# Audit 2014
# Volume 4 Kaymaz
# Resources and Reserves
# Koza Altın İşletmeleri A.Ş.
# Turkey

**Report Prepared for**

## Koza Altın İşletmeleri A.Ş.

**Report Prepared by**

SRK Consulting (U.S.), Inc.
SRK Project Number 173600.130
January 31, 2015

# Audit 2014
# Volume 4 Kaymaz
# Resources and Reserves
# Koza Altın İşletmeleri A.Ş.
# Turkey

## Koza Altın İşletmeleri A.Ş.

Istanbul
Yolu 10. Km. No: 310
Batikent – ANKARA

**SRK Consulting (U.S.), Inc.**
7175 West Jefferson Avenue, Suite 3000
Lakewood, CO  80235

e-mail: denver@srk.com
website: www.srk.com
Tel: +1.303.985.1333
Fax: +1.303.985.9947

**SRK Project Number 173600.130**

**January 31, 2015**

**Contributors:**
Gözde Kaya Akbaş, M.Sc., METU, Consultant GIS Specialist
Nilüfer Gümüş, B.Sc., Consultant Environmental Engineer
Ahmet Oğuz Öztürk, B.SC., Principal Mining Engineer

**Endorsed by CPs:**
Leah Mach, MSc Geology, CPG, Principal Consultant (Resource Geology)
Bret Swanson, BEng Mining, MAusIMM, MMSAQP, Principal Consultant (Mining Engineer)
Eric J. Olin, MSc, MBA, SME-RM, Principal Consultant (Metallurgy
Dorinda Bair, BSc Geology, CPG, Principal Consultant (Geologist)

**Reviewed by:**
Bart Stryhas, PhD, CPG, Principal Consultant (Resource Geologist)
Peter Clarke, BSc Mining, MBA, PEng, Principal Consultant (Mining Engineer)

# Table of Contents

# List of Tables

# List of Figures

# Disclaimer & Copyright

# List of Abbreviations

The metric system has been used throughout this report unless otherwise stated. All currency is in U.S. dollars unless stated otherwise. Market prices are reported in US$ per troy oz of gold and silver. Tonnes are metric of 1,000 kg, or 2,204.6 lb, unless otherwise stated. The following abbreviations are typical to the mining industry and may be used in this report.

| Abbreviation | Unit or Term |
|---|---|
| ° | degree |
| % | percent |
| AA | atomic absorption |
| AAS | atomic absorption spectroscopy |
| Ag | silver |
| Amsl | above mean sea level |
| Au | gold |
| BLEG | Bulk Leach Extractible Gold |
| BWI | Bond Work Index |
| C | Celsius |
| CoG | cutoff grade |
| CIP | carbon in pulp |
| cm | centimeter |
| CP | Competent Person |
| CPR | Competent Person's Report |
| CRP | Community Relations Plan |
| CRM | Certified Reference Material |
| Cu | copper |
| dia. | diameter |
| Eq | equivalent |
| EIA | Environmental Impact Assessment |
| F | Fahrenheit |
| ft | feet/foot |
| g | gram |
| g/cm | grams per centimeter |
| g/t | grams per tonne |
| ha | hectares |
| HG | high-grade |
| hr | hour |
| ID2 | Inverse Distance Squared |
| ID3 | Inverse Distance Cubed |
| in | inch |
| IP | Induced Polarization |
| kg | kilogram |
| km | kilometer |
| koz | thousand troy ounce |
| kt | thousand tonnes |
| kV | kilovolt |
| kVA | kilovolt-amps |
| L | liter |
| lb | pound |
| LHD | load haul dump |
| LG | low-grade |
| LoM | life of mine |
| m | meter |
| M | million |
| m.a. | million annum |
| min | minute |
| mm | millimeter |
| Mm | million meter |
| Moz | million ounces |

| Abbreviation | Unit or Term |
|---|---|
| Mt | million tonnes |
| Mt/y | million tonnes per year |
| MTA | Mining, Research and Exploration Institute of Turkey |
| MVA | million volts amperes |
| NN | Nearest Neighbor |
| NPV | net present value |
| OK | Ordinary Kriging |
| OP | open pit |
| oz | ounce |
| ppb | parts per billion |
| ppm | parts per million |
| QA/QC | Quality Assurance/Quality Control |
| RC | reverse circulation |
| RoM | run of mine |
| SART | sulfidization, acidification, recycling, and thickening |
| t | tonne(s) |
| t/h | tonnes per hour |
| t/d | tonnes per day |
| t/m | tonnes per month |
| t/y | tonnes per year |
| TEM | Technical Economic Model |
| μ | micron |
| UG | underground |
| V | volt |
| WAD | weak acid dissociable |
| Zn | zinc |

# 1 Introduction

SRK Consulting (U.S.), Inc. (SRK) was commissioned by Koza Altın İşletmeleri A.Ş. (Koza) to audit Koza's gold resources and reserves and exploration projects as of the end of December, 2013. Koza's Mining Assets are located in the Ovacık Mining District, Mastra Mining District, and Kaymaz District, including Söğüt, as well as Mollakara in the Diyadin District in Eastern Turkey and Himmetdede in Central Turkey.

This report is Volume 4 Kaymaz Resources and Reserves of the following ten volumes reports:

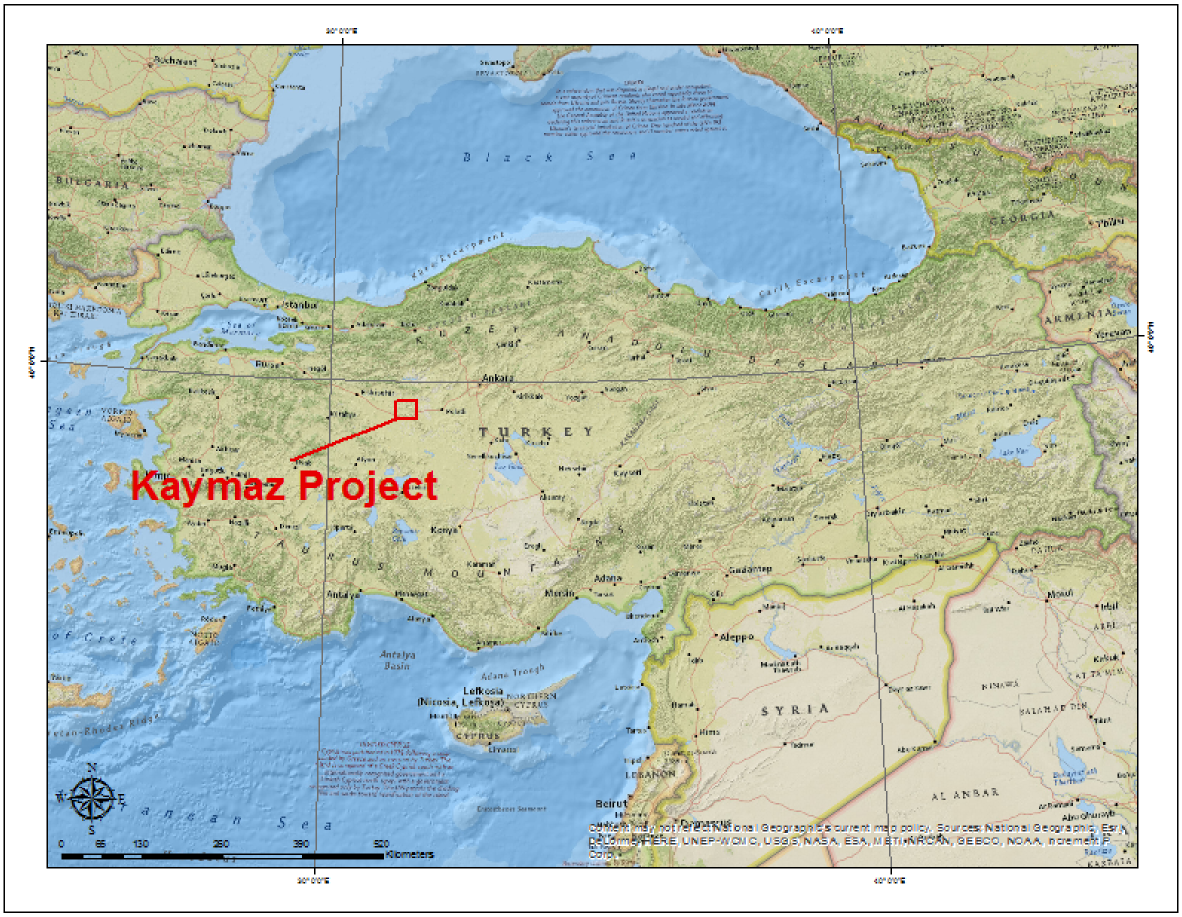
- Volume 1 Executive Summary;
- Volume 2 Ovacık Resources and Reserves;
- Volume 3 Mastra Resources and Reserves;
- **Volume 4 Kaymaz Resources and Reserves;**
- Volume 5 Söğüt Resources and Reserves
- Volume 6 Himmetdede Resources and Reserves;
- Volume 7 Mollakara Resources and Reserves;
- Volume 8 Technical Economics;
- Volume 9 Hasandağ and Işıkdere Resource Areas; and
- Volume 10 Exploration Projects.

This report is prepared in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code 2012).

Volume I Executive Summary contains the Terms of Reference and Property Descriptions relevant to all volumes of this audit.

## 1.1 Kaymaz District

The Kaymaz District includes Damdamca, Main Zone, Mermerlik and Kizilagil. The climate, physiology and regional geology of these mines and projects are discussed in this section of Volume 4. The Location of the Kaymaz District is shown in Figure 1.1.1.




Source: Modified from ESRI Basemap NatGeo_World_Map, 2013

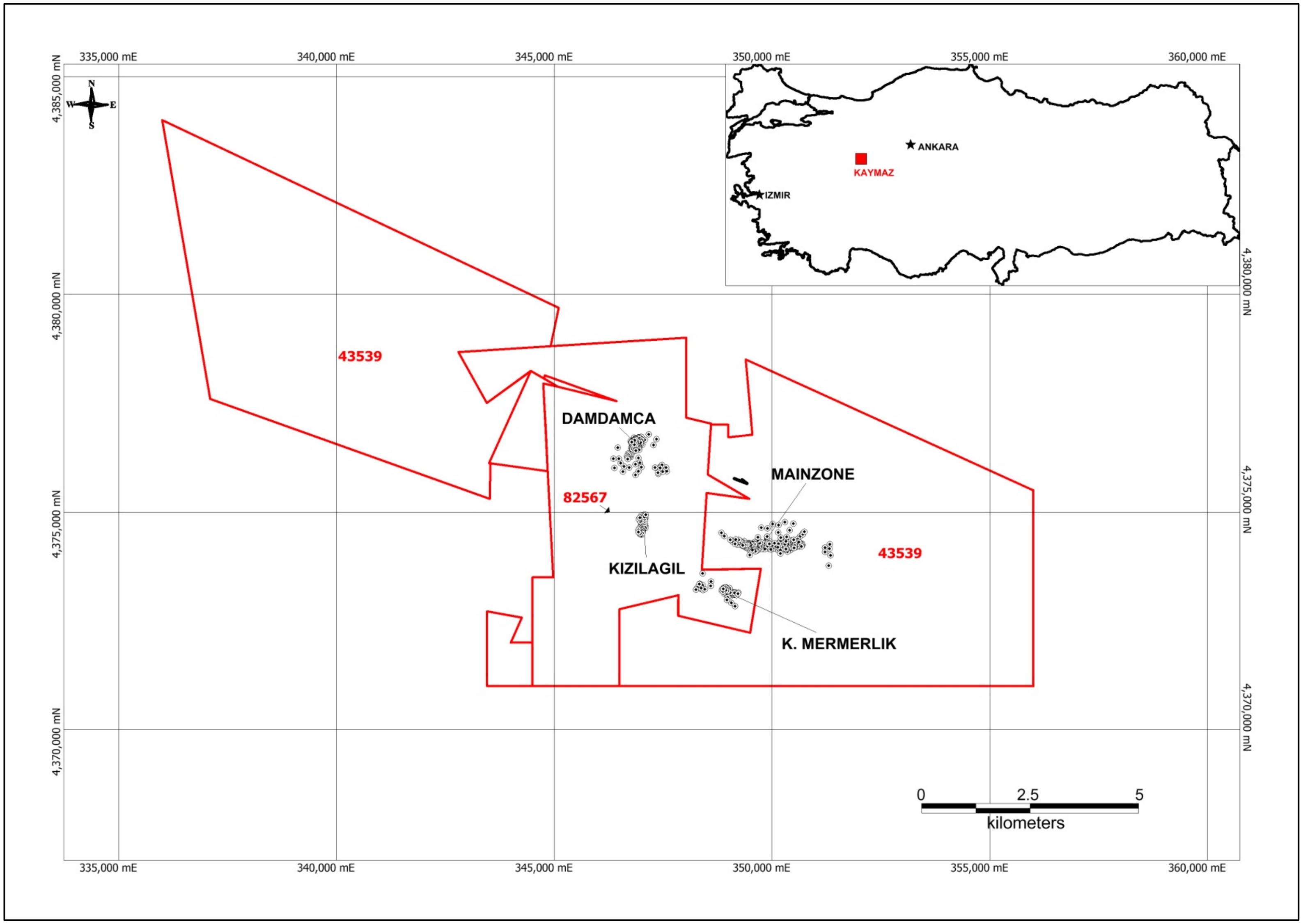
**Figure 1.1.1: Location Map Showing Kaymaz**

# 2 Kaymaz Mine Resources and Reserves

## 2.1 Property Description and Location

The Kaymaz project is located in northwestern Turkey, 65 km southeast of Eskişehir and 150 km west of Ankara. The nearest towns and villages to the project area are Sivrihisar located 20 km east, Karakaya located 3 km east and Kaymaz located 3 km west. The project is located 2 km north of the Ankara–Eskişehir Highway. Kaymaz is located at approximate UTM coordinates 4377000 N, 346000 E to 4373000 N, 35000 E in ED1950 Zone 36.

Koza has two operation licenses at Kaymaz. These include operation license numbers 82567 and 43539 (IR.5262) which total approximately 11,904 ha. Operation license number 82567 has one operation permit for gold and silver totaling 1,070.47 ha. Operation license number 43539 has two operation permits, one for gold and one for silver covering 479.01 ha. Land tenure is shown in Figure 2.1.1. Licenses are identified by their IR number.




Source: Koza, 2015

**Figure 2.1.1: Kaymaz Location and Land Tenure**

## 2.2 Climate and Physiography

The Kaymaz Project, located in Central Anatolia, experiences a continental climate with cold, harsh winters and dry summers with moderate to hot temperatures. Average temperatures range from 0°C in January to 22°C in July and August. The maximum temperatures may reach 40°C in the summer. Local rainfall data indicates average annual precipitation is 350 to 400 mm, which falls as rain during the summer months and snow during the winter months.

The project is near the eastern end of the Sündiken mountain range that is near the center of a broad regional valley between the Sakarya and Porsuk Rivers. Kaymaz is at approximately 1,100 m amsl in an area of low relief with gentle hills.

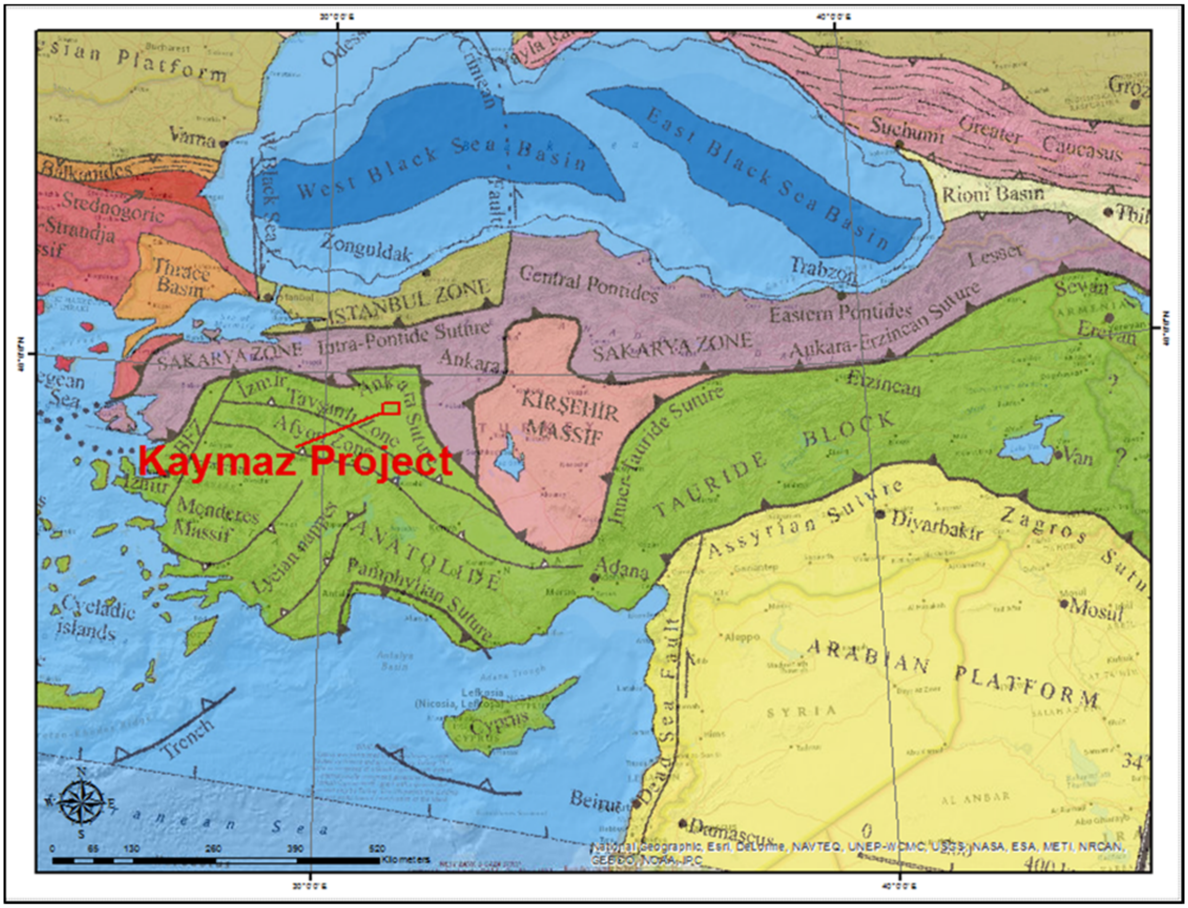
## 2.3 History

The Kaymaz property was held by Tüprag Metal Madencilik (Tüprag) from 1988 until 2005. Although Normandy Madencilik, A.Ş. (Normandy) never held the property, from 1993 to 2006 it performed regional exploration in the area and collected samples on land that is now held by Koza. Normandy also collected ten bulk samples from the area. Work completed by Tüprag included mapping at regional and detailed scales, geophysical surveys, over 80 Reverse Circulation (RC) and core holes, and rock chip sampling. In 1994, Gencor, Ltd., the parent company of Tüprag, performed a project feasibility study at Kaymaz. Koza acquired the property in 2005.

## 2.4 Geology

### 2.4.1 Regional Geology of the Kaymaz District

The Kaymaz project is located 65 km southeast of Eskişehir in northwestern Turkey on the Anatolian Plateau and is situated in the northern margin of the Anatolide-Tauride Block along the İzmir-Ankara Suture. This area is in the Western Anatolian Extensional Tectonic Province extending from north central Turkey to the Aegean Sea and is noted to contain low and high sulfidation epithermal deposits and porphyry copper deposits. Kaymaz is specifically located near the north central area of this province. Deposits within this zone are linked to Paleogene and Neogene period volcanism and Upper Mesozoic to Tertiary age intrusive events (Okay, 2008). Figure 2.4.1.1 shows the location of the Kaymaz in the Anatolide-Tauride Terrane.

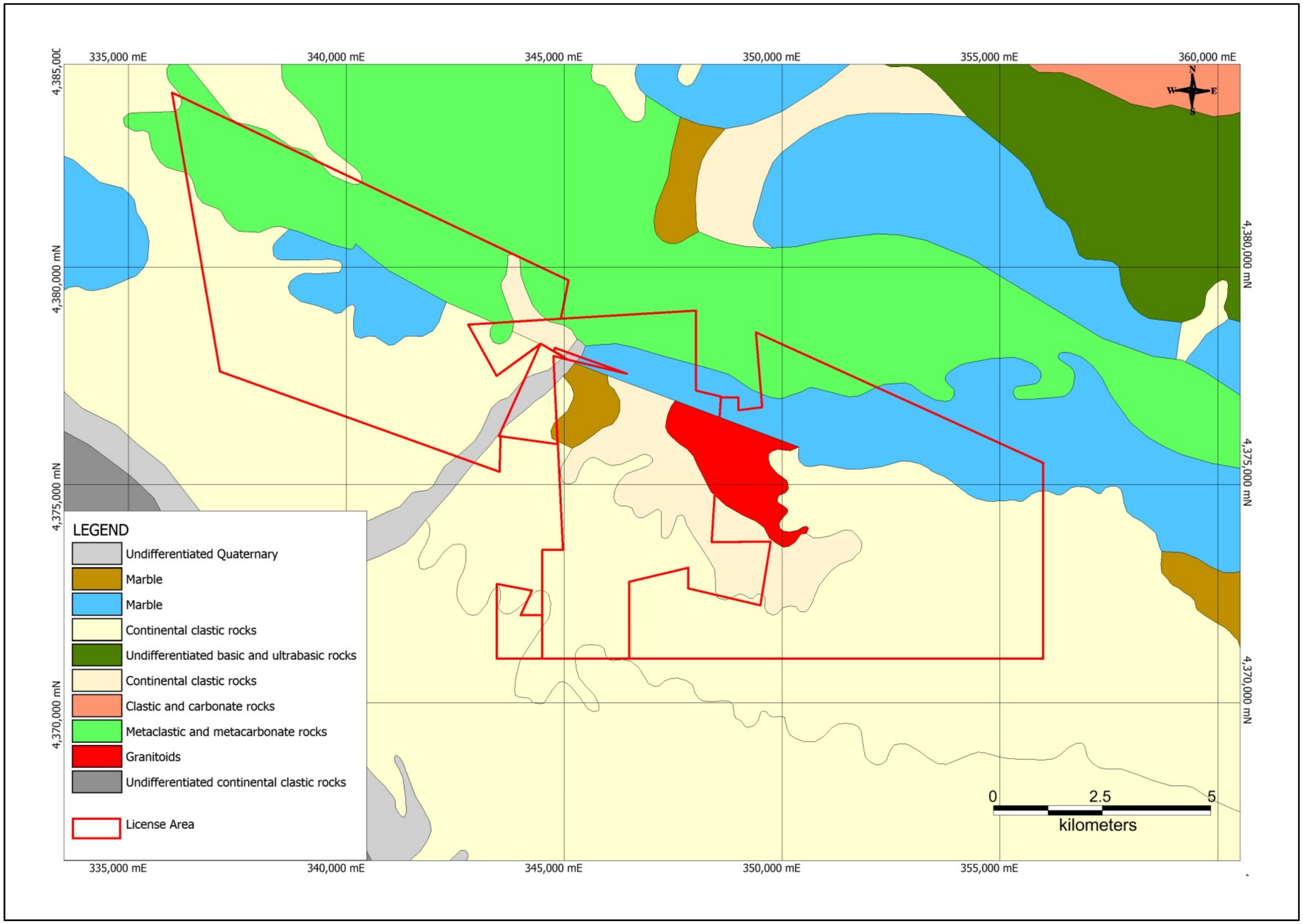



Source: Modified from Okay et al, 2010; Basemap = ESRI Basemap NatGeo_World_Map, 2013

**Figure 2.4.1.1: Location of the Kaymaz Project Relative to the Anatolide-Tauride Terrane**

In this area, seafloor ophiolites and sediments, some of which underwent blue schist metamorphism, were obducted onto the Anatolide-Tauride platform during subduction of the Sakarya Terrane in the Triassic period (Okay et al., 2004; Okay, 2008). During this tectonic period, large sections of ultra-mafic rocks within the ophiolite sequence were metamorphosed to serpentinite. These serpentinites are part of the Karakaya Complex. Subsequent uplift and erosion has left a regional pattern of west-northwest trending troughs, with exposed serpentinites, separated by uplifts of older basement known locally as the Permian to Triassic age Sivrihisar and Dumrek Formations. A number of small chromite-magnetite and magnesite deposits are associated with the serpentinite bodies (Kara, 2007).

The project is located south of a Paleocene volcanic center, in basement rocks composed of limestones and marbles intruded by the Karakaya Granite. Intrusive activity in the region ranges in age from early Mesozoic to early Tertiary, and the Karakaya granite was intruded during the Cretaceous age after the formation of the İzmir-Ankara Suture. This suture cross-cuts both the basement (Sivrihisar Formation) and the overthrust Karakaya Complex ophiolites (Kara, 2007). Figure 2.4.1.2 shows the regional geology of the Kaymaz District.



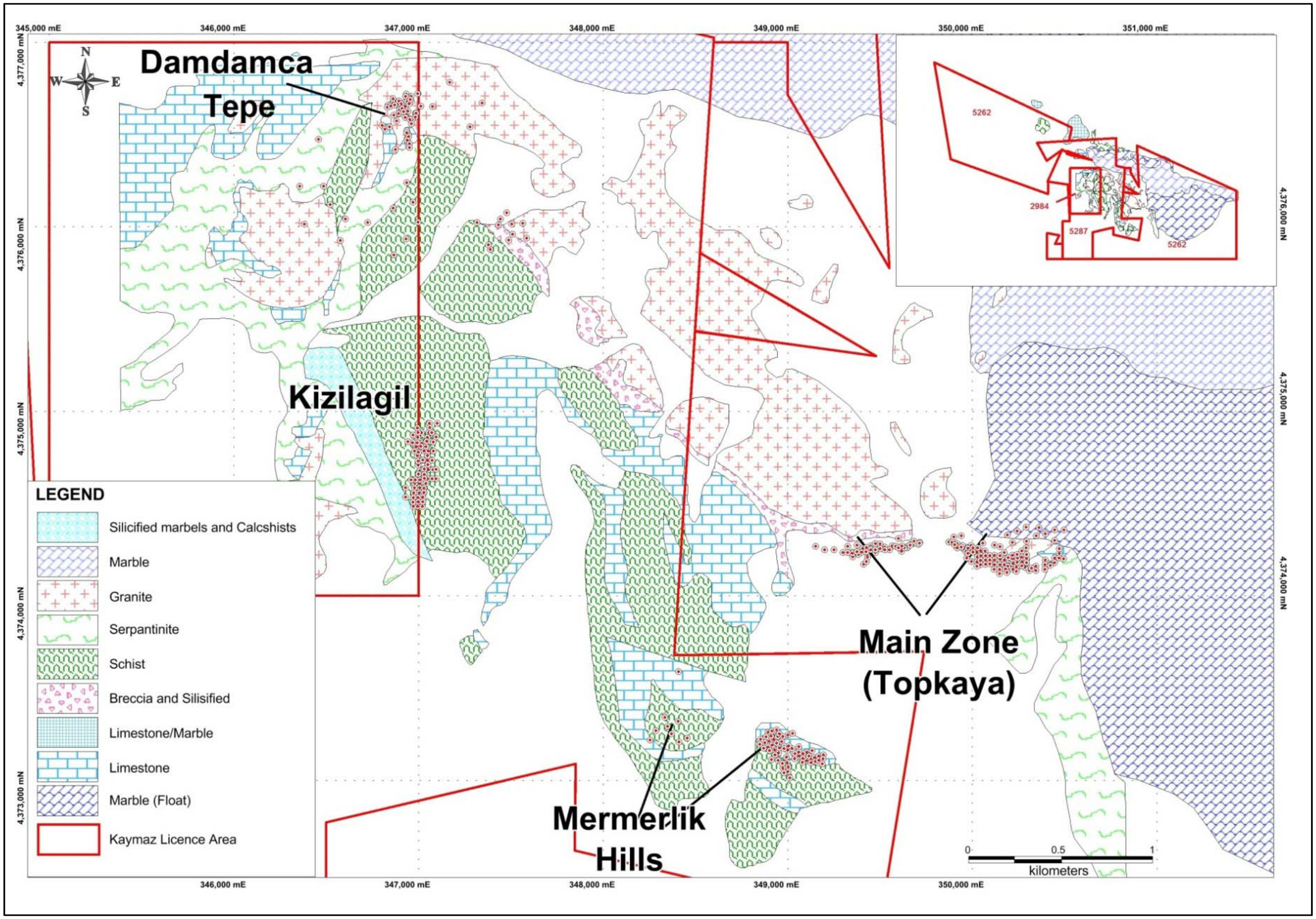

Source: Koza, 2015

**Figure 2.4.1.2: Regional Geology of Kaymaz**

### 2.4.2 Local Geology of the Kaymaz Project

At Kaymaz, the basement rock is composed of the Sivrihisar Formation, which locally consists of marine sediments and some interbedded mafic volcanics, which have been metamorphosed to marbles, mafic schists and metasandstones. These are overlain by serpentinite of the Karakaya Complex. Both of these units have been intruded by the Karakaya Granite. Mineralization at Kaymaz is concentrated in highly silicified serpentinite of the Karakaya Complex and spatially related to the contact with the Karakaya Granite. The granite is not mineralized. The structural fabric and outcrop patterns in the project area are dominated by this granite and its satellite intrusions. South of the intrusion, the Sivrihisar Formation is exposed in the center of a broad north-south trending anticline. Serpentinites located above the Sivrihisar Formation outcrop on either side of the anticline. The metasediments have fold axes oriented parallel to the southern contact of the Karakaya Granite. These folds developed as a result of compressional buckling during the emplacement of the granite.

Drilling has shown that the dip direction of the granite contact changes direction from south to north with depth. This change in dip is in part fault controlled. The dominant fault directions in the Kaymaz area are striking east-west to N30°W and are interpreted to be dextral strike-slip faults. In general these faults dip steeply to the north, and are offset by a north-south striking fault set. Both the dextral faults and the north-south striking faults were controls for mineralizing fluids. The north-south fault set exhibits post mineralization movement of a few meters to several 10's of meters (Kara, 2007). Geology is shown in Figure 2.4.2.1.




Source: Koza, 2012

**Figure 2.4.2.1: Geology of the Kaymaz Project**

The Kaymaz project consists of four areas. These are, from north to south, Damdamcatepe, Kizilagil, the Main Zone also referred to as Topkaya, and the Mermerlik Hills (Figure 2.4.2.1). Near Damdamcatepe and the Main Zone resource areas, the granite forms a northeasterly dipping sill, which outcrops in the southeastern portion of the Kaymaz tenement. At Damdamcatepe the footwall of the mineralization is controlled by a granite dike. It is interpreted that fluids moved along this dike and laterally beneath the granite sill at Damdamcatepe. Post-mineralization strike-slip faulting along the dikes footwall contact has also modified the mineralized zone. Mineralization at Damdamcatepe is traced over 220 m along a north-south strike and dips 45 to 60° E. Mineralization in the Main Zone is similar to that at Damdamcatepe. At the Main Zone, the mineralization strikes east-west over a length of 1,100 m and dips 45º to 60º S (Gencor, 1994; Kara, 2007; Chapman, 2007a; Koza, 2011).

Kizilagil mineralization is oriented approximately north-south and dips steeply to the west. The mineralization has been traced up to 350 m along strike and can be up to 25 to 30 m thick. Mineralization at Kizilagil is characterized as silica replacement in carbonate and metamorphic rocks. Small-scale strike-slip faults striking northeast-southwest have offset mineralization in places (Gencor, 1994; Kara, 2007; Chapman, 2007a; Koza, 2011).

Unlike the other three areas, mineralization at the Mermerlik Hills shows no obvious proximal relationship with the granite intrusive and is near the base of the serpentinite thrust sheet. Exploration has been focused on the southeast Kucuk Mermerlik Hill where mineralization is characterized as silica replacement along reverse faults hosted in serpentine. Mineralization is found in a northwest-southeast elongate zone that has a shallow dip to the northeast and has been offset by northeast striking faults with 15 to 20 m of displacement. The mineralized zone is approximately 250 m along strike and can be up to 20 m thick (Gencor, 1994; Kara, 2007; Chapman, 2007a; Koza, 2011).

Kaymaz includes a number of different mineralization styles. These are manto-type skarn mineralization, quartz stockworks, quartz veinlets and multi-phase brecciation adjacent to the granite dike. The quartz within stockworks and veinlets include chalcedony, massive quartz and drusy quartz. Quartz-tourmaline veinlets have also been found. Quartz is dark gray to black in color. Sulfide minerals include disseminated pyrite, galena and chalcopyrite. Alteration associated with mineralization, is silicification at the hanging wall contact grading outward to a carbonate zone and then into unmineralized serpentinite. Koza is using a low sulfidation epithermal model for the Kaymaz area which is consistent with observed mineralization.

## 2.5 Exploration

Koza acquired the property in 2005 and to date has completed an extensive soil sampling project of 1,259 samples in four different target areas. Koza also collected over 140 rock chip samples for analytical confirmation of previous samples and six bulk samples of their own to further support previous work. Koza has mapped the project at a 1:10,000 scale, drilled 226 core holes and completed additional geophysical surveys of the area.

Koza still has exploration potential at Kaymaz-Mermerlik of approximately 20% to 30% of the mineral resource. Koza expects to verify this potential in the next 6 to 12 months beginning in July 2015. Verification will be through drilling. Exploration budget at this project is handled by the mine. Koza has a preliminary budget of approximately TL30,000 (US$13,000) to cover basic licensing. Drilling will require an additional budget.

### 2.5.1 Exploration Sample Collection

Soil sample grids were designed to cover mineralized areas of interest and were collected where soil was available on regular grids. The soil grids were oriented in the cardinal directions. Over the Main Zone, Koza used a sampling grid of 100 m between lines and 50 m between samples along the line. In other areas Koza used wider spaced grids up to 200 m x 200 m. Samples were collected from the B horizon and typically 3 to 4 kg of sample was collected.

Rock samples were chip type collected at locations across the width of the exposed mineralization and were typically 3 to 4 kg in weight. Collection points ranged from 25 to 200 m apart along the veins strike and were selected based on field observations, conditions and accessibility to the vein.

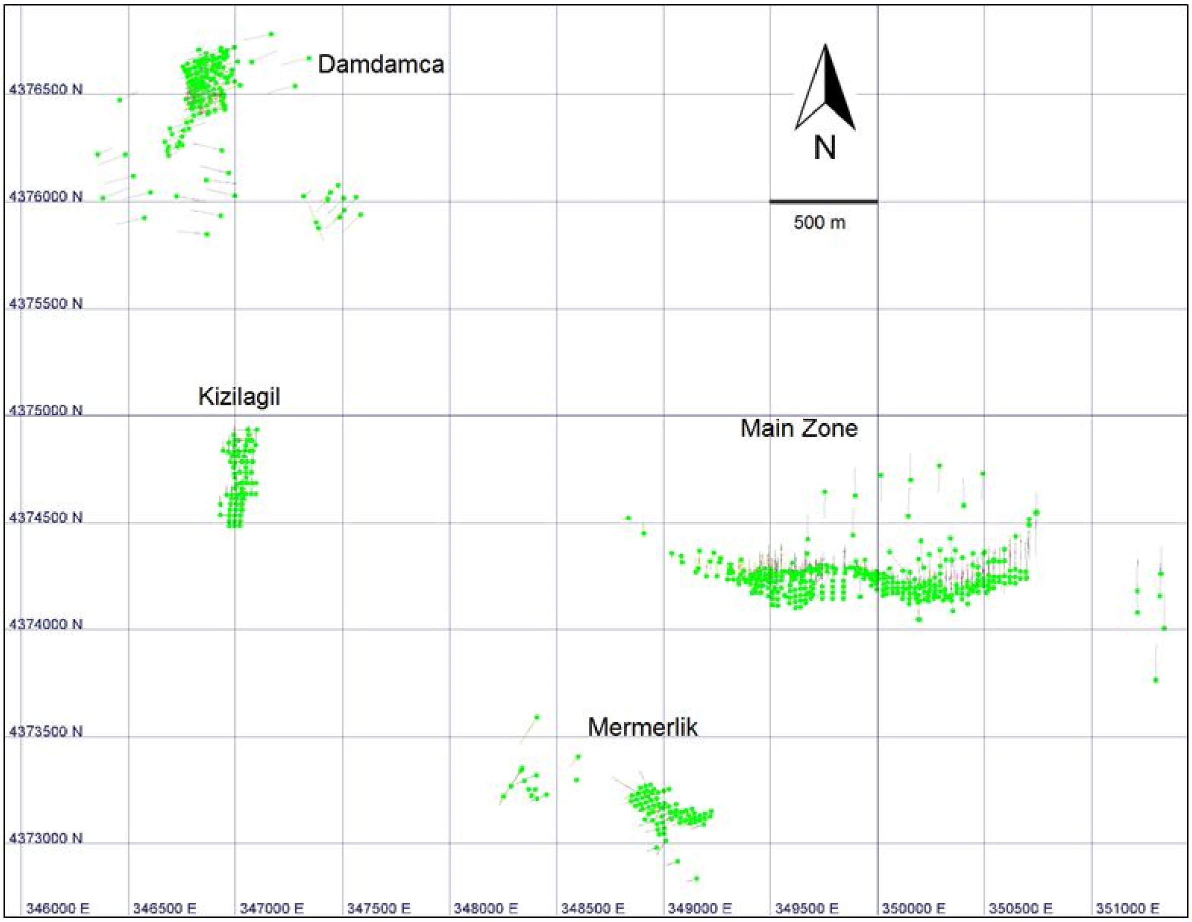
Drill core sampling is discussed in Section 2.5.2.

### 2.5.2 Drilling/Sampling Procedures

The Kaymaz resource is located in three separate areas: Kizilagil, Main and Mermerlik. The Damdamca resource was mined out in 2014. A summary of the drilling at Kaymaz is shown in Table 2.5.2.1. Tüprag conducted drilling and trenching in the 1990’s. Koza has conducted all the trenching and drilling since then. A drillhole location map is shown in Figure 2.5.2.1.

**Table 2.5.2.1: Summary of Drilling at Kaymaz**

| Area | Company | Core | | RC | | Trenches | |
|---|---|---|---|---|---|---|---|
| | | Number | Meters | Number | Meters | Number | Meters |
| Damdamca | Tüprag | | | 96 | 7,303 | 18 | 2,800 |
| | Koza | 63 | 11,487 | | | 15 | 1,994 |
| Main | Tüprag | | | 75 | 3,920 | | |
| | Koza | 304 | 54,005 | | | 22 | 1,445 |
| Mermerlik | Tüprag | | | 6 | 305 | | |
| | Koza | 81 | 8,364 | | | 4 | 410 |
| Kizilagil | Koza | 76 | 4,027 | | | 7 | 722 |
| **Total** | | **524** | **77,883** | **177** | **11,527** | **66** | **7,371** |




Source: SRK, 2014

**Figure 2.5.2.1: Kaymaz Drillhole Location Map**

The drilling at Main Zone, Mermerlik and Kizilagil is on 25 by 25 m grids. At the Main Zone, the drillholes are angled to the north; at Mermerlik, the drillholes are angled to the west-southwest; and at Kizilagil, the drillholes are angled to the west or east. The majority of drillholes have inclinations of about 50°.

All the drilling at Kaymaz prior to 2008 was completed by Tüprag. According to Datamine International (2008), all RC holes were sampled on 1.5 m intervals. The drill cuttings were collected at the drill and the samples were prepared at Tüprag's laboratory in Çanakkale. The analysis was completed by the Robertson Group Plc in the United Kingdom until June 1991 and after that date by SGS-XRAL in Canada. Tüprag submitted Quality Assurance/Quality Control (QA/QC) samples at the rate of 1 in 20; the QA/QC samples included standards, duplicates and blanks.

Koza has acquired the historic drillhole data from paper cross-sections, paper drill logs and digital files acquired from Tüprag. In order to validate the database, Koza resampled five core holes from different locations at Kaymaz, taking one-half of the remaining half-core and assaying at ALS. The results compared within acceptable limits with the original assays. SRK has spot checked about 10% of the data with the paper cross-sections and has found no errors.

Sample lengths typically are 1 m, 1.5 m, or 2 m in length in the Damdamca zone and are all 1.5 m in the Main zone. The core sample intervals were adjusted at geologic boundaries, resulting in some intervals of irregular length.

There are several gold grades of exactly 10 g/t, and it is SRK's opinion that these are upper detection limit values and that these samples were not re-run with a procedure that would produce more accurate results.

The historic Tüprag database at Kaymaz is not readily verifiable; however, the drilling was conducted by a reputable company that presumably used practices that were industry standard at the time and it is SRK's opinion that the historic database is acceptable for resource estimation. The Tüprag data represents about 7% of the total meters drilled at the Main Zone and 4% of the total meters drilled at Mermerlik. At Damdamca, the Tüprag drilling was about 64% of the total database, but that resource has been mined out and is no longer included in the Kaymaz resource.

Between 2008 and 2014, Koza drilled 524 core holes in the four areas, using its standard drilling, logging, sampling and assaying methods. The Koza holes account for all of the core drilling and about 87% of the total meters drilled at the project. Koza's QA/QC samples include standards, duplicates and blanks and the results are monitored on a regular basis. The core recovery ranges from 0 to 100%, averaging 94%.

### 2.5.3 Sample Preparation and Analysis

Samples are in the control of Koza personnel either in a locked field vehicle or at a mine site in a locked building until they are submitted to the laboratory for analysis. Once the samples are submitted to the laboratory, chain of custody is controlled by the laboratory. This is industry best practice.

Drillhole and trench samples have been analyzed at three laboratories:

- ALS Global in Vancouver, British Columbia Canada (ALS Vancouver);
- Koza laboratory at the Ovacik Mine; and
- Koza laboratory at the Kaymaz Mine

**ALS Vancouver**

Samples were submitted to ALS Global in Vancouver, Canada (ALS Vancouver) for preparation and analysis. ALS Vancouver has ISO 17025:2005 accreditation, which is specific to analytical methods, through the Standards Council of Canada valid through May 18, 2017.

Soil samples were analyzed using ALS code ME-MS41, a 51 element package with ultra-trace level sensitivity typically used for rock samples and drill core. This analysis uses an aqua regia digestion and is finished using both Inductively Coupled Plasma-Atomic Emission Spectroscopy (ICP-AES) and Inductively Coupled Plasma-Mass Spectroscopy (ICP-MS). Because of the sample size, ME-MS41 is considered a semi-quantitative method for gold. Because of this, Koza also analysis for gold using ALS code Au-ICP22, which is a FA method using a 50 g charge and ICP-AES finish. The aqua regia digestion used in method ME-MS41 may not provide representative results for refractory minerals and elements such as molybdenum (ALS Global, 2014). The analytical method is appropriate for the mineralization. Table 2.5.3.1 presents the analytes with upper and lower detection limits for ALS ME-MS41 and Au-ICP22.

**Table 2.5.3.1: Analytes and Upper and Lower Detection Limits for ALS Codes ME-MS41 and Au-ICP22 in ppm Unless Otherwise Noted**

| Method | Analyte | Range | Method | Analyte | Range | Method | Analyte | Range |
|---|---|---|---|---|---|---|---|---|
| Au-ICP22 | Au | 0.001-10 | ME-MS41 | Hf | 0.02-500 | ME-MS41 | Sc | 0.1-10,000 |
| ME-MS41 | Ag | 0.01-100 | ME-MS41 | Hg | 0.01-10,000 | ME-MS41 | Se | 0.2-1,000 |
| ME-MS41 | Al | 0.01-25% | ME-MS41 | In | 0.005-500 | ME-MS41 | Sn | 0.2-500 |
| ME-MS41 | Au | 0.2-25 | ME-MS41 | K | 0.01-10% | ME-MS41 | Sr | 0.2-10,000 |
| ME-MS41 | B | 10-10,000 | ME-MS41 | La | 0.2-10,000 | ME-MS41 | Ta | 0.01-500 |
| ME-MS41 | Ba | 10-10,000 | ME-MS41 | Li | 0.1-10,000 | ME-MS41 | Te | 0.01-500 |
| ME-MS41 | Be | 0.05-1,000 | ME-MS41 | Mg | 0.01-25% | ME-MS41 | Th | 0.2-10,000 |
| ME-MS41 | Bi | 0.01-10,000 | ME-MS41 | Mn | 5-50,000 | ME-MS41 | Ti | 0.005-10% |
| ME-MS41 | Ca | 0.01-25% | ME-MS41 | Mo | 0.05-10,000 | ME-MS41 | Tl | 0.02-10,000 |
| ME-MS41 | Cd | 0.01-1,000 | ME-MS41 | Na | 0.01-10% | ME-MS41 | U | 0.05-10,000 |
| ME-MS41 | Ce | 0.02-500 | ME-MS41 | Nb | 0.05-500 | ME-MS41 | V | 1-10,000 |
| ME-MS41 | Co | 0.1-10,000 | ME-MS41 | Ni | 0.2-10,000 | ME-MS41 | W | 0.05-10,000 |
| ME-MS41 | Cr | 1-10,000 | ME-MS41 | P | 10-10,000 | ME-MS41 | Y | 0.05-500 |
| ME-MS41 | Cs | 0.05-500 | ME-MS41 | Pb | 0.2-10,000 | ME-MS41 | Zn | 2-10,000 |
| ME-MS41 | Cu | 0.2-10,000 | ME-MS41 | Rb | 0.1-10,000 | ME-MS41 | Zr | 0.5-500 |
| ME-MS41 | Fe | 0.01-50% | ME-MS41 | Re | 0.001-50 | | | |
| ME-MS41 | Ga | 0.05-10,000 | ME-MS41 | S | 0.01-10% | | | |
| ME-MS41 | Ge | 0.05-500 | ME-MS41 | Sb | 0.05-10,000 | | | |

Source: ALS Global, 2014

Gold was analyzed at ALS using either ALS code Au-AA24 or, if over limit, either Au-AA26 or Au-GRA21. Both Au-AA24 and Au-AA26 are Fire Assay (FA) using a 50 g charge and an Atomic Absorption Spectroscopy (AAS) finish, but Au-AA24 has an analytical range of 0.005 to 10 ppm while Au-AA26 has a range of 0.01 to 100 ppm. The Au-GRA21 code is for a 30 g charge FA with a gravimetric finish and an analytical range of 0.05 to 1,000 ppm. Silver is analyzed using four acid digestion and Inductively Coupled Atomic Emission Spectroscopy (ICP-AES) under code ME-ICP61, which is a 33 element geochemistry package an analytical range for silver of 0.5 to 100 ppm. Over limit silver is analyzed using code Ag-AA47. This is an aqua regia digestion with an AAS finish and analytical range of 1 to 1,500 ppm. In addition, Hg is analyzed using code ME-MS42, which is an aqua regia digestion and ICP Mass Spectroscopy (MS) method with an analytical range of 0.005 to 25 ppm. Table 2.5.3.2 presents the analytes and upper and lower detection limits for ALS code ME-ICP.

**Table 2.5.3.2: Analytes and Upper and Lower Detection Limits for ALS Codes ME-ICP61 in ppm Unless Otherwise Noted**

| Analyte | Range | Analyte | Range | Analyte | Range |
|---|---|---|---|---|---|
| Ag | 0.5-100 | Fe | 0.01-50% | S | 0.01-10% |
| Al | 0.01-50% | Ga | 10-10,000 | Sb | 5-10,000 |
| As | 5-10,000 | K | 0.01-10% | Sc | 1-10,000 |
| Ba | 10-10,000 | La | 10-10,000 | Sr | 1-10,000 |
| Be | 0.5-1,000 | Mg | 0.01-50% | Th | 20-10,000 |
| Bi | 2-10,000 | Mn | 5-100,000 | Ti | 0.01-10% |
| Ca | 0.01-50% | Mo | 1-10,000 | Tl | 10-10,000 |
| Cd | 0.05-1,000 | Na | 0.01-10% | U | 10-10,000 |
| Co | 1-10,000 | Ni | 1-10,000 | V | 1-10,000 |
| Cr | 1-10,000 | P | 10-10,000 | W | 10-10,000 |
| Cu | 1-10,000 | Pb | 2-10,000 | Zn | 2-10,000 |

Source: ALS Global, 2014

**Koza Laboratories**

The Koza laboratories analyze gold using aqua regia – di-isobutyl ketone (AR-DIBK or DIBK) digestion with Absorption Spectroscopy (AAS) finish. Silver is analyzed using aqua regia digestion and AAS finish. The Kaymaz laboratory has the following analytical capability:

- Au by aqua regia – di-isobutyl ketone (AR-DIBK or DIBK) and Atomic Absorption Spectroscopy (AAS) finish with a lower detection limit of 0.1 ppm;
- Ag by aqua regia and AAS finish with a lower detection limit of 0.2 ppm;
- Cu, Ni, As, Sb and Mn by aqua regia and ICP-MS finish all with a lower detection limit of 0.001 ppm;
- C and S by LECO both having a lower detection limit of 0.01%; and
- Fe by AAS with a lower detection limit of 001%.

The Ovacık laboratory has the following capabilities:

- Au by aqua regia – DIBK (AR-DIBK) with a lower detection limit of 0.1 ppm; and
- Ag by aqua regia and AAS finish with a lower detection limit of 0.2 ppm.

The Ovacık laboratory also conducts Fire Assay (FA) using a 15 g charge with an Atomic Absorption Spectroscopy (AAS) finish. If the sample exceeds 2,000 ppm the laboratory uses a gravimetric finish. The lower detection limit is 0.1 ppm.

The production samples are analyzed at the Kaymaz laboratory.

## 2.5.4 Quality Assurance and Quality Control (QA/QC)

Koza is currently submitting its exploration samples to the Kaymaz laboratory. The QA/QC discussed below is monitoring that laboratory.

**Insertion of Internal Controls**

Koza inserts approximately one blank per drillhole and Reference Materials (RMs) at a frequency of approximately one in 25 samples and duplicate samples at a rate of one or two per drillhole. These samples are numbered in sequence by the core logging geologist. The location of the control samples is noted on the sample log and in the sample database. The 2014 samples were analyzed at the Kaymaz laboratory.

**Reference Materials**

Koza currently uses site-specific Reference Materials (RMs) at Kaymaz. The site-specific RMs were crushed, pulverized and homogenized using a single axis cement mixer at the Koza laboratory as described by Bloom (2013). Koza had ALS analyze 30 samples of each site specific RM for Au and Ag at its Vancouver, Johannesburg and Lima laboratories (ten at each lab). Gold was analyzed with two methods: FA with AAS finish and aqua regia digestion and ICP finish; silver was analyzed by aqua regia digestion with AAS finish. SRK notes that the gold FA method is the same used by ALS for Koza samples and is therefore a suitable standard for use in evaluating the results for samples send to ALS; the aqua regia digestion at ALS does not include the DIBK digestion used by Koza and may not be a suitable standard for evaluating results for samples sent to the Koza laboratories. Silver was analyzed in the RMs using aqua regia digestion and ICP finish this method is the same as the Koza laboratory with the exception that ALS uses ICP while Koza uses AAS. The two instruments should provide essentially the same results depending on calibration.

The site-specific RMs have not undergone a round robin analysis and therefore are not certified. Bloom (2013) also suggested that the RMs may not have been properly homogenized. ALS provided a report with summary statistics for each RM. For all RMs, Koza uses a performance range of ±10% of the mean. For the site-specific RMs produced by Koza, Bloom (2013) recommends using 7% as a threshold for a failure based on her communication with ALS.

Koza used three site specific RMs during 2014. Table 2.5.4.1 presents the expected mean, standard deviations and summaries of the analyses of the Kaymaz site-specific Au RMs. The expected mean is from the aqua regia digestion as this method is closer to that used by the Kaymaz laboratory. Table 2.5.4.2 presents the Ag results from the four RMs used in 2014

**Table 2.5.4.1: Results of Au RM Analyses at Kaymaz –Site Specific RMs**

| CRM | Number of Samples | Expected (ppm) | | Observed (ppm) | | % of Expected | Outside ±7% | |
|---|---|---|---|---|---|---|---|---|
| | | Mean | Std Dev | Mean | Std Dev | | No. Failures | % Failure Rate |
| KY01 | 95 | 0.752 | 0.053 | 0.73 | 0.01 | 97% | 0 | 0 |
| KY02 | 60 | 1.408 | 0.075 | 1.42 | 0.013 | 101% | 0 | 0 |
| KY03 | 28 | 2.806 | 0.129 | 2.86 | 0.018 | 102% | 0 | 0 |
| **Total** | **183** | | | | | | **0** | **0** |

**Table 2.5.4.2: Results of Ag Site Specific RMs**

| CRM | Number of Samples | Expected (ppm) | | Observed (ppm) | | % of Expected | Outside ±7% | |
|---|---|---|---|---|---|---|---|---|
| | | Mean | Std Dev | Mean | Std Dev | | No. Failures | % Failure Rate |
| KY01 | 81 | 3.89 | 0.15 | 3.87 | 0.22 | 99.5 | 27 | 33% |
| KY02 | 45 | 4.45 | 0.34 | 4.65 | 0.22 | 104.5 | 16 | 36% |
| KY03 | 26 | 4.03 | 0.03 | 4.19 | 0.17 | 103.9 | 5 | 19% |
| **Total** | **152** | | | | | | **48** | **32%** |

For Au there were no failures in the Kaymaz site specific RMs and the observed means are between 97% and 102% of the expected values. The silver RMs have a very high failure rate at ±7% of the expected mean. The RMs are site specific, but the method of producing them may not have provided a well homogenized product. Because the gold RMs are performing very well, SRK suspects the problem may be with the silver analysis at the Kaymaz laboratory. SRK recommends that Koza investigate the silver analysis by contacting the laboratory and discussing the failures with the laboratory. SRK also recommends that Koza submit CRMs for silver to the laboratory to check performance.

Koza reviews all QA/QC during drilling programs and contacts the laboratory when analytical failures are identified. If the failure is determined to be a laboratory failure, Koza requests the lab reanalyze the failed sample with 5 samples before and after the failure. If all control samples fail, then Koza requested the entire sample batch be reanalyzed. This is industry best practice.

**Blanks**

Sample blanks test for contamination in preparation and assaying and handling errors. Koza inserted one sample blank per drillhole. Before June 2012, Koza used pulp blanks but has used preparation blanks since then. A blank failure is a result greater than five times the detection limit. SRK has examined the 2014 results for Au and Ag in 56 blank samples and found no failures. The results indicate that the preparation laboratory is performing well.

**Field Duplicates**

Field duplicates are created by sampling a second quarter of the core. The objective of testing field duplicates is to understand the variance of the actual sampling and the first size reduction step.

Koza had prepared field duplicates in the past and has discontinued the practice which is acceptable in operating mines where the variability is known.

**Preparation Duplicates**

Preparation duplicates are created by collecting a second split of the crushed sample (coarse reject) using the same type of splitter and collecting the same weight as the original sample. The objective is to determine if:

- Splitting procedures are applied consistently; and
- Changes are required for the crush size.

Preparation duplicates provide an idea of variability in the mineralization. If there is little variability, then they can also provide an estimate of laboratory precision.

Koza sent preparation duplicates to the Kaymaz lab, for analysis. The 2014 duplicate analysis data provided to SRK includes 79 duplicate pairs with gold and silver analysis. Of these 79 samples, there are 28 gold samples and 79 silver samples above the lower detection limit and six gold samples above the cutoff grade of 0.8 g/t Au for mineral resources at Kaymaz. The gold duplicates are all within ±6% of the original. Silver shows more variability with two samples exceeding ±10% of the original but less than ±20%, which is acceptable performance for a preparation duplicate

During 2013, the preparation duplicates demonstrated remarkable reproducibility at ±0.5% and it was SRK's opinion at that time that the laboratory was reanalyzing the duplicates until the results were extremely close to the original sample. The 2014 results also demonstrate remarkable reproducibility for a gold deposit with reproducibility within ±6%. Preparation duplicates are expected to be reproducible with in ±20% and pulp duplicates are expected to be within ±10%. The results show that the preparation duplicates are providing better reproducibility than pulp duplicates, which is unusual in a gold deposit.

SRK suggests that the geology department confirm that the laboratory is analyzing the duplicates one time and reporting that result. If the laboratory is analyzing the duplicate samples multiple times to get a similar result as the original, then no useful data is being produced regarding preparation duplicates. If the laboratory is analyzing the preparation duplicates multiple times, then the Koza Geology department should reiterate the requirement of analyzing the duplicate only once. If it is determined that the preparation duplicates are actually reproducible within ±6%, then preparation duplicates can be discontinued and Koza should use pulp duplicates to track analytical precision.

**Pulp Duplicates**

Koza has not submitted any pulp duplicate samples to the Kaymaz lab. Pulp duplicates are the primary method of checking the precision of analysis. SRK recommends that the Company begin submitting pulp duplicates as part of its QA/QC program. SRK suggests that the laboratory be instructed to analyze its pulp duplicates once and not reanalyze until the results match the original. The reported value for a sample should be the first analysis.

**Secondary Check Lab Analysis**

A previous check sample study showed a bias between the Kaymaz and SGS the secondary lab, with the SGS showing higher results. Both laboratories used AR-DIBK and AAS for analysis. Koza did not send RMs with the samples so it is unknown which laboratory had better accuracy. Koza did not submit check samples to a secondary laboratory during 2014, but had done so the previous year.

Though it is not possible to check the accuracy of the secondary laboratory results without CRMs, there is less than ±10% difference between the two labs analytical results, demonstrating acceptable precision. Since the Kaymaz results are lower than the secondary lab results, the Kaymaz laboratory was conservative relative to the secondary lab during the monitoring period.

SRK recommends that Koza reinitiate sending pulp samples as a check to a secondary laboratory and send RMs or CRMs with the submission. The submission should include one RM or CRM for every five to six pulp samples. It is also important the primary and the secondary laboratory use the same analytical technique so that direct comparison can be made. This lab check should be done to continue to monitor the precision and accuracy of the Kaymaz laboratory.

**Conclusions and Recommendations**

Koza monitors QA/QC of the laboratory analyses by inserting internal control samples into the sample stream. Reference materials, blanks and preparation duplicates are systematically inserted to ensure reliability and accuracy of the laboratory. Should there be a QA/QC sample failure during a drilling program, Koza investigates the failure to determine why it occurred and takes appropriate action. If the failure is due to laboratory error, then Koza requests that the failed samples and five samples before and after the failure be reanalyzed. If there are multiple control sample failures, then Koza requests that the entire batch be reanalyzed. This is industry best practice.

SRK has the following recommendations:

- The use of the site specific RMs should be discontinued and CRMs as suggested by Bloom (2013) should be used;
- Plot the standards against time to determine if the laboratory has trouble during a certain period;

- Duplicate samples submitted to the Kaymaz lab should be analyzed once and should not be reanalyzed until the duplicate is similar to the original. This defeats the purpose of submitting duplicate samples;
- Pulp duplicates should be prepared and submitted to the primary lab;
- Investigate the silver failures at the laboratory by contacting the laboratory and discussing the failures with the laboratory;
- Submit commercial CRMs for silver to the laboratory to check performance;
- Submit pulp samples to a secondary laboratory as a check of the Kaymaz lab;
- When using a secondary check laboratory, plot QA/QC data individually for each laboratory;
- Use the same analytical methods at the primary and secondary laboratories; and
- RM samples should be submitted with the check assay samples.

Overall the laboratory is performing within acceptable limits and the QA/QC program is sufficiently monitoring laboratory accuracy and reliability.

## 2.6 Mineral Resources

The Mermerlik (Koza, 2011c) and Kizilagil (Koza, 2011b) resources were estimated in 2011 by Koza and the Main Zone (Koza, 2014) resource was estimated by Koza in 2014.

### 2.6.1 Damdamca

The Damdamca pit was completed in 2014 and the area is not included in the 2014 Kaymaz resource. Table 2.6.1.1 compares the mined production to the final resource model.

**Table 2.6.1.1: Comparison of Damdamca Mined Production and Resource Model**

| Material | Production | | | Resource Model | | |
|---|---|---|---|---|---|---|
| | Tonnes | Au gpt | Au ozs | Tonnes | Au gpt | Au ozs |
| ROM | 1,604,721 | 7.10 | 366,240 | 1,932,170 | 5.61 | 348,715 |
| LG | 121,913 | 0.92 | 3,609 | 27,385 | 0.93 | 822 |
| Min Zone | 479,196 | 0.59 | 9,156 | 24,074 | 0.59 | 453 |
| **Total** | **2,205,830** | **5.34** | **379,005** | **1,983,630** | **5.49** | **349,990** |

Source: Koza, 2014

### 2.6.2 Main Zone

**<u>Geologic Model and Assay Statistics</u>**

Koza constructed 41 separate gold grade shell wireframes for the Main Zone, several of which contain a single drillhole. The wireframes were grouped into seven domains depending on location and orientation. The wireframes have an east-west extent of 1800 m, a north-south extent of 185 m and a vertical extent of 260 m. Most of the wireframes dip to the south at about 30°. The thickness of the wireframes ranges from 1 to 25 m, with an average of about 8 to 10 m.

Table 2.6.2.1 presents statistics of the raw drillhole and trench assays within the Main Zone. The drillholes, trenches and wireframes are shown in plan view and oblique view in Figures 2.6.2.1 and 2.6.2.2, respectively.

**Table 2.6.2.1: Statistics of Uncapped Assays at the Main Zone**

| Zone | Metal | Number | Minimum | Maximum | Mean | Std Dev | CV |
|---|---|---|---|---|---|---|---|
| 1 | Au | 581 | 0.00 | 153.00 | 4.10 | 8.42 | 2.05 |
| | Ag | 522 | 0.00 | 94.70 | 5.84 | 7.17 | 1.23 |
| 2 | Au | 237 | 0.00 | 34.65 | 6.19 | 6.27 | 1.01 |
| | Ag | 237 | 0.27 | 144.20 | 4.96 | 6.96 | 1.40 |
| 3 | Au | 478 | 0.00 | 22.58 | 3.39 | 4.14 | 1.22 |
| | Ag | 436 | 0.00 | 21.67 | 4.29 | 3.56 | 0.83 |
| 4 | Au | 1132 | 0.00 | 199.70 | 6.54 | 11.02 | 1.69 |
| | Ag | 1122 | 0.00 | 555.00 | 6.51 | 21.28 | 3.27 |
| 5 | Au | 509 | 0.00 | 23.99 | 2.63 | 3.75 | 1.43 |
| | Ag | 430 | 0.30 | 88.98 | 4.71 | 6.53 | 1.39 |
| 6 | Au | 181 | 0.00 | 21.36 | 4.77 | 4.22 | 0.88 |
| | Ag | 181 | 0.99 | 13.73 | 3.17 | 1.87 | 0.59 |
| 7 | Au | 62 | 0.00 | 22.59 | 5.02 | 5.09 | 1.01 |
| | Ag | 62 | 0.73 | 8.80 | 2.92 | 1.54 | 0.53 |
| All | Au | 3,180 | 0.001 | 199.7 | 4.76 | 8.13 | 1.71 |
| | Ag | 2,990 | 0.001 | 555.00 | 5.40 | 13.72 | 2.54 |

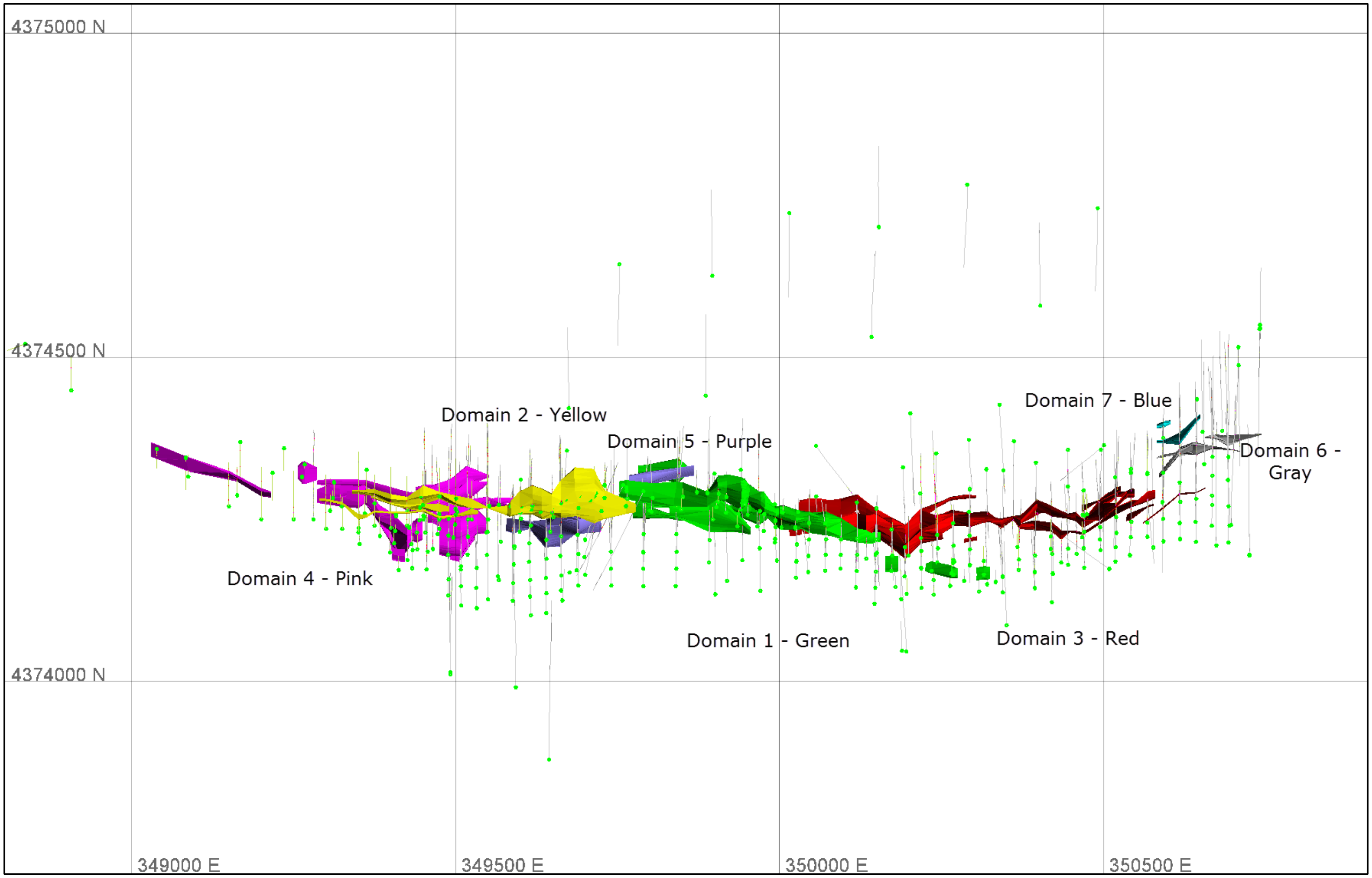



**Figure 2.6.2.1: Main Zone Drillholes and Wireframes in Plan View**

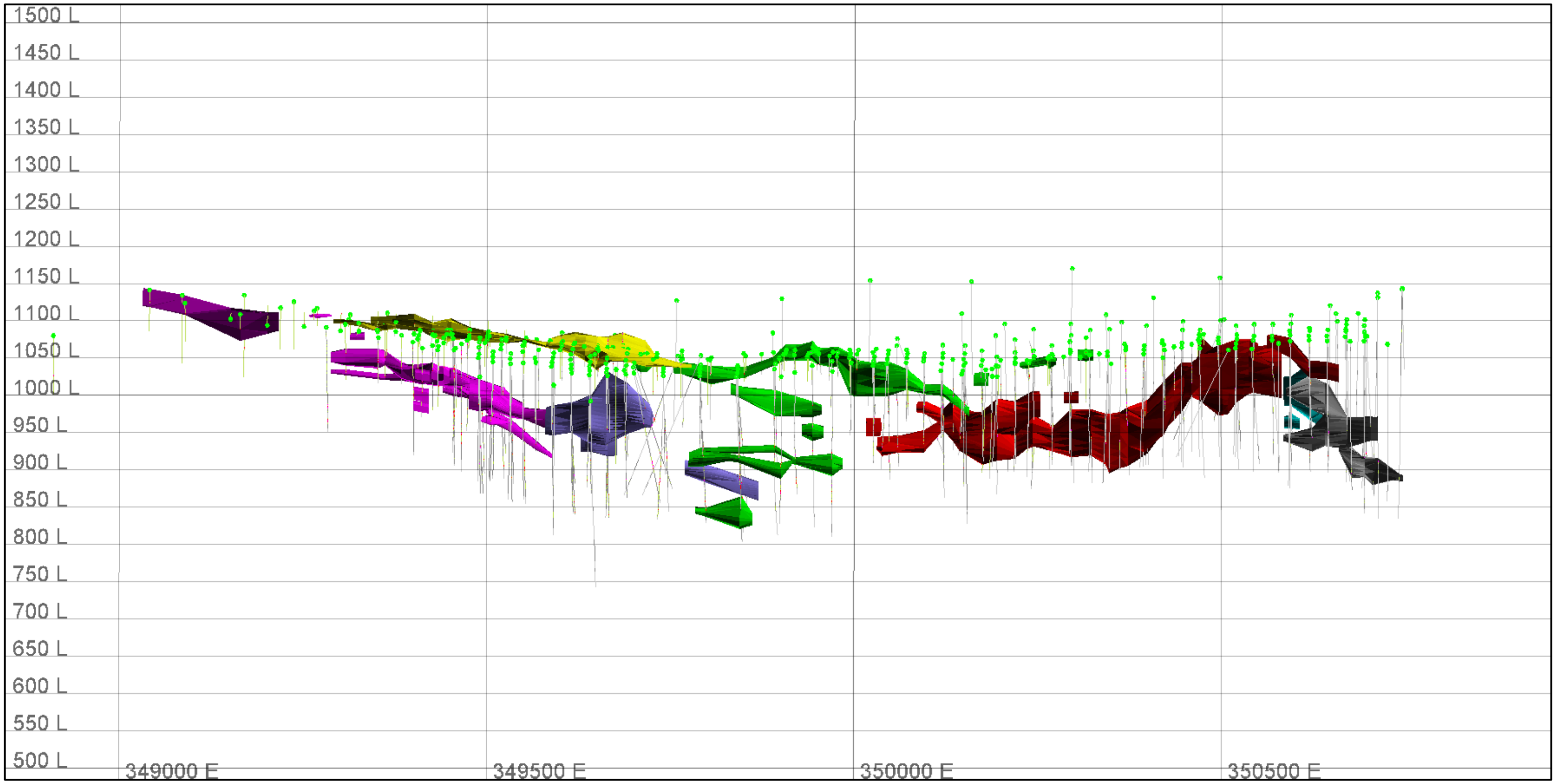



**Figure 2.6.2.2: Oblique View of Main Zone Wireframes, Looking North**

**Capping and Compositing**

The sample lengths at the Main zone are predominately less than or equal to 1 m, but about 25% are greater than 1 m. The samples were composited on 1.5 m lengths. Koza used the option to composite by distribution where the composite lengths are divided equally across the wireframe based on a preferred length of 1.5 m. This method produces variable composite lengths whereas the purpose of compositing is to standardize the lengths. Figure 2.6.2.3 shows the histogram of Koza's composite lengths compared to using a straight 1.5 m length, which produces much more uniform lengths.

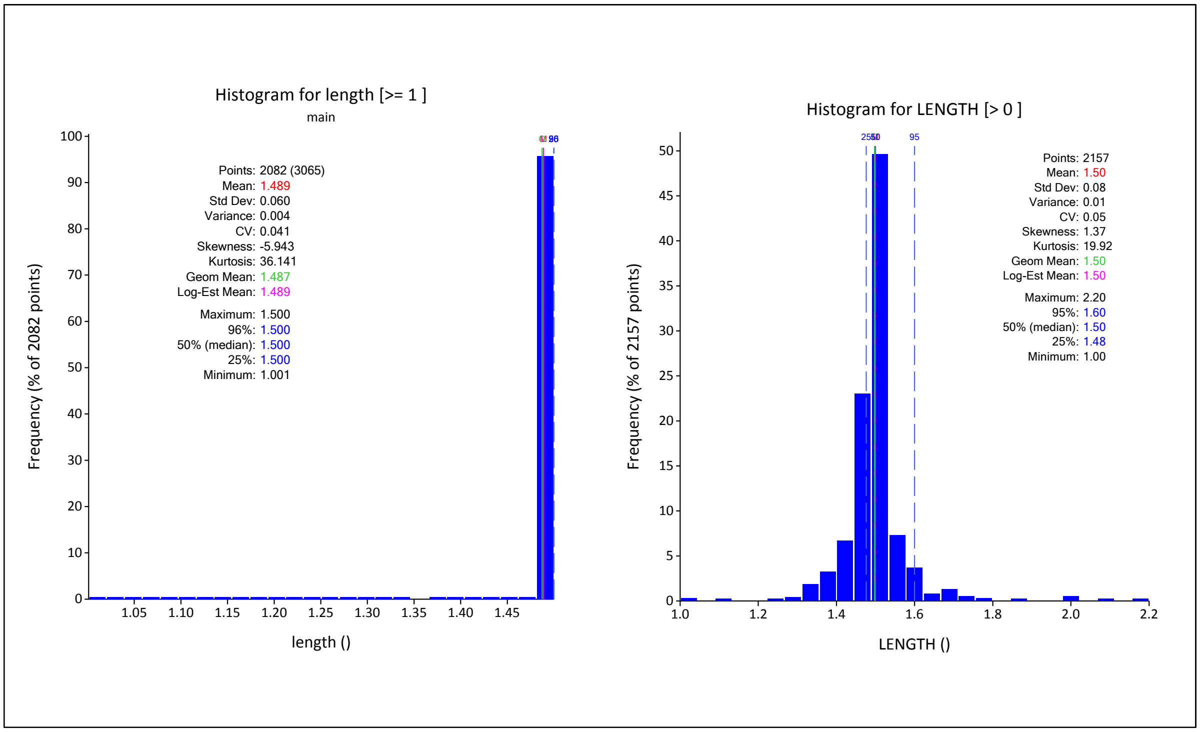



**Figure 2.6.2.3: Comparison of Koza's Composite Lengths by Distribution (right) Compared to Composite Lengths by 1.5 m (left)**

Statistics of uncapped composites are shown in Table 2.6.2.2. The CV has been reduced by compositing.

**Table 2.6.2.2: Statistics of Uncapped Composites at the Main Zone**

| Zone | Metal | Number | Minimum | Maximum | Mean | Std Dev | CV |
|---|---|---|---|---|---|---|---|
| 1 | Au | 401 | 0.00 | 70.32 | 4.10 | 6.42 | 1.57 |
| | Ag | 342 | 0.16 | 52.50 | 5.84 | 5.94 | 1.02 |
| 2 | Au | 338 | 0.06 | 17.84 | 3.37 | 3.50 | 1.04 |
| | Ag | 296 | 0.37 | 17.20 | 4.30 | 3.10 | 0.72 |
| 3 | Au | 730 | 0.00 | 149.57 | 6.52 | 9.92 | 1.52 |
| | Ag | 720 | 0.00 | 555 | 6.49 | 20.90 | 3.22 |
| 4 | Au | 376 | 0.00 | 19.12 | 2.63 | 3.33 | 1.27 |
| | Ag | 302 | 0.34 | 77.60 | 4.71 | 6.16 | 1.31 |
| 5 | Au | 152 | 0.00 | 22.86 | 6.21 | 5.56 | 0.90 |
| | Ag | 152 | 0.31 | 41.52 | 4.97 | 4.56 | 0.92 |
| 6 | Au | 110 | 0.12 | 17.13 | 4.79 | 3.51 | 0.73 |
| | Ag | 110 | 1.04 | 9.37 | 3.18 | 1.53 | 0.48 |
| 7 | Au | 41 | 0.36 | 17.42 | 5.02 | 4.33 | 0.86 |
| | Ag | 41 | 0.81 | 6.59 | 2.92 | 1.26 | 0.43 |
| All | Au | 2,148 | 0.00 | 149.57 | 4.75 | 7.11 | 1.50 |
| | Ag | 1,963 | 0.00 | 555.00 | 5.39 | 13.28 | 2.46 |

Koza reviewed histograms, probability plots and plots of the data to determine the need for capping values. Gold was capped at 40 g/t and silver was capped at 32 g/t for all domains. Table 2.6.2.3 contains statistics of the composites at the Main Zone after capping. The coefficient of variation (CV) has been reduced to under 2 for Domains 1 through 4 and below 1 for Domains 5 through 7.

**Table 2.6.2.3: Statistics of Capped Composites at the Main Zone**

| Zone | Metal | Number | Minimum | Maximum | Mean | Std Dev | CV |
|---|---|---|---|---|---|---|---|
| 1 | Au | 401 | 0.00 | 40.00 | 4.04 | 5.83 | 1.44 |
| | Ag | 342 | 0.16 | 32.00 | 5.78 | 5.54 | 0.96 |
| 2 | Au | 338 | 0.06 | 17.84 | 3.39 | 3.50 | 1.03 |
| | Ag | 296 | 0.37 | 17.20 | 4.31 | 3.10 | 0.72 |
| 3 | Au | 730 | 0.00 | 40.00 | 6.24 | 7.79 | 1.25 |
| | Ag | 720 | 0.00 | 32.00 | 5.78 | 5.33 | 0.92 |
| 4 | Au | 376 | 0.00 | 19.12 | 2.62 | 3.32 | 1.27 |
| | Ag | 302 | 0.34 | 32.00 | 4.55 | 4.65 | 1.02 |
| 5 | Au | 152 | 0.00 | 22.86 | 6.28 | 5.58 | 0.89 |
| | Ag | 152 | 0.31 | 32.00 | 4.93 | 4.11 | 0.83 |
| 6 | Au | 110 | 0.12 | 17.13 | 4.78 | 3.53 | 0.74 |
| | Ag | 110 | 1.04 | 9.37 | 3.18 | 1.54 | 0.48 |
| 7 | Au | 41 | 0.36 | 17.42 | 5.01 | 4.33 | 0.87 |
| | Ag | 41 | 0.81 | 6.59 | 2.92 | 1.26 | 0.43 |
| All | Au | 2,148 | 0.00 | 40.00 | 4.65 | 6.02 | 1.29 |
| | Ag | 1,963 | 0.00 | 32.00 | 5.09 | 4.74 | 0.93 |

**Density**

A total of 180 HQ sized samples were collected covering a geographical range of Main Zone. The samples were collected from 90 drillholes. Samples were grouped according to rock type, alteration and degree of breakage. Initial determinations using Archimedes method were made. Core was covered with wax to preserve pore space and the samples were weighed in water and air. The average density value of 2.65 g/cm$^3$ is used in the Main Zone resource estimation. The density is on a dry tonnage basis.

**Grade Estimation**

The parent block size at the Main zone has dimensions of 15 m x 15 m x 5 m and sub-blocking allowed to 1 m in all directions. The parent block size is about half the drillhole spacing.

Gold and silver were estimated with ID2, ID3 and NN algorithms, using the parameters shown in Table 2.6.2.4. An octant search was used for Domains 1, 2, 3 and 4, requiring a minimum of three octants and a maximum of four samples per octant. A dynamic anisotropic search was used for Domain 3. The parent block size was used in the estimation.

**Table 2.6.2.4: Grade Estimation Parameters at Main Zone**

| Domain | Pass | Search Distance | | | Search Orientation | | | Samples | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Major | Semi | Minor | Major | Semi | Minor | Min | Max | Max/DH |
| 1,2,3*,4 | 1 | 60 | 60 | 10 | 00,090 | -30,180 | 60,000 | 8 | 20 | 5 |
| | 2 | 120 | 120 | 20 | | | | 8 | 20 | 5 |
| | 3 | 240 | 240 | 40 | | | | 4 | 12 | 5 |
| 5,6,7 | 1 | 30 | 30 | 30 | 00,090 | -50,180 | 40,000 | 8 | 20 | 5 |
| | 2 | 60 | 60 | 60 | | | | 8 | 20 | 5 |
| | 3 | 120 | 120 | 120 | | | | 4 | 12 | 5 |

*A dynamic anisotropic search was used for Domain 3

**Block Model Validation**

Koza reviewed cross-sections visually on the computer screen to compare composite and block grades. A comparison of the composites to the estimated grades is shown in Table 2.6.2.5. The ID and NN estimations are within +5% of the composite grades except for gold in Domains 1 and 7, where estimated gold is about 10% less than the composite grade. The final resource model is based on the ID2 estimation.

**Table 2.6.2.5: Main Zone Comparison of Composites and Estimated Grades**

| Zone | Metal | Composites | ID2 | ID3 | NN |
|---|---|---|---|---|---|
| 1 | Au | 4.04 | 3.66 | 3.66 | 3.53 |
| | Ag | 5.78 | 5.72 | 5.75 | 5.74 |
| 2 | Au | 3.39 | 3.39 | 3.40 | 3.21 |
| | Ag | 4.31 | 4.17 | 4.18 | 4.10 |
| 3 | Au | 6.24 | 6.12 | 6.09 | 5.57 |
| | Ag | 5.78 | 5.76 | 5.75 | 5.58 |
| 4 | Au | 2.62 | 2.51 | 2.49 | 2.48 |
| | Ag | 4.55 | 4.75 | 4.80 | 4.85 |
| 5 | Au | 6.28 | 6.20 | 6.17 | 5.49 |
| | Ag | 4.93 | 5.14 | 5.10 | 5.01 |
| 6 | Au | 4.78 | 4.82 | 4.87 | 5.06 |
| | Ag | 3.18 | 3.27 | 3.29 | 3.09 |
| 7 | Au | 5.01 | 4.42 | 4.37 | 3.99 |
| | Ag | 2.92 | 2.96 | 2.93 | 2.62 |
| All | Au | 4.65 | 4.68 | 4.67 | 4.36 |
| | Ag | 5.09 | 5.23 | 5.24 | 5.15 |

SRK suggests that Koza also generate swath plots as a part of the validation.

## Mineral Resource Classification

The resources were classified as follows:

- Measured: Estimated in first pass and the number of drillholes more than 4;
- Indicated: Estimated in first pass and number of drillholes equal to 3 or 4; and
- Inferred: Remaining blocks.

## Mineral Resource Statement

The cutoff grade of 0.80 g/t Au is based on the open pit mining assumptions shown in Table 2.6.2.6. The one year rolling average gold price is US$1,266; the two year average is US$1,339; and the three year average is US$1,449.

**Table 2.6.2.6: Kaymaz Cutoff Grade Assumptions**

| Item | Units | Prices and Costs |
|---|---|---|
| Gold Price | US$/oz | 1,450 |
| Gold Recovery | % | 82 |
| Gold Refining | US$/oz | 3.44 |
| Royalty | % | 0 |
| Government Right | % | 1 |
| Process Cost | US$/t | 22.00 |
| Mining Cost | US$/t | 0.00 |
| G&A Cost | US$/t | 8.00 |
| Final Cutoff grade | g/t | 0.80 |

Source: Koza, 2014

It is becoming an industry practice to state mineral resources within a pit optimization shell. Koza conducted a pit optimization using the parameters in Table 2.6.2.6. Approximately 90% of the Measured and Indicated resources and 45% of the Inferred resources fall within the pit shell.

The Measured, Indicated, and Inferred resources at a cutoff grade of 0.80 g/t Au are listed in Table 2.6.2.7, the tonnage is inclusive of ore reserves.

**Table 2.6.2.7: Main Zone Mineral Resources, Including Ore Reserves, at December 31, 2014**

| Classification | kt | g/t Au | g/t Ag | koz Au | koz Ag |
|---|---|---|---|---|---|
| Measured | 640 | 4.39 | 5.57 | 90 | 115 |
| Indicated | 1,459 | 5.49 | 5.46 | 263 | 261 |
| Measured and Indicated | 2,129 | 5.16 | 5.49 | 353 | 376 |
| Inferred | 990 | 4.47 | 5.02 | 142 | 159 |

Tonnages and grade are rounded to reflect approximation;
Resources are stated at a cutoff grade of 0.80 g/t;
Open pit resources are contained within grade shells but are not constrained by a pit optimization shell; and
Mineral Resources are reported inclusive of Mineral Reserves.

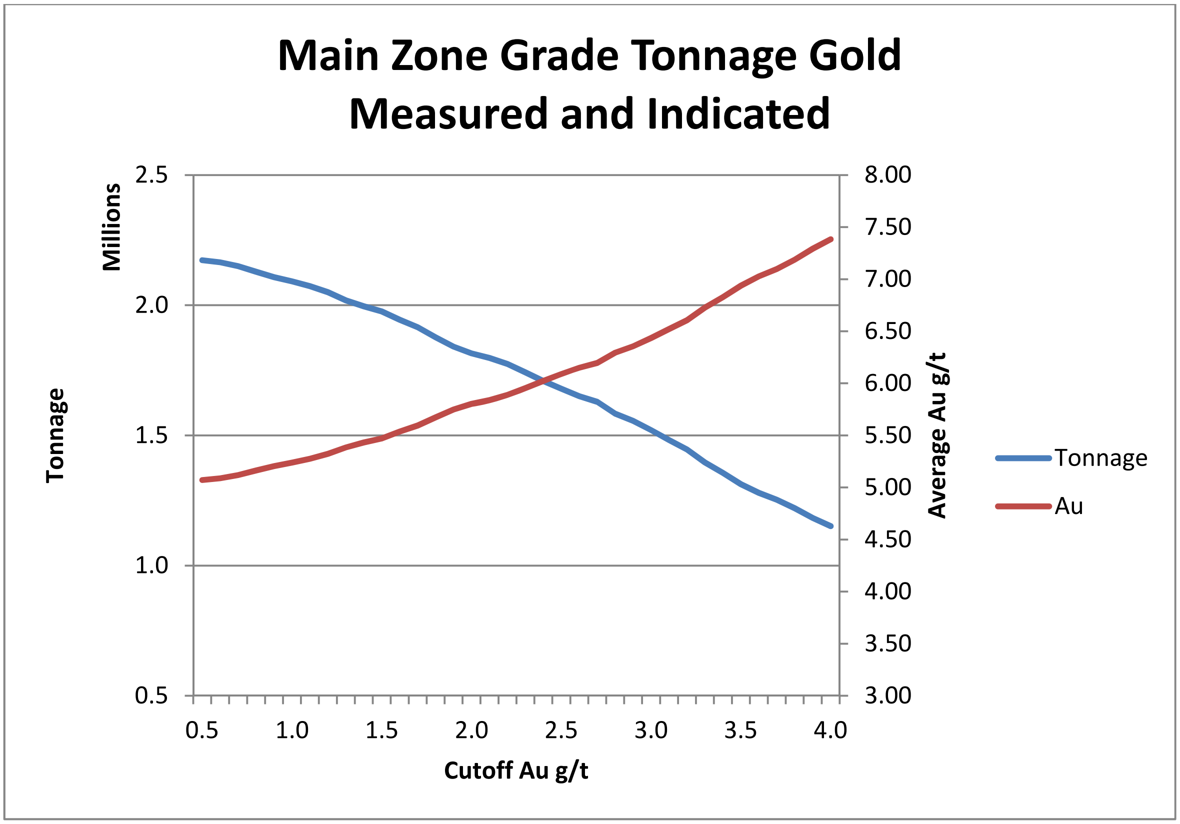
## Mineral Resource Sensitivity

Figure 2.6.2.4 presents grade tonnage curves for Measured and Indicated Resources combined and Inferred Resources separately.

Cutoff grades for the Kaymaz resource at various gold prices are shown in Table 2.6.2.8.

**Table 2.6.2.8: Kaymaz Cutoff Grades vs. Gold Price**

| Gold Price | Cutoff Grade |
|---|---|
| 1600 | 0.72 |
| 1550 | 0.74 |
| 1500 | 0.77 |
| 1450 | 0.79 |
| 1400 | 0.82 |
| 1350 | 0.85 |
| 1300 | 0.89 |
| 1250 | 0.92 |



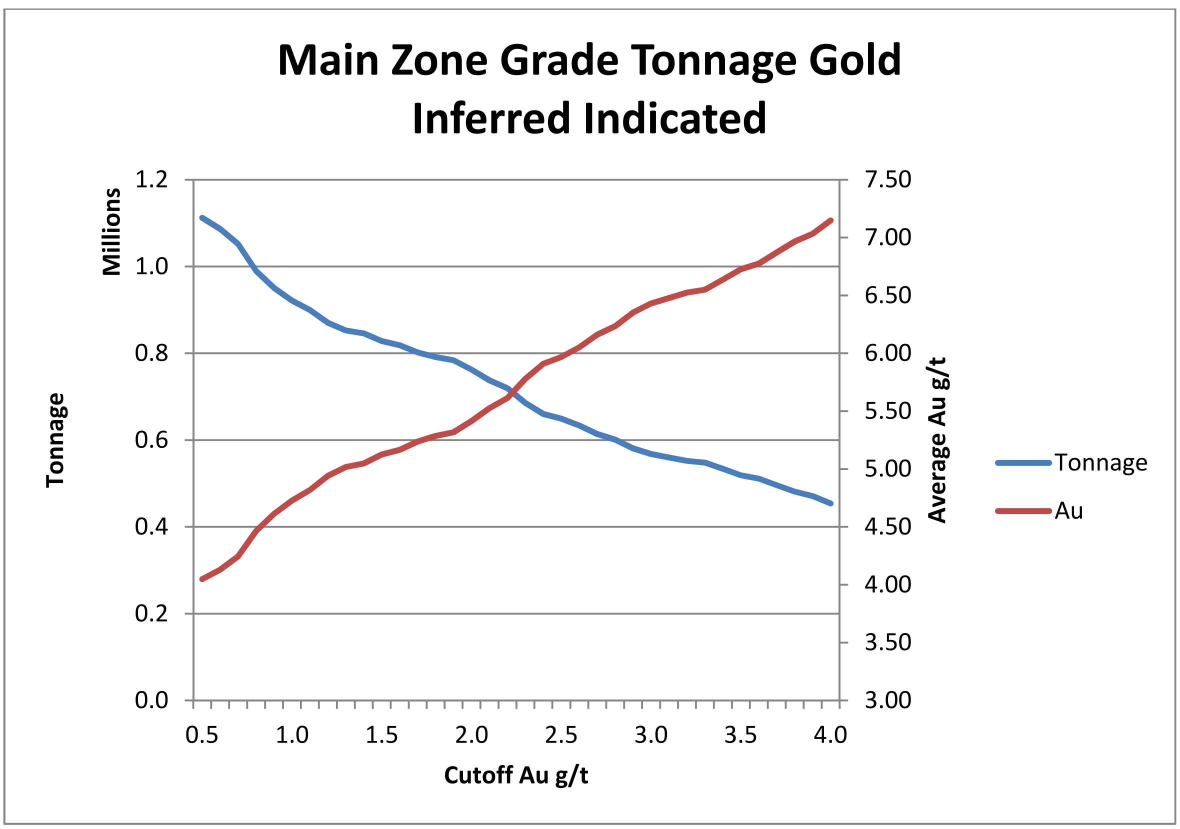




**Figure 2.6.2.4: Grade Tonnage Curves for Main Zone Resource**

### 2.6.3 Mermerlik

The Mermerlik resource was estimated by Koza in 2011 and has not been changed since then.

**Geologic Model and Assay Statistics**

Koza constructed 11 separate gold grade shell wireframes for Mermerlik, several of which contain a single drillhole. The wireframes were grouped into one domain. The wireframes have an east-west extent of 300 m, a north-south extent of 200 m and a vertical extent of 150 m. The wireframes dip to the east-northeast at about 45°. The thickness of the wireframes ranges from 1 to 25 m, with an average of about 10 m. SRK found that some of the wireframes are separated by very thin low grade zones and suggests that these wireframes should be combined into one (Figure 2.6.3.1). Also, two holes were drilled in 2012, which would limit the extent of the wireframes and therefore the tonnage of the resource and these should be incorporated into a revised resource model.

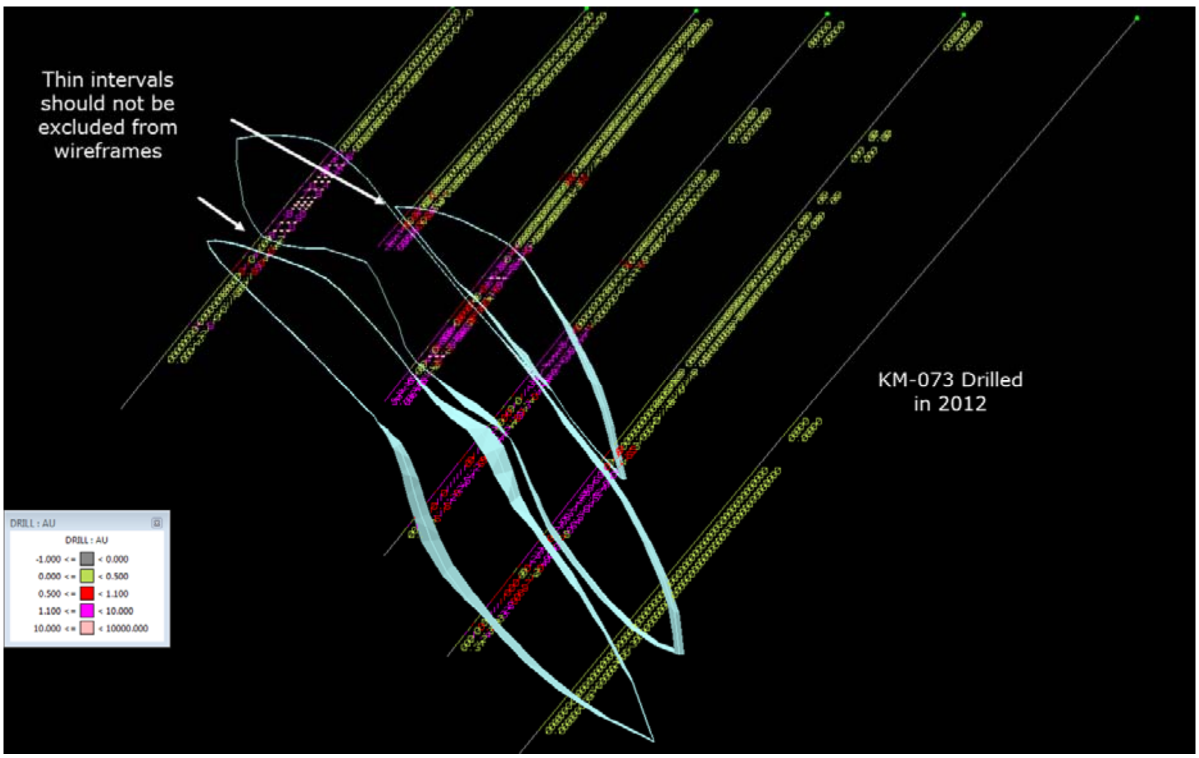


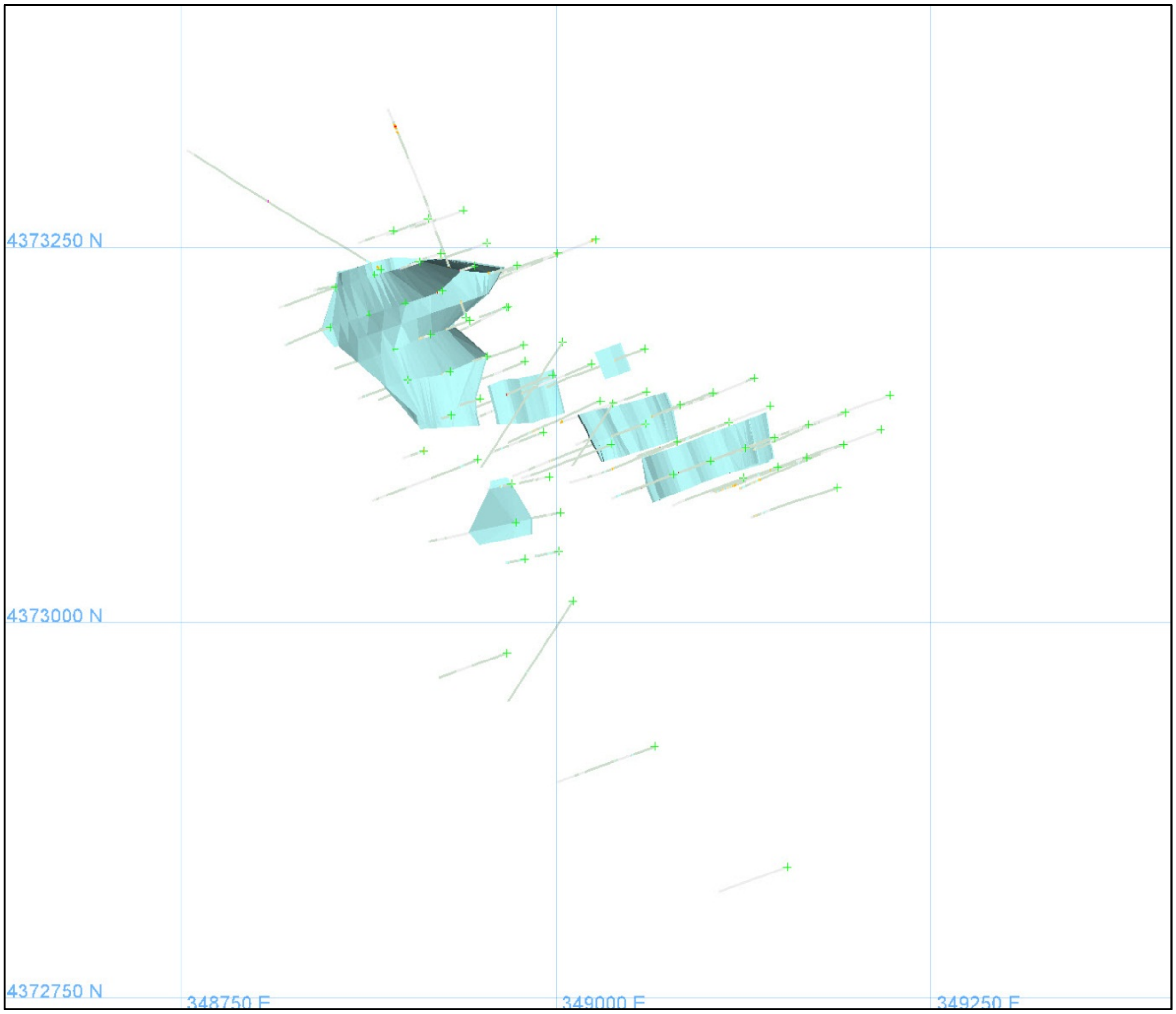

**Figure 2.6.3.1: Cross-Section Showing Wireframes at Mermerlik.**

Table 2.6.3.1 presents statistics of the raw drillhole and trench assays within the Main Zone. The drillholes, trenches and wireframes are shown in plan view and oblique view in Figures 2.6.3.2 and 2.6.3.3, respectively.

**Table 2.6.3.1: Statistics of Uncapped Assays at Mermerlik**

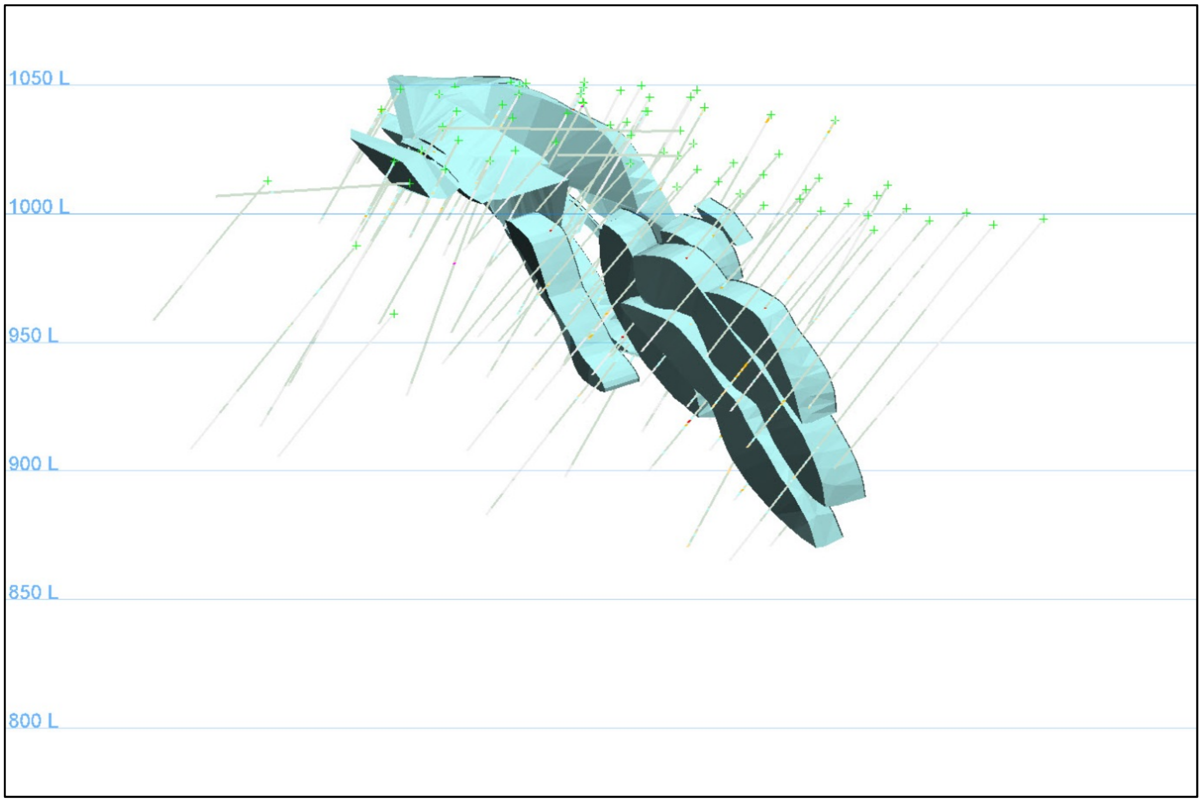
| Variable | Count | Min | Max | Mean | Std Dev | Skewness | CV |
|---|---|---|---|---|---|---|---|
| Au | 744 | 0.020 | 25.5 | 2.46 | 3.03 | 3.19 | 1.23 |
| Ag | 744 | 0.001 | 77.7 | 4.36 | 4.55 | 7.46 | 1.05 |

Source: SRK, 2012




Source: SRK, 2012

**Figure 2.6.3.2: Mermerlik Drillholes and Wireframes in Plan View**




Source: SRK, 2012

**Figure 2.6.3.3: Oblique View of Mermerlik Drillholes and Wireframes, Looking Northwest**

## Capping and Compositing

Koza reviewed the sample lengths to determine the compositing length. Approximately 95% of the samples are 1.5 m in length or less and Koza chose that length for compositing. The samples were composited within the wireframe with breaks at the wireframe boundaries using the distribution method as described in Section 2.6.2. SRK suggests that using a simple run length of 1.5 m would be produce more uniform lengths. Basic statistics of the composites are shown in Table 2.6.3.2.

**Table 2.6.3.2: Statistics of Uncapped Composites at Mermerlik**

| Variable | Count | Min | Max | Mean | Std Dev | Skewness | CV |
|---|---|---|---|---|---|---|---|
| Au | 481 | 0.062 | 21.91 | 2.47 | 2.72 | 2.84 | 1.10 |
| Ag | 481 | 0.001 | 55.20 | 4.31 | 4.19 | 6.16 | 0.97 |

Source: SRK, 2012

Koza performed a quantile analysis to determine an appropriate capping value. The gold composites were capped at 15 g/t and the silver composites were capped at 20 g/t. Table 2.6.3.3 contains the composites at Mermerlik after composites were capped.

**Table 2.6.3.3: Statistics of Capped Composites at Mermerlik**

| Variable | Count | Min | Max | Mean | S.D. | CV |
|---|---|---|---|---|---|---|
| Au | 481 | 0.062 | 15.00 | 2.45 | 2.60 | 1.07 |
| Ag | 481 | 0.001 | 20.00 | 4.16 | 3.09 | 0.74 |

Source: SRK, 2012

**Density**

A total of 159 HQ sized samples were collected covering a geographical range of Mermerlik. The samples were collected from 60 drillholes. Samples were grouped according to rock type, alteration and degree of breakage. Initial determinations using Archimedes method were made. Core was covered with wax to preserve pore space and the samples were weighed in water and air. The average density in the mineralized samples is 2.62 g/cm$^3$ which is used in the resource estimation. The density is on a dry tonnage basis.

**Grade Estimation**

The parent block size at Mermerlik is 5 m x 5 m x 5 m with sub-blocking allowed to 1.25 m in the Y and Z directions and 0.5 m in the X direction. The parent block size is about 25% of the drillhole spacing.

The Mermerlik Zone was estimated with ID2, ID3 and NN approaches in three passes as follows:

- First Pass: Minimum of 10 and maximum of 20 composites, with an octant search requiring a minimum of 1 and maximum of 4 composites within at least 2 octants. A maximum of 4 samples per drillhole were allowed in each estimation. Search ellipsoid with ranges of 60 m x 60 m x 10 m;
- Second Pass: Same as first with 120 m x 120 m x 20 m search; and
- Third Pass: Search ellipsoid of 120 m x 120 m x 20 m and minimum of 3 and maximum of 10 composites.

Only composites within the wireframes were used in the estimations.

**Block Model Validation**

Koza reviewed cross-sections visually on the computer screen to compare composite and block grades. A comparison of the composites to the estimated grades is shown in Table 2.6.3.4. The three estimation types produced results that are very similar to each other, but all are higher than the composite grade. This may be due to uneven drillhole spacing per volume of resource. SRK suggests that Koza also produce swath plots as a method of block validation.

**Table 2.6.3.4: Main Zone Comparison of Composites and Estimated Grades**

| Variable | Composites | ID2 | ID3 | NN |
|---|---|---|---|---|
| Au | 2.45 | 2.50 | 2.51 | 2.51 |
| Ag | 4.16 | 4.34 | 4.35 | 4.34 |

Source: SRK, 2012

**Mineral Resource Classification**

The blocks were classified as follows:

- Measured: First estimation pass and number of drillhole greater than 3;
- Indicated: First estimation pass and number of drillhole equal to 2 or 3; and
- Inferred: Remaining blocks.

**Mineral Resource Statement**

The cutoff grade of 0.80 g/t Au is based on the open pit mining assumptions shown in Table 2.6.2.6. The one year rolling average gold price is US$1,266; the two year average is US$1,339; and the three year average is US$1,449.

**Table 2.6.3.5: Kaymaz Cutoff Grade Assumptions**

| Item | Units | Prices and Costs |
|---|---|---|
| Gold Price | US$/oz | 1,450 |
| Gold Recovery | % | 82 |
| Gold Refining | US$/oz | 3.44 |
| Royalty | % | 0 |
| Government Right | % | 1 |
| Process Cost | US$/t | 22.00 |
| Mining Cost | US$/t | 0.00 |
| G&A Cost | US$/t | 8.00 |
| Calculated Cutoff grade | g/t | 0.77 |
| Final Cutoff grade | g/t | 0.80 |

Source: Koza, 2014

It is becoming an industry practice to state mineral resources within a pit optimization shell. Koza conducted a pit optimization using the parameters in Table 2.6.3.5. Approximately 95% of the Measured and Indicated resources and 78% of the Inferred resources fall within the pit shell.

The Measured, Indicated, and Inferred resources at a cutoff grade of 0.80 g/t Au are listed in Table 2.6.3.6, the tonnage is inclusive of ore reserves.

**Table 2.6.3.6: Mermerlik Mineral Resources, Including Ore Reserves, at December 31, 2014**

| Classification | kt | g/t Au | g/t Ag | koz Au | koz Ag |
|---|---|---|---|---|---|
| Measured | 435 | 2.01 | 3.36 | 28 | 47 |
| Indicated | 413 | 3.06 | 5.20 | 41 | 69 |
| Measured and Indicated | 848 | 2.52 | 4.26 | 69 | 116 |
| Inferred | 179 | 2.52 | 4.87 | 14 | 28 |

Tonnages and grade are rounded to reflect approximation;
Resources are stated at a cutoff grade of 0.80 g/t;
Open pit resources are contained within grade shells but are not constrained by a pit optimization shell; and
Mineral Resources are reported inclusive of Mineral Reserves.

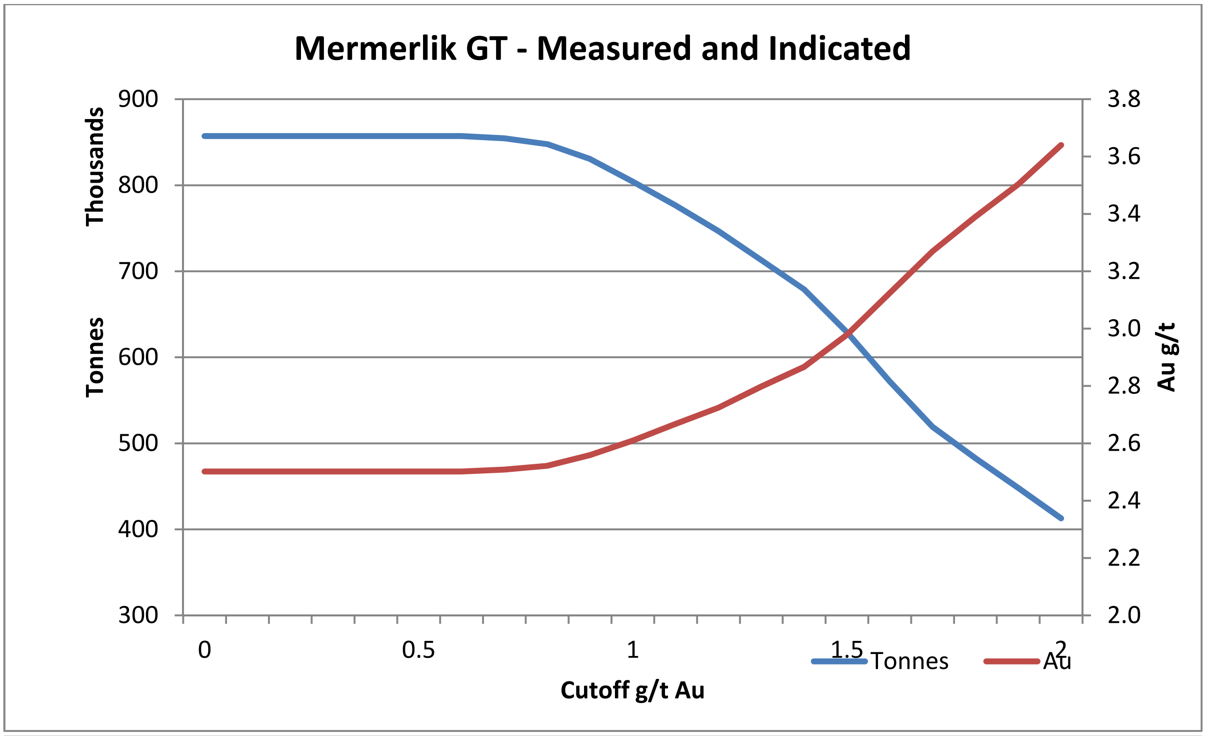
**Mineral Resource Sensitivity**

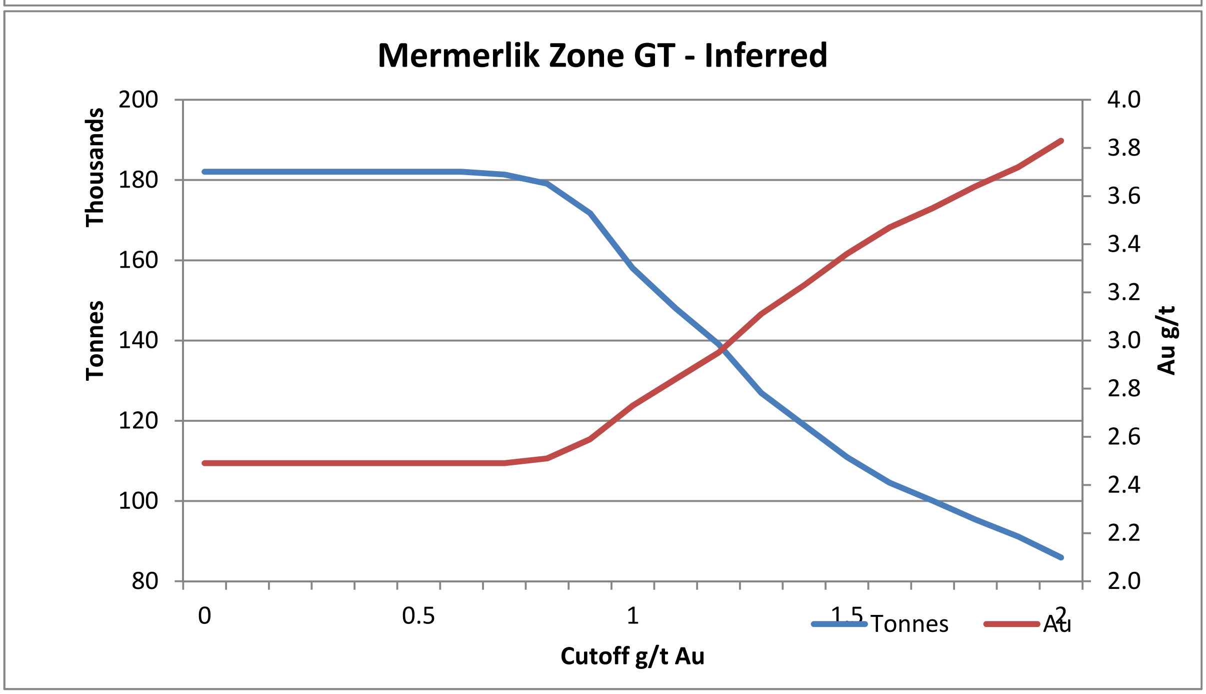
Figure 2.6.3.4 presents grade tonnage curves for Measured and Indicated Resources combined and Inferred Resources separately.

Cutoff grades for the Kaymaz resource at various gold prices are shown in Table 2.6.3.7.

**Table 2.6.3.7: Kaymaz Cutoff Grades vs. Gold Price**

| Gold Price | Cutoff Grade |
|---|---|
| 1600 | 0.72 |
| 1550 | 0.74 |
| 1500 | 0.77 |
| 1450 | 0.79 |
| 1400 | 0.82 |
| 1350 | 0.85 |
| 1300 | 0.89 |
| 1250 | 0.92 |




Source: SRK, 2012

**Figure 2.6.3.4: Grade Tonnage Curves for Mermerlik Resource**

### 2.6.4 Kizilagil

The Kizilagil resource was estimated by Koza in 2012 and has not changed since then.

**Geologic Model and Assay Statistics**

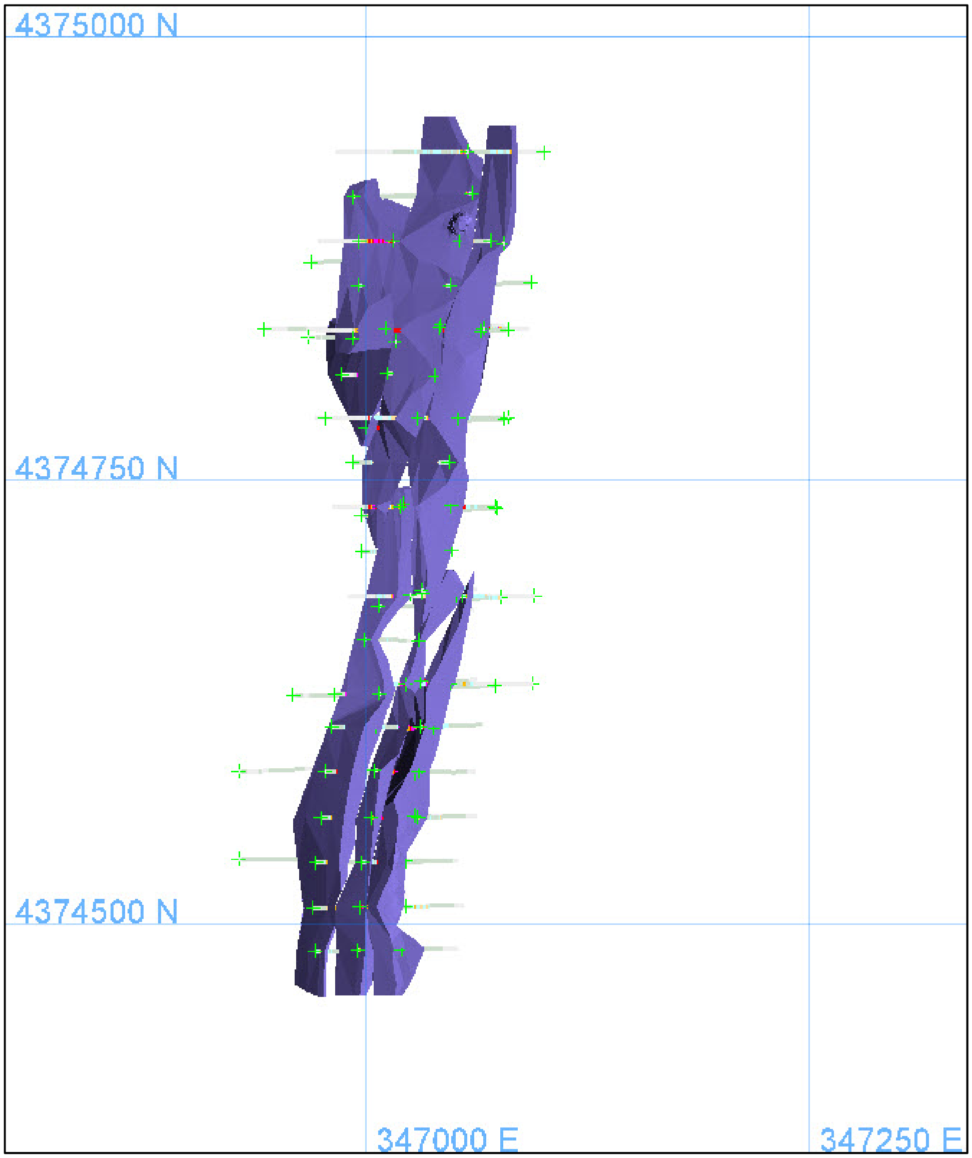
Koza constructed seven separate gold grade shell wireframes for the Kizilagil zone. The wireframes were grouped together in one domain for estimation. The wireframes have an east-west extent of 100 m, a north-south extent of 500 m and a vertical extent of 25 m. The wireframes dip to the west at about 15°. The thickness of the wireframes ranges from 1 to 20 m, with an average of about 8 to 10 m.

Table 2.6.4.1 presents statistics of the raw assays in the Kizilagil zone.

**Table 2.6.4.1: Statistics of Uncapped Assays at Kizilagil**

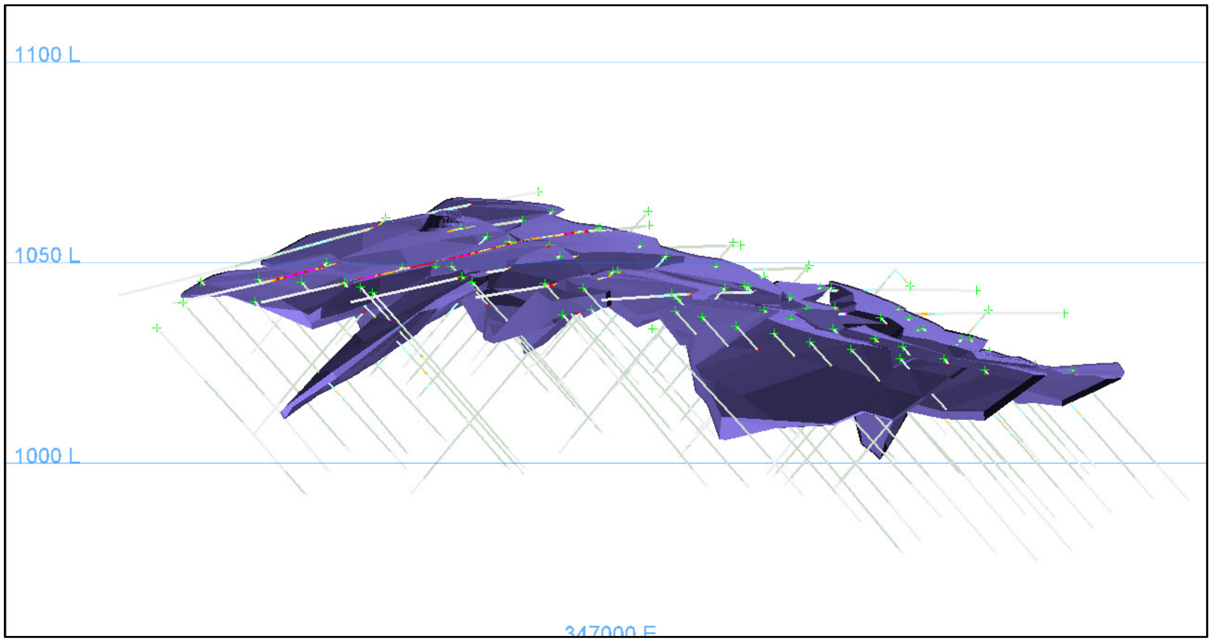
| Zone | Variable | Count | Min | Max | Mean | S.D. | Skewness | CV |
|---|---|---|---|---|---|---|---|---|
| 1 | Au | 911 | 0.001 | 69.62 | 3.07 | 4.67 | 5.89 | 1.52 |
| | Ag | 911 | 0.001 | 136.00 | 5.45 | 7.91 | 7.71 | 1.45 |

Source: SRK, 2012




Source: SRK, 2012

**Figure 2.6.4.1: Kizilagil Drillholes and Wireframes in Plan View**




Source: SRK, 2012

**Figure 2.6.4.2: Oblique View of Kizilagil Drillholes and Wireframes, Looking North Northeast**

## Capping and Compositing

Koza reviewed the sample lengths to determine the compositing length. About 90% of the samples are 1 m in length or less and therefore 1 m was chosen as the composite length. The samples were composited on 1.0 m lengths within the wireframe with breaks at the wireframe boundary. Statistics of the composites are shown in Table 2.6.4.2.

**Table 2.6.4.2: Statistics of Uncapped Composites at Kizilagil**

| Variable | Count | Min | Max | Mean | Std Dev | Skewness | CV |
|---|---|---|---|---|---|---|---|
| Au | 863 | 0.001 | 48.88 | 3.03 | 4.08 | 4.46 | 1.35 |
| Ag | 863 | 0.001 | 131.12 | 5.62 | 8.20 | 7.23 | 1.46 |

Source: SRK, 2012

Koza performed a quantile analysis of the composites to determine the capping value for the composites. The composites were capped at 18 g/t for gold and 40 g/t for silver. Table 2.6.4.3 contains the basic statistics for the capped composites at Kizilagil.

**Table 2.6.4.3: Statistics of Capped Composites at Kizilagil**

| Variable | Count | Min | Max | Mean | Std Dev | Skewness | CV |
|---|---|---|---|---|---|---|---|
| Au | 863 | 0.001 | 18.00 | 2.92 | 3.34 | 5.89 | 1.14 |
| Ag | 863 | 0.001 | 40.00 | 5.35 | 5.93 | 7.71 | 1.11 |

Source: SRK, 2012

**Density**

Specific gravity measurements of Kizilagil Project were completed in August 2011. A total of 61 HQ sized samples were collected covering a geographical range of Kizilagil. The samples were collected from 26 drillholes. Samples were grouped according to rock type, alteration and degree of breakage. Initial determinations using Archimedes method were made. Core was covered with wax to preserve pore space and the samples were weighed in water and air. The average specific density in the mineralized samples is 2.57 g/cm$^3$. The density is on a dry tonnage basis.

**Grade Estimation**

The parent block size at Kizilagil is 5 m x 5 m x 5 m. Sub-blocking to 2.5 m in the X and Y directions and 0.5 m in the Z direction are allowed to provide best correlation to the ore/waste boundary. The parent block size is about 25% of the drill spacing.

The Kizilagil Zone was estimated with ID2, ID3 and NN approaches in three passes as follows:

- First: Minimum of 8 and maximum of 20 composites, with an octant search requiring a minimum of 1 and maximum of 4 composites within at least 2 octants. Search ellipsoid with ranges of 50 m x 50 m x 10 m;
- Second: Same as first with 100 m x 100 m x 20 m search; and
- Third: Search ellipsoid of 150 m x 150 m x 150 m and minimum of 2 and maximum of 12 composites.

Only composites within the wireframes were used in the estimations. A comparison of composite and estimated grades is shown in Table 2.6.4.4. The three estimated gold grades are similar to the composite grades. The estimated silver grades are quite close to each other, but less than the composite grade.

**Table 2.6.4.4: Comparison of Composites and Estimated Grades at Kizilagil**

| Variable | Composite | ID2 | ID3 | NN |
|---|---:|---:|---:|---:|
| Au | 2.92 | 2.83 | 2.82 | 2.70 |
| Ag | 5.35 | 4.69 | 4.67 | 4.39 |

Source: SRK, 2012

**Mineral Resource Classification**

The blocks were categorized as follows:

- Indicated: First estimation pass and more than 3 drillholes; and
- Inferred: Remaining blocks.

**Mineral Resource Statement**

The cutoff grade of 0.80 g/t Au is based on the assumptions shown in Table 2.6.1.6. The one year rolling average gold price is US$1,266; the two year average is US$1,339; and the three year average is US$1,449.

**Table 2.6.4.5: Kaymaz Cutoff Grade Assumptions**

| Item | Units | Prices and Costs |
|---|---|---|
| Gold Price | US$/oz | 1,450 |
| Gold Recovery | % | 82 |
| Gold Refining | US$/oz | 3.44 |
| Royalty | % | 0 |
| Government Right | % | 1 |
| Process Cost | US$/t | 22.00 |
| Mining Cost | US$/t | 0.00 |
| G&A Cost | US$/t | 8.00 |
| Final Cutoff grade | g/t | 0.80 |

Source: Koza, 2014

It is becoming an industry practice to state mineral resources within a pit optimization shell. Koza conducted a pit optimization using the parameters in Table 2.6.4.5. Approximately 99% of the Indicated resources and 92% of the Inferred resources fall within the pit shell.

The Measured, Indicated, and Inferred resources at a cutoff grade of 0.80 g/t Au are listed in Table 2.6.4.6, the tonnage is inclusive of ore reserves.

**Table 2.6.4.6: Kizilagil Mineral Resources, Including Ore Reserves, at December 31, 2014**

| Classification | kt | g/t Au | g/t Ag | koz Au | koz Ag |
|---|---|---|---|---|---|
| Indicated | 432 | 2.88 | 4.76 | 40 | 66 |
| Inferred | 22 | 2.00 | 3.24 | 1 | 2 |

Tonnages and grade are rounded to reflect approximation;
Resources are stated at a cutoff grade of 0.80 g/t;
Open pit resources are contained within grade shells but are not constrained by a pit optimization shell; and
Mineral Resources are reported inclusive of Mineral Reserves.

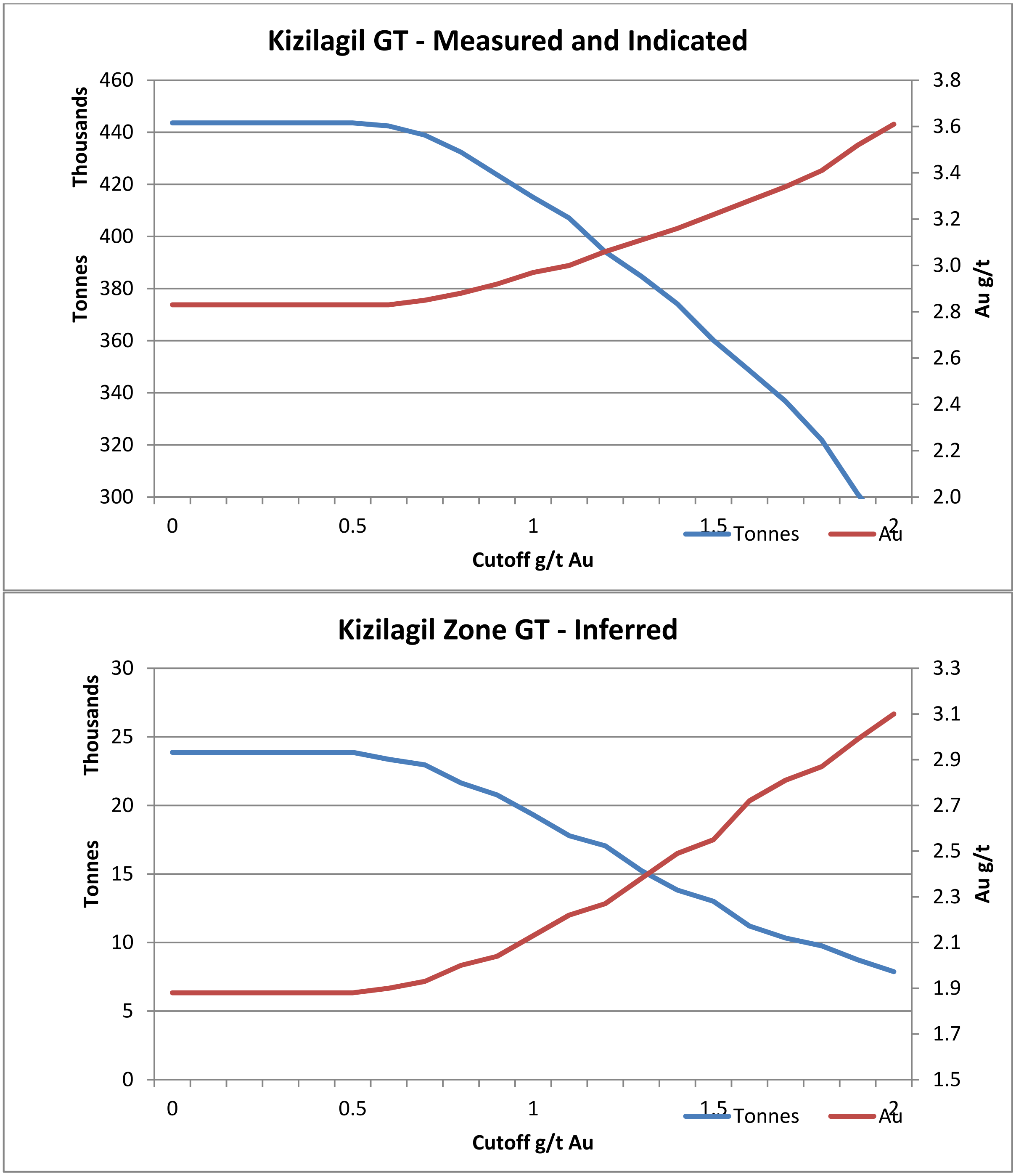
## Mineral Resource Sensitivity

Figure 2.6.4.3 presents grade tonnage curves for Measured and Indicated Resources combined and Inferred Resources separately.

Cutoff grades for the Kaymaz resource at various gold prices are shown in Table 2.6.4.7.

**Table 2.6.4.7: Kaymaz Cutoff Grades vs. Gold Price**

| Gold Price | Cutoff Grade |
|---|---|
| 1600 | 0.72 |
| 1550 | 0.74 |
| 1500 | 0.77 |
| 1450 | 0.79 |
| 1400 | 0.82 |
| 1350 | 0.85 |
| 1300 | 0.89 |
| 1250 | 0.92 |




Source: SRK, 2012

**Figure 2.6.4.3: Grade Tonnage Curves for Kizilagil Resource**

## 2.6.5 Kaymaz Combined Mineral Resource Statement

The cutoff grades were calculated from the costs shown in Table 2.6.5.1 based on an open pit operation with a mill at Kaymaz. The one year rolling average gold price is US$1,266; the two year average is US$1,339; and the three year average is US$1,449.

It is becoming an industry standard to report resource within pit optimization shells to meet JORC requirements that mineral resources be potentially mineable. Material that falls outside the pit shells may be stated at a cutoff grade that represents potential underground mining costs. Pit optimizations were run on the Kaymaz resources and the resulting pit shells at Damdamca and Kizilagil contain more than 90% of the total resource. There are blocks in the Main and Mermerlik zones that are below the pit and may not be mineable by open pit methods and may not have sufficient grade for underground mining. The resources reported in this report are not constrained by a pit optimization shell.

**Table 2.6.5.1: Kaymaz Cutoff Grade Parameters**

| Item | Units | Prices and Costs |
|---|---|---|
| Gold Price | US$/oz | 1,450 |
| Gold Recovery | % | 82 |
| Gold Refining | US$/oz | 3.44 |
| Royalty | % | 0 |
| Government Right | % | 1 |
| Process Cost | US$/t | 22.00 |
| Mining Cost | US$/t | 0.00 |
| G&A Cost | US$/t | 8.00 |
| Cutoff grade | g/t | 080 |

Source: Koza, 2014

Table 2.6.5.2 lists the open pit resources at a cutoff grade of 0.80 g/t Au. The Mineral Resources are inclusive of Ore Reserves.

**Table 2.6.5.2: Kaymaz Mineral Resources, Inclusive of Ore Reserves, at December 31, 2014**

| Classification | kt | g/t Au | g/t Ag | koz Au | koz Ag |
|---|---|---|---|---|---|
| **Main** | | | | | |
| Measured | 640 | 4.39 | 5.57 | 90 | 115 |
| Indicated | 1,489 | 5.49 | 5.46 | 263 | 261 |
| **Measured and Indicated** | **2,129** | **5.16** | **5.49** | **353** | **375** |
| Inferred | 990 | 4.47 | 5.02 | 142 | 159 |
| **Mermerlik** | | | | | |
| Measured | 435 | 2.01 | 3.36 | 28 | 47 |
| Indicated | 413 | 3.06 | 5.20 | 41 | 69 |
| **Measured and Indicated** | **848** | **2.52** | **4.26** | **69** | **116** |
| Inferred | 179 | 2.52 | 4.87 | 15 | 28 |
| **Kizilagil** | | | | | |
| Measured | 0.00 | | | | |
| Indicated | 432 | 2.88 | 4.76 | 40 | 66 |
| **Measured and Indicated** | **432** | **2.88** | **4.76** | **40** | **67** |
| Inferred | 22 | 2.00 | 3.24 | 1 | 2 |
| **Total** | | | | | |
| Measured | 1,074 | 3.43 | 4.68 | 118 | 162 |
| Indicated | 2,335 | 4.58 | 5.28 | 344 | 397 |
| **Measured and Indicated** | 3,409 | 4.22 | 5.09 | 462 | 558 |
| Inferred | 1,190 | 4.13 | 4.96 | 158 | 190 |

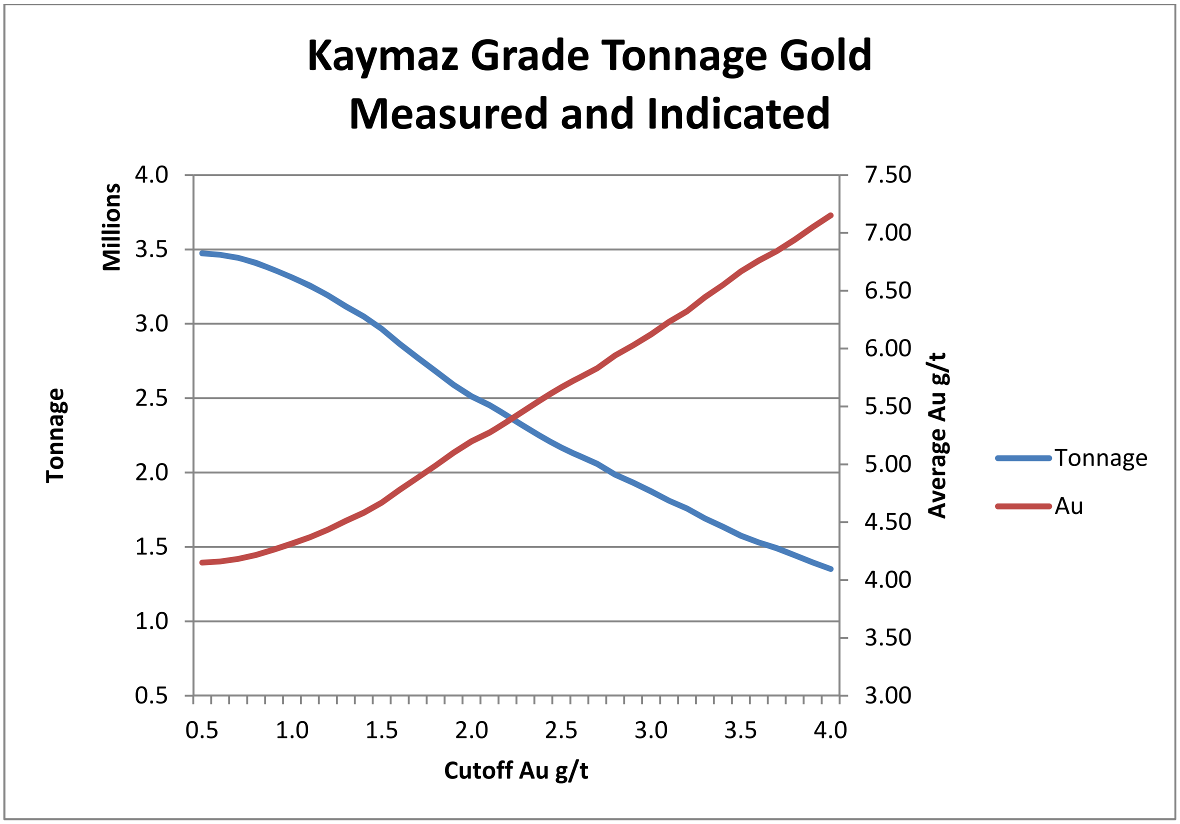
### 2.6.6 Mineral Resource Sensitivity

Grade tonnage curves for the Measured and Indicated Resource and also for the Inferred Resource are shown in Figure 2.6.6.1.

Cutoff grades for the Kaymaz resource at various gold prices are shown in Table 2.6.6.1.

**Table 2.6.6.1: Kaymaz Cutoff Grades vs. Gold Price**

| Gold Price | Cutoff Grade |
|---|---:|
| 1600 | 0.72 |
| 1550 | 0.74 |
| 1500 | 0.77 |
| 1450 | 0.79 |
| 1400 | 0.82 |
| 1350 | 0.85 |
| 1300 | 0.89 |
| 1250 | 0.92 |
| 1200 | 0.72 |



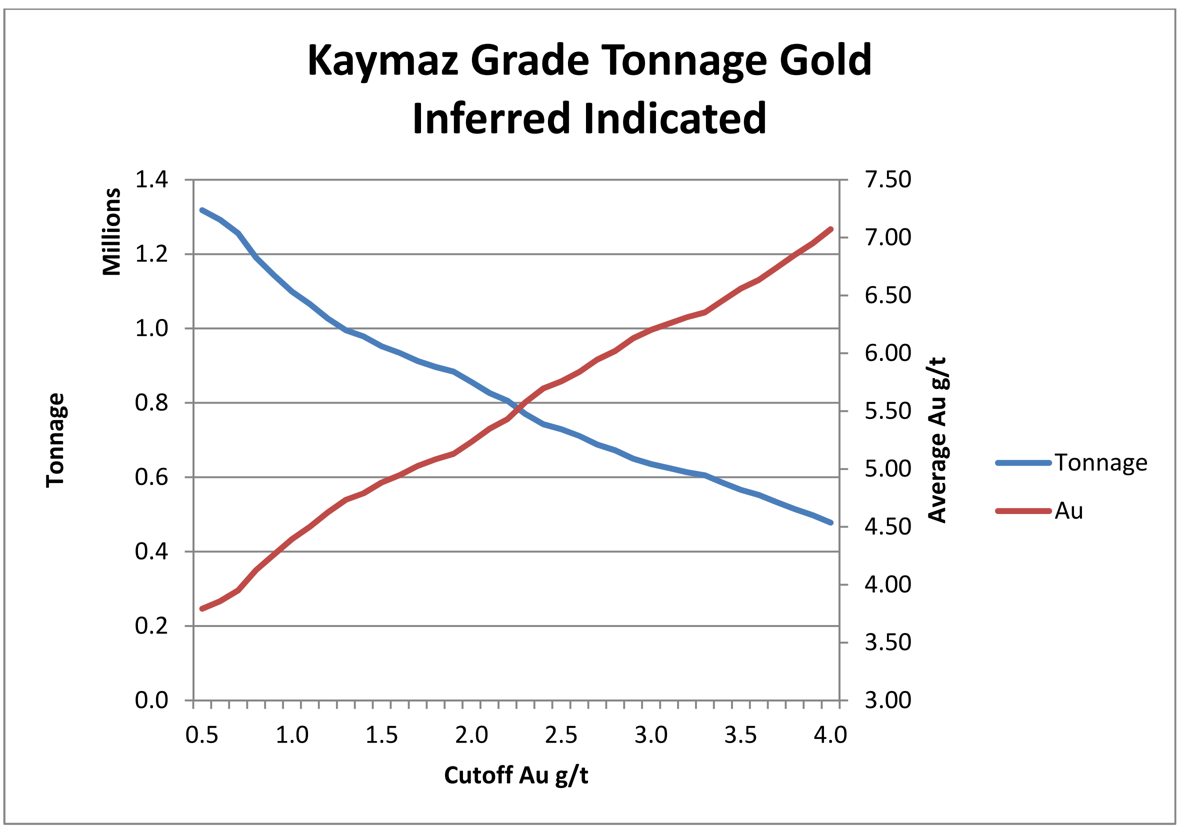




**Figure 2.6.6.1: Grade Tonnage Curves for Kaymaz Resource**

## 2.7 Kaymaz Mine Production

The Kaymaz Project is separated into four separate areas: Damdamca, Kizilagil, Main and Mermerlik. The Damdamca orebody is approximately 3.5 km to the northwest of the Main zone with the Mermerlik orebody approximately 1.5 km to the south.

2014 Mine production was primarily sourced from the Damdamca open pit mining as permitting issues prevented the continuation of pre-stripping activities that had commenced at the Main zone. As no permit was available in 2014 for the Main zone, the only source of ore was from the Damdamca pits and stockpiles previously located on site.

The mining method is a replica of that which is performed at all other Koza open pits in terms of operational procedures, equipment fleets and general operating costs.

The site does not contain any serious topographical limitations to haul routes, waste disposal or pit design, as the terrain is gently undulating and consists of open fields. No major watersheds are present.

Table 2.7.1 shows the reconciliation between the 2014 mine production and the end of 2013 technical economic model (TEM). Because main zone production was suspended the only comparable production was for the Damdamca pit through April 2014 when the pit was mined out.

Kaymaz has performed within expectation for 2013 with a gold grade 3% higher than predicted and total mined ounces 27% over the production schedule estimated. While the grade variance is good from a financial perspective (as a bonus), reconciliation on the whole pit should be undertaken when Damdamca finishes. The estimation parameters can then be adjusted so what was predicted can match what was produced at Damdamca. The goal of the exercise is to use the lessons learnt from resource estimation at Damdamca and apply them to the Main Zone which is of a similar mineralization style.

**Table 2.7.1: 2014 Kaymaz Mine Production**

| 2014 Production | Kaymaz OP RoM Production | | | | Ovacik 2013 TEM | | | | Reconciliation (Predicted vs. Achieved) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Ore Tonne | Au g/t | Ag g/t | Gold Ounces | Ore Tonne | Au g/t | Ag g/t | Au Ounces | Tonnage | Au Grade | Au Ounce |
| January | 75,753 | 4.04 | 3.20 | 9,844 | 75,711 | 4.48 | 2.86 | 10,905 | 0% | 11% | 11% |
| February | 70,452 | 3.82 | 2.81 | 8,651 | 47,485 | 5.04 | 3.20 | 7,694 | -33% | 32% | -11% |
| March | 75,536 | 5.79 | 3.65 | 14,059 | 42,164 | 6.52 | 7.34 | 8,839 | -44% | 13% | -37% |
| April | 21,831 | 9.56 | 5.28 | 6,710 | 17,486 | 2.02 | 3.96 | 1,136 | -20% | -79% | -83% |
| **Total** | **243,572** | **5.01** | **3.41** | **39,264** | **182,846** | **4.86** | **4.09** | **28,574** | **-25%** | **-3%** | **-27%** |

Source: Koza, 2014

## 2.8 Ore Reserve Estimation

LoM plans and resulting reserves are determined based on a gold price of US$1,250/Oz for the Kaymaz project. Reserves stated in this report are as of December 30, 2014 with an exchange rate of 2 Turkish liras to the U.S. dollar.

The ore material is converted from resource to reserve based primarily on positive cash flow pit optimization results, pit design and geological classification of Measured and Indicated resources. The in-situ value is derived from the estimated grade and certain modifying factors. The previous sections discuss the procedures used to estimate gold grade.

### 2.8.1 Modifying Factors

The conversion of resource to reserve entails the evaluation of modifying factors that should be considered in stating a reserve. Table 2.8.1.1 illustrates a reserve checklist and associated commentary on the risk factors involved for the Kaymaz reserve statement.

**Table 2.8.1.1: Kaymaz SRK Reserve Checklist**

| Unit | Data Evaluated | Data Not Evaluated | Not Applicable | Notes |
|---|---|---|---|---|
| **Mining** | | | | |
| Mining Width | X | | | Small mining trucks |
| Open Pit and/or Underground | X | | | Open Pit |
| Density and Bulk handling | X | | | Operating Mine |
| Dilution | X | | | No dilution added |
| Mine Recovery | X | | | Full mine recovery assumed |
| Waste Rock | X | | | Main zone fully designed |
| Grade Control | X | | | Channel sample |
| **Processing** | | | | |
| Representative Sample | X | | | Operating Mine |
| Product Recoveries | X | | | Operating Mine |
| Hardness (Grindability) | X | | | Operating Mine |
| Bulk Density | X | | | Operating Mine |
| Deleterious Elements | X | | | Clay, high nickel stockpiled |
| Process Selection | X | | | CIL |
| **Geotechnical/Hydrological** | | | | |
| Slope Stability (Open Pit) | X | | | Lessons From Damdamca |
| Water Balance | X | | | Water supply |
| Area Hydrology | X | | | Dry climate |
| Seismic Risk | | X | | Assume no limiting factor to mining |
| **Environmental** | | | | |
| Baseline Studies | X | | | Operating Mine |
| Tailing Management | | X | | |
| Waste Rock Management | X | | | Design close to pit rims. |
| Acid Rock Drainage Issues | X | | | EIA |
| Closure and Reclamation Plan | X | | | Project still developing EIA |
| Permitting Schedule | X | | | Operating permits in place |
| **Location and Infrastructure** | | | | |
| Climate | X | | | Dry climate |
| Supply Logistics | X | | | Close to highway |
| Power Source(S) | X | | | Readily available |
| Existing Infrastructure | X | | | Excellent |
| Labor Supply and Skill Level | X | | | Excellent |
| **Marketing Elements or Factors** | | | | |
| Product Specification and Demand | X | | | Gold Market |
| Off-site Treatment Terms and Costs | X | | | Gold Market |
| Transportation Costs | X | | | Gold Market |
| **Legal Elements or Factors** | | | | |
| Security of Tenure | X | | | Operating Mine |
| Ownership Rights and Interests | X | | | Operating Mine |
| Environmental Liability | X | | | Assume no limiting factor reserve - Operating Mine |
| Political Risk (e.g., land claims, sovereign risk) | X | | | Some conflict with government and mine owners – permits take time to be granted. |
| Negotiated Fiscal Regime | X | | | Operating Mine |
| **General Costs and Revenue Elements or Factors** | | | | |
| General and Administrative Costs | X | | | |
| Commodity Price Forecasts | X | | | |
| Foreign Exchange Forecasts | | | X | |
| Inflation | | | X | |
| Royalty Commitments | X | | | |
| Taxes | X | | | |
| Corporative Investment Criteria | X | | | |
| **Social Issues** | | | | |
| Sustainable Development Strategy | X | | | Koza Environmental/Social – Operating Mine |
| Impact Assessment and Mitigation | X | | | Koza Environmental/Social – Operating Mine |
| Negotiated Cost/Benefit Agreement | | X | | Assume no limiting factor to mining |
| Cultural and Social Influences | X | | | Koza Environmental/Social – Operating Mine |

Source: SRK, 2014

Pit optimization inputs for the Kaymaz deposits are given in Table 2.8.1.2

**Table 2.8.1.2: 2014 Kaymaz Pit Optimization - Base Inputs**

| Parameter | Unit | Amount |
|---|---|---|
| Mining Cost | US$/t | 1.19 |
| Rehabilitation Cost | US$/t waste | 0.20 |
| Milling Cost | US$t/ore | 20.00 |
| Selling Cost | US$/oz | 3.44 |
| Grade Control | US$t/ore | 0.50 |
| Administration | US$t/ore | 9.0 |
| Ore Rehandle | US$t/ore | 0.50 |
| Gold Price | US$/oz | 1,250 |
| Silver Price | US$/oz | 20 |
| Gold Recovery | % | 87 |
| Silver Recovery | % | 75 |
| Royalty | % Revenue | 1 |
| Cutoff grade | g/t Au | 0.87 |

Source: Koza, 2014
Recovery of Mermerlik and Kizilagil were estimated at 88%

A royalty of 1% of gross profit is payable to the Turkish Government.

### 2.8.2 Reserve Statement

Ore tonnes which lie within the final pit design shape are classified as Proven or Probable reserves based on the geological classification for Measured and Indicated resources. Proven reserves are Measured resources within the design pit shape and Probable reserves are Indicated resources within the design pit shape. Inferred material, which lies within the pit design, is not included in the reserve statement and is treated as waste in the economic model.

Stockpiles available for processing are considered Proven if they achieve a RoM grade and Probable if Au grade nears the calculated cutoff grade. For low-grade, the removal of administration and grade control costs lower the break-even cutoff grade making processing profitable at the end of mine life as the material movement is considered a sunk cost. The open pit and stockpile reserves are listed in Tables 2.8.2.1 and 2.8.2.2; the emergency stockpile was processed during 2014.

**Table 2.8.2.1: Kaymaz Open Pit Mineral Reserves at December 31, 2014**

| Category | kt | g/t Au | g/t Ag | koz Au | koz-Ag |
|---|---|---|---|---|---|
| Proven Reserve | 935 | 3.69 | 4.9 | 111 | 148 |
| Probable Reserve | 2,037 | 5.04 | 5.5 | 330 | 360 |
| **Total Proven and Probable Reserves** | **2,972** | **4.62** | **5.3** | **441** | **508** |

Source: Koza, 2014
Metal Price: US$1,250/oz-Au, US$20/oz-Ag, Au Recovery 87% and 88%, Ag Recovery 75%, Au cutoff grade 0.87g/t

**Table 2.8.2.2: Kaymaz LG Stockpile Reserve, at December 31, 2014**

| Category | kt | g/t Au | g/t Ag | Contained koz Au | Contained koz Ag |
|---|---|---|---|---|---|
| Probable Reserve | 67 | 0.93 | 2.7 | 2 | 6 |
| **Total Proven and Probable Reserves** | **67** | **0.93** | **2.7** | **2** | **6** |

Source: Koza, 2014
Metal Price: US$1,250/oz-Au, US$20/oz-Ag, Au Recovery 87% and 88%, Ag Recovery 75%, no cutoff as already mined

## 2.9 Mining

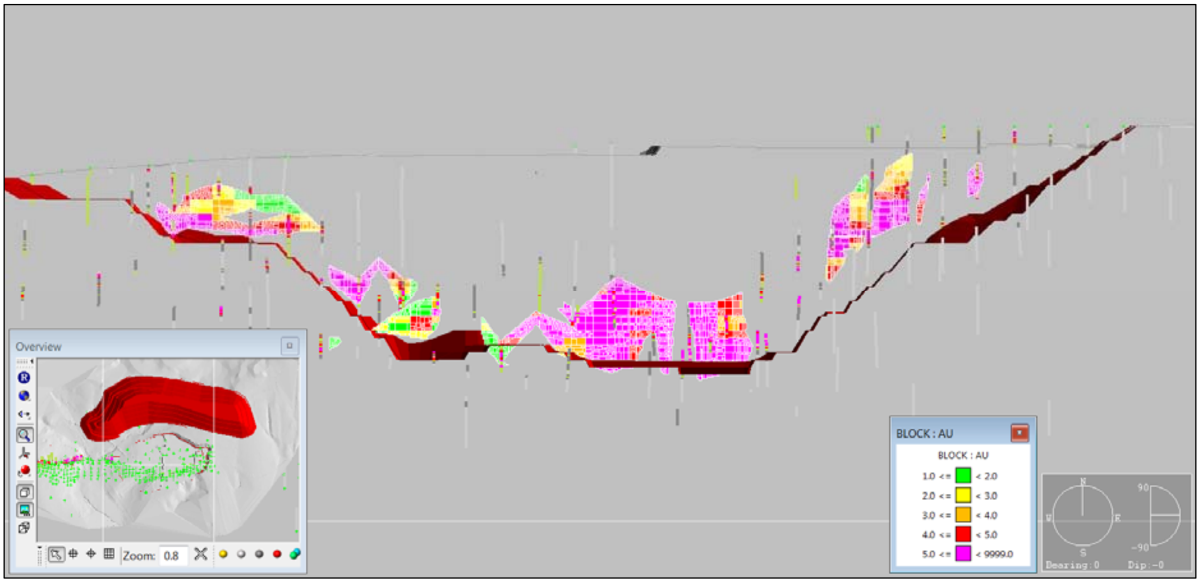
Mine operations are similar to other Koza operations with contractors used for open pit excavation, waste disposal and ancillary operations. Koza act as mine owners and grade control technicians.

Mining of the Damdamca open pit was finished in April 2014, thus successfully completing three years of operations that began in 2011. The contractor employed by Koza, Koseoglu Mining, is new to Kaymaz but operations appear well run and the ability to selectively mine in a productive fashion has continued at Kaymaz.

No cost escalation has been applied to the transportation of Damdamca ore over the Main or Mermerlik zones. The costs will only be incremental as the distances of the satellite ores are within 3.5 km and when a detailed design and schedule has been applied contractor negotiations will ensue, but typically a 0.15 c/t km haul cost is applied. Figure 2.9.1 shows the final Damdamca pit. Figures 2.9.2 through 2.9.5 illustrate cross sections of the pit designs and block model for Main, Mermerlik, and Kizilagil.

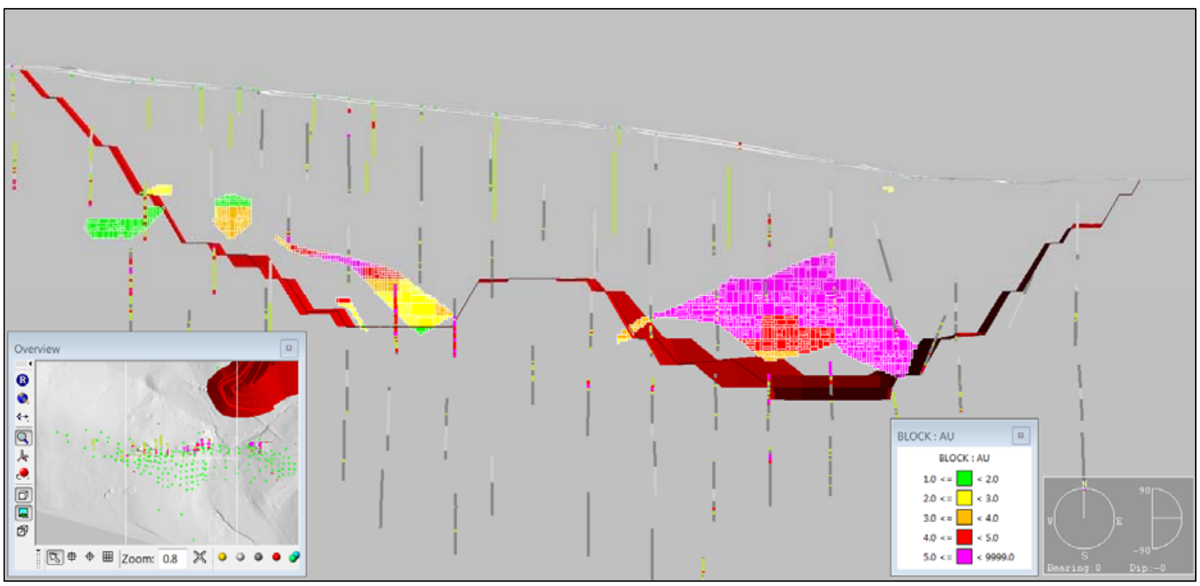
Source: Koza, 2014

**Figure 2.9.1: Final Damdamca Open Pit – April 2014**



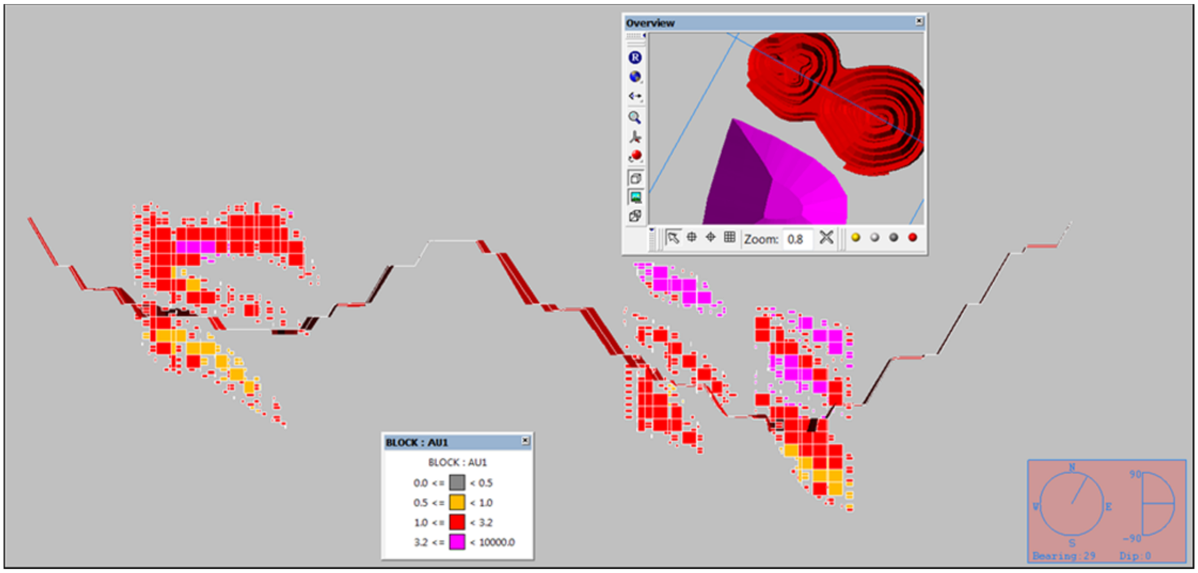

Source: SRK, 2014

**Figure 2.9.2: Perspective View of Main Zone Open Pit – East Pit**



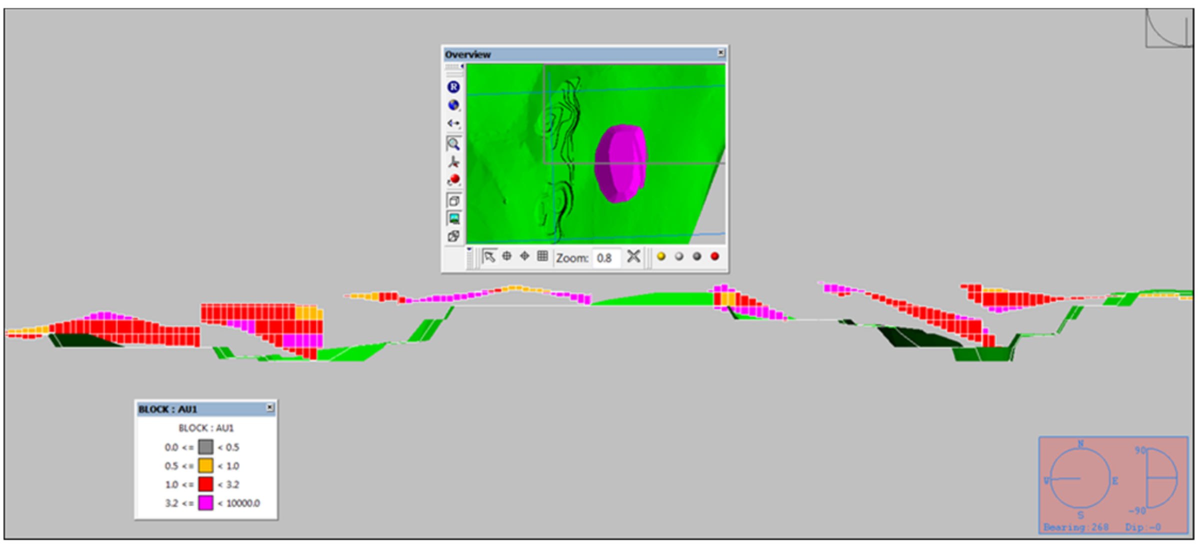

Source: SRK, 2014

**Figure 2.9.3: Perspective View of Main Zone Open Pit – West Pit**




Source: SRK, 2013

**Figure 2.9.4: Perspective View of Mermerlik Open Pit**




Source: SRK, 2013

**Figure 2.9.5: Perspective Long Section View of Kizilagil Open Pit**

## 2.9.1 Commentary on Mine Operations

The Main Zone targets total material movement at an average of 60,000 t/d through the end of 2015. With this production profile, an average of 80,000 tonnes per month has been estimated for 2015. As only a single phase is operating at any one time, stockpiling is required for steady delivery of ore to the plant. No stockpiled ore is available on site.

**Main Zone**

As of November 2013, operations were commencing at the Main Zone. At the end of 2013, mining had progressed to the 1150 level with five mining benches established.

Source: SRK, 2013

**Figure 2.9.1.1: Main Zone Pre-Production Mining**

Table 2.9.1.1 shows the major mining fleet used for both Damdamca and Main Zone operations at the end of 2013. The mining fleet is based on 40 t haul trucks and associated loading, drilling and ancillary operations to support that class of truck.

**Table 2.9.1.1: Kaymaz Contractor Mining Equipment Fleet**

| Units | Make | Type | Specification |
|---|---|---|---|
| 3 | CAT | Excavator | 349 |
| 2 | CAT | Excavator | 385 |
| 1 | CAT | Grader | 140 M |
| 1 | HITACHI | Excavator | 450 |
| 1 | JCB | Roller | |
| 1 | CAT | Excavator | 324 |
| 1 | JCB | Excavator | |
| 2 | LIEBER | Excavator | 944 |
| 1 | JCB | Roller | |
| 2 | CAT | Front End Loader | 966 |
| 1 | VOLVO | Front End Loader | 150 |
| 2 | Atlas COPCO | Drill Rig | D7 |
| 2 | Atlas COPCO | Drill Rig | T 35 - SMART |
| 40 | MAN | Haul Truck (40t) | 41400 |
| 2 | FORD | Water Truck | CARGO |
| 2 | FORD | Fuel Truck | CARGO |
| 2 | MAN | Lowboy | |
| 2 | IVECO | Maintenance Truck | |
| 3 | | Light Plants | |
| 2 | CAT | DOZER | D8 |
| 1 | CAT | LOADER | 963 |

Source: Koza, 2013

## 2.9.2 Geotechnical Analysis

Open pit geotechnical analysis is carried out by a dedicated geotechnical engineer employed by Koza. The pits at Kaymaz are designed using a triple bench configuration with overall slope angles ranging from 31° to 46° depending on section orientation. Geotechnical factors of safety are calculated using Slide 6.0 and DIPS 5.0 and generally are considered safe above a factor of safety greater than 1.3. The main source of production will be the Main Zone where ten geotechnical holes were drilled and sampled to determine the geotechnical parameters shown in Table 2.9.2.1. The site experience from Damdamca has also been used to classify high, medium and low strength granites that were the cause of stability problems at Damdamca.

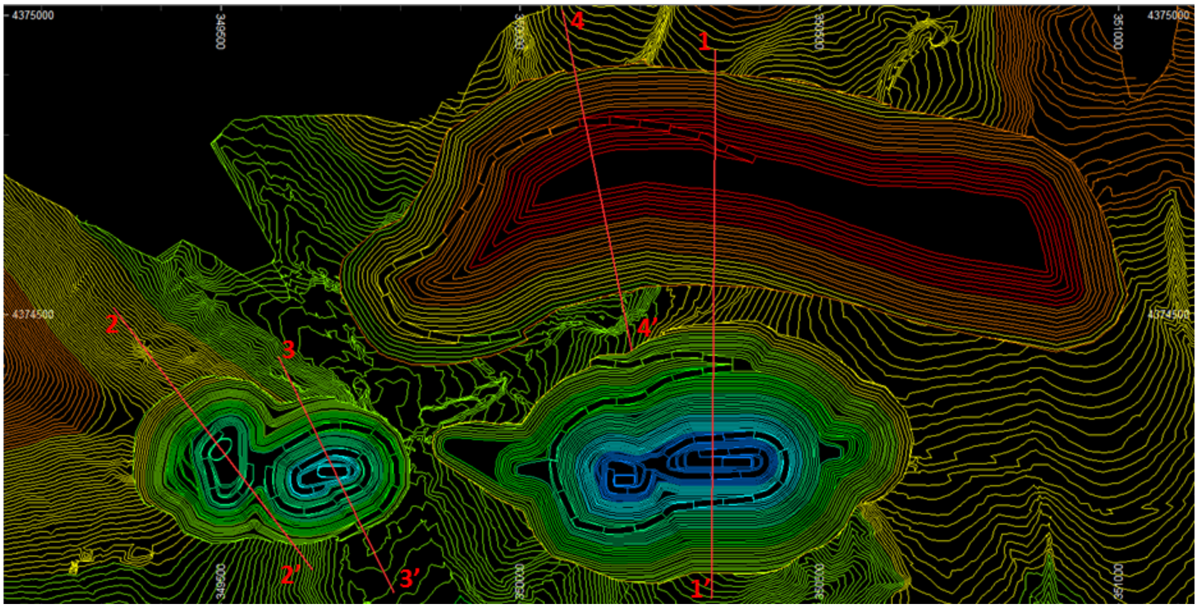
**Table 2.9.2.1: Kaymaz Geotechnical Parameters**

| Lithological Formation | C (kPa) | Φ (°) |
|---|---|---|
| Granite (H) | 420 | 34 |
| Granite (M) | 370 | 33 |
| Granite (L) | 180 | 32 |
| Serpentinite (H) | 400 | 32 |
| Serpentinite (M) | 350 | 31 |

Source: Koza, 2014

**Main Zone**

Figure 2.9.2.1 details the section lines that were analyzed by Koza engineers. Table 2.9.2.2 shows the pit geometry and calculated factors of safety for the current Main Zone pit design.




Source: Koza, 2013

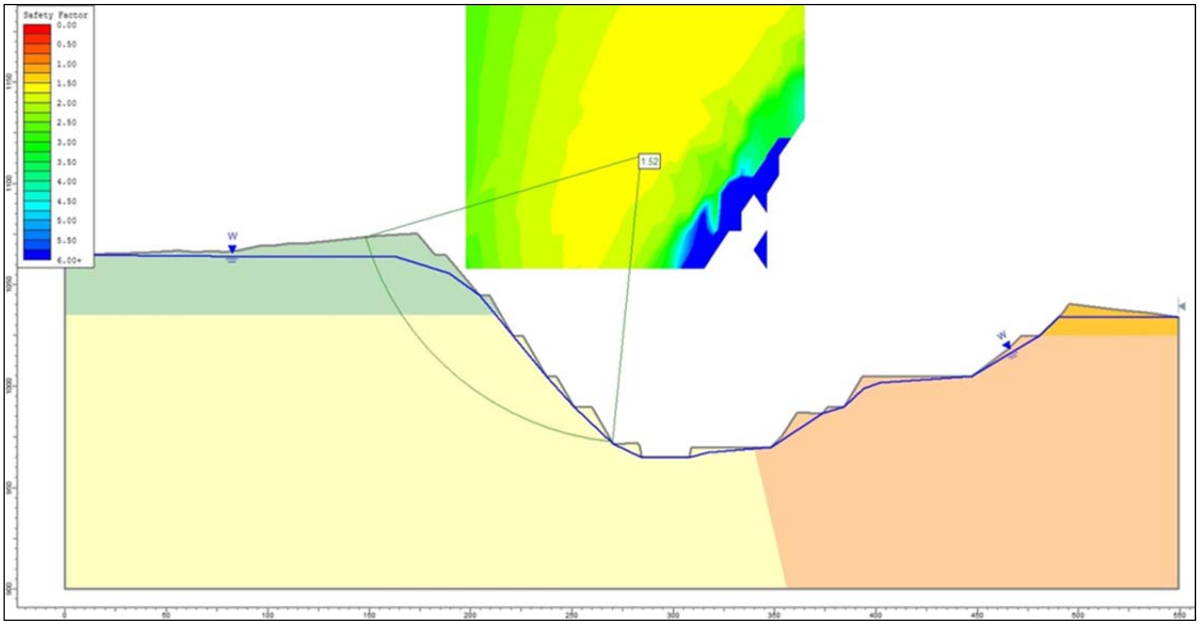
**Figure 2.9.2.1: Main Zone Geotechnical Sections**

**Table 2.9.2.2: Main Zone Slope Stability**

| Cross-section | Slope Height (m) | Slope Angle (°) | FOS |
|---|---|---|---|
| Section 1-1' (North) | 157 | 36 | 1.53 |
| Section 1-1' (South) | 150 | 40 | 1.56 |
| Section 2-2' (North) | 110 | 45 | 1.52 |
| Section 2-2' (South) | 89 | 42 | 1.91 |
| Section 3-3' (North) | 90 | 26 | 1.54 |

Source: Koza, 2014

Figure 2.9.2.2 illustrates the highest risk section 2 and rock type modeling used in the analysis. It should also be noted that the groundwater has been modeled quite close to the surface indicating the pit walls will be saturated.

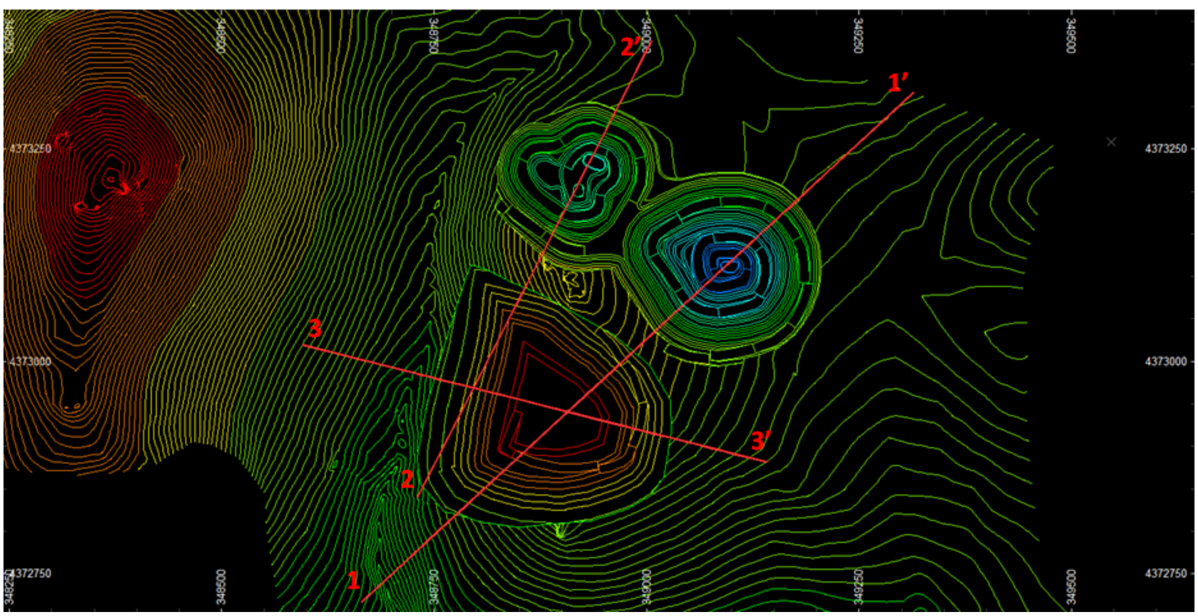


Source: Koza, 2013

**Figure 2.9.2.2: Main Zone Slide Analysis**

**<u>Mermerlik</u>**

Figure 2.9.2.3 details the section lines that were analyzed by Koza engineers. Table 2.9.2.3 shows pit geometry and calculated factors of safety for the current Mermerlik pit design.




Source: Koza, 2012

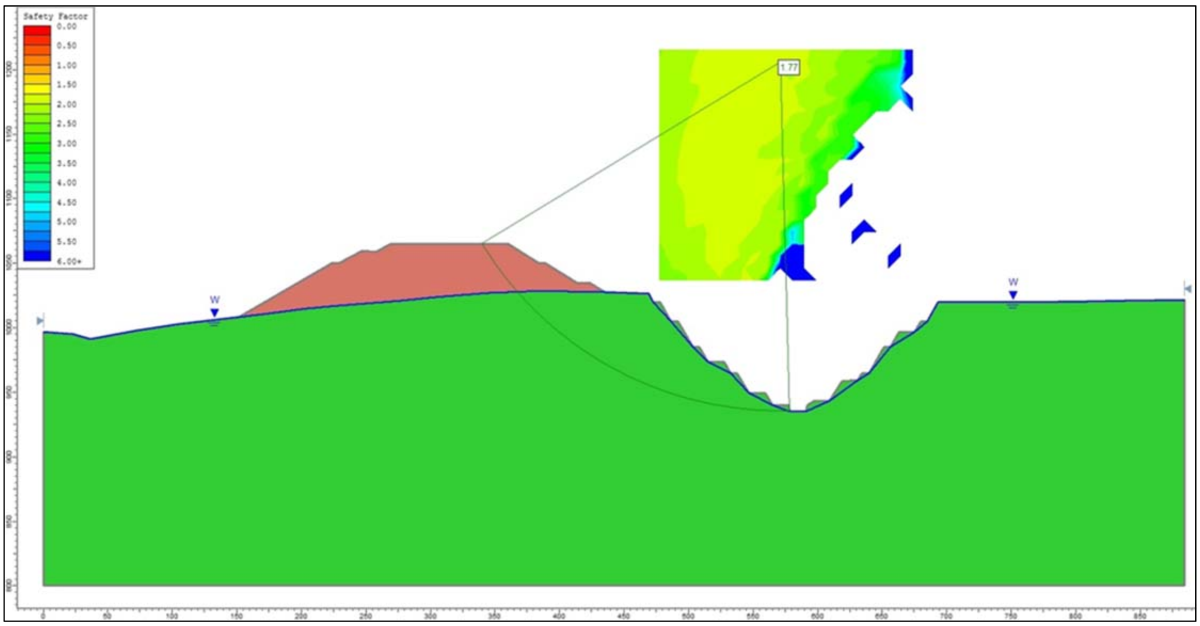
**Figure 2.9.2.3: Mermerlik Geotechnical Sections**

**Table 2.9.2.3: Mermerlik Slope Stability**

| Cross-section | Slope Height (m) | Slope Angle (°) | FOS |
|---|---|---|---|
| Section 1-1 (South) | 91 | 40 | 1.77 |
| Section 1-1 (North) | 85 | 40 | 2.02 |
| Section 2-2 (South) | 58 | 35 | 2.85 |
| Section 2-2' (North) | 52 | 41 | 2.79 |
| Section 3-3 (West) | 59 | 31 | 1.63 |
| Section 3-3 (South) | 42 | 27 | 1.99 |

Source: Koza, 2012

Figure 2.9.2.4 illustrates the highest risk section 1 (south) and rock type modeling used in the analysis. It should also be noted that the groundwater has been modeled to the surface indicating the pit walls will be saturated. The section 3 analyses are waste dump stability analyses and do not apply to in situ rock.




Source: Koza, 2012

**Figure 2.9.2.4: Mermerlik Slide Analysis**

# 2.10 Metallurgy, Process Plant and Infrastructure

## 2.10.1 Metallurgical Testing

The most recent metallurgical investigations were conducted by SGS Canada (SGS) for Koza and documented in their report: "An Investigation into the Recovery of Gold and Silver from Kaymaz Samples", December 16, 2010. Testwork was conducted on three Kaymaz samples representing low sulfide and high sulfide ore. As summarized in Table 2.10.1.1., the low sulfide ore composite (Low SS) assayed 7.06 g/t Au, 14.9 g/t Ag and 0.3% S and the first high sulfide composite (High SS) assayed 5.04 g/t Au, 16.7 g/t Ag and 4.07% S. The second high sulfide composite (High SS2) was used primarily for CIP modeling and assayed 8.52 g/t Au, 8.9 g/t Ag and 4.67% S. A rapid mineral scan was run on both the low sulfide and high sulfide composites and the results are reported in Table 2.10.1.2.

**Table 2.10.1.1: Head Analyses for Kaymaz Metallurgical Test Composites**

| Element | Low SS | High SS | High SS2 |
|---|---|---|---|
| Au 1 (g/t) | 6.99 | 4.93 | 8.55 |
| Au 2 (g/t) | 7.13 | 5.14 | 8.48 |
| Au Avg. (g/t) | 7.06 | 5.04 | 8.52 |
| Ag (g/t) | 14.9 | 16.7 | 8.9 |
| S (%) | 0.30 | 4.07 | 4.67 |

Source: SGS 2010

**Table 2.10.1.2: Results of Rapid Mineral Scan on Kaymaz Test Composites**

| Sample | | Low SS | High SS |
|---|---|---|---|
| Mineral Mass (%) | Cu-Sulfides | 0.02 | 0.00 |
| | Pyrites | 0.42 | 10.73 |
| | Ni-Sulfides | 0.02 | 0.07 |
| | Arsenopyrite | 0.04 | 0.10 |
| | Feldspars | 0.47 | 0.25 |
| | Fe-Sulfates | 0.08 | 0.29 |
| | Quartz | 91.87 | 83.51 |
| | Mica/Clays | 1.45 | 0.65 |
| | Chlorites | 0.28 | 0.13 |
| | Calcite/Dolomite | 0.05 | 0.03 |
| | Fe-Carbonates | 0.01 | 0.02 |
| | Fe-Oxides | 5.02 | 3.80 |
| | Ti-Minerals | 0.07 | 0.04 |
| | Barite | 0.16 | 0.36 |
| | Other | 0.02 | 0.03 |
| | **Total** | **100.00** | **100.00** |
| Mean Grain Size by Frequency (µm) | Cu-Sulfides | 11 | 0 |
| | Pyrite | 9 | 13 |
| | Ni-Sulfides | 6 | 7 |
| | Arsenopyrite | 8 | 6 |
| | Feldspars | 6 | 6 |
| | Fe-Sulfates | 6 | 13 |
| | Quartz | 17 | 24 |
| | Mica/Clays | 7 | 8 |
| | Chlorites | 7 | 8 |
| | Calcite/Dolomite | 6 | 7 |
| | Fe-Carbonates | 5 | 5 |
| | Fe-Oxides | 13 | 19 |
| | Ti-Minerals | 7 | 6 |
| | Barite | 7 | 13 |
| | Other | 7 | 6 |

Source: SGS, 2010

## Abrasion Testwork and Bond Work Index

A separate sample was used to run a Bond abrasion test, and was found to be highly abrasive with an abrasion index of 0.571, which SGS reported to be in the 84% percentile of most abrasive ores. No Bond ball mill Work index (BWi) determinations were run on the test composites, but the Kaymaz ore is reported in other sources to be very hard with a BWi of about 21 kWh/mt.

**Cyanidation Testwork**

Cyanidation tests were conducted on both the low sulfide and high sulfide composites to investigate the effect of grind fineness and the potential benefits of pre-aeration. Tests CN1 and CN2 were conducted in open vessels to evaluate the effect of omitting pre-aeration, all other tests were conducted in bottle rolls. In all cases the tests were conducted with the following conditions:

- Slurry density 45% solids;
- Retention time 48 hours;
- Cyanide Conc. 0.4 g/L NaCN; and
- pH 9.5-11.

The results of these tests are summarized in Table 2.10.1.3. Gold extractions from the low sulfide composite increased from 88.5% at a grind fineness of $P_{80}$ 60 µm to 89.3% at a grind of $P_{80}$ 20 µm. At the very fine grind of $P_{80}$ 11µm gold extraction increased to 93.7%. Silver extractions ranged from 45 to 70% over the grinds tested.

Gold extractions from the high sulfide composite were constant at 72.7% at grinds of $P_{80}$ 70 and $P_{80}$ 28 µm. At the very fine grind of $P_{80}$ 10µm gold extraction increased to 77%. Silver extractions ranged from about 50 to 60% over the grind sizes tested.

It should be noted that pre-aeration for 10 hours had the effect of significantly reducing cyanide consumption for both the low sulfide and high sulfide ore composites.

Several cyanidation tests were run on the high sulfide composite to investigate leach kinetics. The results of these tests are summarized in Table 2.10.1.4 and demonstrate that leaching kinetics on the high sulfide composite are very rapid and that near complete extraction of recoverable gold and silver can be achieved within 12 hours.

**Table 2.10.1.3: Summary of Bottle Roll Tests On High and Low Sulfide Test Composites**

| Test | Sample | $K_{80}$ | Preaer | Preg Sol'n (mg/L) | | Reagent Cons. (kg/t) | | Au Extr'n | Residue | Head | Ag Extr'n | Residue | Head |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No. | | (µm) | (h) | Fe | Ni | NaCN | CaO | (%) | Au (g/t) | Au (g/t) | (%) | Ag (g/t) | Ag (g/t) |
| CN3 | Low SS | 60 | 10 | 6.36 | 3.38 | 0.55 | 0.61 | 88.5 | 0.85 | 7.37 | 45.5 | 8.3 | 15.2 |
| CN1 | Low SS | 60 | 0 | 71.0 | | 0.83 | 1.80 | 87.4 | 0.93 | 7.40 | 44.3 | 8.4 | 15.1 |
| CN6 | Low SS | 20 | 10 | | | 0.46 | 1.00 | 89.3 | 0.78 | 7.31 | 53.8 | 6.8 | 14.7 |
| CN5 | Low SS | 11 | 10 | | | 0.59 | 1.15 | 93.7 | 0.47 | 7.50 | 69.9 | 4.4 | 14.6 |
| CN4 | High SS | 70 | 10 | 57.3 | 68.2 | 1.30 | 1.11 | 72.7 | 1.41 | 5.16 | 50.9 | 8.6 | 17.5 |
| CN2 | High SS | 70 | 0 | 117 | | 2.27 | 2.10 | 67.5 | 1.70 | 5.22 | 26.5 | 12.9 | 17.5 |
| CN8 | High SS | 28 | 10 | 19.6 | | 1.38 | 0.94 | 72.7 | 1.39 | 5.09 | 51.5 | 8.1 | 16.7 |
| CN7 | High SS | 10 | 10 | 4.80 | | 0.80 | 1.55 | 77.0 | 1.14 | 5.04 | 61.4 | 6.5 | 16.7 |

Source: SGS, 2010

**Table 2.10.1.4: Summary of Leach Kinetic Test Results on High Sulfide Composites**

| Test No. | Feed | Au Recovery (%) | | | | Ag Recovery (%) | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | 12 | 24 | 30 | 48 | 12 | 24 | 30 | 48 |
| CN4 | High SS | | | | 72.7 | | | | 50.9 |
| CN9 | High SS | 71 | 72 | 72 | 73.0 | 52 | 53 | 54 | 55.9 |
| CN10 | High SS | 74 | 74 | | 74.1 | 54 | 54 | | 55.0 |
| CN12 | High $SS_2$ | 62 | 62 | | 63.2 | | | | |

Source: SGS, 2010

**CIP Modeling**

Extensive modeling of the carbon-in-pulp (CIP) circuit requirements was conducted as part of the SGS test program and included the development of a carbon adsorption equilibrium isotherm followed by carbon adsorption kinetic tests. The data obtained from this work was used to mathematically model a variety of CIP circuit configurations. It was determined that Koza's CIP circuit specifications, which included eight stages of CIP with each tank having a volume of 210 $m^3$, would be more than adequate.

**Cyanide Destruction Testwork**

The destruction of cyanide contained in the final cyanidation tailing was investigated using the $SO_2$/air process for both the low sulfide and high sulfide ore composites. The results of both batch and continuous tests are summarized in Table 2.10.1.5 and show that it was possible to effectively treat the residue from the low sulfide composite with the $SO_2$/air process to achieve the target $CN_{WAD}$ concentration of 10 mg/L. The reagent requirement was approximately 4.5 g of $SO_2$ equivalent per gram of $CN_{WAD}$ in the cyanidation residue (destruct circuit feed), and 1 g hydrated lime and 0.1 g copper per gram of $CN_{WAD}$.

The residue from the high sulfide composite did not respond well to the $SO_2$/air process. The poor performance is attributed to the very high concentrations of nickel in the leach residues. Treating the pulp with 3.8 g $SO_2$ equivalent, 1.9 g hydrated lime and 1 g Cu per gram of $CN_{WAD}$ in the feed reduced the $CN_{WAD}$ and nickel concentrations from 147 mg/L and 68 mg/L to 29/mg/L and 16/mg/L, respectively. Additional testwork will be required to define the reagent levels that will be required to achieve $CN_{WAD}$ target guidelines when processing high sulfide ores.

**Table 2.10.1.5: Summary of Cyanide Destruction Test Results**

| Test | Mode | Reten. Time | Composition (Solution Phase) | | | | | | | | Cumulative Reagent Add'n[1] | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | pH | $CN_T$ | $CN_{WAD}$ $CN_{Picric}$ | Cu | Fe | Ni | Zn | SCN | g/g $CN_{WAD}$ | | | Cu |
| | | | | | | | | | | | $SO_2$ Equiv. | Lime | Cu | mg/L Sol'n |
| | | min | | mg/L | mg/L | mg/L | mg/L | mg/L | mg/L | mg/L | | | | |
| *Pulp from Test CN-3* | | | *10.4* | *227* | *206* | *10.9* | *6.36* | *3.38* | *1.88* | *25* | ... | ... | ... | ... |
| *Target* | | | ... | ... | *10* | *5.0* | ... | *1.0* | *5.0* | ... | | | | |
| CND 1A | Batch | 75 | 8.5 | ... | 0.6 | ... | ... | ... | ... | ... | 5.59 | 3.73 | 0.15 | 30 |
| CND 1B | Continuous | 61 | 8.5 | 17[2] | 5.2 | 1.05 | 4.09 | 1.58 | ... | ... | 4.98 | 1.08 | 0.10 | 21 |
| | After approx. 1 week | | 8.1 | 14 | 2.2 | 0.08 | 4.05 | 0.97 | ... | ... | ... | ... | ... | ... |
| CND 1C | Continuous | 58 | 8.5 | 19[2] | 5.6 | 1.13 | 4.81 | 2.33 | ... | ... | 4.51 | 1.06 | 0.10 | 20 |
| | After approx. 1 week | | 8.1 | 10 | 3.1 | 0.12 | 4.17 | 1.49 | ... | ... | ... | ... | ... | ... |
| CND 1D | Continuous | 60 | 8.6 | 26[2] | 3.8 | 1.01 | 7.82 | 2.45 | ... | ... | 3.73 | 0.72 | 0.10 | 21 |
| | After approx. 1 week | | 8.1 | 8.3 | 3.5 | 0.12 | 5.20 | 1.62 | ... | ... | ... | ... | ... | ... |
| *Pulp from Test CN-4* | | | *9.8* | *369* | *147* | *12.9* | *57.3* | *68.2* | *<0.02* | *210* | ... | ... | ... | ... |
| *Target* | | | ... | *10* | ... | *5.0* | ... | *1.0* | *5.0* | ... | | | | |
| CND 2A | Batch | 75 | 8.5 | ... | 14 | ... | ... | ... | ... | ... | 7.75 | 4.32 | 0.88 | 130 |
| | | 105 | 8.5 | 43[2] | 19 | 1.24 | 8.45 | 4.83 | ... | ... | 10.8 | 6.37 | 0.88 | 130 |
| CND 2B | Continuous | 57 | 9.0 | 90[2] | 29 | 5.70 | 21.7 | 16.4 | ... | ... | 3.81 | 1.90 | 1.01 | 149 |
| | After approx. 1 week | | 8.2 | 64 | 30 | 2.09 | 12.1 | 14.3 | ... | ... | ... | ... | ... | ... |
| CND 2C | Continuous | 56 | 9.0 | 60[2] | 29 | 3.07 | 11.1 | 16.5 | ... | ... | 5.91 | 2.98 | 1.20 | 177 |
| *Pulp from Test CN-10 CIP* | | | *10.1* | *52* | *52* | *7.9* | *0.8* | *15.0* | | | | | | |
| *Target* | | | ... | *10* | ... | *5.0* | ... | *1.0* | *5.0* | ... | | | | |
| CND 3A | Batch | 120 | 9.1 | 0.9 | 0.9 | 0.10 | 0.18 | <0.05 | ... | ... | 9.12 | 4.22 | 0.96 | 50 |

[1] Cu added as $CuSO_4\ 5H_2O$, $SO_2$ added as $Na_2S_2O_5$

[2] Calculated based on $CN_T = CN_{WAD} + CN$ in $Fe(CN)_6$

Source: SGS, 2010

## 2.10.2 Process Plant

The Kaymaz process plant initiated operations during September 2011 and was designed to process ore from the Kaymaz open pit mine at the rate of 45 t/h at a grind size of $P_{80}$ 75µm, but was soon operating at a rate of almost 60 t/h (1,440 t/d) at an average grind size of $P_{80}$ 68 µm. The process plant capacity was expanded to about 105 t/h (2,500 t/d) during 2013 with the installation of additional crushing and grinding capacity. The expanded process plant was commissioned during October 2013.

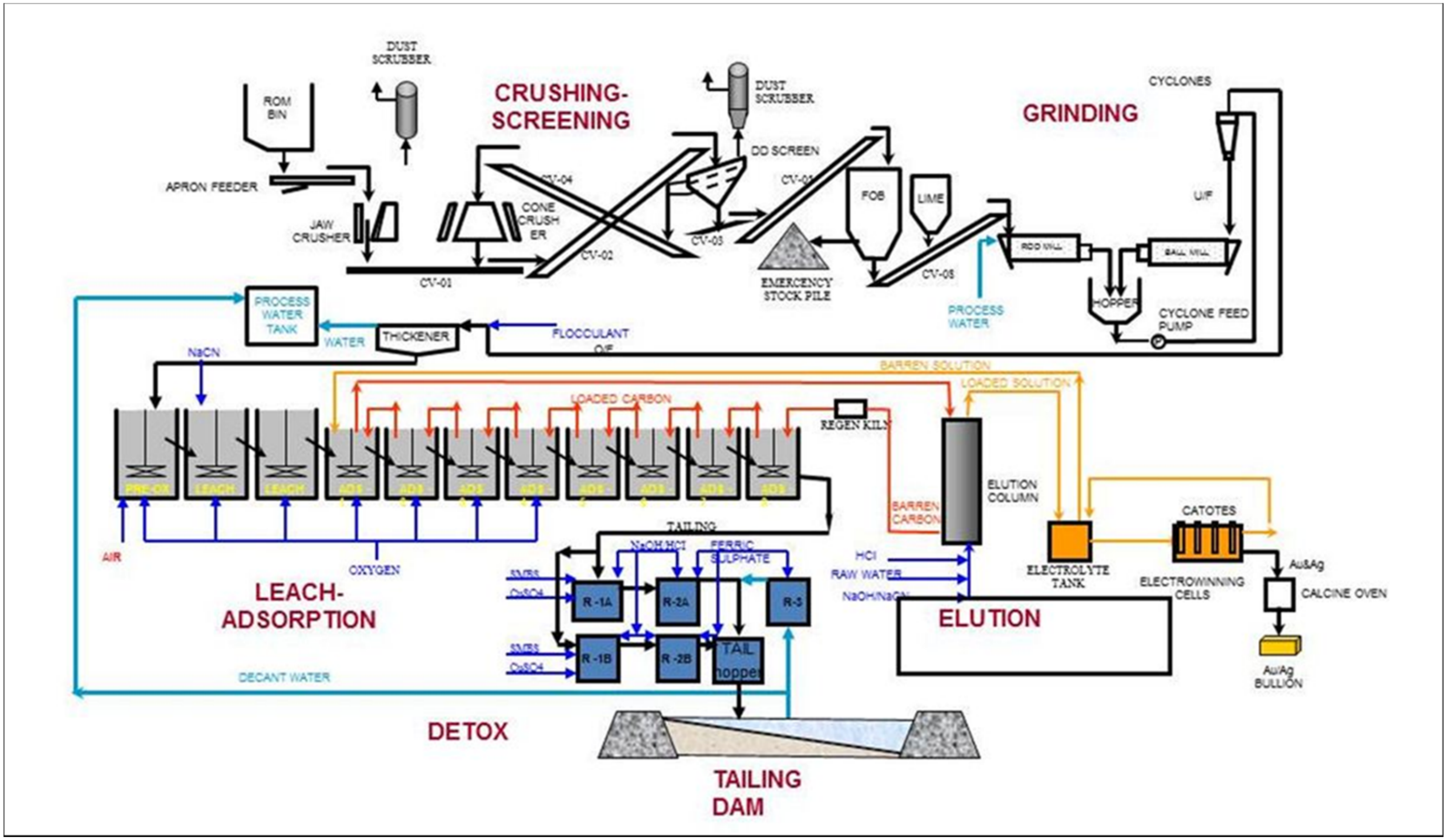
**Process Plant Description: Pre-Expansion**

The Kaymaz process plant incorporates a conventional carbon-in-pulp (CIP) cyanidation flowsheet that consists of two-stage crushing, two-stage grinding, pre-aeration, cyanide leaching with oxygen addition, CIP adsorption of the dissolved gold, carbon elution, electrowinning and smelting through to doré metal. A schematic flowsheet of the pre-expansion process circuit is presented in Figure 2.10.2.1 and a list of major equipment is presented in Table 2.10.2.1

**Table 2.10.2.1: Kaymaz Process Plant: Pre-Expansion Major Equipment List**

| Equipment Item | Units | Description | kW |
|---|---|---|---|
| Primary Jaw Crusher | 1 | Metso to C110 - 1.10 m x 0.88 m Double toggle | 150 |
| Secondary Crusher | 1 | Metso Omni cone 1560 | 315 |
| Crushing Circuit Screen | 1 | 8 ft x 20 ft Double Deck ( 50 mm and 18 to 20 mm decks) | 30 |
| Crushed Ore Silo | 1 | 1,400 t capacity | |
| Rod Mill | 1 | 2.7 m dia x 4.1 m long | 315 |
| Ball Mill | 1 | 3.6 m dia x 5.7.m long | 1,250 |
| Classifying Hydrocyclones | 9 | 250 mm diameter | |
| Preleach Thickener | 1 | 10 m diameter Outotec High Rate | |
| Agitated Leach Tanks | 3 | 720 $m^3$ nominal capacity | 60 |
| CIP Tanks | 8 | 210 $m^3$ nominal capacity | 25 |
| Interstage Screen | 10 | RPA-4-200SSH | |
| Elution Column (AARL) | 1 | 4 t capacity | |
| Electrowinning Cells | 2 | 100 $ft^3$ 13*2 cathodes and 14*2 anodes | 800A/2.2V |
| Regeneration Kiln | 1 | Metso - 250 kg/hr | |
| Smelting Furnace | 1 | Duraline Induction Furnace - 125 kg capacity | 125 |

Source: Koza, 2011




Source: Koza, 2012

**Figure 2.10.2.1: Kaymaz Process Plant Flow Sheet - Pre-Expansion**

Crushing

Ore is trucked to the stockpile area and separated into different stockpiles from the different feed sources, and then blended as needed with a front end loader prior to feeding to the primary jaw crusher. Primary crushed ore is conveyed to a double-deck vibrating screen, which operates in closed circuit with a secondary cone crusher to produce a final crushed product which is conveyed to the 1,400 t crushed ore silo at a nominal size of $P_{80}$ 16 to 18 mm. Dust collection is installed at all major emission and transfer points.

Milling and Classification

Mill feed is reclaimed from the storage silo with two belt feeders that discharge onto the rod mill feed conveyor. Hydrated lime is added to the crushed ore as it is conveyed to the rod mill in order to maintain the necessary slurry alkalinity. The rod mill operates in open circuit and discharges into a common mill discharge sump. From the sump, the ground ore is pumped to a cluster of eight 250 mm hydrocyclones with the underflow feeding the ball mill and the overflow being advanced to the grinding control thickener where it is thickened to about 45% solids prior to being pumped to the cyanidation circuit at a grind fineness of about $P_{80}$ 65µm.

Cyanidation

The cyanide leach circuit consists of one stage of pre-aeration followed by cyanidation in two 720 $m^3$ agitated leach tanks that provide about 16 hours retention time at the design feed rate of 60 t/h. The pre-aeration stage is incorporated into the flowsheet in order to reduce cyanide consumption when processing high sulfide ores. Cyanide is added to the first of the two cyanidation leach tanks at an initial concentration of 200 ppm and the leach slurry pH is maintained at 10.5 to 11. Oxygen is injected into the leach tanks through an annulus in the agitator shaft.

Discharge from the leach tanks flows by gravity to an eight-stage counter-current CIP circuit with oxygen injected into the first four CIP tanks. The CIP tanks are all installed on the same level with cylindrical basket-type inter-stage screens installed to retain the carbon in each tank while allowing the slurry to progress to the next stage. Overall retention time in the cyanidation/CIP circuit is about 34 hours.

Carbon is maintained at a concentration of about 6 g/L in each adsorption tank, and is transferred counter-currently to the slurry flow with vertical pumps. Carbon from the first CIP tank is loaded to about 3,000 to 4,000 g/t gold. Tailing solution discharging from the CIP circuit typically grades 0.01 to 0.02 g/t Au with tailing solids ranging from 0.4 to 0.7 g/t Au. After the final stage of CIP, slurry passes over a carbon safety screen prior to advancing to the tailings detoxification circuit.

Tailings Detoxification and Disposal

Tailings from the CIP circuit are treated for the destruction of residual cyanide using a conventional $SO_2$/air detoxification circuit employing copper sulfate and sodium metabisulfite as the active reagents. Detoxification is achieved in agitated tanks before the tailings are pumped to the tailings storage facility (TSF). Provision is also available to add ferric chloride in the tailings treatment circuit although it is anticipated that this will only be used during periods of elevated arsenic levels. Kaymaz is required to detox the tailings to only 10 ppm $CN_{WAD,}$ but stated that they are actively treating to <1 ppm $CN_{WAD}$.

Carbon Elution and Regeneration

Loaded carbon from the CIP circuit is pumped to the loaded carbon screen where it is drained and washed before passing into a pressure Anglo American Research Laboratories (AARL) elution column. Prior to elution the carbon is initially acid washed with hydrochloric acid to remove scale and other contaminants. Elution is then accomplished with a solution of 12% NaOH and 15% NaCN heated to $110^{0}$C. The capacity of the elution circuit is 4 t per batch and each elution cycle takes about 6 hours to complete.

The carbon regeneration kiln has a capacity of 250 kg/hour. After every second elution, cycle carbon is hydraulically transferred to the regeneration section where the carbon is dewatered, and thermally regenerated in a gas fired rotary kiln at 700ºC. This removes organic components, adsorbed along with the gold in the CIP circuit, and restores the carbon activity. Regenerated carbon is quenched and hydraulically transferred, screened on a static sieve-bend screen to remove fines generated during regeneration and carbon handling, and then returned to the last CIP stage.

Gold Electrowinning and Smelting

Electrolyte from the eluate storage tank is routed to the electrowinning cells located in the refinery where the gold and silver are plated onto stainless steel wire mesh cathodes by recirculating the solution until the majority of precious metals have been recovered. Gold and silver plated in the cell are stripped from the loaded cathodes using a high pressure water spray and the sludge produced is filtered, dried, calcined and smelted to doré.

Tailing Storage Facility

Single stage pumps are used to transfer tailings slurry to the new tailings storage facility (TSF), which consists of a plastic lined earthwork embankment behind which the tails slurry is deposited.

Plant Monitoring and Accounting

Feed tonnage to the plant is monitored by a weightometer on the mill feed conveyor which controls the belt feeder below the crushed ore silo and is used for accounting. The cyclone overflow represents the plant feed and is sampled automatically every hour to formulate 12 hour shift composites. The plant tails, after detoxification, are sampled using an automatic cutter with samples taken every two hours and prepared into a 12 hour shift composite for analysis. Tails samples are taken both before and after detoxification. In addition tails samples are taken and analyzed every two hours to monitor $CN_{WAD}$ levels.

Plant accounting assays are generally based on aqua regia digestions and Atomic Absorption Spectroscopy analysis (AAS) for solids and AAS for solutions on both gold and silver. Carbon samples are roasted and digested prior to AAS. Fire assays on solids are only undertaken on high sulfur/high copper samples.

**Process Plant Description: Post-Expansion**

Koza expanded the capacity of the Kaymaz process plant from about 60 t/h (1,440 t/d) to 105 t/h (2,500 t/d) during 2013, with commissioning during October 2013. The plant expansion included installation of new primary and secondary crushers along with the addition of a new rod mill, ball mill and grinding control thickener and installation of larger inter-stage screens in the CIP circuit. The leach circuit tankage was not expanded, resulting in the reduction of overall leach retention time from 34 hours to about 17 hours. Koza has conducted metallurgical investigations that confirm that 17 hour leach retention is sufficient to achieve target gold extraction. A schematic flowsheet of the expanded Kaymaz process plant is shown in Figure 2.10.2.2 and a list of major equipment is shown in Table 2.10.2.2. The basic process flowsheet is identical to the previous flowsheet; however the grind size at the higher capacity is somewhat finer at about $P_{80}$ 58 µm. Due to the highly abrasive ore, consumption of wear materials is high. Jaw crusher liners are rotated once per week and replaced every four weeks, and cone crusher liners are replaced every two weeks.

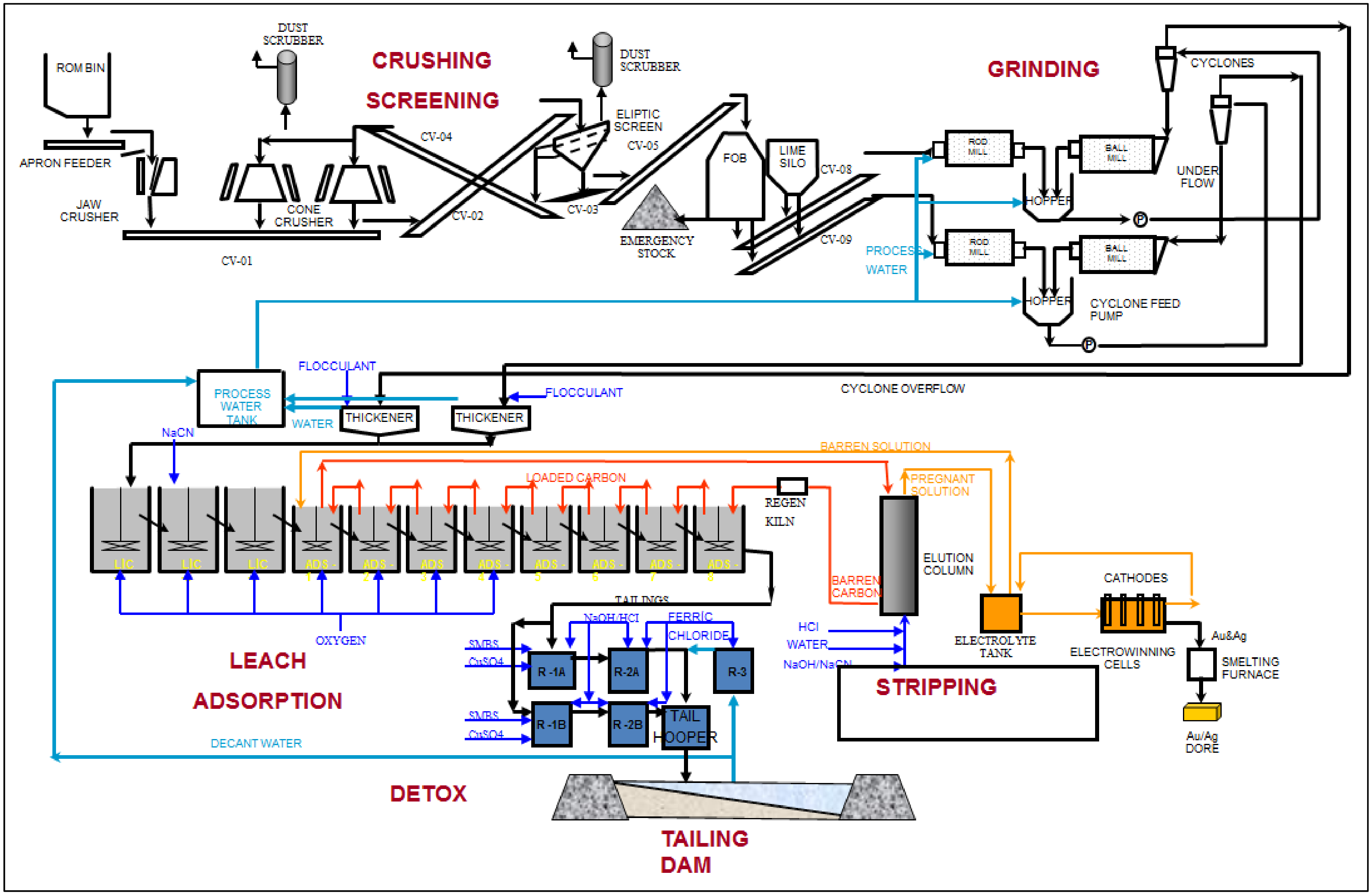



**Figure 2.10.2.2: Kaymaz Process Plant: Post-Expansion Flowsheet**

**Table 2.10.2.2: Kaymaz Process Plant: Post-Expansion Major Equipment List**

| Equipment Item | Units | Description | kW |
|---|---|---|---|
| **Crushing Circuit** | | | |
| Primary Jaw Crusher | 1 | Metso C-120 | |
| Secondary Crusher | 2 | Nordberg GP-550 | |
| Crushing Circuit Screen | 1 | Metso EF-2461 Double Deck | |
| Crushed Ore Silo | 1 | 1,400 t capacity | |
| **Grinding Circuit** | | | |
| Rod Mill | 1 | 2.7 m dia x 4.5 m long | 315 |
| | 1 | 2.7 m dia x 4.1 m long | |
| Ball Mill | 1 | 3.6 m dia x 6.3.m long | 1,250 |
| | 1 | 3.6 m dia x 5.7.m long | |
| Classifying Hydrocyclones | 9 | 250 mm diameter | |
| | 10 | 250 mm diameter | |
| Preleach Thickener | 2 | 10 m diameter Outotec High Rate | |
| **Cyanidation/ Gold Recovery** | | | |
| Agitated Leach Tanks | 3 | 720 $m^3$ nominal capacity | 60 |
| CIP Tanks | 8 | 210 $m^3$ nominal capacity | 25 |
| Interstage Screen | 10 | RPA-5-250SSH | |
| Elution Column (AARL) | 1 | 4 t capacity | |
| Electrowinning Cells | 2 | 100 $ft^3$ - 26 cathodes and 28 anodes | 800A/2.2V |
| Regeneration Kiln | 1 | Metso - 250 kg/hr | |
| Smelting Furnace | 1 | Duraline Induction Furnace - 125 kg capacity | 125 |

Source: Koza, 2013

## 2.10.3 Plant Performance

**Metallurgical Recoveries and Plant Throughput**

Mining activities during 2014 at Kaymaz were curtailed after April, and much of the production for the balance of the year was derived from stockpiled ore and from ore hauled from the Söğüt open pit mine. Process operations were suspended at the end of November with no production reported for December. Table 2.10.3.1 provides a summary of plant performance during 2014. A total of 659,177 t of ore were processed at an average grade of 4.88 g/t Au and 3.66 g/t Ag. Overall gold recovery averaged 85.8% and overall silver recovery averaged 58.9%, resulting in the production of 92,482 poured ounces of gold and 46,889 poured ounces of silver.

**Table 2.10.3.1: Summary of Kaymaz Process Plant Monthly Performance - 2014**

| | Feed Tonnes | | | | Damdamca Grade | | Main Zone Grade | | Söğüt Grade | | Recovery % | | Poured Ounces | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Month** | **Damdamca** | **Main Zone** | **Söğüt** | **Total** | **Au, g/t** | **Ag, g/t** | **Au, g/t** | **Ag, g/t** | **Au, g/t** | **Ag, g/t** | **Au** | **Ag** | **Au** | **Ag** |
| January | 76,568 | | | 76,568 | 5.41 | 4.09 | | | | | 81.5 | 58.4 | 12,753 | 5,923 |
| February | 69,772 | | | 69,772 | 5.15 | 3.54 | | | | | 84.0 | 60.8 | 9,418 | 4,661 |
| March | 76,151 | | | 76,151 | 4.89 | 3.48 | | | | | 84.1 | 58.5 | 9,224 | 4,820 |
| April | 72,079 | | | 72,079 | 5.25 | 3.62 | | | | | 84.3 | 59.6 | 11,823 | 5,454 |
| May | 64,063 | | 9,318 | 73,380 | 5.91 | 4.67 | | | 3.56 | 1.2 | 85.5 | 62.1 | 10,337 | 5,522 |
| June | 59,865 | 12,704 | | 72,569 | 4.31 | 3.23 | 5.94 | 4.53 | | | 86.7 | 57.5 | 9,049 | 4,684 |
| July | 47,061 | 17,405 | | 64,465 | 4.08 | 2.97 | 4.1 | 4.07 | | | 87.0 | 59.0 | 7,742 | 3,778 |
| August | 12,241 | 11,479 | 16,318 | 40,038 | 6.93 | 4.65 | 3.37 | 3.64 | 3.67 | 1.55 | 90.4 | 57.7 | 6,448 | 3,309 |
| September | 6,231 | 14,847 | 17,368 | 38,447 | 10.59 | 7.6 | 3.21 | 3.46 | 4.25 | 1.53 | 92.4 | 53.5 | 6,113 | 2,353 |
| October | 31,695 | 2,589 | 5,566 | 39,850 | 4.36 | 4.37 | 3.15 | 3.22 | 5.2 | 1.58 | 87.7 | 60.3 | 4,730 | 2,509 |
| November | 29,906 | 5,952 | | 35,858 | 4.38 | 4.61 | 1.02 | 1.97 | | | 87.5 | 65.7 | 3,804 | 3,088 |
| December | | | | | | | | | | | | | 1,041 | 789 |
| **Total** | **545,632** | **64,976** | **48,570** | **659,177** | **5.08** | **3.85** | **3.81** | **3.71** | **4.03** | **1.48** | **85.8** | **58.9** | **92,482** | **46,889** |

Source: Koza, 2014

**Operating Costs**

Process plant unit operating costs for 2013 and 2014 are summarized in Table 2.10.3.2. Unit processing costs for 2013 averaged US$37.11/t and during 2014 unit processing costs averaged US$17.91. The lower unit operating cost for 2014 is, in part, attributed to the higher production rates following the plant expansion.

**Table 2.10.3.2: Summary Kaymaz Process Plant Operating Costs**

| | US$/t | |
|---|---|---|
| **Cost Area** | **2013** | **2014** |
| Chemicals | 7.93 | 4.12 |
| Materials | 8.98 | 4.69 |
| Hourly Labor | 2.23 | 0.99 |
| Salaries | 0.77 | 0.25 |
| Energy | 8.85 | 4.23 |
| Maintenance | 4.88 | 2.14 |
| Contractors | 2.07 | 1.09 |
| Other | 1.40 | 0.41 |
| **Total** | **37.11** | **17.91** |
| Exchange Rate (TL:US$) | 1.90 | 2.20 |

Source: Koza, 2013/2014

## 2.11 Tailings Storage Facility

Tailings from the CIP plant are detoxified using the Inco sulfur dioxide/air process to ~ 1ppm $Cn_{WAD}$, below the permit requirement of 10 ppm required to be in compliance with EU Directives. Detoxified tailings are then pumped at a slurry density of 40 to 45% solids to the tailing storage facility (TSF). The TSF was designed as a zero discharge facility, with the supernatant being reclaimed and recycled back to the process.

The Kaymaz TSF was designed by Hidromark Engineering and Consultancy Ltd. in 2009. Design parameters for dynamic/seismic and static stability are based on the specialized studies documented in the following reports:

- Kaymaz TSF Geological and Geotechnical Study Report (SIAL, 2009);
- Study Report on Kaymaz Fault, Activity and Impact on Koza Kaymaz Prospect (Osmangazi University, 2009); and
- Earthquake Risk Analysis Report (Kaptan C., 2009).

The TSF construction and operation is being carried out in three stages. Stage one has been completed and the TSF started to receive process tailings from the Kaymaz deposit in 2010. Stage two was completed in 2012. The final capacity at closure will be approximately 3 $Mm^3$. The TSF will have sufficient capacity to meet a 100 year storm event. Diversion ditches are constructed around the TSF. The TSF is a zero discharge facility and was constructed according to the following specifications:

- Geotextile (500 $g/m^2$) spread over compacted ground;
- 0.5 m compacted clay layer ($k=1x10^{-8}$ m/s);
- 1.5 mm HDPE geomembrane ($k=1x10^{-13}$ m/s), and
- Geosynthetic drainage layer.

# 2.12 Environmental

## 2.12.1 Permitting

The EIA permit for the Damdamca and Main Zone open pits and the Tailings Storage Facility (TSF) was obtained on November 2, 2009. Open pit mining at Damdamca and TSF operation began in 2011. A second EIA permit was obtained for mine expansion on November 15, 2012. The mine expansion involved the Mermerlik and Kızılagıl open pits. All environmental permits for the first EIA have been obtained. The temporary operation license was obtained on December 16, 2014 and is valid for one year. The environmental operation permit process is ongoing. The land permits involving change of status of pasture land has not been obtained yet for the Kızılagıl area. All other environmental permits for the Kaymaz mine have been secured. Currently, a third EIA study is in progress for the expansion of the Topkaya open pit.

## 2.12.2 Environmental Management and Monitoring

A Mine Closure and Reclamation Plan (MCRP) for the Kaymaz mine does not exist. However, Koza has made some preliminary estimates for mine closure based on earlier EIA and technical studies and legal commitments. The preliminary mine closure cost estimate is approximately US$34 million; a portion of that will be used for partial back filling of the pits. However, recent geochemical assessment conducted with more data indicated that backfilling of the open pits may not necessarily provide better conditions with regard to the ARD formation. Without the partial back-filling, the open pits are anticipated to remain near neutral, with certain constituents such as sulfate, fluoride, arsenic, chloride, and base metals showing increases over time. In the absence of any groundwater pollution potential from the post-closure pit lakes, it may be possible to avoid the partial back-filling of the open pits. This condition would of course have to be studied further. However, in such an event, the current mine closure cost estimates may be significantly reduced.

# 3 Conclusions and Recommendations

## 3.1 Geology and Resources

SRK recommends that Koza report resources within pit optimization shell. This has become a standard procedure for mining companies internationally.

SRK recommends that Koza composite on run length intervals rather than using the distribution option in order to standardize the sample length.

The wireframes at Mermerlik should be reviewed to incorporate narrow zones of waste into the wireframes and to incorporate all drilling.

Swath plots should be generated as another method of resource model validation.

In regard to QA/QC, SRK recommends the following:

- The use of the site specific RMs should be discontinued and CRMs as suggested by Bloom (2013) should be used;
- Plot the standards against time to determine if the laboratory has trouble during a certain period;
- Duplicate samples submitted to the Kaymaz lab should be analyzed once and should not be reanalyzed until the duplicate is similar to the original. This defeats the purpose of submitting duplicate samples;
- Pulp duplicates should be prepared and submitted to the primary lab;
- Investigate the silver failures at the laboratory by contacting the laboratory and discussing the failures with the laboratory;
- Submit commercial CRMs for silver to the laboratory to check performance;
- Submit pulp samples to a secondary laboratory as a check of the Kaymaz lab;
- When using a secondary check laboratory, plot QA/QC data individually for each laboratory;
- Use the same analytical method at both the primary and secondary laboratories; and
- CRM or RM samples should be submitted with the check assay samples.

## 3.2 Mining

Since the beginning of 2014, operations have been suspended at the Main Zone and Koza only received permits to operate in December of that year. Koza did successfully mine out the Damdamça open pit through April 2014.

Koza performs global stability analysis on the pit designs at Kaymaz. It may be prudent for Koza staff to implement local stability analysis into their model runs for the Main Zone given the slope stability problems encountered at Damdamça. This will lead to improved inter-ramp angle optimization based on material hardness rather than applying an overall wall angle to the entire depth of the deposit.

SRK recommends that additional time be spent on the pit design of Main Zone so that the walls can be straightened and noses removed where potential slope instability will be focused. SRK would also recommend a phased approach be applied at Main Zone rather than a single top down bench approach that was used at Damdamça. The east pit of main zone is 700 m wide and the west pit of main zone 430 m wide. These are very large pits to be taken in a single phase.

## 3.3 Metallurgy and Process

SRK offers the following conclusions and recommendations regarding the Kaymaz process plant:

- The Kaymaz process plant has been well designed and constructed, and is similar in many respects to the Ovacık and Mastra process plants;
- Plant capacity was doubled to about 2,500 t/d in 2013 and 2014 was the first full year of production through the expanded facility. However, full capacity was not realized due to curtailment of mining operations;
- Gold recovery during 2014 averaged 85.80% and silver recovery averaged 59.35%; and
- Process plant operating costs averaged US$17.91/t of ore processed during 2014 (January to November).

## 3.4 Environmental

The second EIA permit was obtained for mine expansion on November 15, 2012. The mine expansion involved the Mermerlik and Kızılagıl open pits. All environmental permits for the first EIA have been obtained. The land permits involving change of status of pasture land has not been obtained yet for the Kızılagıl area. All other environmental permits for the Kaymaz mine have been secured. Currently, a third EIA study is in progress for the expansion of the Topkaya open pit.

# 4 References

Bloom, L. (Bloom), 2013. *Ankara geochemistry, assaying and quality control sessions* report to Koza Gold, dated May 31, 2013, 15p.

Datamine International (September 2009), *Mineral Resource Estimate for Kaymaz Prospect, Turkey, Technical Report.*

ESRI Basemap NatGeo_World_Map, 2013, National Geographic, Esri, DeLorme, NAVTEQ, UNEP-WCMC, USGS, NASA, ESA, METI, NRCAN, GEBCO, NOAA, iPC, Accessed January 2013.

ESRI Basemap World_Topo_Map, 2013, Sources: Esri, DeLorme, NAVTEQ, TomTom, Intermap, increment P Corp., GEBCO, USGS, FAO, NPS, NRCAN, GeoBase, IGN, Kadaster NL, Ordnance Survey, Esri Japan, METI, Esri China (Hong Kong), and the GIS User Community, Accessed January 2013.

EUROGOLD MADENCİLİK A.Ş., 1996, *Kızılsaray (D4-17) (Söğüt/Eskişehir) Prospect*, Unpublished Report, 10p.

Gencor, 1994, *Kaymaz Gold Project Feasibility Study, Prepared for Tüprag Metal Madenlicik Ltd. Şti a subsidiary of Gencor*, Unpublished Report by Gencor Ltd., South Africa.

Kara, Z., 2007, Personal communication on the Kaymaz Project.

Koza Altın İşletmeleri A.Ş. (Fall 2013), *Mineral Resource Estimate for the Main Zone Prospect, Turkey.*

Koza Altın İşletmeleri A.Ş. (Fall 2013a) *Mineral Resource Estimate for the Akbaştepe Prospect, Turkey.*

Koza Altın İşletmeleri A.Ş. (Fall 2013b) *Mineral Resource Estimate for the Korudanlık Prospect, Turkey.*

Koza Altın İşletmeleri A.Ş. (Fall 2013c) *Mineral Resource Estimate for the Hayriye Prospect, Turkey.*

Koza Altın İşletmeleri A.Ş. (Fall 2011a), *Mineral Resource Estimate for the Damdamca Prospect, Turkey*

Koza Altın İşletmeleri A.Ş. (Fall 2011b), *Mineral Resource Estimate for the Kizilagil Prospect, Turkey.*

Koza Altın İşletmeleri A.Ş. (Fall 2011c), *Mineral Resource Estimate for the Mermerlik Prospect, Turkey.*

Koza Altın İşletmeleri A.Ş. (2012e) *Kaymaz Project Slope Stability.*

Koza Altın İşletmeleri A.Ş. (2012f) GIS Database.

Okay, A.I. and Göncüoğlu, C., 2004, *The Karakaya Complex: A Review of Data and Concepts*, in: Turkish Journal of Earth Sciences, V.13, pp.77-95.

Okay, A. I. (2008), *Geology of Turkey: A Synopsis,* Anschnitt, Vol. 21, pp 19-42.

Okay, A.I. and Whitney, D.L., 2010, Blueschists, ophiolites, and suture zones in northwest Turkey, in: Tectonic Crossroads: Evolving Orogens of Eurasia-Africa-Arabia October 4-8, 2010, Ankara, Turkey Conference Paper, 54p.

SRK, 2012, Audit 2013 Volume 4 Kaymaz, Including Söğüt Resources and Reserves Koza Altın İşletmeleri A.Ş. Turkey, 122 p.

# 5 Glossary

## 5.1 Mineral Resources and Reserves

The JORC Code 2012 was used in this report to define resources and reserves.

A ‘Mineral Resource’ is a concentration or occurrence of material of intrinsic economic interest in or on the Earth’s crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.

An ‘Inferred Mineral Resource’ is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes which may be limited or of uncertain quality and reliability.

An ‘Indicated Mineral Resource’ is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.

A ‘Measured Mineral Resource’ is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes. The locations are spaced closely enough to confirm geological and grade continuity.

## 5.2 Glossary of Terms

### Table 6.2.1: Glossary

| Term | Definition |
|---|---|
| Assay | The chemical analysis of mineral samples to determine the metal content. |
| Capital Expenditure | All other expenditures not classified as operating costs. |
| Composite | Combining more than one sample result to give an average result over a larger distance. |
| Concentrate | A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore. |
| Crushing | Initial process of reducing ore particle size to render it more amenable for further processing. |
| Cutoff Grade | The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration. |
| Dilution | Waste, which is unavoidably mined with ore. |
| Dip | Angle of inclination of a geological feature/rock from the horizontal. |
| Fault | The surface of a fracture along which movement has occurred. |
| Flitch | Mining horizon within a bench. Basis of Selective Mining Unit and excavator dig depth. |
| Footwall | The underlying side of an orebody or stope. |
| Grade | The measure of concentration of gold within mineralized rock. |
| Haulage | A horizontal underground excavation which is used to transport mined ore. |
| Igneous | Primary crystalline rock formed by the solidification of magma. |
| Kriging | An interpolation method of assigning values from samples to blocks that minimizes the estimation error. |
| Level | Horizontal tunnel the primary purpose is the transportation of personnel and materials. |
| Milling | A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product. |
| Mining Assets | The Material Properties and Significant Exploration Properties. |
| SAG Mill | Semi-autogenous grinding mill, a rotating mill similar to a ball mill that utilizes the feed rock material as the primary grinding media. |
| Sedimentary | Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks. |
| Sill | A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness. |
| Smelting | A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase. |
| Spigotted | Tap/valve for controlling the release of tailings. |
| Stope | Underground void created by mining. |
| Strike | Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction. |
| Sulfide | A sulfur bearing mineral. |
| Tailings | Finely ground waste rock from which valuable minerals or metals have been extracted. |
| Thickening | The process of concentrating solid particles in suspension. |
| Variogram | A statistical representation of the characteristics (usually grade). |

# 6 Date and Signature Page

Signed on this 31st Day of January, 2015.

### Endorsed by CPs:

This signature was scanned for the exclusive use in this document with the author's approval; any other use is not authorized.

Leah Mach, MSc Geology, CPG

This signature was scanned for the exclusive use in this document with the author's approval; any other use is not authorized.

Dorinda Bair, BSc Geology, CPG

This signature was scanned for the exclusive use in this document with the author's approval; any other use is not authorized.

Bret Swanson, BEng Mining, MAusIMM, MMSAQP

This signature was scanned for the exclusive use in this document with the author's approval; any other use is not authorized.

Eric Olin, MSc, MBA, SME-RM

### Reviewed by:

This signature was scanned for the exclusive use in this document with the author's approval; any other use is not authorized.

Peter Clarke, BSc Mining, MBA, PEng

This signature was scanned for the exclusive use in this document with the author's approval; any other use is not authorized.

Bart Stryhas, PhD. CPG

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 industry practices.

## Disclaimer

The opinions expressed in this Report have been based on the information supplied to SRK Consulting (U.S.), Inc. (SRK) by Koza Altın İşletmeleri A.Ş. (Koza). These opinions are provided in response to a specific request from Koza to do so, and are subject to the contractual terms between SRK and Koza. SRK has exercised all due care in reviewing the supplied information. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information and does not accept any consequential liability arising from commercial decisions or actions resulting from them. Opinions presented in this report apply to the site conditions and features as they existed at the time of SRK's investigations, and those reasonably foreseeable. These opinions do not necessarily apply to conditions and features that may arise after the date of this Report.

## Copyright