow | M$ | | (64) | (365) | (889) | (401) | (164) | 84 | 250 | 437 | 444 | 481 | 583 | 669 | 738 | 817 | 970 | 1,120 | 1,209 | 1,115 |
| 7% | Discounted After-tax Cash-flow | M$ | 474 | (64) | (281) | (458) | 398 | 181 | 177 | 110 | 117 | 4 | 20 | 52 | 41 | 31 | 33 | 59 | 54 | 30 | (30) |
| | Discounted Cumulative After-tax Cash-flow | M$ | 1,200 | (64) | (345) | (803) | (405) | (224) | (47) | 63 | 180 | 184 | 204 | 256 | 297 | 328 | 361 | 420 | 474 | 504 | 474 |

## 20.4 Economic Results

The results of the economic analyses are shown in Table 20.4. Based on the assumptions made in this study, the pre-tax net present value, using a discount rate of 7%, ("PT $NPV_{7\%}$") is estimated to be $675M yielding an internal rate of return ("IRR") of 25%. The PT $NPV_{7\%}$ and IRR go up considerably with Cases B and C as higher metal prices were used in these Cases. The after-tax ("AT") $NPV_{7\%}$ for Case A was estimated to be $474M with an IRR of 20%. Clearly, the potential economic return of the project would be dependent upon the selection of long-term copper and gold prices.

**Table 20.4: Economic Results by Case**

| Parameter | Unit | Pre-tax Results | After-tax Results |
|---|---|---|---|
| **Case A ($2.90/lb Cu and $1,200/oz Au)** | | | |
| $NPV_{0\%}$ | M$ | 1,418 | 1,115 |
| $NPV_{7\%}$ | M$ | 675 | 474 |
| IRR | % | 26 | 20 |
| Payback period | Production years | 2.1 | 2.7 |
| **Case B ($3.30/lb Cu and $1,350/oz Au)** | | | |
| $NPV_{0\%}$ | M$ | 2,099 | 1,660 |
| $NPV_{7\%}$ | M$ | 1,081 | 799 |
| IRR | % | 35 | 27 |
| Payback period | Production years | 1.5 | 2.1 |
| **Case C ($3.70/lb Cu and $1,500/oz Au)** | | | |
| $NPV_{0\%}$ | M$ | 2,779 | 2,204 |
| $NPV_{7\%}$ | M$ | 1,487 | 1,124 |
| IRR | % | 43 | 34 |
| Payback period | Production years | 1.2 | 1.6 |

## 20.5 Sensitivity Analysis

Sensitivity analysis was performed using metal prices, mill head grade, CAPEX and OPEX as variables. The value of each variable was changed plus and minus 20% independently while all other variables were held constant.

The results of the sensitivity analysis are shown on graphs in Figure 20.1 to Figure 20.3 and Table 20.5 and Table 20.6.

As with most mining projects, the NPV of the project is most affected by the price of metal and in this case, the price of Cu and gold. This holds true for all the cases in this study. In Case A, for example, a 20% increase in metal prices leads to an increase in the after -tax ("AT") $NPV_{7\%}$ from $4740M to $958M. The plant feed grade and recovery also have a similar large impact with a $484M AT $NPV_{7\%}$ gain for a 20% increase in grade or recovery. Conversely, decreases in metal prices, plant feed grades or recoveries have a correspondingly large negative effect on the AT $NPV_{7\%}$.

All cases indicate they are slightly more sensitive to OPEX than CAPEX. For Case A, a 20% increase in OPEX drops the AT $NPV_{7\%}$ to $282M or -40% from $474M. A 20% increase in the Case A CAPEX yields a $167M drop in AT $NPV_{7\%}$ from $474M to $307M.

**Table 20.5: Pre-tax $NPV_{7\%}$ Sensitivities by Case**

| Case | Variable | Pre-tax $NPV_{7\%}$ (M$) | | |
|---|---|---|---|---|
| | | -20% Variance | 0% Variance | 20% Variance |
| Case A (Base Case) | CAPEX | 868 | 675 | 482 |
| | OPEX | 913 | 675 | 437 |
| | Metal Price or Grade | 70 | 675 | 1,2799 |
| Case B | CAPEX | 1,274 | 1,081 | 888 |
| | OPEX | 1,319 | 1,081 | 843 |
| | Metal Price or Grade | 395 | 1,081 | 1,767 |
| Case C | CAPEX | 1,680 | 1,487 | 1,293 |
| | OPEX | 1,725 | 1,487 | 1,248 |
| | Metal Price or Grade | 719 | 1,487 | 2,254 |

**Table 20.6: After-tax $NPV_{7\%}$ Sensitivities by Case**

| Case | Variable | After-tax $NPV_{7\%}$ (M$) | | |
|---|---|---|---|---|
| | | -20% Variance | 0% Variance | 20% Variance |
| Case A (Base Case) | CAPEX | 642 | 474 | 307 |
| | OPEX | 666 | 474 | 282 |
| | Metal Price or Grade | -17 | 474 | 958 |
| Case B | CAPEX | 967 | 799 | 632 |
| | OPEX | 990 | 799 | 608 |
| | Metal Price or Grade | 250 | 799 | 1,348 |
| Case C | CAPEX | 1,292 | 1,124 | 956 |
| | OPEX | 1,315 | 1,124 | 933 |
| | Metal Price or Grade | 510 | 1,124 | 1,738 |

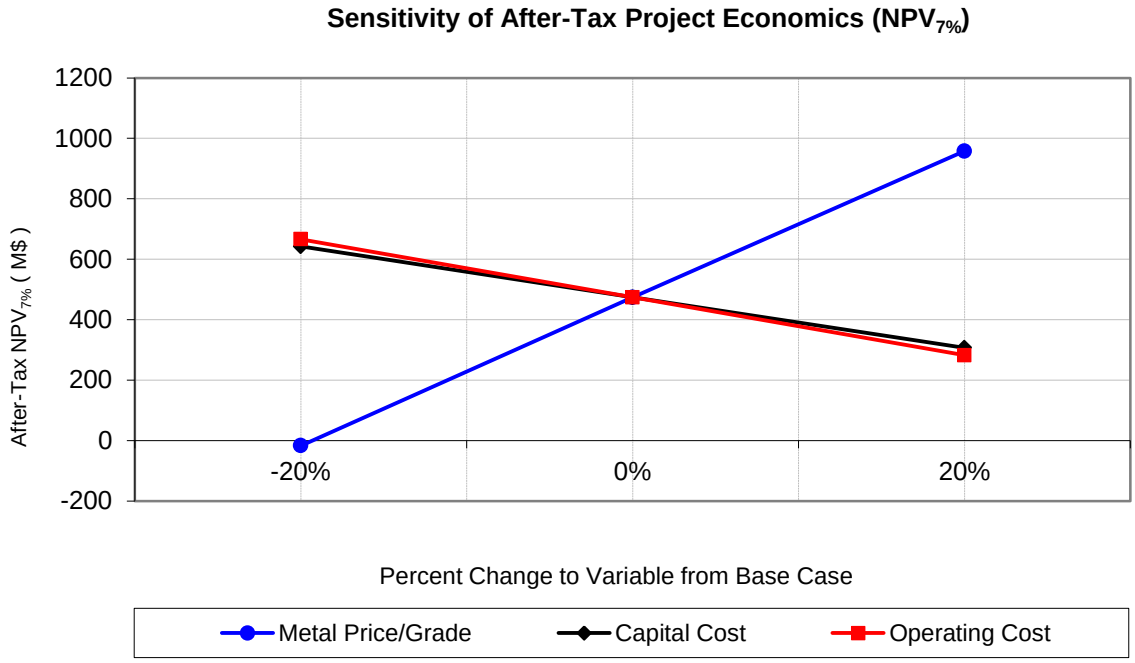



**Figure 20.1: Case A After-Tax Net Present Value Sensitivity Graph**

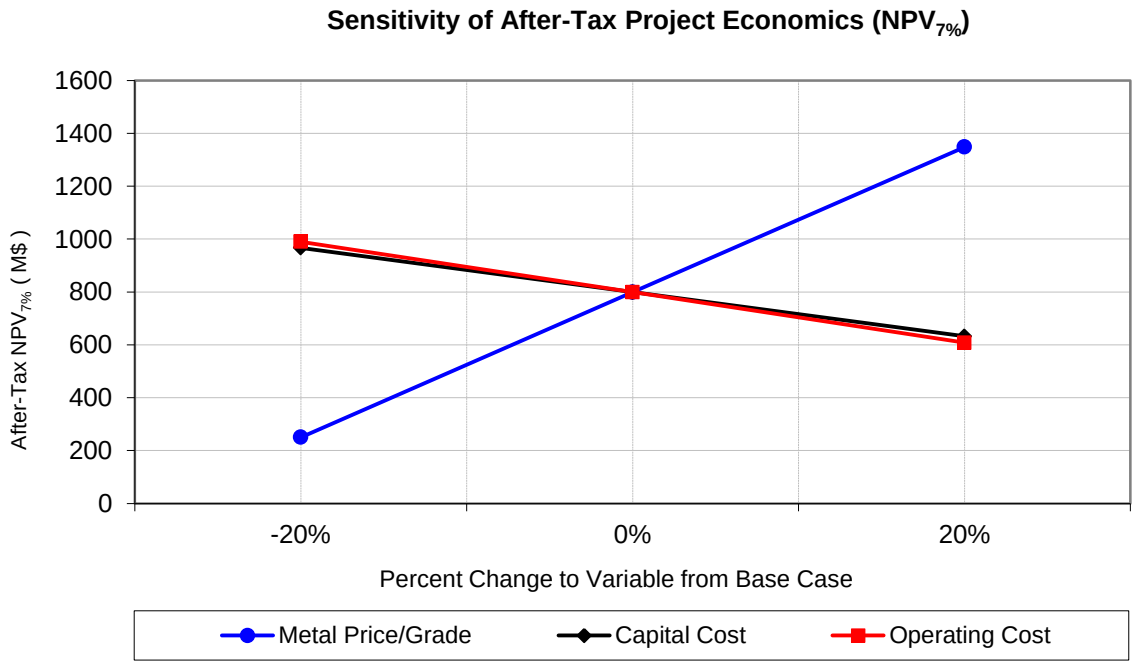



**Figure 20.2: Case B Net Present Value Sensitivity Graph**

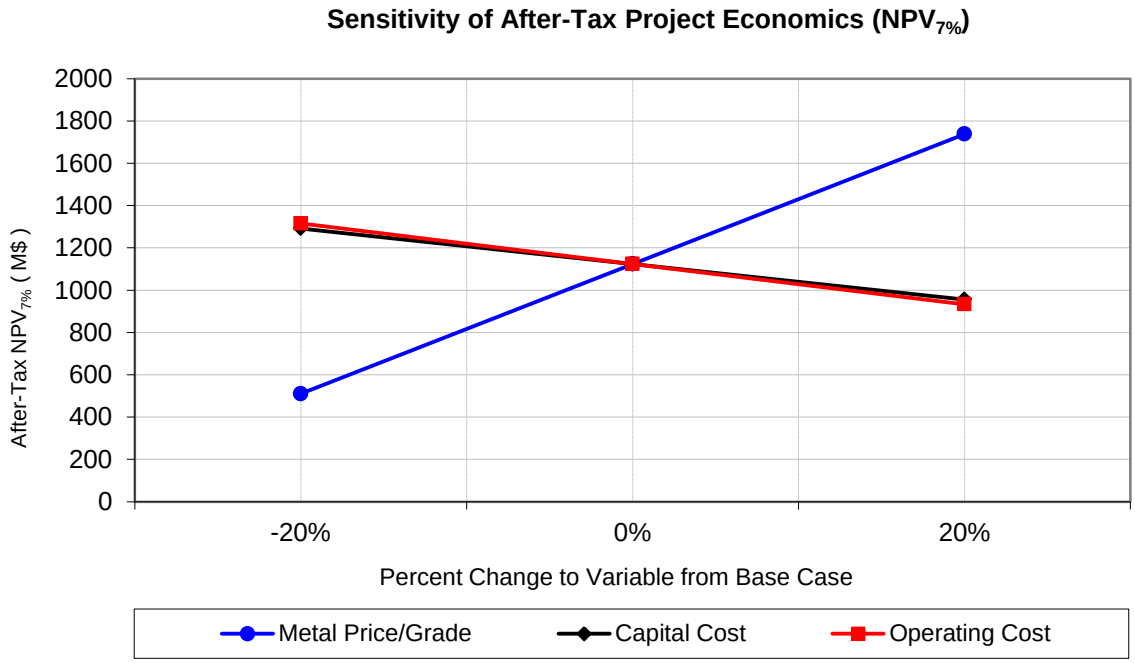



**Figure 20.3: Case C Net Present Value Sensitivity Graph**

### Limitations of Sensitivity Analysis

The selection of 20% to use as a variation is standard, but arguably somewhat arbitrary as it does not reflect the variability and uncertainty with respect to those input parameters.

## 20.6 Break-even Scenarios

Using the pit design developed for the PEA LOM plan, and a discount rate of 7%, the after-tax break-even metal prices for the project are 19% lower than those used in Case A. I.e. if metal prices fall by 19% from Case A to approximately \$972/oz Au and \$2.35/lb Cu, the project shows an after-tax $NPV_{7\%}$ of \$0 and an IRR of 7%.

## 20.7 Payback Period

Payback period for the after-tax base case (Case A) at a discount rate of 7% is approximately 2.7 years. The short payback period is driven by the mining of a low strip ratio high-grade zone in the first year of operation. The average grade of this zone is more than double the overall LOM plan grade (both gold and copper grades).

## 20.8 Mine Life

The LOM is 14 production years. The first year of production includes near-surface high-grade material that does not require pre-stripping in year -1. Waste mining ends in Year 12 with Years 13 and 14 projecting only in mineralized material being mined.

# 21 Adjacent Properties

## 21.1 Adjacent Properties

The majority of adjacent exploration licenses are auction licenses which would be put up for auction in 2012. The closest operation license, namely the Ağı Dağı project, is located 12 km SE of Kestane and is owned by Alamos Gold Inc. (“Alamos Gold”) (see Figure 21.1). Alamos Gold is planning to put the Ağı Dağı project into operation within 2 years.

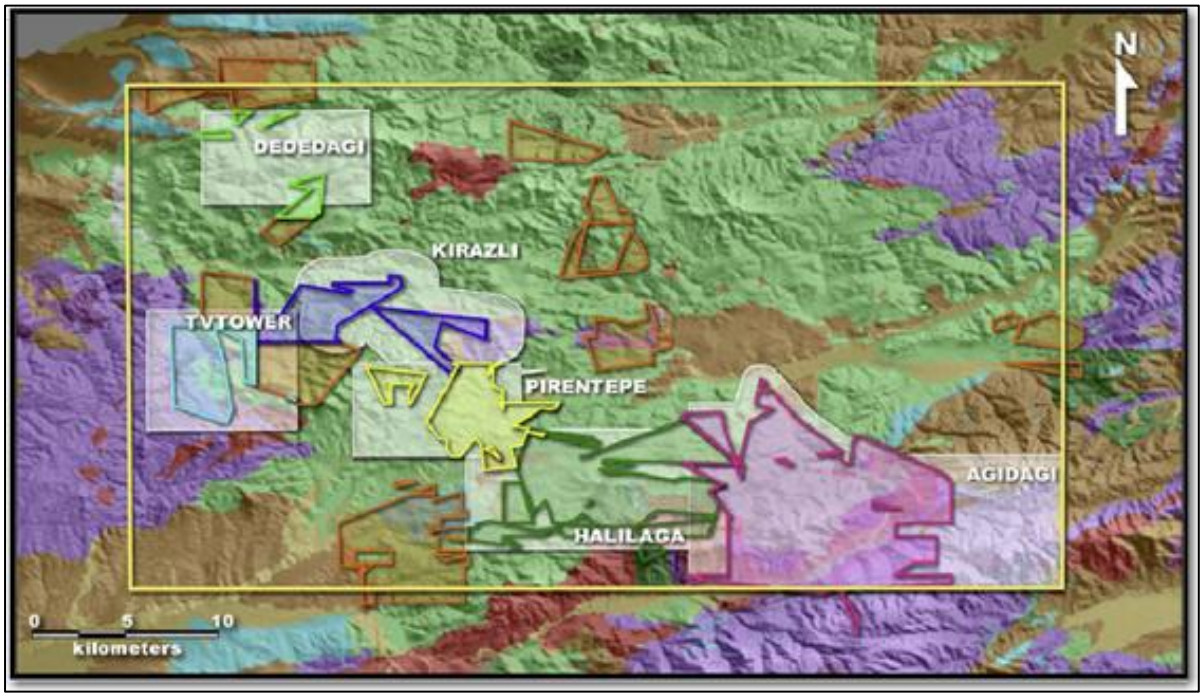



**Figure 21.1: Location of Halilağa and Adjacent Properties**

In January of 2010, the Ağı Dağı and Kirazli Projects were purchased by Alamos Gold for a total consideration of $40 million cash and issued an aggregate of 4 million Alamos common shares in total consideration to Teck (60%) and Fronteer (40%) for acquiring these two projects through the purchase of certain Turkish subsidiaries held by Teck and Fronteer. The Alamos Gold website describes the Ağı Dağı and Kirazli Projects as high sulphidation epithermal gold and silver deposits. Table 21.1 summarizes the contained resources. The reader is cautioned that this information is supplied for information purposes only and in the interest of providing a complete report. However, there has been no work in the creation of this report to link the deposits or to draw definitive comparisons or associations. Information provided in this section is not necessarily indicative of the mineralization on the Halilağa property. In addition, the author has not confirmed this publicly available disclosure and has not talked to Alamos to confirm the data.

The previously named Pirentepe and Halilağa North projects are now within the Halilağa Project and are owned by Truva Bakir.

**Table 21.1: 2010 Resource for the Ağı Dağı and Kirazli Properties (as taken from Gray and Kirkham 2012)**

| 2010 Ağı Dağı and Kirazli Mineral Resources[1]<br>Measured and Indicated and Inferred Mineral Resources | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Resource Area | Measured and Indicated Mineral Resources[2] | | | | | Inferred Resources[3] | | | | |
| | Tonnes (000s) | Grade (g/t Au)[3] | Grade (g/t Ag) | Contained Ounces Au | Contained Ounces Ag | Tonnes (000s) | Grade (g/t Au)[3] | Grade (g/t Ag) | Contained Ounces Au | Contained Ounces Ag |
| Ağı Dağı | 66,323 | 0.61 | 3.59 | 1,299,266 | 7,648,072 | 22,341 | 0.53 | 2.71 | 383,383 | 1,945,179 |
| Kirazli | 27,060 | 0.76 | 8.92 | 660,528 | 7,762,855 | 4,108 | 0.56 | 11.21 | 73,612 | 1,480,766 |
| **TOTAL** | **93,383** | **0.65** | **5.13** | **1,959,794** | **15,410,927** | **26,449** | **0.53** | **4.03** | **456,995** | **3,425,945** |

[1]The economic cut-off grade is 0.2 g/t Au

[2]Measured and indicated resources for the Ağı Dağı project, which includes Kirazli and the Baba, Deli and Fire tower zones, are pit constrained, with cut-off determined as a net of process value of $0.10 per tonne, for each model block. The determinant was based on a $1,150 per ounce gold price and $22.50 per ounce silver price, a June 2011 resource model for Ağı Dağı and a July 2011 resource model for Kirazli, average pit slope angle of 38˚ and estimated costs and recoveries based on the ongoing pre-feasibility study specifications. The resources were then tabulated by gold cut-off grade. Indicated mineral resources were for the Kirazli project are pit constrained, with cut-off determined as a net of process value of $0.10 per tonne, for each model block. The determination was based on a $1,025 per ounce gold price, a February 2011 resource model and estimated metal recovery and costs based on the March 2010 Preliminary Economic Assessment. Mineral resources were then tabulated by gold cut-off grade.

[3]Inferred resources for the Ağı Dağı project which includes Kirazli and the Baba, Deli and Fire tower zones, are pit constrained, with cut-off determined as a net process value of $0.10 per tonne, for each model block. The determination was based on a $1,025 per ounce gold price and a $22.50 per ounce silver price, a June 2011 resource model, average pit slope angle of 38˚ and estimated costs and recoveries based on the ongoing pre-feasibility study specifications. Resources were then tabulated by gold cut-off grade.

# 22 Interpretation and Conclusions

## 22.1 Interpretation and Conclusions

Industry standard mining, process design, construction methods and economic evaluation practices were used to assess the Halilağa Project. In SRK's opinion, there is adequate geological and other pertinent data available to generate a PEA.

Based on current knowledge and assumptions, the results of this study show that the project has positive economics (within the very preliminary parameters of a PEA) and should be advanced to the next level of study, a preliminary feasibility study ("PFS").

As with almost all mining ventures, there are a large number of risks and opportunities that can influence the outcome of the Halilağa Project. Most of these risks and opportunities are based on a lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal price, exchange rates, etc.). The following section identifies the most significant potential risks and opportunities currently identified for the project, almost all of which are common to mining projects at this early stage of project development.

Subsequent higher-level engineering studies would need to further refine these risks and opportunities, identify new ones, and define mitigation or opportunity implementation plans.

While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the project.

The study achieved its original objective of providing a preliminary review of the potential economic viability of the Halilağa Project.

# 23 Risks and Opportunities

## 23.1 Risks

As with most early-stage projects there are a multitude of risks that could affect the economic potential of the project. Many of these risks are based on lack of knowledge and can be managed with appropriate engineering. External risks are out of the project proponents control and are much harder to anticipate and mitigate although, in many instances, some risk reduction can be achieved. Table 23.1 and Table 23.2 identify some of the most significant internal and external project risks, potential severity and possible mitigation approaches.

**Table 23.1: Internal Project Risks**

| Risk | Explanation | Potential Impact | Possible Risk Mitigation |
|---|---|---|---|
| Recoveries | Flotation recoveries are largely based on a limited number of samples and tests. | If life-of-mine recovery of Cu or gold is lower than projected, project economics could be negatively impacted. | Conduct a flotation variability study to determine how material from different areas responds and what the average Cu and gold recoveries are likely to be. |
| Water Supply | The ability to provide water to the processing facility requires the establishment of a water reservoir. | Failure to secure an adequate water supply could reduce the size of the operation and impact economics due to possible competition with agricultural usage in the project area. | Investigations on water rights need to continue and be documented in the next level of study. |
| CAPEX and OPEX | The ability to achieve the estimated CAPEX and OPEX costs would be an important element of success. The size of the OP is very sensitive to OPEX | An increase in OPEX of 20% would reduce the PT $NPV_{7\%}$ and the size of the optimized OP. | Further cost accuracy with the next level of study as well as the active investigation of potential cost-reduction measures |
| Permit Acquisition | The ability to secure a mining permit is of paramount importance as is the negotiation with current farmers and other stakeholders. Currently, the tailings dam lies partly out of the claim boundary | Failure to secure a mining permit would stop the project.<br>Failure to secure the location of the tailings dam could lead to significant increases in TMF CAPEX. | The development of close relationship with the communities and government along with a thorough Environmental Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. |
| Development Schedule | The development could be delayed for a number of reasons and could impact project economics and the ability to mine during a period of high metals prices. | A change in schedule would alter the project economics once project construction and mining commences. | If an aggressive schedule is to be followed, PFS field work should begin ASAP. |
| Water Management and Geochemistry | It has been assumed that the waste dumps and tailings ponds do not need to be lined and acid rock drainage and metal leaching (“ARD/ML”) will not be a problem. No water treatment facility was budgeted. | If ARD/ML testing indicates that geochemistry would be an issue the liners may have to be placed under the TMF and/or the WRFs. This could add CAPEX and OPEX costs to the project but insufficient testing has been done to date. | Adequate testing of tailings and waste materials needs to be done to determine if there is an ARD/ML issue. |
| Inability to upgrade inferred resources to measured or indicated | The PEA mine plan uses 44% inferred resources which cannot be used at a higher level of study | If none of the inferred resources can be upgraded to indicated then the mineable tonnage would be cut | A well planned definition drilling campaign, renewed geostatistical analysis and resource estimation needs |

| Risk | Explanation | Potential Impact | Possible Risk Mitigation |
|---|---|---|---|
| | | byhalf of what is presented here and would make the project uneconomic | to be undertaken to determine the amount of inferred resource that can be converted |
| TMF Location and Stability | The geotechnical condition of the soils under the water dam and waste facilities must be investigated to confirm the location suitability and design adequacy. | A more robust dam design or having to move the Waste facilities could significantly impact both OPEX and CAPEX. | Field investigations are critical at the next level of study. |
| Smelter Terms | Smelter terms used in the study are only preliminary and could affect the project economics if the terms (payable %, deductions and/or penalties) change. | A reduction in the net smelter return would have a direct effect on project economics.<br>Low concentrate grade and/or the presence of deleterious elements in the concentrates could impact the desirability of the concentrate and the price smelters are willing to pay. | More work on identifying any potential deleterious elements and ways to increase Cu concentrate should be undertaken |
| Power Supply | It was assumed that sufficient power would be available locally forom the grid. | If power is not available, site generated power may have to be investigated or an upgrade to the regional grid capacity. These options would likely add costs to the project and potential delays. | Obtaining a clearer understanding of the regional power availability and early communication with Turkish power authorities should be undertaken. |
| Plant Process Copper and Gold Recovery | A reduction in gold recovery would have a negative impact on the project economics. | A copper process recovery of 86% and gold recovery of 61% was assumed in the economic analysis based on preliminary tests. A reduction in recovery by1% would lower the Case A AT $NPV_{7\%}$ by approximately $30 M. | Further testwork would confirm recoveries. |

## 23.2 Opportunities

**Table 23.2: Project Opportunities**

| Opportunity | Explanation | Potential Benefit |
|---|---|---|
| Exploration Potential | The expansion of the deposit may be possible with further resource drilling | The expansion of the deposit resources could potentially lead to a bigger pit and reduced strip ratio. |
| Copper/Gold Price | Metal price has the biggest single impact on the project economics. | An increase in metal value from the prices assumed in this study would have a significant economic benefit. In Case A, a 1% increase in Au price increases the PT $NPV_{7\%}$ of about $25 M. Using current spot metal prices of $1615/oz Au and 3.34/lb Cu the AT NPV7% is $960M with an IRR of 31%.<br>The open pit is not backfilled and this allows for expansion (push-back) of the pit should metal prices increase from Case A. |
| Resource Grade | An increase in resource grade would have a significant impact on the economic viability of the project. | A 1% increase in gold grade, while maintaining tonnage, increases the Case A PT $NPV_{7\%}$ by approximately $25 M. |
| Contract Mining | The use of contract mining may be a viable alternative to an owner-operator fleet as it would reduce the initial CAPEX. It would however, increase OPEX. | Approximately $190M in Capex could be saved if the mining was done by a contractor. |
| Power Cost | The cost of power is relatively high at $0.14/kWhr and a reduction in power costs would decrease OPEX. | Electrical power costs are 50% of the processing costs Therefore, a 10% reduction in power costs would lead to a 5% reduction in processing OPEX. |
| Project Strategy and Optimization | With additional data, the project should be subjected to a series of strategic option reviews, aimed at determining the most valuable strategy for exploiting the resource. | Planning and executing the project at the correct scale and with the optimum mine design and processing systems will result in the maximum possible value to shareholders and other economic stakeholders. |
| Metallugical Recoveries | The potential for optimized recoveries vs. concentrate grade could improve the project economics. | The NPV of the project may be improved with optimization of metallurgical recoveries and concentrate grade. |
| Staged Ramp-up | The project could include a staged ramp-up whereby a smaller facility could be built to reduce initial CAPEX and then expand production with the use of revenue. | Reducing initial CAPEX may offer financing benefits, although the overall NPV may be reduced with a staged start-up |

| Opportunity | Explanation | Potential Benefit |
|---|---|---|
| Tax and Investment Incentives | Tax and investment incentives may be available for the project that could reduce payable taxes and improve project economics. The simplified tax calculations done in this PEA report do not consider any tax breaks beyond standard depreciation. | Reducing taxation or providing other forms of investment incentive could improve the project NPV. |

# 24 Acronyms, Abbreviations and Units of Measure

| Distance | |
|---|---|
| µm | micron (micrometre) |
| mm | millimetre |
| cm | centimetre |
| m | metre |
| km | km |
| ” | inch |
| in | inch |
| ’ | foot |
| ft | foot |
| **Area** | |
| $m^2$ | square metre |
| $km^2$ | square km |
| ac | acre |
| Ha | hectare |
| **Volume** | |
| l | litre |
| $m^3$ | cubic metre |
| $ft^3$ | cubic foot |
| usg | US gallon |
| lcm | loose cubic metre |
| bcm | bank cubic metre |
| Mbcm | million bcm |
| **Mass** | |
| kg | kilogram |
| g | gram |
| t | metric tonne |
| Kt | kilotonne |
| lb | pound |
| Mt | megatonne |
| oz | troy ounce |
| wmt | wet metric tonne |
| dmt | dry metric tonne |
| **Pressure** | |
| psi | pounds per square inch |
| Pa | pascal |
| kPa | kilopascal |
| MPa | megapascal |
| **Elements and Compounds** | |
| Au | gold |
| Ag | silver |
| Cu | copper |
| Fe | iron |
| S | sulphur |
| CN | cyanide |
| NaCN | sodium cyanide |

| Other | |
|---|---|
| $^oC$ | degree Celsius |
| $^oF$ | degree Fahrenheit |
| Btu | British Thermal Unit |
| cfm | cubic feet per minute |
| elev | elevation above sea level |
| masl | m above sea level |
| hp | horsepower |
| hr | hour |
| kW | kilowatt |
| kWh | kilowatt hour |
| M | Million |
| mph | miles per hour |
| ppb | parts per billion |
| ppm | parts per million |
| s | second |
| s.g. | specific gravity |
| usgpm | US gallon per minute |
| V | volt |
| W | watt |
| Ω | ohm |
| A | ampere |
| tph | tonnes per hour |
| tpd | tonnes per day |
| mtpa | million tonnes per annum |
| Ø | diameter |
| **Acronyms** | |
| SRK | SRK Consulting (Canada) Inc. |
| CIM | Canadian Institute of Mining |
| NI 43-101 | National Instrument 43-101 |
| ABA | Acid- base accounting |
| AP | Acid potential |
| NP | Neutralization potential |
| NPTIC | Carbonate neutralization potential |
| ML/ARD | Metal leaching/ acid rock drainage |
| PAG | Potentially acid generating |
| non-PAG | Non-potentially acid generating |
| RC | reverse circulation |
| IP | induced polarization |
| COG | cut-off grade |
| NSR | net smelter return |
| NPV | net present value |
| LOM | life of mine |
| **Conversion Factors** | |
| 1 tonne | 2,204.62 lb |
| 1 oz | 31.1035 g |

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# 26 Date and Signature Page

This technical report was written by the following “Qualified Persons” and contributing authors. The effective date of this technical report is August 27, 2012.

| Qualified Person | Signature | Date |
|---|---|---|
| Gordon Doerksen, P.Eng | “original signed” | October 10, 2012 |
| James Gray, P.Geo. | “original signed” | October 10, 2012 |
| Garth Kirkham, P.Geo | “original signed” | October 10, 2012 |
| Dino Pilotto, P.Eng. | “original signed” | October 10, 2012 |
| Maritz Rykaart, P.Eng. | “original signed” | October 10, 2012 |
| Kevin Scott, P.Eng. | “original signed” | October 10, 2012 |

Reviewed by

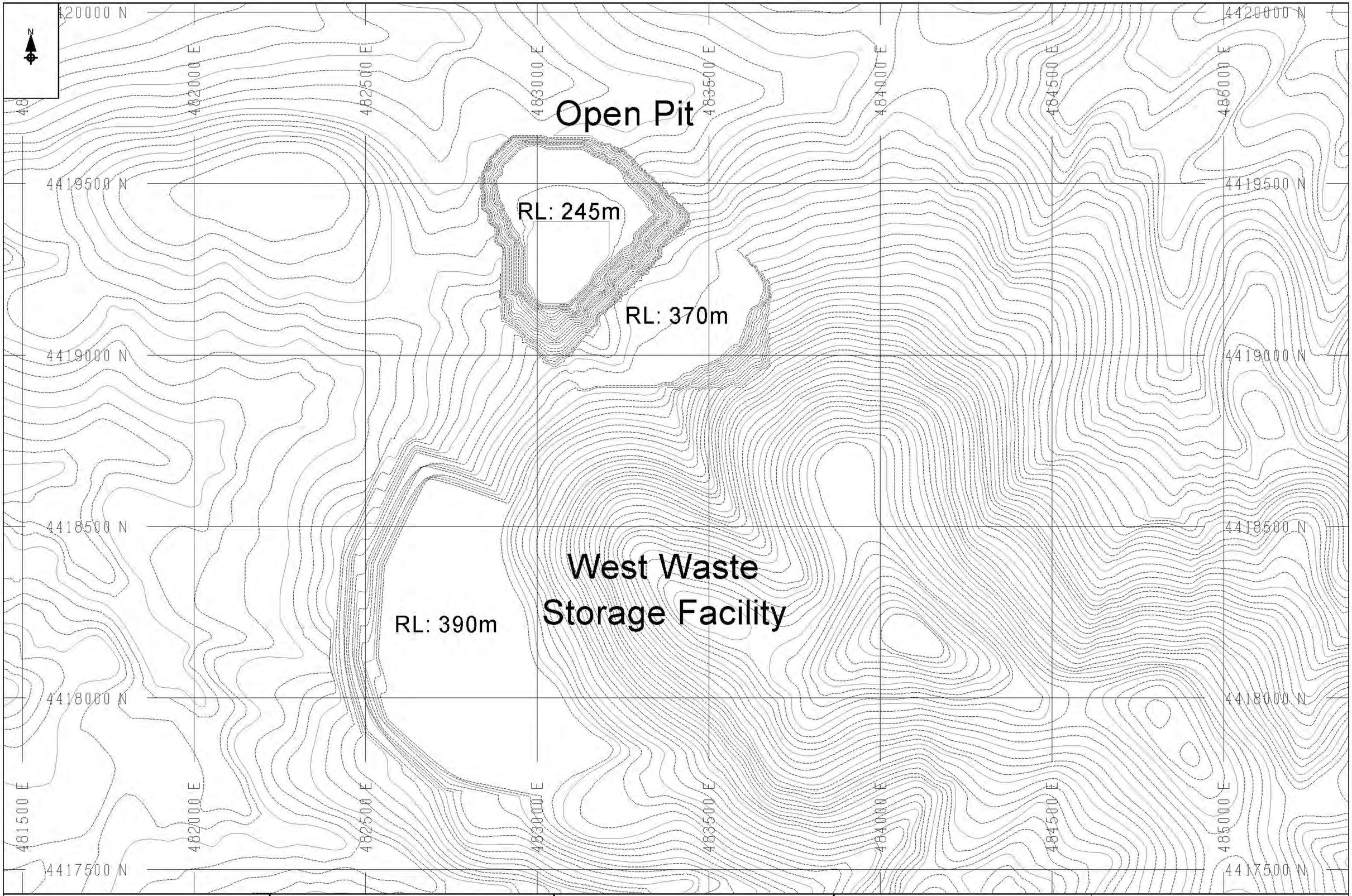
Original Signed

Neil Winkelmann

Report Reviewer

All data used as source material plus the text, Tables, Figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices

# Appendix A
# End of Period Maps

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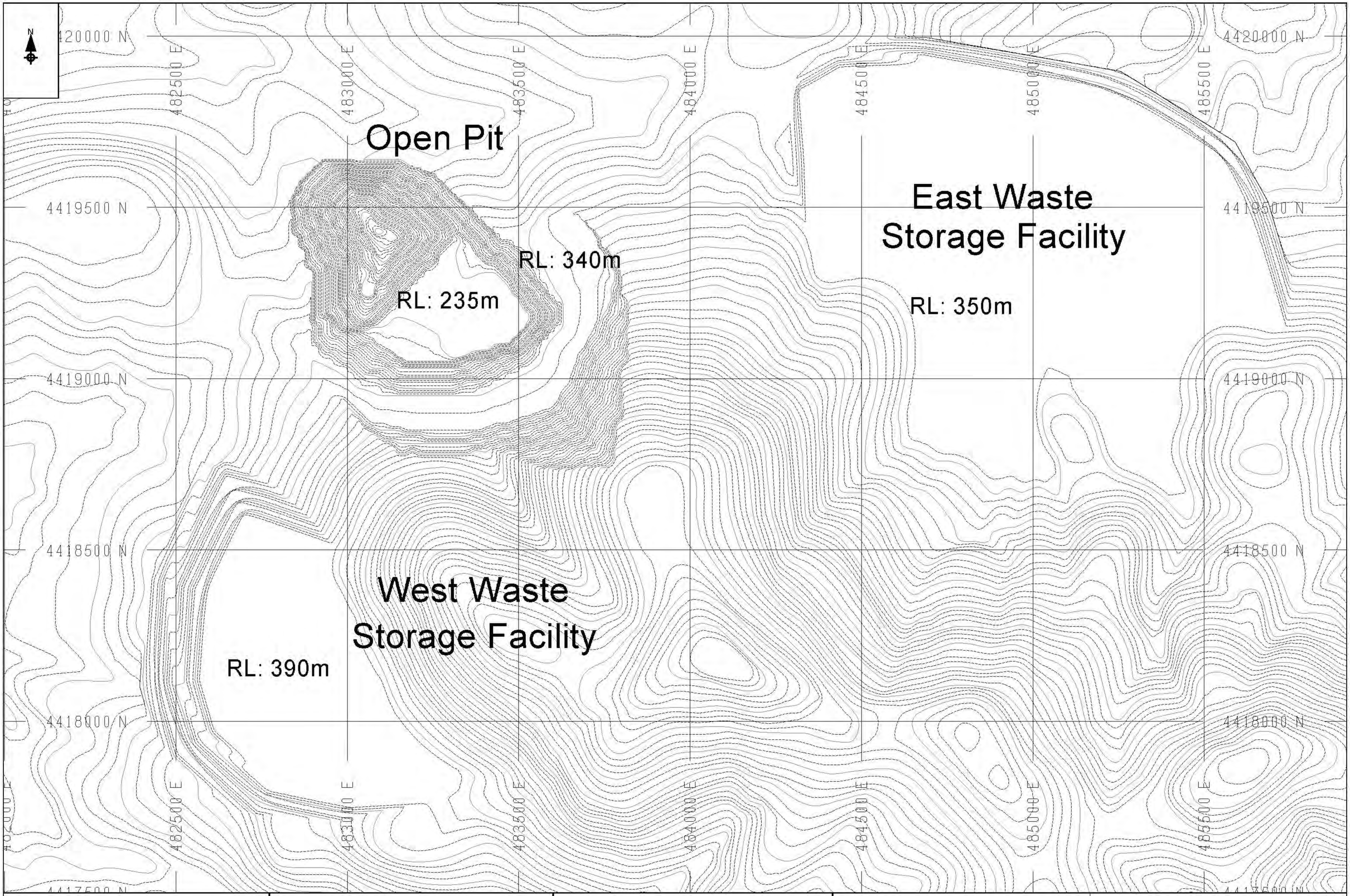
| DESCRIPTION: | End of Year 2 |
| DATE: | 2012/05/29 |

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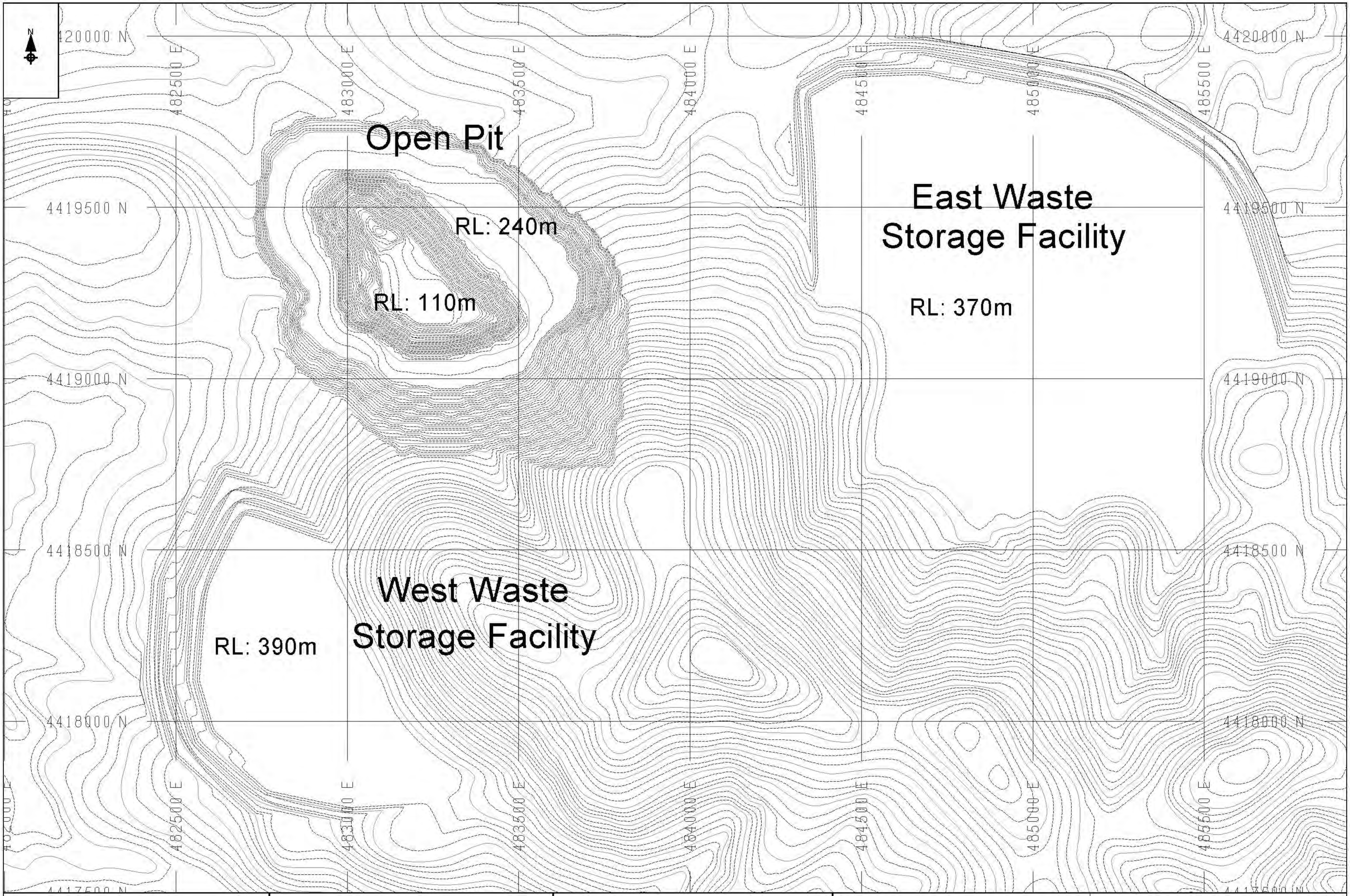


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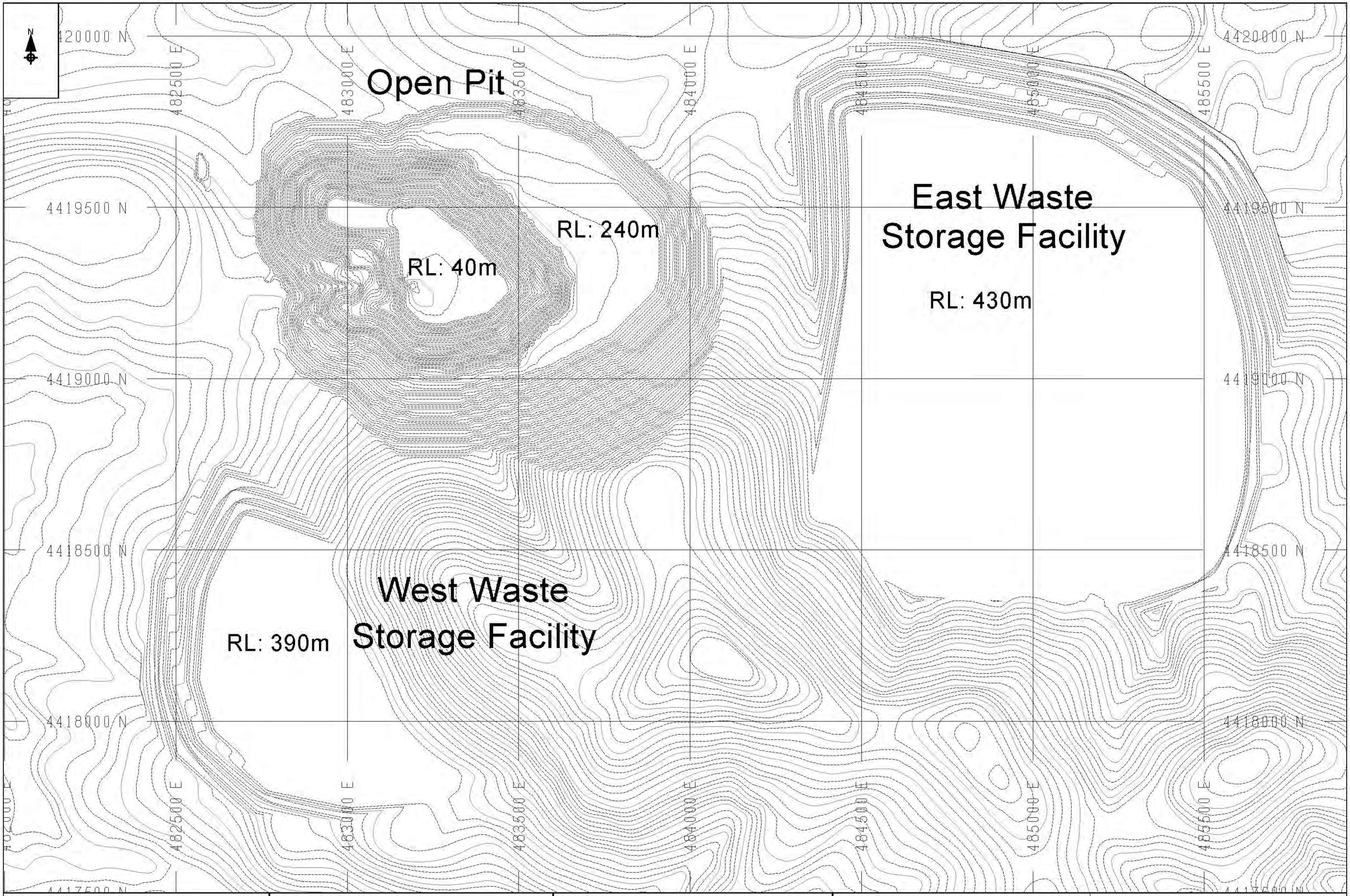


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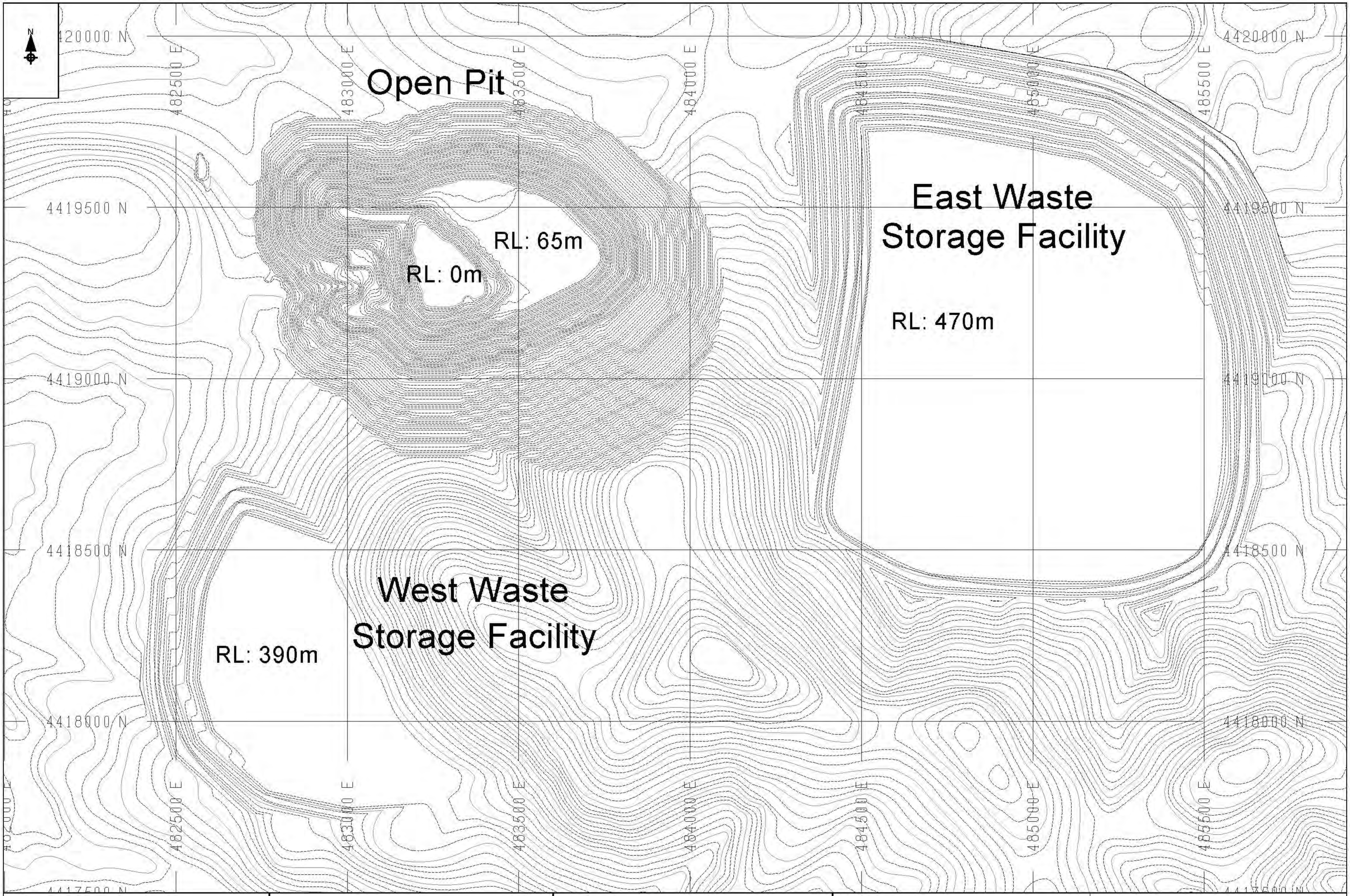


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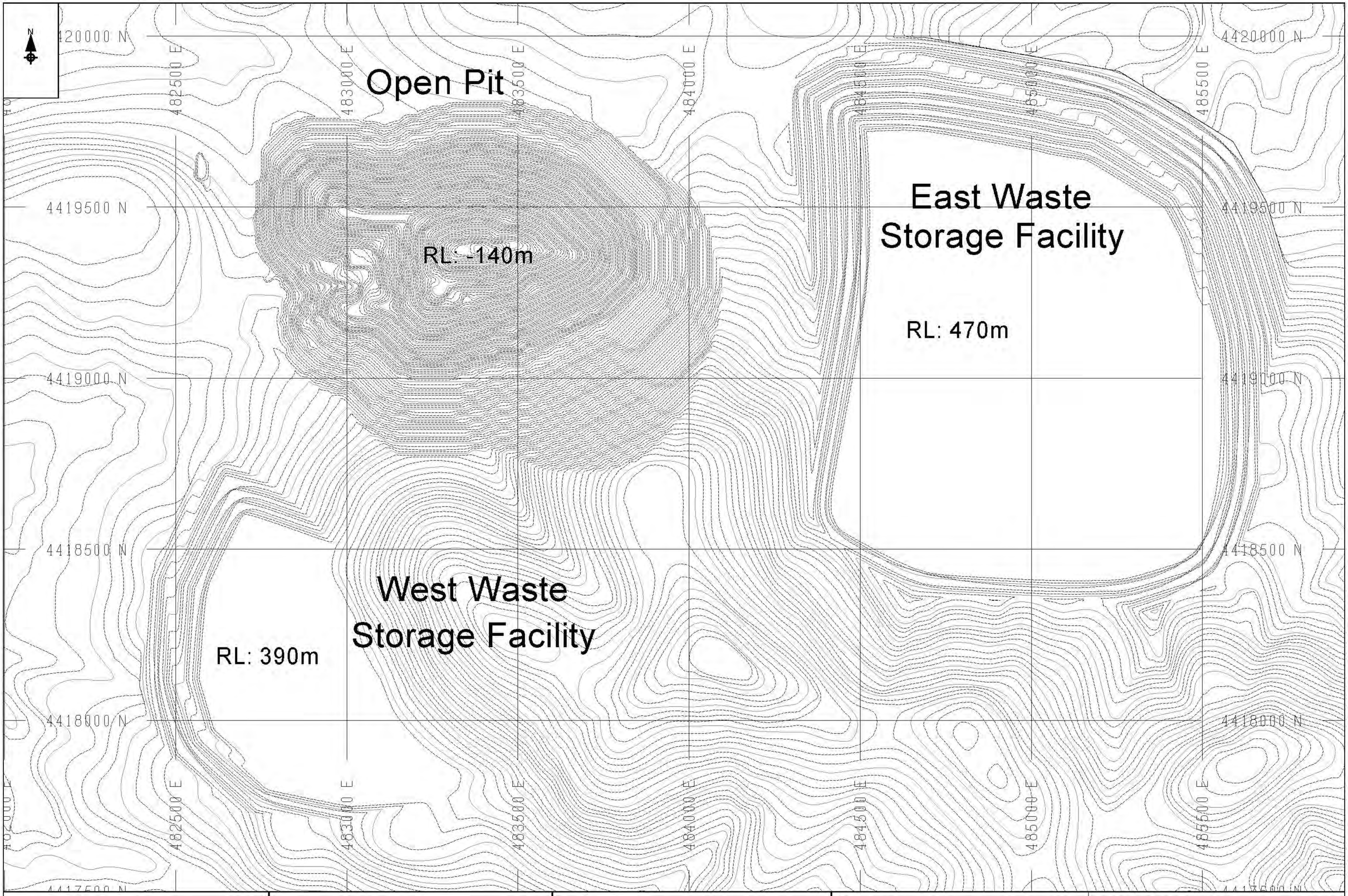


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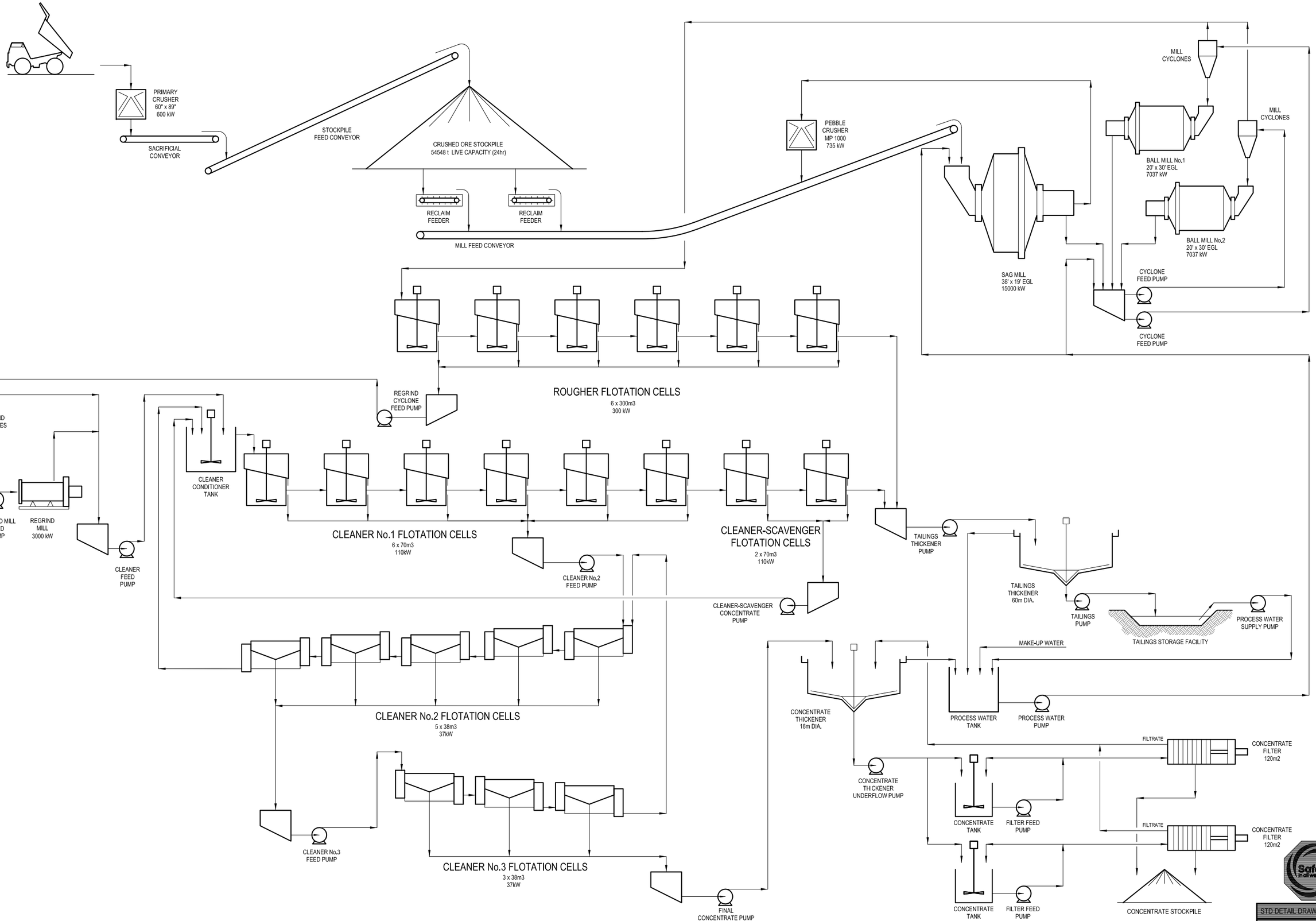
# Appendix B
# Process Flow Sheet Designs

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Vancouver BC
Canada V6B 2W2
T +1 778 374 6998
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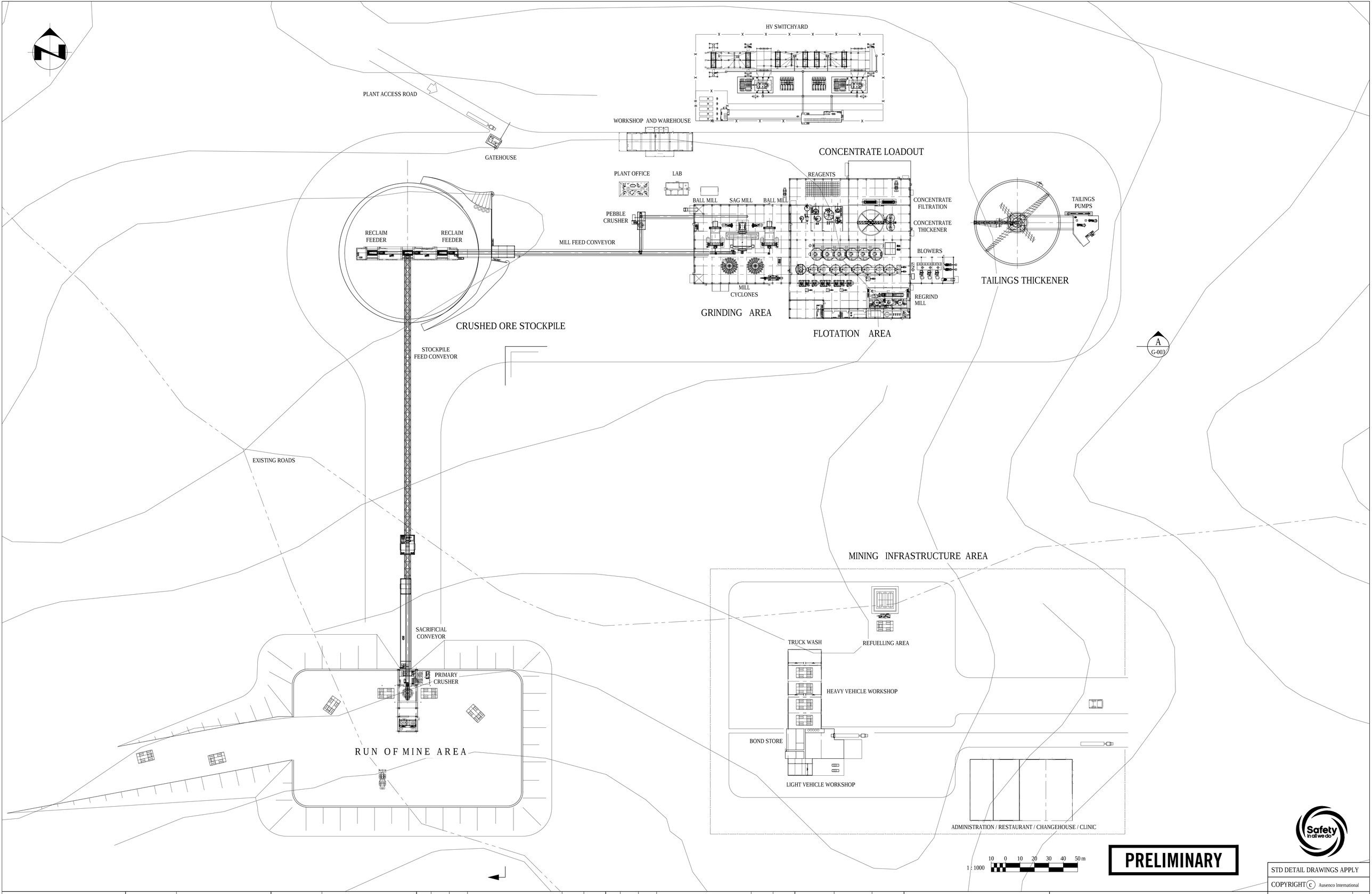
CLIENT: PILOT GOLD INC

TITLE: HALILAGA PROJECT
OVERALL PROCESS PLANT
PROCESS FLOW DIAGRAM

SCALE N T S
ALL DIMENSIONS IN MILLIMETRES
SIZE B1
REV No A B
DRAWING No 2249-F-001

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| No | BY | DATE | REVISION DETAILS | CHKD | APPR |
|---|---|---|---|---|---|
| B | PGD | 25-05-12 | ISSUED FOR SCOPING STUDY | | |
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CLIENT: PILOT GOLD INC

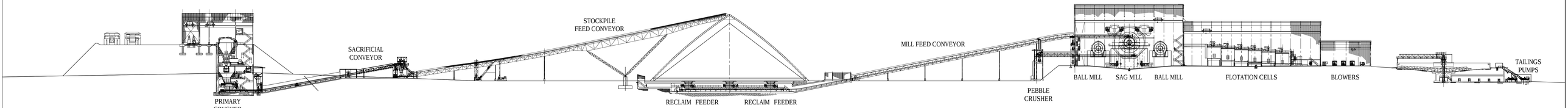
TITLE: HALILAGA PROJECT
OVERALL PROCESS PLANT PLAN

SCALE 1 : 1000 | SIZE B1

ALL DIMENSIONS IN MILLIMETRES

REV No A B

DRAWING No 2249-G-002




RUN OF MINE AREA | CRUSHED ORE STOCKPILE | GRINDING AREA | FLOTATION AREA | TAILINGS THICKENER

ELEVATION A
1 : 1000
G-002

1 : 1000 10 0 10 20 30 40 50 m

**PRELIMINARY**

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| No | BY | DATE | REVISION DETAILS | CHKD | APPR |
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CLIENT
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TITLE
HALILAGA PROJECT
OVERALL PROCESS PLANT
GENERAL ARRANGEMENT - ELEVATION

SCALE 1 : 1000
SIZE B1
ALL DIMENSIONS IN MILLIMETRES

REV No A B

DRAWING No
2249-G-003