Audit 2014
Volume 2 Ovacık District
Koza Altın İşletmeleri A.Ş.
Turkey

Report Prepared for

Koza Altın İşletmeleri A.Ş.

Report Prepared by

srk consulting

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

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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 spectography
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

Abbreviation	Unit or Term
Mm	million meter
Moz	million ounces
Mt	million tonnes
Mt/y	million tonnes per year
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, 2014. 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 2 Ovacık 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).

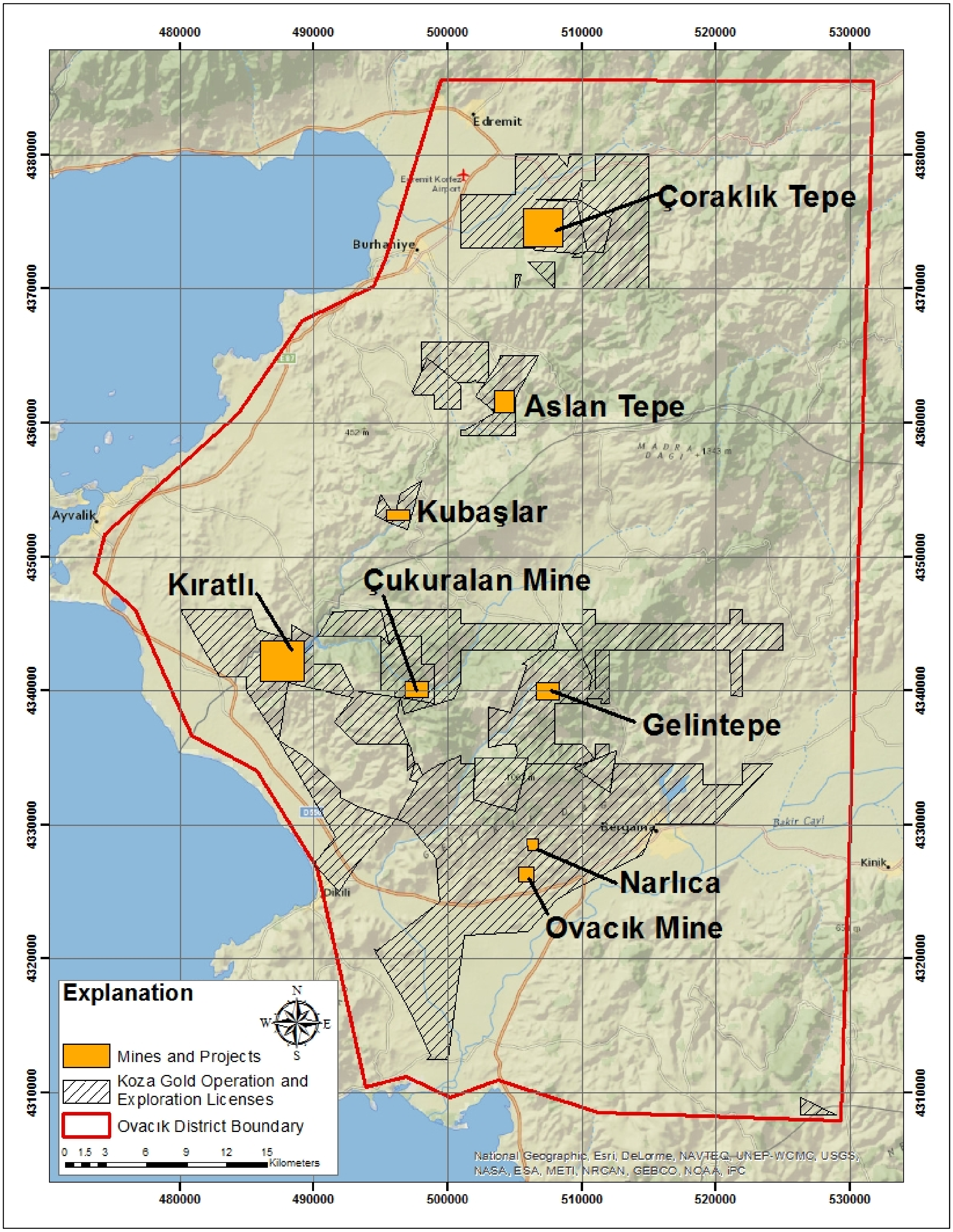
Volume I Executive Summary contains the Terms of Reference and Property Descriptions relevant to all volumes of this audit.

1.1 Ovacık District

The Ovacık District includes the Ovacık and Çukuralan Mines and the Kıratlı, Kubaşlar, Aslantepe, Gelintepe and Narlıca advanced exploration projects. The Çoraklıktepe open pit mine was completed in 2014 and the resource and reserve has been depleted. The climate, physiology and regional geology of these mines and projects are discussed in this section of Volume 2. The Location of the Ovacık District is shown in Figure 1.1.1. Individual project locations within the Ovacık District are shown in Figure 1.1.2. Specific locations and access are discussed individually by mine or project.

Source: Basemap = ESRI Basemap NatGeo_World_Map, 2013

Figure 1.1.1: Location of the Ovacık District



Source: Koza, 2012 GIS; Basemap = ESRI Basemap NatGeo_World_Map, 2013

Figure 1.1.2: Individual Project Locations within the Ovacık District

1.1.1 Climate and Physiography of the Ovacık District

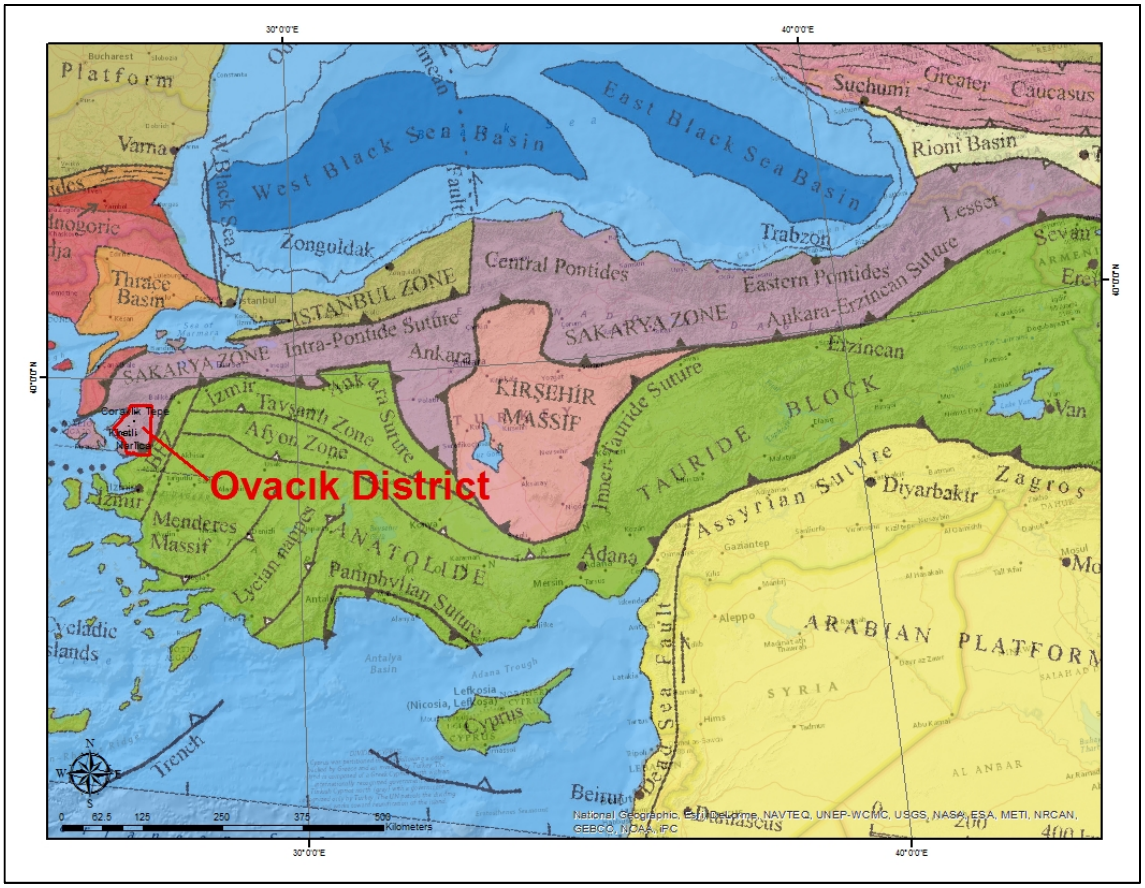
The Ovacık District is located between Bergama and Havran and extends from the Aegean Sea eastward into the adjacent mountains. Near the coast, the regional weather is typical of a Mediterranean climate, characterized by hot, dry summer months and warm, wet winter months. Frost and snow rarely occur in this region. Mediterranean climate effects are observed inland up to elevations of 800 m above mean sea level (masl); the climate becomes more continental further inland. The coastal mountains here reach an elevation of 1,000 m amsl.

The hottest and coldest months are July and January, respectively. The maximum and minimum temperatures recorded in Bergama are 45°C (July 2000) and –11.4°C (January 1968), respectively. The maximum and minimum temperatures recorded at Dikili, on the coast, are 41.8°C (July 1987) and –8.6°C (January 1942), respectively. The yearly average temperature is 16.2°C in Bergama and 16.4°C in Dikili. The annual precipitation is distributed equally between spring and autumn. Precipitation is normally in the form of rain and the annual average precipitation is 646.2 mm inland near Ovacık Mine and 629.2 mm at the coast.

The terrain in the Ovacık District is flat to rolling hills near the Aegean Sea, rising quickly to approximately 500 m amsl at the Çukuralan Mine, Kıratlı, Kubaşlar, and Aslantepe. Gelintepe is at the highest elevation between 800 and 900 m and the Ovacık Mine, Narlıca and Çoraklıktepe are at the lowest elevations ranging from 80 to 210 m. The relief at the project areas near the coast is primarily low rolling hills while the relief at the inland projects is moderate to steep.

1.1.2 Regional Geology of the Ovacık District

The Ovacık District is located in the Western Anatolian Extensional Tectonic Province in a belt of low and high sulfidation epithermal deposits. This belt extends from north central Turkey to the Aegean Sea, and straddles the İzmir-Ankara Suture, which formed during Cretaceous age as a result of collision and subsequent subduction of the Anatolide-Tauride block beneath the Sakarya Terrane during closure of the Tethyan Sea. This was followed by two periods of rift related extension caused by a change in plate motion and resulting development of NNE-SSW and NE-SW trending grabens. The Ovacık District is located in the Sakarya Terrane, north of the İzmir-Ankara Suture Zone. Deposits within this zone are commonly associated with Paleogene and Neogene age volcanism and Upper Mesozoic to Tertiary age intrusive events (Yilmaz, 2002; Okay et al., 2004; Okay, 2008). Figure 1.1.2.1 shows the position of the Ovacık District in the Sakarya Terrane.



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

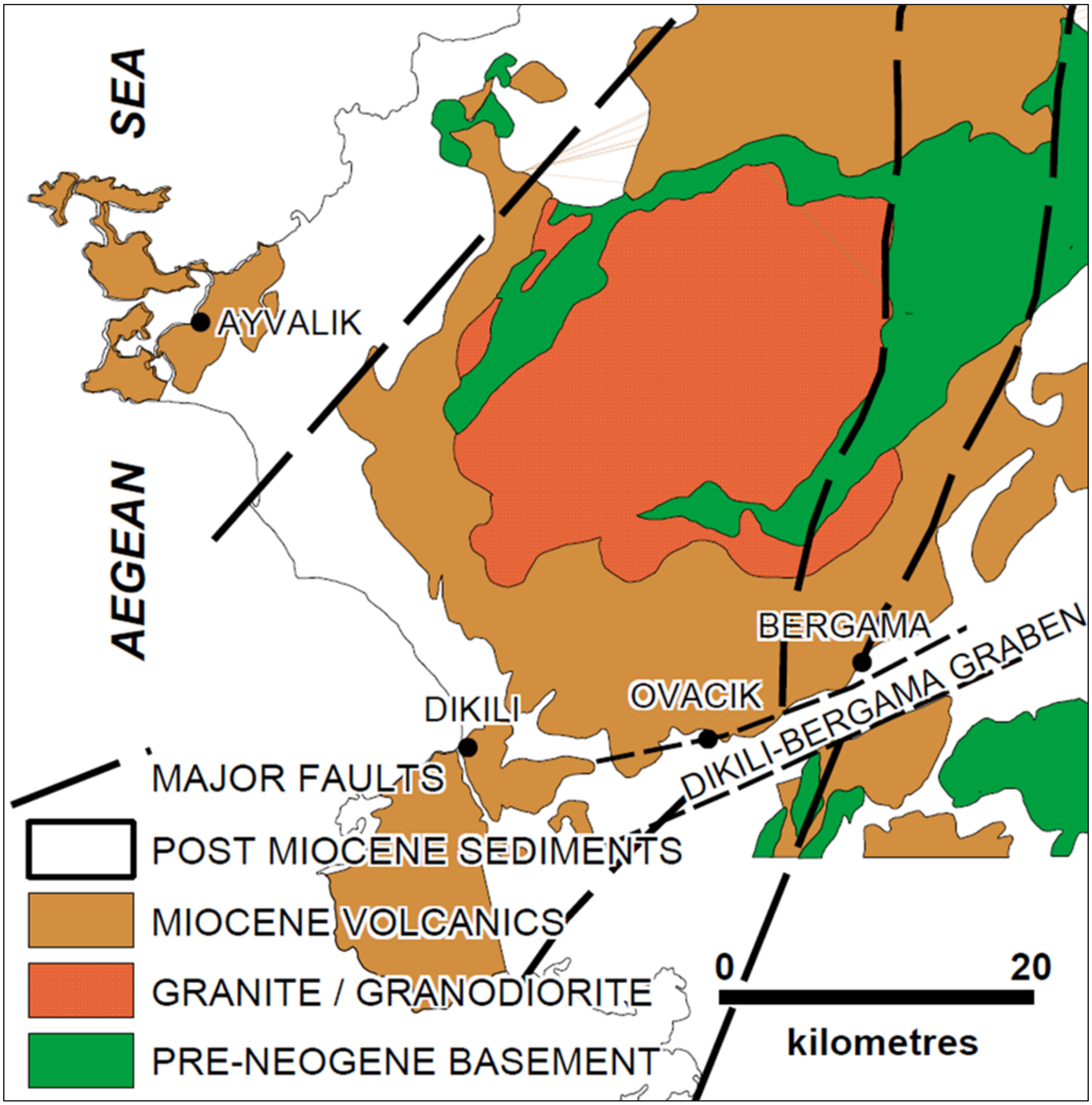
Figure 1.1.2.1: Ovacık District Relative to the Terrane Map of Turkey

The regional geology of the Ovacık District is described by Koza as an area underlain by the Triassic-age Karakaya Metamorphic Complex represented locally by the Kinik Formation. This metamorphic complex was intruded during the Oligocene and Miocene by the Kozak Magmatic Complex. In places, both the Kozak Magmatic and Karakaya Metamorphic complexes have been overlain by Miocene age volcanic rocks and Quaternary alluvium (Kara, 2004).

The Kinik Formation consists of conglomerate, sandstone, siltstone, limestone, metasediments and metavolcanics. The rocks of the Kozak Magmatic Complex include plutonic, hypabyssal and volcanic rocks thought to be related to caldera formation and collapse with the subsequent emplacement of a shallow level granitic intrusion. To the south of the district are Permian to Triassic age intrusions of granite and granodiorite rocks, which are correlative to the Karakaya complex. The Miocene age volcanic rocks are divided into two stages. The first stage is commonly porphyritic andesite and second stage is andesitic lava with pyroclastics associated with volcanic necks and felsic dikes. Numerous hot springs can be found throughout the district.

There are several regional fault zones in the area related to the middle to late Miocene age extensional event and include a series of NNE-SSW to NE-SW trending grabens and normal faults that become a well-developed horst and graben system farther to the south. Many of the normal faults show secondary sinistral, strike-slip movement. There are two different types of epithermal systems in the district, which are high and low sulfidation. Epithermal low sulfidation systems are related to late stage volcanism and extension, and high sulfidation is related with early stage volcanic phases and may be more proximal to the intrusive center.

The Ovacık Mine is located on the north side of the Dikili-Bergama Graben and quartz veins at the mine are oriented sub-parallel to this structure. At Ovacık, the veins are orientated N60-85°E (Kara, 2004). Figure 1.1.2.2 shows the regional geology of the Ovacık District.



Source: Koza, 2013

Figure 1.1.2.2: Regional Geology of Ovacık District

1.1.3 Exploration

The Ovacık District includes the Ovacık and Çukuralan Mines, the Kubaşlar reserve area and the Aslantepe, Gelintepe, Kıratlı and the Narlıca resource areas. The Çoraklıktepe Mine was completed in 2014 and the resource and reserve have been depleted. The resource and reserve projects are within a 50 km radius of the Ovacık Mine, which has been used by Koza as the exploration model within the district. Koza has expanded its exploration model to include those deposit types consistent with island arc volcanics and extension zones in association with porphyry intrusions and has identified variations on the epithermal model within the district. This model presents additional geological targets for exploration in the Ovacık District.

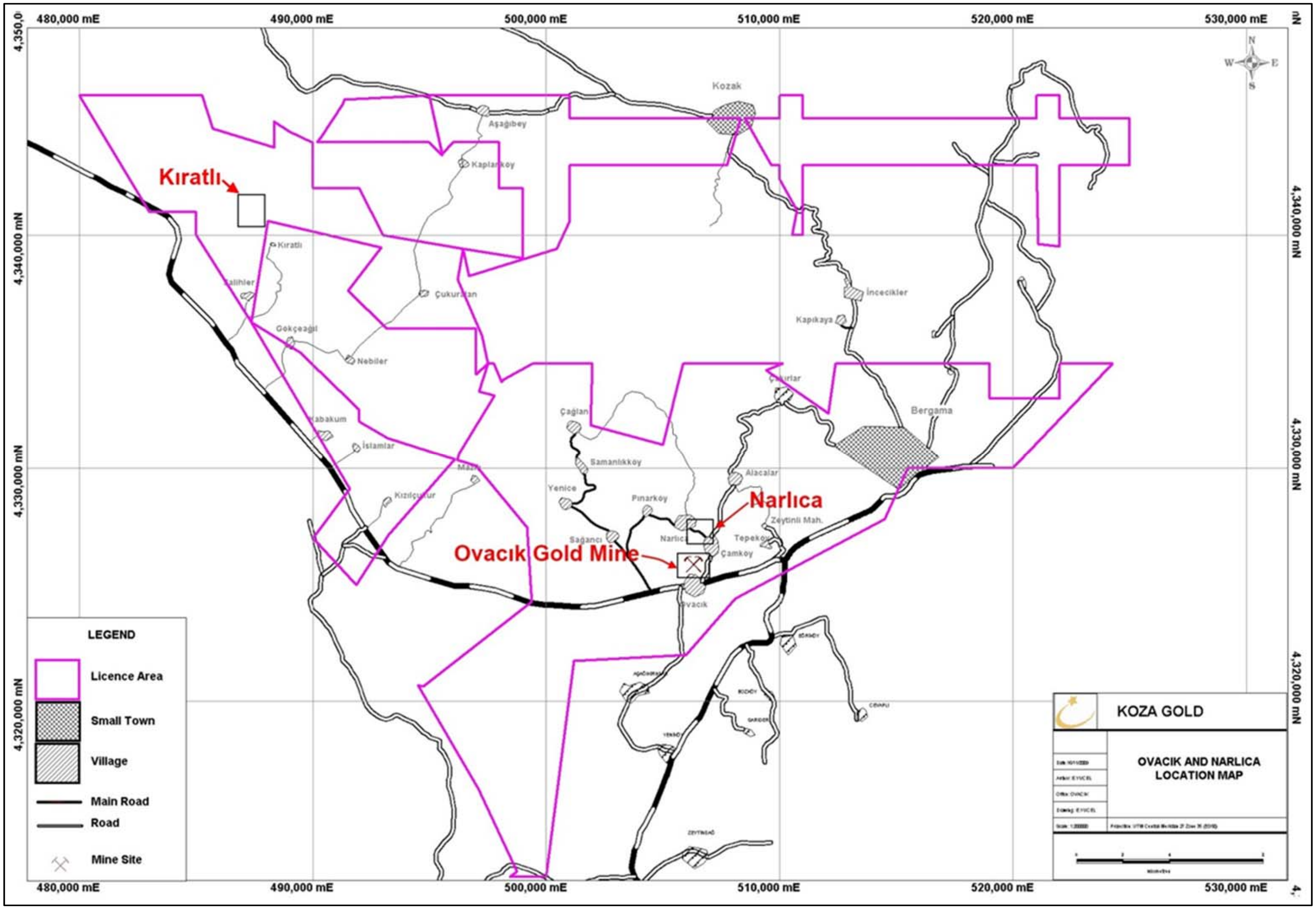
Koza uses industry best practice in its exploration work. Within the exploration team, there is an established, logical progression of steps that are used at each project using a standard set of procedures. This progression begins with identification of the target area and mapping at ever increasing detail. In tandem with this, Koza incorporates stream sediment, chip channel and soil sampling to better define a target for drilling. Koza also uses any geophysics tools at its disposal, including IP, resistivity and magnetic surveys and Portable Infrared Mineral Analyzer (PIMA) mapping for alteration. Once drilling begins, Koza continues to use industry best practice in its chain of custody, core logging, core photography, sample collection, sample submission, Quality Assurance/Quality Control (QA/QC) and database management.

2 Ovacık Mine Resources and Reserves

2.1 Property Description and Location

The Ovacık Mine is located near Ovacık Village in western Turkey, approximately 100 km north of İzmir, 15 km east of Dikili and 8 km west of Bergama. The mine is north of, and accessed from, a divided highway connecting İzmir and Çanakkale called the İzmir–Çanakkale Highway (D550). Access is along a mine road immediately north of the highway. Ovacık is located between Universal Transverse Mercator (UTM) coordinates 4,326,000 N, 506,000 E to 4,327,000 N, 507,000 E in European Datum (ED) 1950 Zone 35.

Koza has extensive land holdings in the Ovacık District in both operating and exploration licenses. Ovacık Mine is currently under operation license number 18201, which comprises approximately 41,515 ha. Both Narlica and the Kıratlı exploration projects lie within this operating license. Koza holds two mining permits covering the same area as the operation license. These two operation permits are for gold and silver. Land tenure for the Ovacık Mine and properties within operating license 18201 are shown in Figure 2.1.1.



Source: Koza, 2013 GIS

Figure 2.1.1: Ovacık and Narlıca Location Map

2.2 History

The Ovacık Mine area was first explored in 1989 by Eurogold Madencilik A.Ş. (Eurogold), a joint venture company formed between Australian Consolidated Minerals Gold Ltd (ACM) and Canadian Metal Mining Ltd. Later, ACM sold its shares in the joint venture to Normandy Mining in Australia and the company name was changed to Normandy Madencilik A.Ş. (Normandy). In 2002, Newmont Mining Ltd, Normandy Mining Ltd and Franco Nevada merged and retained the name Normandy Madencilik A.Ş. Normandy held the property until 2005 when Koza acquired the Ovacık Mine.

Historic pits and regional stream sediment sampling led to the discovery of the Ovacık deposit in 1989. Mineral potential was confirmed by diamond drilling, and a feasibility study was completed in 1991 and approved in 1996. Construction of the mine facilities and infrastructure was completed in 1997. Between 1997 and 2001, the Turkish Supreme Court stopped progress due to environmental concerns voiced by local and international groups. The Turkish government commissioned a separate review from a Turkish scientific institute and the Ministry of the Environment, which found that risks had been reduced or eliminated and work was resumed at the site in 2001 and mining started the same year. However, the Project was shut down by a court ordered injunction in 2002 for just one day and operations resumed the following day. This injunction was overturned in December 2002 by the court and the formal permitting process was restarted (Clow and Wahl, 2004).

Koza acquired Ovacık in 2005. When Koza acquired the mine, the operation was idle and experiencing resistance for continued production from the local communities. Koza put the mine back into production and improved local community relations. The mine has been in continuous operation since 2007.

Production has come from open pit and underground sources. The open pit operations were completed in October 2007 and all ore is currently being produced from underground. Koza is exploring extensions of the Ovacık mineralized system.

The Ovacık Mill processes ore from the Ovacık and Çukuralan Mines and stockpiles from the Küçükdere and Çoraklıktepe Mines, which are now closed. Koza plans to continue to use the Ovacık Mill as the processing hub for all of the mines developed in the Ovacık District.

To date, approximately 2.0 Moz of gold and 1.5 Moz of silver have been processed from 7.8 Mt of ore at the Ovacık Mill from Koza's mining operations in the district. The majority of the production has been from the Ovacık Mine with inputs from the Küçükdere, Çukuralan and Çoraklıktepe Mines. Historic production at the Ovacık Mill by year from 2001 through 2014 is listed in Table 2.2.1 with ore from Küçükdere included in years 2006 through 2010, Çukuralan included in 2011 through 2014 and Çoraklıktepe in 2013 through 2014.

Table 2.2.1: Historic Production at the Ovacık Mill, including Ovacık, Küçükdere, Çukuralan and Çoraklıktepe Ore

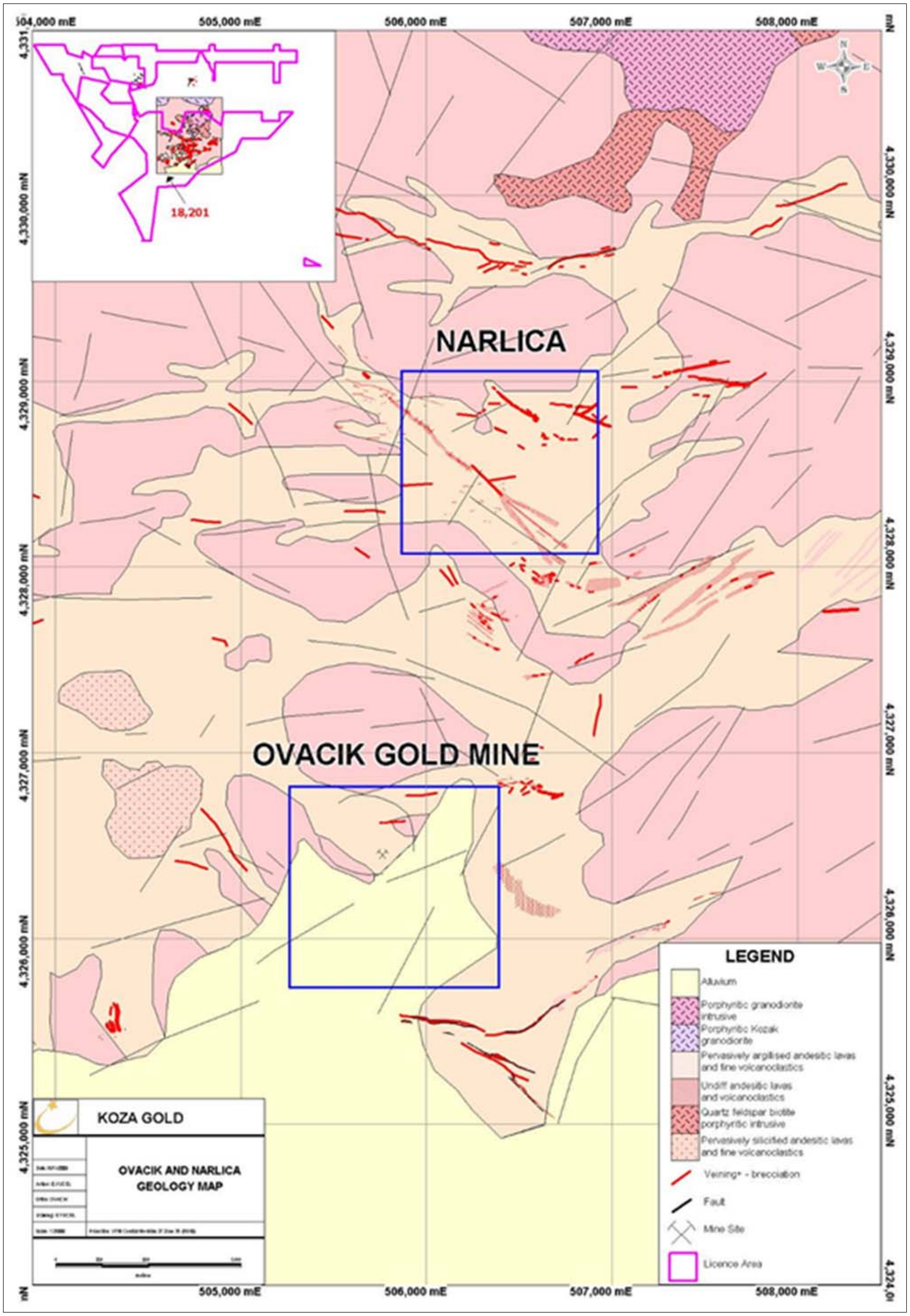
Operator	Year	Tonnes	g/t Au	g/t Ag	oz Au	oz Ag
Normandy	2001	153,283	9.93	13.74	45,862	53,813
Normandy	2002	373,768	12.28	18,43	138,319	182,708
Normandy	2003	483,967	11.75	12.97	172,325	159,595
Normandy	2004	299,869	10.97	11.05	100,912	82,778
Koza	2005	277,889	15.68	12.57	134,959	82,806
Koza	2006	587,521	10.37	10.26	187,171	131,805
Koza	2007	658,050	9.14	8.38	187,372	116,788
Koza	2008	758,382	7.18	11.57	167,059	170,330
Koza	2009	808,136	5.74	11.93	140,485	175,197
Koza	2010	827,498	4.81	6.47	120,577	86,651
Koza	2011	832,777	5.01	3.38	124,635	52,936
Koza	2012	876,185	4.67	2.80	125,127	49,912
Koza	2013	879,411	5.80	3.82	156,492	67,664
Koza	2014	866,867	7.71	4.56	203,962	81,016

Source: Koza, 2014

2.3 Ovacık Mine Geology

2.3.1 Local Geology of the Ovacık Mine

The Ovacık deposit is a high-angle, low sulfidation, epithermal vein deposit composed of quartz-adularia vein zones with breccia, colloform/crustiform banding and carbonate replacement textures. Four vein systems have been identified. The two main vein systems are the M and S Veins. The D and G Veins are subordinate systems. The S, D and G are single veins, while the M Vein has been offset by faults and exhibits splays and brecciation in the hangingwall and footwall. Near the top of the system, a late stage hematitic, quartz breccia is observed around the quartz vein zone. Deeper in the system, hydrothermal quartz and breccias with pyrite, silica and clay matrix are observed adjacent to the main vein system. The entire vein system is hosted by Miocene, porphyritic andesite, which has been brecciated near the vein zone. This andesite breccia has a silica-clay matrix. Alteration adjacent to the brecciation zone includes silica flooding of the wallrock with a larger argillic alteration halo grading out to unaltered host rock. The Ovacık vein system is oriented approximately N60-85°W with the exception of the G Vein, which is oriented N40°E. Mining activity has been centered on the M and S Veins both in the pit and underground while the G Vein has been the focus of exploration. Sulfide mineralization includes pyrite, chalcopyrite, galena and sphalerite. Figure 2.3.1.1 shows the local geology of the Ovacık Mine and adjacent Narlıca Project.



Source: Koza, 2013 GIS

Figure 2.3.1.1: Ovacık and Narlıca Geology Map

2.4 Exploration

The Ovacık Mine and Narlıca mineralization are low sulfidation epithermal deposits. Koza is using this type of deposit as an exploration model in this area of the Ovacık District. Exploration is primarily in-fill and step-out drilling along strike and down dip of the current mining operations. Koza has budgeted TL1.5 million (US$687,000) for the 2015 exploration efforts for projects in the Ovacık Mine concession.

2.5 Drilling/Sampling Procedures

The surface drilling on the Project consists predominately of HQ or PQ sized core, and the underground drilling consists of HQ and BQ sized core. Most of the surface holes were pre-collared to a depth of approximately 3 m prior to coring. Koza has also undertaken underground drilling and underground grade control sampling on the M and S veins. All samples are used in resource estimation. Table 2.5.1 summarizes the drilling and sampling on the M and S veins at Ovacık and Figure 2.5.1 shows the drillholes in plan view.

Table 2.5.1: Summary of Drilling and Face Samples at the Ovacık Mine

Type	Number	Meters		
		Grade Control	RC	Core
M Vein - Surface	328		214.6	61169.1
S Vein - Surface	135		155	18960.4
M Vein - Underground	297			9401.8
S Vein - Underground	61			889.25
Total Drillholes	821		370	90,421
M Vein - Face Samples	3,005	14,682		
S Vein Face Samples	1,899	9,094		
Total Face Samples	**4,904**	**23,776**		

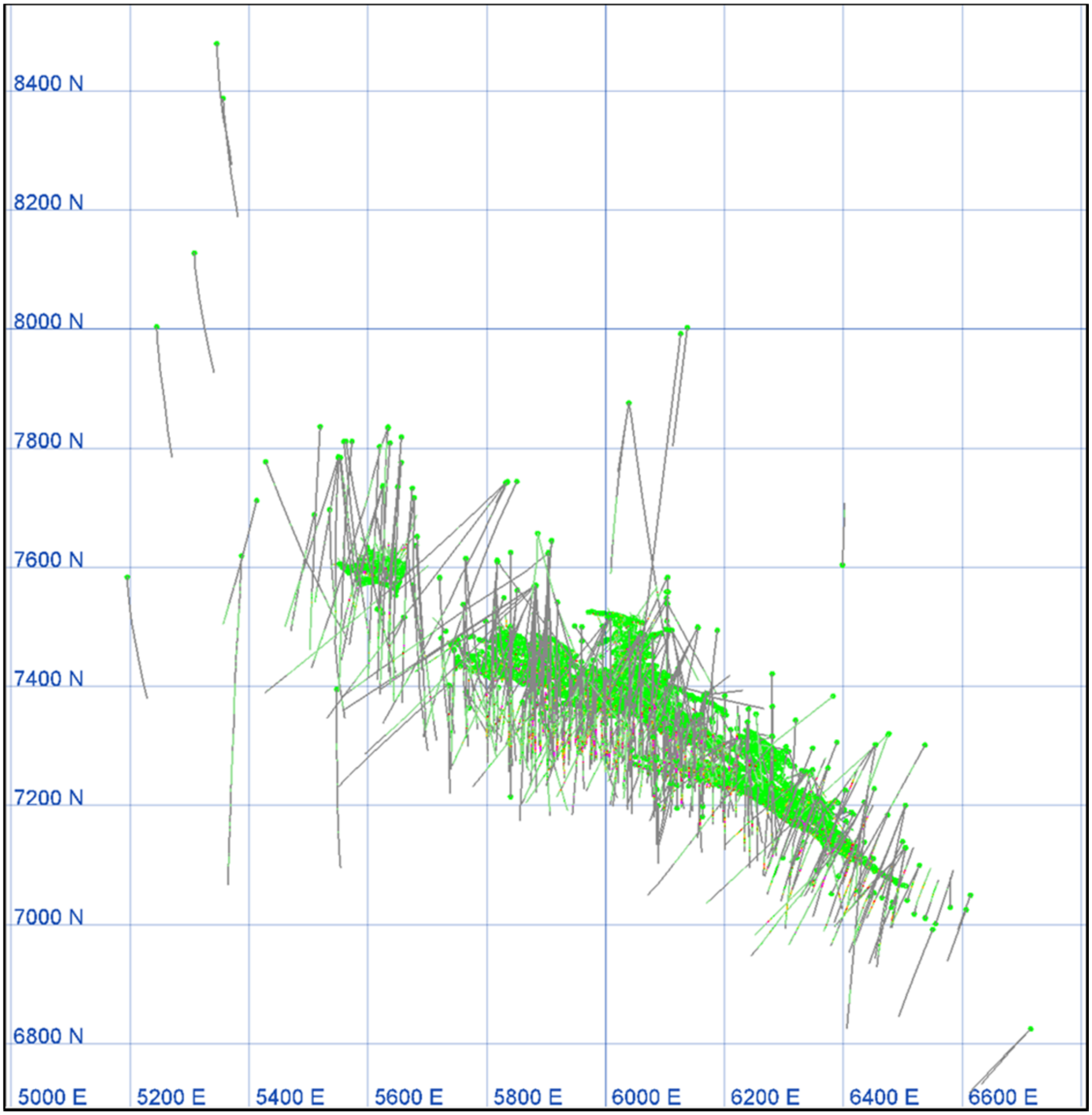


Figure 2.5.1: Ovacık Drillhole Location Map

The surface holes are generally drilled to the south with inclinations of about 60°, nearly perpendicular to the steeply dipping veins. Downhole surveys were taken with a single shot Eastman camera or a multi-shot Flexit tool. The collars were surveyed by the company survey team.

The underground drilling is conducted by Koza with its own drills. The holes are mainly horizontal, but because they are generally less than 10 m in length, they do not require downhole surveys. The collar locations are picked up by the underground survey crew.

The underground grade control samples are collected as horizontal channel samples at a height of 1 m. The samples are taken at 1 m intervals within the channel.

The core is sampled on 1 m intervals with adjustments at geologic boundaries. The sampling is based on the presence of quartz, with visually barren zones unsampled. Sufficient host rock is sampled to ensure that a proper geologic model can be prepared. Core is split in half lengthwise with a core saw or by hand with a flat blade if it is soft or broken.

The core is stored in wooden boxes within a secure yard. Much of the historic core has been destroyed due to poor practices in the past; however, photographs of the core remain.

The samples from the historic drilling were analyzed at Caleb Brett Laboratories in the UK, with a small amount of check analyses at Rapley Wilkinson in Australia, SGS Canada, and Knights in the UK. Eurogold later established a sample prep facility in İzmir to produce a pulp sample for shipment to the UK. When operations started, ALS Chemex in Canada (ALS Canada) was under contract for assaying services at the mine site. Subsequently, Koza has undertaken the sample prep and assaying at the mine lab.

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.

Bulk density measurements were taken on 103 quartz vein and andesite samples. The procedure included wrapping the samples in plastic and weighing in air and water. This work was completed by the Koza geology department. The resource estimate is based on the average of the 103 samples, 2.2 g/cm^3, for all mineralized blocks.

Koza's drilling and sampling programs are well conceived and executed, and follow industry best practices.

2.5.1 Quality Assurance and Quality Control

Koza has a laboratory QA/QC program in place which consists of:

- Reference Material samples;
- Blanks; and
- Preparation Duplicates.

The Koza QA/QC procedures were reviewed by Lynda Bloom, Principal at Analytical Solutions LTD in 2013 (Bloom, 2013).

Insertion of Internal Controls

Koza inserts QA/QC control samples into the sample stream at approximately one blank per drillhole, 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 inserted into the sample stream in sampling numbering sequence by the core logging geologist. The location of the control samples is noted on the sample log and in the sample database. Internal control samples have the same numbering system as the drill core samples.

All Ovacık control samples have been monitored for Au. The Ag results for the control samples were not provided to SRK and it does not appear that the Company is currently monitoring the Ag control samples data. Because Ag is included in the resource, SRK recommends Koza also monitor Ag results.

Reference Materials

For 2013, two different RMs have been used at Ovacık; both site-specific RMs produced with material from the Ovacık Mine. 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 for Au and Ag at its Vancouver, Johannesburg and Lima laboratories (ten at each lab). ALS provided a report with summary statistics for each of the RMs. For all RMs, Koza uses a performance range of ±10% of the mean. For these RMs, Bloom (2014) recommends using 7% as a threshold for a failure based on her communication with ALS.

Table 2.5.1.1 presents the expected mean, standard deviations and summaries of the analyses of the Au RMs. It should be noted that the standard deviation listed in the ALS report is the within-lab standard deviation which is the standard deviation of the three average values from the three labs. Although this is the standard method recommended for establishing the standard deviation of the RM, during the certification process, 15 labs are used, not just three and the standard deviation would presumably be higher. Because of the method used to produce these materials, they are not considered certified.

Table 2.5.1.1: Results of 2013 Au RM Analyses at Ovacık

CRM	Number of Samples	Expected (ppm)		Observed (ppm)		% of Expected	2 SD		3 SD	
		Mean	Std Dev	Mean	Std Dev		No. Failures	% Failure Rate	No. Failures	% Failure Rate
OV23	8	1.395	0.020	1.433	0.02	102.7	4	50.0	1	12.5
OV24	14	1.799	0.064	1.81	0.02	100.6	0	0.0	0	0.0
Total	**22**						**4**	**18.2**	**1**	**4.5**

Using the 7% threshold, there are no failures for OV23 or OV24. The OV23 results are all greater than the expected value, but within the 7% threshold. Five of the 14 OV24 results are below the expected value and nine are above.

The cutoff grade at Ovacık is 1.65 g/t Au and the average grade is over 5 g/t. Current RMs used at Ovacık represent grades near the cutoff grade. SRK recommends Koza include additional RMs with grades that represent a wider range of the resource grade, including the average grade and at the 75^{th} percentile.

Bloom (2013) has made the following recommendations regarding Reference Material samples:

- Use commercially available Certified Reference Materials (CRM)s for common deposit types (i.e., there are many available CRMs for high and low sulfidation systems);
- Purchase appropriate equipment for pulverizing, mixing and sub-sampling large batches (>500 kg); or

- Prepare a custom standard at a facility that specializes in CRM production, such as OREAS that can prepare up to 5,000 kg at one time. Costs are usually half of the cost of purchasing pre-packaged commercial RMs.

The current practice of preparing 100 kg with inadequate round robin data is not recommended.

Blanks

Sample blanks test for contamination in preparation and assaying and handling errors. Koza inserted one sample blank per drillhole using pulp blanks up until June 2012 and preparation blanks since then. A blank failure is a result greater than five times the detection limit. SRK has examined the results for gold in the blank samples and finds that there were seven Ovacık samples submitted with zero failures. The results indicate that the preparation laboratory is generally performing well.

There was a limited number of blank data for Ovacık in 2013. The seven analyses do not provide a sufficient number of analytical results to properly assess performance. SRK recommends Koza continue inserting blanks so that a more statistically valid number of data becomes available.

Preparation Duplicates

Preparation duplicates are created by splitting a second cut of the crushed sample (coarse reject) in the same way and for 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.

Koza sent preparation duplicates to ALS Chemex, the primary lab, for analysis. The duplicate analysis data provided to SRK includes 12 duplicate pairs with Au. After filtering out pairs with at least one value less than detection limit, three duplicate pairs were available for QA/QC review.

A summary of the analytical results are presented in Table 2.5.1.2.

Table 2.5.1.2: Summary of Preparation Duplicate Au Analysis at Ovacık

Criteria	Number of Samples	Original>Dup	Dup>Original	Original = Dup	Within +/- 20%
All samples	3	0	2	1	3
		0%	67%	33%	100%

All duplicate results were within 20% of the original value.

For a mature mine such as Ovacık, it may not be necessary to continue using preparation duplicates. Instead, Koza could request that the results from the duplicates routinely prepared and analyzed by Koza be made available to them.

Pulp Duplicates

Koza has not submitted any pulp duplicate samples to the primary lab. Pulp duplicates are the primary method of checking the precision of analysis. SRK recommends that the Company monitor the internal pulp duplicates produced and analyzed by its primary lab.

Secondary Check Lab Analysis

Koza has not sent any check samples in the form of pulps from the Ovacık projects to a secondary laboratory for check analysis. SRK recommends that Koza add this type of QA/QC samples to its program. Check assays should be done on the original pulp and must be analyzed at the secondary laboratory, using the same method as the primary lab, and RMs must be submitted with the pulps.

Conclusions and Recommendations

Koza monitors QA/QC of the laboratory analyses by inserting internal control samples into the sample stream. Reference materials, blanks, preparation duplicates and secondary check lab analyses 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 entire batch be reanalyzed. This is industry best practice.

SRK has the following recommendations:

- Ag results for the RMs, blanks and duplicates should be monitored;
- The use of the OV23 and OV24 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;
- Include additional CRMs with grades that represent a wider range of the resource grade;
- The blank samples should all be preparation blanks, not pulp blanks, and more should be inserted;
- Duplicate samples should be within the resource grade range;
- Pulp duplicates prepared and analyzed by the primary lab as part of its internal QAQC program should be monitored; and
- Submit original pulps to a secondary laboratory for check assay.

Overall the laboratory is performing well and, although the QA/QC program needs improvement, it is sufficiently monitoring laboratory accuracy and reliability.

2.6 Mineral Resources

The mineral resources were estimated in 2014 by Koza (Koza, 2014a). The estimation methodology follows Newmont's procedures set in 2004 (Kara, 2004).

2.6.1 Geological Modeling

The geologic model is prepared using all mapping, underground sampling, and drill data. In the S Vein, it appears that most mineralization is contained within the main quartz vein and brecciated vein structure. In the M Vein, mineralization also exists in the andesite and breccia zones in the hanging wall immediately adjacent to the vein. High angle faults cut both veins and terminate mineralization. Three dimensional solids of the S and M Veins were constructed from the drillholes at a cutoff grade of 0.5 g/t Au. The M Vein consists of five zones separated by faults. M Veins 1, 2 and 3 have been combined into a single wireframe; M123, and M4 and M5 are the combined zones. There are also six mineralized zones (Zones 1 to 6) in the andesite in the hangingwall and the footwall of the M vein. Zone 4 has been separated into an Upper and Lower Zone. There are eleven mineralized domains as shown below:

- M Vein – 1, 2, 3 (combined into M123), M4, M5, Zones 1 through 6, including Zone 4 Upper and Lower; and

- S Vein.

Together, the wireframes cover a strike length of about 1,100 m and extend about 260 m vertically below the open pit.

In addition, lower grade zones have been constructed around the M123, M4 and M5 Veins and the S Vein. The high grade cores of the veins are categorized as vein type1 and the surrounding lower grade mineralization is categorized as vein type2. Figure 2.6.1.1 is a plan view of the wireframes of the M and S Veins and Zones 4 Upper and Lower. Figure 2.6.1.2 is a long-section of the hangingwall showing drilling and the M and S Veins and Zones 1, 2, 3 and 4. Figure 2.6.1.3 is a long-section of the footwall showing drilling and the M and S Veins and Zones 1, 4, 5 and 6. Figure 2.6.1.4 shows the underground samples in the M and S Veins. Table 2.6.1.1 summarizes basic statistics of the drill samples and Table 2.6.1.2 summarizes statistics of the grade control samples.

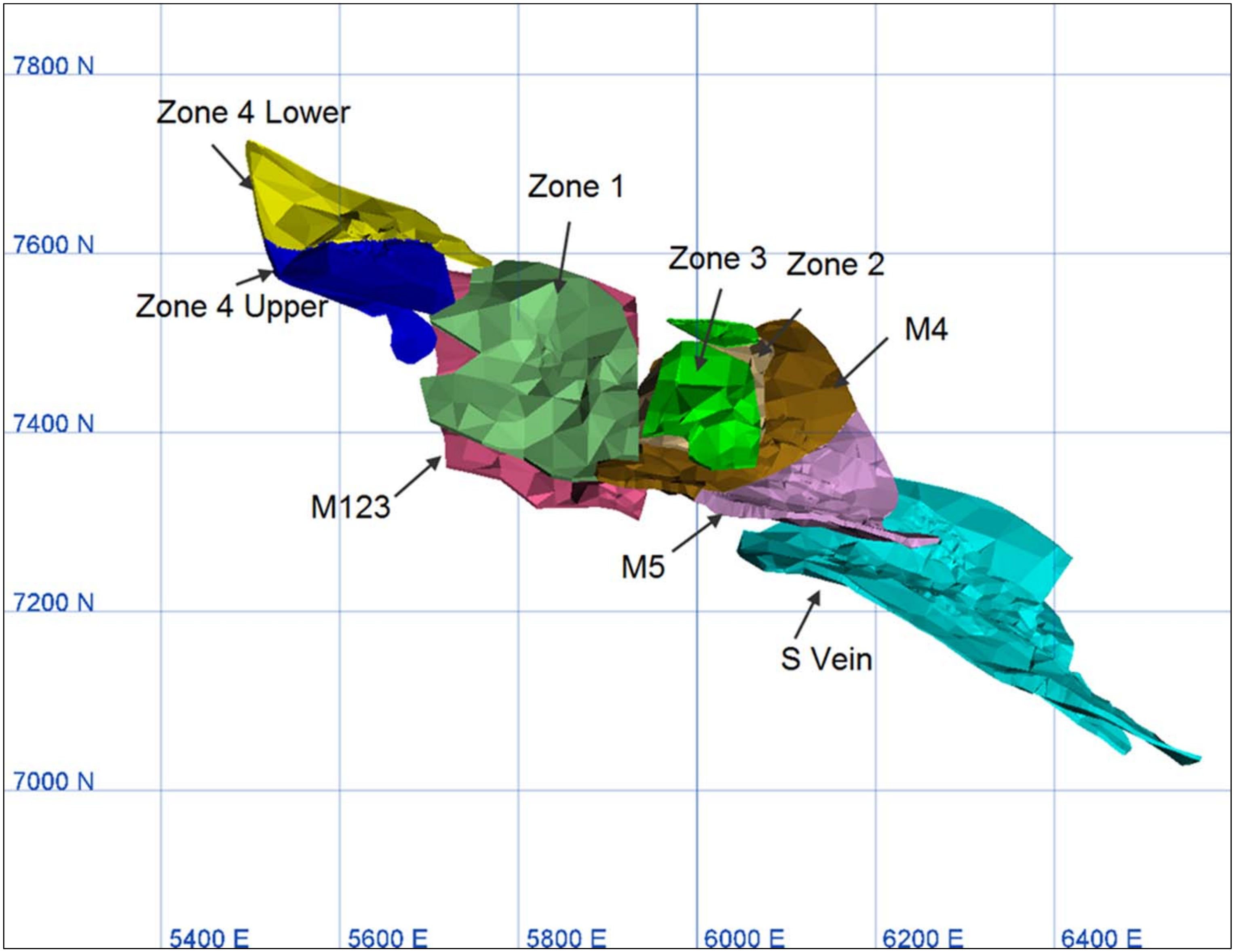


Figure 2.6.1.1: Ovacık Domains Defined by Vein Wireframes in Plan View

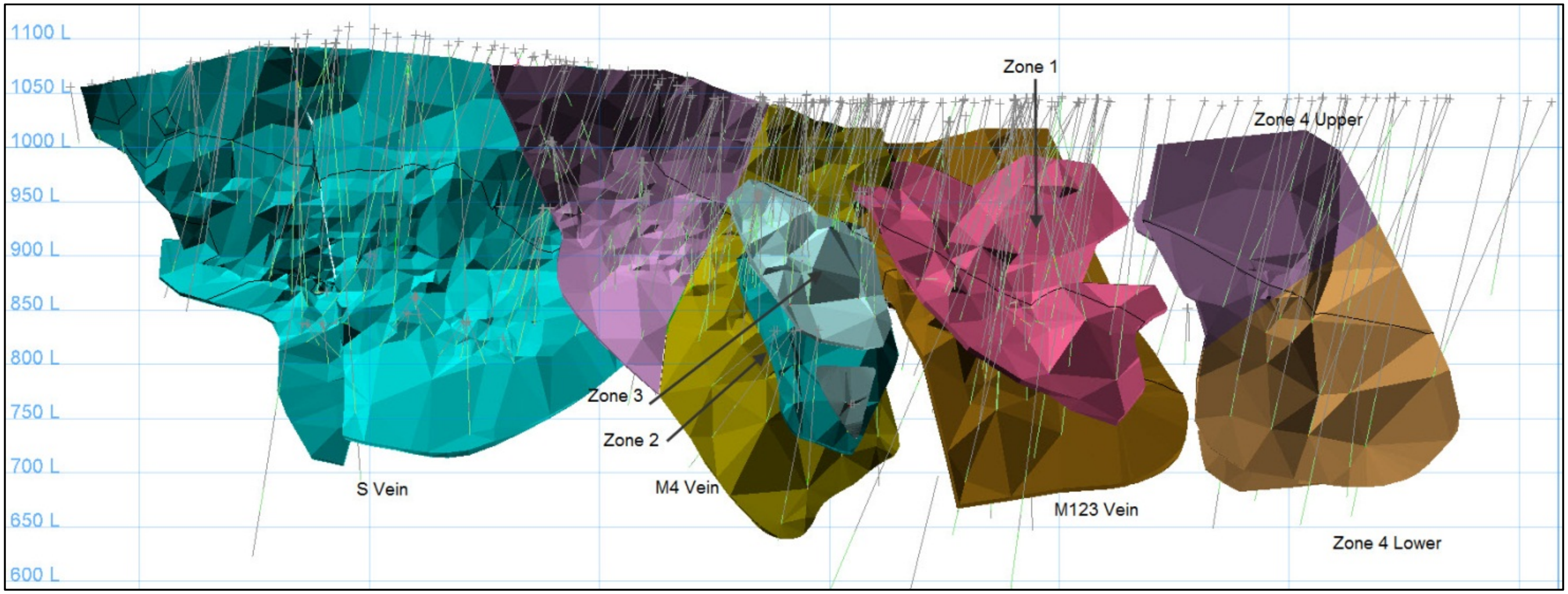


Figure 2.6.1.2: Ovacık Drilling and Long Section of the Hangingwall Showing Domains

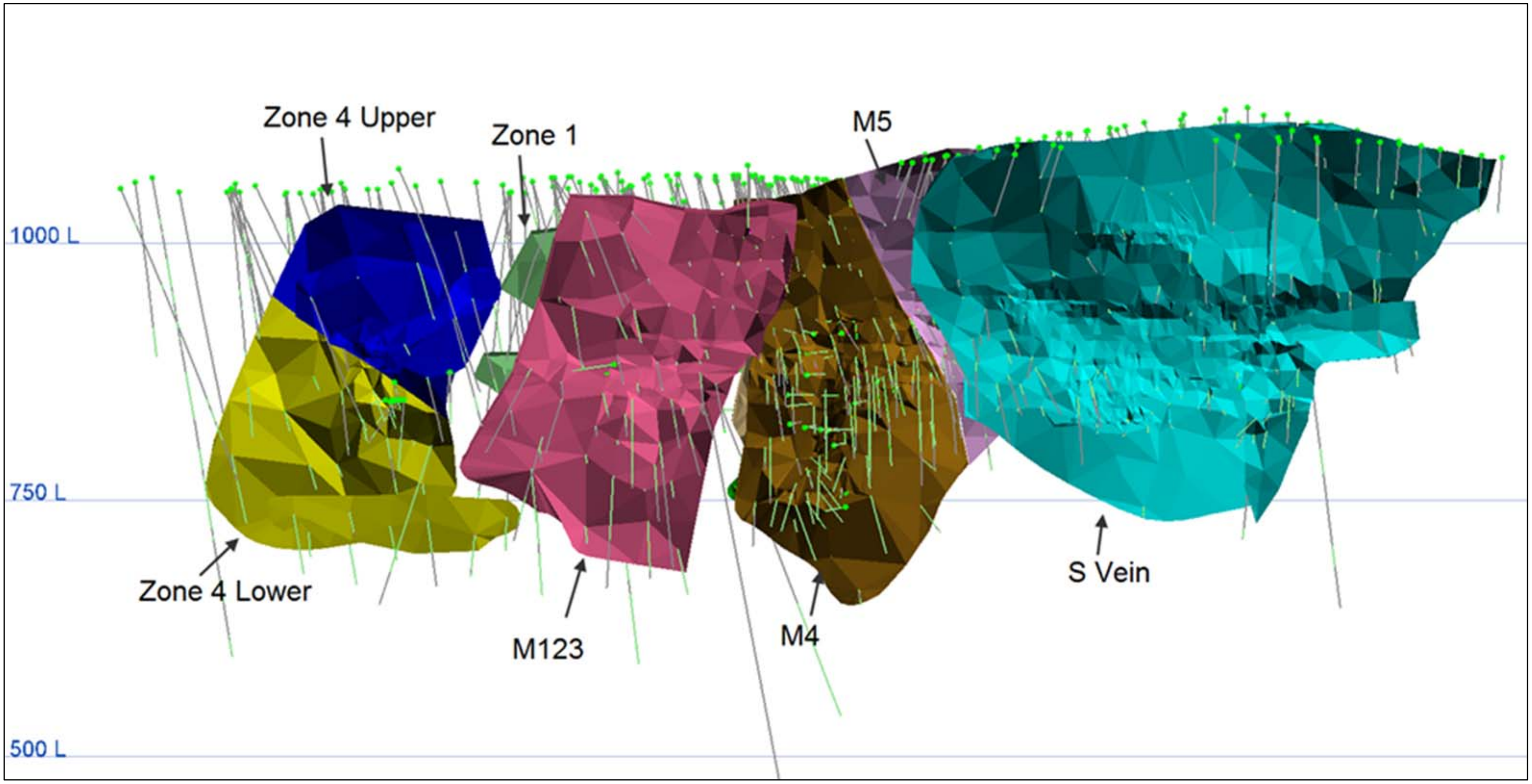


Figure 2.6.1.3: Ovacık Drilling and Long Section of the Footwall Showing Domains

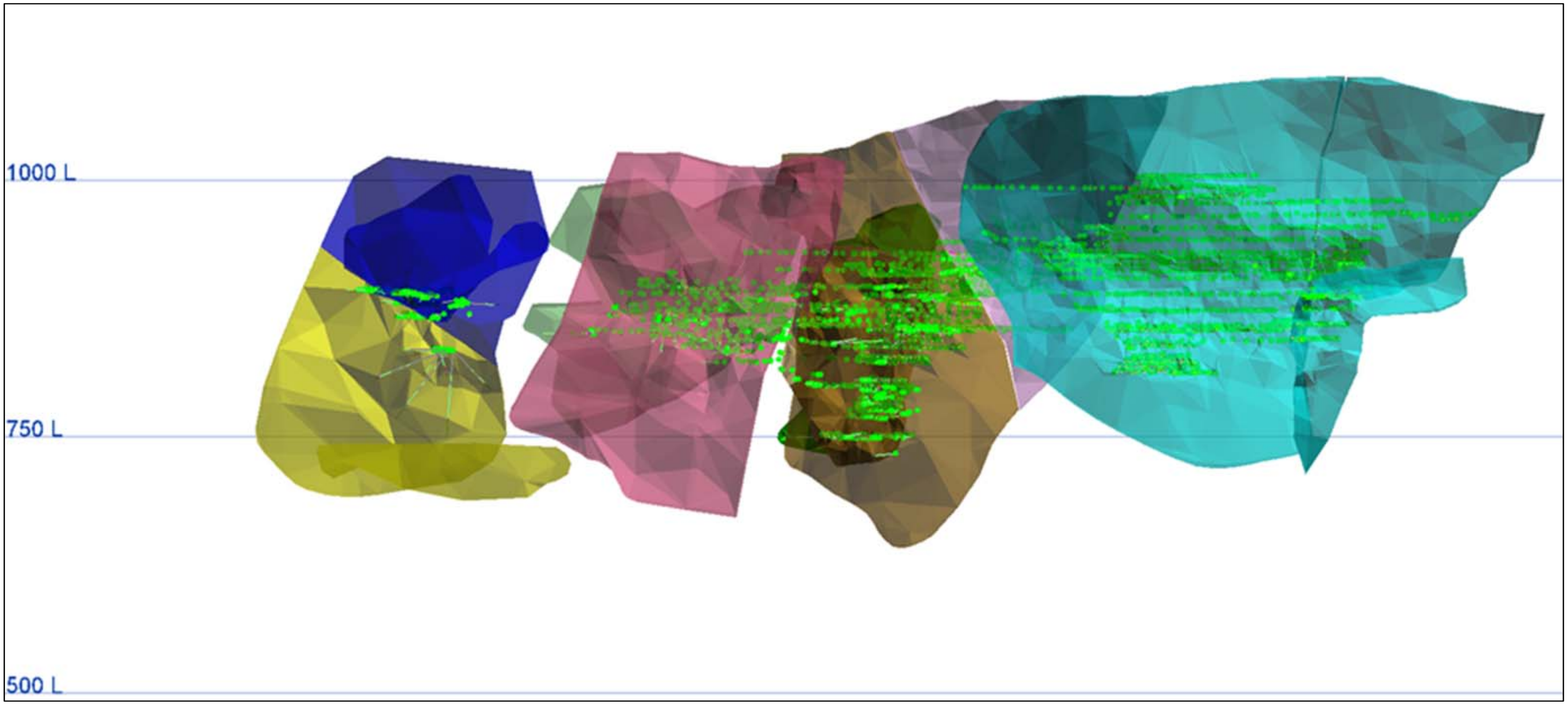


Figure 2.6.1.4: Ovacık Long Section of the Footwall showing Underground Grade Control Samples

Table 2.6.1.1: Ovacık Statistics of Drill Samples

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
S	Au	2,411	0.01	255.70	5.14	13.34	2.59
	Ag	2,411	0.00	340.00	5.64	16.93	3.00
M123	Au	1,549	0.01	805.00	8.85	39.23	4.43
	Ag	1,537	0.00	2,774.00	11.26	83.83	7.44
M4	Au	2,291	0.04	345.50	6.72	15.46	2.30
	Ag	2,291	0.00	300.00	6.29	16.97	2.70
M5	Au	991	0.06	104.00	7.73	12.47	1.61
	Ag	991	0.00	128.00	6.69	13.12	1.96
Z1	Au	764	0.01	100.00	1.87	7.51	4.01
	Ag	760	0.00	69.81	1.15	4.41	3.84
Z2	Au	1,151	0.01	76.51	2.28	5.24	2.30
	Ag	1,151	0.00	57.80	1.51	3.49	2.30
Z3	Au	555	0.03	110.00	1.95	5.58	2.85
	Ag	555	0.00	29.54	1.05	1.95	1.86
Z4 Lower	Au	341	0.09	218.50	7.90	26.38	3.34
	Ag	341	0.20	58.62	3.14	5.99	1.91
Z4 Upper	Au	56	0.09	91.62	3.50	10.53	3.01
	Ag	56	0.30	26.40	1.59	3.73	2.35
Z5	Au	319	0.09	92.00	3.25	9.09	2.79
	Ag	319	0.00	31.70	1.39	2.87	2.06
Z6	Au	255	0.09	108.05	3.95	11.89	3.01
	Ag	255	0.00	106.50	1.88	7.46	3.97
Total	**Au**	**10,683**	**0.01**	**805.00**	**5.51**	**19.41**	**3.52**
	Ag	**10,667**	**0.00**	**2,774.00**	**5.34**	**34.71**	**6.49**

Table 2.6.1.2: Ovacık Statistics of Grade Control Samples

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
S	Au	7913	0.00	301.90	5.14	10.23	1.99
	Ag	7913	0.00	345.15	4.16	9.54	2.30
M123	Au	2732	0.09	2,146.00	9.63	48.22	5.01
	Ag	2732	0.00	423.40	4.08	12.24	3.00
M4	Au	5436	0.00	1,221.00	7.43	29.91	4.03
	Ag	5436	0.00	476.10	6.73	18.66	2.77
M5	Au	1403	0.09	1,251.00	8.75	35.50	4.06
	Ag	1403	0.09	266.10	5.22	12.43	2.38
Z1	Au	448	0.09	237.20	6.65	22.42	3.37
	Ag	448	0.09	50.22	2.11	5.19	2.46
Z2	Au	1,377	0.09	123.00	3.40	7.16	2.11
	Ag	1,377	0.13	476.10	2.65	14.07	5.30
Z3	Au	709	0.09	110.00	5.00	10.28	2.05
	Ag	709	0.25	476.10	4.14	20.72	5.01
Z4 Lower	Au	719	0.09	604.10	24.59	62.01	2.52
	Ag	719	0.30	154.80	7.23	14.42	2.00
Z4 Upper	Au	476	0.09	461.00	10.00	23.33	2.33
	Ag	476	0.21	202.30	4.39	9.01	2.05
Z5	Au	270	0.09	110.80	5.35	12.05	2.25
	Ag	270	0.19	35.59	2.26	3.76	1.67
Z6	Au	337	0.09	100.00	4.51	10.90	2.41
	Ag	337	0.19	26.79	1.72	2.85	1.66
Total	**Au**	**21,820**	**0.00**	**2,146.00**	**7.09**	**28.05**	**3.95**
	Ag	**21,820**	**0.00**	**476.10**	**4.75**	**13.58**	**2.86**

Compositing

Samples are composited on 1 m intervals, with breaks at the vein boundaries. Tables 2.6.1.3 and 2.6.1.4 summarize statistics of the composited drill and grade control samples, respectively.

Table 2.6.1.3: Ovacık Uncapped Drill Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
S	Au	2,232	0.04	255.70	5.11	12.42	2.43
	Ag	2,232	0.00	267.45	5.60	14.89	2.66
M123	Au	1,512	0.01	504.87	8.83	33.90	3.84
	Ag	1,491	0.00	2,153.41	11.22	77.66	6.92
M4	Au	2,118	0.04	158.47	6.73	13.87	2.06
	Ag	2,118	0.00	175.73	6.31	15.31	2.43
M5	Au	904	0.09	102.60	7.62	11.52	1.51
	Ag	904	0.00	113.26	6.61	11.97	1.81
Z1	Au	795	0.02	100.00	1.87	6.95	3.72
	Ag	787	0.00	69.81	1.15	4.20	3.67
Z2	Au	1,105	0.09	73.37	2.27	4.89	2.16
	Ag	1,105	0.00	55.17	1.51	3.24	2.15
Z3	Au	540	0.03	85.72	1.96	5.06	2.58
	Ag	540	0.00	23.08	1.05	1.77	1.69
Z4 Lower	Au	297	0.09	209.40	7.78	23.61	3.03
	Ag	297	0.20	58.62	3.13	5.46	1.74
Z4 Upper	Au	50	0.20	48.59	3.60	8.34	2.32
	Ag	50	0.30	18.80	1.64	3.19	1.95
Z5	Au	307	0.09	83.20	3.23	8.08	2.50
	Ag	307	0.00	31.70	1.39	2.76	1.99
Z6	Au	235	0.09	100.00	3.97	11.02	2.78
	Ag	235	0.00	106.50	1.88	7.34	3.90
Total	**Au**	**10,095**	**0.01**	**504.87**	**5.51**	**17.19**	**3.12**
	Ag	**10,066**	**0.00**	**2,153.41**	**5.54**	**32.04**	**6.01**

Table 2.6.1.4: Ovacık Uncapped Grade Control Sample Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
S	Au	8,648	0.00	301.90	5.11	9.80	1.92
	Ag	8,648	0.00	345.15	4.16	9.31	2.24
M123	Au	2,832	0.09	1,800.19	9.60	42.55	4.43
	Ag	2,832	0.00	355.52	4.08	11.21	2.74
M4	Au	5,632	0.00	836.90	7.45	26.57	3.57
	Ag	5,632	0.00	476.10	6.80	17.42	2.56
M5	Au	1,455	0.09	1,017.22	8.76	30.80	3.52
	Ag	1,455	0.09	216.57	5.23	11.53	2.20
Z1	Au	455	0.09	237.20	6.59	21.71	3.30
	Ag	455	0.19	50.22	2.10	5.02	2.40
Z2	Au	1,468	0.09	81.89	3.45	6.59	1.91
	Ag	1,468	0.17	476.10	2.65	13.25	5.00
Z3	Au	704	0.09	85.72	5.05	9.06	1.79
	Ag	704	0.25	476.10	4.12	19.54	4.74
Z4 Lower	Au	740	0.09	570.93	23.99	53.64	2.24
	Ag	740	0.30	147.84	7.13	12.52	1.76
Z4 Upper	Au	481	0.09	333.04	9.94	21.18	2.13
	Ag	481	0.23	145.06	4.34	8.01	1.84
Z5	Au	276	0.09	88.85	5.28	10.91	2.07
	Ag	276	0.19	28.65	2.24	3.42	1.53
Z6	Au	348	0.09	100.00	4.56	10.56	2.32
	Ag	348	0.19	26.31	1.73	2.70	1.56
Total	**Au**	**23,039**	**0.00**	**1,800.19**	**7.09**	**24.97**	**3.52**
	Ag	**23,039**	**0.00**	**476.10**	**4.76**	**12.75**	**2.68**

Koza reviewed quantile analyses, probability plots and histograms to determine the need for capping and the appropriate capping values for the composites. Table 2.6.1.5 presents the capping values for gold and silver for the composited drill samples and grade control samples. Tables 2.6.1.6 and 2.6.1.7 present statistics for the capped composites. SRK has reviewed the data and agrees with the capping values.

Table 2.6.1.5: Ovacık Capping Values for Gold and Silver

	Drill Samples				Face Samples			
Zone	Max Au	Capped Value	Max Ag	Capped Value	Max Au	Capped Value	Max Ag	Capped Value
S	255.7	100.0	267.5	100.0	301.9	80.0	345.2	80.0
M123	504.9	110.0	2,153.4	130.0	1800.2	120.0	355.5	65.0
M4	158.5	100.0	175.7	110.0	836.9	130.0	476.1	105.0
M5	102.6	80.0	113.3	85.0	1017.22	50.0	216.57	50.0
Z1	100.0	50.0	69.8	30.0	237.2	70.0	50.2	40.0
Z2	73.37	25.0	55.2	30.0	81.89	25.0	476.1	25.0
Z3	85.72	25.0	23.08	-	85.72	18.0	23.1	5.0
Z4u	48.6	35.0	18.8	-	333.0	83.0	145.1	31.0
Z4d	209.4	32.0	58.6	-	570.93	125.0	58.6	11.0
Z5	83.2	40.0	31.7	-	88.9	55.0	28.7	-
Z6	100.0	40.0	106.5	30.0	100.0	37.0	26.3	12.0

Table 2.6.1.6: Ovacık Capped Drill Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
S	Cutau	2,232	0.04	100	5.01	10.86	2.17
	Cutag	2,196	0	100	5.30	11.38	2.15
M123	Cutau	1,512	0.01	110	6.97	17.92	2.57
	Cutag	1,491	0	130	6.46	19.25	2.98
M4	Cutau	2,118	0.04	100	6.67	13.42	2.01
	Cutag	2,118	0	110	6.19	14.38	2.32
M5	Cutau	904	0.01	80	7.73	11.47	1.48
	Cutag	904	0	85	6.67	11.85	1.78
Z1	Cutau	795	0.02	50	1.71	5.03	2.95
	Cutag	787	0	30	1.03	2.62	2.53
Z2	Cutau	1,105	0.09	25	2.10	3.29	1.57
	Cutag	1,105	0	30	1.48	2.82	1.91
Z3	Cutau	540	0.03	25	1.81	3.39	1.88
	Cutag	540	0	23.1	1.04	1.74	1.68
Z4U	Cutau	50	0.20	35	3.23	6.89	2.14
	Cutag	50	0.30	18.8	1.59	3.11	1.96
Z4L	Cutau	297	0.09	32	4.72	8.23	1.74
	Cutag	297	0.20	58.6	3.14	5.49	1.75
Z5	Cutau	307	0.09	40	3.01	6.41	2.13
	Cutag	307	0	31.7	1.39	2.77	1.99
Z6	Cutau	235	0.09	40	3.34	6.68	2.00
	Cutag	235	0	30	1.55	3.25	2.1
Total	**Cutau**	**10,095**	**0.01**	**110**	**5.03**	**11.68**	**2.32**
	Cutag	**10,066**	**0**	**130**	**4.51**	**12.15**	**2.70**

Table 2.6.1.7: Ovacık Capped Grade Control Sample Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
S	Cutau	8,648	0.00	80	5.03	8.22	1.64
	Cutag	8,648	0.00	80	4.06	7.72	1.90
M123	Cutau	2,832	0.09	120	8.32	17.30	2.08
	Cutag	2,832	0.00	65	3.87	7.33	1.90
M4	Cutau	5,632	0.00	130	6.68	15.27	2.29
	Cutag	5,632	0.00	105	6.46	14.13	2.19
M5	Cutau	1,455	0.09	50	7.43	9.97	1.34
	Cutag	1,455	0.09	50	4.84	8.17	1.69
Z1	Cutau	455	0.09	70	5.30	13.69	2.58
	Cutag	455	0.19	40	2.07	4.87	2.34
Z2	Cutau	1,468	0.09	25	3.15	4.53	1.44
	Cutag	1,468	0.17	25	2.20	3.42	1.56
Z3	Cutau	704	0.09	18	4.10	4.54	1.11
	Cutag	700	0.25	5	2.03	1.46	0.72
Z4U	Cutau	516	0.09	83	9.43	15.17	1.61
	Cutag	516	0.23	31.1*	4.08	4.80	1.18
Z4L	Cutau	705	0.09	125	20.04	33.86	1.69
	Cutag	705	0.30	11	4.52	3.75	0.83
Z5	Cutau	276	0.09	55	5.21	10.09	1.94
	Cutag	276	0.19	28.7	2.26	3.41	1.51
Z6	Cutau	348	0.09	37	3.73	5.81	1.56
	Cutag	348	0.19	26.3**	1.70	2.61	1.54
Total	**Cutau**	**23,039**	**0.00**	**130**	**6.39**	**13.46**	**2.11**
	Cutag	**23,039**	**0.00**	**105**	**4.41**	**9.30**	**2.11**

*Should have been capped at 31; affects 1 composite
**Should have been capped at 12; affects 3 composites

2.6.2 Density

In 1991, 103 quartz and 43 andesite samples were taken from 9 PQ sized core holes. Samples were grouped according to rock type, alteration, degree of breakage and nature of the quartz. The specific gravity was determined using Archimedes Principle where the core was covered with wax and the samples were weighed in water and in air. Later, mineralized intercepts from 17 holes were chosen and specific gravity determinations made on those intercepts. The samples were grouped as quartz or andesite and as above and below 1000 RL. All quartz samples averaged 2.2 g/cm^3 and andesite samples above 1000 RL average 2.0 g/cm^3, and samples below 1000 RL average 2.2 g/cm^3. Koza is using 2.2 g/cm^3 for all lithologies. It is SRK's opinion that this is reasonable given that most samples are of the quartz type.

2.6.3 Variography

Variography studies were performed by Newmont on the 1 m composites. Koza has reviewed variography, but has not updated the variography parameters used in the resource estimation. The variogram parameters used in the estimation are listed in Table 2.6.3.1. Koza's updated variograms indicate that the maximum range is about 100 m, rather than ranges between 175 and 320 shown in Table 2.6.3.1. SRK suggests that Koza use their updated variography because there is much more data now compared to when Newmont owned the mine.

Table 2.6.3.1: Ovacık Variogram Parameters

Vein	Axis	Orientation	Nugget	Sill 1	Sill2	Range1 (m)	Range2 (m)
M Vein 123,Zones 1,2,3	Major	10°,098°	0.43	0.379	0.19	9	59
	Semi-major	76°,145°				9	42
	Minor	10°,190°				39	320
M Vein 4, Zones 5,6	Major	-10°,098°	0.10	0.642	0.26	9	157
	Semi-major	76°,145°				30	35
	Minor	-10°,190°				1	175
M Vein 5, Zone 4	Major	00°,280°	0.04	0.769	0.188	6	304
	Semi-major	-90°,270°				63	50
	Minor	00°,190°				1	71
S Vein	Major	00°,280°	0.22	0.66	0.12	1	182
	Semi-major	-90°,270°				7	62
	Minor	00°,190°				462	251

2.6.4 Grade Estimation

The block model was constructed in Datamine with a block size of 10 m x 2 m x 10 m (X, Y, Z) with sub-blocking to 1.25 m x 0.5 m x 0.01 m. Grade control samples and drill samples were used separately in the resource estimation, as explained below. Estimation was accomplished by ordinary kriging (OK) with search ellipse directions matching the variogram directions (Table 2.6.3.1) and primary search distances shown in Table 2.6.4.1.

Table 2.6.4.1: Ovacık Primary Search Distances (m)

Domain	Au			Ag		
	Major	Semi Major	Minor	Major	Semi Major	Minor
Drillhole Samples						
S Vein	95	60	10	85	85	10
M Vein , Z1-6	105	80	10	75	65	10
Grade Control Samples						
S Vein	30	20	5	30	20	5
M123, 4, 5 Veins	25	35	5	25	35	5
Z1-6 of M Vein	20	20	5	20	20	5

Grade Control Samples

In order not to extrapolate higher grades from the grade control samples into the main part of the orebody, a smaller search ellipse was used with an octant restriction on the samples to be used. The estimation parameters included the following:

- A single estimation pass, using the primary distances shown in Table 2.6.4.1;
- Octant search requiring a minimum of three octants with a minimum of two and a maximum of five samples per octant; and
- Minimum of 10 and a maximum of 20 samples, with a maximum of 4 per drillhole.

Estimation was done to the parent cell size. Datamine's dynamic anisotropy search option was used where the search ellipse orientation closely follows the vein structure. For the M and S veins, vein types 1 and 2 were estimated separately. The estimation with the grade control samples took precedence over the estimation with surface and underground core holes.

Drill Samples

Three estimation passes were used with the surface and underground core samples. The search parameters are as follows:

- First pass with the primary distances shown in Table 2.6.4.1;
- Octant search was applied with a minimum of 1 and a maximum of 4 samples per octant with samples in a minimum of two octants;
- Minimum of 2 and maximum of 20 samples, with no maximum per drillhole; and
- Successive iterations were made with 1.5 and 2 times the first search with the same sample number requirements as the first pass.

A dynamic search was used to follow the vein structure.

The structural domains were used as hard boundaries and only composites with the same code were used for each individual domain. For the M and S veins, vein types 1 and 2 were estimated separately.

2.6.5 Block Model Validation

Koza validated the model by:

- Visual comparison of composites to block grades on vertical sections;
- Comparison of grades estimated by kriging, ID2 and ID5 in the drillhole model and ID2, NN and ID5 in the grade control model; and
- Comparison of composite and block grades.

Table 2.6.5.1 shows average gold grades estimated by OK, ID2 and ID5 for the block models estimated with drillholes and with grade control samples.

Table 2.6.5.1: Comparison of Gold Grades Estimated by OK, ID2 and ID5

Domain	Drillhole Model			Grade Control Model		
	OK	ID2	ID5	ID2	NN	ID5
S	5.33	5.23	5.19	4.44	4.41	4.42
M4	5.83	5.94	5.87	5.36	5.31	5.32
M5	8.01	8.07	7.93	6.81	7.02	6.76
M123	5.37	5.20	5.11	7.61	7.62	7.60
Z1	2.07	NA	NA	3.60	3.62	3.38
Z2	1.95	NA	NA	2.80	2.72	2.77
Z3	2.15	NA	NA	3.39	3.52	3.36
Z4u	3.92	NA	NA	10.08	10.92	9.90
Z4d	2.93	NA	NA	17.47	15.40	16.54
Z5	2.61	NA	NA	5.32	3.82	5.07
Z6	3.10	NA	NA	3.54	3.38	3.54

Tables 2.6.5.2 and 2.6.5.3 show the average block gold and silver grades estimated by OK for the drillhole model and ID2 for the grade control model and the average composite grades for both composite types. Generally, the estimated grades should not be higher than the composite grades and should not be more than about 10% lower than the composite grades. Koza should investigate the areas where the estimated grades are outside this range.

Table 2.6.5.2: Comparison of Estimated Gold and Silver Grades and Composite Grades in the Drillhole Model

Domain	Au			Ag		
	Block	Composite	% Diff	Block	Composite	% Diff
S	5.33	5.01	6	5.67	5.30	7
M123	5.37	6.97	-23	4.13	6.46	-36
M4	5.83	6.67	-13	5.46	6.19	-12
M5	8.01	7.73	4	7.20	6.67	8
Z1	2.07	1.71	21	1.14	1.03	11
Z2	1.95	2.10	-7	1.36	1.48	-8
Z3	2.15	1.81	19	1.15	1.04	11
Z4u	3.92	3.23	21	1.98	1.59	25
Z4d	2.93	4.72	-38	2.60	3.14	-17
Z5	2.61	3.01	-13	1.24	1.39	-11
Z6	3.10	3.34	-7	1.43	1.55	-8

Table 2.6.5.3: Comparison of Estimated Gold and Silver Grades and Composite Grades in the Grade Control Model

Domain	Au			Ag		
	Block	Composite	% Diff	Block	Composite	% Diff
S	4.44	5.03	-12	3.96	4.06	-2
M123	7.61	8.32	-9	3.99	3.87	3
M4	5.36	6.68	-20	4.91	6.46	-24
M5	6.81	7.43	-8	4.75	4.84	-2
Z1	3.60	5.30	-32	1.51	2.07	-27
Z2	2.81	3.15	-11	1.90	2.20	-14
Z3	3.39	4.10	-17	1.82	2.03	-10
Z4u	10.08	9.43	7	4.25	4.08	-4
Z4d	17.47	20.04	-13	4.29	4.52	-5
Z5	5.32	5.21	2	2.45	2.26	8
Z6	3.54	3.74	-5	1.58	1.70	-7

SRK Validation

SRK found an error in the estimation during its validation process in late January 2015. The grade control block model was estimated with uncapped composites in Zone 4D. Koza corrected this error when SRK notified the resource geologist of the problem. The resource Table 2.6.9.2 contains the corrected resource numbers and Table 2.6.9.3 contains the original resource numbers prior to correction.

2.6.6 Reconciliation

Koza performed a reconciliation of 2014 mined tonnes and grade to tonnes and grade predicted by the 2013 model. Table 2.6.6.1 shows the reconciliation numbers provided by Koza. Koza ascribes the differences to variations in predicted and actual vein width and grade. The block model does not contain waste blocks, so the calculation of tonnes and grade from the resource model does not include dilution.

Table 2.6.6.1: Koza Reconciliation Ovacık 2013 Model to 2014 Production

Item - Ovacık	Tonnes	AUCUTOK (g/t)	Ounces
Resource Model	48,847	7.23	11,593
Production	97,738	9.02	28,333
Milling Reconciliation + Stockpile	116,016	7.96	29,699
Percent Mill/Production	119%	88%	105%

Source: Koza, 2014

The planned production for Ovacık in the EOY 2013 Technical Economic Model was 205,000 t at 4.21 g/t Au, containing 27,800 oz. Koza evidently chose to mine fewer tonnes in higher grade areas than called for in the mine plan, resulting in about the same number of produced ounces.

2.6.7 Depletion

The block model is depleted for production using the wireframes for the underground as-builts and the open pit final topography. The mined out blocks are completely removed from the block model, which limits the ability to validate the block model with drill and grade control data. A better method is to use a variable denoting percentage depleted from, or percentage remaining in, the resource.

Koza also performs some depletion of the block model to remove remnant blocks that are outside the as-builts and are either too small for mine plans or which need additional drilling before the mine engineers will build mine plans on them. It is SRK's opinion that these remnant blocks form a large tonnage and that more depletion needs to take place. There are also many resource blocks which are too isolated to meet the criterion as potentially mineable and should be excluded from the resource. Figure 2.6.7.1 is a long-section of Ovacık, showing only the S zone and Zone 123 resource blocks. The S zone has been depleted for remnant blocks around the mine as-builts, as is appropriate (Figure 2.6.7.2). However, there are isolated blocks in the S Zone (Figure 2.6.7.3) that do not form shapes that are potentially mineable and which should be excluded from the resource. Figure 2.6.7.4 is a cross-section of Zone 123 showing remnant blocks around the as-builts that should be depleted from the resource model. The mine engineers have made mine plans on the blocks that are large enough, and blocks which do not meet their criteria should be excluded from the resource.

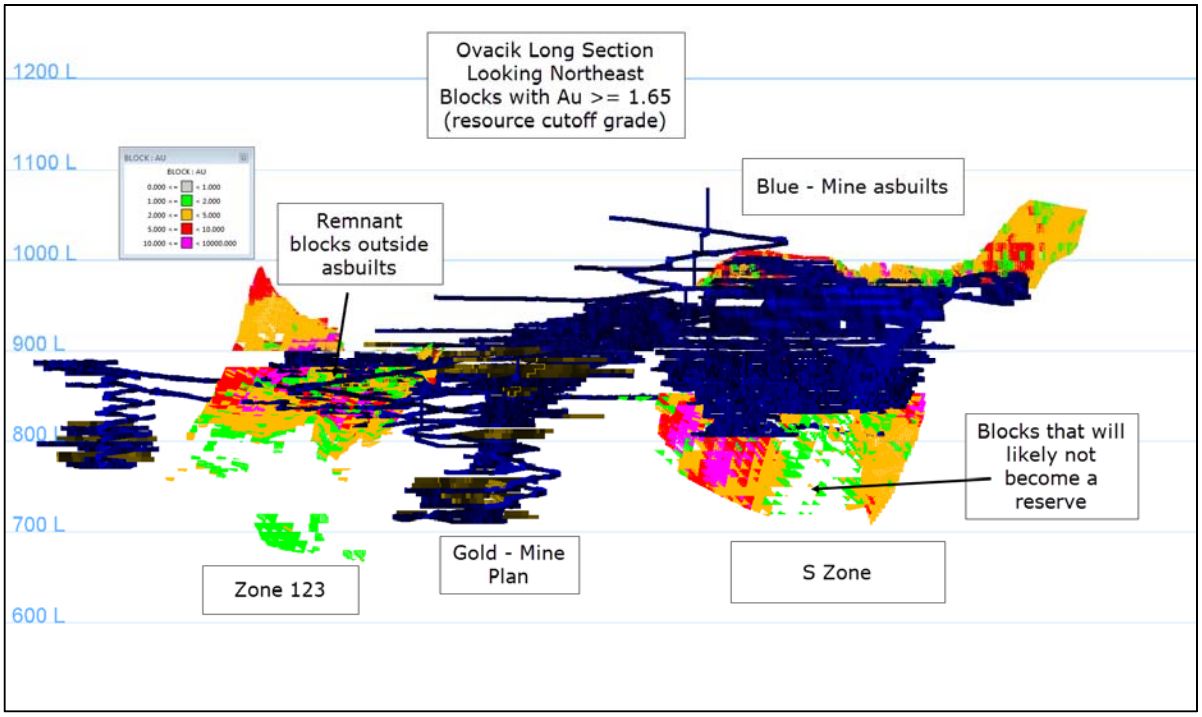


Figure 2.6.7.1: Ovacık Long-section with S Zone and Zone 123 Resource Blocks

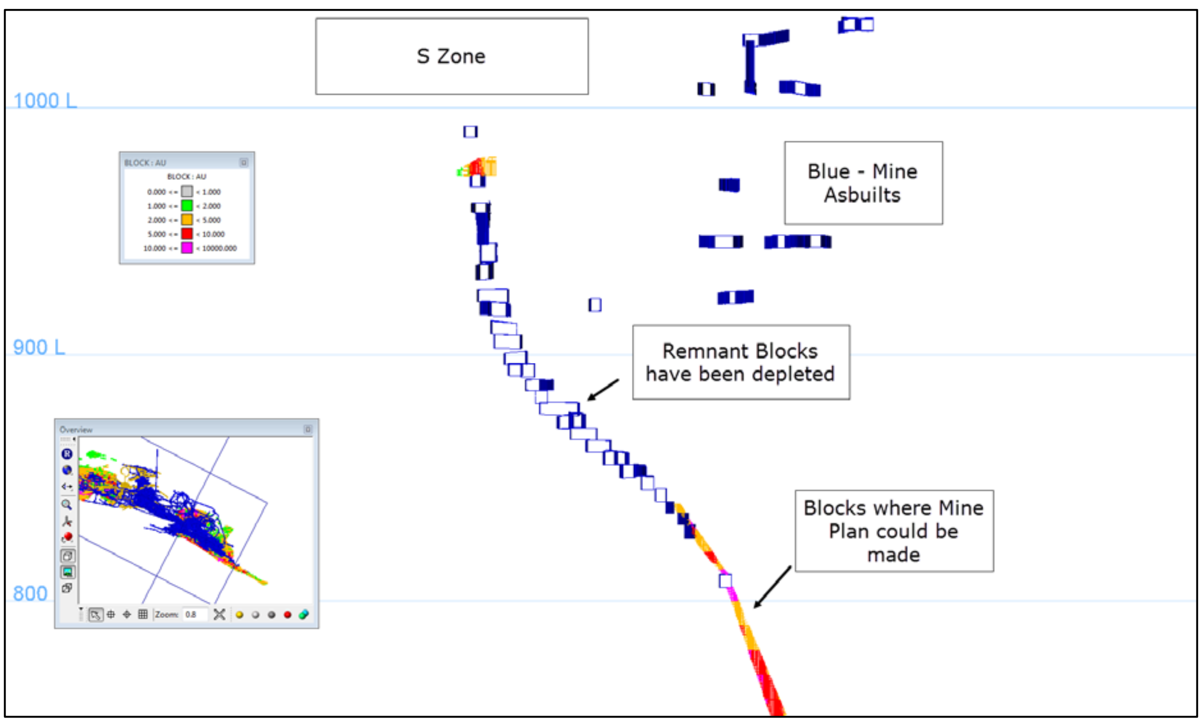


Figure 2.6.7.2: Ovacık Cross-section with S Zone Resource Blocks, Showing Depletion

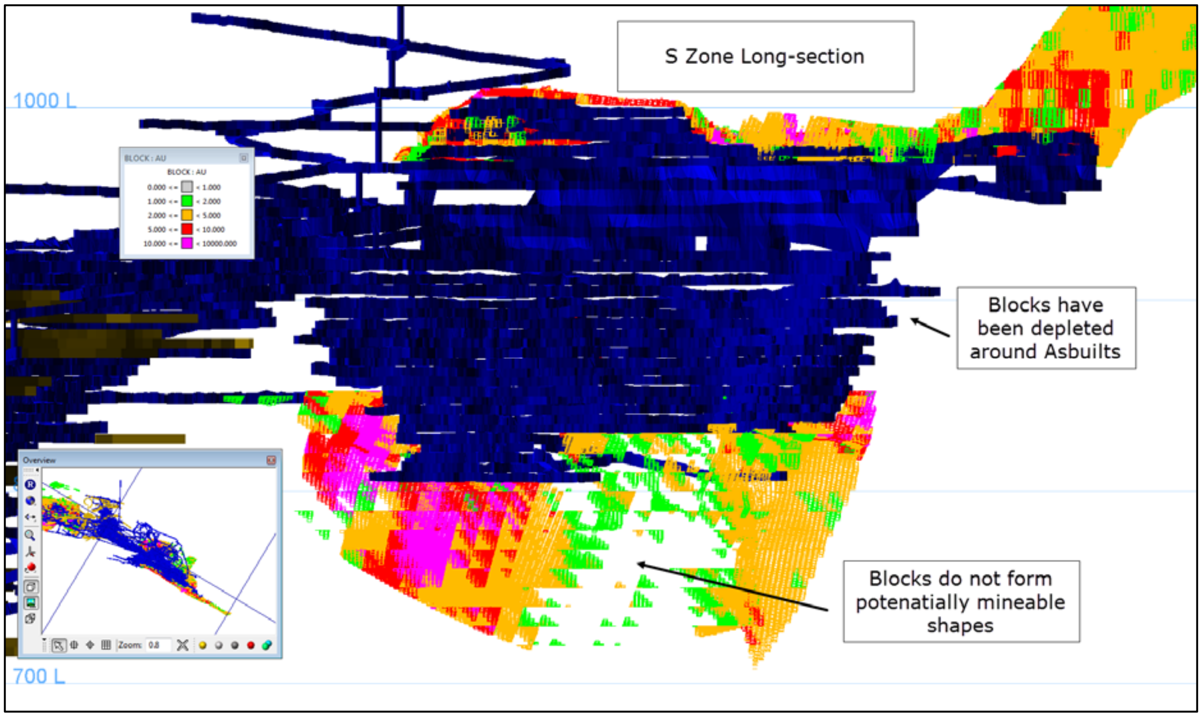


Figure 2.6.7.3: Ovacık Long-section with S Zone Resource Blocks, Showing Depletion and Blocks that should be excluded from Resource

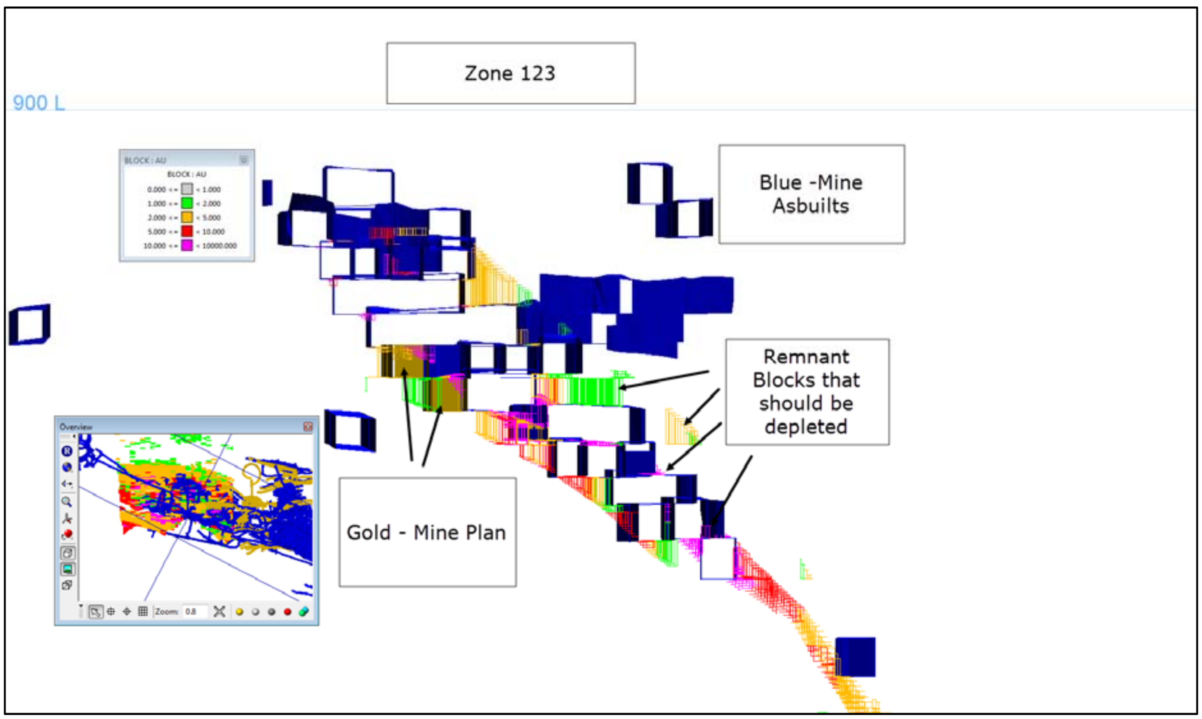


Figure 2.6.7.4: Ovacık Cross-section with Zone 123 Resource Blocks that should be depleted

2.6.8 Mineral Resource Classification

Kriging variance was used for classification of the resources. A kriging variance value of 0.45 was used to create an "Indicated" envelope. The blocks were classified as Measured, Indicated, or Inferred based on the kriging variance and the "Indicated" envelope. Blocks within the envelope and with a kriging variance less than 0.2 were classified as Measured. All other blocks inside the envelope were classified as Indicated. Resources outside the envelope or which were coded as andesite were defined as Inferred.

2.6.9 Mineral Resource Statement

The resource block model is depleted for material removed during mining operations.

The cutoff grade of 1.65 g/t Au is based on the assumptions shown in Table 2.6.9.1. 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 2.6.9.1: Ovacık Cutoff Grade Assumptions

Item	Units	Prices and Costs
Gold Price	US$/oz	1,450
Gold Recovery	%	0.95
Gold Refining	US$/oz	3.44
Royalty	%	3
Government Right	%	1
Process Cost	US$/t	11.00
Mining Cost	US$/t	45.00
G&A Cost	US$/t	15.00
Calculated Cutoff grade	g/t	1.67
Final Cutoff grade	g/t	1.65

Source: Koza, 2014

As noted in Section 2.6.5, SRK noted an error in the estimation using the grade control samples in late January 2015 and Koza corrected the error at that time. The Corrected Measured, Indicated, and Inferred resources at a cutoff grade of 1.65 g/t Au are listed in Table 2.6.9.2; the tonnage is inclusive of ore reserves. The resources prior to the correction are shown in Table 2.6.9.3. The difference is a reduction of about 8,000 oz of gold in Measured and Indicated and 1,000 oz in Inferred.

Table 2.6.9.2: Ovacık Mineral Resources Corrected, Including Ore Reserves, at December 31, 2014

Classification	kt	g/t Au	g/t Ag	oz Au	oz Ag
Measured	1,616	5.06	3.3	263	173
Indicated	703	3.20	1.9	72	43
Measured and Indicated	2,319	4.50	2.9	335	215
Inferred	251	4.00	2.1	32	17

*Cutoff grade of US$1.65
*Inclusive of Reserves

Table 2.6.9.3: Ovacık Mineral Resources Prior to Correction, Including Ore Reserves, at December 31, 2014

Classification	kt	g/t Au	g/t Ag	koz Au	koz Ag
Measured	1,611	5.23	3.3	271	172
Indicated	699	3.23	1.9	73	43
Measured and Indicated	2,310	4.62	2.9	343	215
Inferred	260	3.99	2.1	33	18

*Cutoff grade of US$1.65
*Inclusive of Reserves

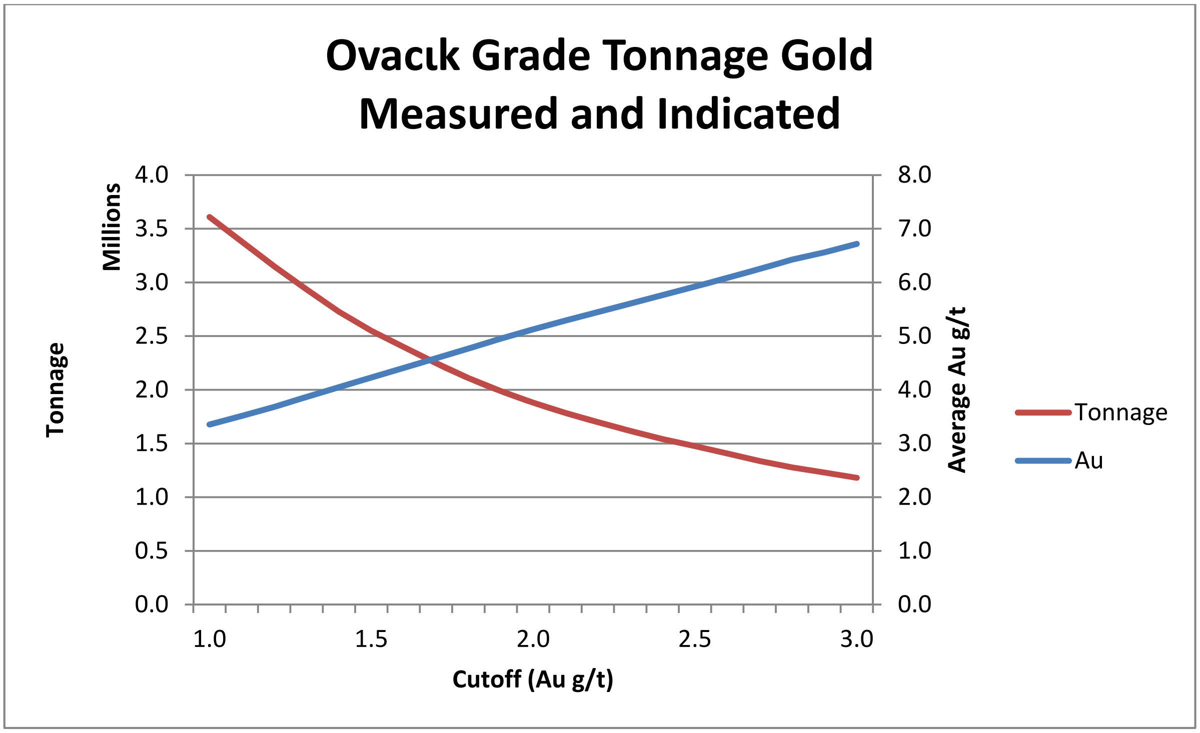
2.6.10 Mineral Resource Sensitivity

Figure 2.6.10.1 presents grade tonnage curves for Measured and Indicated Resources combined and Inferred Resources separately.

Cutoff grades for the Ovacık resource at various gold prices are shown in Table 2.6.10.1.

Table 2.6.10.1: Ovacık Cutoff Grades vs. Gold Price

Gold Price	Cutoff Grade
1600	1.52
1550	1.57
1500	1.62
1450	1.67
1400	1.73
1350	1.80
1300	1.87
1250	1.94
1200	2.02



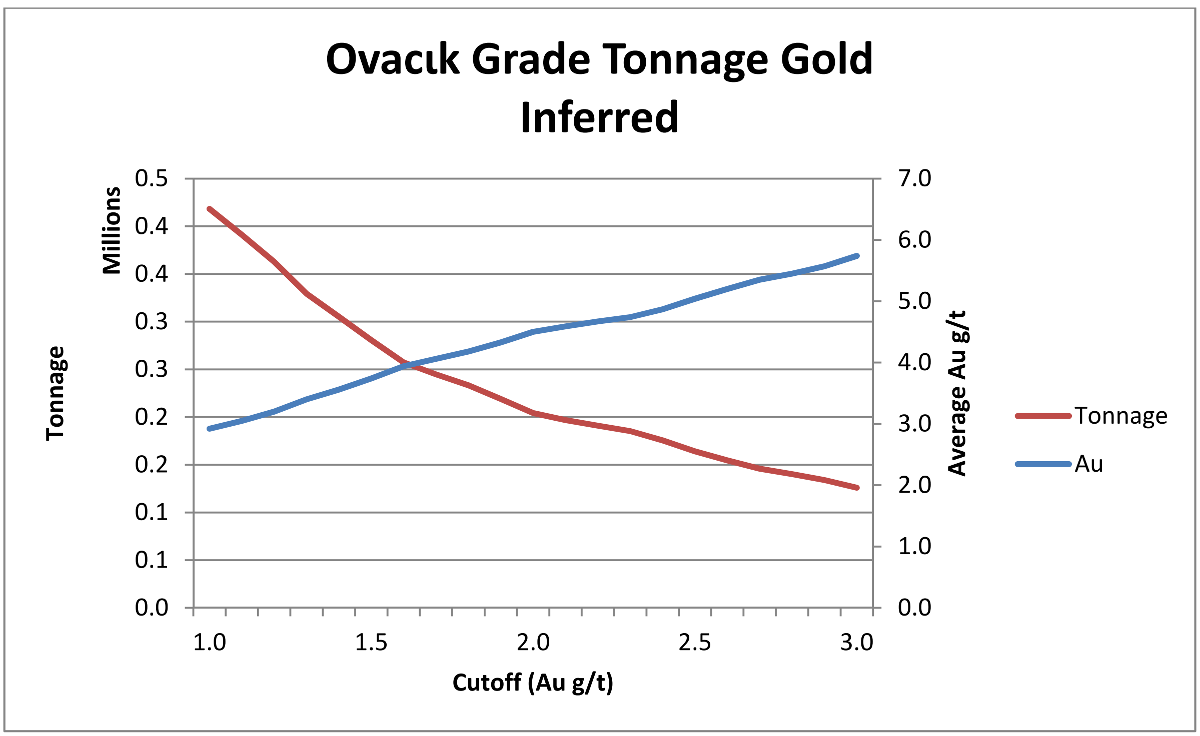


Figure 2.6.10.1: Grade Tonnage Curves for Ovacık Resource

2.7 2013 Ovacık Mine Production

2013 mine production figures for Ovacık are based on the full year production from underground operations. Run of Mine (RoM) grade is shipped directly to the Ovacık processing facility. Table 2.7.1 details the high grade mine production for the Ovacık Mine for 2014 compared to that estimated in the 2013 technical economic model for yearly reconciliation purposes.

Table 2.7.1: 2014 Ovacık RoM Mine Production vs. 2013 Technical Economic Model Estimates

2014 Production	Ovacik UG 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	10,868	9.89	6.15	3,456	11,337	4.06	2.28	1,480	4%	-59%	-57%
February	10,076	11.65	4.90	3,774	13,411	6.10	3.53	2,630	33%	-48%	-30%
March	10,446	6.05	5.22	2,032	14,628	5.04	3.32	2,370	40%	-17%	17%
April	7,503	8.73	12.55	2,106	11,684	4.23	2.53	1,589	56%	-52%	-25%
May	7,677	6.87	4.83	1,696	15,341	4.41	2.49	2,175	100%	-36%	28%
June	9,512	7.84	7.19	2,398	9,928	4.80	2.51	1,532	4%	-39%	-36%
July	7,737	10.84	5.29	2,696	12,872	6.45	5.63	2,669	66%	-40%	-1%
August	7,554	10.66	9.47	2,589	11,628	5.04	3.07	1,884	54%	-53%	-27%
September	5,375	5.67	3.96	980	8,787	6.83	5.81	1,930	63%	20%	97%
October	6,357	10.69	6.02	2,185	14,913	5.37	3.88	2,575	135%	-50%	18%
November	7,184	8.57	3.84	1,979	18,214	4.70	2.72	2,752	154%	-45%	39%
December	7,449	10.20	3.90	2,443	9,854	3.84	2.03	1,217	32%	-62%	-50%
Total	**97,738**	**9.02**	**6.13**	**28,333**	**152,597**	**5.06**	**3.28**	**24,803**	**56%**	**-44%**	**-12%**

The production schedule for Ovacik underground is quite flexible and no great demand was placed on operations through 2014. This is because the grade from the Çukuralan underground is higher than the grade scheduled from Ovacik. As a result, the Ovacik underground is essentially high graded so it does not replace Çukuralan ore, but rather adds value to the processing stream. For 2014, the grade of the mine production was 44% higher than that planned and apart from a single month the grade was consistently higher. Because the grade was so high, not as many ore tonnes were mined to produce the planned ounces. As it stands, over 12% additional ounces were mined than scheduled. Due to the grades of Çukuralan, the mine production forecast has been curtailed for Ovacik with a move to a single crew in 2015.

2.8 Ore Reserve Estimation

LoM plans and resulting reserves are determined based on a gold price of US$1,250/oz for the underground mine. Reserves stated in this report are as of December 31, 2014.

The open pit at Ovacık was completed in September 2007 and all ore production from the site is now sourced from stockpiles and underground production at Ovacık. Feed for the Ovacık Mill will come from the Ovacık underground, Ovacık stockpiles, Çoraklıktepe, Çukuralan underground, Çukuralan open pit and Kubaşlar open pit.

The ore at Ovacık is extracted using cut and fill and short hole open stoping methods. The ore material is converted from resource to reserve based primarily on in-situ value, resource classification and its location relative to current underground mine workings. The in-situ value is derived from the estimated grade and various modifying factors.

2.8.1 Modifying Factors

Ore reserves are based on the economic balance between the value per tonne of rock and the cost to mine and process each tonne of rock. The value is based on estimated metal concentration, estimated metal value and milling recovery. The costs include development, mining, processing, transportation, and operating overhead.

To define the value per tonne of rock, the estimated concentration of gold is factored by an estimated long-term value. The long-term gold price used by Koza in the cutoff grade calculation is US$1,250/oz. In the opinion of SRK, this gold value is reasonable and appropriate for ore reserve estimation.

The second factor is the process recovery, which is based on mill head grade, recovered metal and tail grade. The reserve uses a mill recovery value of 95% at Ovacık. The final factor is based on a royalty of 3% and government mining rights of 1%.

Table 2.8.1.1 summarizes the cutoff grade calculation for the Ovacık underground orebody.

Table 2.8.1.1: Ovacık Mine Underground Cutoff grade Calculation 2014

Parameter	Unit	Cut and Fill
Mining cost	US$/t ore	55
Processing cost	US$/t ore	10.85
Admin cost	US$/t ore	18.10
Total cost	**US$/t**	**83.95**
Gold price	US$/oz	1,250
Mill recovery	%	95
Royalty	%	4
Refining	US$/oz	3.44
Cutoff grade	**g/t Au**	**2.3**

Source: Koza, 2014

2.8.2 Reserve Classification

The underground mine design process at Ovacık entails filtering the resource model for Measured and Indicated resources above the 2.30 g/t Au cutoff grade and then designing cut and fill stopes and bench stopes based on the orebody wireframes. Mining dilution is accounted for during the design process by modeling the expected final stope and development shapes based on the as-built excavations. Where stope and development designs contain Inferred resource material, the Inferred gold content is removed from the design but the tonnage is retained. Where the stope and development designs contain material that is outside the geological wireframe, this material is included as planned dilution at zero grade.

Proved and Probable reserve categories are determined directly from the Measured and Indicated categories. Table 2.8.2.1 presents the mineral reserve for the Ovacık underground mine as of December 31, 2014.

Table 2.8.2.1: Ovacık Underground Reserve, as at December 31, 2014

Category	kt	g/t Au	g/t Ag	koz Au	koz Ag
Proven Reserve	190	5.31	3.0	32,	18,
Probable Reserve	37	2.95	2.8	4	3
Total Proven and Probable	**227**	**4.92**	**3.0**	**36**	**22**

Source: Koza, 2014
Reserves based on December 31, 2014, Metal Price US$1,250/oz-Au, Au Recovery 95%, Au cutoff grade 2.30 g/t.

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 (LG), the removal of administration and grade control costs lower the break-even-cutoff grade making processing profitable at the end of mine life. The RoM and LG reserves produced from underground operations are listed in Table 2.8.2.2. Low grade material stockpiled at Küçükdere is available for transportation to the Ovacık facility if mine production is interrupted at the other mine sites. It is planned that the stockpile will be processed when the Küçükdere LG material does not displace processing of RoM feed from the other underground and open pit operations. The Küçükdere stockpile reserve is detailed in Table 2.8.2.3 and emergency stockpile reserve in Table 2.8.2.4.

Table 2.8.2.2: Ovacık RoM and LG Stockpile Reserve, at December 31, 2014

Category	kt	g/t Au	g/t Ag	koz Au	koz Ag
Proven Reserve	95	5.89	4.0	18	12
Probable Reserve	158	1.40	1.5	7	8
Total Proven and Probable Reserves	**253**	**3.09**	**2.4**	**25**	**20**

Source: Koza, 2014
Reserves based on stockpile balance on December 31, 2014, Metal Price US$1,250/oz-Au, Au Recovery 95%

Table 2.8.2.3: Küçükdere LG Stockpile Reserve, at December 31, 2014

Category	kt	g/t Au	g/t Ag	koz Au	koz Ag
Probable Reserve	389	1.36	6.3	17	79
Total Proven and Probable Reserves	**389**	**1.36**	**6.3**	**17**	**79**

Source: Koza, 2014
Reserves based on stockpile balance on December 31, 2014, Metal Price US$1,250/oz-Au, Au Recovery 95%

Table 2.8.2.4: Ovacık Emergency Stockpile Reserve, at December 31, 2014

Category	kt	g/t Au	g/t Ag	Contained koz Au	Contained koz Ag
Proven Reserve	61	6.59	4.5	13	9
Total Proven and Probable Reserves	61	6.59	4.5	13	9

Source: Koza, 2014
Reserves based on stockpile balance on December 31, 2014, Metal Price US$1,250/oz-Au, Au Recovery 95%

2.9 Mining Engineering

The Ovacık Mine is an underground and open pit complex. The open pit completed operation in September 2007. It has been partially backfilled and no further production is currently planned. Studies are continuing into the suitability of using the historic open pit as a tailings facility storage area.

Two primary vein systems are found on the property. The M Vein system tends to be higher grade and more shallowly dipping and is mined using a cut and fill method. The S Vein is lower grade, steeper and mined using an open stoping method with backfill.

Underground mining commenced in June 2005 and the current ore reserve gives a lifespan through July 2021. Koza plans to continue mining at a 500 t/d rate at Ovacik through July 2015, after which time the mining rate will decrease to just 100 t/d for the remainder of the mine life. This will be achieved by reducing mine operations to a single shift per day with the other shifts relocated to the second Çukuralan portal. The low-grade cutoff is calculated by removing mining cost from cutoff grade calculation. This material is mined through to access ore and is stockpiled rather than wasted.

2.9.1 Mining Method

Access to the underground orebody is via a portal at the top of the north side of the pit. The ramp spirals down in the hangingwall of the orebody to a depth of about 160 m, whereupon the ramp crosses over into the footwall. All access development is driven at a nominal 5 m x 5 m size. Figure 2.9.1.1 shows a long section view of the final pit, the underground drifts and stoping, and the future mine planning (magenta).

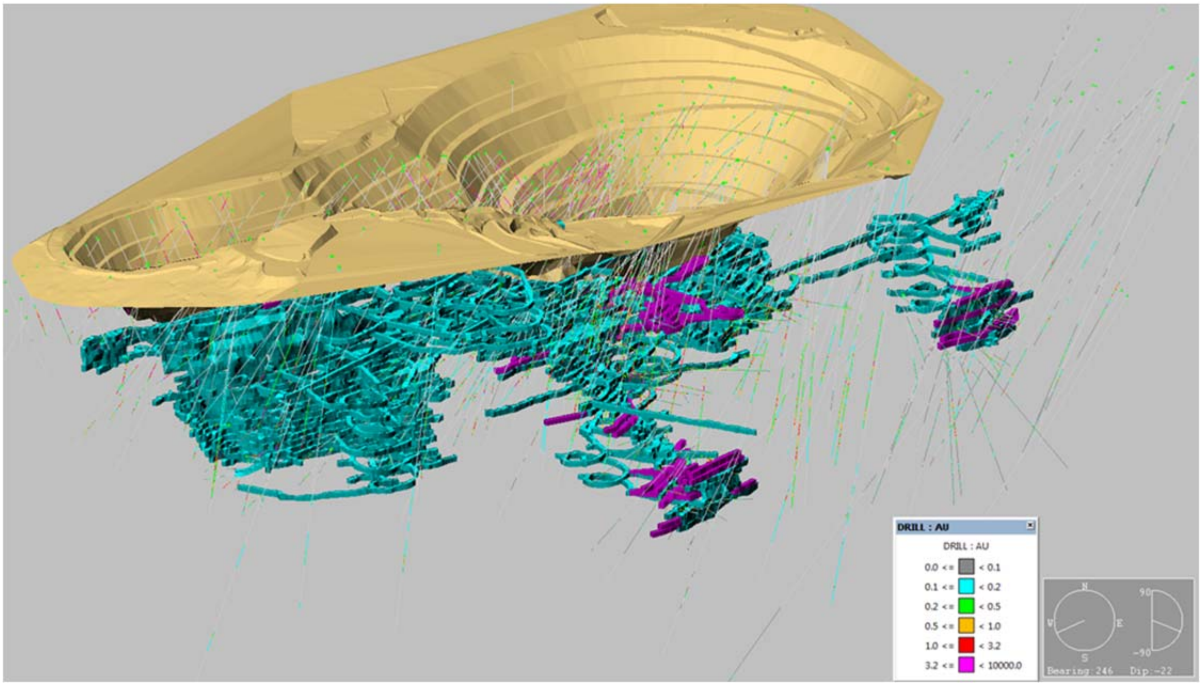
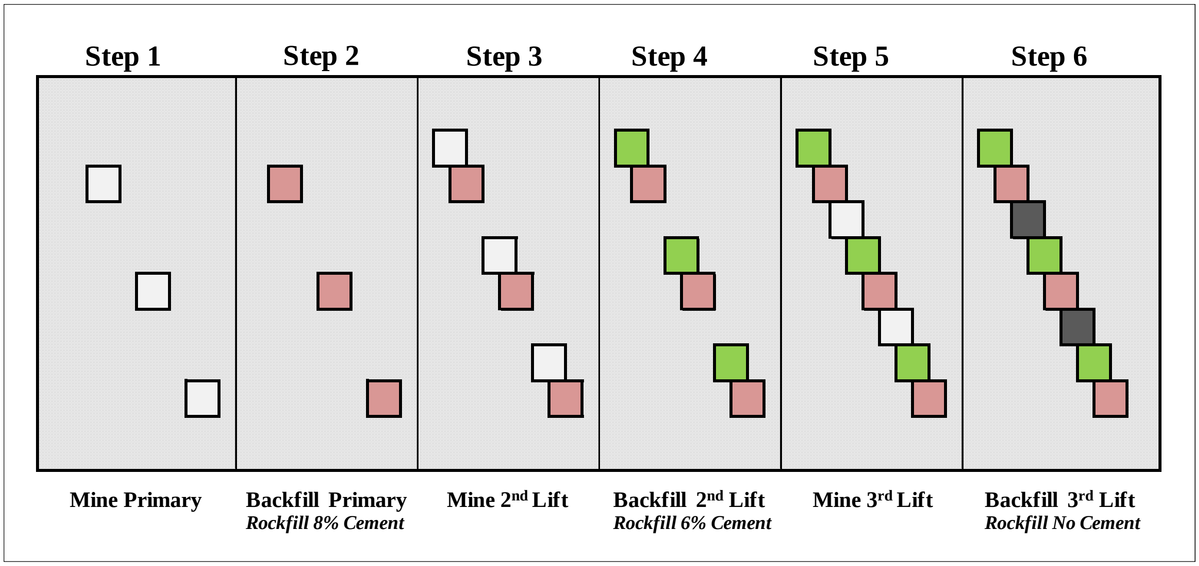


Figure 2.9.1.1: Ovacık Mine Long Section View Looking South West

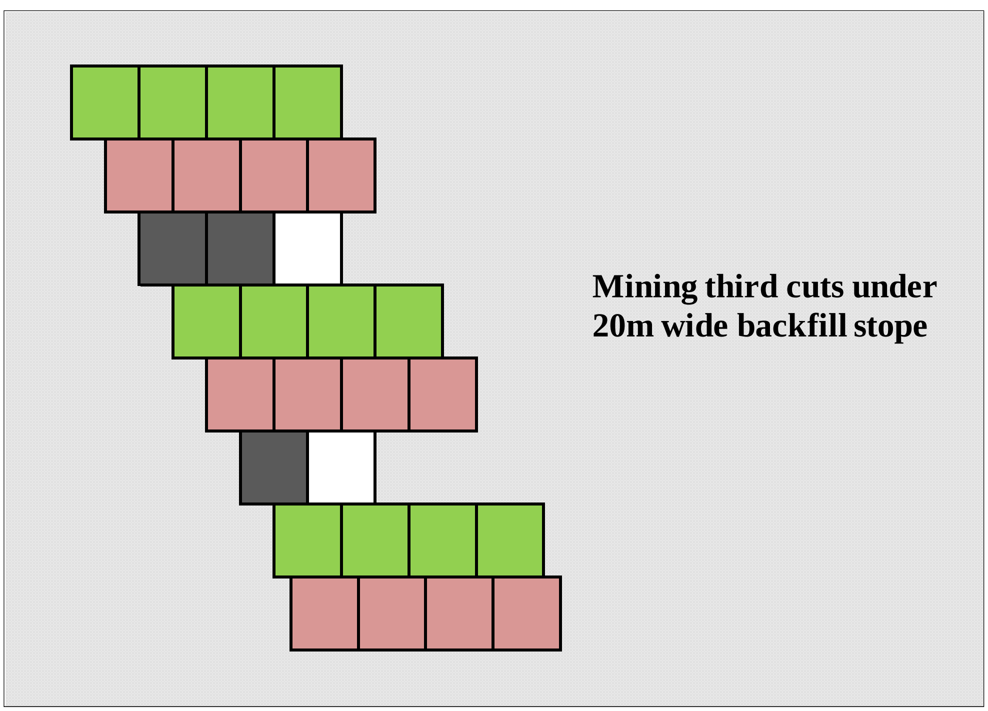
The M Vein system is mined using a cut and fill method. Primary cuts are driven 5 m high at a spacing of 10 m or 15 m back to floor, allowing either two or three cuts to be mined between the primaries respectively. Figure 2.9.1.2 shows an idealized mining and backfill sequence for a stope block with a 10 m separation between primary stopes. Primary stopes are mined and filled with rockfill containing an 8% cement binder. This high cement content is required as these stopes will be undercut by the third lift at a later date. The second lift is mined over the first and filled with development waste, unless a parallel drift is being mined on the same horizon, in which case backfill with 6% cement binder is installed. This percentage of cement is required so that equipment mining for the third cut has a stable working platform and to provide increased strength for wider stope areas. The final lift is mined in an undercut fashion with the backfill from the primary cut forming the stope back. This final cut is generally left open.



Source: Koza, 2011

Figure 2.9.1.2: Ovacık Underground Cut and Fill Mining and Backfilling Sequence

In wide orebody areas the cuts are still mined as 5 m x 5 m but in this case multiple cuts are mined side by side with each lift. Some areas of the orebody are mined in four side-by-side 5 m drifts giving a total cut span of 20 m (Figure 2.9.1.3) although this practice is becoming less common as mining gets deeper. In this situation, the quality of the backfill and the strength obtained using a 6% cement binder is important.



Source: Koza, 2011

Figure 2.9.1.3: Undercut Mining Wide Stope Areas

The upper S Vein is mined using a bench stoping method. In this case primary access drifts are driven in ore at a back to floor spacing of about 8 m. Blasthole drilling is carried out from the bottom drift using sub-vertical holes drilled using a development jumbo with extension steel. A raise is blasted at the end of the stope to provide breaking room and the blastholes are retreat blasted into the raises in short 8 m blocks. The ore is mucked from the bottom access using a remote controlled load haul dump (LHD). At the end of the stope, or when the hangingwall starts to become unstable, the stope is backfilled. The hangingwall span generally does not exceed 12 m.

All development is drilled using twin boom jumbos. Faces are mucked out using LHD's and primary support in the form of 10 cm of polypropylene fiber reinforced shotcrete is applied. Split set tendon support is installed in the back and upper walls using a single boom jumbo. In areas of poor ground the shotcrete and split sets are supplemented with wire screen installed over the shotcrete and held in place by the split sets.

Ore and waste are transported from the face using dump trucks to either the ore stockpiles or the waste dumps on surface. The trucks back-haul cemented backfill to the cut and fill stopes underground.

A comprehensive backfill design and testing program led to the finding that the waste material sourced from underground development would not provide enough strength for undercut mining at wider stope widths, thus a waste material with suitable strength properties is sourced from a quarry outside the mine. Backfill is produced by mixing the crushed waste material with a cement and water mixture. The crushed waste is loaded onto a conveyor using a front-end loader and conveyed up to a batch mixing plant on a raised platform. Cement and water are added to the mixer and the material is mixed for one minute before being dumped out of the bottom of the mixer directly into the truck below. When full, a process that takes about fifteen minutes, the trucks back-haul the material down the ramp to the stope being filled. In the case of cut and fill mining, the trucks dump the material in the drift and the material is pushed up to the stope back using a pusher-plate assembly mounted on the boom of an LHD specially modified for the task. In the case of open stope mining the trucks dump backfill material in the drift at the top of the stope and remote controlled LHD's push the material into the stope. This process leads to a very consistent high quality stope backfill.

Shotcrete material is produced on site using the same conveyor and mixing arrangement as the backfill. In this case, fine gravel and sand are loaded onto the conveyor in the correct proportions and conveyed to the mixer, where cement, water and plasticizers are added. The mixed material is dumped directly from the batch mixer into a shotcrete transmixer (underground truck with a rotating mixer drum attached). The transmixer drives to the face, reverses up to the Spraymec (remote controlled, robot arm, shotcrete application unit) and dumps shotcrete material directly into the pump unit. The Spraymec operator controls the boom and applies shotcrete to the back and walls of the excavation from a safe location away from the unsupported ground.

SRK found the underground mining operation to be efficient, clean and well organized. Development, mining, ground support and general housekeeping standards were of a high quality and consistent with a world-class mining operation. There is potential for optimization of mining costs, geotechnical application of ground support and mining method. At this time, Koza continues a strategy to minimize risk and use proven mine methodology given the current gold price.

2.9.2 Mining Equipment

Table 2.9.2.1 presents the Ovacık underground mining equipment fleet.

Table 2.9.2.1: Ovacık Underground Mining Equipment

Task	Equipment	Quantity	Supplier
Truck haulage	MT 2000 MT 2010	1 2	Atlas Copco Atlas Copco
Mucking	ST 710 SLF 65	1 3	Atlas Copco Schopf
Shotcrete	Spraymec UG Mixer Truck	1 1	Normet shotcrete sprayer Normet shotcrete transmixer truck
Drilling	Jumbo -Rocketboomer 281 Jumbo -Rocketboomer	1 2	Atlas Copco
Integrated Tool Carrier	Telehandler - T40140 Telehandler - TM310 Telehandler - TM320	1 1 1	BOBCAT JCB JCB
Wheel Loader	CAT 960	1	CAT
	Backhoe Loader	2	Hidromek
Tractor	N.Holland 4030-6020	3	New Holland
Light Vehicle	Pick-up	4	Mitsubishi

Source: Koza, 2014

2.9.3 Geotechnical Designs

The ground conditions appear to be well controlled in the underground mine. Systematic use of shotcrete provides excellent areal coverage and pattern bolting ties the surface support into the unfractured rock mass away from the excavations. There was no sign of significant shotcrete failure in any of the excavations visited on the underground tour (2011) in either the M or S Vein areas. Visual observation of the shotcrete confirmed that it is being applied at a consistent 10 cm+ thickness. In discussion with mine engineers and the underground production manager there are some areas where the ground conditions are extremely poor especially in the M Vein. It also appears as though ground conditions in the S Vein are deteriorating with depth. As a result of this, all mining planned below the current operations will be cut and fill in the S Vein. The cut and fill provides better opportunity for ground control than the stoping method.

There is a geotechnical engineer based at the site who provides ground control and design support to the mine department and there is a process of review by a senior geotechnical engineer and regular consultants that provide overview and strategic comment to ensure that good long-term decisions are being made.

2.9.4 Mine Planning

Mine planning is carried out by the Ankara-based engineering department and is of a high quality. There is good communication between the mine operations department and the engineers. Regular operational meetings and discussions are held to ensure that both short and long-term issues are addressed and that the plan is understood by all parties.

The engineering department uses Datamine for mine design and Mine2-4D for development and stope scheduling. SRK is of the opinion that the scheduling process carried out in Mine2-4D is efficient and effective. This package links directly to Datamine and provides an integrated, graphical design and scheduling interface that removes much of the tedious and error prone spreadsheet work.

The mine design process is as follows:

- Obtain updated ore wireframes and block model information from geology;
- Identify areas of the orebody above cutoff grade;
- Based on the ore location and characteristics, decide on the mining method;
- In cut and fill areas, each 5 m high primary cut and subsequent lift is designed according to the vein profile and block model grade at that elevation. The cuts are broken down into 4m long segments for ore reserve evaluation purposes. Wireframes are created for the 4m drift segments and these are evaluated against the block model for tonnage, grade and resource category;
- In longhole stope areas the primary drifts are designed using the same methodology as cut and fill. Stope wireframes are then built between the primary drifts based on the vein wireframes and block model grades;
- The results of the evaluation are exported to a spreadsheet and sorted by stope name;
- In order to report the reserve, the Inferred category metal content is removed and the results are reported according to reserve category; and
- To produce the LoM plan, the extraction sequence of the ore drives and stopes are scheduled based on an average historical development advance rate. The development is balanced between all the production areas while keeping in mind the overall mining sequence. Waste access and ramp extension development is added to the schedule to support opening up of new cut and fill and stope access drifts.

The schedule is modified to include changes in geological understanding and actual development performance on an as required basis. Short term plans are generated monthly based on the long-term plan. In this manner the operation keeps a direct relationship between short and long-term plans and ensures that the strategic direction is followed.

2.9.5 Mine Ventilation

The mine is well ventilated in the main access, services areas and production areas, utilizing an exhaust system. Two 132 kW surface fans running in parallel are situated at the top of the exhaust raise provide primary ventilation. A series of underground auxiliary fans distribute the air to the working areas and maintain the flow in the main ramps. The nominal flow capacity of the system is 130 m^3/s for a demand of 110 m^3/s. Ventilation surveys are conducted by the mine engineer on a weekly and monthly basis, and the schematic ventilation plan is updated frequently. In addition, the ventilation is simulated using Ventsim software to identify any deficiencies in the ventilation plans for future mining areas and any proposed modification in ventilation system.

2.10 Metallurgy, Process Plant and Infrastructure

During 2014, plant feed material continued to come from both the both Ovacık and Çukuralan mining operations and was supplemented by ore from Çoraklıktepe. The Ovacık process plant was initially designed for a throughput of 300,000 t/y although it became evident that the design grind size (80% -38µm) for the Ovacık feed was not optimal and that increased throughput could be achieved with only a minor loss in recovery at a coarser grind (80% -75µm). The plant processed 866,867 t of ore during 2014, which was only slightly less than the 876,185 t that were processed during 2013.

2.10.1 Ore Mineralogy and Metallurgy

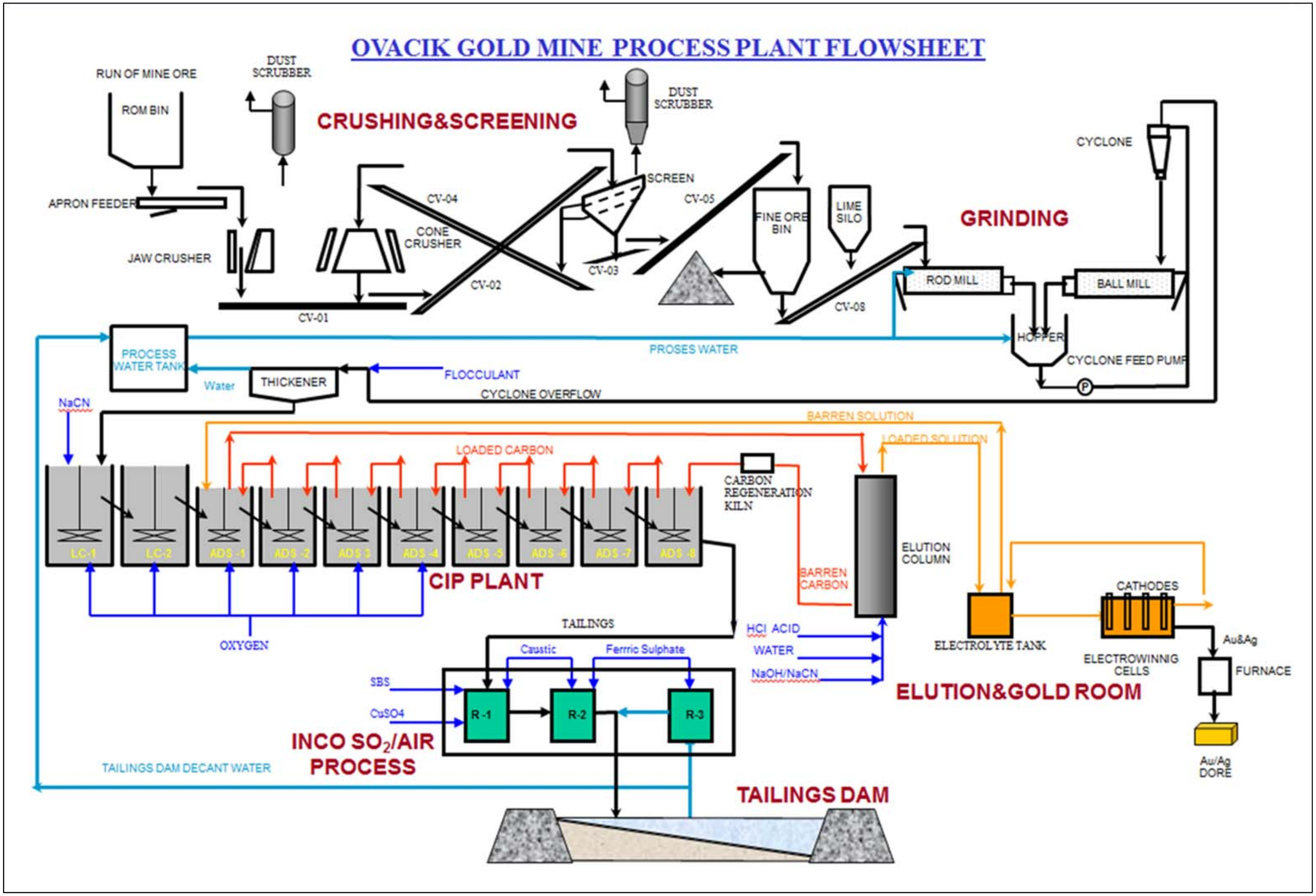
The Ovacık and Çukuralan ores are epithermal quartz vein type deposits with relatively low levels of base metal sulfides.

Testwork on ore samples from Çukuralan has demonstrated gold recoveries in the range of 94 to 95% at a grind size of 80% -75µm with a cyanidation retention time of 24 hours. Crushing and grindability tests undertaken by SGS have indicated that the ore is similar in hardness to the Ovacık ore. Preliminary data on the Çukuralan ore indicated reagent consumptions of around 0.50 to 0.55 kg/t sodium cyanide and 0.8 to 1 kg/t for lime.

2.10.2 Process Plant Description

The Ovacık process plant incorporates a conventional carbon-in-pulp (CIP) cyanidation flowsheet that consists of two-stage crushing, two-stage grinding, cyanide leaching with oxygen injection, CIP adsorption of the dissolved gold, carbon elution, electrowinning and smelting through to doré metal.

A schematic flowsheet of the current circuit is presented in Figure 2.10.2.1. A list of the major equipment installed at Ovacık is presented in Table 2.10.2.1.



Source: Koza, 2014

Figure 2.10.2.1: Ovacık Process Plant Flowsheet

Table 2.10.2.1: Major Equipment Sizes

Equipment Item	No. units	Details	Motor Size kW
Primary Jaw Crusher	1	1.10 m x 0.88 m Double toggle	150
Secondary Crusher	1	Metso GP550 Cone	315
Crushing Circuit Screen	1	7.3 m x 3.0 m Metso RF SH 3000x7300 Double deck (rubber) 50 mm and 24 mm	30
Crushed Ore Silo	1	1,500 t capacity	
Rod Mill	1	2.7 m dia x 4.2 m long	315
Ball Mill	1	3.8 m dia x 6.2 m long	1,300
Classifying Hydrocyclones	8+2	250 mm diameter	
Preleach Thickener	1	10 m diameter High Rate	
Leach Tanks	2	9.5 m dia x 11 m high 700 m^3 nominal capacity	37
Adsorption/CIP Tanks	8	6.5 m dia x 8 m high 210 m^3 capacity	18.5
Elution Column (AARL)	1	4 t capacity	
Electrowinning Cells	4	11 cathode units	500A 3-5V
Regeneration Kilns	1	200 to 250 kg/h Rotary Kiln	

Source: Koza, 2014

Crushing

Ore is trucked to the stockpile area and separated into different high and low-grade 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 1,500 t capacity crushed ore silo at a nominal size of P_{80} 16-18 mm. Dust collection is installed at all major emission and transfer points and is routed to individual wet scrubbers in the crushing and screening areas.

Milling and Classification

Mill feed is reclaimed from the storage silo with 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 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 target grind fineness of P_{80} 75μm.

Cyanidation

The cyanide leach circuit consists of two 700 m^3 agitated leach tanks that provide a nominal 24 hours retention time. The dissolved oxygen level in the first leach tank is maintained at 30 ppm with oxygen injection and the free cyanide level at is maintained at 150 to 160 ppm. The slurry pH is maintained at 10.5.

Discharge from the cyanidation tanks flow by gravity to an eight-stage counter-current CIP circuit with air 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.

Carbon is transferred counter-currently to the slurry flow with vertical pumps. Carbon from the first CIP tank is loaded to about 8,000 g/t gold from a solution head grade of about 5 g/t Au while in the final stage of CIP the gold in solution level is reduced to about 0.02 to 0.03 g/t Au. Tailing solids after CIP are typically 0.2 to 0.3 g/t Au. The CIP tailing 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 sulfate in the tailings treatment circuit although this is only used during periods of elevated arsenic levels. Residual weak acid dissociable CN_{WAD} levels after treatment are <1ppm.

Carbon Elution and Regeneration

Loaded carbon from the CIP circuit is pumped to the loaded carbon screen where it is drained and washed to remove residual slurry before passing into a pressure Anglo American Research Laboratories (AARL) elution column. The capacity of the elution circuit is 4 t/batch and around four or five elution cycles are undertaken each week. A complete elution cycle typically takes 8 to 10 hours to complete. In the elution column the carbon is initially acid washed using hydrochloric acid to remove scale and other contaminants, rinsed with water and the gold and silver are then stripped from the carbon using caustic and cyanide pretreatment and water at elevated temperature. A split elution system is employed using low-grade solution from the previous elution cycle in the first step of gold elution.

Following elution, every second cycle of 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 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 resulting sludge is filtered, dried, calcined and smelted to doré. Smelting of the gold sludge is undertaken once a week.

Tailing Storage Facility

Single stage pumps are used to transfer tailings slurry to the TSF facility, which consists of a plastic lined earthwork embankment behind which the tails slurry is deposited. Tailings pass along a single line and then via two individual lines which run around either side of the dam to discharge spigots. The TSF was said to have two remaining years of storage capacity. An alternative TSF will need to be designed and constructed before the existing TSF is filled to capacity.

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 relative quantity of Ovacık and Çukuralan ore processed is calculated by loader weights. Ovacık ore grade is taken as reported from the mine and Çukuralan ore grade is calculated based on total feed and ore grade to the plant. The cyclone overflow represents the plant feed and is sampled manually every hour to formulate 12-hour shift composites. The plant tails, after detoxification, are sampled using an automatic cutter with samples taken every 15 minutes and prepared into a 12-hour shift composite for analysis. Tailing samples are taken both before and after detoxification. In addition, tailing samples are taken and analyzed every two hours to monitor WAD cyanide levels.

Plant accounting assays are based on aqua regia digestions and Atomic Absorption Spectrography 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. Facilities are also available for ICP analyses.

Metallurgical balances are reasonable with variations between the assayed and back-calculated gold head grade varying by generally <10% on a monthly basis. The calculated head grade is generally higher than the assay head.

2.10.3 Plant Performance

Metallurgical Recoveries and Plant Throughput

Table 2.10.3.1 provides a summary of Ovacık process plant performance for the period from 2003 through 2014. It is notable that plant annual throughput increased steadily to 658,050 ore tonnes by 2007 during the period when ore was sourced from the Ovacık open pit and underground mines. This was accomplished by recognizing that the ore could be processed at a coarser grind than the plant had been designed for without losing gold recovery. During the period from 2009 to 2010 plant throughput was further increased to over 800,000 t. This was due to the processing of ore sourced from the Küçükdere mine, which was significantly softer than Ovacık ore, and as a result, could be processed at a higher tonnage rate while maintaining the target grind and recovery levels. During 2011 to 2014, ore has been sourced predominantly from the Çukuralan mine with some ore contributed from the Ovacık underground mine and the Çoraklıktepe open pit mine. During this period, annual production steadily increased to as high as 879,411 t. Monthly production from the Ovacık process plant for 2013 and 2014 is tabulated in Table 2.10.3.2 and Table 2.10.3.3.

Table 2.10.3.1: Summary of Ovacık Process Plant Annual Performance

Year	Ore Tonnes	Average TPM	Head Grade		Recovery %		Poured Ounces	
			Au, g/t	Ag, g/t	Au	Ag	Au	Ag
2003	483,969	40,331	11.77	12.96	94.2	77.8	172,560	157,931
2004	299,869	24,989	10.97	11.05	95.5	77.6	102,009	83,912
2005	277,889	23,157	14.53	12.57	96.4	73.7	133,969	82,142
2006	584,713	48,726	10.50	10.43	95.8	70.0	187,172	132,789
2007	658,050	54,838	9.05	8.21	96.9	66.1	184,624	115,381
2008	758,382	63,199	7.18	11.57	95.7	61.9	167,059	170,330
2009	808,136	67,345	5.74	11.93	94.6	55.3	140,485	175,197
2010	827,498	68,958	4.81	6.47	93.9	49.0	120,577	88,651
2011	832,777	69,398	5.01	3.38	94.1	59.5	124,635	52,936
2012	876,185	73,015	4.67	2.80	94.9	62.7	125,127	49,912
2013	879,411	73,285	5.80	3.82	95.4	62.7	156,492	67,664
2014	866,867	72,239	7.71	4.56	95.4	64.0	203,962	81,016

Source: Koza, 2014

Table 2.10.3.2: Summary of Ovacık Process Plant Monthly Performance - 2013

Month	Ore Tonnes				Feed Grade		Recovery %		Poured Ounces	
	Ovacık	Çukuralan	Çoraklıktepe	Total	Au, g/t	Ag, g/t	Au	Ag	Au	Ag
January	17,876	56,462		74,338	3.53	2.34	94.7	57.9	9,169	4,080
February	12,574	52,794		65,368	4.90	3.07	96.0	66.9	9,006	3,664
March	14,754	57,950		72,704	4.76	2.60	94.7	62.5	9,544	3,594
April	5,609	65,223		70,832	5.44	2.80	95.2	65.1	13,576	4,795
May	701	63,646	10,496	74,843	6.28	3.78	95.1	62.1	12,035	4,690
June	828	54,388	13,688	68,904	5.21	4.05	95.5	64.5	11,444	5,443
July	90	63,068	13,400	76,558	5.47	4.54	95.5	57.5	13,858	6,959
August	1,270	61,800	15,078	78,148	6.28	4.93	95.4	63.1	13,023	6,175
September	4,808	57,986	11,864	74,658	6.47	4.93	95.6	58.5	13,857	7,232
October	9,935	58,981	7,051	75,967	6.94	4.57	95.6	61.1	18,174	7,883
November	5,674	63,781	3,769	73,224	7.24	4.38	95.6	67.3	15,115	6,181
December	11,430	62,006	431	73,867	6.87	3.56	95.5	67.8	17,693	6,968
Total	**85,549**	**718,085**	**75,777**	**879,411**	**5.80**	**3.82**	**95.4**	**62.7**	**156,494**	**67,664**

Source: Koza 2013

Table 2.10.3.3: Summary of Ovacık Process Plant Monthly Performance - 2014

Month	Ore Tonnes				Feed Grade		Recovery %		Poured Ounces	
	Ovacık	Çukuralan	Çoraklıktepe	Total	Au, g/t	Ag, g/t	Au	Ag	Au	Ag
January	4,715	56,188	13,092	73,995	6.60	3.91	95.1	62.5	13,388	5,194
February	2,077	52,399	12,786	67,262	8.13	4.28	95.4	65.5	15,837	6,137
March	10,414	53,356	9,536	73,306	7.21	4.21	94.5	63.8	15,426	5,850
April	6,795	53,772	9,149	69,716	6.99	4.62	94.6	69.1	17,232	7,688
May	9,427	59,443	829	69,699	8.28	4.42	94.8	70.0	14,995	6,156
June	12,614	56,492	2,311	71,417	8.12	5.31	95.1	62.0	16,966	6,810
July	14,968	55,565	3,347	73,880	8.35	4.47	95.3	69.7	16,473	6,808
August	17,822	51,847	4,938	74,607	7.20	4.62	95.5	66.0	19,260	8,121
September	3,356	60,832	8,659	72,847	7.91	4.81	95.6	61.2	21,029	8,713
October	1,950	63,749	9,157	74,856	8.27	4.61	96.3	62.1	17,066	5,919
November	3,720	58,877	9,022	71,619	8.22	4.47	96.3	62.8	16,908	6,176
December	21,001	43,130	9,533	73,664	7.34	4.98	96.1	53.8	19,383	7,443
Total	**108,858**	**665,650**	**92,359**	866,868	**7.71**	**4.56**	**95.4**	**64.0**	**203,962**	**81,015**

Source: Koza 2014

Gold recovery from the Ovacık process plant has been remarkably consistent over the years, ranging from 94 to 96%. During 2014 gold recovery averaged 95.4%. Silver recovery has been much more variable over the years, ranging from almost 78% during the initial production years and declining to as low as 49% (2010) as the silver grade declined and ore sources changed. Silver recovery averaged almost 64% during 2014. Ore will continue to be sourced from Çukuralan during the next several years and it is reasonable to expect gold and silver recoveries in the range of 95% and 60%, respectively.

Operating Costs

Unit operating costs for the Ovacık process plant are presented in Table 2.10.3.4 for the years 2011 to 2014. Operating costs for 2013 were US$11.61/t, and declined to US$10.87/t during 2014.

Table 2.10.3.4: Summary of Ovacık Process Plant Operating Costs

Item	2011		2012		2013		2014	
	TL	US$	TL	US$	TL	US$	TL	US$
Chemicals	3.93	2.31	4.00	2.25	3.89	2.06	3.63	1.92
Materials	4.07	2.39	4.78	2.69	4.55	2.41	4.67	2.47
Hourly Labor	1.51	0.89	1.57	0.88	1.76	0.93	1.76	0.93
Salaries	1.08	0.64	1.02	0.57	1.08	0.57	1.17	0.62
Energy	4.45	2.62	5.20	2.92	5.58	2.95	4.71	2.49
Maintenance	2.54	1.49	2.86	1.61	2.74	1.45	2.46	1.30
Contractors	1.72	1.01	2.01	1.13	2.15	1.14	1.98	1.05
Other	0.26	0.15	0.61	0.34	0.19	0.10	0.17	0.09
Total per tonne	**19.57**	**11.51**	**22.06**	**12.39**	**21.94**	**11.61**	**20.54**	**10.87**
Average Exchange Rate		**1.7**		**1.8**		**1.9**		**2.2**

Source: Koza 2014

Plant Labor

Plant operations are based on four shifts of five people per shift including a shift foreman. Two people are employed in the gold room and a further two plant metallurgists plus one for the metallurgical laboratory making a total complement for plant operations of 25 people. Additional personnel are available on day shift for cleanup and for other miscellaneous activities although these are not covered under the plant budget.

Maintenance has a common management and maintenance planning department for both the underground mining and plant. Total maintenance personnel dedicated to the plant include four technicians for electrical/instrumentation, and nine mechanics plus two supervisors.

The laboratory has a total staff of 14 people, mainly on day shift. The operation of the laboratory, which analyzes geological, mining and metallurgical samples, is monitored under a separate cost center and the costs are not included under the plant costs.

2.10.4 Infrastructure and Services

Electrical Power

Electric power for the plant is supplied from the Turkish national grid and reliability is reported to be good (>99% availability). Line and supply system maintenance downtime is scheduled in advanced and the time available used for plant maintenance. A 1,000 kVA standby emergency generator is available on site for critical drives in the event of power failure (leach and CIP tank agitation, thickener, etc.), which replaced the 500 kVA unit originally installed.

Water

The majority of the plant process water requirement is obtained from reclaim water returned from the tailings pond. Make-up water is obtained from Ovacık underground workings and from water wells in the valley close to the mine site.

2.11 Tailings Storage Facility

Parts of the following section were excerpted from the Ovacık Gold Mine TSF Management System Report Eurogold Madencilik A.Ş., December 1999 and formatted to fit this report. Information on the second TSF is based on the EIA of Ovacık Gold Mine Capacity Increase Project.

2.11.1 TSF Construction Parameters

Ovacık Gold Mine consists of an open pit and underground operation as well as a CIP plant originally designed at a capacity to process 300,000 t/y but it is now processing up to 900,000 t/y ore. The original tailings storage capacity was 1.6 Mm^3, but the TSF was expanded to a total capacity of 3.84 Mm^3.

To accommodate tailings from the Ovacık Mill processing ore from other nearby Koza deposits, the TSF capacity was increased. The final increase was accomplished by adding a 3 m lift to the Ovacık TSF. The first TSF has reached its capacity and is currently being closed and reclaimed.

The TSF consists of a main embankment and upstream embankment within an existing dry streambed. The TSF includes the embankments, surface water collection pond, diversion channels and other surface water runoff controls. Treated tailings (cyanide destruction, metal stabilization) from the CIP plant are pumped to the TSF for disposal. Supernatant is collected in the center of the TSF and returned to the plant for reuse.

The lining system in the Ovacık TSF consists of a 1.5 mm thick geomembrane between two layers of clay. The lower clay layer is 50 cm thick. The lower clay layer is overlain by a high-density polyethylene (HDPE) type geomembrane with a thickness of 1.5 mm. The HDPE geomembrane is covered with a 20 cm layer of clay. The upper clay layer is covered by a protective 20 cm drainage/filter layer. The lining system extends to the interior crest of both embankments. Drainage piping is spaced on 20 m centers. Tailings water drains via gravity flow in the drainage piping to the center of the TSF.

A second TSF with a total capacity of 4.87 Mm^3 was constructed to the southwest of the first TSF in two stages. The first stage of the new TSF was completed during 2009 and the second stage was completed in June 2011. The second TSF was established on the adjacent agricultural land using waste rock from the Ovacık mine operations. The second TSF is lined in a similar fashion to the first TSF. The second TSF has two years of remaining storage capacity.

A third TSF is currently being designed within the depleted Ovacık open pit. The technical designs as well as the EIA permitting process for the third TSF is currently continuing.

2.12 Environmental

2.12.1 Mine Operations

Ovacık ore production is performed by underground mining methods. The open pit, which operated in the past, ceased production in September 2007. The site contains a cyanide tank leach facility for ore beneficiation and a TSF for the deposition of detoxified tailings from gold extraction process. The first TSF is currently under closure and reclamation work. A second TSF was constructed and is currently in use. The construction of the TSF was conducted under the supervision of the State Hydraulic Works (DSİ). In addition to the ore from the Ovacık underground mine, the mill receives ore from the Çukuralan, and Çoraklıktepe deposits and will receive ore from Kubaşlar and other nearby deposits that Koza may decide to mine in the future.

Mine operations and waste disposal at the Ovacık Mine are carried out in compliance with the environmental regulations provided by the Ministry of Environment and Urban Planning (MoEU). Groundwater and the direct precipitation into the open pit are mixed with water from underground operations and then are discharged to the nearby Nardal Stream. The quality of water from underground operations is continuously monitored. The mine water is treated in a settling pond before mixing with water in the open pit. An open pit and TSF water quality monitoring program has been developed in accordance with the official requests. The monitoring results are reported on a monthly basis.

2.12.2 Permitting

The Ovacık Mine operations started in 2001. The facility has been owned and operated by Koza since 2005. The mine ceased operation for a one-month period in February 2009 as a result of a lawsuit regarding the EIA procedures followed by the MoEU. The EIA report was revised to take into account the court's decision. Following the approval of the revised EIA report, and renewal of other environmental permits, the mine facility resumed its operation in March 2009. The Ovacık Mine currently has the environmental permit and license valid to December 02, 2018 from the MoEU.

In June 2009, the EIA permit for the second TSF was obtained. Currently, the EIA permitting process for the third TSF is being conducted. The EIA process for the 3rd capacity increasing is ongoing.

2.12.3 Waste Control and Monitoring

The geochemical analyses for Ovacık mine lithologies indicate that the waste rock does not pose Acid Rock Drainage (ARD) generation risk. Therefore, complex closure designs for particular ARD prevention and control measures are not required for waste rock disposal areas. Waste rock is suitable to be used as general fill material. Disposal areas can be rehabilitated by providing organic soil cover to support revegetation.

Metals immobilization and cyanide destruction in tailings material is achieved through detoxification. After detoxification, the cyanide level in the tailings drops below one tenth of EU standard (European Directive 2006/21/EC), 1 mg/L. According to the waste characterization analysis, tailings material is classified as non-hazardous waste. Following detoxification, tailings are sent to on-site TSF's for disposal. The storage facility is constructed with an impermeable liner. The TSF is operated in accordance with the "zero discharge principle". The TSF design and construction is based on appropriate stability constraints.

It has been determined that the entire operation is in compliance with the current regulations, especially in regard to collection of solid wastes, disposal of waste water, collection and disposal of waste oil, and other hazardous waste materials. The Ovacık mine operates in accordance with the appropriate regulatory authorizations and government approvals.

A Reference Document on Best Available Technique (BAT) for Management of Tailings and Waste Rock in Mining Activities published in 2009 by European Commission European IPPC Bureau cites tailings and waste rock management practices and waste rock dump rehabilitation applications in Ovacık Mine as a successful application of BAT.

2.12.4 Closure

The Ovacık Mine Reclamation Plan based on official regulatory formats was submitted in December 2008 to MoEU. On the other hand, a comprehensive Mine Closure and Reclamation Plan detailing the closure costs and plans does not yet exist. However, Koza has made some preliminary cost estimations for mine closure. Accordingly, Koza estimates that it will cost approximately US$7.8 million to close the Ovacık mine, of which US$3.8 million accounts for closure of the open pit. However, this cost estimate does not take into account the currently designed third TSF and other factors such as staff compensation etc. The new TSF design involving utilization of the open pit is likely to change the total closure costs. The change will depend on the geochemical and hydrogeological assessment of the new TSF. Any mitigation measure required further to the existing ones may drive up the final closure costs. However, the savings that will be achieved by not partially filling the open pit may offset those extra costs. The actual effect of these different factors should be determined within a Mine Closure and Reclamation Plan.

It should be noted that the first TSF is currently being closed and reclaimed. The first TSF closure involves drying out of TSF by evaporation, covering with 1 m of waste rock, spreading multilayered cover preventing infiltration into TSF, and revegetation of surface for new land use. The closure of the first TSF is roughly estimated to be US$1.3 million, which is currently being expended as part of the progressive closure.

Koza is liable for the environmental performance of the Ovacık operations for a period of 30 years after the closure of the mine.

2.13 Conclusions and Recommendations

2.13.1 Geology and Resources

Koza is using all grade control samples in the estimation as well as drillhole samples. Because the grade control samples are much more closely spaced than the drillhole samples, Koza has used a restricted search in order to limit the influence of the higher grade samples. Koza is using the wireframes as hard boundaries and uses only composites within the individual wireframes for estimation. SRK considers these good estimation practices.

The resource classification system is based on kriging variance and on drillhole spacing.

Koza validates the model by visual comparison of composites versus block grades and comparison of the block and composite grades. These are acceptable methods for validation for an operating mine. However, Koza is not comparing the block and composite grades for each vein type. The comparison should be done by domain and vein type for the grade control and drillhole models separately. If the model does not validate properly, then the parameters should be changed and the model re-run and re-validated. SRK suggests that Koza should also use swath plots as another validation method. The swath plots are valuable for showing problem areas where the resource grades may be over- or under-estimated.

The very large discrepancy between the planned mining tonnes predicted by the 2013 resource model/mine plan and what was actually mined in 2014 (ascribed to variation in predicted and actual vein width and grade) should be investigated and appropriate corrective actions taken going forward. It is SRK's opinion that it is very difficult to produce a reconciliation where waste blocks are excluded because that tonnage is not accounted for in the mined tonnage.

SRK recommends that Koza review the model to locate areas that have not been depleted by the mining as-builts so that remnant and/or isolated blocks are depleted from the model.

The conversion rate of Measured and Indicated Resources to Proven and Probable Ore is very low at 10%. SRK would expect that the conversion rate would be much higher for a mature mine. This brings into question of whether much of the resource is actually potentially mineable.

2.13.2 Mining and Reserves

The underground operations at Ovacik are planned to wind down in 2015 where mine production rates are planned to fall to just 100t/d through the planned end of mining in 2021.

2.13.3 Metallurgy and Process

The Ovacık plant is in good condition and achieves consistently high metallurgical performance.

The plant operations at Ovacık are running well and few technical risks are envisaged for the treatment of ores similar to those from Ovacık, Küçükdere, and Çukuralan. Additional proposed feeds would need to be tested for amenability to the relatively simple leach and CIP process installed at Ovacık and to confirm relevant operating parameters (optimum grind size, reagent additions etc.).

2.13.4 Environmental

Mine operations and waste disposal at the Ovacık Mine are carried out in compliance with the environmental regulations provided by the Ministry of Environment and Urban Planning. The Ovacık Mine currently has the environmental permit and license valid to December 02, 2018 from the MoEU.

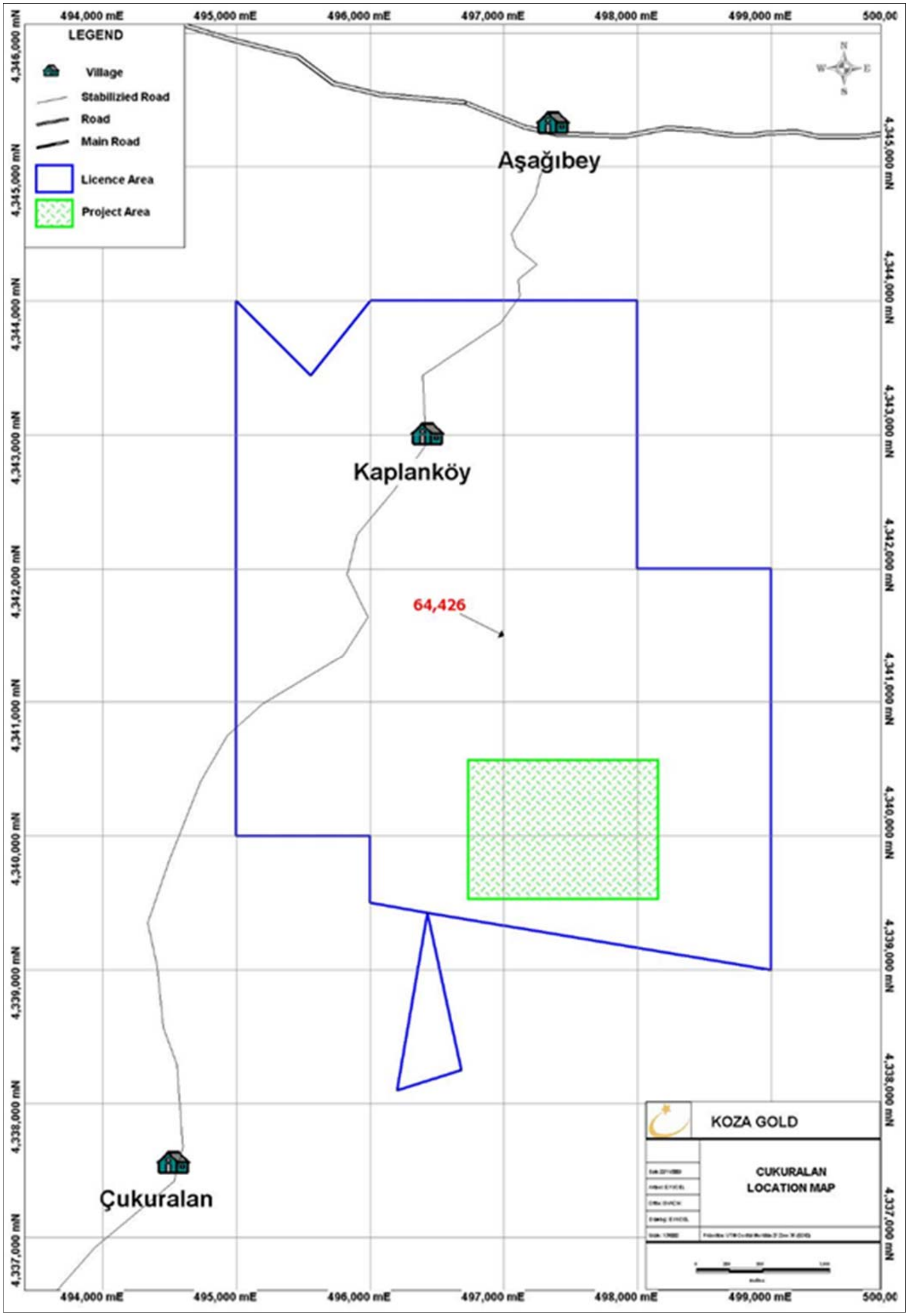
3 Çukuralan Mine Resources and Reserves

3.1 Property Description and Location

The Çukuralan Mine is located near the village of Kaplan in western Turkey at UTM coordinates 4340500 N, 496500 E to 4339500 N, 498500 E, ED1950 Zone 35. Çukuralan is located approximately 40 km northwest of the Ovacık Mine within a National Forest and upland from the Aegean Sea. The Çukuralan Mine is located at approximately 200 m amsl.

The Çukuralan Mine is currently under operation license 64426, which comprises approximately 1,628 ha and a permit for gold and silver covering approximately 525 ha of the operation license. This license will require renewal in 2018. Land Tenure for the mine is shown in Figure 3.1.1.

Operations started at the Çukuralan Mine in late 2010 and ore is shipped to the Ovacık Mill for processing.



Source: Koza, 2012 GIS

Figure 3.1.1: Çukuralan Location Map

3.2 Climate and Physiography

The Çukuralan Project is located in the Ovacık District and experiences a typical Mediterranean climate. Detailed discussion of the climate and physiography of the Ovacık District is found above in Section 1.1.1.

3.3 History

There is evidence of Roman era, small-scale open pit and underground mining on the property, some of which have been exposed in the current open pit. The Çukuralan exploration project was previously held by Normandy, a subsidiary of Newmont. Normandy performed regional sampling in the area in 1995, collecting five stream samples for BLEG analysis near the Çukuralan property. Although, a favorable anomaly was found in this area, no follow up was performed by Normandy. Koza acquired the property in 2005.

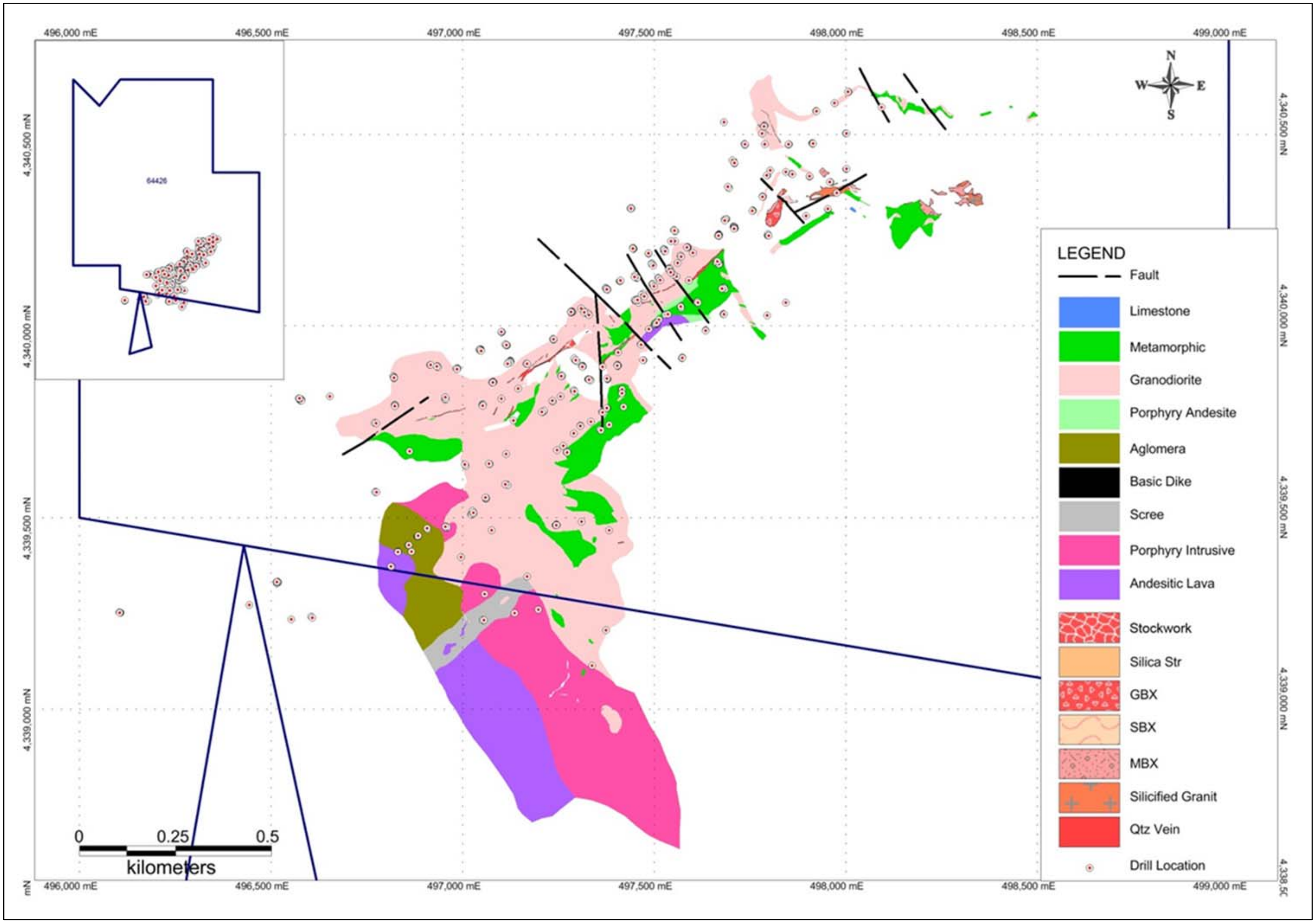
3.4 Çukuralan Mine Geology

The Çukuralan Mine is located the Ovacık District. Regional geology of the Ovacık District is discussed in Section 1.1.2.

Çukuralan is described as a low sulfidation, epithermal gold vein system with associated breccias. Mineralized veins and breccia zones strike approximately N60°E and dip from 65°NNW to near vertical in the southwest and 80°SSE near the northeast section of the vein zone. Locally, the strike can vary by 20 to 30°. The exposed vein system ranges from approximately 2 m in thickness at the southwest end to 40 m at the northeast end of the system, and mineralization can be traced up to 1.7 km in outcrop.

Çukuralan mineralization is structurally controlled and has been preferentially emplaced along a contact between late Oligocene to early Miocene age Kozak Magmatic Complex multi-phase intrusive rocks and xenoliths of the Triassic age Kapıkaya Metamorphic Complex. The intrusive rocks are composed of feldspar, quartz and biotite, while the older xenoliths are hornfels schist from meta-sediments and marble derived from limestone. This contact zone extends for 2 km and has epithermal style alteration along its entire length. It is interpreted that the intrusive rocks were emplaced during caldera collapse as a shallow-level granitic-granodiorite intrusion preferentially along the contact with the hornfels schist xenoliths as a result of more open space. The hornfels schist is the more favorable host rock. Graphite present in the xenoliths also contributed to deposition of sulfide minerals. Other xenoliths found in the area include marble, fine-grained sedimentary rocks and conglomerate. In the southern part of the project, the granite-granodiorite rocks are overlain by trachytic andesite volcanic rocks of Miocene age.

Pathfinder elements including As, Hg, Sb and Ag in the soil geochemistry indicate that the vein outcropping at surface is near the top of the hydrothermal system suggesting that most of the hydrothermal system is still present. Koza has interpreted the presence of copper and molybdenum in the soil samples as indicating a porphyry component to the mineralizing event and possible intermediate sulfidation, epithermal formation. Sulfide mineralization within the vein system is limited to pyrite and chalcopyrite, and alteration is composed of intense silicification near the veins and propylitic alteration in the intrusive host rocks. This vein system has been crosscut by a late stage northwest striking set of veins characterized as a low sulfidation, porphyry-related gold mineralization event. Figure 3.4.1 shows the local geology of the Çukuralan Mine.



Source: Koza, 2012 GIS

Figure 3.4.1: Çukuralan Geology Map

3.5 Exploration

Koza acquired the property in 2005 and work performed between 2005 and 2006 included detailed mapping, stream sediment sampling and soil sampling. This work identified a vein system which has been mapped over a strike length of 1.7 km with evidence in places of historic Roman pits and adits along the vein outcrop. Geophysics completed in 2007 and 2008 identified additional targets to the northwest and southeast of the main vein, which correlate directly with previous soil geochemistry work and has similar response to the outcropping vein in the anomalous area. In the northwest part of the main vein this response has been observed to a depth of approximately 200 m. Koza mapped the alteration assemblages using a Portable Infrared Mineral Analyzer (PIMA) as an additional technique of defining exploration targets during 2013. The Main Zone remains open at depth and in both the northeast and southwest directions.

At the end of 2014, Koza had completed 582 core holes testing the main vein zone to a depth of more than 400 m and intersecting additional veins to the northwest that directly correspond to the geophysics and soil geochemistry anomalies. Drill spacing ranges from 25 m in the center of the mineralization to 90 m on the edges. Koza has identified two additional target zones to the southeast. These are the NW Zone that has been traced in continuous and discontinuous veins over 3,000 m and the KOZ Zone, which exhibits more offsets and less continuity but the zone has been traced over 1,500 m. Koza did not drill in the extension zones in 2013 and 2014 because they were waiting on permit approval for drill sites. Exploration was focused on mapping and surface sampling. Koza has budgeted TL781,500 (US$347,000) for drilling during 2015.

3.6 Drilling/Sampling Procedures

Koza uses its standard methods for drilling, logging, and sampling at Çukuralan. Koza has a drilling foreman who is a geologist that oversees all coring operations company wide. Core is drilled by a contractor and is in the custody of the contractor until a Koza employee removes the core and brings it to the core facility for logging. This is a secure facility within the mine site and controlled by Koza security. The drillhole collars were originally surveyed by the Ovacık Mine survey team and are now surveyed by the Çukuralan survey team. The holes are measured for downhole deviation by the drill contractor, using a Flexit SmartTool Drillhole Survey System.

Core is photographed and then logged onto paper logs and transferred to the computer for both lithologic and geotechnical properties. Core is marked for sampling by a geologist and the core is cut lengthwise using a core saw. The core is sampled on 1 m intervals, with adjustments at geologic boundaries. The core is selectively sampled based on the presence of quartz and quartz veinlets. Samples are collected in a plastic bag with sample numbers written on the bag and a sample tag is inserted into the bag.

The surface core recovery averages 97% and ranges from 0 to 100%; the underground core recovery averages 89% and ranges from 0 to 100%.

Koza also collects underground grade control samples at Çukuralan. All samples are used in the resource estimation. Table 3.6.1 summarizes the drilling and sampling at Çukuralan and Figure 3.6.1 shows the drillholes and face samples in plan view.

Table 3.6.1: Summary of Drilling and Sampling at the Çukuralan Mine

Type	Drillholes			Face Samples	
	Number	Meters	Samples	Number	Meters
Surface Drilling	582	182,047	94,667		
Underground Drilling	273	10,782	8,719		
Face Samples				17,841	21,083
Total	**855**	**192,829**	**103,386**	**17,841**	**21,083**

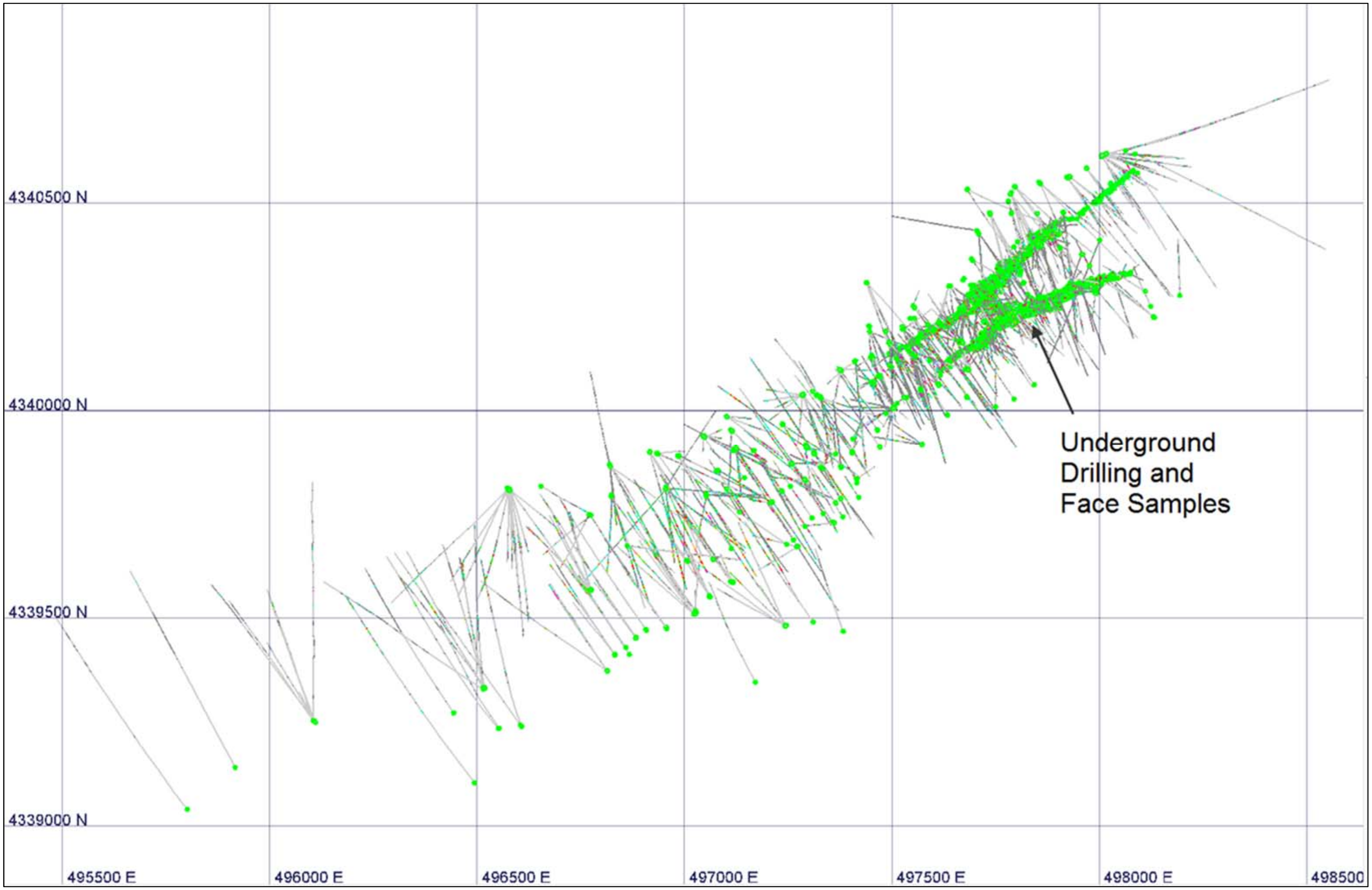


Figure 3.6.1: Drilling at Çukuralan, W Face Samples

3.6.1 Laboratory Procedures

Through mid-2013, gold was analyzed 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), 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 was 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 3.6.1.1 presents the upper and lower detection limits for ALS cod ME-ICP61.

Table 3.6.1.1: Analytes and Upper and Lower Detection Limits for ALS Code 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

After 2013, samples were analyzed at either the Kaymaz or Ovacık laboratories. 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; and
- Ag by aqua regia and AAS finish with a lower detection limit of 0.2 ppm.

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.

3.6.2 Quality Assurance and Quality Control (QA/QC)

Koza has a laboratory QA/QC program in place which consists of:

- Reference Material samples;
- Blanks; and
- Preparation Duplicates.

The Koza QA/QC procedures were reviewed by Lynda Bloom, Principal at Analytical Solutions LTD in 2013 (Bloom, 2013).

All Çukuralan control samples have been monitored for Au. The Ag results for the control samples were not provided to SRK and it does not appear that the Company is currently monitoring the Ag control samples data. Because Ag is included in the resource, SRK recommends Koza also monitor Ag results.

All of the core samples with the exception of samples from one drillhole were sent to the Koza lab. One drillhole was submitted to ALS.

Reference Materials

Throughout the life of the Project, 25 different RMs have been used at Çukuralan: five were CRMs produced by Rock Labs based in New Zealand and 18 site-specific RMs with material from the Ovacık (OV), Kaymaz (K) and Çukuralan (C) mines.

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 by FA with AAS finish and silver was analyzed by FA with ICP finish.

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 site-specific RMs produced by Koza, Bloom (2013) recommends using ±7% as a threshold for a failure based on her communication with ALS.

Table 3.6.2.1 presents the expected mean, standard deviations and summaries of the analyses of the Au RMs used in 2013. These are the RMs submitted to the Koza lab. The four RMs are from material specific to Çukuralan.

Table 3.6.2.1: Results of 2013 Au RM Analyses at Çukuralan – Site Specific

RM	Number of Samples	Expected (ppm)		Observed (ppm)		% of Expected	Outside ±7%	
		Mean	Std Dev	Mean	Std Dev		No. Failures	% Failure Rate
C1	51	0.674	0.004	0.705	0.014	104.6	0	0.0
C2	46	1.387	0.046	1.440	0.015	103.8	0	0.0
C5	24	0.803	*	0.808	0.012	100.6	0	0.0
C6	4	2.155	*	2.112	0.083	98.0	0	0.0
Total	**125**						**0**	0.0

*SRK does not have the certificate standard deviation for this RM. However, standard deviation is not used in the performance gate.

There were no RM failures for gold during the drilling program. All the results for C1 and C2 were higher than the expected mean. There were 16 results (66.7%) that were higher than the expected mean for C5 and all of the C6 results were lower. There are only four results for C6, but C6 is the highest grade RM and this is where the lab has low results. The three lower grade RMs indicate that there may be a high bias in that grade range. However, the performance for C6 suggests that there may be concern in the higher grades. More data is required to fully assess C6, but SRK recommends that Koza monitor C6 performance to determine if there is an analytical problem with higher grade samples.

There were four RMs submitted to ALS; two C1 and two C2. There were no failures in the RM submissions but both RMs had results that were higher than the expected mean. C1 was 109.7% and C2 was 103.1% of the expected mean.

The performance of the RMs may be related to the manner in which they were created and assigned grades.

Bloom (2013) has made the following recommendations regarding RM samples:

- Use commercially available CRMs for common deposit types (i.e., there are many available RMs for low sulfidation systems);
- For site-specific RMs, purchase appropriate equipment for pulverizing, mixing and sub-sampling large batches (>500 kg); or
- Prepare a site-specific standard at a facility that specializes in RM production, such as OREAS that can prepare up to 5,000 kg at one time. Costs are usually half of the cost of purchasing pre-packaged commercial RMs; and
- The current practice of preparing 100 kg with inadequate round robin data is not recommended.

Blanks

Sample blanks test for contamination in preparation and assaying and handling errors. Koza inserted one sample blank per drillhole using pulp blanks up until June 2012 and preparation blanks since then. A blank failure is a result greater than five times the detection limit. SRK has examined the results for gold in the blank samples and finds that there were no failures in the 2014 data.

Field Duplicates

Field duplicates are created by sampling a second quarter of the split core. The objective of testing field duplicates is to estimate the variance of the actual sampling and the first size reduction step. Field duplicates provide information on deposit mineralization but not necessarily on sampling and assaying QA/QC. Once the variability within the deposit is understood, then the use of field duplicates could be discontinued.

In the past, Koza had prepared 561 field duplicates and submitted them to the primary lab. The data showed a slight bias between original and duplicate with bias for the original over the duplicate. Fifty-seven percent of the field duplicates were outside +/- 30%, which is not uncommon for gold deposits. Koza has assessed the variability of mineralization and taken action to compensate for this in submitting larger sample splits. It is SRK's opinion that this is an acceptable practice at Çukuralan.

Preparation Duplicates

Preparation duplicates are created by splitting a second cut of the crushed sample (coarse reject) in the same way and for 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.

Koza sent preparation duplicates to the primary lab for analysis. The 2014 duplicate analysis data provided to SRK includes 61 duplicate pairs with Au values. Of these, 22 duplicate pairs had samples at or above the detection limit and four were above the resource 1.65 g/t cutoff grade for gold. A summary of the analytical results are presented in Table 3.6.2.2 for those samples at or above the detection limit.

Table 3.6.2.2: Summary of Preparation Duplicate Au Analysis at Çukuralan at or above the Detection Limit

Criteria	Number of Samples	Original>Dup	Dup>Original	Original = Dup	Within +/- 20%
All samples	22	12	2	8	21
		54.5%	9.0%	36.3%	95.4%

Overall, the results of the Çukuralan preparation duplicates indicate that the sample preparation is adequate for the analysis and that the preparation duplicates are demonstrating analytical precision. Preparation duplicates need to be submitted from mineralized material in order to test the variability of the mineralization and confirm analytical precision in the range of mineralization. SRK recommends that Koza continue to submit coarse duplicates, and that samples should be submitted in the resource grade range rather than at low gold levels.

Pulp Duplicates

Koza has not submitted any pulp duplicate samples to the primary lab. Pulp duplicates are the primary method of checking the precision of analysis. SRK recommends that the Company begin sending pulp duplicates as part of its QA/QC program or monitor the internal pulp duplicates produced and analyzed by the primary lab as part of its internal QA/QC program.

Secondary Check Lab Analysis

In 2013, Koza used SGS in Ankara as the secondary laboratory. Koza submitted a total of 25 duplicates originally analyzed at ALS to SGS Ankara for verification of results. The data showed significant bias between the laboratories with ALS higher than SGS. There were no CRMs submitted with the data so it was difficult to determine which laboratory had better accuracy.

Currently the Çukuralan samples are being submitted to the Koza laboratories. SRK recommends that Koza conduct a check assay program inserting CRM samples at a frequency of one CRM for every five to six pulps sent to the secondary lab. This way Koza can assess the reproducibility between the laboratories and accuracy of analysis at the secondary laboratory.

Conclusions and Recommendations

Koza monitors QA/QC of the laboratory analyses by inserting internal control samples into the sample stream. Certified reference materials, blanks, preparation duplicates and secondary check lab analyses 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 entire batch be reanalyzed. This is industry best practice.

SRK has the following recommendations:

- Ag results for the RMs, blanks and duplicates should be monitored;
- 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;

- Plot QA/QC data individually for each laboratory;
- Duplicate samples should be within the resource grade range;
- Pulp duplicates should be prepared and submitted to the primary laboratory; and
- A check assay program should be reinstated and CRMs samples should be submitted with the check assay samples.

Overall, the QA/QC program is sufficiently monitoring laboratory accuracy and reliability.

3.7 Mineral Resources

The mineral resources were estimated by Koza in 2014 (Koza, 2014b).

3.7.1 Geological Model

A total of 103,386 drillhole samples and 21,083 underground grade control samples were assayed. Koza generated more than 90 separate wireframes of the mineralization using a cutoff grade of 0.5 g/t Au. Several faults displace the mineralized zones. The wireframes have been grouped into five domains based on structural orientation. Domains 1, 2, 4 and 5 strike to the east-northeast and together cover a strike length of 2,700 m. Domains 1 and 5 dip about 55° to the northwest and Domains 2 and 4 dip about 60° to the southeast. Domain 3 strikes to the northwest and dips about 55° to the northeast. Together, the five domains have a vertical extent of about 300 m.

There are 10,804 drillhole samples and 8,995 face samples located within the wireframes. The five wireframe domains are shown in plan view in Figure 3.7.1.1 and oblique view in Figure 3.7.1.2. Tables 3.7.1.1 and 3.7.1.2 present statistics of the drill and grade control samples within the wireframes.

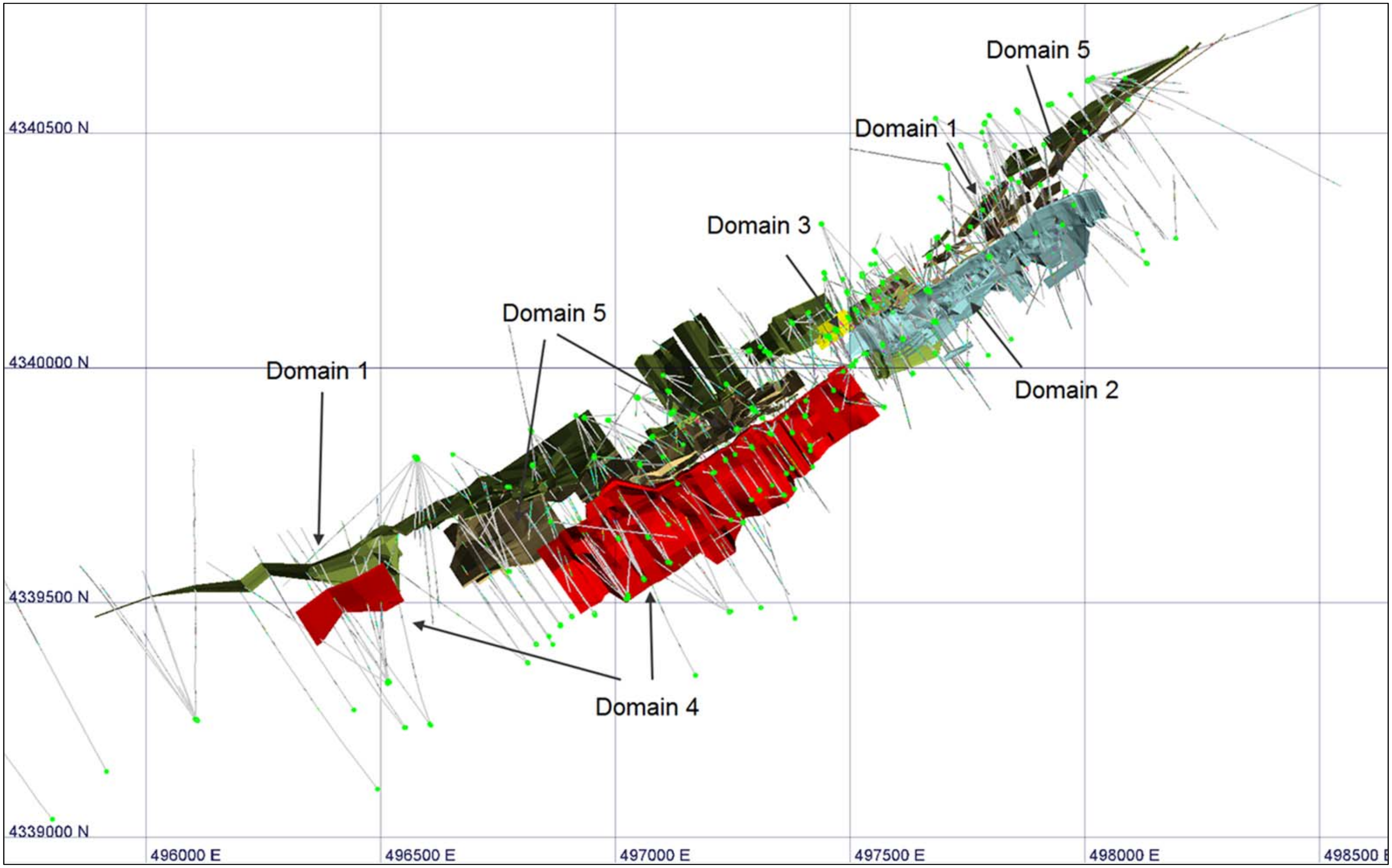


Figure 3.7.1.1: Plan View of Drilling and Mineralization Wireframes at Çukuralan

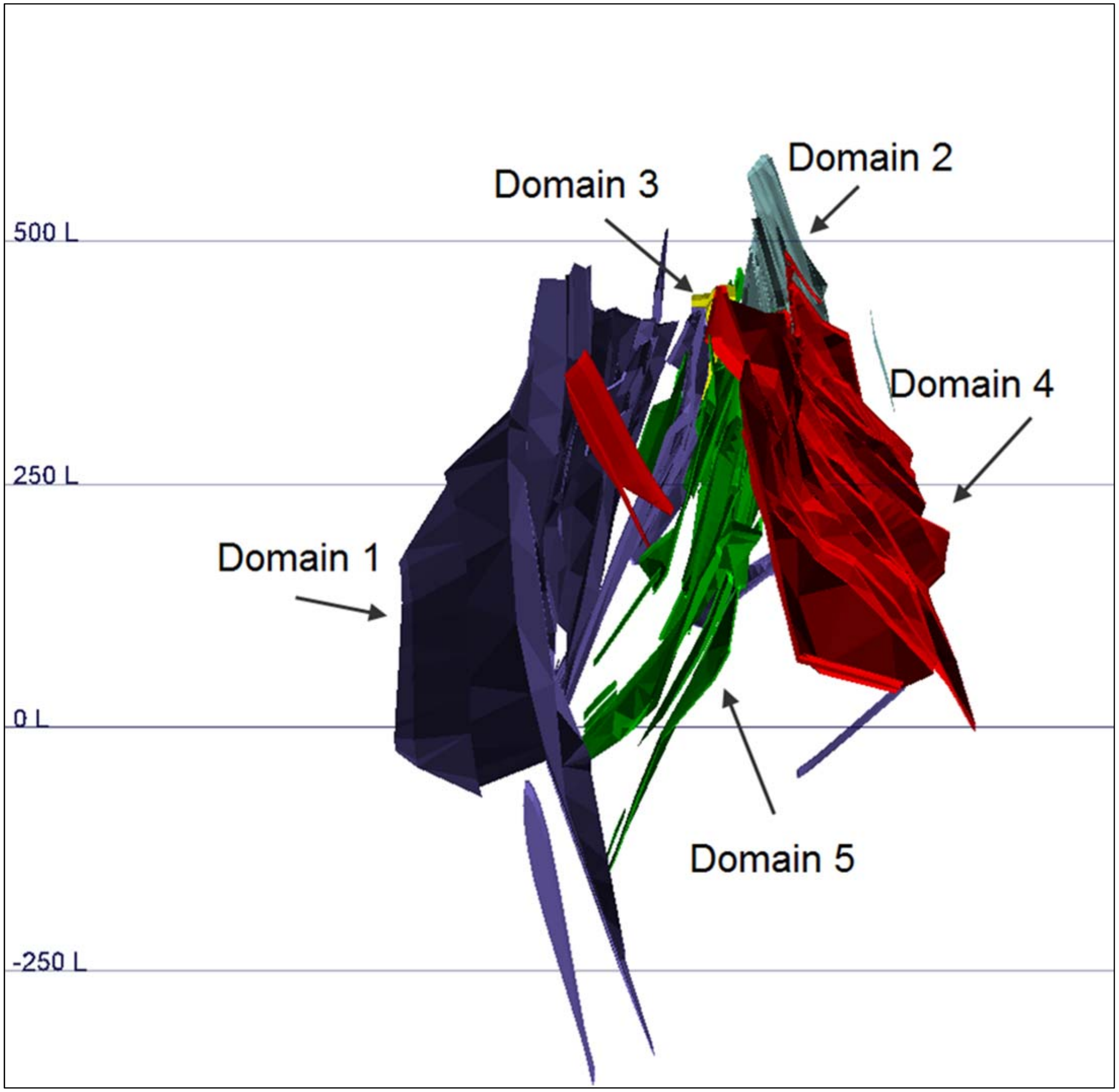


Figure 3.7.1.2: Oblique View of Mineralization Wireframes at Çukuralan, Looking Northeast

Table 3.7.1.1: Çukuralan Statistics of Drill Samples in the Domains

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
1	Au	3,820	0	187.5	4.94	13.41	2.72
	Ag	3,879	0	1,490.0	1.96	25.35	12.95
2	Au	2,545	0	712.9	6.22	24.86	3.99
	Ag	2,589	0	236.5	1.35	6.45	4.78
3	Au	112	0	79.7	6.81	12.56	1.84
	Ag	114	0	25.4	1.73	3.76	2.18
4	Au	2,060	0	285.9	4.15	15.91	3.84
	Ag	2,060	0	79.3	1.08	3.73	3.46
5	Au	2,267	0	268.0	4.84	15.08	3.12
	Ag	2,268	0	82.0	1.68	4.93	2.94
All	Au	10,804	0.00	712.9	4.84	17.55	3.45
	Ag	10,910	0.00	1,490.0	1.68	15.56	9.85

Table 3.7.1.2: Çukuralan Statistics of Grade Control Samples in the Domains

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
1	Au	2,758	0	307.1	5.50	10.41	1.92
	Ag	2,758	0	108.2	2.95	3.68	1.34
2	Au	6,071	0	3,017.0	15.52	35.78	2.86
	Ag	6,071	0	385.3	5.58	8.95	1.94
5	Au	166	0	84.4	7.15	12.18	1.9
	Ag	166	0	45.7	3.99	5.61	1.54
All	Au	8,995	0	3,017.0	6.72	32.36	4.81
	Ag	8,995	0	385.3	2.90	7.69	2.65

3.7.2 Density

Density measurements have been taken on 537 HQ sized core samples. The results have been sorted by lithology and by depth below surface. The average of all vein samples is 2.47 g/cm^3 and that value is used in the resource estimate.

3.7.3 Core Recovery

The core recovery is excellent overall, averaging in excess of 97%. However, there are some intervals within the wireframe where the recovery is low and, therefore, intervals with less than 50% recovery were excluded from the resource estimation. This has affected about 10 samples.

3.7.4 Capping and Compositing

Koza conducted an investigation of sample lengths to determine the compositing length. About 90% of the drill samples are 1 m in length. Koza chose 1 m as the composting length for the drill samples. The lengths of the face samples show more variation and Koza chose to composite over the width of the vein. This results in considerable variation in the composite length, ranging from 0.1 to 8.5 m and averaging 1.15 m. SRK suggests that it would be better to use the same compositing length of 1 m or another standard length for the face samples. Statistics of the composites are shown in Tables 3.7.4.1 and 3.7.4.2.

Table 3.7.4.1: Çukuralan Uncapped Drill Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
1	Au	3,547	0	182.5	4.94	12.07	2.44
	Ag	3,568	0	918.08	2.02	20.85	10.32
2	Au	2,393	0	461.78	6.22	21.12	3.39
	Ag	2,394	0	145.99	1.36	5.43	4.01
3	Au	103	0.07	79.7	6.98	11.21	1.61
	Ag	111	0	25.4	1.72	3.41	1.98
4	Au	2,001	0	285.9	4.16	14.6	3.51
	Ag	2,001	0	79.28	1.08	3.37	3.12
5	Au	2,120	0	217	4.86	13.32	2.74
	Ag	2,120	0	81	1.69	4.56	2.70
All	Au	10,164	0	461.78	5.09	15.38	3.02
	Ag	10,194	0	918.08	1.61	12.88	8.01

Table 3.7.4.2: Çukuralan Uncapped Grade Control Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
1	Au	886	0	48.90	5.41	6.44	1.19
	Ag	886	0.16	18.61	2.74	2.34	0.85
2	Au	1,581	0	1,209.33	12.49	20.69	1.66
	Ag	1,581	0.18	78.53	4.60	4.95	1.08
5	Au	81	0	84.41	6.40	9.62	1.50
	Ag	81	0.34	45.66	3.64	4.68	1.29
All	Au	2,548	0	1,209.33	10.12	17.60	1.74
	Ag	2,548	0.16	78.53	3.99	4.37	1.10

Koza conducted a quantile analysis of the composite data to identify outlier grades. The capping values by domain are shown in Table 3.7.4.3. Tables 3.7.4.4 and 3.7.4.5 present statistics of the capped drill and face sample composites. The CV has been reduced to more acceptable values for resource estimation. SRK has reviewed the data and finds the capping values to be acceptable. In some cases, SRK would have chosen somewhat higher values.

Table 3.7.4.3: Çukuralan Capping Values for Gold and Silver

Variable	Domain	Drill Samples		Face Samples	
		Maximum	Capped Value	Maximum	Capped Value
Au	1	182.5	75	48.9	20
	2	461.8	82	1209.3	50
	3	79.7	14	NA	NA
	4	285.9	25	NA	NA
	5	217.0	50	84.4	20
Ag	1	918.1	25	18.6	13
	2	146.04	13	78.5	23
	3	25.4	6	NA	NA
	4	79.3	11	NA	NA
	5	81.0	33	45.7	7

Table 3.7.4.4: Çukuralan Capped Drill Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
1	Au	3,547	0	75	4.75	10.49	2.21
	Ag	3,568	0	25	1.48	3.09	2.09
2	Au	2,393	0	82	5.47	12.87	2.35
	Ag	2,394	0	13	1.05	2.41	2.29
3	Au	103	0	14	4.90	5.44	1.11
	Ag	111	0	6	1.32	1.89	1.44
4	Au	2,001	0	25	3.12	5.06	1.62
	Ag	2,001	0	11	0.93	1.66	1.77
5	Au	2,120	0	50	4.24	8.69	2.05
	Ag	2,120	0	33	1.62	3.91	2.42
All	Au	10,164	0	82	4.50	9.95	2.21
	Ag	10,194	0	33	1.30	2.92	2.25

Table 3.7.4.5: Çukuralan Capped Face Sample Composites

Domain	Metal	Number	Minimum	Maximum	Mean	Std Dev	CV
1	Au	886	0	20	5.07	5.07	1
	Ag	886	0	13	2.72	2.24	0.82
2	Au	1,581	0	50	11.87	12.52	1.05
	Ag	1,581	0	23	4.52	4.34	0.96
5	Au	81	0	20	5.71	6.77	1.19
	Ag	81	0.34	7	2.96	2.40	0.81
All	Au	2,548	0	65	9.82	11.80	1.20
	Ag	2,548	0.16	23	3.93	3.86	0.99

3.7.5 Variography

Koza conducted extensive variography studies on gold and silver composites in 2012 and updated Domain 5 in 2013. The results are shown in Table 3.7.5.1. In the gold variograms, the nugget is fairly high, ranging from about 30% to 50% of the total sill; in the silver variograms, the nugget ranges from 3% to 58% of the total sill.

Table 3.7.5.1: Çukuralan Variogram Parameters by Domain

Domain	Axis	Au						Ag					
		Orientation, Major Axis	Nugget	Sill 1	Sill 2	Range1 (m)	Range 2 (m)	Orientation	Nugget	Sill 1	Sill 2	Range1 (m)	Range 2 (m)
	Major					29	51					30	60
1	Semi-major	70, 326	35.0	32.47	48.30	26	75	70, 326	15.91	4.54	6.89	30	60
	Minor					5	15					30	60
	Major					16	43					47	132
2	Semi-major	70, 145	70.0	38.50	123.74	11	79	70, 145	3.84	2.13	30.19	47	132
	Minor					5	15					47	132
	Major					5	25					26	
3	Semi-major	60, 55	15.0	13.24	23.78	5	25	60, 55	3.60	5.24		26	
	Minor					5	25					26	
	Major					14	38					25	
4	Semi-major	- 60, 330	12.5	0.07	13.55	63	134	- 60, 330	1.86	2.24		25	
	Minor					5	15					25	
	Major					51	125					26	76
5	Semi-major	70, 326	25.0	19.50	32.22	14	75	70, 326	0.64	9.6	14.00	26	76
	Minor					5	15					26	76

3.7.6 Grade Estimation

A block model was created with a block size of 10 m x 10 m x 5 m with sub-blocking at 2 m by 1 m by 1 m in the x, y, z directions, respectively. The resulting blocks can be as small as 2 m x 1 m x 1 m, which is quite small for an open pit resource estimation. All blocks above current topography or which had been mined from underground were removed from the wireframe blocks. SRK suggests that in the future that mined out blocks not be removed, so that mined to model reconciliations can be done.

The block grades were estimated with ordinary kriging (OK) in five passes and the blocks were classified as Measured, Indicated or Inferred according to the pass in which they were estimated. Estimations using inverse distance squared (ID2) and nearest neighbor (NN) approaches were also conducted for comparison to the OK estimation. The estimation with face samples included only ID2. Dynamic anisotropy was used in the estimation whereby the search orientation is calculated on a block by block basis to provide for irregularities in the wireframe shape. Table 3.7.6.1 shows the primary search distances for the estimation. These reflect the range of the Au variogram in each domain

Table 3.7.6.1: Çukuralan Primary Search Distances (m)

Domain	Au, Ag		
	Major	Semi Major	Minor
1	51	75	15
2	43	79	15
3	25	25	25
4	38	134	15
5	125	75	15
Grade Control Model			
1, 2, 5	35	35	10

Grade Control Samples

In order not to extrapolate higher grades from the face samples into the main part of the orebody, a smaller search ellipse was used with a restriction the samples to be used. The estimation parameters included the following:

- A single estimation pass, using the primary distances shown in Table 3.7.6.1;
- Estimation by ID2 and NN; and
- Minimum of five and a maximum of 10 samples, with no maximum per drillhole.

The parent cell size was used in the estimation. A dynamic search was used where by the search ellipse closely follows the vein orientation. The estimation with the face samples took precedence over the estimation with surface and underground core holes.

Drill Samples

Five estimation passes were used with the surface and underground core samples. The search parameters are shown in Table 3.7.6.2.

Table 3.7.6.2: Çukuralan Estimation Parameters for Drillhole Model

Pass	Multiple of Primary Search Distance	Samples		
		Minimum	Maximum	Maximum/DH
1	0.66	15	30	9
2	0.66	10	15	9
3	1	5	20	4
4	1.5	5	20	4
5	5	2	20	4

Estimations were made with OK, ID2 and NN. A dynamic search was used to follow the vein structure.

The structural domains were used as hard boundaries and only composites with the same code were used for each individual domain.

3.7.7 Block Model Validation

Koza validated the model by visual comparison of composites to block grades, comparison of mean composite grades to mean block model grades (Tables 3.7.7.1 and 3.7.7.2), comparison of the three estimation techniques (Table 3.7.7.3) and swath plots showing composite and block grades by domain.

The estimated gold values in the drillhole model are higher than the composite grades in Domains 1, 3 and 5 and the estimated silver grades are higher than the composite grades in all domains except Domain 3. Typically the estimated grades should be close to the composite grades or slightly lower. An acceptable range would be within 0 to -10% of the composite grade. The grade control model has higher block grades for gold and silver in Domains 1 and 5. Again, the estimated grades should be close to the composite grades or slightly lower. SRK suggests that Koza review its estimation procedures and review the swath plots to see where the problem areas are.

Table 3.7.7.1: Çukuralan Comparison of Block Grades to Composite Grades in the Drillhole Model

Domain	Au			Ag		
	Block	Composite	% Diff	Block	Composite	% Diff
1	4.90	4.75	3%	1.69	1.48	14%
2	5.07	5.47	-7%	1.12	1.05	7%
3	5.10	4.90	4%	1.22	1.32	-8%
4	2.99	3.12	-4%	1.04	0.93	12%
5	4.52	4.24	7%	1.66	1.62	2%

Table 3.7.7.2: Çukuralan Comparison of Block Grades to Composite Grades in the Grade Control Model

Domain	Au			Ag		
	Block	Composite	% Diff	Block	Composite	% Diff
1	5.41	5.07	7%	2.97	2.72	9%
2	11.2	11.87	-6%	4.23	4.52	-6%
5	6.22	5.71	9%	3.15	2.96	6%

The estimations by ID2 and OK are very close to each other in the drillhole model. In the grade control model, the NN estimation is lower in Domains 1 and 2 than the ID2 estimation. SRK considers the differences to be within acceptable ranges.

Table 3.7.7.3: Çukuralan Comparison of Estimation Methods for Gold

Domain	Based on Drillhole Composite			Based on Face Samp Composite		
	NN	ID2	OK	NN	ID2	OK
1	5.06	4.86	4.90	4.93	5.41	-
2	5.35	4.98	5.07	9.98	11.20	-
3	5.47	4.94	5.10	-	-	-
4	2.89	2.93	2.99	-	-	-
5	4.41	4.38	4.52	7.25	6.22	-

3.7.8 Reconciliation

Koza also validated the model through reconciliation of mined tonnes to tonnes and grade predicted by the model. Table 3.7.8.1 shows the reconciliation numbers. The tonnes mined from the open pit are about 20% higher than the model predicted at about 95% of the modeled grade, producing about 14% more gold ounces. Koza suggests that this discrepancy may be caused by additional low grade material encountered in the pit.

The reconciliation between the resource model and underground production is more difficult to assess because the block model contains blocks only inside the domain wireframes as shown in

Figure 3.7.8.1. If the resource tonnage is calculated from the as-builts and the as-builts deviate from the domain blocks, then resource tonnage will be lower than expected because there will be no blocks in the as-builts.

Table 3.7.8.1: Reconciliation Çukuralan Model to Production

Çukuralan	Tonnes	Au (g/t)	Ounces
Resource Model – Open Pit	513,094	7.10	117,141
Production – Open Pit	616,076	6.75	133,677
Resource Model – Underground	83,368	9.18	24,600
Production – Underground	196,191	9.07	57,192
Resource Model - Total	596,462	7.39	141,741
Production - Total	812,267	7.31	190,869

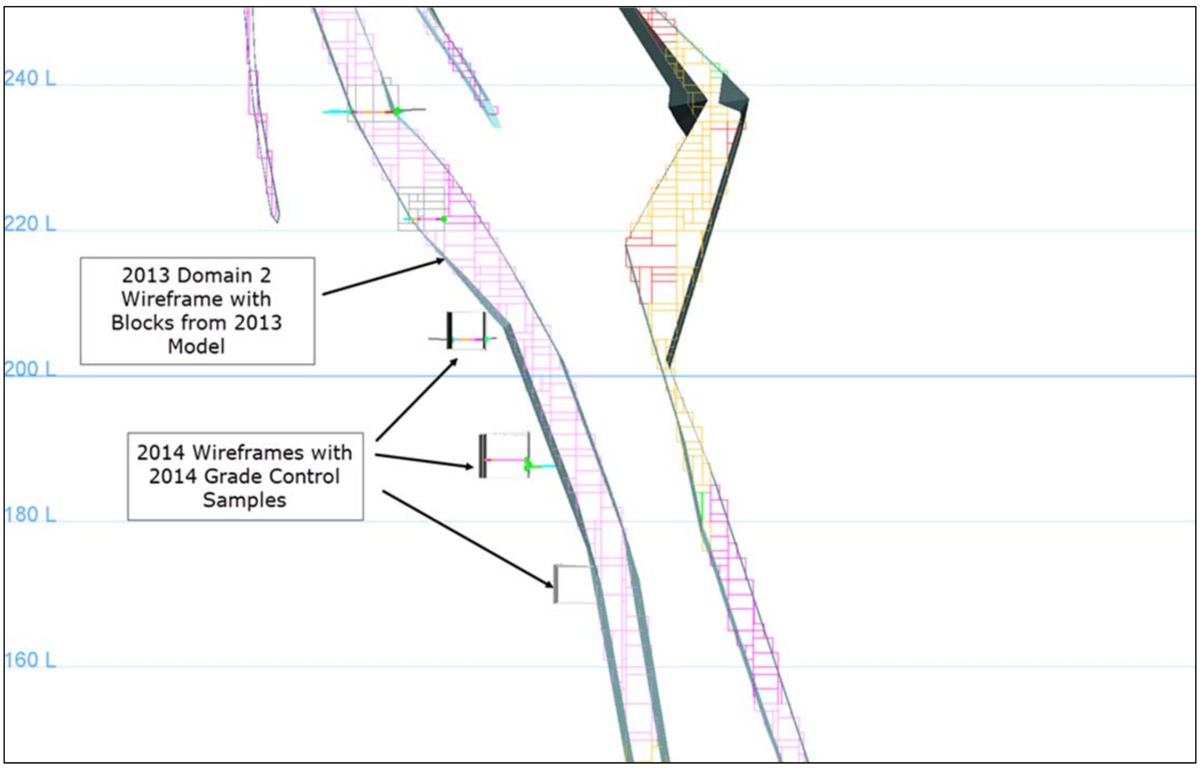


Figure 3.7.8.1: Cross-section Showing Domain 2 Wireframe and 2013 Blocks and 2014 As-builts

3.7.9 Resource Classification

The resources were classified as Measured, Indicated or Inferred according to the estimation pass and the number of samples used in the estimation as shown in Table 3.7.9.1. After the classes were assigned to the blocks, Koza generated a few wireframes for reclassification to reduce the “spotted dog” effect.

Table 3.7.9.1: Çukuralan Resource Classification Methodology

Category	Multiples of Search Volume	Min. Number of Samples	Max. Number of Samples
Inferred	5	2	20
Inferred	1.5	5	20
Indicated	1	5	20
Indicated	0.66	10	15
Measured	0.66	15	30

3.7.10 Mineral Resource Statement

The Çukuralan reserves include both underground and open pit reserves. The open pit resources are limited by the reserve pit shell because Koza does not plan to expand the pit at this time. Open pit resources are inside the pit shell and are stated at a cutoff grade of 0.80 g/t Au which excludes mining costs. Underground resources are outside the shell and are stated at a cutoff grade of 1.85 g/t Au. 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 3.7.10.1: Çukuralan Cutoff Grade Parameters

Prices and Costs	Units	UG	OP
Gold Price	US$/oz	1,450	1,450
Gold Recovery	%	0.95	0.95
Gold Refining	US$/oz	3.44	3.44
Royalty	%	0	0
Government Right	%	1	1
Process Cost	US$/t	11.00	11.00
Mining Cost	US$/t	47.00	0.00
G&A Cost	US$/t	15.00	15.00
Transport Cost	US$/t	8.00	8.00
Calculated Cutoff grade	g/t	1.85	0.78
Final Cutoff grade	g/t	1.85	0.80

The resources are listed in Table 3.7.10.2, and have been depleted for production through December 31, 2014. The historic Roman underground workings have not been depleted from the model as it is not possible to map the excavations, however, it appears that most of the workings have been mined out and the tonnage would not be significant.

Table 3.7.10.2: Çukuralan Mineral Resources, including Ore Reserves, at December 31, 2014

Classification	kt	Au (g/t)	Ag (g/t)	Au(oz)	Ag(oz)
Open Pit (1)					
Measured	1,988	4.37	1.4	279	88
Indicated	739	4.38	1.5	104	35
Measured and Indicated	2,728	4.37	1.4	384	122
Inferred	32	2.87	1.3	3	1
Underground (2)					
Measured	5,488	5.82	2.1	1,027	374
Indicated	4,382	5.04	1.4	710	200
Measured and Indicated	9,870	5.47	1.8	1,737	574
Inferred	3,123	5.33	2.1	535	212
Total					
Measured	7,476	5.43	1.92	1,306	462
Indicated	5,121	4.94	1.42	814	235
Measured and Indicated	12,597	5.24	1.72	2,121	696
Inferred	3,155	5.30	2.10	538	213

1 Open pit cut-off grade is 0.8 g/t Au
2 Underground cut-off grade is 1.85 g/t Au

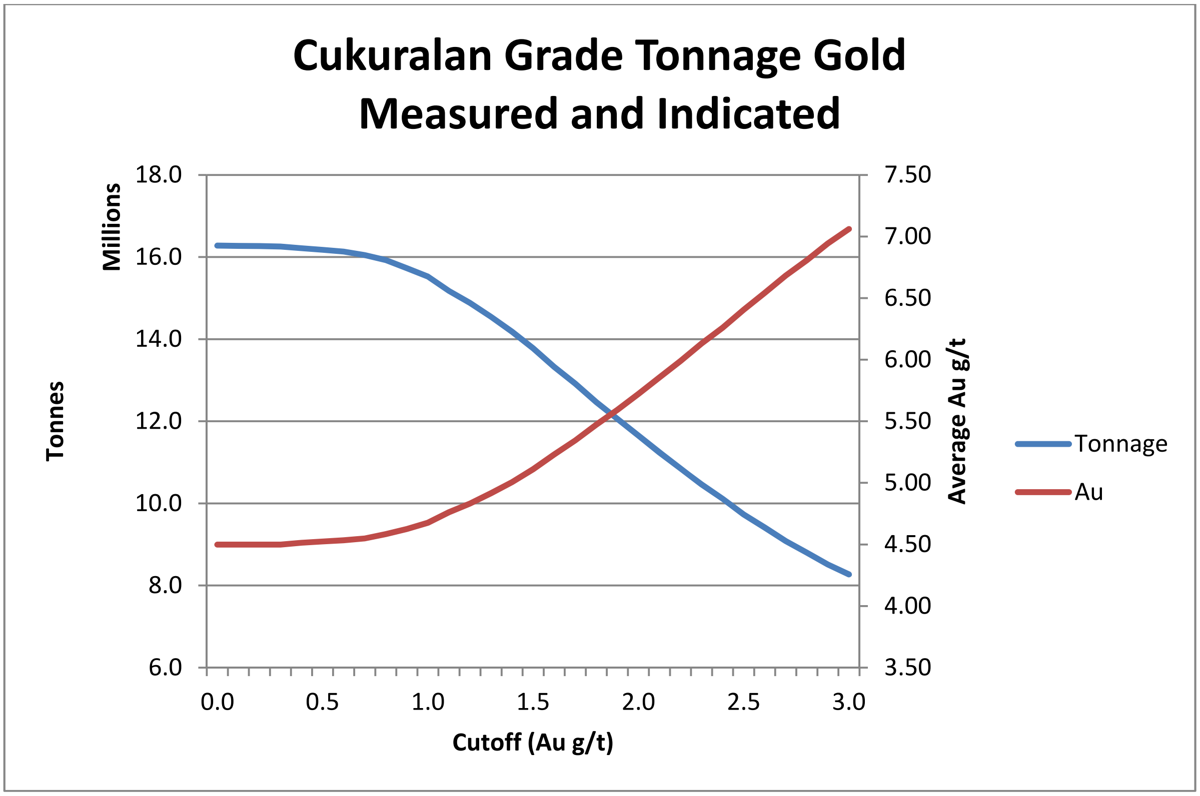
3.7.11 Mineral Resource Sensitivity

Figure 3.7.11.1 presents grade tonnage curves for Measured and Indicated Resources and also for Inferred Resources.

Cutoff grades for the Çukuralan resource at various gold prices are shown in Table 3.7.11.1.

Table 3.7.11.1: Çukuralan Cutoff Grades vs. Gold Price

Gold Price	Open Pit Au Cutoff Grade	Underground Au Cutoff Grade
1600	0.70	1.68
1550	0.73	1.73
1500	0.75	1.79
1450	0.78	1.85
1400	0.81	1.92
1350	0.84	1.99
1300	0.87	2.07
1250	0.90	2.15
1200	0.94	2.24



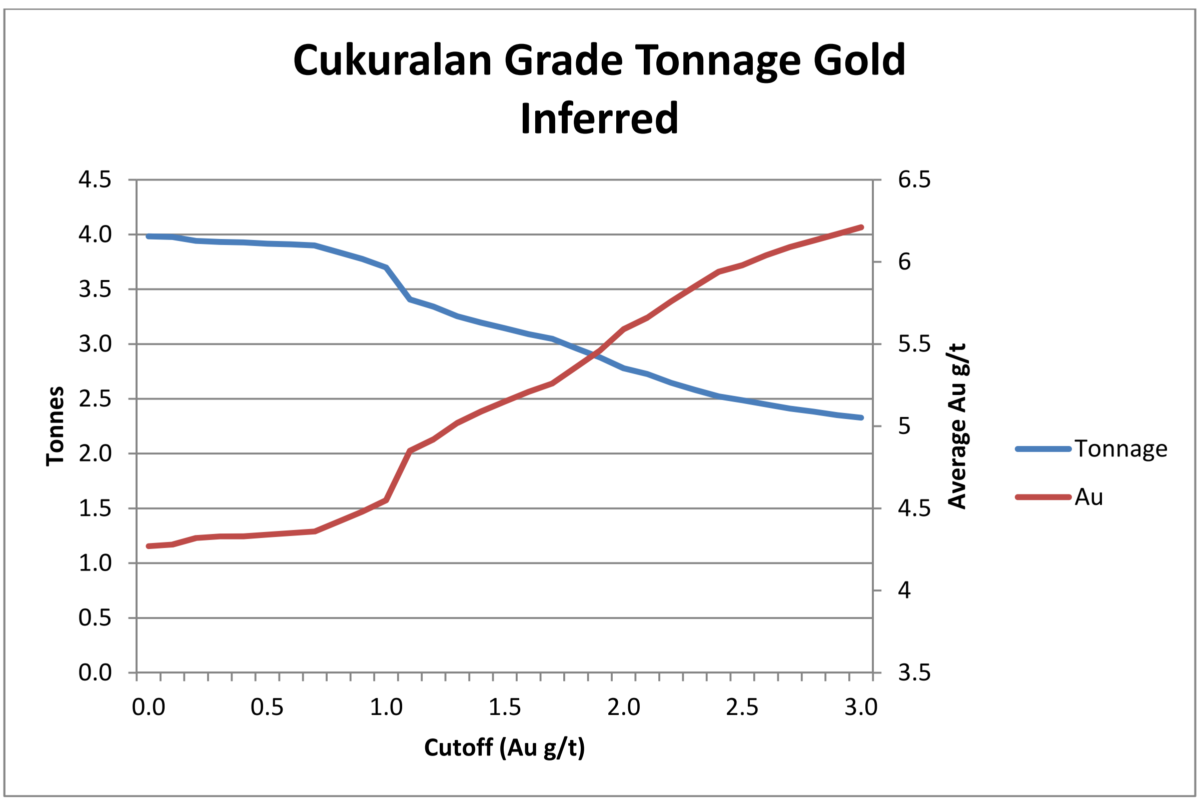


Figure 3.7.11.1: Grade Tonnage Curves for Çukuralan Resource

3.8 Çukuralan Mine Production

Both streams of RoM grade from open pit and underground operations are shipped directly to the Ovacık processing facility, whereas low grade (LG) produced as part of underground mining is stockpiled onsite and used on demand. Table 3.8.1 details the 2014 open pit mine RoM production compared to that estimated in the 2013 Technical Economic Model. Tables 3.8.2 and 3.8.3 detail the 2014 underground mine production for the Çukuralan Mine and is compared to the 2013 economic model for RoM ore. Negative percentages represent an underestimation of grade, tonnage and ounces by the economic model compared to actual production.

Overall the reconciliation on a monthly basis varies considerable when compared to that estimated from the mine production schedule. Fortunately, the combination of underground and open pit ore streams complement one another so ore exposure is not critical for the open pit. Interestingly, the overall reconciliation was essentially quite correct, but given the wild swings on a monthly basis this is not a true reflection of how well the production model compares to mine production.

Table 3.8.1: 2014 Çukuralan Open Pit Mine Production versus 2013 Technical Economic Model Production Schedule

2014 Production	Çukuralan OP ROM				Çukuralan 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	63,094	7.75	4.80	15,730	72,781	9.40	1.49	21,996	15%	21%	40%
February	78,676	7.28	4.47	18,410	90,116	7.21	2.10	20,889	15%	-1%	13%
March	109,590	8.15	4.78	28,703	95,507	6.15	2.28	18,884	-13%	-25%	-34%
April	118,126	7.51	4.21	28,504	88,209	6.89	2.77	19,540	-25%	-8%	-31%
May	96,141	6.44	3.86	19,914	70,565	9.41	3.75	21,349	-27%	46%	7%
June	30,074	6.92	3.94	6,694	77,138	5.36	2.06	13,293	156%	-23%	99%
July	17,666	2.64	1.99	1,501	52,431	5.09	2.47	8,580	197%	93%	472%
August	24,937	3.16	2.33	2,532	10,960	5.29	1.67	1,864	-56%	68%	-26%
September	21,380	4.76	2.73	3,274	13,482	5.20	1.58	2,254	-37%	9%	-31%
October	15,848	5.22	2.62	2,662	14,335	4.24	1.45	1,954	-10%	-19%	-27%
November	24,158	4.31	2.15	3,349	16,415	3.62	1.33	1,910	-32%	-16%	-43%
December	16,386	4.56	1.90	2,405	15,595	3.36	1.23	1,685	-5%	-26%	-30%
Total	**616,076**	**6.75**	**3.96**	**133,677**	**617,534**	**6.76**	**2.29**	**134,198**	**0%**	**0%**	**0%**

Table 3.8.2: 2014 Çukuralan Underground High Grade Mine Production versus 2013 Technical Economic Model Production Schedule

2014 Production	Çukuralan UG RoM Production				Çukuralan 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	12,441	9.36	4.01	3,744	17,753	7.35	2.36	4,195	43%	-21%	12%
February	12,872	8.83	3.77	3,654	14,195	6.78	2.01	3,094	10%	-23%	-15%
March	19,056	12.84	4.65	7,867	14,133	5.91	1.76	2,685	-26%	-54%	-66%
April	15,194	10.93	3.96	5,339	15,264	7.67	1.99	3,764	0%	-30%	-30%
May	14,346	9.33	3.59	4,303	15,763	7.30	2.00	3,700	10%	-22%	-14%
June	14,839	11.00	3.79	5,248	16,423	8.48	2.36	4,478	11%	-23%	-15%
July	12,599	7.61	3.12	3,083	17,340	9.23	2.38	5,146	38%	21%	67%
August	15,663	7.11	3.42	3,581	16,740	6.98	1.68	3,757	7%	-2%	5%
September	19,328	9.41	3.87	5,848	16,329	6.55	1.90	3,439	-16%	-30%	-41%
October	18,107	8.78	3.93	5,111	15,349	5.68	1.31	2,803	-15%	-35%	-45%
November	20,457	6.52	3.15	4,288	15,412	5.98	1.27	2,963	-25%	-8%	-31%
December	21,287	7.49	3.72	5,126	18,852	5.54	1.36	3,358	-11%	-26%	-34%
Total	**196,191**	**9.07**	**3.76**	**57,192**	**193,553**	**6.97**	**1.87**	**43,381**	**-1%**	**-23%**	**-24%**

The major surprise for underground operations during 2013 that continued into 2014, was the significantly higher gold grade that was achieved compared to that estimated in the 2013 Technical Economic Model production schedule. While tonnages reconciled quite well for RoM ore, the gold grade was higher by 23% and gold ounces 24% better than estimated. The monthly variation for tonnage, grade and ounces is generally within a plus minus 20% range which is reasonable.

3.9 Ore Reserve Estimation

LoM plans and resulting reserves are determined based on a gold price of US$1,250/oz for the underground and open pit mines and projects. Reserves stated in this report are as of December 31, 2014.

The ore at Çukuralan is extracted using open pit and underground mining methods and is being processed at the Ovacık mill. The ore material is converted from resource to reserve based primarily on positive cash flow pit optimization results, pit and underground mine design and geological classification of Measured and Indicated resources. The in-situ value is derived from the estimated grade and various modifying factors. The previous section discusses the procedures used to estimate gold grade. The modifying factors include the metal value and recovery.

3.9.1 Modifying Factors

The conversion of resource to reserve entails the evaluation of modifying factors that should be considered stating a reserve. Table 3.9.1.1 illustrates a reserve checklist and associated commentary on the risk factors involved for the Çukuralan reserve statement.

Table 3.9.1.1: Çukuralan 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/Underground
Density and Bulk Handling	X			Operating Mine
Dilution	X			No dilution added to open pit; Planned and unplanned for underground
Mine Recovery	X			Full mine recovery assumed
Waste Rock	X			Waste dump strategy in place and sufficient volume
Grade Control	X			Channel sample
Processing				
Representative Sample	X			Operating Mine
Deleterious Elements	X			No
Process Selection	X			CIL
Geotechnical/Hydrological				
Slope Stability (Open Pit)	X			Slope stability study complete, bench scale stability should be looked at, full shotcreting UG
Area Hydrology	X			Underground requires pumping. Risk of open pit underground conductivity of water.
Seismic Risk		X		Assume no limiting factor to mining
Environmental				
Baseline Studies	X			Operating Mine
Tailing Management	X			No tailings at site
Waste Rock Management	X			Stability OK; MBA management ongoing
Acid Rock Drainage Issues	X			Water management in place
Closure and Reclamation Plan	X			Project still developing EIA
Permitting Schedule	X			Forestry permits are obstacle to advancement
Legal Elements or Factors				
Security of Tenure	X			
Ownership Rights and Interests	X			
Environmental Liability	X			
Political Risk (e.g., land claims, sovereign risk)	X			Local government forced shutdown of operations for two weeks in early 2013. Court injunction to recommence operations and rectify permits. Still problem with forestry permits to continue exploration
Negotiated Fiscal Regime	X			
General Costs and Revenue Elements or Factors				
General and Administrative Costs	X			
Commodity Price Forecasts	X			
Royalty Commitments	X			
Taxes	X			
Corporative Investment Criteria	X			

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
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 Influence		X		Koza Environmental/Social – Operating Mine

Open Pit

Table 3.9.1.2 illustrates the cost inputs used as the basis for pit optimization at Çukuralan.

Table 3.9.1.2: Çukuralan Pit Optimization Inputs (as of December 31, 2014)

Parameter	Unit	Amount
Mining Cost	US$/t material	1.93
Rehabilitation Cost	US$/t waste	0.20
Milling Cost	US$t/ore	10.85
Selling Cost	US$/oz	3.44
Grade Control	US$t/ore	0.8
Administration	US$t/ore	18.10
Ore Rehandle	US$t/ore	0.97
Transport	US$t/ore	8.00
Gold Price	US$/oz	1,250
Silver Price	US$/oz	20
Gold Recovery	%	95
Silver Recovery	%	75
Cutoff grade	g/t Au	1.03

Source: Koza, 2014

Çukuralan ore is transported via highway to the Ovacık processing facility. Koza is working in conjunction with a local community co-operative for the transportation of ore. As a result, the cost of transporting ore has been contracted to US$8.00/ore-t with the additional social benefit from improved landowner relationships along the route from the mine to the processing plant. Koza has also invested in upgrading the main haul route from the main highway to the site given the large increase in traffic caused by the ore shipments and personnel movements.

Underground

The costs used for calculating the underground mining reserves are based primarily on historical information available from the Ovacık underground mine, and current costs seen at Çukuralan since underground mining commencement in 2011.

Table 3.9.1.3 summarizes the cutoff grade calculation for the Çukuralan underground orebody.

Table 3.9.1.3: 2014 Çukuralan Underground Cutoff grade Calculation

Parameter	Unit	Cut and Fill
Mining cost	US$/t mined	38.00
Processing cost	US$/t ore	10.85
Admin cost	US$/t ore	18.1
Total cost	**US$/t**	**66.95**
Gold price	US$/oz	1,250
Mill recovery	%	95
Government Right	%	1
Refining	US$/oz	3.44
Transport	US$/t ore	8
Cutoff grade	**g/t Au**	**1.98**

Source: Koza, 2014

3.10 Ore Reserves Statement

3.10.1 Open Pit

The terrain surrounding Çukuralan, combined with water diversion and waste disposal constraints, has dictated a reduced open pit footprint and careful phase design. Where mine sequencing is affected by environmental constraints, resulting in reduced open pit reserves, underground operations will be applied instead.

Ore tonnes which lie within the final pit design shape are able to be 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. The Çukuralan open pit reserves are shown in Table 3.10.1.1.

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 (LG), the removal of administration and grade control costs lower the break-even-cutoff grade making processing profitable at the end of mine life. The RoM and LG reserves are listed in Table 3.10.1.2. No emergency stockpile exists for Çukuralan as it is associated with the Ovacık processing facility.

Table 3.10.1.1: Çukuralan Open Pit Mineral Reserves at December 31, 2014

Category	kt	g/t Au	g/t Ag	koz Au	koz Ag
Proven Reserve	1,920	4.46	1.4	275	85
Probable Reserve	693	4.59	1.5	102	33
Total Proven and Probable Reserves	2,614	4.49	1.4	378	118

Source: Koza, 2014
Metal Price: US$1,250/oz-Au, US$20/oz-Ag, Au Recovery 95%, Ag Recovery 75%, Au cutoff grade 1.03g/t.

Table 3.10.1.2: Çukuralan RoM and LG Stockpile Reserve, at December 31, 2014

Category	kt	g/t Au	g/t Ag	koz Au	koz Ag
Proven Reserve	206	5.11	2.8	34	19
Probable Reserve	573	0.86	1.2	16	22
Total Proven and Probable Reserves	779	1.98	1.6	50	41

Source: Koza, 2014
Reserves based on stockpile balance on December 31, 2014 survey. Au Recovery 95%, Ag Recovery 75%.

3.10.2 Underground

Proven and Probable reserve categories are determined directly from the Measured and Indicated categories. SRK is of the opinion that the reserve classification used by Koza is valid for the Çukuralan underground mine.

Table 3.10.2.1 presents the mineral reserve for the Çukuralan underground mine as of December 31, 2014.

Table 3.10.2.1: Çukuralan Underground Mineral Reserve, at December 31, 2014

Category	kt	g/t Au	g/t Ag	koz Au	koz Ag
Proven Reserve	4,554	4.74	1.7	694	246
Probable Reserve	3,240	4.23	1.1	441	115
Total Proven and Probable Reserve	7,793	4.54	1.4	1,135	361

Source: Koza, 2014
Reserves based on December 31, 2014, Metal Price US$1,250/oz-Au, Au Recovery 95%, Au cutoff grade 1.98 g/t.

3.10.3 Commentary on Open Pit Mine Operations

Open Pit

The Çukuralan open pit is located approximately 40 km from the Ovacık processing facility in mountainous terrain. Mining is based on truck and excavator operations comprising blast, load and haul operations. Excavators are used for the removal of blasted overburden to load highway trucks for waste disposal. Grade control engineers determine what material is ore from each mining flitch within a 5 m bench based on samples from trenches excavated every 10 m along strike. Based on grade, ore is either classified as low-grade, RoM or high-grade. High grade and RoM ores are transported to the Ovacık facility based on demand. Surplus low-grade material is stockpiled for processing at a later date.

The stripping ratio is quite high at 28:1 waste versus ore, but is not unreasonable given the high gold grade.

As is common with all the open pit operations, small excavators and loaders are used as the primary mine fleet. In Turkey, the combination of fuel price and wage competitiveness does not dictate the use of dedicated mining equipment as is the case in many other countries. Using highway trucks (essentially a civil operation) lowers fuel burn rates at the expense of equipment life and is not burdened by high operator labor costs. Koza and the contractors do a very good job of maintaining haul roads but do have to stop operations during inclement weather.

Source: SRK, 2013

Figure 3.10.3.1: Çukuralan Bench Face and Grade Control

<u>Waste Rock Dump</u>

As part of the permitting approval process, there are strict controls over the management of waste and water flow around the Çukuralan waste rock dumps. The main waste dump is nearing completion and rehabilitation of the valley facing faces has begun with the new application of topsoil and re-grading. There is only a small amount of space available at the very top of the dump but the main location for waste disposal has now moved to in pit backfilling of the Çukuralan Phase 1 pit.

SRK confirmed that the three major dumps contain enough capacity to dispose of 26.25 Mm^3 of material. Currently waste tonnage is projected to be 72.3 Mt of material so with a swell factor of 20%, 24 Mm^3 of space will be required. SRK is of the opinion that sufficient capacity exists on site to dispose of mine waste given no waste dump sequencing problems.

Figure 3.10.3.2 shows the clean and dirty water control structures below the main waste dump plus ongoing rehabilitation efforts

Source: SRK, 2014

Figure 3.10.3.2: Waste Dump and Water Control Measures

An integral part of the pit sequencing will be the diversion of a small stream which bisects the Çukuralan pit. A combination of cutoff and diversion channels are being built and integrated into the phase sequencing of the pit. There will still be a requirement for clean and dirty water separation downstream of the disturbed areas. Dirty water is pumped to a treatment facility before discharge.

Figure 3.10.3.3 shows the construction of the main diversion channel running between the pit and back filled pit areas.

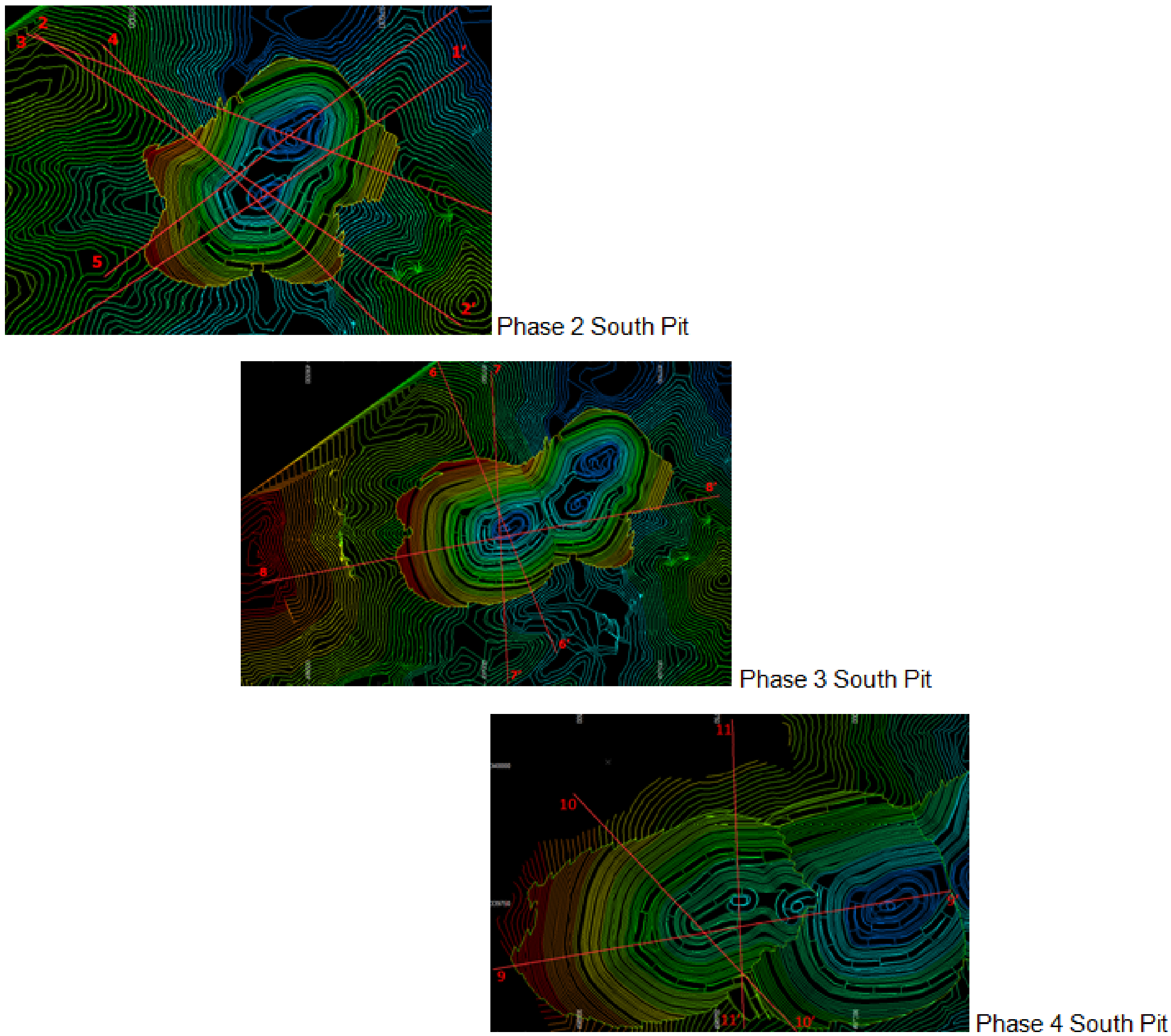
Source: SRK, 2014

Figure 3.10.3.3: Çukuralan Water Diversion Structure

Geotechnical

Open pit geotechnical analysis is carried out by a dedicated geotechnical engineer employed by Koza (Koza, 2012a). The pit has been designed using a triple bench configuration with overall slope angles varying by sector from 35° to 51° in different pit sectors and phases. Geotechnical factors of safety are calculated using Slide 6.0 and DIPS 5.0 and are generally above 1.4. While the usual production supervision will be required for all high walls, the current geotechnical analysis indicates the pit walls will are quite strong and the possibility for more aggressive slope angles exists.

The North pit was completed in 2014 and is currently being backfilled so there is no need to show the geotechnical slope angles. The south pit phase designs and geotechnical analysis of those phases has not changed since 2012. Figure 3.10.3.4 illustrates the sections analyzed.



Source: Koza, 2012

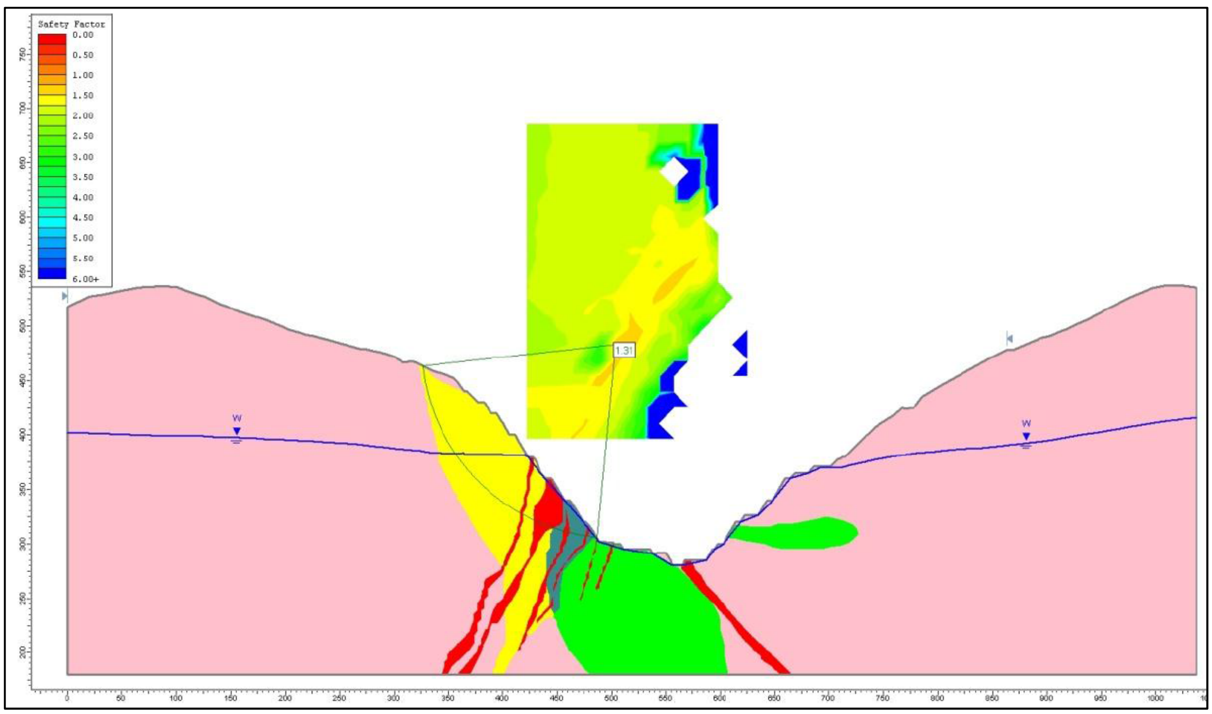
Figure 3.10.3.4: Analysis for South Pit Phase 2, 3, 4

Table 3.10.3.1 details the factors of safety for the Southern Pit extensions. As with the North Pit the factors of safety are generally high, except Section 2 of Phase 2 (South Pit) that suffers from a “nose” feature in the design. Koza plans on backfilling this wall to improve stability after extraction and that will improve the factor of safety to 1.62. Figure 3.10.3.5 illustrates the high risk wall defined by Section 2 for the Phase 2 south pit.

Table 3.10.3.1: Analysis for South Pit Phase 2, 3, 4

Phase	Slope Height	Overall Slope Angle	Factor Of Safety
Phase 2 Sections			
Section 1-1' (West)	180	35	2.16
Section 2-2' (North)	149	48	1.38
Section 2-2' (North after backfill)	110	48	1.62
Section 3-3' (South East)	210	35	2.02
Section 4-4' (South)	145	36	2.50
Section 5-5' (East)	95	51	1.72
Phase 3 Sections			
Section 6-6' (North)	225	47	1.45
Section 7-7' (South)	149	39	2.39
Section 8-8' (North)	211	38	1.71
Phase 4 Sections			
Section 9-9' (West)	294	42	1.68
Section 10-10' (North)	193	44	1.82
Section 11-11' (North)	145	44	2.05

Source: Koza, 2012c



Source: Koza, 2012c

Figure 3.10.3.5: Limit Equilibrium Results for Section 2

During SRK's site visit in 2014, the Phase 2 south pit walls were showing considerable daylighting where the orientated structures on the pit wall were close in angle to the bench face angle. This leads to benches being lost as the resultant wedges in the pit face fall out or daylight.

Source: Koza, 2012c

Figure 3.10.3.6: Bench Scale Stability of Phase 2 Pit Walls

Even though the Phase 2 south wall is an internal wall and not a final design wall, the major concern with the inability to keep catch benches in place is that a rock has no way of slowing down if it begins to move. Because the wall is internal, Koza should consider an artificial catch bench that is wide enough so the bench won't fail, but also wide enough to retard any rock falls and protect workers below. These benches should be placed every 50 to 100 m from the pit crest as the current phase design is over 185m deep.

3.10.4 Comment on Underground Mine Operations

The Çukuralan Mine is an underground and open pit complex where both mining methods are being applied simultaneously. Çukuralan underground has a similar layout to the Ovacık underground and utilizes cut and fill mining methods with the inclusion of some open stopes.

The vein system at Çukuralan is made up of five domains categorized by vein dip. Each domain contains many individual veinlets and grade/vein continuity does not appear to be as consistent as that seen at Ovacık. Grade control is important in this operation as veins change grade and shape with each round. The rock is generally harder than Ovacık requiring an estimated 30% more drilling and explosives, 15% less ground support, and 50% less shotcrete than Ovacık. Figure 3.10.4.1 shows a long section view looking south of the final pits, the underground drifts and stoping and the future mine planning.

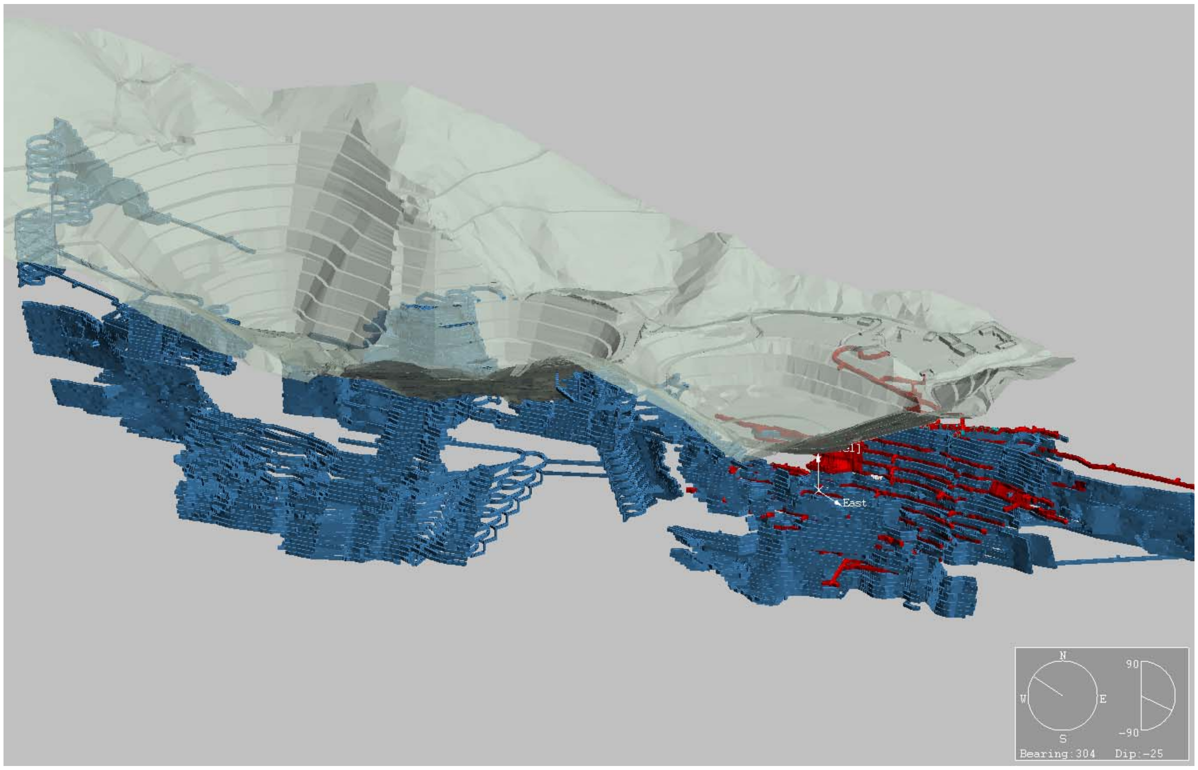


Figure 3.10.4.1: Çukuralan Perspective View Long Sections

Figure 3.10.4.2 illustrates a typical cross section of the reserve mine plan and multiple ore bodies intersected by the development plan. Of particular note is the proximity of underground workings to the open pit.

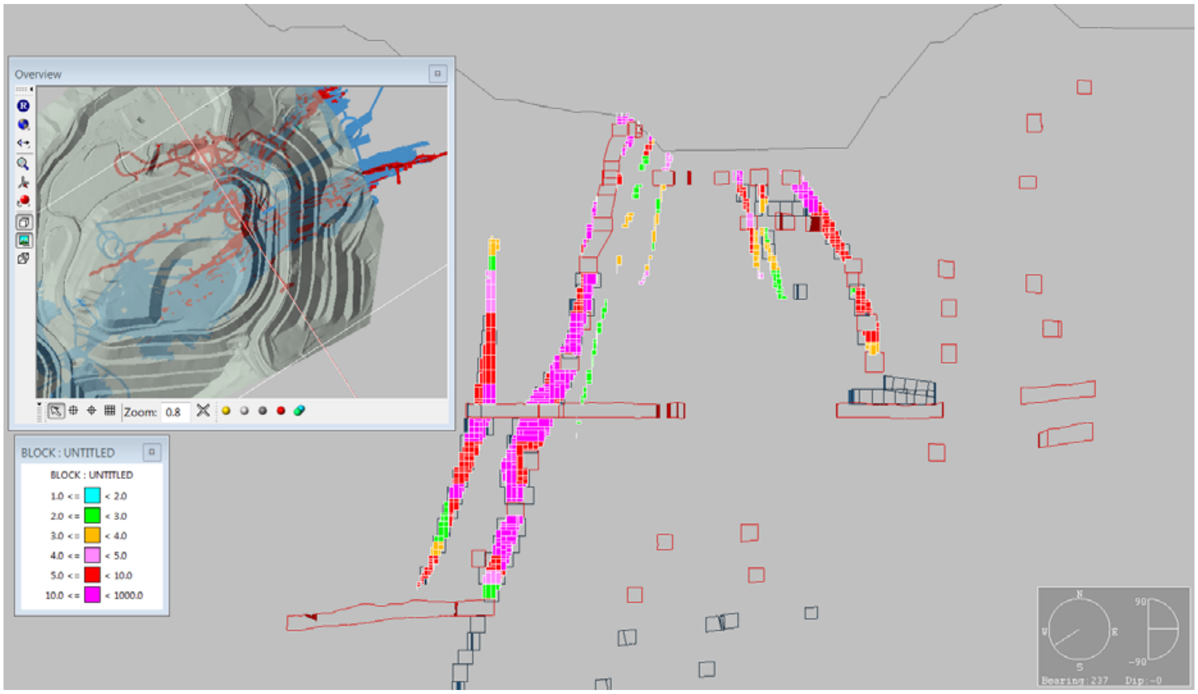
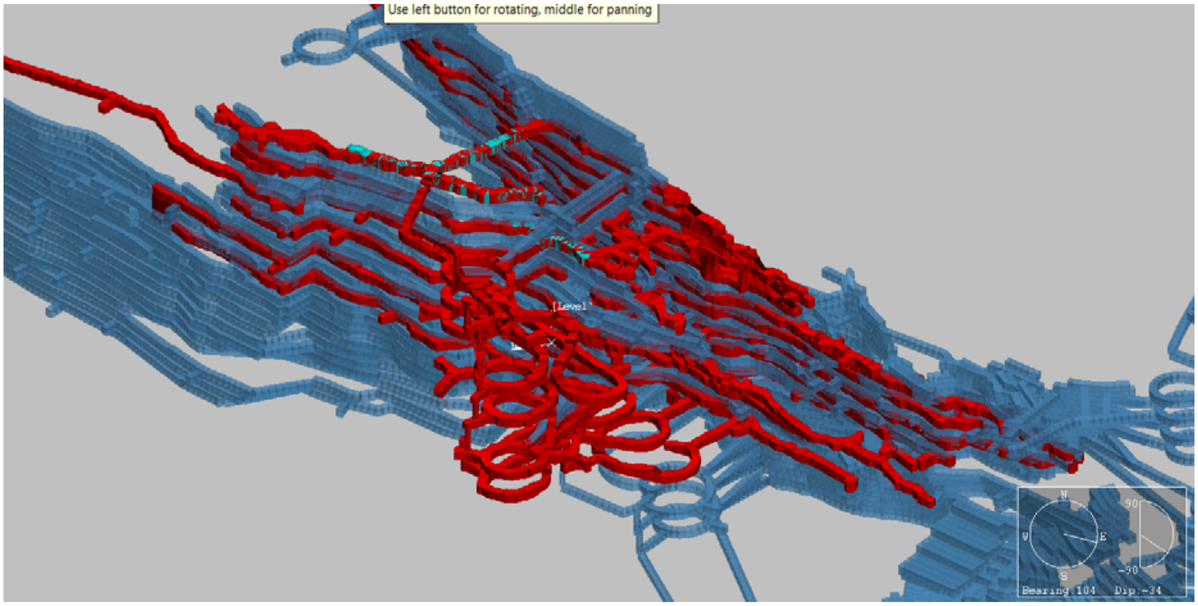


Figure 3.10.4.2: Çukuralan Cross Sections

Underground Mining Method

Access to the underground orebody is via a portal located on the north side of the pit. The ramp spirals down in the hanging wall of the orebody about 134 m, whereupon the ramp crosses over into the footwall. All access development is driven at a nominal 5 m x 5 m. Figure 3.10.4.3 shows the 2014 as built triangulations (red) versus LoM plan development (blue) and figure 3.10.4.4 shows a picture of the main portal.



Source: SRK, 2014

Figure 3.10.4.3: Çukuralan Mine Workings and Life of Mine Design

Source: SRK, 2013

Figure 3.10.4.4: Çukuralan Portal

A cut and fill method is used. Primary cuts are driven 5 m high at a spacing of 10 m or 15 m back to floor, allowing either two or three cuts to be mined between the primaries respectively. Primary stopes are mined and filled with rockfill containing an 8% cement binder. This high cement content is required as these stopes will be undercut by the third lift at a later date. The second lift is mined over the first and filled with development waste, unless a parallel drift is being mined on the same horizon, in which case backfill with 6% cement binder is installed. This percentage of cement is required so that equipment mining the third cut has a stable working platform and to provide increased strength for wider stope areas. The final lift is mined in an undercut fashion with the backfill from the primary cut forming the stope back. This final cut is generally left open.

Historically mined areas have and will continue to be encountered during underground mining. Generally, when drilling around a historically mined area, the material is softer and has a more oxidized color. Care must be taken in these areas to ensure safety and Koza has successfully detected and mined through such areas at Çukuralan.

All development is drilled using twin boom jumbos. Faces are mucked out using LHD's and primary support in the form of 10 cm of polypropylene fiber reinforced shotcrete is applied. Split set tendon support is installed in the back and upper walls using a single boom jumbo. In areas of poor ground the shotcrete and split sets are supplemented with wire screen installed over the shotcrete and held in place by the split sets.

Ore and waste are transported from the face using dump trucks to either the ore stockpiles or the waste dumps on surface. The trucks back-haul cemented backfill to the cut and fill stopes underground.

Backfill is produced by mixing waste material sourced from site with a cement and water mixture. The crushed waste is loaded onto a conveyor using a front-end loader and conveyed up to a batch mixing plant on a raised platform. Cement and water are added to the mixer and the material is mixed for one minute before being dumped out of the bottom of the mixer directly into the truck below. When full, a process that takes about fifteen minutes, the trucks back-haul the material down the ramp to the stope being filled. The trucks dump the material in the drift and the material is pushed up to the stope back using a pusher-plate assembly mounted on the boom of an LHD specially modified for the task.

Shotcrete material is produced on site using the same conveyor and mixing arrangement as the backfill. In this case, fine gravel and sand are loaded onto the conveyor in the correct proportions and conveyed to the mixer, where cement, water and plasticizers are added. The mixed material is dumped directly from the batch mixer into a shotcrete transmixer (underground truck with a rotating mixer drum attached). The transmixer drives to the face, reverses up to the Spraymech (remote controlled, robot arm, shotcrete application unit) and dumps shotcrete material directly into the pump unit. The Spraymec operator controls the boom and applies shotcrete to the back and walls of the excavation from a safe location away from the unsupported ground.

SRK found the underground mining operation to be efficient, clean and well organized. Development, mining, ground support and general housekeeping standards were of a high quality and consistent with a world-class mining operation.

Mining Equipment

Table 3.10.4.1 presents the Çukuralan underground mining equipment fleet.

Table 3.10.4.1: Çukuralan Underground Mining Equipment

Task	Equipment	Quantity	Supplier
Truck haulage	MT 416	1	Atlas Copco Wagner
	MT 2010	4	Atlas Copco Wagner
Mucking			
	SLF 65	4	Schopf
Shotcrete	Spraymec	1	Normet shotcrete sprayer
	UG Mixer Truck	1	Normet shotcrete transmixer truck
Drilling	Jumbo	3	Atlas Copco 2 twin boom and 1 single boom
	Simba	1	Atlas Copco long hole drilling rig
	Diamec U-6	1	Atlas Copco Core drilling rig
Service	ITC	2	JCB TM 310
	Loader	1	JCB 456 ezx
	Pick-up	3	Ford Ranger
	Tractor	1	New Holland T6020
		1	New Holland T4030
		1	Fiat 5560
	Personnel Carrier (portal to UG/UG to portal)	1	Titan
	Back Hoe	2	JCB 4CX
Total		**28**	

Source: Koza, 2014

A second underground portal has been planned for Çukuralan. Table 3.10.4.2 shows the planned equipment list for this secondary portal.

Table 3.10.4.2: Çukuralan Underground Mining Equipment – Second Portal

Task	Equipment	Quantity	Supplier
Truck haulage	MT 2010	2	Atlas Copco Wagner
Mucking	SLF 65	2	Schopf
Shotcrete	Spraymec	1	Normet shotcrete sprayer
	UG Mixer Truck	1	Normet shotcrete transmixer truck
Drilling	Jumbo	1	Atlas Copco 2 twin boom and 1 single boom
Total		7	

Source: Koza, 2014

Geotechnical Designs

The ground conditions appear to be well controlled in the underground mine. Systematic use of shotcrete provides excellent areal coverage and pattern bolting ties the surface support into the unfractured rock mass away from the excavations. There was no sign of significant shotcrete failure in any of the excavations visited on the underground tour. Visual observation of the shotcrete confirmed that it is being applied at a consistent 5 cm+ thickness. In discussion with mine engineers and the underground production staff, it is expected that similar ground conditions will be encountered as the mine is further developed.

There is a geotechnical engineer based at the site who provides ground control and design support to the mine department and there is a process of regular external review by consultants that provides overview and strategic comment to ensure that good long-term decisions are being made.

Mine Planning

Mine planning is carried out by the Ankara-based engineering department using the same methodology as Ovacık. There is good communication between the mine operations department and the engineers. Regular operational meetings and discussions are held to ensure that both short and long-term issues are addressed and that the plan is understood by all parties.

The engineering department uses Datamine for mine design and Mine2-4D for development and stope scheduling. SRK is of the opinion that the scheduling process carried out in Mine2-4D is efficient and effective. This package links directly to Datamine and provides an integrated, graphical design and scheduling interface that removes much of the tedious and error prone spreadsheet work.

The mine design process is as follows:

- Obtain updated ore wireframes and block model information from geology;
- Identify areas of the orebody above cutoff grade;
- Each 5 m high primary cut and subsequent lift is designed according to the vein profile and block model grade at that elevation. The cuts are broken down into 4 m long segments for ore reserve evaluation purposes. Wireframes are created for the 4 m drift segments and these are evaluated against the block model for tonnage, grade and resource category;
- The results of the evaluation are exported to a spreadsheet and sorted by stope name;
- A 10% planned dilution is applied using a zero grade;
- In order to report the reserve, the Inferred category metal content is removed and the results are reported according to reserve category; and
- To produce the LoM plan the extraction sequence of the ore drives and stopes are scheduled based on an average historical development advance rate. The development is balanced between all the production areas while keeping in mind the overall mining sequence. Waste access and ramp extension development is added to the schedule to support opening up of new cut and fill and stope access drifts.

The schedule is modified to include changes in geological understanding and actual development performance on an as required basis. Short term plans are generated monthly based on the long-term plan. In this manner the operation keeps a direct relationship between short and long-term plans and ensures that the strategic direction is followed.

In general six mucking faces are available at any point in time to provide various ore sources for blending, operational ease, and backup in case of difficulties in any one area. The mine operates six days a week, three 8-hour shifts each day, with approximately seven effective hours per shift.

Mine Ventilation

The mine is well ventilated in the main access, services areas and production areas utilizing an exhaust system. Two 132 kW surface fans running in parallel and provide primary ventilation. A series of underground auxiliary fans distribute the air to the working areas and maintain the flow in the main ramps. The nominal flow capacity of the system is 130 m^3/s for a demand of 110 m^3/s. Ventilation surveys are conducted by the mine engineer on a weekly and monthly basis and the schematic ventilation plan is updated frequently. In addition, the ventilation is simulated using Ventsim software to identify any deficiencies in the ventilation plans for future mining areas and any proposed modification in ventilation system.

3.11 Metallurgy, Process Plant and Infrastructure

All material mined at Çukuralan Mine is processed in the Ovacık Mill and tailings will report to the Ovacık TSF.

3.12 Environmental

3.12.1 Permitting

Çukuralan is mined using open pit and underground mining methods. There is no ore processing on-site. Ore is trucked over public roads to the Ovacık Mill, 40 km southeast of Çukuralan, for processing. Waste rock from ore extraction is stored on-site.

The EIA permit for the Çukuralan mine was first obtained on September 2, 2009. Following the first EIA three revisions were performed on the mine design which resulted in three more EIA permits. The EIA permit for the first revision was obtained on November 3, 2010. The EIA permit for the second revision was obtained on March 11, 2011. The EIA permitting process for the third revision is on-going.

Due to environmental sensitivities in the Çukuralan Project area, additional environmental studies are critical for further stages of environmental permitting, and for a solid environmental management system. These studies include:

- Development of closure plans for waste rock dump and open pits; and
- A Mine Dewatering Study.

The studies have been completed for the initial stages of the operation. Revision of the studies and update of resulting plans are required for the full operation of the Project. The relevant geochemical and hydrogeological studies that will contribute to the closure plan are in progress. The results of both studies will be submitted to the relevant state authorities for approval and consultation. Both studies may present additional environmental constraints to Koza than those given in the EIA report.

3.12.2 Environmental Management and Mine Closure

The Project baseline environmental assessments indicate that the land use type is forest with proximity to a forest protection reserve area and stone pine (Pinus pinea) plantation areas. The permitting with respect to forestry has completed prior to operation. Another important issue is the proximity of the Madra Reservoir catchment area that provides potable water to the Balıkesir Province. Water and effluent quality management including Acid Rock Drainage (ARD) control are the main environmental issues of the Çukuralan Project. As part of the EIA studies addition of phosphate minerals to the dump material for prevention of ARD and heavy metal mobilization is an option. Most of the waste rock from the Southwest Pit will be used for the closure of the Northeast Pit. The construction of diversion channels to control run-off water has been in progress. A leakage collection pond has been established on the downstream part of the waste rock dump site, and the water quality monitoring program has begun. The monitoring program will continue until 2026.

A Mine Closure and Reclamation Plan does not exist for the Çukuralan Mine. However, Koza has made some preliminary mine closure estimates. The estimated total closure cost is US$10.6 million, of which US$8.9 million is for back filling of the open pits. However, US$6 million estimated for the backfilling of the Northeast Pit may be misallocated as closure costs, instead of operating costs. In-pit filling will be exercised for the North Pit as part of the mine operation. Therefore, the closure costs for the Çukuralan mine would likely be less than given here. However, costs such as employee compensation packages may increase the overall closure costs. It is recommended that a comprehensive Mine Closure and Reclamation Plan be prepared to clarify the closure costs.

3.13 Conclusions and Recommendations

3.13.1 Geology and Resources

Koza has used a number of CRMs at Çukuralan. Although there are a number of failures related to some of these CRMs, there are sufficient standards that show reliable accuracy and bracket the range of grades in the deposit to support the resource estimate. SRK also observed that there was low reproducibility in the preparation duplicates. SRK recommends that Koza investigate the reproducibility by first examining homogenization procedures and then possibly adjusting the volume of material submitted for pulverization. SRK also recommends that Koza add pulp duplicates to monitor analytical precision and check samples sent to a secondary laboratory to verify samples at the primary laboratory.

SRK recommends that the block model and grade control data be closely compared to find areas where there is not good agreement. Special attention should be paid to areas where the grade control samples were used in the estimation to determine if they are contributing to an over-estimation of grade that is suggested in the reconciliation.

The block model for Çukuralan is focused on two main structures but there are multiple areas of mineralization that occur between these two features that are only partially accounted for in the wireframes. SRK recommends that a leapfrog wireframe analysis be conducted to aid in the identification of these small areas of mineralization rather than relying on manual interpretation. Alternatively, an indicator model to identify these areas may be considered.

The open pit mine production was about 20% higher in tonnes and 5% lower in gold grade compared to the resource model resulting in 15% more ounces. Koza suggests that the increased tonnage is due to lower grade zones that have not been modeled in the resource.

3.13.2 Mining and Reserves

The Çukuralan project has shown Koza can operate a successful open pit and underground operation simultaneously while dealing with adverse waste haulage requirements, diversion of stream flows and limitations based on land access. The orebody remains open at depth and along strike for further underground operations in the future.

As the open pit transitions to the south pit and in-pit backfilling continues, further operating efficiencies and reduced contractor costs are likely.

SRK recommends further optimization of the pit walls which would allow improved open pit extraction. Enough information on water table, joint orientation and lithology’s from past operations should now be incorporated into geotechnical models with the recommendation that all slopes have a factor of safety above 1.3 but less than 2. It is also recommended that local stability analysis be performed with enhanced lithological modelling. This will test the local stability in relation to changes in lithology that may be missed in a global analysis for the high wall. Re-optimization of bench heights and bench face angles should be considered for the South Pit as it was evident that joint set orientation was unfavorable for keeping benches stable on the high wall as they were daylighting consistently.

The ramp systems of the south phase designs should be optimized to interlink where possible. The current design is based on individual pit phases working in isolation rather than harmoniously. As such, ramps have been designed to remain in the highwalls serving no useful purpose and will lead to unnecessary increases in stripping ratio and/or ore loss.

The block model for Çukuralan is focused on two main veins but there are multiple areas of mineralization that occur between these two features that are only accounted for during operations. SRK recommends that a leapfrog wireframe analysis be conducted to aid in the identification of these small areas of mineralization rather than relying on manual interpretation. Alternatively, an indicator model to identify these areas may be considered.

SRK found the underground mining operation to be efficient, clean and well organized. Development, mining, ground support and general housekeeping standards were of a high quality and consistent with a world-class mining operation.

The mine grades achieved from underground operations have been consistently good for 2013 and 2014 and there is no reason to believe that the underground will not produce good grade (5+ g/t Au) for years to come. That being said, SRK would recommend a groundwater analysis be conducted on the LoM plans to predict future mine pump requirements and capacities. As the underground deepens, the quantity and cost of disposing this water will likely increase and should be planned for.

3.13.3 Metallurgy and Process

All material mined at Çukuralan Mine is processed in the Ovacık Mill and tailings report to the Ovacık TSF.

3.13.4 Environmental

The EIA permit for the second revision was obtained on March 11, 2011. The EIA permitting process for the third revision is on-going. Due to environmental sensitivities in the Çukuralan Project area, additional environmental studies are critical for further stages of environmental permitting, and for a solid environmental management system.

4 Çoraklıktepe Development Project Resources and Reserves

The Çoraklıktepe Mine was completed in 2014 and the resources and reserves have been depleted from the 2014 resource statement.

4.1 Çoraklıktepe Mine Production

Mine operations at Çoraklıktepe finished in October of 2014 after a successful operation that produced 30% more ounces than predicted by the 2013 technical economic model achieved through more tonnes at a slightly reduced grade. There are no plans for further development at Çoraklıktepe.

Table 4.1.1 details the Çoraklıktepe production compared to that predicted in the economic model.

Table 4.1.1: Çoraklıktepe Mine Production

2014 Production	Çoraklıktepe 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	19,892	3.83	8.58	2,448	4,503	4.17	3.37	604	-77%	9%	-75%
February	24,484	4.58	9.07	3,609	6,766	4.03	3.18	877	-72%	-12%	-76%
March	31,236	3.82	9.51	3,838	11,350	4.21	4.17	1,536	-64%	10%	-60%
April	14,846	5.69	12.60	2,717	13,463	4.75	6.48	2,056	-9%	-17%	-24%
May	28,168	4.17	12.53	3,773	19,808	6.16	11.05	3,923	-30%	48%	4%
June	23,032	5.73	10.08	4,243	17,871	6.72	13.70	3,861	-22%	17%	-9%
July	472	3.41	4.49	52	11,802	6.52	13.98	2,474	2400%	91%	4681%
August	4,392	7.41	18.00	1,046	823	7.30	14.37	193	-81%	-1%	-82%
September	835	4.47	16.57	120				-	-100%	-100%	-100%
October	2,287	3.37	11.76	248				-	-100%	-100%	-100%
Total	**149,644**	**4.59**	**10.58**	**22,094**	**86,386**	**5.59**	**9.40**	**15,524**	**-42%**	**22%**	**-30%**

While no further mining operations are planned, there still remains stockpiled material that may be processed if higher grade RoM material from Çukuralan and Ovacik is not forthcoming.

Table 4.1.1: Çoraklıktepe Reserves

Area	Process	Proven and Probable				
		kt	g/t Au	g/t Ag	koz Au	koz Ag
Çoraklıktepe Rom Stockpile	Ovacik Mill	234	3.19	6.5	24	49

4.2 Environmental

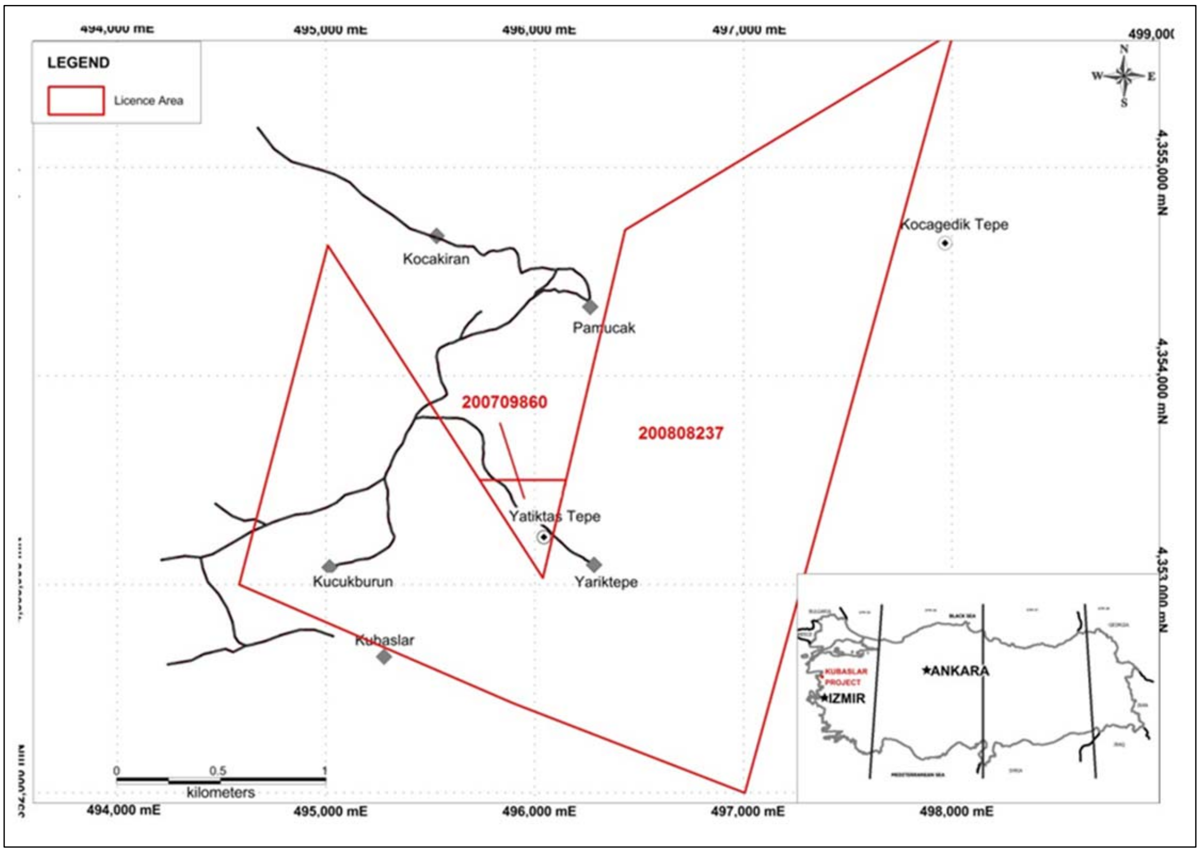
Çoraklıktepe Property is located in Balıkesir Province close to the Küçükdere Property. The EIA permit was received on November 15, 2012. The environmental permit was received on September 24, 2013 and is valid until September 24, 2018. Open pit operations were completed in June 2014. Rehabilitation activities will start after communication with the related authority. The mine closure and rehabilitation involves partial backfilling of the open pit. Koza estimates that the mine closure will roughly cost US$1.4 million.

5 Kubaşlar Project

There was no drilling at the Kubaşlar site in 2014. This section is unchanged from the EOY 2013 report except Section 5.6.7 and 5.6.8 where the cutoff grade calculation and the resource statement are updated with the 2014 gold price.

5.1 Property Description and Location

The Kubaşlar Project is located approximately 12 km southeast of Gömeç, a town located along D550. The Project is between UTM coordinates 4355000 N, 494000 E and 44351000 N, 494000 E ED1950 Zone 35. The Project is accessed from Gömeç by following village roads southeast for approximately 14 km to the village of Kubaşlar. The Project is located immediately east of the village, between approximately 400 and 540 m elevation, within exploration licenses 200709860 and 200808237. The licenses total approximately 548 ha and are shown in Figure 5.1.1.



Source: Koza, 2012 GIS

Figure 5.1.1: Kubaşlar Location Map

5.2 Climate and Physiography

The Kubaşlar Project is located in the Ovacık District and experiences a typical Mediterranean climate. Discussion of the climate and physiography of the Ovacık District is found above in Section 1.1.1.

5.3 History

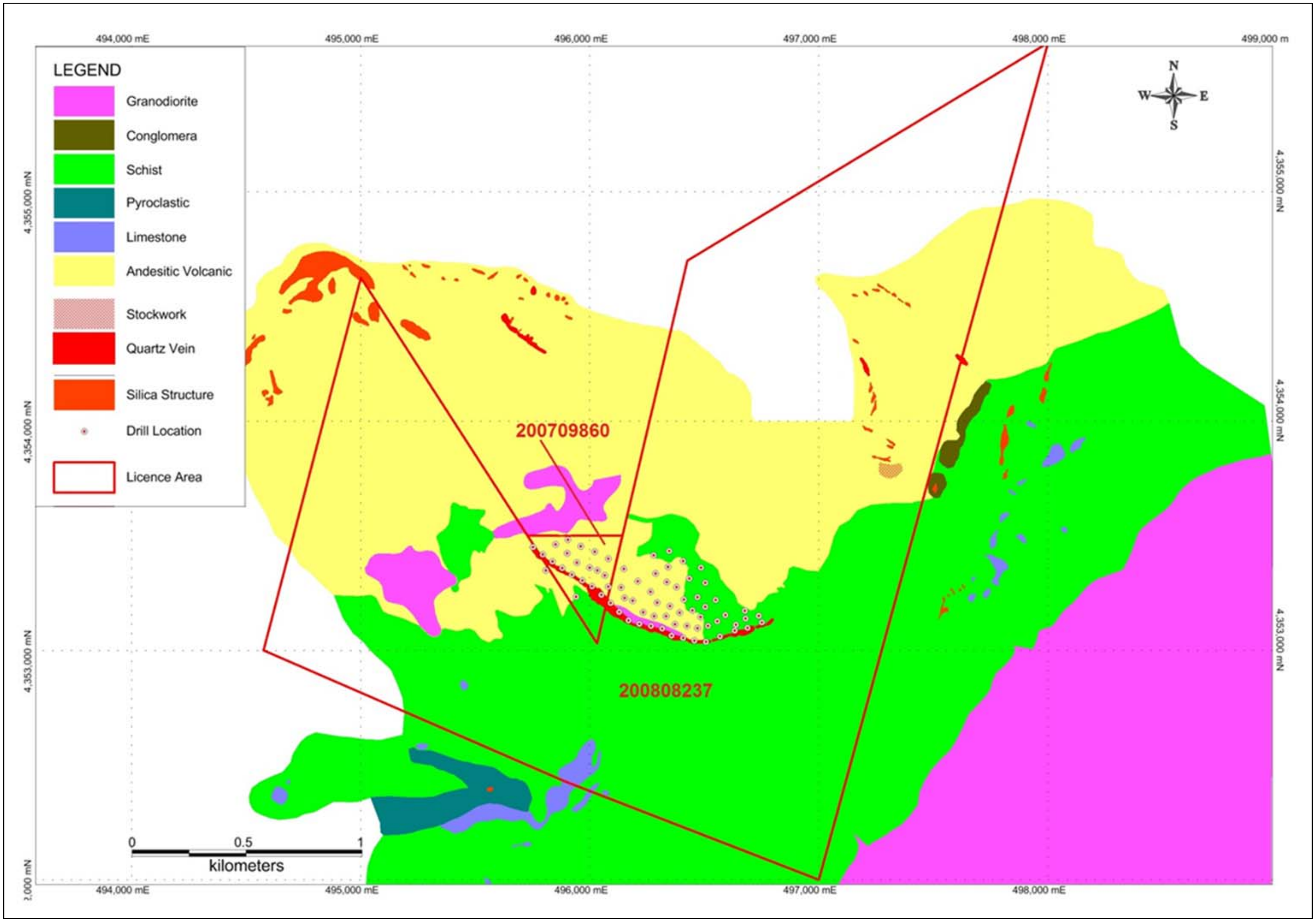
The Kubaşlar Project was held by Tüprag Metal Madencilik (Tüprag) a subsidiary of Eldorado Gold Corporation (Eldorado) between 2003 and 2006. During this time Tüprag completed eight RC holes along the length of the vein.

5.4 Geology

The Kubaşlar Project is located the Ovacık District. Regional geology of the Ovacık District is discussed in Section 1.1.2.

The Kubaşlar Project is hosted by the Triassic age Karakaya Metamorphic Complex and the lower to middle Miocene volcanic rocks, which form the capping sequence in the area. Kubaşlar mineralization is associated with the intrusion of the Kozak Granodiorite, between the early Miocene and late Oligocene. Locally the metamorphic rocks are primarily limestone, schist, serpentinite and metavolcanic units, and the volcanic rocks are composed of andesitic and dacitic calc-alkaline flows.

The Kubaşlar Project includes a quartz vein along the contact between the volcanic units and the metamorphic rocks. This vein strikes east-west for approximately 450 m then turns N50°W. The vein has approximately 1.3 km of total strike length. The vein averages 5 to 10 m in width and has been intercepted at 175 m below surface in drilling conducted by Koza. At surface, the vein has a vertical extent from exposures in streams of approximately 150 m. The veins are characterized by banding, carbonate replacement and breccia textures and include chalcedony, opaline, sugary and crystalline quartz. In addition, there is a correlation between As, Ag, Au and Sb in rock geochemistry. The geochemistry, quartz habits and textures suggest that this is the top of the epithermal system. This is further supported by the fact that the western part of the vein extends toward a flat silica cap. Also within the project area are zones of advanced argillic, propylitic and silica alteration as well as discontinuous quartz vein zones and breccias possibly indicating a larger target or a porphyry association. Figure 5.4.1 presents the geology of the Kubaşlar Project.



Source: Koza, 2012 GIS

Figure 5.4.1: Kubaşlar Geology Map

5.5 Exploration

Koza acquired the Kubaşlar Project in 2009. Between 2009 and 2010, Koza collected 391 soil samples and 360 rock chip and channel samples. During this time, Koza also completed geophysical surveys that included 12.8 line km of Induced Polarization (IP) and resistivity and ±40 line km of ground magnetics. The IP and resistivity identified favorable chargeable and resistive anomalies that are being explored by drilling. Koza has mapped the project area at a scale of 1:5,000 and is currently conducting detailed mapping at a 1:2,000 scale. Koza has also conducted Portable Infrared Mineral Analyzer (PIMA) mapping in the project area.

Between 2010 and 2011, Koza drilled 56 drill holes totaling 8,198.45 m. Drill spacing is approximately 50 to 100 m. As part of the drilling program, Koza collected 242 HQ sized samples for density measurements. These samples were collected along the length and depth of the Main Zone vein and represent both vein (QV) and breccia samples (QBX). The average density for the QV is 2.48 g/cm^3 and for QBX is 2.46 g/cm^3. Koza will continue exploration by drilling the high resistivity and chargeable anomalies identified by geophysics in the deeper portions of the mineralization. Based on geophysics, Koza is interpreting Kubaşlar as a potential porphyry target. No drilling was completed between 2013 and 2014 while drill permits were being acquired. Permits are still pending and Koza has budgeted TL24,000 (US$11,000) for licensing fees during the 2015 exploration year.

5.5.1 Sample Collection

Between 2009 and 2010, Koza collected 391 soil, and 360 rock chip samples at the project. Soil samples were collected using a regular grid. Samples were collected from the B horizon and typically 3 to 4 kg of sample was collected.

There were 47 rock chip channel samples collected at Kırıntı between 2011 and 2012. These are chip samples collected perpendicular to mineralized structures. Rock chip samples were typically 3 to 4 kg in weight. Collection points ranged from 200 to 25 m apart along the structures trend and were selected based on field observations, conditions and accessibility to the structures and veins.

5.5.2 Drilling/Sampling Procedures

Eldorado drilled seven RC holes for a total of 435 m and Turkey's General Directorate of Mineral Research and Exploration (MTA) drilled six core holes for a total for 720.25 m. In 2010, Koza collected 333 channel samples in 22 trenches (359.75 m) across the outcropping quartz vein. In 2010 and 2011, Koza drilled 8,198.45 m of core in 56 holes. Six of the drillholes were vertical and the remaining seven were drilled at inclinations between -40° and -70° to the southwest. The drillholes are on section lines approximately 50 m apart. There are between one and four drillholes on the section lines. The core recovery is excellent, averaging 99%. Table 5.5.2.1 summarizes the drilling at Kubaşlar.

Table 5.5.2.1: Kubaşlar Resource Database

Company	Type	Number	Meters	Sample Number	Meters
MTA	Core	6	720.25	605	746.21
Eldorado	RC	4	230.00	165	182.00
Koza	Core	56	8,198.45	2,643	2,609.50
	Channel	22	359.75	333	341.65
Total		**88**	**9,508.45**	**3,746**	**3,879.4**

The MTA samples were analyzed at its own laboratory in Turkey. The Eldorado and Koza samples were assayed at ALS Chemex.

5.5.3 Sample Preparation and Analysis

Core samples are held in the custody of Koza until it is shipped to the laboratory for analysis in a locked core logging facility or at the nearest mine site in a locked building. Core samples are either delivered to the laboratory by Koza personnel or shipped via commercial trucking. This is industry best practice.

Samples submitted between 2011 and 2012, were prepared at ALS İzmir. Analysis was conducted at two different laboratories in the ALS Global system. The ALS Vancouver laboratory conducted ICP multi-element analysis and ALS Romania conducted gold FA analysis. ALS Vancouver and ALS Romania have ISO 17025 accreditation for specific analytical methods through the Standards Council of Canada. ALS Vancouver's accreditation is valid through May 18, 2017 and ALS Romania's is valid through March 27, 2016.

Once the samples arrived at the laboratory, they were bar coded and entered into the Laboratory Information Management System (LIMS). All samples were dried to a maximum temperature of 60°C in order to avoid or limit volatilization of elements such as mercury (ALS code DRY-22). Soil and stream sediment samples were screened to -180 micron (80 mesh) to remove organic matter and large particles. Soil samples were then analyzed. Stream sediment samples were pulverized to 85% passing 75 microns (ALS code PUL-31) prior to digestion and analysis.

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. In this analysis, a minimum 1 g of sample is digested using aqua regia and 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 requested 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). Table 5.5.3.1 presents the analytes with upper and lower detection limits for ALS ME-MS41 and Au-ICP22.

Table 5.5.3.1: 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

After drying using ALS code DRY-22, rock chip and channel samples were crushed to 70% passing -2 mm (ALS code CRU-31) and a 1,000 g split was collected using a riffle splitter (ALS code SPL-21). The 1,000 g split was pulverized to 85% passing 75 microns (ALS code PUL-32). Koza requests a larger split pulverized to help mitigate the nugget affect.

Rock samples were analyzed using ALS code ME-ICP61, a 33 element package with trace level sensitivity. A 1g sample is put into solution using a four acid digestion and the sample is analyzed using ICP-AES. Gold was analyzed using ALS code Au-AA24, which is gold by FA using a 50g charge with an Atomic Absorption Spectroscopy (AAS) finish. The samples were also analyzed for mercury using Hg-CV41. By this method, mercury content is determined using aqua regia digestion and cold vapor AAS. Table 5.5.3.2 presents the analytes with upper and lower detection limits for ALS ME-ICP61, Hg-CV41 and Au-AA24.

Table 5.5.3.2: Analytes and Upper and Lower Detection Limits for ALS Codes ME-ICP61, Hg-CV41 and Au-AA24 in ppm Unless Otherwise Noted

Method	Analyte	Range	Method	Analyte	Range	Method	Analyte	Range
Au-AA24	Au	0.005-10	ME-ICP61	Cu	1-10,000	ME-ICP61	S	0.01-10%
Hg-CV41	Hg	0.01-100	ME-ICP61	Fe	0.01-50%	ME-ICP61	Sb	5-10,000
ME-ICP61	Ag	0.5-100	ME-ICP61	Ga	10-10,000	ME-ICP61	Sc	1-10,000
ME-ICP61	Al	0.01-50%	ME-ICP61	K	0.01-10%	ME-ICP61	Sr	1-10,000
ME-ICP61	As	5-10,000	ME-ICP61	La	10-10,000	ME-ICP61	Th	20-10,000
ME-ICP61	Ba	10-10,000	ME-ICP61	Mg	0.01-50%	ME-ICP61	Ti	0.01-10%
ME-ICP61	Be	0.5-1,000	ME-ICP61	Mn	5-100,000	ME-ICP61	Tl	10-10,000
ME-ICP61	Bi	2-10,000	ME-ICP61	Mo	1-10,000	ME-ICP61	U	10-10,000
ME-ICP61	Ca	0.01-50%	ME-ICP61	Na	0.01-10%	ME-ICP61	V	1-10,000
ME-ICP61	Cd	0.05-1,000	ME-ICP61	Ni	1-10,000	ME-ICP61	W	10-10,000
ME-ICP61	Co	1-10,000	ME-ICP61	P	10-10,000	ME-ICP61	Zn	2-10,000
ME-ICP61	Cr	1-10,000	ME-ICP61	Pb	2-10,000			

Source: ALS Global, 2014

5.5.4 Quality Assurance and Quality Control

There has been no drilling at the Kubaşlar Project since 2010; therefore, this section has not changed since the 2010 report. SRK notes that Bloom (2013) has suggested changes in the QA/QC program and SRK concurs with these recommendations.

Insertion of Internal controls

Koza inserts QA/QC control samples into the sample stream at approximately one blank per drillhole, 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 inserted into the sample stream in core sequence by the core logging geologist. The location of the control samples is noted on the sample log and in the sample database. Internal control samples have the same numbering system as the drill core samples.

Reference Materials

Koza uses one RM and one in-house standard generated from material from the Ovacık Mine. The CRM OXE74 was purchased from RockLabs. The in-house standard is OV018. For both the in-house standard and CRM, Koza uses a performance range of ±10% of the mean determined during certification of the standards. Table 5.5.4.1 presents the expected mean, standard deviations and summaries of the analyses of the Au RMs.

Table 5.5.4.1: Results of Au RM Analyses at Kubaşlar

Standard	Number of Samples	Expected (ppm)		Observed (ppm)		% of Expected	Number Failures	% Failure Rate
		Mean	Std Dev	Mean	Std Dev			
OXE74	80	0.615	0.017	0.613	0.018	99	3	4
OV018	7	1.858	0.042	1.787	0.165	96	0	0
Total	**87**						**3**	**4**

There are only seven analytical results for OV018, which is a statistically small data set, but there were no failures. For CRM OXE74 there were three failures. The results for both of the control samples are within the performance range indicating that the laboratory has good accuracy. Both these control samples are performing low and should continue to be monitored.

Koza is using ±10% as a performance gate for RMs at the project. SRK recommends that Koza consider the following performance gates for CRMs:

- If one analysis is outside of ±2 standard deviations it is a warning;
- Two or more consecutive analyses outside of ±2 standard deviations is a failure;
- If an analysis is outside ±3 standard deviations it is a failure if ±3 standard deviations does not exceed ±10% of the mean; and
- If the ±3 standard deviations exceed ±10% of the mean, then ±5 to ±10% should be used.

Ore Research & Exploration (OREAS), who manufactures CRMs, recommends using these performance gates and has started printing this information on CRM certificates as part of a guide for use of the CRM. ALS Global uses ±3 standard deviations during analysis as a performance gate for internal CRMs (ALS Global, 2012).

Blanks

Sample blanks test for contamination in preparation and assaying and handling errors. Koza inserts at least one sample blank per drillhole using pulp blanks up until June 2012, and preparation blanks since then. A blank failure is a result greater than five times the detection limit. SRK has examined the results for gold in the blank samples. There were no failures in 48 blank sample analyses.

Preparation Duplicates

Preparation duplicates are created by taking a second split of the crushed sample (coarse reject) using the same method 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.

Koza sent 22 preparation duplicates to the original lab for Au analysis. A summary of the analytical results are presented in Table 5.5.4.2.

Table 5.5.4.2: Summary of Duplicate Au Analysis at Kubaşlar

Criteria	Number of Samples	Original>Dup	Dup>Original	Original = Dup	Within +/- 20%
All samples	22	12	7	3	22
		54%	32%	14%	100%

The results from this small database indicate that Kubaşlar preparation duplicates have good reproducibility between original and duplicates. This indicates that the preparation procedures are appropriate for the material at Kubaşlar.

Pulp Duplicates

Koza does not submit pulp duplicates at this time. Pulp duplicates test the analytical reproducibility or precision of the analysis. SRK recommends that Koza add pulp duplicates to its QA/QC program.

Secondary Check Lab Analysis

Koza has not sent any Kubaşlar check samples in the form of pulp duplicates to a secondary laboratory as verification of the ALS Chemex analytical results. SRK recommends that Koza add this type of QA/QC samples to its program. Check samples must be analyzed at the secondary laboratory using the same method as ALS Chemex and standards must be submitted with the check samples.

SRK is of the opinion that the QA/QC data supports use of the database in resource estimation.

5.6 Mineral Resources

The Mineral Resources were estimated by Koza in 2010 (Koza, 2010).

5.6.1 Geological Model

Koza constructed three wireframe solids of the vein based on a cutoff grade of 0.50 g/t Au. These represent the main mineralized vein and smaller hanging wall and footwall zones. Figure 5.6.1.1 illustrates the wireframes and the drillholes. The wireframes are oriented to the northwest and dip to the northeast at about 55° near surface and at about 30° at depth. The wireframes have a strike length of 1100 m and a vertical extent of about 200 m. The main zone has thickness up to 20 m, with an average of about 10 m; the hangingwall and footwall zones are thinner, averaging about 2 m.

Within the wireframe there are 850 samples with basic statistics as shown in Table 5.6.1.1.

Table 5.6.1.1: Statistics of Assays within the Kubaşlar Grade Shell

Domain	Metal	Count	Min	Max	Mean	S.D.	CV
All	Au	830	0.04	17.10	1.28	1.62	1.27
	Ag	674	0.50	177.00	9.95	14.48	1.46
Footwall	Au	31	016	6.35	1.28	1.24	0.97
	Ag	19	1.00	6.40	3.11	1.73	0.56
Hangingwall	Au	32	0.13	16.40	1.65	2.96	1.80
	Ag	10	1.60	50.40	16.81	16.44	0.99
Main	Au	767	0.04	17.10	1.27	1.56	1.23
	Ag	645	0.50	177	10.06	14.61	1.45

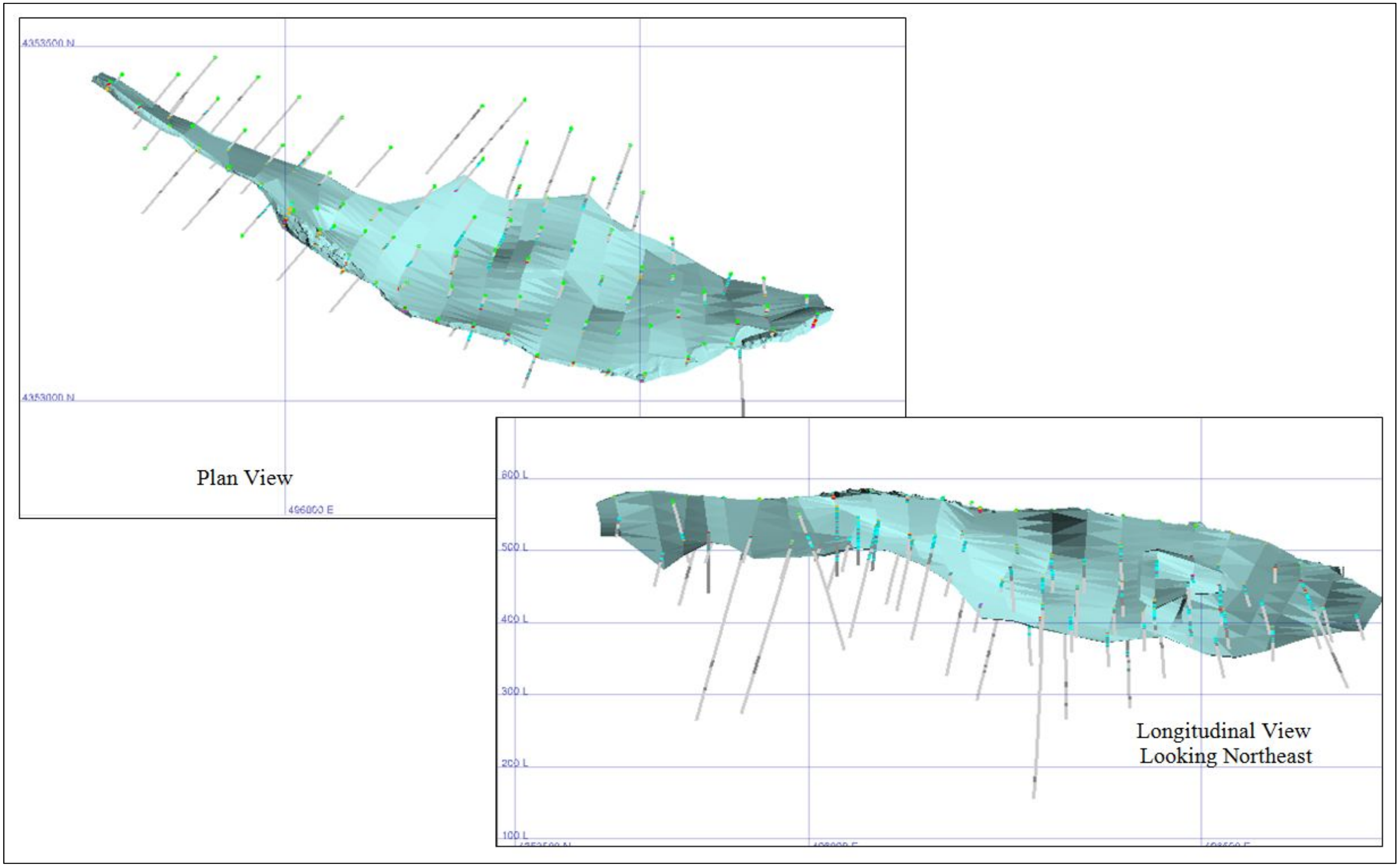


Figure 5.6.1.1: Drilling and Veins at Kubaşlar in Plan View and Oblique View Looking West

5.6.2 Capping and Compositing

Over 90% of the sample lengths are less than 1 m in length and therefore the assays were composited into 1 m lengths within the wireframes. The basic statistics of the composites are shown in Table 5.6.2.1. The unassayed silver intervals have been ignored in the statistics.

Table 5.6.2.1: Statistics of Composites within the Kubaşlar Grade Shell

Domain	Metal	Count	Min	Max	Mean	S.D.	CV
All	Au	813	0.04	17.10	1.29	1.55	1.20
	Ag	688	0.00	117.18	9.98	13.21	1.32
Footwall	Au	31	0.26	6.12	1.29	1.15	0.89
	Ag	19	1.00	6.29	3.09	1.61	0.52
Hangingwall	Au	31	0.13	16.40	1.64	2.93	1.78
	Ag	9	1.63	42.04	17.08	14.23	0.83
Main	Au	751	0.04	17.10	1.27	1.48	1.16
	Ag	640	0.00	117.18	10.08	13.32	1.32

The need for capping was reviewed with a quantile analysis which showed that the top 1% of the Au samples (8) accounted for 9% of the contained metal, and Koza capped those intervals at 7 g/t Au. A similar exercise was done for Ag, resulting in a capping value of 67.5. The coefficient of variation in the silver values was slightly reduced by the capping.

Statistics of the capped composites are shown in Table 5.6.2.2.

Table 5.6.2.2: Statistics of Capped Composites within the Kubaşlar Grade Shell

Domain	Metal	Count	Min	Max	Mean	S.D.	CV
All	Au	813	0.04	7.00	1.25	1.29	1.03
	Ag	688	0.00	67.50	9.77	12.02	1.23
Footwall	Au	31	0.26	6.12	1.29	1.15	0.89
	Ag	19	1.00	6.29	3.09	1.61	0.52
Hangingwall	Au	31	0.13	7.00	1.34	1.47	1.10
	Ag	9	1.63	42.04	17.08	14.23	0.83
Main	Au	751	0.04	7.00	1.24	1.29	1.03
	Ag	640	0.00	67.5	9.87	12.09	1.26

5.6.3 Density

Koza conducted density determinations on 242 samples from the 56 HQ sized core holes. The samples were grouped according to rock type, alteration, degree of breakage and mineralized/non-mineralized. A density of 2.50 g/cm^3 was used in the resource estimation based on the average value of the mineralized samples.

5.6.4 Variography

Because of the limited number of samples, it was not possible to conduct a variography study.

5.6.5 Grade Estimation

A block model was created with a block size of 10 m x 5 m x 5 m, with sub-blocking to 1 m.

The estimation was done using ID2 in three passes with the search ellipsoid oriented along the strike and dip of the vein and sample selection as follows:

- First Pass: minimum of 5, maximum of 12 composites, search 50 m x 50 m x 10 m;
- Second Pass: minimum of 5, maximum of 20 composites, search 100 m x 100 m x 20 m;
- Third Pass: minimum of 3, maximum of 12 composites, search 150 m x 150 m x 30 m; and
- Only composites within the wireframe were used for estimation.

In addition, an octant search was used for the first pass where a minimum of two octants were required with a minimum of one sample and a maximum of four samples per octant. This effectively requires a minimum of two drillholes for the first pass.

The resource was validated by visually inspecting the composites and the block grades on cross-sections and reviewing statistics for both. In addition, estimations were done using ID3 and NN methodologies and the results compared to the ID2 results. Table 5.6.5.1 presents the statistics of the block gold and silver grades.

Table 5.6.5.1: Statistics of Blocks within the Kubaşlar Grade Shell

Domain	Au				Ag			
	ID2	ID3	NN	Composite	ID2	ID3	NN	Composite
All	1.30	1.30	1.30	1.25	10.95	10.95	11.05	9.77
Main	1.31	1.30	1.30	1.24	10.87	10.84	10.69	9.87
Footwall	1.18	1.17	1.10	1.29	2.31	5.07	3.94	3.09
Hangingwall	1.36	1.43	1.43	1.34	17.94	18.56	24.89	17.08

The Au estimated grades compare well with the composite grades. The average estimated Ag grade of the main and hangingwall zones are higher than the average Ag grade of the composite by about 10%. It is generally unacceptable for the estimated grade to be higher than the composite grade and Koza should try to find the reason why, perhaps by using swath plots to identify areas where the block and composite grades are out of line.

5.6.6 Mineral Resource Classification

The blocks estimated on the first pass were classified as Indicated and blocks estimated in the second and third pass were classified as Inferred.

5.6.7 Mineral Resource Statement

The resources at Kubaşlar are stated at a cutoff grade of 1.10 g/t Au based on the parameters shown in Table 5.6.7.1 and the assumption that mining would be by open pit methods and reserves would be transported to and processed at the Ovacık mill. 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 5.6.7.1: Kubaşlar Cutoff Grade Parameters

Prices and Costs	Units	OP
Gold Price	US$/oz	1,450
Gold Recovery	%	0.74
Gold Refining	US$/oz	3.44
Royalty	%	3
Government Right	%	1
Process Cost	US$/t	11.00
Mining Cost	US$/t	0.00
G&A Cost	US$/t	15.00
Transport Cost	US$/t	11.00
Calculated Cutoff grade	g/t	1.12
Final Cutoff grade	g/t	1.10

Source: Koza, 2014

It is becoming an industry practice to state mineral resources within a pit optimization shell. Koza conducted a pit optimization exercise in 2011 and nearly 90% of the Indicated resource and 40% of the Inferred resource fall within the pit shell. Mineral Resources stated in this report are not pit shell constrained and SRK recommends that Koza generate a pit shell for that purpose.

The mineral resources at December 31, 2014 are listed in Table 5.6.7.2.

Table 5.6.7.2: Kubaşlar Mineral Resources, at December 31, 2014, Including Ore Reserves

Classification	kt	g/t Au	g/t Ag	koz Au	koz Ag
Indicated	1,726	1.91	13.6	106	754
Inferred	204	2.17	12.5	14	82

- Tonnages and grade are rounded to reflect approximation;
- Resources are stated at a cutoff grade of 1.10 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.

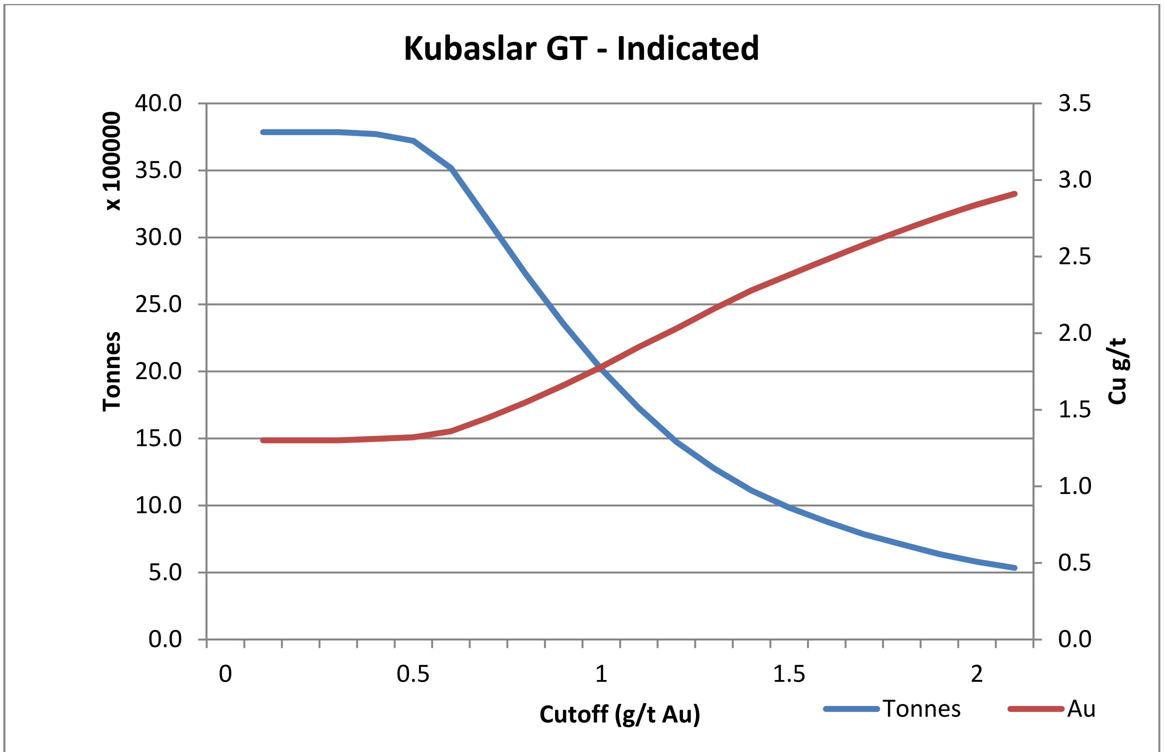
5.6.8 Mineral Resource Sensitivity

Figure 5.6.8.1 presents grade tonnage curves for the Indicated and Inferred Resources.

Cutoff grades for the Kubaşlar resource at various gold prices are shown in Table 5.6.8.1.

Table 5.6.8.1: Kubaşlar Cutoff Grades vs. Gold Price

Gold Price	Cutoff Grade
1600	1.01
1550	1.05
1500	1.08
1450	1.12
1400	1.16
1350	1.20
1300	1.25
1250	1.30
1200	1.35



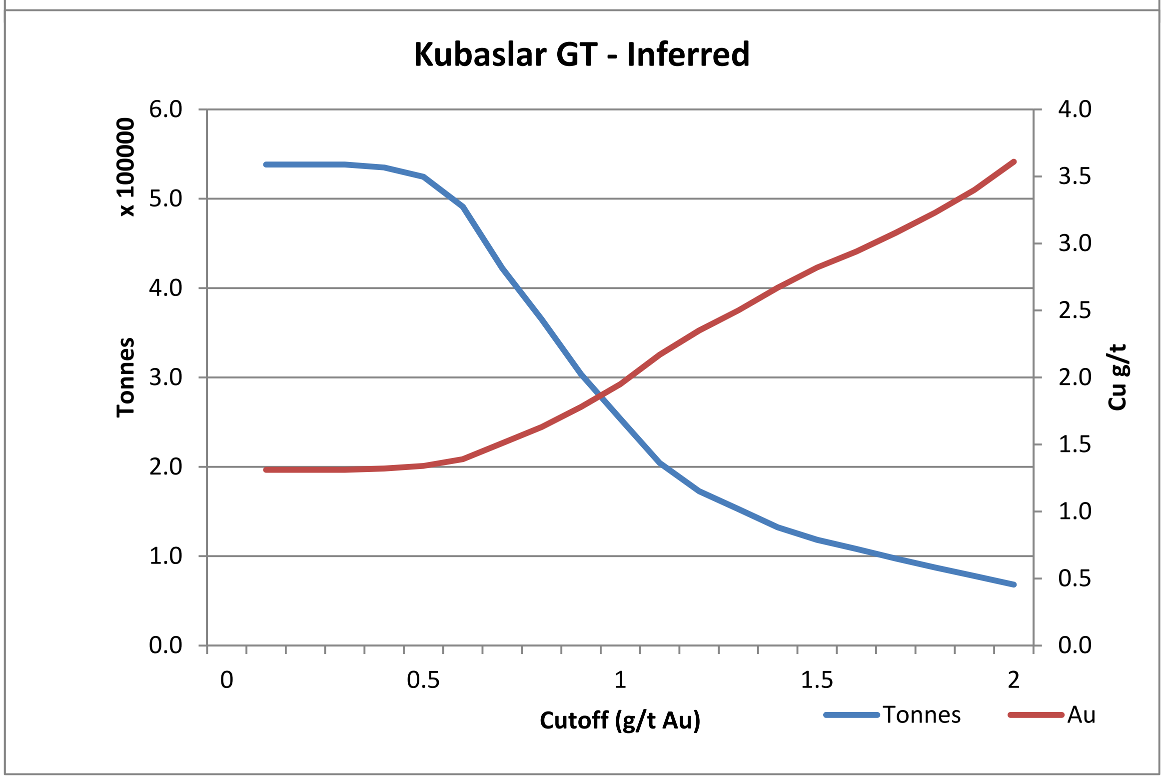


Figure 5.6.8.1: Grade Tonnage Curves for Kubaşlar Resource

5.7 Ore Reserve Estimation

Kubaşlar is a Project at the prefeasibility stage and is due to commence ore production in January of 2024 and finish by October the same year. Kubaşlar will be a small open pit operation. Ore is to be transported to the Ovacık processing facility and mining is expected to be carried out by contractors.

The ore at Kubaşlar is to be extracted using open pit mining methods. 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 section discusses the procedures used to estimate gold grade. The modifying factors include the metal value and recovery. Only material that can be mined for a profit is considered as a reserve.

5.7.1 Modifying Factors

Kubaşlar is similar to the majority of gold occurrences mined by Koza at the Ovacık processing facility. As such the material modifying factors to the reserve quantum is the transportation cost of ore to the Ovacık processing facility and the reduced recovery estimate of 71%. Koza has estimated this cost at US$11.00/ore per tonne which is a reasonable estimation.

The pit optimization parameters are given in Table 5.7.1.1.

Table 5.7.1.1: Kubaşlar Pit Optimization Inputs

Parameter	Unit	Amount
Mining Cost	US$/t	1.68
Rehabilitation Cost	US$/t waste	0.20
Milling Cost	US$t/ore	11.88
Selling Cost	US$/oz	3.44
Grade Control	US$t/ore	0.50
Administration	US$t/ore	19.42
Ore Rehandle	US$t/ore	0.50
Transport	US$t/ore	11.00
Gold Price	US$/oz	1,250
Silver Price	US$/oz	20
Gold Recovery	%	71
Silver Recovery	%	75
Cutoff grade	g/t Au	1.54

Source: Koza, 2014

As with other small satellite deposits feeding the Ovacık facility, reserves are based on the assumption that there are no onerous environmental or permitting issues.

5.7.2 Reserve Classification

Ore tonnes which lie within the final pit design shape are able to be 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 technical economic model. The Kubaşlar open pit reserves are given in Table 5.7.2.1.

Table 5.7.2.1: Kubaşlar Open Pit Mineral Reserves at December 31, 2014

Category	kt	g/t Au	g/t Ag	koz Au	koz Ag
Probable Reserve	927	2.31	14.5	69	433
Total Probable Reserves	**927**	**2.31**	**14.5**	**69**	**433**

Source: Koza, 2014
Metal Price: US$1,250/oz-Au, US$20/oz-Ag, Au Recovery 71%, Ag Recovery 75%, Au cutoff grade 1.54 g/t

5.8 Mining

As with other satellite deposits around the Ovacık processing facility, Kubaşlar will be mined using contractors with ore transported via highway to Ovacık. The pit will have a moderately low stripping ratio of 3.3:1 (Waste:Ore) and ore will be available without the need for pre-stripping.

Mining is expected to commence in January 2024 mining approximately 100,000 t of ore per month when in full production.

Operations will include drill, blast, load and haul processes utilizing a small excavator and trucks. Mine benches are sized at 5 m with two internal 2.5 m flitches which characterize the selective mining face. Waste will be disposed of around the pit edges.

Long term mine planning is carried out at the Ankara office and is consistent with all other Koza operations. A single mining engineer employed by Koza will oversee contractor operation.

Figure 5.8.1 illustrates the Kubaşlar site overview, waste dump and pit design.

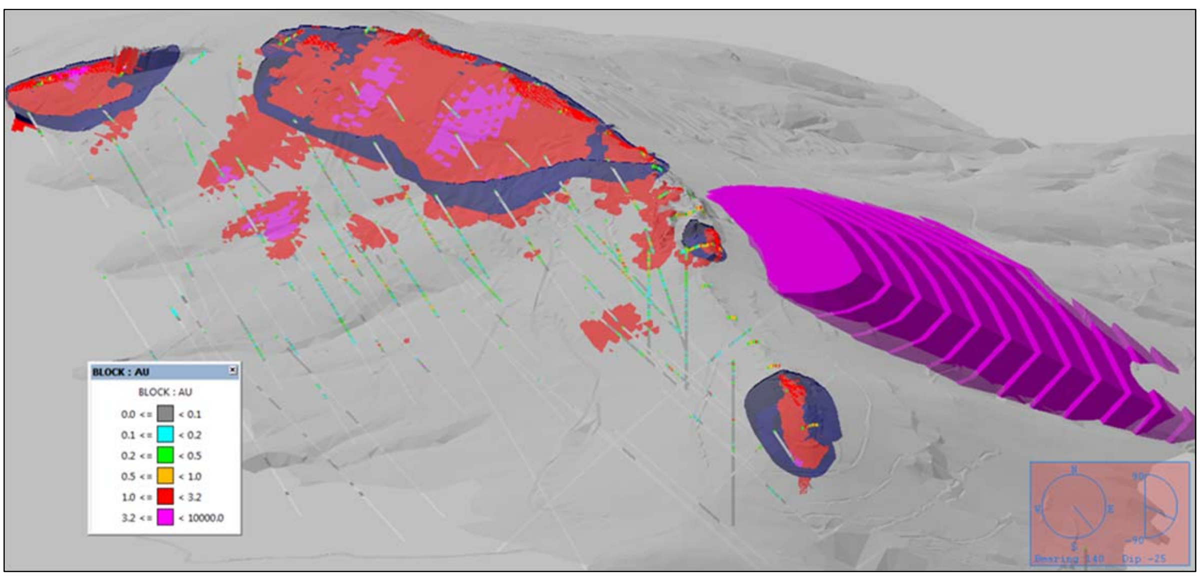
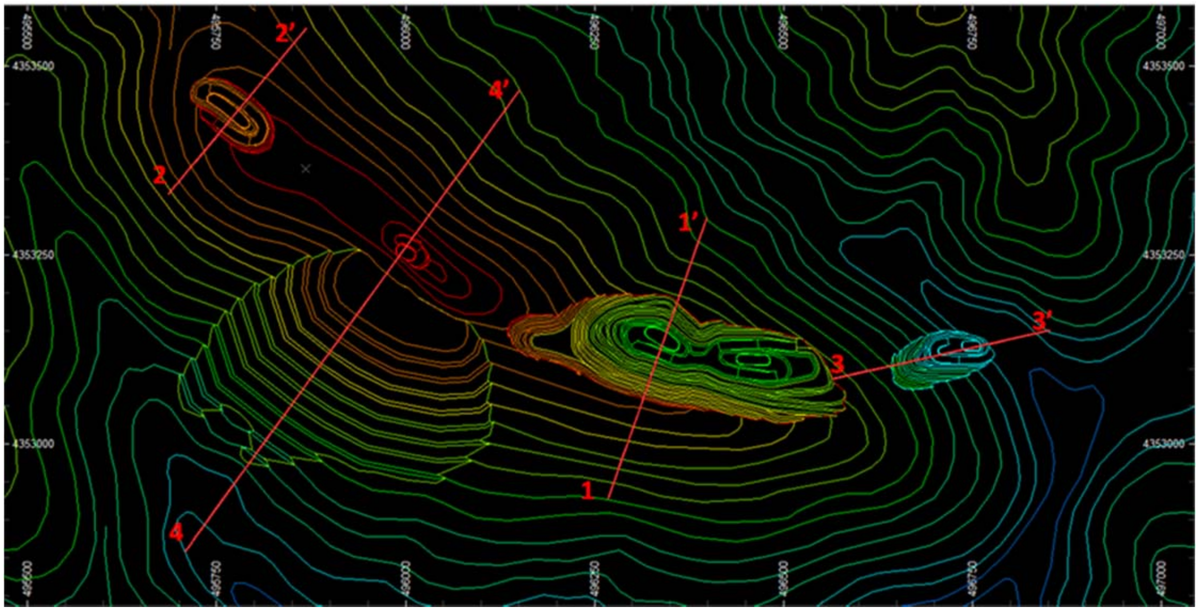


Figure 5.8.1: Kubaşlar Site Overview

5.8.1 Geotechnical

Geotechnical analysis performed on Kubaşlar is based on a limit equilibrium analysis using section lines displayed in Figure 5.8.1.1. An internal friction angle of 35º and cohesion strength of 350 kPa were assumed.



Source: Koza, 2012e

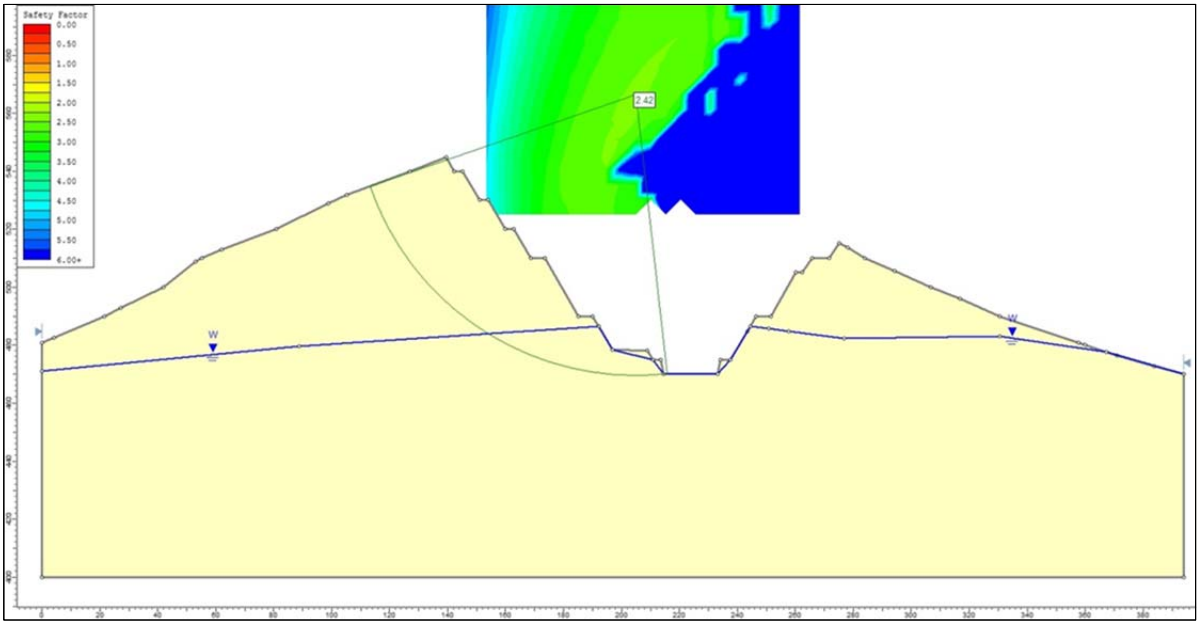
Figure 5.8.1.1: Geotechnical Sections

The resultant factors of safety are detailed in Table 5.8.1.1. As any factor of safety above 1.3 is usually considered to be safe, there is unlikely to be major stability problems for the Çoraklıktepe pit.

Table 5.8.1.1: Kubaşlar Geotechnical Factors of Safety

Section	Slope Height	Slope Angle	Factor Of Safety
Section 1-1'	75	45	2.42
Section 2-2'	27	47	4.55
Section 3-3'	48	42	2.75
Section 4-4'	116	26	1.78

Source: Koza, 2013



Source: Koza, 2013

Figure 5.8.1.2: Geotechnical Analysis

5.9 Metallurgy and Process

Koza has conducted bottle roll tests on Kubaşlar samples. The results were provided to SRK after the audit process and have not been reviewed.

5.10 Environmental

The Kubaşlar property is located 12 km north of Çukuralan, 25 km southwest of Küçükdere, and 27 km northwest of the Ovacık property. The environmental baseline study, including the geochemical and hydrogeological aspects and EIA studies of the Kubaşlar Project were conducted in 2011 and the EIA permit was received on February 7, 2013. A temporary environmental permit was received on December 30, 2013 and was valid until December 30, 2014. Operations have not been started since other permissions have not been obtained. Currently, partial backfilling is planned at the Kubaşlar mine for open pit closure. Koza estimates that the total closure cost will be about US$1.1 million. However, the geochemical assessment is on-going. The mine closure costs and plans should be revisited after this assessment is completed.

5.11 Conclusions and Recommendations

SRK recommends that Koza generate a pit shell and that it be used to confine the reported resources.

SRK observed that the QA/QC program in place at Kubaşlar is providing appropriate data for the Project. SRK notes that the QA/QC data is limited and recommends that Koza continue to monitor the CRMs, which are performing low and the preparation duplicates to confirm that sample preparation is appropriate. SRK also recommends that Koza add pulp duplicates to monitor analytical precision and check samples to verify analytical results at the primary laboratory. Koza is using ±10% as a performance gate for RMs at the project. SRK recommends that Koza consider the following performance gates for CRMs:

- If one analysis is outside of ±2 standard deviations it is a warning;
- Two or more consecutive analyses outside of ±2 standard deviations is a failure;
- If an analysis is outside ±3 standard deviations it is a failure if ±3 standard deviations does not exceed ±10% of the mean; and
- If the ±3 standard deviations exceed ±10% of the mean, then ±5 to ±10% should be used.

Ore Research & Exploration (OREAS), who manufactures CRMs, recommends using these performance gates and has started printing this information on CRM certificates as part of a guide for use of the CRM. ALS Global uses ±3 standard deviations during analysis as a performance gate for internal CRMs (ALS Global, 2012).

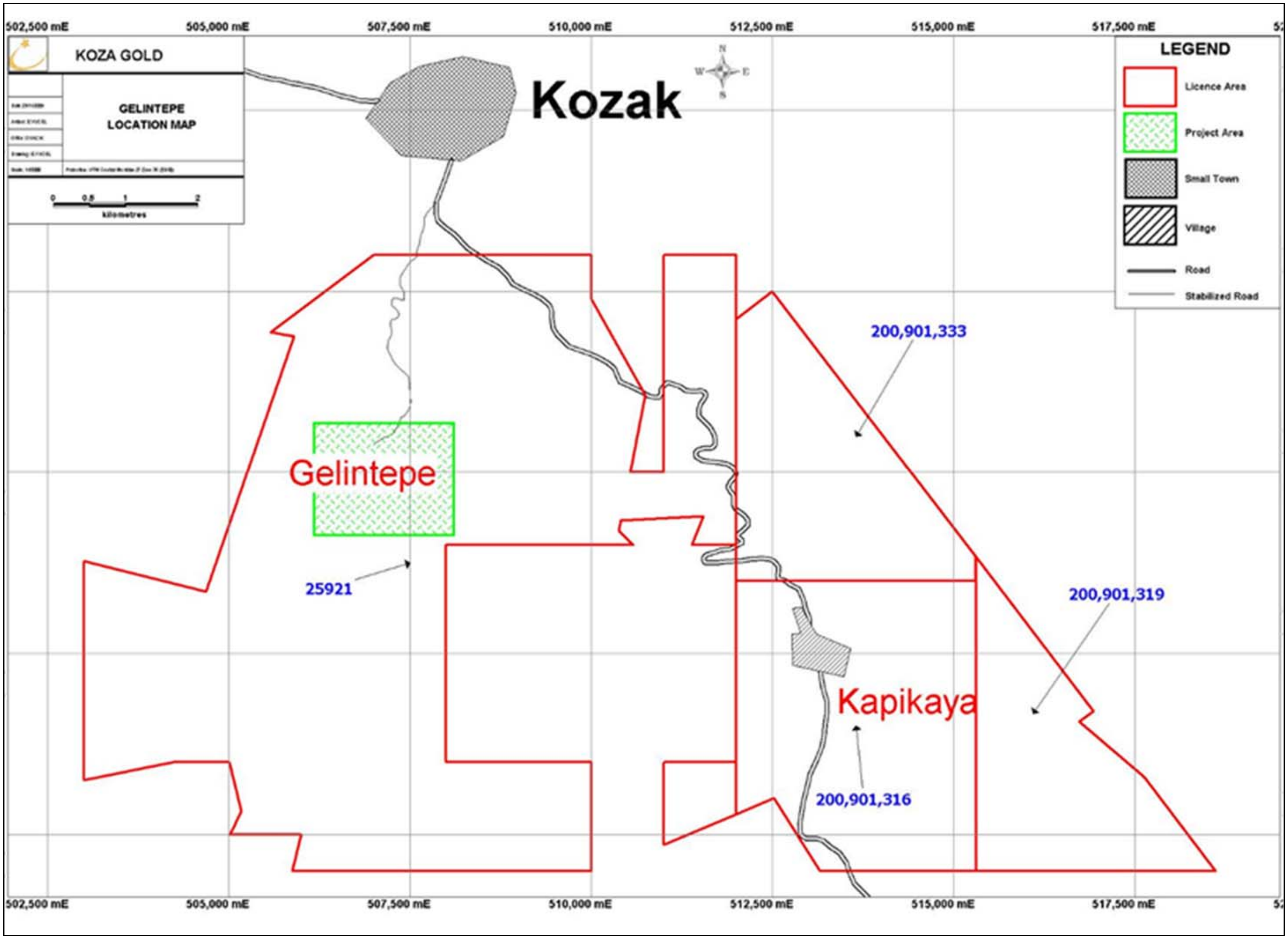
6 Gelintepe Project

There was no drilling at the Gelintepe site in 2014. This section is unchanged from the EOY 2013 report except Section 6.6.5 where the cutoff grade calculation and the resource statement are updated with the 2014 gold price.

6.1 Property Description and Location

The Gelintepe Project is located approximately 30 km northwest of the Ovacık Mine and can be accessed from Bergama, Turkey. This Project is located near the village of Kozak, which is 30 km north of Bergama. Gelintepe is approximately 5 km west of Kozak between UTM coordinates 4340500 N, 506000 E to 4339000 N, 508000 E in ED1950 Zone 35, at an elevation of 700 to 900 m.

Gelintepe is located within operating license 25921 totaling approximately 4,543 ha. Koza holds one operation permit for gold, which comprises 400 ha within operation license 25921. Land tenure for the Gelintepe Project is shown in Figure 6.1.1.



Source: Koza, 2012 GIS

Figure 6.1.1: Gelintepe Location Map

6.2 Climate and Physiography

The Gelintepe Project is located in the Ovacık District and experiences a typical Mediterranean climate. Discussion of the climate and physiography of the Ovacık District is found above in Section 1.1.1.

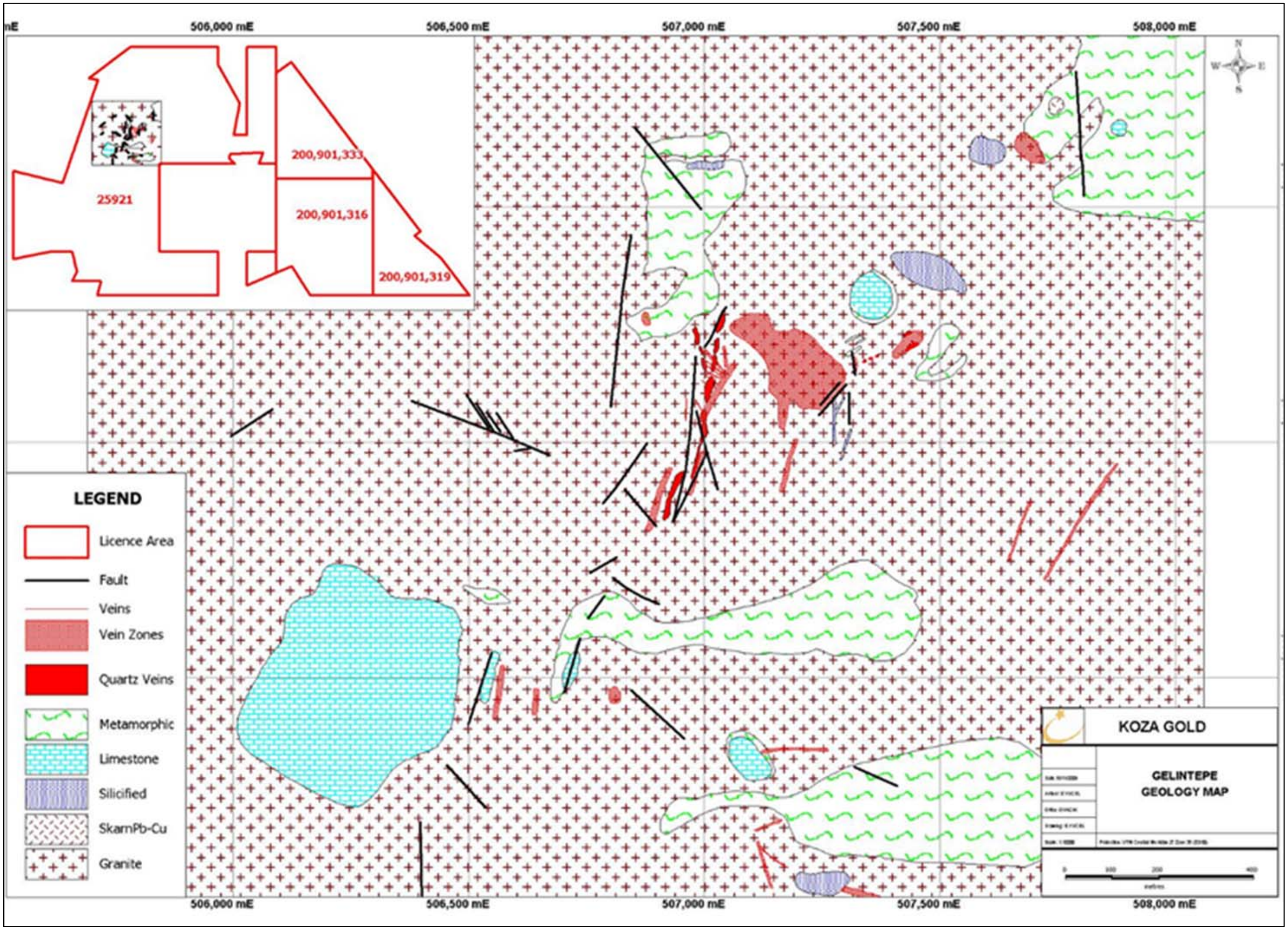
6.3 History

From 1990 to 2005, the Gelintepe Project area was held by Eurogold and Normandy who collected 421 soil samples and mapped the area at a 1:1,000 scale.

6.4 Geology

The Gelintepe Project is located in the Ovacık District. Regional geology of the Ovacık District is discussed in Section 1.1.2.

The Gelintepe Project is located 30 km north of Ovacık Mine and is described as a high temperature, intermediate sulfidation, gold vein system related to, and hosted by, early Miocene age Kozak Granite. The granite has intruded Triassic age limestone of the Kapıkaya Metamorphic Complex and in places, Pb-Zn skarn has developed. Mineralization is exposed over an area approximately 1.8 km x 0.9 km and is composed of a series of massive quartz veins, quartz stockwork and silica structures at surface. The most continuous veins are oriented at N30-65°E and N to N5°W. A smaller discontinuous set is oriented at N55°W. Sulfide mineralization includes chalcopyrite, pyrite, sphalerite and galena. Figure 6.4.1 shows the local geology of the Gelintepe Project.



Source: Koza, 2012 GIS

Figure 6.4.1: Gelintepe Geology Map

6.5 Exploration

Koza acquired the property in 2005. Between 2005 and 2007, Koza collected stream sediment and soil samples, as well as rock chip samples, completed three trenches and drilled eleven core holes. Koza has also mapped the property at 1:1,000 and 1:500 scales. Gelintepe is under the Kapakaya exploration budget. Koza has budgeted TL564,000 (US$251,000) for exploration at these projects during 2015.

6.5.1 Sample Collection

Stream sediment samples were collected along master streams above and below the inflow of tributary creeks. Samples were collected to be as representative as possible by collecting a composite sample from the same depositional environment in the stream bed at each sample location. Koza screens stream sediment samples to -80 mesh and typically collects 3 to 4 kg of sample.

Soil samples were collected over a regular grid from the B horizon and were typically 3 to 4 kg of material.

Koza also collected trench samples from three trenches. The trench samples were continuous chip samples collected to be as representative as possible of a cut channel sample but may vary in harder or softer material. Channel samples are typically 1 m long but vary in depth and width depending on field conditions and lithological contacts. Widths range from 5 to 15 cm and depths range from 15 to 20 cm. Sample weights range from 3 to 4 kg. Samples may be shorter or slightly longer than 1 m to accommodate changes in lithology.

6.5.2 Drilling/Sampling Procedures

Koza drilled 11 core holes in 2007; in addition, 33 lines of grade control channel samples were taken (Figure 6.5.2.1). The grade control samples are the standard method of sampling at Ovacık and Küçükdere Mines whereby lines are marked on the ground at 10 m intervals perpendicular to the strike of the vein. The sample locations on the lines are surveyed at 1 m intervals and the surface is cleaned with a backhoe. A jackhammer is used to break up the rock and a sample of about 3 kg is collected by the sample collectors. This procedure only gives two-dimensional information on the metal grade for the bench. However, with steeply dipping veins, it is an acceptable, although not optimal method of sampling.

The core was sampled and assayed according to the Ovacık exploration procedures and assayed at the Ovacık lab.

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.

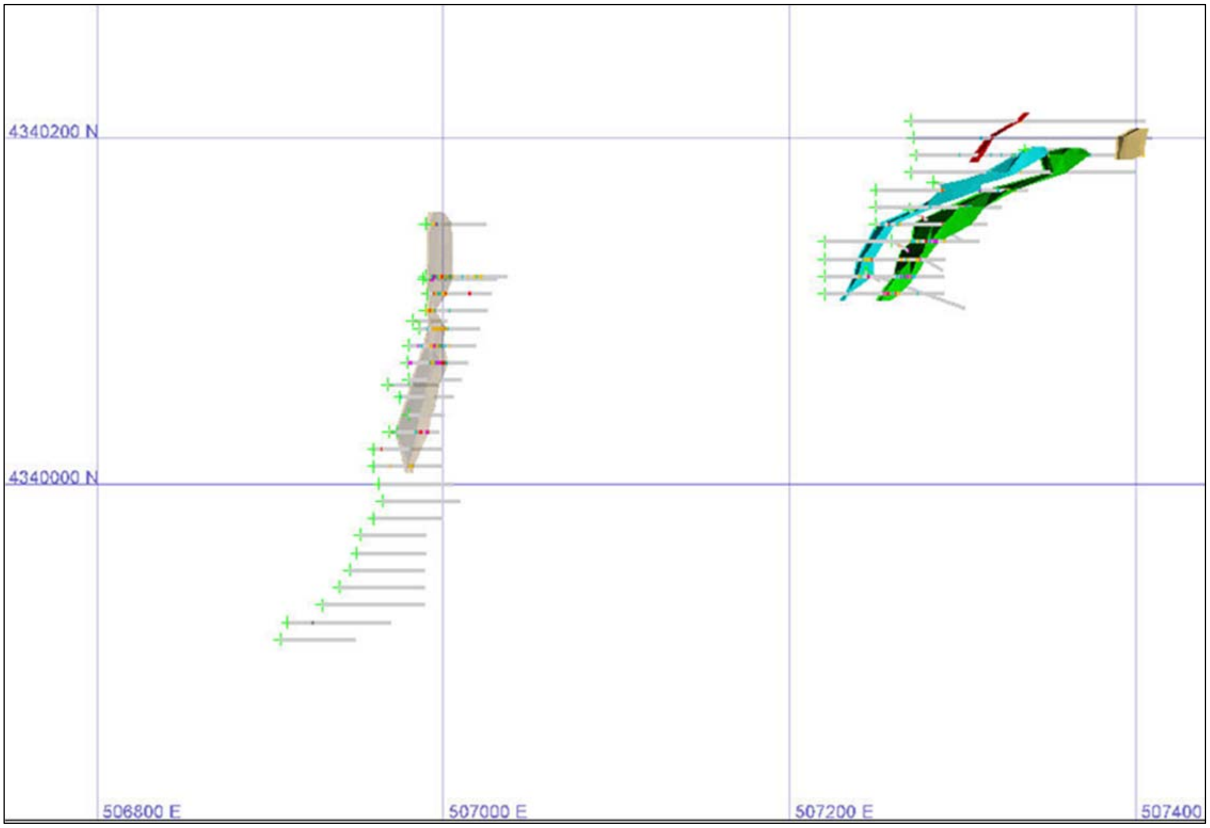


Figure 6.5.2.1: Drilling and Grade Control Samples and Veins at Gelintepe, in Plan View

6.5.3 Quality Assurance and Quality Control

There has been no drilling at the Gelintepe Project since 2010; therefore, this section has not changed since the 2010 report. SRK notes that Bloom (2013) has suggested changes in the QA/QC program and SRK concurs with these recommendations.

Insertion of Internal controls

Koza inserts QA/QC control samples into the sample stream at approximately one RM per 25 samples. These samples are inserted into the sample stream in sampling numbering sequence by the core logging geologist. The location of the control samples is noted on the sample log and in the sample database. Internal control samples have the same numbering system as the drill core samples.

Reference Materials

Since 2007, two different RMs have been used at Gelintepe: both produced in-house with material from both the Ovacık Mine and Çukuralan mine. The in-house standards were developed in conjunction with ALS Chemex and were analyzed either by ALS Canada or ALS Australia. For the in-house standards Koza uses a performance range of ±10% of the certified mean. Table 6.5.3.1 presents the expected mean, standard deviations and summaries of the analyses of the Au RMs.

Table 6.5.3.1: Results of Au RM Analyses at Gelintepe

Standard	Number of Samples	Expected (ppm)		Observed (ppm)		% of Expected Mean	Number Failures	% Failure Rate
		Mean	Std Dev	Mean	Std Dev			
OV10	25	11.71	0.596	11.954	0.151	102.1	0	0.0
OV12	18	46.86	1.994	47.378	0.862	101.1	0	0.0
Total	**43**						**0**	**0.0**

There were no failures in the Gelintepe data and the observed means are between 101.1 and 102.1% of the expected values for Au. Standard OV10 samples all performed higher than the mean with the exception of two which equaled the mean. Of the OV12 standard samples, 13 performed higher than the mean and five performed lower than the mean. The results indicate that the laboratory has relatively good accuracy. SRK recommends adding a lower and a middle grade RM to this Project that would bracket the grade range at Gelintepe.

Koza reviews all QA/QC during drilling programs and contacts the laboratory when analytical failures are identified. This is industry best practice.

The results indicate that the laboratory is providing accurate data.

Blanks

Coarse blanks test for contamination during preparation and assaying as well as handling errors. Koza has not inserted blank samples during drilling programs at Gelintepe. SRK recommends that Koza add coarse blank samples to its QA/QC program.

Preparation Duplicates

Koza has not had any Gelintepe preparation duplicates analyzed at this time. Preparation duplicates are created by taking a second split of the crushed sample (coarse reject) using the same method 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.

SRK recommends that Koza add preparation duplicates to its QA/QC program.

Pulp Duplicates

Koza does not submit pulp duplicates at this time. Pulp duplicates test the analytical reproducibility or precision of the analysis. SRK recommends that Koza add pulp duplicates to its QA/QC program.

Secondary Check Lab Analysis

Koza has not sent any Gelintepe check samples in the form of pulp duplicates to a secondary laboratory as verification of the ALS Chemex analytical results. SRK recommends that Koza add this type of QA/QC samples to its program. Check samples must be analyzed at the secondary laboratory using the same method as ALS Chemex and standards must be submitted with the check samples.

6.6 Resource Estimation

The Mineral Resource was estimated by SRK in 2010.

6.6.1 Geological Model

Five, 3D solids were created from the drillholes and channel samples (Figure 6.5.1.1). Because the channel samples only give information at the surface and because the drillholes only test about 10 m below the surface, the solids do not have a large down dip extent.

6.6.2 Compositing and Capping

The drillholes were composited on 1 m intervals with breaks at the solid boundaries. The probability plots of the composites were inspected and it was deemed that capping was not necessary.

6.6.3 Variography

Because of the limited number of samples, it was not possible to conduct a variography study.

6.6.4 Grade Estimation

A block model was created with a block size of 6 m x 6 m x 2.5 m, with sub-blocking to 1 m x 1 m x 1 m. A minimum of three and maximum of 12 composites were used to estimate grade. ID2 was used for grade estimation, using only composites within the 3D solid to estimate grade within the vein. The search was oriented along the strike and dip of the vein, with search ranges of 50 m x 50 m x 50 m.

The resource was validated by visually inspecting the composites and the block grades and reviewing statistics for both.

6.6.5 Mineral Resource Classification and Statement

The blocks were all classified as Inferred because of the geologic uncertainty below surface.

The resources at Gelintepe are stated at a cutoff grade of 1.85 g/t Au based on the underground mining parameters shown in Table 6.6.5.1 and the assumption that the reserves would be transported to and processed at the Ovacık mill. The one year rolling average gold price is US$1,411; the two year average is US$1,540; and the three year average is US$1,549.

Table 6.6.5.1: Gelintepe Cutoff Grade Parameters

Prices and Costs	Units	OP
Gold Price	US$/oz	1,450
Gold Recovery	%	0.95
Gold Refining	US$/oz	3.44
Royalty	%	3
Government Right	%	1
Process Cost	US$/t	11.00
Mining Cost	US$/t	45.00
G&A Cost	US$/t	15.00
Transport Cost	US$/t	8.00
Calculated Cutoff grade	g/t	1.86
Final Cutoff grade	g/t	1.85

Source: Koza, 2014

Table 6.6.5.2 lists the resources at Gelintepe at a cutoff grade of 1.85 g/t Au with the assumption that mined material would be trucked to the Ovacık Mill. It is becoming an industry practice to state mineral resources within a pit optimization shell. Koza has not conducted a pit optimization exercise on the Gelintepe deposit.

Table 6.6.5.2: Gelintepe Mineral Resources, at December 31, 2014

Classification	kt	g/t Au	g/t Ag	koz Au	koz Ag
Inferred	48	3.54	2.3	5	3

- Tonnages and grade are rounded to reflect approximation;
- Resources are stated at a cutoff grade of 1.85 g/t; and
- Open pit resources are contained within grade shells but are not constrained by a pit optimization shell.

6.7 Environmental

The Gelintepe Project will involve mining in two open pits and trucking to the Ovacık mill for processing. Koza received the EIA permit in September 2009. The EIA has been canceled by the Ministry. Operations have not been started yet in Gelintepe.

Koza indicates that both open pits will be backfilled to the original topography after mine closure. The site will be reforested. Hydrogeological assessments conducted to date have shown that the bottom elevations of both pits will be above the groundwater table and no groundwater impact is anticipated. According to geochemical analysis of the surface rock samples, waste rock from the site does not create ARD risk. Project environmental monitoring will be initiated during the construction stage and will be conducted for 10 years following the mine closure.

6.8 Conclusions and Recommendations

At Gelintepe, Koza has only used two very high grade CRMs for QA/QC. SRK recommends that Koza add the following control samples to its QA/QC program:

- Low grade and middle grade CRM to bracket the deposit grade range monitor analytical accuracy;
- Coarse blank samples to its QA/QC program to monitor cross contamination;
- Preparation duplicates to its QA/QC program to assess sample preparation;
- Pulp duplicates to assess analytical precision; and
- Check samples sent to a secondary laboratory to validate analyses at the primary laboratory.

Koza is using ±10% as a performance gate for RMs at the project. SRK recommends that Koza consider the following performance gates for CRMs:

- If one analysis is outside of ±2 standard deviations it is a warning;
- Two or more consecutive analyses outside of ±2 standard deviations is a failure;
- If an analysis is outside ±3 standard deviations it is a failure if ±3 standard deviations does not exceed ±10% of the mean; and
- If the ±3 standard deviations exceed ±10% of the mean, then ±5 to ±10% should be used.

Ore Research & Exploration (OREAS), who manufactures CRMs, recommends using these performance gates and has started printing this information on CRM certificates as part of a guide for use of the CRM. ALS Global uses ±3 standard deviations during analysis as a performance gate for internal CRMs (ALS Global, 2012).

7 Narlıca Project

There was no drilling at the Narlıca site in 2014. This section is unchanged from the EOY 2013 report except Section 7.6.6 and 7.6.7 where the cutoff grade calculation and the resource statement are updated with the 2014 gold price.

7.1 Property Description and Location

The Narlıca Project is located approximately 500 m northeast of Ovacık Mine. Access to Narlıca is north from the Ovacık Mine property. Narlıca is between UTM 4329500 N, 505000 E to 4327500 N, 507000 E in ED1950 Zone 35 at an elevation of approximately 280 m. Koza has extensive land holdings in the area with both operation and exploration licenses. Narlıca is under the same operation license and operation permits for gold and silver as the Ovacık Mine. This operation license number is 18201 and comprises approximately 41,515 ha. Land tenure for Narlıca and the Project location are shown in Figure 2.1.1 in Section 2.

7.2 Climate and Physiography

The Narlıca Project is located in the Ovacık District adjacent to the Ovacık Mine and experiences a typical Mediterranean climate. Discussion of the climate and physiography of the Ovacık District is found above in Section 1.1.1.

7.3 History

Eurogold held the property from 1990 to 2005. Previous work by Eurogold included soil samples, rock chip samples, drilling and geophysics surveys.

7.4 Geology

The Narlıca Project is located in the Ovacık District. Regional geology is discussed in Section 1.1.2.

Narlıca is located 500 m northeast of Ovacık Mine and is described as an epithermal vein system with associated breccias hosted in middle to lower Miocene andesite. The main vein strikes N45°W and is offset in places by northeasterly trending faults. Near the center of the main vein is a silver-rich zone referred to as Guvemtepe. This zone is bounded by northeasterly faults. Approximately 750 m south of this zone, the vein system splays and becomes discontinuous with swarms of quartz and silica structures developed to the southwest of the main system. Here the system is faulted and changes orientation to approximately N55°E. Mineralization changes to predominately silica structures with less quartz vein development. In the northeast end of the system, small discontinuous veins striking east-west have been mapped to the northeast and southwest of the main vein zone. All quartz veins and silica structures have pervasive argillic and propylitic alteration extending up to 400 m into the host rock. The system has been mapped over an area approximately 2.5 km x 1.5 km. The northwest vein system can be traced for 1.5 km and is open to the northwest and at depth. The southeast part of the system has had less exploration but appears to be open along strike and at depth. Figure 2.3.1.1 shows the local geology of the Narlıca Project within the Ovacık area.

7.5 Exploration

Koza acquired Narlıca from Eurogold in 2005 and has been expanding the exploration around the site. This included enlarging the soil sample grid, collecting rock chip samples, drilling 42 core holes and mapping the property at 1:5,000 scale. Narlıca is within the Ovacik Mine exploration budget and exploration in the deposits around the mine will be part of a TL1.5 million (US$687,000) exploration effort during 2015. Verification will be through drilling.

7.5.1 Sample Collection

Soil samples were collected over a regular grid from the B horizon and were typically 3 to 4 kg of material. Koza also collected rock chip samples. Rock chip samples were collected perpendicular to structures and to be as representative as possible of the mineralization but may vary in harder or softer material. Sample weights range from 3 to 4 kg.

7.5.2 Drilling/Sampling Procedures

The drilling at Narlıca was conducted by Koza using its standard procedures for exploration drilling. The database includes 69 HQ sized core holes (9,006 m) and 8 channels sampled on the outcrop. Forty-two of the holes were drilled by Koza between 2005 and 2008 and 27 were drilled by Normandy in 2003. The core recovery ranges between 6 and 100% with an average of 91%.

The holes are drilled to either the north-northeast or south-southwest in order to intersect the nearly vertical mineralization at a high angle. The holes are spaced at roughly 50 m apart.

The core was sampled and assayed according to the Ovacık exploration procedures and assayed at the Ovacık lab.

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.

7.5.3 Quality Assurance and Quality Control

There has been no drilling at the Narlıca Project since 2009; therefore, this section has not changed since the 2009 report. SRK notes that Bloom (2013) has suggested changes in the QA/QC program and SRK concurs with these recommendations.

Insertion of Internal controls

Koza inserts QA/QC control samples into the sample stream at approximately one blank per drillhole, RMs at a frequency of approximately one per drillhole and duplicate samples at a rate of approximately one per drillhole. These samples are inserted into the sample stream in sampling numbering sequence by the core logging geologist. The location of the control samples is noted on the sample log and in the sample database. Internal control samples have the same numbering system as the drill core samples.

Reference Materials

During the course of the drilling programs at Narlıca, Koza has used five RMs: one produced by Rock Labs based in New Zealand and four produced in-house with material from both the Ovacık Mine and Çukuralan mine. The in-house standards were developed in conjunction with ALS Chemex and were analyzed either by ALS Canada or ALS Australia. For the in-house standards and the Rock Labs standard Koza uses a performance range of ±10% of the mean. Only one standard sample was analyzed for each of OV15, KA01, and OX60, in order for these standards to be statistically meaningful, SRK recommends that more of these standards be submitted for analysis. Table 7.5.3.1 presents the expected mean, standard deviations and summaries of the analyses of the Au RMs.

Table 7.5.3.1: Results of Au RM Analyses at Narlıca

Standard	Number of Samples	Expected (ppm)		Observed (ppm)		% of Expected Mean	Number Failures	% Failure Rate
		Mean	Std Dev	Mean	Std Dev			
OV14	12	3.091	0.182	2.906	0.033	94.0	0	0.0
OV15	1	5.400	0.269	-	-	-	0	0.0
OV16	3	8.390	0.307	8.670	0.111	103.3	0	0.0
OXG60	1	1.025	0.028	-	-	-	0	0.0
KA01	1	1.000	0.067	-	-	-	0	0.0
Total	**18**						**0**	**0.0**

There are no failures in the Narlıca data and the observed means are between 94.0 and 103.3% of the expected values for Au. Of the standards and CRMs, 14 performed higher than the mean while the remaining four performed lower than the mean. SRK notes that there are limited RM analyses for a statically meaningful assessment of these standards. However, they are providing results within the performance range at this time. SRK recommends adding a low grade RM near the cutoff grade of 0.55 g/t Au for Narlıca in order bracket the grade range of mineralization. It is important that the analyses are accurate at the cutoff grade for the deposit.

Koza reviews all QA/QC during drilling programs and contacts the laboratory when analytical failures are identified. This is industry best practice.

The results indicate that the laboratory is providing accurate data.

Blanks

Sample blanks test for contamination in preparation and assaying and handling errors. Koza inserts one sample blank per drillhole using pulp blanks up until June 2012, and preparation blanks since then. A blank failure is a result greater than five times the detection limit. SRK has examined the results for gold in the blank samples and finds that there were 11 Narlıca samples submitted with zero failures. The results indicate that the preparation laboratory is performing well.

Preparation Duplicates

Preparation duplicates are created by taking a second split of the crushed sample (coarse reject) using the same method 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.

Koza sent 11 preparation duplicates to the original lab for Au analysis. A summary of the analytical results are presented in Table 7.5.3.2.

Table 7.5.3.2: Summary of Duplicate Au Analysis at Narlıca

Criteria	Number of Samples	Original>Dup	Dup>Original	Original = Dup	Within +/- 20%
All samples	11	2	6	3	7
		18	55	27	64

The duplicate results from Narlıca are from a limited database of 11 submissions. Of the four failures, two had both original and duplicate analyses that were below 0.40 g/t Au, which is below the cutoff grade of 0.55 g/t Au for the deposit. Of the two other failures there was a pair where the original analysis was 2.65 g/t Au and the duplicate was 0.59 g/t Au, and a second pair where the original was 0.10 g/t Au while the duplicate was 0.50 g/t Au, which is approaching the cutoff grade. Although this is a very small database, these two failures could be an indication that there is problem with sample preparation. SRK recommends that Koza continue to insert preparation duplicates in order to increase the database. Should there be more failures of this type, Koza should examine sample preparation procedures at Narlıca.

Pulp Duplicates

Koza does not submit pulp duplicates at this time. Pulp duplicates test the analytical reproducibility or precision of the analysis. SRK recommends that Koza add pulp duplicates to its QA/QC program.

Secondary Check Lab Analysis

Koza has not sent any Narlıca check samples in the form of pulp duplicates to a secondary laboratory as verification of the ALS Chemex analytical results. SRK recommends that Koza add this type of QA/QC samples to its program. Check samples must be analyzed at the secondary laboratory using the same method as ALS Chemex and standards must be submitted with the check samples.

7.6 Grade Estimation

The Mineral Resources were estimated by Koza in 2009 (Koza, 2009a).

7.6.1 Geological Modeling and Grade Estimation

In 2009, Koza prepared five wireframes based on a gold cutoff grade of 0.50 g/t (Figure 7.6.1.1). The wireframes strike to the north-northwest and have a steep dip. The strike length (north-northwest) is about 800 m, although the mineralization is not continuous across the entire length. The thickness of the vein varies from less than 1 m to about 10 m with an average of about 5 m.

Table 7.6.1.1 presents statistics of the drillhole samples within the wireframe.

Table 7.6.1.1: Statistics of Narlica Assays

Metal	Count	Min	Max	Mean	S.D.	CV
Au	302	0.09	49.60	2.91	5.47	1.88
Ag	296	0.00	500	14.11	43.34	3.07

7.6.2 Compositing and Capping

The drillholes were composited on 1 m lengths with breaks at the wireframe boundaries. Capping values were set at 21 g/t for Au and 110 g/t for Ag were set after inspecting probability plots of the composites. SRK did not receive the composite file and has therefore reconstructed the file using Vulcan software. The statistics of the composites are shown in Table 7.6.2.1.

Table 7.6.2.1: Statistics of Narlica Composites

Metal	Count	Min	Max	Mean	S.D.	CV
Au	292	0.14	49.60	2.87	5.16	1.80
Ag	292	0.00	500	13.29	38.63	2.91
Au_capped	292	0.14	21	2.65	3.72	1.41
Ag_capped	292	0.00	110	10.84	18.80	1.74

7.6.3 Variography

Because of the limited number of samples, it was not possible to conduct a variography study.

7.6.4 Density

The density data includes 37 samples from 9 core holes. The average value of the ore-grade material is 2.46 g/cm^3.

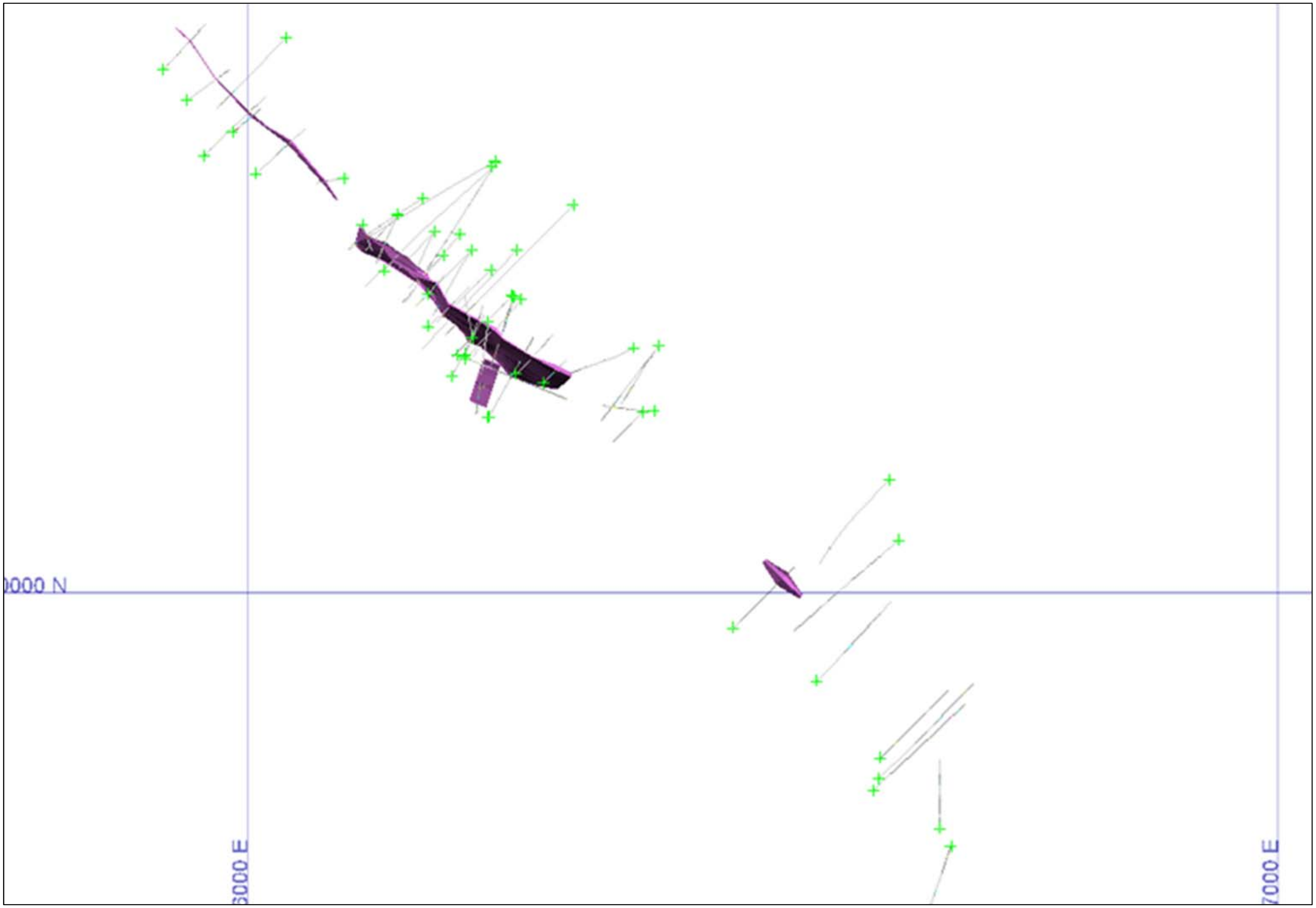


Figure 7.6.1.1: Drilling and Vein Wireframes at Narlıca, in Plan View

7.6.5 Grade Estimation

A block model was created with a block size of 5 m x 5 m x 5 m, with sub-blocking to less than 0.1 m. A minimum of three and maximum of 12 composites were used to estimate grade. ID2 was used for grade estimation, using only composites within the 3D solid to estimate grade within the vein. The search was oriented along the strike and dip of the vein, with four passes as follows:

- First: minimum of 5, maximum of 20 composites, search 50 m x 50 m x 5 m;
- Second: minimum of 5, maximum of 20 composites, search 100 m x 100 m x 10 m;
- Third: minimum of 1, maximum of 20 composites, search 250 m x 250 m x 25 m; and
- Fourth: minimum of 1, maximum of 20 composites, search 500 m x 500 m x 50 m.

Additional estimations were done with ID3 and NN approaches for resource validation. The resource was validated by visually inspecting the composites and the block grades on cross-sections and reviewing statistics for both Table 7.6.5.1. The ID2 results were compared with the ID3 and NN results as another check. The results show a good comparison between the estimated and composite grades. SRK suggests that Koza also generate swath plots as a means of block model validation.

Table 7.6.5.1: Comparison of Estimated Grades and Composite Grades

Au				Ag			
ID2	ID3	NN	Composite	ID2	ID3	NN	Composite
2.58	2.59	2.64	2.65	10.69	10.74	10.29	10.84

7.6.6 Mineral Resource Classification and Statement

The blocks were classified by Koza as Indicated if they were estimated in the first or second pass and as Inferred if estimated in the third or fourth pass. About one quarter of the resource was estimated in the third or fourth pass. SRK later re-classified blocks in the small portion of the wireframe in the southeast as Inferred because the wireframe is based on only a single drillhole.

The resources are tabulated at a cutoff grade of 0.70 g/t Au based on the open pit mining parameters shown in Table 7.6.6.1. 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 7.6.6.1: Narlıca Cutoff Grade Parameters

Prices and Costs	Units	OP
Gold Price	US$/oz	1,450
Gold Recovery	%	0.95
Gold Refining	US$/oz	3.44
Government Right	%	1
Royalty	%	3
Process Cost	US$/t	11.00
Mining Cost	US$/t	0.00
G&A Cost	US$/t	15.00
Transport Cost	US$/t	4.00
Calculated Cutoff grade	g/t	0.71
Final Cutoff grade	g/t	0.70

Source: Koza, 2014

It is becoming an industry practice to state mineral resources within a pit optimization shell. Koza has not conducted a pit optimization exercise on the Narlica deposit and SRK strongly suggests that it do so.

Table 7.6.6.2 lists the resources at Narlıca at a cutoff of 0.70/t Au based on trucking material to the Ovacık Mill.

Table 7.6.6.2: Narlıca Mineral Resources, at December 31, 2014

Classification	kt	g/t Au	g/t Ag	oz Au	oz Ag
Indicated	376.0	2.48	10.80	30,000	131,000
Inferred	140.5	3.02	11.01	14,000	50,000

- Tonnages and grade are rounded to reflect approximation;
- Resources are stated at a cutoff grade of 0.70 g/t; and
- Open pit resources are contained within grade shells but are not constrained by a pit optimization shell.

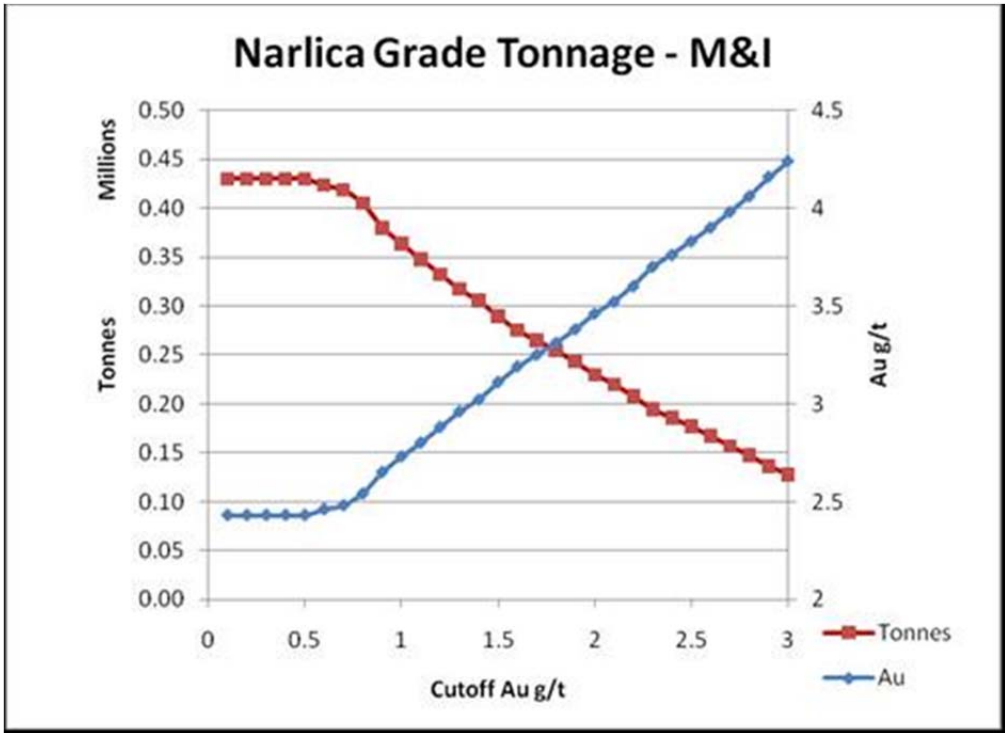
7.6.7 Mineral Resource Sensitivity

Figure 7.6.7.1 presents grade tonnage curves for Indicated and Inferred Resources.

Cutoff grades for the Narlica resource at various gold prices are shown in Table 7.6.7.1.

Table 7.6.7.1: Çukuralan Cutoff Grades vs. Gold Price

Gold Price	Cutoff Grade
1600	0.64
1550	0.66
1500	0.68
1450	0.71
1400	0.73
1350	0.76
1300	0.79
1250	0.82
1200	0.86



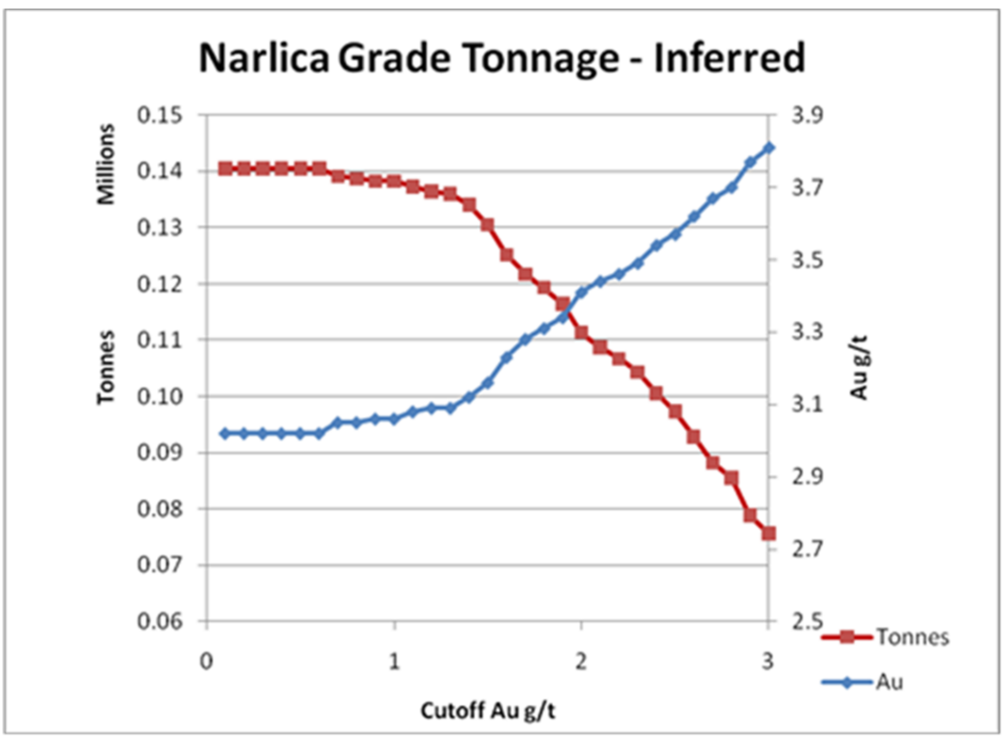


Figure 7.6.7.1: Grade Tonnage Curves for Narlıca Resource

7.7 Environmental

The Narlıca Prospect is located within the license area of Ovacık Property. The environmental characteristics and sensitivities are expected to be similar to the Ovacık mine. The EIA decision was received on August 26, 2010.

7.8 Conclusions and Recommendations

SRK recommends that Koza state resources within a pit optimization shell.

Koza is using CRMs that are twice the cutoff grade of the resource database. Koza also has a number of failures in its preparation duplicate database. SRK recommends adding a low grade CRM near the cutoff grade of 0.55 g/t Au for Narlıca in order bracket the grade range of mineralization and that Koza continue to insert preparation duplicates in order to increase the duplicate database. Should there be more failures of this in the preparation duplicates, Koza should examine sample preparation procedures at Narlıca. SRK also recommends that Koza add pulp duplicates and check samples sent to a secondary laboratory to its QA/QC program.

SRK recommends that Koza consider the following performance gates for CRMs and RMs:

- If one analysis is outside of ±2 standard deviations it is a warning;
- Two or more consecutive analyses outside of ±2 standard deviations is a failure;
- If an analysis is outside ±3 standard deviations it is a failure if ±3 standard deviations does not exceed ±10% of the mean; and
- If the ±3 standard deviations exceed ±10% of the mean, then ±5 to ±10% should be used.

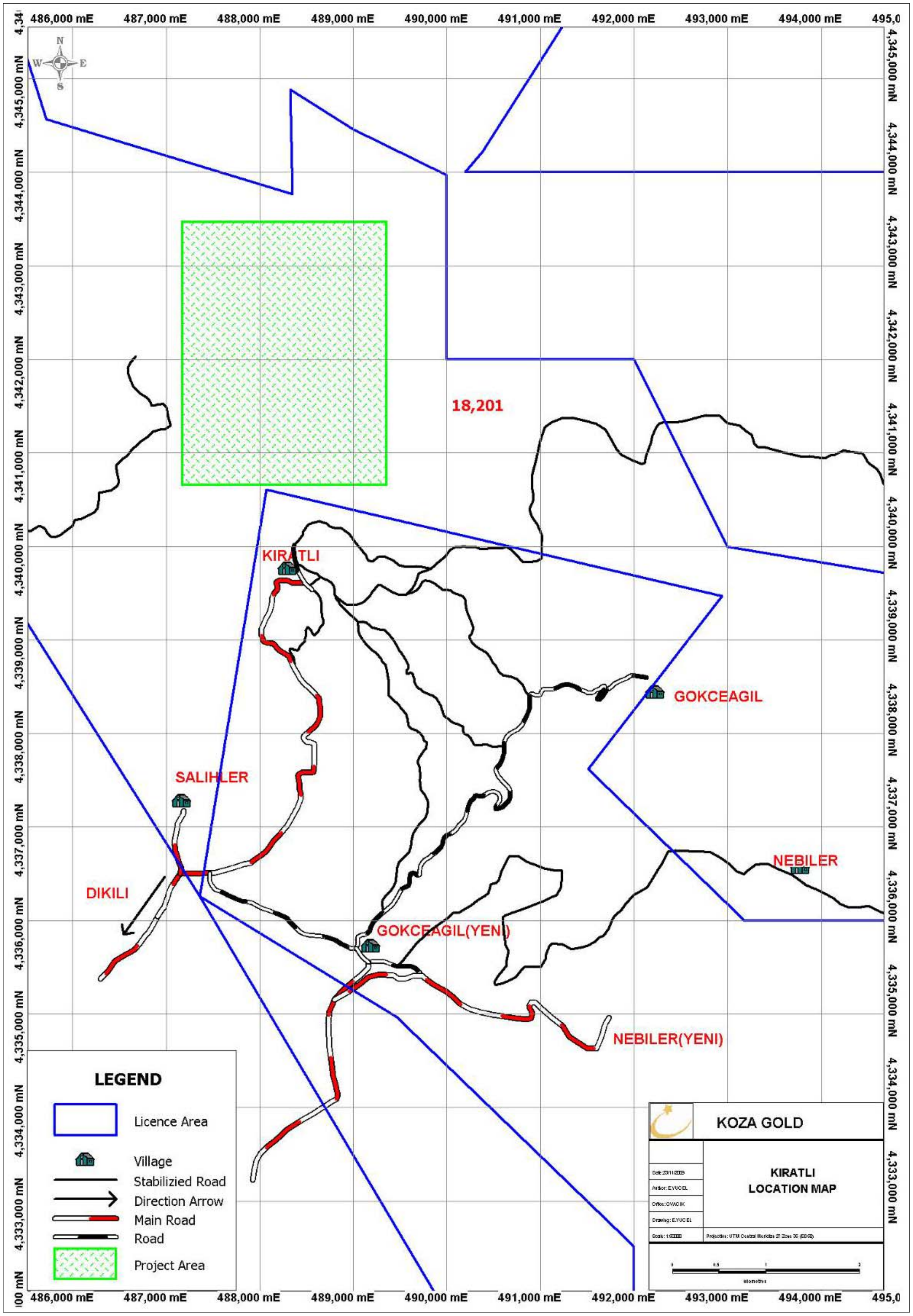
Ore Research & Exploration (OREAS), who manufactures CRMs, recommends using these performance gates and has started printing this information on CRM certificates as part of a guide for use of the CRM. ALS Global uses ±3 standard deviations during analysis as a performance gate for internal CRMs (ALS Global, 2012).

8 Kıratlı Project

There was no drilling at the Kıratlı site in 2014. This section is unchanged from the EOY 2013 report except Section 8.6.6 and 8.6.7 where the cutoff grade calculation and the resource statement are updated with the 2014 gold price.

8.1 Property Description and Location

The Kıratlı Project is located 20 km northwest of the Ovacık Mine at UTM coordinates 4343000 N, 487500 E to 4341750 N, 489000 E ED1950 Zone 35, at an elevation of 120 to 480 m. The Project is accessed from the Ovacık Mine by following highway D550 northwest for approximately 25 km and taking a mountain road for an additional 10 km to the northeast. Kıratlı is included in the Ovacık operation license 18201, which contains approximately 41,515 ha. The land tenure for Kıratlı is shown in Figure 8.1.1.



Source: Koza, 2012 GIS

Figure 8.1.1: Kıratlı Location Map

8.2 Climate and Physiography

The Kıratlı Project is located in the Ovacık District and experiences a typical Mediterranean climate. Discussion of the climate and physiography of the Ovacık District is found above in Section 1.1.1.

8.3 History

The Kıratlı exploration property was previously held by Normandy. In 1995, Normandy performed regional sampling in the collecting 28 stream samples five of which were analyzed using BLEG analysis.

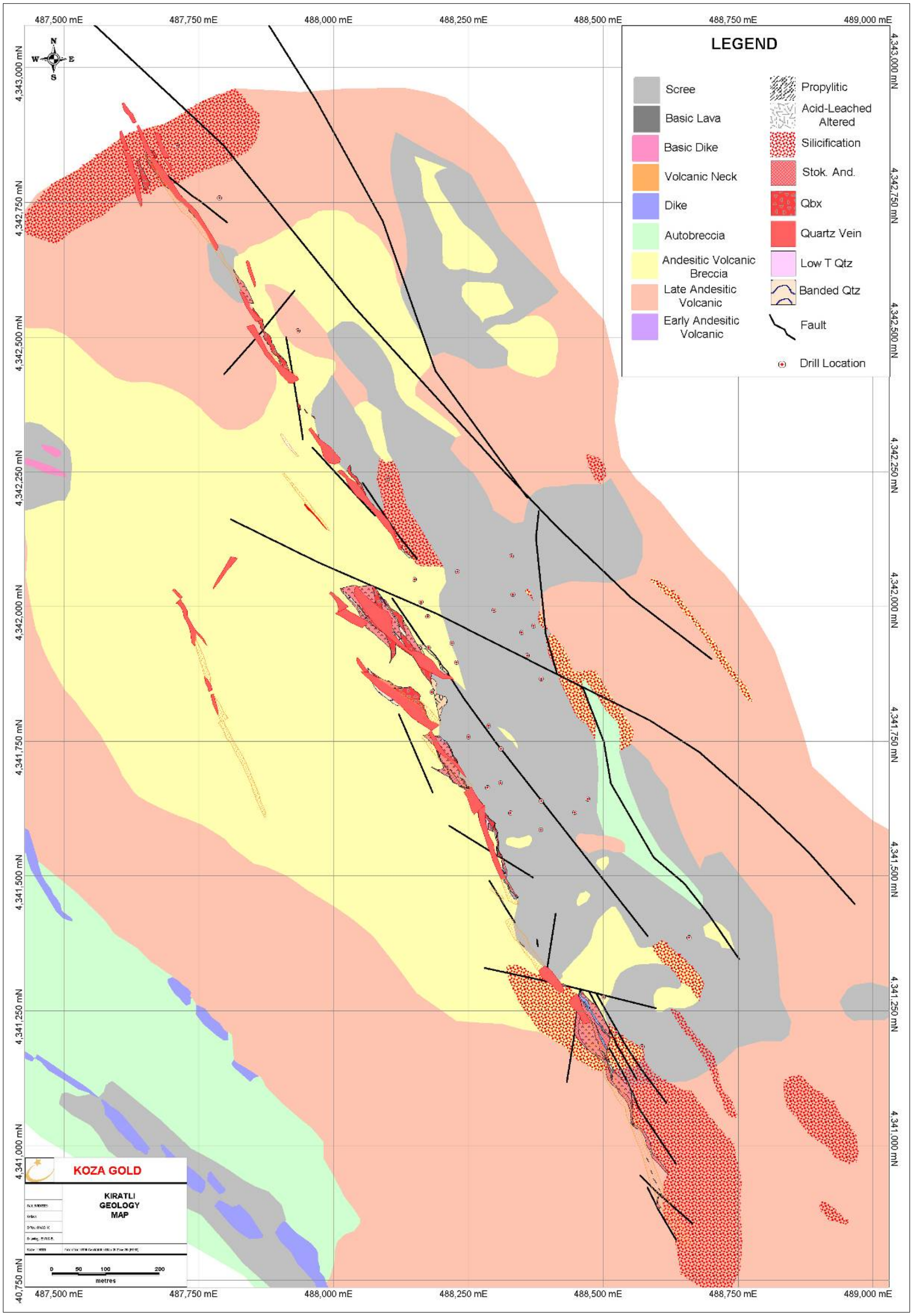
8.4 Geology

Kıratlı is located in the Ovacık District. Regional geology of the Ovacık District is discussed in Section 1.1.2.

The Kıratlı Project is centered on a highly visible quartz vein that can be traced for 2.5 km. The average width of this vein is 10 m with vertical exposures up to 20 m. A second vein has been mapped approximately 280 m to the southwest of the primary vein and has been traced over 450 m. This second vein has apparent widths up to 13 m. Although the Kıratlı vein system is highly visible, prior to Koza's acquisition the vein was relatively unexplored.

Kıratlı is a high-angle, low sulfidation, epithermal gold vein system with associated breccias hosted by Miocene andesite. The system strikes approximately N30°W and dips steeply. Quartz within the vein system shows episodic deposition with mapable zones of banding, brecciation and drusy quartz overgrowths. Chalcedonic and opaline quartz are the most common quartz types, and both high and low temperature quartz has been identified in the vein system. Carbonate replacement has also been observed. The vein system also includes zones of silicified andesite breccia near the selvages of the vein system. Sulfide minerals include pyrite and chalcopyrite and alteration consists of intense silicification in the host rock adjacent to the vein system. Based on these vein textures and geochemistry, the vein outcrop is interpreted to be high in the hydrothermal system suggesting most of the hydrothermal system is still present.

Koza has recently identified an epithermal vein to the southwest of the main vein near the contact between the early stage porphyry andesitic unit and hornfels. This vein strikes E-W and can be traced approximately 450 m. The average width of this vein is 13 m. Gangue mineralization includes calcite, ankerite, manganese oxide and quartz and mineralization is characterized by banded and replacement textures. Immediately southwest of this vein, alteration and mineralization changes to a high sulfidation epithermal system. This includes the appearance of silica and acid leached caps, broad zones of argillic alteration, pyrite halos, silica sulfide alteration, vuggy silica, all of which are typical of high sulfidation systems. Geology is shown in Figure 8.4.1.



Source: Koza, 2012 GIS

Figure 8.4.1: Kıratlı Geology Map

8.5 Exploration

Koza is the first company to do any significant exploration work at this site. Kıratlı was acquired by Koza in 2006 and subsequent work has included soil sampling, channel sampling of the exposed vein, mapping at a 1:5000 scale and PIMA mapping. Since 2008, exploration has been limited to field mapping and surface sampling pending acquisition of drilling permits. Kiratli is within the Ovacik Mine exploration budget and exploration in the deposits around the mine will be part of a TL1.5 million (US$687,000) exploration effort in 2015. Verification will be through drilling.

8.5.1 Sample Collection

Soil samples were collected over a regular grid spacing. Samples were collected from the B horizon and typically 3 to 4 kg of sample was collected. Rock channel samples are collected using a gas powered concrete saw with a diamond blade. They are typically 1 m long but vary in depth and width depending on field conditions and lithological contacts. Widths range from 5 to 15 cm and depths range from 15 to 20 cm. Sample weights range from 2 to 3 kg. Samples may be shorter or slightly longer than 1 m to accommodate changes in lithology.

8.5.2 Drilling/Sampling Procedures

Koza drilled a total of 37 HQ sized core holes at Kıratlı in 2007 and 2008. The drilling and sampling procedures conform to Koza's standard exploration procedures. The samples were assayed by FA and ICP at ALS Canada and later by FA at ALS Romania. Sample preparation was conducted at the ALS preparation laboratory located in İzmir.

The drilling is located on section lines spaced about 50 m apart and oriented to the west-west-southwest. There are one or two holes on each section line. Drill recovery ranges between 0 and 100% with an average of 95%.

8.5.3 Sample Preparation and Analysis

Core and exploration samples are held in the custody of Koza until it is shipped to the laboratory for analysis in a locked, in a locked core logging facility or at the nearest mine site in a locked building. Core samples are either delivered to the laboratory by Koza personnel or shipped via commercial trucking. This is industry best practice.

Samples were prepared at ALS İzmir. All early analysis was conducted at the ALS Vancouver laboratory but later was only ICP multi-element analysis was completed at ALS Vancouver. Later in the program, FA was conducted at ALS Romania. ALS Vancouver and ALS Romania have ISO 17025 accreditation for specific analytical methods through the Standards Council of Canada. ALS Vancouver's accreditation is valid through May 18, 2017 and ALS Romania's is valid through March 27, 2016.

Once the samples arrived at the laboratory, they were bar coded and entered into the Laboratory Information Management System (LIMS). All samples were dried to a maximum temperature of 60°C in order to avoid or limit volatilization of elements such as mercury (ALS code DRY-22). Soil samples were screened to -180 micron (80 mesh) to remove organic matter and large particles. Soil samples were then analyzed.

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. In this analysis, a 1 g sample is digested using aqua regia and analyzed 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. Koza also analyzed 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). This analytical method is appropriate for the deposit type and level of investigation. Table 8.5.3.1 presents the analytes with upper and lower detection limits for ALS ME-MS41 and Au-ICP22.

Table 8.5.3.1: 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

After drying using ALS code DRY-22, rock chip and core samples were crushed to 70% passing -2 mm (ALS code CRU-31) and a 1,000 g split was collected using a riffle splitter (ALS code SPL-21). The 1,000 g split was pulverized to 85% passing 75 microns (ALS code PUL-32). Koza requests a larger split pulverized to help mitigate the nugget affect.

Rock and core samples were analyzed using ALS code ME-ICP61, a 33 element package with trace level sensitivity. A 1 g sample is put into solution using a four acid digestion and the sample is analyzed using ICP-AES. Gold was analyzed using ALS code Au-AA24, which is gold by FA using a 50 g charge with an Atomic Absorption Spectroscopy (AAS) finish. The samples were also analyzed for mercury using Hg-CV41. By this method, mercury content is determined using aqua regia digestion and cold vapor AAS. Table 8.5.3.2 presents the analytes with upper and lower detection limits for ALS ME-ICP61, Hg-CV41 and Au-AA24.

Table 8.5.3.2: Analytes and Upper and Lower Detection Limits for ALS Codes ME-ICP61, Hg-CV41 and Au-AA24 in ppm Unless Otherwise Noted

Method	Analyte	Range	Method	Analyte	Range	Method	Analyte	Range
Au-AA24	Au	0.005-10	ME-ICP61	Cu	1-10,000	ME-ICP61	S	0.01-10%
Hg-CV41	Hg	0.01-100	ME-ICP61	Fe	0.01-50%	ME-ICP61	Sb	5-10,000
ME-ICP61	Ag	0.5-100	ME-ICP61	Ga	10-10,000	ME-ICP61	Sc	1-10,000
ME-ICP61	Al	0.01-50%	ME-ICP61	K	0.01-10%	ME-ICP61	Sr	1-10,000
ME-ICP61	As	5-10,000	ME-ICP61	La	10-10,000	ME-ICP61	Th	20-10,000
ME-ICP61	Ba	10-10,000	ME-ICP61	Mg	0.01-50%	ME-ICP61	Ti	0.01-10%
ME-ICP61	Be	0.5-1,000	ME-ICP61	Mn	5-100,000	ME-ICP61	Tl	10-10,000
ME-ICP61	Bi	2-10,000	ME-ICP61	Mo	1-10,000	ME-ICP61	U	10-10,000
ME-ICP61	Ca	0.01-50%	ME-ICP61	Na	0.01-10%	ME-ICP61	V	1-10,000
ME-ICP61	Cd	0.05-1,000	ME-ICP61	Ni	1-10,000	ME-ICP61	W	10-10,000
ME-ICP61	Co	1-10,000	ME-ICP61	P	10-10,000	ME-ICP61	Zn	2-10,000
ME-ICP61	Cr	1-10,000	ME-ICP61	Pb	2-10,000			

Source: ALS Global, 2014

8.5.4 Quality Assessment and Quality Control

There has been no drilling at the Kıratlı Project since 2008; therefore, this section has not changed since the 2009 report. SRK notes that Bloom (2013) has suggested changes in the QAQC program and SRK concurs with these recommendations.

Insertion of Internal controls

Koza inserts QA/QC control samples into the sample stream at approximately one blank per drillhole, RMs at a frequency of approximately one per drillhole and duplicate samples at a rate of one to two per drillhole. These samples are inserted into the sample stream in sampling numbering sequence by the core logging geologist. The location of the control samples is noted on the sample log and in the sample database. Internal control samples have the same numbering system as the drill core samples.

Reference Materials

Since 2007, three different RMs have been used at Kıratlı: all produced in-house with material from both the Ovacık Mine and Çukuralan mine. The in-house standards were developed in conjunction with ALS Chemex and were analyzed either by ALS Canada or ALS Australia. Koza generally uses ±2 standard deviations or ±10% of the observed mean to define a performance range for its RMs and reference materials and analyses falling outside of this range are considered failures. General industry practice is to use the following method for determining failures:

- If one analysis is outside of ±2 standard deviations it is a warning;
- If two or more consecutive analyses are outside of ±2 standard deviations is a failure; and
- If an analysis is outside ±3 standard deviations it is a failure.

For the in-house standards, Koza uses a performance range of ±10% of the mean. Table 8.5.4.1 presents the expected mean, standard deviations and summaries of the analyses of the Au RMs.

Table 8.5.4.1: Results of Au RM Analyses at Kıratlı

Standard	Number of Samples	Expected (ppm)		Observed (ppm)		% of Expected	Number Failures	% Failure Rate
		Mean	Std Dev	Mean	Std Dev			
OV14	29	3.091	0.182	3.040	0.210	98.3	2	6.9
OV15	23	5.400	0.269	5.230	0.194	96.9	1	4.3
OV16	8	8.390	0.307	8.679	0.139	103.4	0	0.0
Total	**60**						**3**	**5.0**

There is a 5.0% failure rate in the Kıratlı data and the observed means are between 96.9 and 103.4% of the expected values for Au. Of the standards and RMs, 23 performed higher than the mean while the remaining 37 performed lower than the mean. The RMs used bracket only higher grade analyses. SRK recommends that Koza add a low grade RM to the standards used at Kıratlı.

Koza reviews all QA/QC during drilling programs and contacts the laboratory when analytical failures are identified. This is industry best practice.

The results indicate that the laboratory is providing accurate data.

Blanks

Sample blanks test for contamination in preparation and assaying and handling errors. Koza inserts one sample blank per drillhole using pulp blanks up until June 2012, and preparation blanks since then. A blank failure is a result greater than five times the detection limit. SRK has examined the results for gold in the blank samples and finds that there were 32 Kıratlı samples submitted with two failures, or a 6% failure rate. The results indicate that the preparation laboratory is generally providing samples without cross contamination. However, cross contamination issues must be addressed with follow-up with the preparation laboratory to correct the problem. Koza has addressed these two failures by follow-up with the preparation laboratory.

Preparation Duplicates

Preparation duplicates are created by taking a second split of the crushed sample (coarse reject) using the same method 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.

Koza sent 34 preparation duplicates to the original lab for Au analysis. . A summary of the analytical results are presented in Table 8.5.4.2.

Table 8.5.4.2: Summary of Duplicate Au Analysis at Kıratlı

Criteria	Number of Samples	Original>Dup	Dup>Original	Original = Dup	Within +/- 20%
All samples	34	8	8	18	12
		24	24	53	35

All of the pairs except for one were for samples with analytical results less than 0.45 g/t Au, which is less than the cutoff grade of the resource estimate of 0.55 g/t Au. The data does not test the precision of the laboratory within the resource range. SRK recommends that Koza submit samples with the resource range of the deposit to determine if the laboratory can provide reproducible results in the range of resource grades.

Although the bulk of the data is at low grade, the results appear to indicate that Kıratlı preparation duplicates have moderately low reproducibility. This suggests that sample preparation may need to be investigated to determine if an adequate volume of material is being submitted for analysis and that the sample is being pulverized appropriately. SRK recommends that Koza submit more samples in the resource grade range and if the data continue to show low reproducibility then Koza should investigate the reproducibility by first examining homogenization procedures and then possibly adjusting the volume of material submitted for pulverization.

Pulp Duplicates

Koza does not submit pulp duplicates at this time. Pulp duplicates test the analytical reproducibility or precision of the analysis. Many commercial laboratories routinely run pulp duplicates as part of the internal laboratory QA/QC. It is industry practice to either request the ALS Chemex pulp duplicate data or if Koza prefers request that ALS Chemex create a pulp duplicate of selected intervals. SRK recommends that Koza add assessment of pulp duplicates to its QA/QC program to monitor laboratory precision during its next drilling program.

Secondary Check Lab Analysis

Koza has not sent any Kıratlı check samples in the form of pulp duplicates to a secondary laboratory as verification of the ALS Chemex analytical results. SRK recommends that Koza add this type of QA/QC samples to its program. Check samples must be analyzed at the secondary laboratory using the same method as ALS Chemex and standards must be submitted with the check samples.

8.6 Mineral Resources

The Mineral Resources were estimated by Koza in 2009 (Koza, 2009b).

8.6.1 Geological Modeling and Grade Estimation

In 2009, Koza constructed seven wireframe solids of the veins based on a cutoff grade of 0.50 g/t Au and a minimum thickness of 1 m (Figure 8.6.1.1). The wireframes are oriented, striking to the north-northwest and cover an area of 1,000 m northwest-southeast, 250 m northeast-southwest and 330 m vertically. The drilling and the wireframes are not continuous across the 1000 m extent northwest-southeast. The thickness of the veins varies from less than 1 m to over 5 m with an average between 2 and 3 m.

Statistics of the samples within the wireframe are given in Table 8.5.2.1.

Table 8.5.2.1: Statistics of Kıratlı Assays

Metal	Count	Min	Max	Mean	S.D.	CV
Au	155	0.03	27.20	2.03	3.49	1.72
Ag	155	0.00	362	36.31	47.73	1.31

8.6.2 Compositing and Capping

Koza did not composite the samples, but used the original sample length assays for resource estimation with length weighting. There are 155 samples within the wireframe; the length of the samples ranges from 0.4 to 1.7 with an average length of 1. SRK recommends that Koza composite the drillhole samples at a standard length.

Koza did not cap grades prior to grade estimation and SRK suggests that capping gold at a grade of 12 g/t (8 samples) and silver at a grade of 96 g/t (14 samples) should be utilized in further studies.

8.6.3 Variography

Because of the limited number of samples, it was not possible to conduct a variography study.

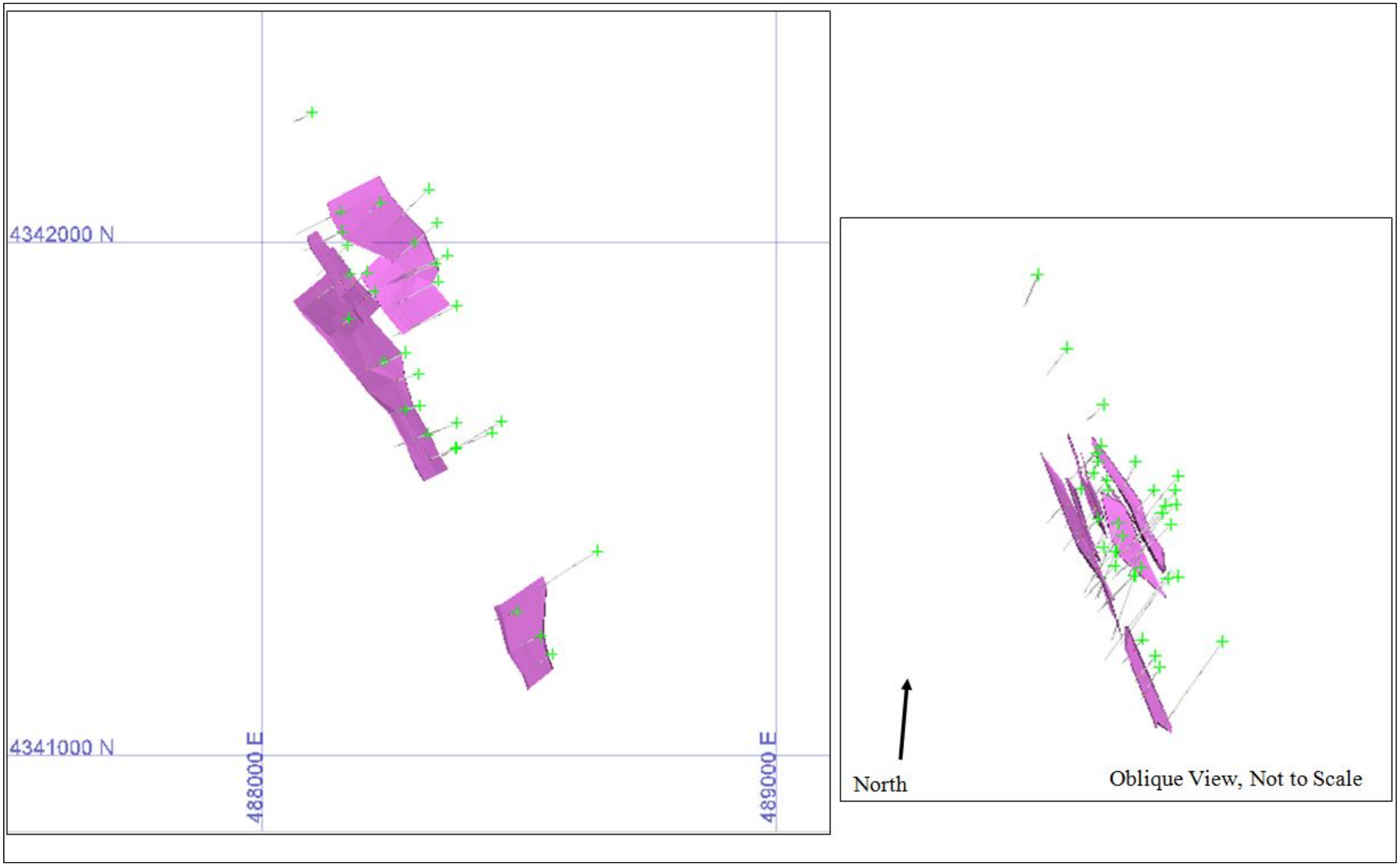


Figure 8.6.1.1: Drilling and Veins at Kıratlı in Plan View and Oblique View Looking NNE

8.6.4 Density

A density value of 2.40 g/cm^3 was used in the resource estimation based on core samples taken from 37 drillholes.

8.6.5 Grade Estimation

A block model was created with a block size of 5 m x 5 m x 2 m, with sub-blocking to less than 0.1 m.

The estimation was done in four passes with the search ellipsoid oriented parallel to the strike and dip of the vein and sample selection as follows:

- First Pass: minimum of 5, maximum of 20 composites, search 50 m x 50 m x 5 m;
- Second Pass: minimum of 5, maximum of 20 composites, search 100 m x 100 m x 10 m;
- Third Pass: minimum of 1, maximum of 20 composites, search 250 m x 250 m x 25 m; and
- Fourth Pass: minimum of 1, maximum of 20 composites, search 500 m x 500 m x 50 m.

Only samples within the wireframe were used for estimation.

The resource was validated by visually inspecting the drillholes and the block grades on cross-sections and reviewing statistics for both. In addition, estimations were done using ID3 and NN methodologies and the results compared to the ID2 results. A comparison of block grades and composite grades are shown in Table 8.6.5.1. The results are very close to the composite grades. SRK suggests that Koza also produce swath plots as a form of validation.

Table 8.6.5.1: Comparison of Estimated Grades and Composite Grades

Au				Ag			
ID2	ID3	NN	Composite	ID2	ID3	NN	Composite
2.06	1.05	2.00	2.03	36.83	36.56	36.47	36.31

8.6.6 Mineral Resource Classification and Statement

All blocks were classified as Inferred based on the low number of drillholes.

The resources are tabulated at a cutoff grade of 0.75 g/t Au based on the open pit mining parameters shown in Table 8.6.6.1. 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 8.6.6.1: Kıratlı Cutoff Grade Parameters

Prices and Costs	Units	Open Pit
Gold Price	US$/oz	1,450
Gold Recovery	%	0.94
Gold Refining	US$/oz	3.44
Government Right	%	1
Royalty	%	3
Process Cost	US$/t	11.00
Mining Cost	US$/t	0.00
G&A Cost	US$/t	15.00
Transport Cost	US$/t	5.33
Calculated Cutoff grade	g/t	0.75
Final Cutoff grade	g/t	0.75

Source: Koza, 2014

It is becoming an industry practice to state mineral resources within a pit optimization shell. Koza has not conducted a pit optimization exercise on the Kıratlı deposit and SRK highly recommends that it do so.

Table 8.6.6.2: Kıratlı Mineral Resources, at December 31, 2014

Classification	kt	g/t Au	g/t Ag	koz Au	koz Ag
Inferred	1,786	2.32	38.6	133	2,216

- Tonnages and grade are rounded to reflect approximation;
- Resources are stated at a cutoff grade of 0.75 g/t; and
- Open pit resources are contained within grade shells but are not constrained by a pit optimization shell.

8.6.7 Mineral Resource Sensitivity

Figure 8.6.7.1 presents grade tonnage curves for the Inferred Resources.

Cutoff grades for the Kıratlı resource at various gold prices are shown in Table 8.6.7.1.

Table 8.6.7.1: Kıratlı Cutoff Grades vs. Gold Price

Gold Price	Grade
1600	0.68
1550	0.70
1500	0.72
1450	0.75
1400	0.77
1350	0.80
1300	0.83
1250	0.87
1200	0.90

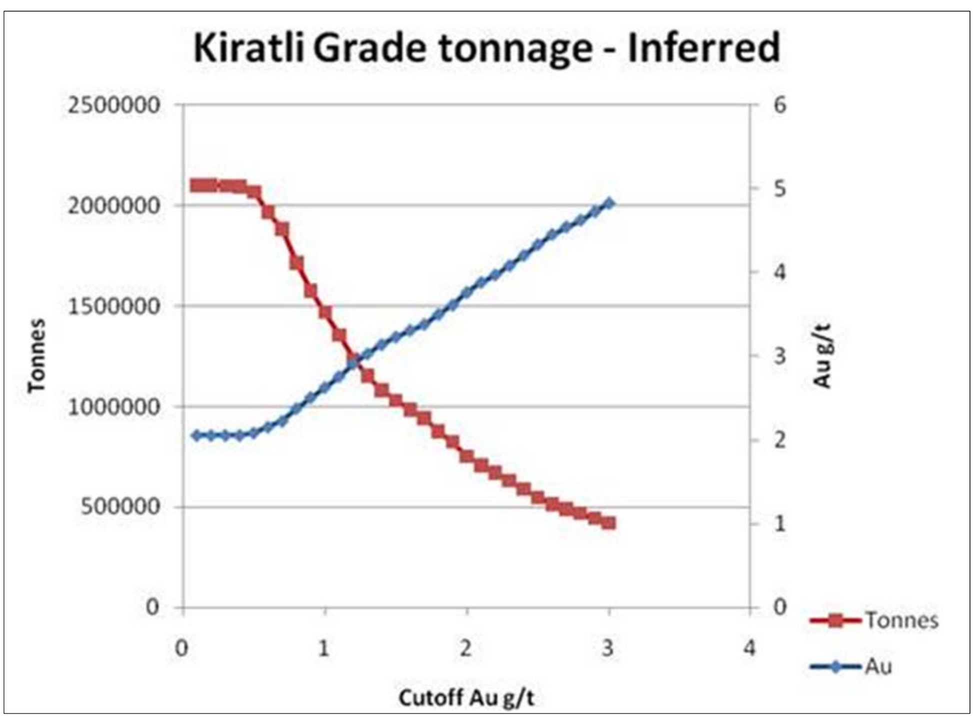


Figure 8.6.7.1: Grade Tonnage Curves for Kıratlı Resource

8.7 Environment

The Kıratlı Prospect is located within the license area of Ovacık Property. The environmental characteristics and sensitivities are expected to be similar to the Ovacık mine. There are no environmental studies in this area.

8.8 Conclusions and Recommendations

SRK recommends that future resource estimations investigate compositing length and capping values for gold and silver. SRK also recommends that Koza state the resource within a pit optimization shell.

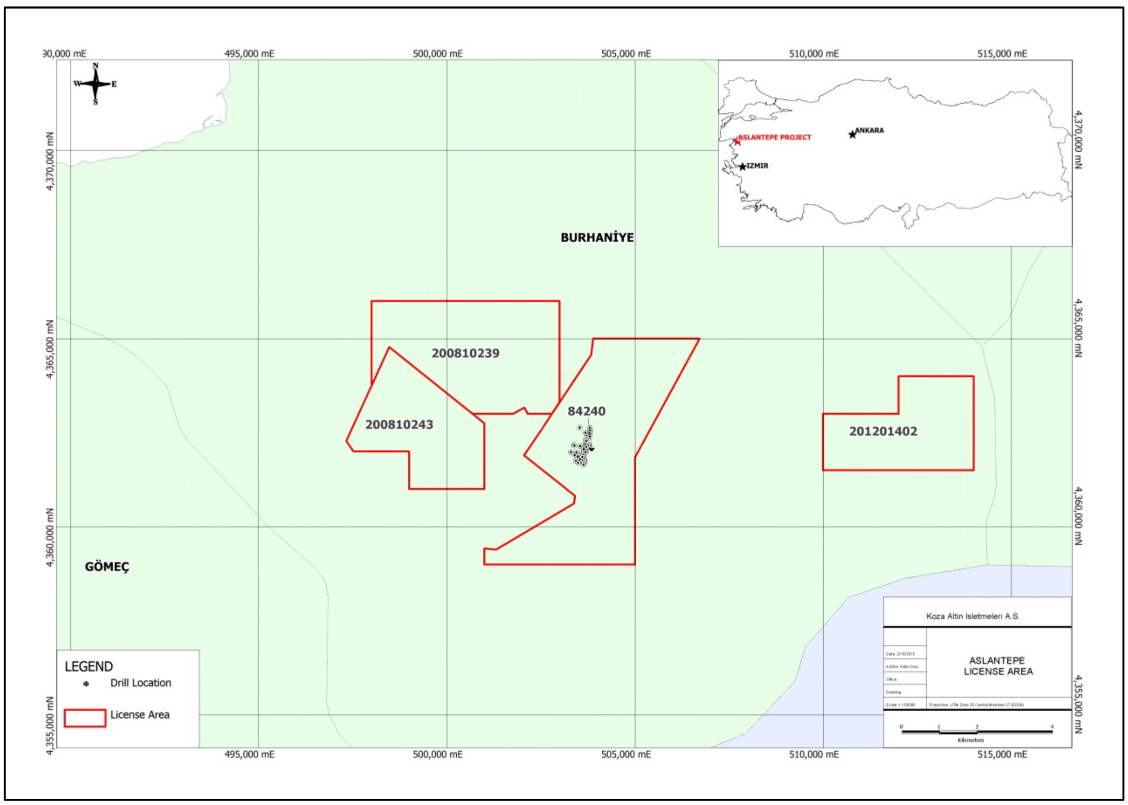
Koza monitors its QA/QC on an ongoing basis during drilling and addresses failures as they occur. SRK observed that Koza uses only one high grade RM at Kıratlı. SRK also observed that there was a lack of precision in coarse duplicates. SRK recommends that Koza add a low grade RM to the standards used at Kıratlı, and that Koza submit more coarse duplicates to increase the coarse duplicate database. If the database continues to show low reproducibility, then Koza should investigate the reproducibility by first examining homogenization procedures and then possibly adjusting the volume of material submitted for pulverization. SRK also recommends that Koza submit samples with the resource range of the deposit to determine if the laboratory can provide reproducible results in the range of resource grades. Additions to the QA/QC program also should include pulp duplicates to test analytical precision and check samples sent to a secondary laboratory to test analysis of the primary laboratory.

9 Aslantepe Project

There was no drilling at the Aslantepe site in 2014. This section is unchanged from the EOY 2013 report except Section 9.6.7 and 9.6.8 where the cutoff grade calculation and the resource statement are updated with the 2014 gold price.

9.1 Property Description and Location

The Aslantepe Project is located in the Ovacık District mining district approximately 12 km south-southeast Burhaniye and approximately 35 km north of the Ovacık Mine. Burhaniye is located along state road D550. The Project is accessed from Burhaniye by following village roads south-southeast for approximately 15 km to the village of Avunduk. The Project is located immediately southeast of the village, within operation license 84240 and exploration licenses 200810239, 200810243 and, 201201402 totaling approximately 4,485 ha. Land tenure is shown in Figure 13.1.1. Aslantepe is between UTM coordinates 4366000 N, 498000 E, 4365000 N, 506700 E ED50 Zone 35 at approximately 400 to 620 m elevation (Figure 9.1.1).



Source: Koza, 2015 GIS

Figure 9.1.1: Land Tenure of the Aslantepe Project

9.2 Climate and Physiography

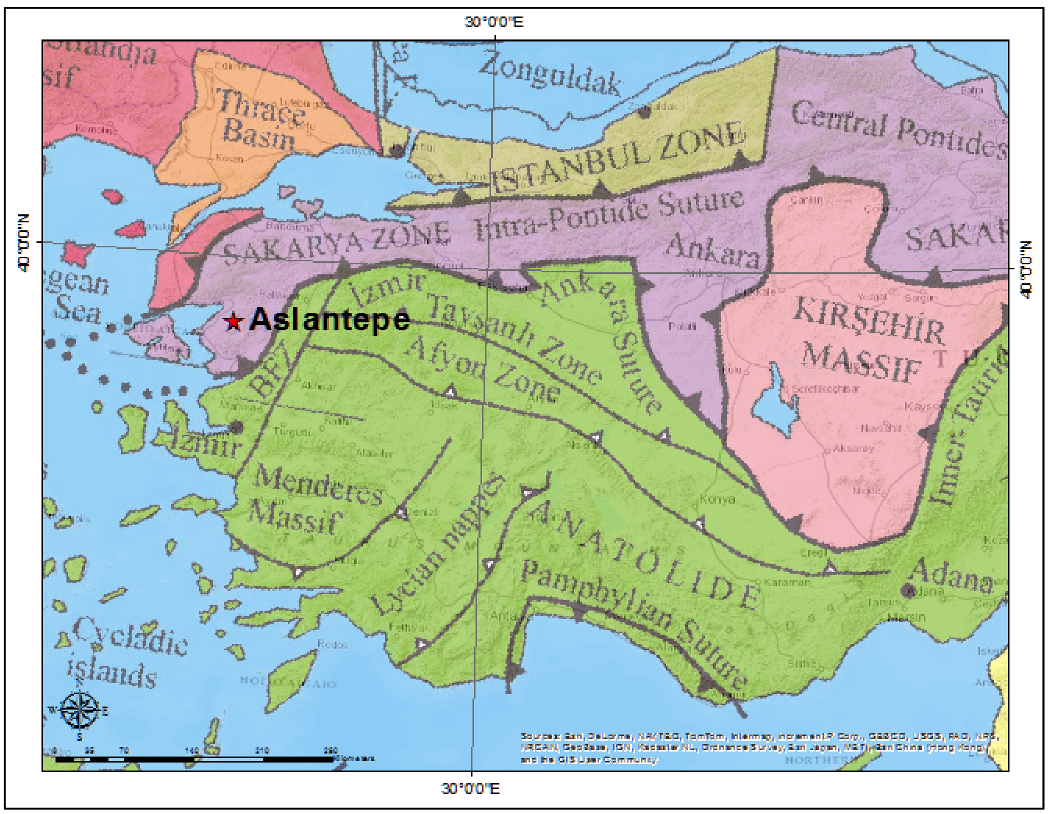
The Aslantepe Project is located in the Ovacık District and experiences a typical Mediterranean climate. Discussion of the climate and physiography of the Ovacık District is found in Section 2.1.

9.3 History

The Aslantepe Project was previously held by Newmont, Eurogold and Tüprag (Eldorado) between 2001 and 2003. Koza acquired the license through auction in 2007. Newmont collected five rock chip samples and Eurogold collected 31 Bulk Leach Extractable Gold (BLEG) samples, 57 stream sediment samples and 40 rock chip channel samples.

9.4 Geology

The Aslantepe Project is in the Ovacık District, located in the Western Anatolian Extensional Tectonic Province in a zone of low and high sulfidation epithermal deposits. This zone extends from north central Turkey to the Aegean Sea, and straddles the İzmir-Ankara Suture. This suture formed in the Cretaceous age as a result of collision and subsequent subduction of the Anatolide-Tauride block beneath the Sakarya Terrane during the closure of the Tethyan Sea. This was followed by two periods of rift related extension caused by a change in plate motion and resulted in the development of NNE-SSW and NE-SW trending grabens (Yilmaz, 2002; Okay et al., 2004; Okay, 2008). Figure 9.4.1 shows Aslantepe relative to the İzmir-Ankara Suture in the Sakarya Terrane of Western Turkey.



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

Figure 9.4.1: Geology of the Aslantepe Project

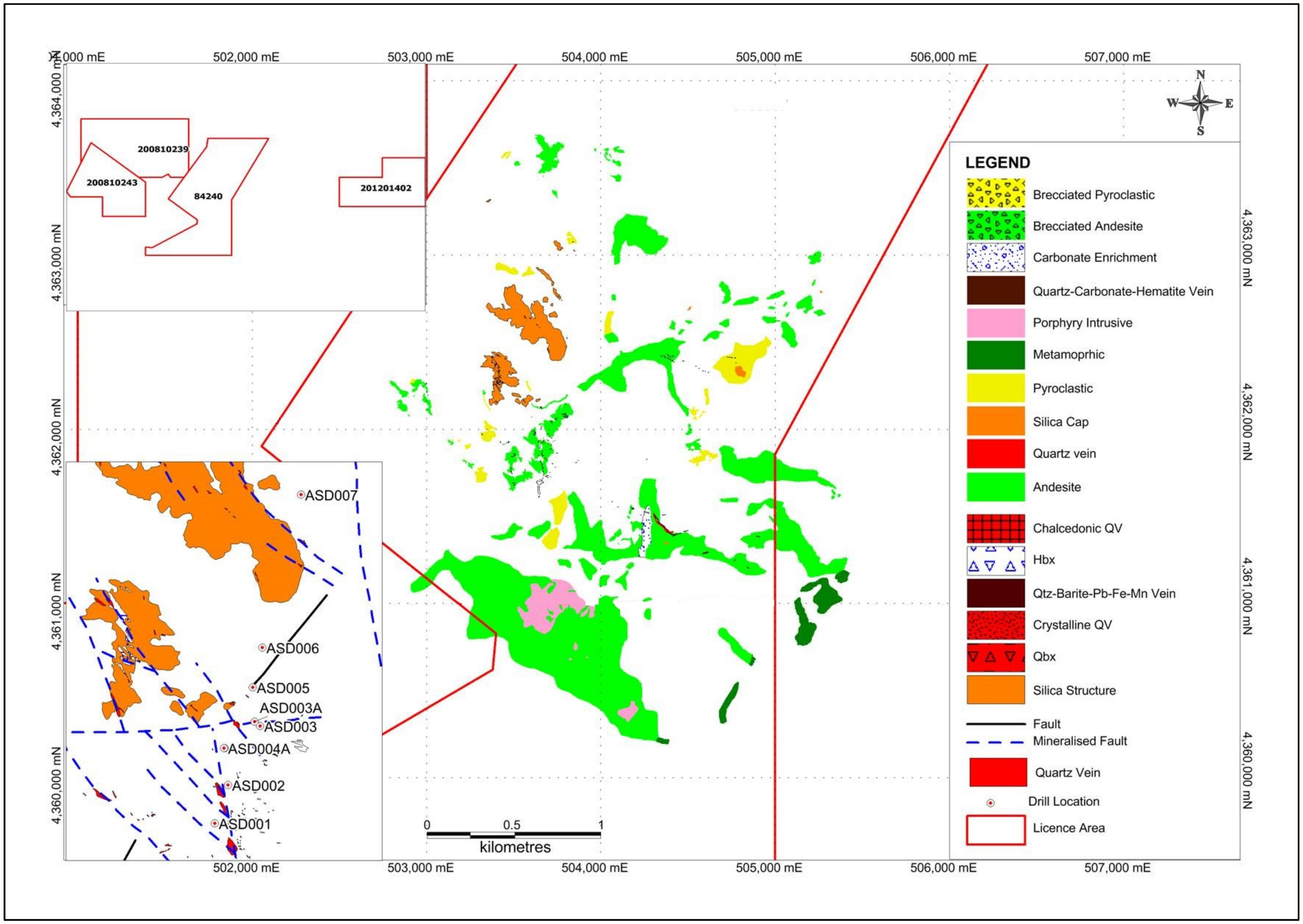
The Ovacık District is located in the Sakarya Terrane, immediately north of the İzmir-Ankara Suture Zone. The regional geology of the Ovacık District is described by Koza as an area underlain by the Triassic-age Karakaya Metamorphic Complex represented locally by the Kinik Formation. The metamorphic complex was intruded by the Kozak Magmatic Complex during the Oligocene and Miocene. In places, both the Kozak Magmatic and Karakaya Metamorphic complexes have been overlain by Miocene age volcanic rocks and Quaternary alluvium (Kara, 2004).

Deposits within the Ovacık District are commonly associated with Paleogene and Neogene-age volcanism and Upper Mesozoic to Tertiary-age intrusive events (Yilmaz, 2002; Okay et al., 2004; Okay, 2008). Aslantepe mineralization is associated with the intrusion of the early Miocene and late Oligocene age Kozak Magmatic Complex and is hosted by middle Miocene age volcanic rocks. These rocks represent the capping sequence in the area. Metamorphic rocks at the Project are primarily limestone, schist and metavolcanic units of the Karakaya Metamorphic Complex.

Mineralization at the Aslantepe Project includes a quartz vein zone hosted by Miocene volcanic units that are primarily andesitic in composition. Quartz veins vary in orientation and length. However, the primary structure strikes north-south and contains a silicified zone with an approximate 750 m strike length that ranges from 10 to 70 m in width. The structure dips approximately 75º NE. Quartz veins at Aslantepe are characterized by crystalline, saccharoidal, milky, chalcedonic, drusy, cocks comb and replacement textures, which are consistent with low sulfidation epithermal mineralization. Brecciation is also found within the vein zone. The silicified zone is offset by northwest and northeast faulting.

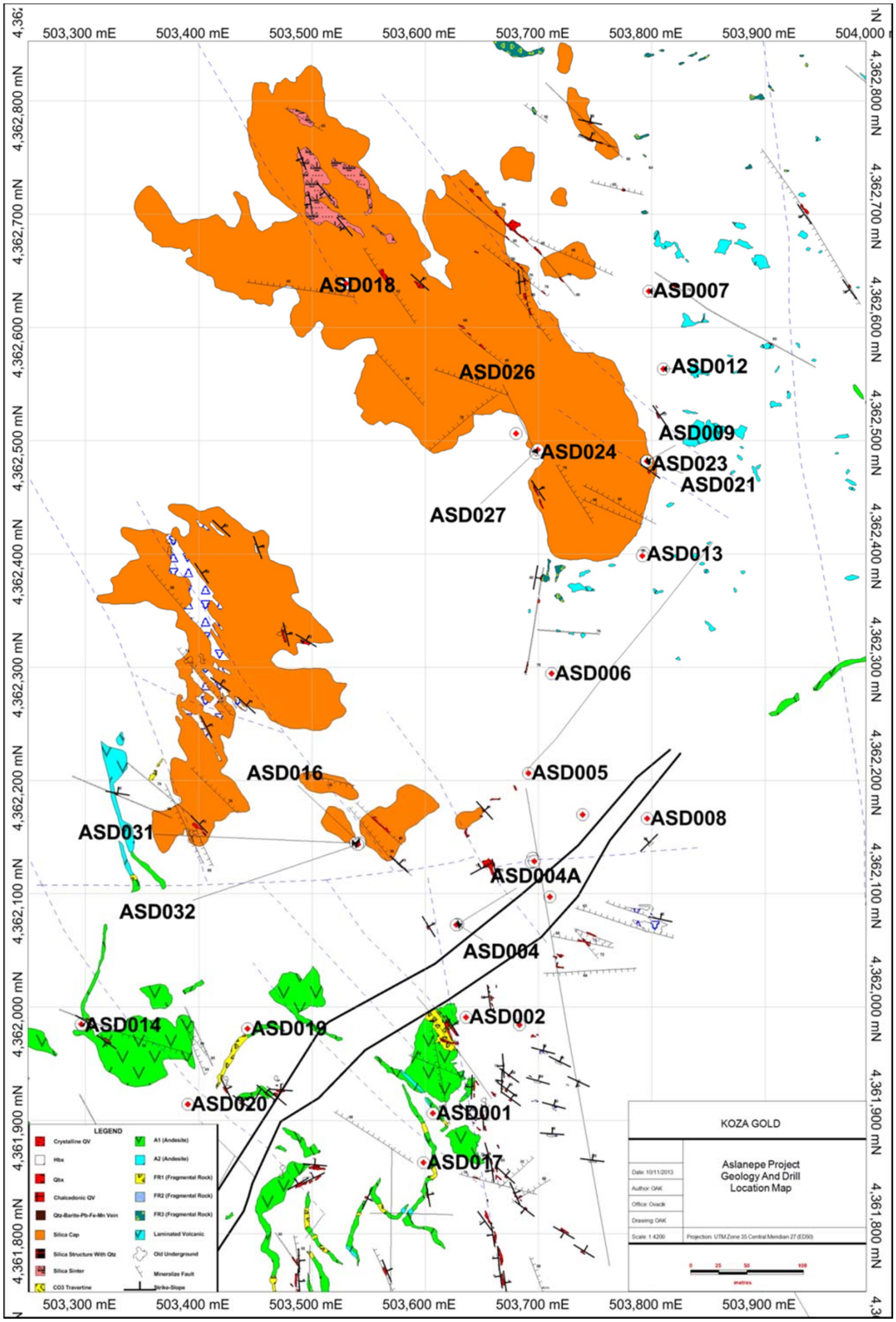
In 2013, Koza identified siliceous sinter float in the Project area during field mapping. Samples collected from the float material have anomalous As, Hg, Sb and Ag. Siliceous sinter with these pathfinder elements is often found at the top of low sulfidation epithermal mineralization. Typical zoning in simple uneroded low sulfidation epithermal systems has a silica cap, a vein/feeder with an argillic and phyllic alteration zone adjacent to the vein and a distal propylitic alteration envelope. Au, Ag and base metals are concentrated below the cap, in and near the zone of boiling. Quartz in this part of the system is commonly flow banded and may be brecciated. Au and Ag are highest in the system. Silver increases with depth and finally deep in the system base metals are predominant. Epithermal systems are usually complicated by overprinting of subsequent hydrothermal events. If the source of siliceous sinter is associated with mineralization found at Aslantepe, it indicates that the mineralization at the Project may be at the top of the hydrothermal system.

Koza has also identified five historic underground workings along the vein zone. Geology and drillhole location of Aslantepe is shown in Figure 9.4.2 with insets showing drilling and general location within the license. Figure 9.4.3 is a close up map of the drilled area.



Source: Koza, 2015 GIS

Figure 9.4.2: Geology and Drillhole Location Map for the Aslantepe Project



Source: Koza, 2013 GIS

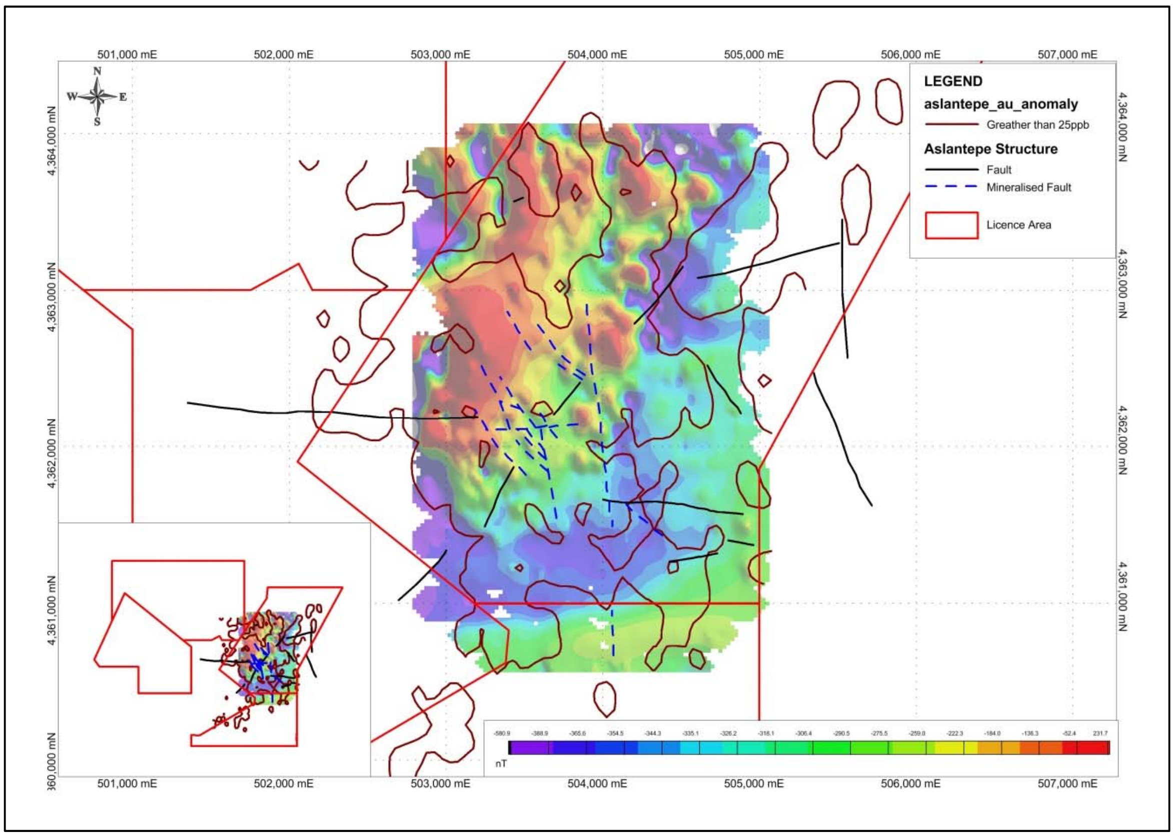
Figure 9.4.3: Drillhole Location Map with Detailed Geology for the Aslantepe Project

9.5 Exploration

Koza acquired the Aslantepe Project through auction in 2007 but did not begin exploration until 2008. Since then, Koza has collected 220 stream sediment samples, 902 soil samples, 570 continuous chip channel and 36 rock chip samples. Koza has mapped mineralization at Aslantepe at 1:5,000, 1:2000 and 1:1000 scales. In addition, Koza has completed geophysical surveys including 11.6 line km IP/resistivity and ±65 line km of ground based magnetics. Koza excavated 35 trenches over a total length of 745 m. Koza has budgeted TL2.1 million (US$952,000) for exploration drilling during 2015.

9.5.1 Geophysical Surveys

The geophysical surveys completed by Koza included 11.6 line km IP/resistivity and ±65 line km of ground based magnetic surveys. Figure 9.5.1.1 shows the magnetic survey completed by Koza superimposed on gold contours from surface sample data. Gold contours are at 25 ppb gold. Drilling at the project has targeted the largest magnetic high shown in the figure.



Source: Koza GIS, 2015

Figure 9.5.1.1: Ground Base Magnetic Survey Map with Gold Contours at 25 ppb

9.5.2 Sample Collection

Stream sediment samples were collected along master streams above and below the inflow of tributary creeks. Samples were collected to be as representative as possible. This was done by collecting a composite sample at each location from the same depositional environment in the stream bed. Koza screens stream sediment samples to -80 mesh and typically collects 3 to 4 kg of sample. Soil samples were collected using a regular grid and are collected from the B horizon.

Rock chip channel collected perpendicular to mineralized structures. Rock chip samples were typically 3 to 4 kg in weight. Samples are typically 1 m long but may be shorter or longer depending on lithological contacts. Samples are collected to be as representative as possible. Variability results from differing hardness in sampled material. Collection points ranged from 200 to 25 m apart along the structures trend and were selected based on field conditions and accessibility to the structures and veins.

9.5.3 Drilling/Sampling Procedures

Between 2012 and 2014, Koza completed 46 drillholes totaling 12,105.2 m of drilling. The drilling is still in the exploratory stage and Koza has not drilled on a regular grid. The drillholes and trenches are shown in Figure 9.5.3.1. Core recovery ranges between 13 and 100%, with an average of 99%. Koza used company owned drill rigs and crews to drill at Aslantepe.

Koza records drill core data onto paper and then enters the data into the computer. Data captured during core logging includes core recovery, RQD, fractures counts and orientation, vein orientation, rock type alteration and sulfide and oxide percentages. Core samples are selected and marked by the geologist. Sample intervals are selected by the geologist and are typically 1 m in length. Samples may be shorter or slightly longer than 1 m to accommodate changes in lithology. The core is cut in half lengthwise with half sent for assay and half archived for reference or future analysis.

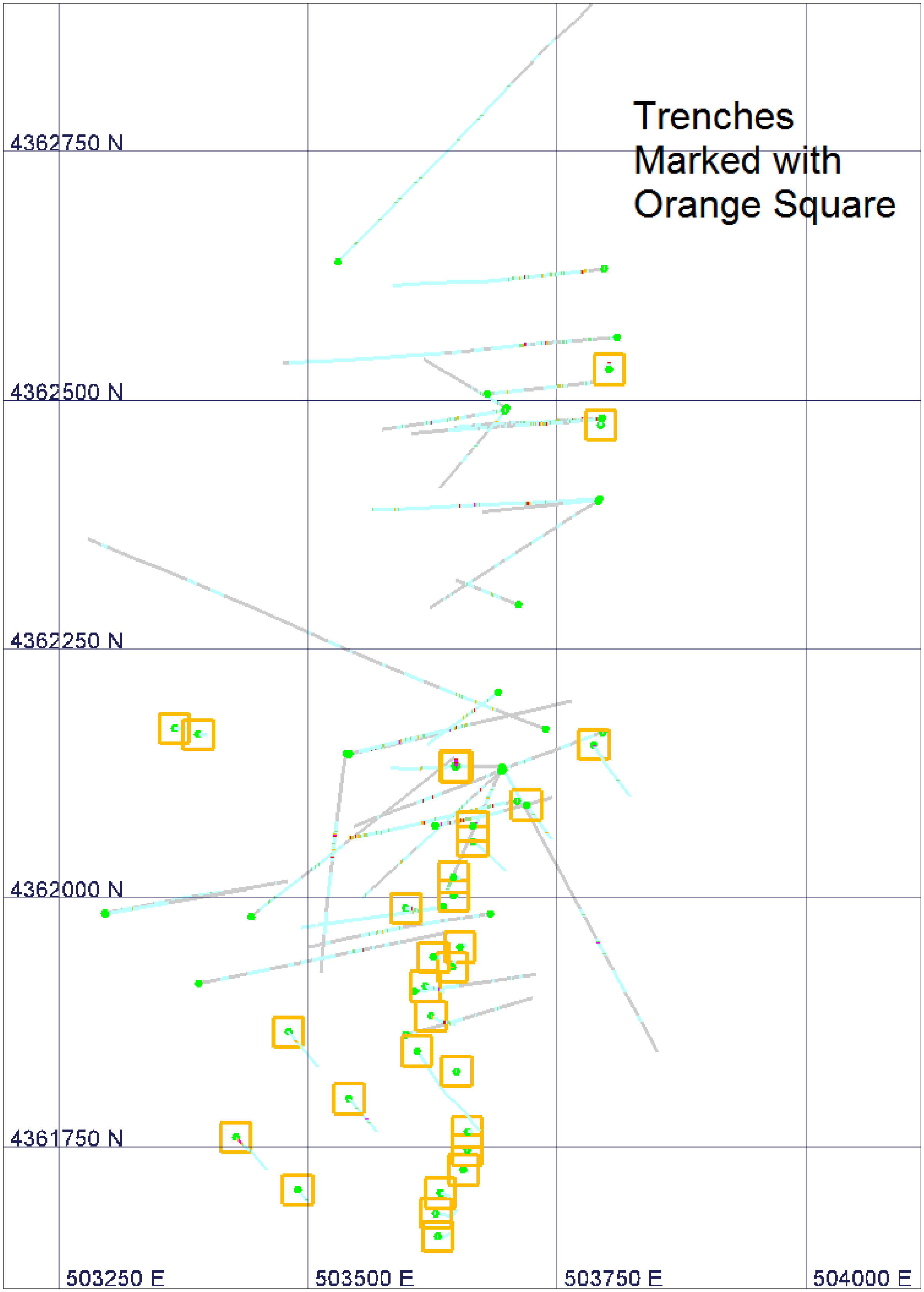


Figure 9.5.3.1: Drillholes and Trenches at Aslantepe

9.5.4 Sample Preparation and Analysis

Core and exploration samples are held in the custody of Koza until they are shipped to the laboratory for analysis in a locked vehicle, in a locked core logging facility or at the nearest mine site in a locked building. Core samples are either delivered to the laboratory by Koza personnel or shipped via commercial trucking. This is industry best practice.

Samples were prepared at ALS İzmir. The ALS Vancouver laboratory conducted ICP multi-element analysis and gold FA. ALS Vancouver has ISO 17025 accreditation for specific analytical methods through the Standards Council of Canada. ALS Vancouver's accreditation is valid through May 18, 2017.

Once the samples arrived at the laboratory, they were bar coded and entered into the Laboratory Information Management System (LIMS). All samples were dried to a maximum temperature of 60°C in order to avoid or limit volatilization of elements such as mercury (ALS code DRY-22). Soil and stream sediment samples were screened to -180 micron (80 mesh) to remove organic matter and large particles. Soil samples were then analyzed. Stream sediment samples were pulverized to 85% passing 75 microns (ALS code PUL-31) prior to digestion and analysis.

Soil and stream sediment 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. In this analysis, a 1 g sample is digested using aqua regia and 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. Koza also analyzed 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 deposit type. Table 9.5.3.1 presents the analytes with upper and lower detection limits for ALS ME-MS41 and Au-ICP22.

Table 9.5.3.1: 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

After drying using ALS code DRY-22, rock chip and core samples were crushed to 70% passing -2 mm (ALS code CRU-31) and a 1,000 g split was collected using a riffle splitter (ALS code SPL-21). The 1,000 g split was pulverized to 85% passing 75 microns (ALS code PUL-32). Koza requests a larger split pulverized to help mitigate the nugget affect.

Rock chip and core samples were analyzed using ALS code ME-ICP61, a 33 element package with trace level sensitivity. A 1 g sample is put into solution using a four acid digestion and the sample is analyzed using ICP-AES. Gold was analyzed using ALS code Au-AA24, which is gold by FA using a 50g charge with an Atomic Absorption Spectroscopy (AAS) finish. The samples were also analyzed for mercury using Hg-CV41. By this method, mercury content is determined using aqua regia digestion and cold vapor AAS. Table 9.5.3.2 presents the analytes with upper and lower detection limits for ALS ME-ICP61, Hg-CV41 and Au-AA24.

Table 9.5.3.2: Analytes and Upper and Lower Detection Limits for ALS Codes ME-ICP61, Hg-CV41 and Au-AA24 in ppm Unless Otherwise Noted

Method	Analyte	Range	Method	Analyte	Range	Method	Analyte	Range
Au-AA24	Au	0.005-10	ME-ICP61	Cu	1-10,000	ME-ICP61	S	0.01-10%
Hg-CV41	Hg	0.01-100	ME-ICP61	Fe	0.01-50%	ME-ICP61	Sb	5-10,000
ME-ICP61	Ag	0.5-100	ME-ICP61	Ga	10-10,000	ME-ICP61	Sc	1-10,000
ME-ICP61	Al	0.01-50%	ME-ICP61	K	0.01-10%	ME-ICP61	Sr	1-10,000
ME-ICP61	As	5-10,000	ME-ICP61	La	10-10,000	ME-ICP61	Th	20-10,000
ME-ICP61	Ba	10-10,000	ME-ICP61	Mg	0.01-50%	ME-ICP61	Ti	0.01-10%
ME-ICP61	Be	0.5-1,000	ME-ICP61	Mn	5-100,000	ME-ICP61	Tl	10-10,000
ME-ICP61	Bi	2-10,000	ME-ICP61	Mo	1-10,000	ME-ICP61	U	10-10,000
ME-ICP61	Ca	0.01-50%	ME-ICP61	Na	0.01-10%	ME-ICP61	V	1-10,000
ME-ICP61	Cd	0.05-1,000	ME-ICP61	Ni	1-10,000	ME-ICP61	W	10-10,000
ME-ICP61	Co	1-10,000	ME-ICP61	P	10-10,000	ME-ICP61	Zn	2-10,000
ME-ICP61	Cr	1-10,000	ME-ICP61	Pb	2-10,000			

Source: ALS Global, 2014

9.5.5 Quality Assurance and Quality Control

Insertion of Internal Controls

Location of the QA/QC is samples determined by the core logging geologist and are inserted into the drill core sequence. The QA/QC samples are bagged and submitted by the core sampler under the direction of the geologist who logged the core. The location of the control samples is noted on the sample log and in the sample database. The QA/QC samples have the same numbering system as the drill core samples. Sample blanks and preparation duplicates are inserted into the sample stream at a rate of one in every 30 samples. CRMs are inserted at a rate of one in every 50 samples.

Certified Reference Materials

In 2012 Koza began its drilling program using a low grade standard, OREAS 501, from Ore Research and Exploration (OREAS) based in Australia. With the 2013 drilling program, Koza added a second CRM, OxE86, purchased from RockLabs, based in New Zealand and during the 2014 drilling program Koza added a third CRM, OREAS 61e.

Koza generally uses ± 2 standard deviations or ±10% of the observed mean to define a performance range for its CRMs and reference materials and analyses falling outside of this range are considered failures. General industry practice is to use the following method for determining failures:

- If one analysis is outside of ±2 standard deviations it is a warning;
- Two or more consecutive analyses outside of ±2 standard deviations is a failure; and
- If an analysis is outside ±3 standard deviations it is a failure.

Using ± 2 standard deviations, there is one warning for each standard and no failures. The observed mean for OREAS 501 is 98% of the expected value for Au, for OxE86 is 99.0% of the expected value for Au and for OREAS 61e is 100.1% of the expected mean for Au. For silver, the observed mean for OREAS 501 is 85% of the expected value for Ag and for OREAS 61e is 98% of the expected value for Ag. Table 9.5.5.1 presents the expected mean, standard deviation and summary of the analysis of the Au CRM. Table 9.5.5.2 shows the same data for Ag CRMs.

Table 9.5.5.1: Results of Au CRM Analyses Aslantepe

Standard	Number Samples	Expected (ppm)		Observed (ppm)		% of Expected Mean	Number ±2 <3 Std Dev	Number ±3 Std Dev	% Failure Rate
		Mean	Std Dev	Mean	Std Dev				
OREAS 501	66	0.204	0.011	0.200	0.006	98.0	0	0	0
OxE86	50	0.613	0.021	0.609	0.017	99.0	2	0	0
OREAS 61e	3	4.43	0.15	4.45	0.157	100.1	0	0	0
Total	**116**						**2**	0	**0**

Table 9.5.5.2: Results of Ag CRM Analyses Aslantepe

Standard	Number Samples	Expected (ppm)		Observed (ppm)		% of Expected Mean	Number ±2 and < 3Std Dev	Number ±3 Std Dev	% Failure Rate
		Mean	Std Dev	Mean	Std Dev				
OREAS 501	57	0.840	0157	0.714	0.24	85	0	3	5
OREAS 61e	3	5.27	0.43	5.27	0.15	98	0	0	0
Total	**60**						**0**	3	**5**

The results show that with the exception of OREAS 61e for gold analysis the CRMs are performing low. Two of the gold CRMs are performing but all are within acceptable limits. The CRMs for silver are also low, but OREAS501 has many more warnings at two standard deviations. The mean for silver for OREAS501 is close to the detection limit of the analysis and the CRM performance can be attributed to this.

Koza is using three CRMs for this drilling program two of which are very low grade and below the 0.8 g/t Au cutoff grade for the Inferred resource. It is industry best practice to use at least two CRMs to bracket the grade range of a deposit to determine accuracy of the analysis in a range of grades. SRK recommends that Koza discontinue using OREAS 501 with an expected range of 0.204 g/t Au and 0.804 g/t Ag, and add at least one more CRM to its QA/QC program of a higher grade value for silver.

Blanks

Sample blanks test for contamination in preparation and assaying and handling errors. Koza used pulp blanks up until June 2012. After June 2012, pulp blanks were replaced with preparation blanks. A blank failure is a result greater than five times the detection limit. There were no blank failures in the 63 samples submitted at Aslantepe. The results indicate that there are no cross contamination issues at the preparation laboratory.

Preparation Duplicates

Preparation duplicates are collected from a second cut of the crushed sample (coarse reject) using the same splitting method and 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.

Koza sent 64 preparation duplicate to the ALS Chemex for Au analysis and 74 for Ag. A summary of the analytical results are presented in Tables 9.5.5.3 and 9.5.5.4 for Au and Ag, respectively.

Table 9.5.5.3: Summary of Duplicate Au Analysis at Aslantepe

Criteria	Number of Samples	Original>Dup	Dup>Original	Original = Dup	Within +/- 20%
All samples	64	17	17	30	57
		27%	27%	47%	89%

Table 9.5.5.4: Summary of Duplicate Ag Analysis at Aslantepe

Criteria	Number of Samples	ALS>SGS	SGS>ALS	ALS = SGS	Within +/- 20%
All samples	74	17	16	41	63
		23%	22%	55%	85%

Of the 64 samples submitted, four are above the 0.80 g/t Au cutoff grade of the Inferred resource. However, all of those samples were within ±20% and were acceptable. The seven samples that were outside the ±20% performance range were all below 0.2 g/t Au and are not considered material. However, the range of samples submitted for analysis is not representative of the mineralization at Aslantepe. The results of this small database indicate that sample preparation is adequate for the analysis for low ranges.

Silver submissions that were failures were predominantly in the lower ranges at the detection limit. There were also two failures in the range of mineralization at the deposit. SRK notes that removing the failures near the detection limit shows a 95% reproducibility rate for crushed samples in the range of silver mineralization indicating excellent precision.

SRK recommends that preparation duplicates continue to be submitted selecting duplicates from core that is representative of the mineralization at Aslantepe.

Pulp Duplicates

Koza has not sent pulp duplicates to ALS Chemex. Pulp duplicates are used to test the laboratory’s precision. SRK commonly recommends that clients request that a split of selected pulps be requested and these pulp duplicates be analyzed by the laboratory to assess laboratory precision during drilling programs. Alternatively, commercial laboratories routinely run pulp duplicates as part of internal laboratory QA/QC. The internal QA/QC can be requested and reviewed by Koza as an acceptable procedure to assess the laboratory precision, without submitting additional samples. However, this eliminates Koza’s ability to select specific sample intervals for pulp duplicate analysis.

Secondary Check Lab Analysis

Koza sent 76 Aslantepe samples to SGS Ankara to verify results from ALS Chemex. The samples were submitted with CRM OREAS 501. The analysis code at SGS Ankara was Au-FAA 505 which is FA method using a 50 g charge with an AAS finish. This was selected to match ALS Chemex code Au-AA24, and the two methods are equivalent except that the two methods have different lower detection limits. At SGS Ankara, the lower detection limit is 0.01 ppm and at ALS Chemex it is 0.005 ppm.

OREAS 501 was initially submitted to SGS Ankara a total of three times. Of these two were outside the performance range. In both cases the failures were above the upper acceptable limit for the CRM. Koza requested a reanalysis of the 25 samples with the insertion of two samples of OREAS 501. The CRMs in the second submission were within the performance range. All five submissions of OREAS 501, failures and those that passed, performed high at SGS Ankara. The same CRM reported low overall at ALS Chemex with 55 submissions.

Since check assays are done on the same pulp, the secondary laboratory is expected to return analytical results within ±10% of the original analysis at the primary laboratory. The analysis of Aslantepe samples show that 54% of the results from SGS Ankara were within ±10% of the original sample. Of those samples submitted as checks, 37 were above the cutoff grade of 0.80 g/t Au and 56.7% of that set were within ±10%. Overall, ALS Chemex analyses were higher than those from SGS Ankara. Table 9.5.5.5 presents the pulp duplicate results.

Table 9.5.5.5: Summary of Duplicate Au Analysis at Aslantepe

Criteria	Number of Samples	ALS>SGS	SGS>ALS	ALS = SGS	Within +/- 10%
All samples	76	44	25	7	41
		59%	33%	9%	54%

Some of the difference in precision could be the result of nugget affect in the higher grade samples where coarser grained gold was not evenly distributed. It could also be related to the sensitivity in the two analyses. The lower detection limit at ALS Chemex is half that of SGS Ankara. The CRM performed low overall at ALS Chemex. The CRM dataset at SGS Ankara is too small to be statistically meaningful; however, the CRM performed high at SGS Ankara. Because of this, it is difficult to assess why there are differences in the laboratory results. SRK recommends that Koza continue submitting samples for check assays, ensuring that the same standards are submitted to both laboratories and that both laboratories are using the same digestion and analytical techniques.

The CRM sample submitted is low grade and below the cutoff grade for the mineralization at Aslantepe. Koza is also using CRM OxE86, which with a mean of 0.603 g/t Au is closer to the cutoff grade of 0.8 g/t Au and more appropriate for the resource range. SRK has recommended discontinuing use of OREAS 501.

9.6 Mineral Resources

The Mineral Resources were estimated by Koza in 2013 (Koza, 2013d).

9.6.1 Geological Modeling and Grade Estimation

Koza constructed wireframes of the veins based on a cutoff grade of 0.50 g/t Au and a minimum thickness of 1 m. There are 14 wireframes which have been grouped into 4 domains based on orientation. The wireframes cover an area that extends 700 m north-south, 350 m east-west and 250 m vertically. Domain 1 comprises six wireframes striking north-south with steep dips; the average thickness is less than 3 m. Domain 2 consists of two, nearly horizontal wireframes, one stacked over the other; the upper wireframe is less than 3 m in thickness and the lower is up to 16 m in thickness. Domain 3 consists of four wireframes striking north-northeast with steep dips; the thickness of the wireframes is less than 3 m, except for one where the thickness reaches 5 m. Domain 4 consists of two wireframes, one striking east-west and the other northwest, both with steep dips and 4 to 10 m in thickness. The domains are shown in Figure 9.6.1.1. Statistics of the assays within the wireframes are presented in Table 9.6.1.1.

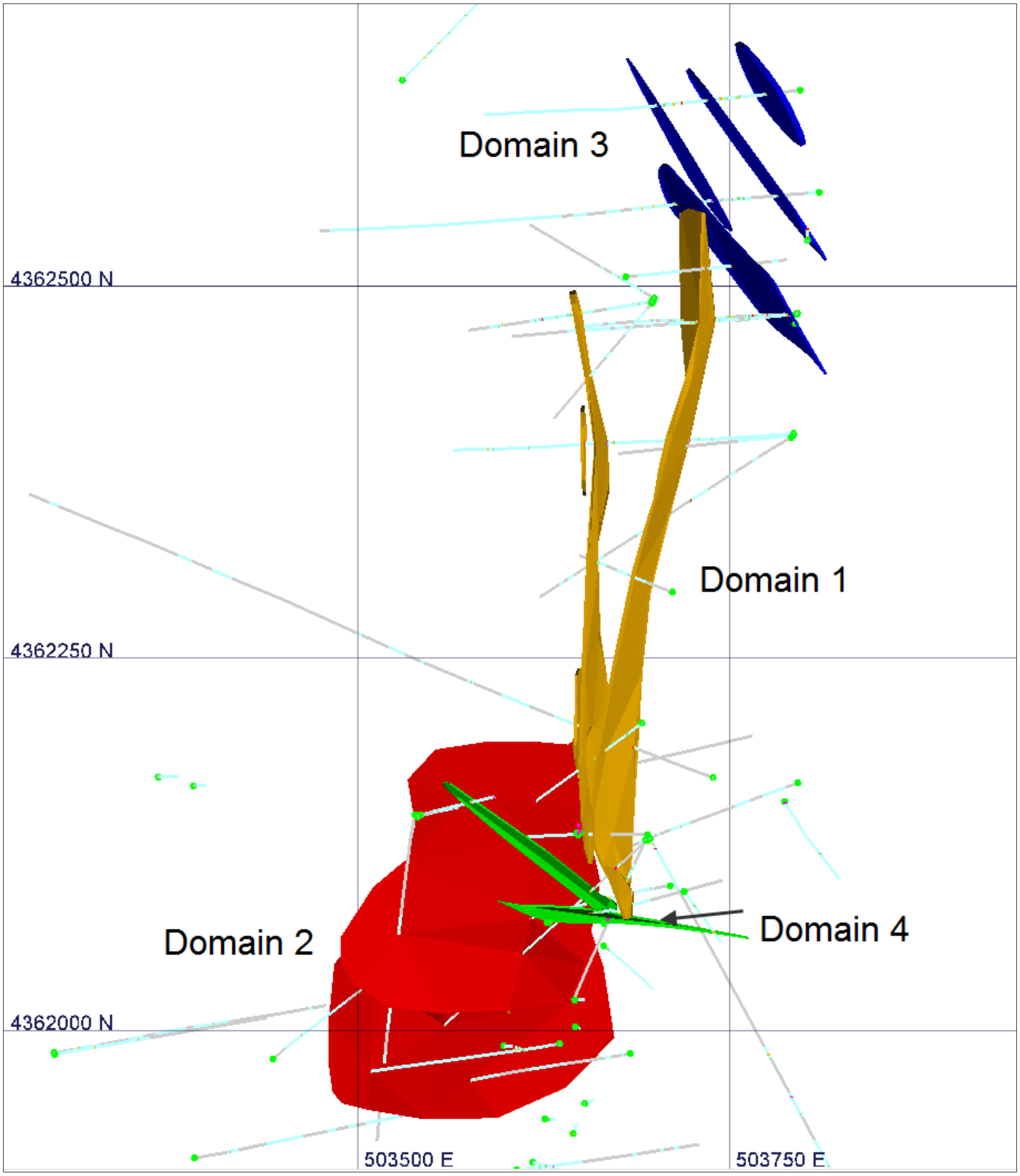


Figure 9.6.1.1: Aslantepe Wireframes and Domains

Table 9.6.1.1: Statistics of Assays within the Aslantepe Wireframes

Metal	Domain	Samples	Minimum	Maximum	Mean	Std Dev	CV
Au	1	118	0.03	24.20	2.22	3.62	1.63
	2	84	0.03	30.50	2.07	3.87	1.87
	3	59	0.13	14.00	1.81	2.38	1.31
	4	128	0.00	20.22	1.86	2.95	1.59
Ag	1	118	0.00	60.90	5.10	8.20	1.61
	2	81	0.00	32.90	2.36	4.62	1.96
	3	59	0.80	27.30	4.23	4.61	1.09
	4	128	0.00	105.10	13.32	17.96	1.35

9.6.2 Compositing and Capping

The raw assay sample lengths average 0.97 m and 88% are 1 m or less in length. Koza chose 1 m as the compositing length. Statistics of the uncapped composites are presented in Table 9.6.2.1. .

Table 9.6.2.1: Statistics of Uncapped Composites at Aslantepe

Metal	Samples	Domain	Minimum	Maximum	Mean	Std Dev	CV
Au	107	1	0.00	19.70	1.93	2.88	1.49
	83	2	0.03	30.50	2.16	3.98	1.84
	58	3	0.09	14.00	1.67	2.13	1.27
	101	4	0.01	26.70	2.53	4.94	1.95
Ag	107	1	0.00	35.90	4.54	6.01	1.32
	83	2	0.00	32.90	2.29	4.58	2.00
	58	3	0.80	27.30	4.07	4.32	1.06
	101	4	0.00	83.40	12.53	16.39	1.31

Koza reviewed histograms, probability plots and distribution of gold and silver grades to select an appropriate capping value. For gold, the composites were capped at 11 g/t; and for silver, the composites were capped at 48 g/t. Statistics of the capped composites are shown in Table 9.6.2.2. SRK suggests that the capping be reviewed on a domain basis, especially for silver, where the capping is influenced by the high values in Domain 4.

Table 9.6.2.2: Statistics of Capped Composites at Aslantepe

Metal	Domain	Samples	Minimum	Maximum	Mean	Std Dev	CV
Au	1	107	0.00	11.00	1.83	2.41	1.31
	2	83	0.03	11.00	1.89	2.49	1.32
	3	58	0.09	11.00	1.62	1.85	1.14
	4	101	0.01	11.00	1.96	2.76	1.41
Ag	1	107	0.00	35.90	4.54	6.01	1.32
	2	83	0.00	32.90	2.29	4.58	2.00
	3	58	0.80	27.30	4.07	4.32	1.06
	4	101	0.00	48.00	11.74	13.72	1.17

9.6.3 Variography

Because of the limited number of samples, it was not possible to conduct a variography study.

9.6.4 Density

A density value of 2.36 g/cm^3 was used in the resource estimation based on 24 core samples taken from the Aslantepe deposit. The density is on a dry tonnage basis.

9.6.5 Grade Estimation

A block model was created with a block size of 5 m x 5 m x 5 m, with sub-blocking allowed to 1 m. The block size is much smaller than the drillhole spacing, but is comparable to the vein thickness.

The estimation was done in three passes with the search ellipsoid oriented along the strike and dip of the vein with search criteria as follows:

- Firs Pass: minimum of 10, maximum of 20 composites, search 60 m x 60 m x 10 m;
- Second Pass: minimum of 5, maximum of 20 composites, search 120 m x 120 m x 20 m; and
- Third Pass: minimum of 2, maximum of 20 composites, search 240 m x 240 m x 40 m.

This search was used for Domains 1, 2 and 3. In Domain 4, the search was omnidirectional using the maximum search shown in the bullets above. Only composites within the wireframe were used for estimation. A maximum of five composites could be used in the block estimation, thus requiring at least two drillholes for estimation in the first pass. ID2 was used as the estimation method.

9.6.6 Block Model Validation

Koza validated the block model by conducting estimations with ID3 and NN and comparing to the ID2 estimation. Koza also compared the average block grades to the composite grades. Composite and block grades were also compared on cross-sections. Table 9.6.6.1 presents the average block grades by estimation method and composite grades by domain.

The ID2 gold estimation is higher than the composite grade in Domains 1 and 3 and slightly lower in Domain 4. The estimated silver grade is also high in Domains 1 and 3 and lower in Domains 2 and 4. It appears that the reason for this is that in an area of widely spaced drilling, a single drillhole with high grades has an influence on a large tonnage. With more drilling, this problem should be alleviated.

Table 9.6.6.1: Comparison of Estimated Block Grades and Composites at Aslantepe

Domain	Au				Ag	
	ID2	ID3	NN	Composite	ID2	Composite
1	2.01	2.05	2.56	1.83	4.77	4.54
2	1.89	1.89	1.88	1.89	2.15	2.29
3	1.84	1.84	1.40	1.62	4.50	4.07
4	1.88	1.76	1.61	1.96	10.56	11.74
Total	1.93	1.95	2.12	1.85	8.44	6.01

9.6.7 Mineral Resource Classification and Statement

All blocks were classified as Inferred based on the low number of drillholes.

The resources are tabulated at a cutoff grade of 0.85 g/t Au based on the open pit mining parameters shown in Table 9.6.7.1. 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 9.6.7.1: Aslantepe Cutoff Grade Parameters

Prices and Costs	Units	Open Pit
Gold Price	US$/oz	1,450
Gold Recovery	%	0.95
Gold Refining	US$/oz	3.44
Government Right	%	1
Process Cost	US$/t	11.00
Mining Cost	US$/t	0.00
G&A Cost	US$/t	15.00
Transport Cost	US$/t	11.00
Calculated Cutoff grade	g/t	0.85
Final Cutoff grade	g/t	0.85

Source: Koza, 2014

Koza generated a pit shell to constrain the resources. The parameters used in the pit optimization study are shown in Table 9.6.7.2.

Table 9.6.7.2: Pit Optimization Parameters for Aslantepe

Prices and Costs	Units	Open Pit
Gold Price	US$/oz	1,450
Silver Price	US$/oz	30
Gold Recovery	%	0.95
Silver Recovery	%	0.75
Government Right	%	1
Process Cost	US$/t	11.88
Mining Cost	US$/t	0.00
G&A Cost	US$/t	19.42
Transport Cost	US$/t	14.00

Source: Koza, 2013

The Aslantepe Inferred Resources are stated in Table 9.6.7.3.

Table 9.6.7.3: Aslantepe Mineral Resources, at December 31, 2014

Classification	kt	g/t Au	g/t Ag	koz Au	koz Ag
Inferred	263	2.71	8.4	23	71

- Tonnages and grade are rounded to reflect approximation;
- Resources are stated at a cutoff grade of 0.85 g/t; and
- Open pit resources are contained within grade shells but are not constrained by a pit optimization shell.

9.6.8 Mineral Resource Sensitivity

Figure 9.6.8.1 presents grade tonnage curves for the Inferred Resources.

Cutoff grades for the Aslantepe resource at various gold prices are shown in Table 9.6.8.1.

Table 9.6.8.1: Aslantepe Cutoff Grades vs. Gold Price

Gold Price	Cutoff Grade
1600	0.77
1550	0.79
1500	0.82
1450	0.85
1400	0.88
1350	0.91
1300	0.94
1250	0.98
1200	1.02

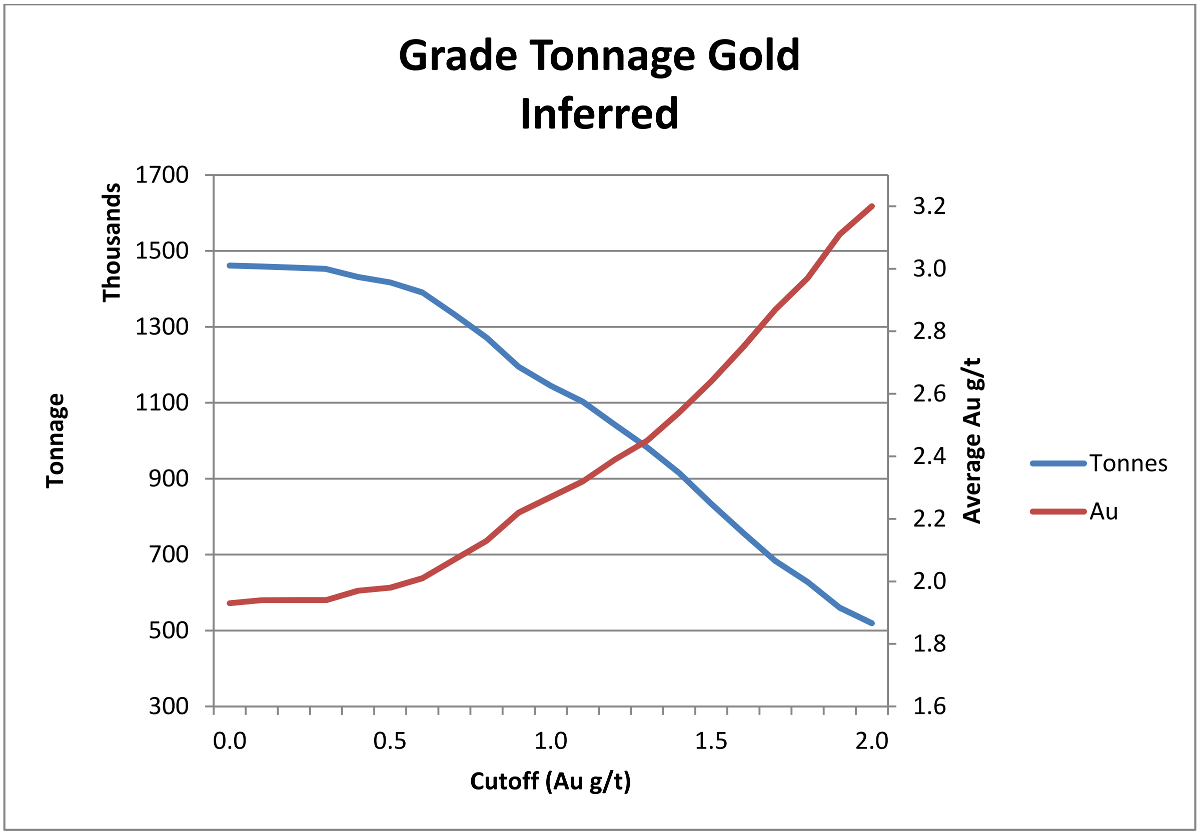


Figure 9.6.8.1: Grade Tonnage Curves for Aslantepe Inferred Resource

9.7 Environmental

The EIA process was cancelled by the Ministry on December 25, 2014. There is no existing environmental information available for this prospect since it is still in exploration phase. SRK is currently not aware of any environmental and/or social limitations.

9.8 Conclusions and Recommendations

SRK suggests that Koza continue to drill at Aslantepe in order to better understand the geology and to increase the resource.

SRK suggests that Koza review the capping values. The highest silver values are found in Domain 4 and these values have been used to determine the capping value. The other three domains have significantly lower grades and the capping value should be lower in those domains.

Results from the Aslantepe QA/QC indicate that the analytical and preparation used for samples at this deposit are appropriate. SRK recommends that Koza add at least one more CRM to its QA/QC program of a higher grade value for silver, that Koza request the ALS Chemex pulp duplicate data or if Koza prefers request that ALS Chemex create a pulp duplicate of selected intervals during its next drilling program in addition to the preparation duplicates. SRK also recommends that Koza continue submitting check samples to SGS ensuring that standards are included and that both laboratories are using the same digestion and analytical techniques.

10 Conclusions and Recommendations

10.1.1 Exploration

Koza uses industry best practices for its exploration activities.

Koza follows has a QA/QC program that supports resource estimation. Koza monitors the QA/QC at its projects during drilling and addresses identified failures immediately, which is industry best practice. Koza uses CRMs and in-house standards at every project and has recently started to use preparation blanks to better monitor cross contamination during sample preparation. Koza uses preparation duplicates. Improvements to the QA/QC program include the following:

- Using two to three CRMs at each deposit to bracket the grade range of each deposit;
- Submitting duplicate samples within the grade range of the deposit;
- Based on performance of coarse duplicates, Koza may need to address the sample preparation at Kıratlı, Narlıca and Çukuralan;
- Adding pulp duplicates to the QA/QC program that would monitor the precision of the analysis; and
- Submitting check samples to a secondary laboratory for verification of analysis at the primary laboratory.

SRK also recommends that that Koza monitor QA/QC results for every element reported in a resource estimate at each deposit.

10.1.2 Geology and Resources

See specific project conclusions and recommendations.

10.1.3 Mining and Reserves

See specific project conclusions and recommendations.

10.1.4 Metallurgy

See specific project conclusions and recommendations.

10.1.5 Environmental and Permitting

See specific project conclusions and recommendations.

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12 Glossary

12.1 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.

12.2 Glossary of Terms

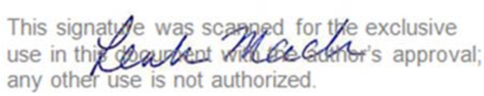
Table 11.2.1: Glossary

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.
PIMA	Portable Infrared Mineral Analyzer
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).

13 Date and Signature Page

Signed on this 31st Day of January, 2015.

Endorsed by CPs:



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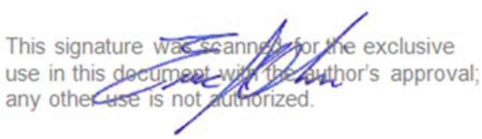

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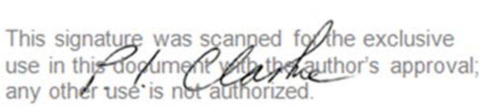

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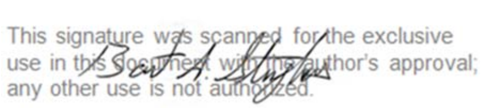

Bart Stryhas, PhD. CPG

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

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