CENTERRA GOLD INC.

TECHNICAL REPORT ON THE ÖKSÜT GOLD PROJECT, TURKEY

NI 43-101 Report

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1 SUMMARY

BACKGROUND AND PROJECT DESCRIPTION

The Öksüt Gold Project (the Öksüt Project or the Project) is located in the Kayseri province of south-central Turkey, 295 km to the southeast of the capital city of Ankara and 48 km directly south of the city of Kayseri. This Technical Report supports the disclosure of the Project's Mineral Resource and Mineral Reserve estimates at June 30, 2015 and has been prepared in accordance with National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101). All dollar amounts in this Technical Report are stated in United States dollars (US$) unless otherwise noted.

In August 2009, Centerra Gold Inc. (Centerra) entered into a joint venture with Stratex International Plc (Stratex), a publicly listed company on the London-based Alternative Investment Market (AIM), for the Öksüt Project through its now wholly-owned Turkish subsidiary, Öksüt Madencilik Sanayi ve Ticaret A.Ş. (OMAS). In October 2012, Centerra increased its interest in the joint venture to 70% after completing total expenditures of $6M on exploration. In January 2013, Centerra purchased Stratex's remaining 30% interest for a cash payment of $20M and a 1% net smelter return (NSR) royalty up to a maximum of $20M. In addition to the royalty payable to Stratex's wholly-owned subsidiary Stratex Gold AG (Stratex Gold), the Project is also subject to a sliding scale NSR royalty in favour of Teck Resources Limited (Teck) as well as a sliding scale royalty in favour of the Turkish Government. Closing of the purchase was also conditional on the granting of two Operation Licences by the Turkish authorities, which occurred on January 16, 2013. Centerra now owns 100% of the Project.

The Öksüt Project was first discovered in 2007 when geological staff of Stratex identified gold mineralization in reconnaissance rock chip sampling from outcrops located on what is now referred to as the Güneytepe Deposit area. Prior to this, there is no record of any modern exploration for gold being conducted on the property. In 2008, Stratex followed up these surface sampling results with an initial 13-hole diamond drilling program located within the Güneytepe Deposit area.

Following the formation of the joint venture, several additional drilling programs were completed on the property resulting in the first Centerra publication of a Mineral Resource estimate on the Project in February 2013 (effective at December 31, 2012) subsequently followed by a revised Mineral Resource estimate published in February 2014 (effective at December 31, 2013). On February 19, 2014, Centerra announced the results of a preliminary economic assessment (PEA) on the Project. An updated Mineral Resource estimate was published in February 2015 (effective at December 31, 2014). On July 28, 2015, Centerra announced the results of a Feasibility Study (FS) on the Öksüt Project.

The Öksüt Project is planned as a conventional truck and shovel open pit heap leach mining operation. A total of approximately 26.1 Mt of ore at a grade of 1.4 g/t Au, containing a total of approximately 1.2 million ounces of gold, is planned to be mined and stacked over a mine life of eight years from two open pits, the Keltepe and the smaller Güneytepe pit.

Table 1-1 provides a summary of Mineral Reserves and Mineral Resources in addition to Mineral Reserves with an effective date of June 30, 2015. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves adopted on May 10, 2014 (CIM definitions) were followed for this estimate.

TABLE 1-1 ÖKSÜT PROJECT MINERAL RESERVE AND MINERAL RESOURCE SUMMARY (AT JUNE 30, 2015)

Mineral Reserves (1) (4) (7) (8)									
(tonnes and ounces in thousands)									
	Proven			Probable			Total Proven and Probable		
Deposit	Tonnes	Grade (g/t)	Contained Gold (oz)	Tonnes	Grade (g/t)	Contained Gold (oz)	Tonnes	Grade (g/t)	Contained Gold (oz)
Keltepe	-	-	-	22,821	1.4	1,036	22,821	1.4	1,036
Güneytepe	-	-	-	3,316	1.2	125	3,316	1.2	125
Total	**-**	**-**	**-**	**26,137**	**1.4**	**1,162**	**26,137**	**1.4**	**1,162**

Mineral Resources (2)(3)(5)(6)(7)(8)									
Measured and Indicated Mineral Resources									
(tonnes and ounces in thousands)									
	Measured			Indicated			Total Measured and Indicated		
Deposit	Tonnes	Grade (g/t)	Contained Gold (oz)	Tonnes	Grade (g/t)	Contained Gold (oz)	Tonnes	Grade (g/t)	Contained Gold (oz)
Keltepe	2,024	0.7	44	4,450	0.7	106	6,474	0.7	150
Güneytepe	76	0.5	1	248	0.7	5	324	0.6	7
Total	**2,100**	**0.7**	**46**	**4,698**	**0.7**	**111**	**6,798**	**0.7**	**157**

Inferred Mineral Resources			
(tonnes and ounces in thousands)			
Deposit	Tonnes	Grade (g/t)	Contained Gold (oz)
Keltepe	1,705	0.8	44
Güneytepe	675	1.0	21
Total	**2,380**	**0.8**	**64**

Notes:
1. Mineral Reserves have been estimated based on a gold price of US$1,250 per ounce.
2. Mineral Resources are in addition to Mineral Reserves. Mineral Resources do not have demonstrated economic viability.
3. Mineral Resources are constrained within an optimized pit shell based on a gold price of $1,450 per ounce.
4. Mineral Reserves are estimated based on a cut-off grade of 0.3 g/t Au.
5. Mineral Resources are estimated based on a cut-off grade of 0.2 g/t Au.
6. Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the Inferred Mineral Resources will ever be upgraded to a higher category.
7. A conversion factor of 31.10348 grams per ounce of gold is used in the reserve and resource estimates.
8. Numbers may not add up due to rounding.

PROPERTY LOCATION, DESCRIPTION AND CLIMATE

The Öksüt Project is located in south-central Turkey, 295 km to the southeast of the capital city of Ankara and 48 km directly south of the city of Kayseri which has a population of 1.1 million. The nearest administrative centre is at Develi (population 64,000) located approximately 10 km north of the Project. Ankara and Kayseri have international airports and are serviced by international and domestic airlines.

Currently, travel to the Project is via the paved dual lane road from Kayseri southwards to Develi over the eastern flank of Mount Erciyes. From there, the road connects Develi to the town of Gömedi, 33 km to the south-southwest, and then via minor access roads to the village of Zile. The Project site is accessed via a narrow agricultural dirt road from the village of Zile and, as a result, access to the site in winter is presently limited.

The Project is located in the Develi Mountains on a north-south trending topographic high. The topographic relief comprises steep-sided V-shaped valleys, and locally, cliffs tens of metres high, capped by flat-lying mesas and plateaus. The Project site is located at an elevation of approximately 1,800 m. The valleys are extensively farmed, with the local population living in a number of small villages including the villages of Öksüt and Zile.

The climate is continental with cold, snowy winters and warm, dry summers with cool nights due to the area's high elevation. Temperatures range from 10°C to 29°C in summer and -5°C to 7°C in winter. The annual rainfall is approximately 370 mm, generally falling as mixed rain and snow in winter and as rain during spring. Vegetation at higher elevations is predominantly scrub oak, small shrubs, and grasses.

The Develi area is well-serviced with electrical power lines. The Project is expected to be supplied from a substation 28 km away, near the town of Sendrimeke. The proposed all weather access to the Project is from Epçe. Water for the Project will be supplied from two wells located just south of the village of Epçe. These wells have been fully tested, and the permits and land rights have been received. Preliminary reviews of the region suggest that there is an abundant level of mining support services and skilled to semi-skilled labour available to support a potential mining operation.

GEOLOGY AND MINERALIZATION

The Öksüt Project is a high-sulphidation epithermal gold deposit within the Central Anatolian Volcanic Province, part of the Tethyan Metallogenic Belt. The belt extends from southeastern Europe across Turkey, the Caucasus, and on into Pakistan and contains a number of important gold and porphyry copper deposits. Magmatic activity and related ore forming processes are the result of the closure of the Tethyan Ocean in response to the collision between the north-moving Arabian Plate with the Eurasian Plate that began in the late Cretaceous period and continues today.

Öksüt gold mineralization is hosted within the Develidağ Volcanic Complex, one of the numerous stratovolcanoes situated along the Central Anatolian Fault Zone (CAFZ). The volcanic complex is composed of Miocene basaltic-andesitic volcanic domes, pyroclastic rocks, and lava flows. Flow-banded Pliocene andesite overlies these sequences and the Öksüt mineralization to the north and east.

There are several gold occurrences in the Öksüt Project area, the most important of which is the Keltepe Deposit. The distribution of the alteration assemblages and the gold grades at the Keltepe Deposit are strongly zoned, with a central massive silica breccia having the highest gold grade. This core is surrounded by quartz-alunite altered volcanic rocks, and as the alteration intensity diminishes outwardly, the gold grade decreases.

The Keltepe Deposit has been oxidized to depth, up to 400 m below the surface. The original copper content of the deposit has been completely leached out of the current resources, however, zones of oxide copper enrichment are found deeper within the deposit, below the planned open pit. An irregular zone of supergene enrichment exists below the oxide zone, with some high grade sulphide copper intersections. It is surmised that the oxidation of the deposit has liberated the gold allowing heap leaching at a relatively coarse crush size.

The nearby Güneytepe Deposit is significantly smaller based on the drilling information collected to the end of 2014 and does not show the more straightforward zonation and continuity of alteration and gold grades as observed on the Keltepe Deposit. Silicification is intense, however, the host rocks are much less porous, and, as a result, oxidation is restricted to the upper 50 m to 75 m of this deposit.

MINERAL RESOURCES

Mineral Resources for the Öksüt Project were estimated using a block model constrained with three dimensional (3D) wireframes of the principal mineralized domains and incorporating all the drilling completed to the date of the resource estimate. Values for gold were interpolated into blocks using ordinary kriging (OK). The Mineral Resource estimate, exclusive of Mineral Reserves, is summarized in Table 1-1.

The resource model update for the Öksüt Project was prepared by Centerra in May 2015, using all of the drill holes available as of that date. ARANZ Leapfrog software was used to update the principal mineralized domains at Keltepe and Güneytepe and values for gold were interpolated into blocks using OK in GEOVIA GEMS software.

MINERAL RESERVES

Mineral Reserves are that part of the Mineral Resource that can be legally, safely, and profitably mined given a specific set of technical and economic parameters. These include the gold price, mine, mill, and administrative operating costs, metallurgical recovery, geotechnical behaviour of the rocks in the future pit walls, and equipment size parameters. Computer software "optimizes" the pit shape by interrogating each block of the block model as to its ability to pay for its removal plus the incremental tonnage of waste that must be removed to mine the block. This process results in the creation of one or more "pit shells" which recover the economic part of the Mineral Resources and which are then engineered in detail by adding ramps for mining access and by smoothing of the pit walls.

The Mineral Reserve estimate at a 0.3 g/t Au cut-off grade for the Keltepe and Güneytepe Deposits is summarized in Table 1-1. All oxide resources classified as either Measured or Indicated inside the reserve pits have been classified as Probable Mineral Reserves.

MINING METHODS

The Öksüt Project is planned as a conventional truck and shovel open pit mine with two pits planned to be mined simultaneously, the main Keltepe pit and the small satellite Güneytepe pit. It is planned to use a mining contractor to do all mining using small excavators and 36 tonne trucks. The use of this equipment among mining contractors is common in Turkey. The mining contractor will provide and maintain all equipment, and will perform drill, blast, load, haul, and road and dump maintenance on a unit cost basis. Centerra will provide oversight of the mining operations, grade control, survey control, mine planning, and other required technical services.

The Keltepe pit will be developed in three cutbacks in order to smooth stripping requirements and mine higher grade material earlier in the mine life. Due to its small size, the Güneytepe pit will be developed in a single cutback. Lower grade material will be stockpiled throughout the Project for processing at the end of the mine life. This allows higher grade material to be processed earlier, increasing the Project net present value (NPV). Tables 1-2 and 1-3 show the life-of-mine (LOM) mining and processing schedules.

TABLE 1-2 LIFE-OF-MINE MINING SCHEDULE

	Total	2016	2017	2018	2019	2020	2021	2022	2023
Ore Mined (Mt)	**26.1**	0.2	3.8	4.0	5.7	2.3	3.4	5.4	1.4
Grade (g/t)	**1.38**	2.08	1.47	1.41	1.71	1.95	1.00	0.93	1.32
Contained Gold (koz)	**1,162**	15.9	178.4	182.2	310.6	146.1	109.4	161.8	57.3
Ore Rehandled (Mt)	**6.3**	-	0.5	0.7	-	2.2	0.6	0.1	2.2
Waste Mined (Mt)	**51.1**	5.1	8.9	8.8	7.1	10.4	8.6	2.1	0.1
Total Material Mined (Mt)	**77.3**	5.4	12.7	12.8	12.8	12.7	12.0	7.4	1.5

TABLE 1-3 LIFE-OF-MINE PROCESSING SCHEDULE

	Total	2016	2017	2018	2019	2020	2021	2022	2023	2024
Ore Processed (Mt)	**26.1**	-	3.30	4.02	4.02	4.02	3.53	4.02	3.22	-
Grade Processed (g/t)	**1.38**	-	1.69	1.44	2.07	1.47	0.98	1.09	0.83	-
Contained Gold (koz)	**1,162**	-	180	186	267	190	111	141	86	-
Gold Produced (koz)	**895**	-	90	179	199	151	93	106	70	7

METALLURGY

Metallurgical testing has focused on supporting the development of the Öksüt Project as a heap leach operation. Testing has focused on gold recovery at coarse particle sizes. Metallurgical testing was initiated in 2012 using samples from existing exploration diamond drill holes. A second program completed in 2012 utilized samples from a single large diameter hole to provide the bulk of the sample. The second program included the first column leach tests. In 2013, four large diameter drill holes were drilled (three in the Keltepe Deposit and one in the Güneytepe Deposit) to provide samples for two large scale column leach test programs. A mineralogy program was also completed on the samples from this program. In 2014, a further five large diameter drill holes (one in the Güneytepe and four in the Keltepe Deposit) were completed to provide samples for additional large scale column leach tests and further mineralogical analysis.

The results from all programs show that samples from both the Keltepe and Güneytepe Deposits are amenable to heap leach processing. Leach rates are relatively fast with comparatively high final recoveries. Size by size analysis of the column leach test feed and tails samples shows gold evenly distributed among the size classes, roughly following the mass splits.

Since the Keltepe Deposit currently contains approximately 90% of the contained gold for the Öksüt Project, the leach characteristics for the Keltepe Deposit will predominate. Güneytepe Deposit leach characteristics are expected to be as good as or better than leach characteristics of the Keltepe Deposit. Analyzing all the column test results using a Keltepe average grade of 1.24 g/t Au, the corresponding leach residue grade will be 0.23 g/t Au, with a resulting recovery of 81.4%. For scale-up purposes, this is discounted by 4% and an Adsorption-Desorption-Recovery (ADR) efficiency factor of 99% is applied, which results in a design recovery of 77%.

RECOVERY METHODS

The flowsheet for the Öksüt Project was derived from metallurgical test work primarily performed by Kappes, Cassiday & Associates (KCA). The test work results and interpretation are described in Section 13 of this report. The flowsheet is based on an 11,000 tpd heap leach operation. It includes primary crushing, screening and secondary crushing, heap stacking and cyanide leaching, carbon adsorption, carbon stripping and regeneration, electrowinning and refining. The concepts and data presented in this section are taken from the Process Flowsheets and Process Design Criteria developed for the Project.

PRIMARY CRUSHING

Run-of-mine ore will be delivered by 36 tonne haul trucks to the primary crusher. The ore will be dumped on the stationary grizzly installed over the 80 tonne truck dump hopper. Oversize rocks will be handled by a rock breaker. The ore will be withdrawn from the dump hopper via a 2.0 m wide x 4.5 m long grizzly feeder. The grizzly oversize will feed the 1.5 m x 2.0 m jaw crusher that will reduce the rock size to minus 150 mm prior to being conveyed by a 1.4 m wide x 95.5 m long belt conveyor to the secondary crushing circuit, along with the grizzly feeder undersize. A self-cleaning belt magnet will be installed over the conveyor belt feeding the secondary crusher building. A metal detector installed after the belt magnet will identify any remaining piece of metal and the conveyor will be stopped to allow manual removal by an operator.

SECONDARY CRUSHING

The product from the primary crushing circuit will feed a 2.4 m wide x 6.1 m long double-deck screen. The screen oversize will feed a 600 kW cone crusher while the screen undersize will report with the cone crusher product and will be transported by a 1.1 m wide x 50.7 m long belt conveyor to a radial stacker after quicklime has been added to the crushing circuit product. A 10,000 t capacity stockpile will be formed by the 1.1 m wide x 39 m long stacker installation.

Dust collection units will be provided at the crushers' discharge and transfer points in both crushing buildings and a dry fog system will be installed at the truck dump. If required, additional dust control measures could be implemented. A compressor will provide compressed air for process and instrumentation application. Fresh water will be distributed to the crushing area.

HEAP STACKING

The crushed ore will be trucked from the crushing facility to the heap leach pad (HLP). The leach pad will be developed in three phases. The HLP will be able to accommodate up to 40 Mt.

HEAP LEACHING

The heap will be irrigated with a diluted cyanide solution recirculated from the ADR plant, via a network of piping covering the surface area under leach. The barren leach solution will be pumped from the barren tank at the ADR plant to the area under heap leach. The cyanide concentration will be adjusted and the pH will be controlled so that HCN gas formation is inhibited. The solution will be filtered to remove carbon fines prior to distribution over the area under heap leach to minimize emitter plugging. It will be pumped by means of two centrifugal pumps installed in series. The first pump will cover operation for the first three years of operation, which is the end of Phase 1, while the second pump will be required from year four and beyond.

The irrigation distribution piping will consist of a 300 mm diameter main header made of carbon steel from the barren pumps discharge to the heap perimeter followed by high-density polyethylene (HDPE) ending at the ore panels to be irrigated. Drip emitters will be used to provide irrigation. A typical panel piping arrangement will include a 300 mm diameter HDPE header starting from the main header and running for 190 m along the 250 m side of the panel. It is proposed to use two lengths of 300 mm diameter HDPE header in rotation for irrigation, one in operation and the other one available for installation on the next section to be irrigated. Four lateral pipes spaced at every 62.5 m will be branched from the header. Each lateral pipe will include a 150 mm butterfly valve, a pressure gauge, and 75 m of a 150 mm diameter HDPE pipe followed by 75 m of a 100 mm diameter HDPE pipe. Emitter lines will be branched at every 500 mm on the pipes and emitters will be spaced at every 762 mm on the emitter lines.

The pregnant leach solution (PLS) will flow by gravity through a network of collection pipes at the base of the heap to the PLS pond prior to being pumped to the ADR plant for precious metals recovery.

WATER BALANCE AND SOLUTION MANAGEMENT

A probabilistic water balance model was developed for the HLP to simulate the performance of the facility. The objectives of the analysis were to evaluate the demand for makeup water from external sources and the volume of excess water generated during HLP operation (i.e., for use in event pond sizing). The water balance model was developed using GoldSim software.

The water balance model was applied to three different scenarios. Laboratory testing of the sample of post-column leached ore as well as work done by Golder Associates Ltd. (Golder) indicates a representative K_{sat} value may be within a range of 0.5 cm/s to 1.0 cm/s. Factors that may result in lowering the effective vertical hydraulic conductivity of the ore heap include the following: 1) surface trafficking (compaction) on top of previously placed ore, 2) breakdown of ore particles over time, 3) migration of fines particles, or 4) one lift (or a portion of one lift) placed with incrementally higher fines content.

CAPITAL COSTS

All pre-production expenditures and construction capital have been assumed to occur in Years -2 and -1 (2015-2016) and Q1 2017, with mining commencing in mid-2016 and, subsequent to commissioning, gold production commencing in Q2 2017. Total initial direct and indirect capital costs and pre-production operating costs are estimated to be $220.8M. This total includes all applicable Owner's costs, HLP, ADR, infrastructure, engineering, mining, working capital, and project management, with a contingency of 15% upon commencement of construction. The estimates are found in Table 1-4.

TABLE 1-4 PRE-PRODUCTION EXPENDITURES

Item	Total ($M)
Direct Costs	
Mining, including HLP Construction	51.2
Utilities	16.5
Infrastructure	22.0
Processing including Crushing and ADR	27.9
Water Handling	8.3
Working Capital	**10.2**
Total Direct Costs	**136.1**
Indirect Costs	
Owner's Costs	17.6
Permitting/Land Acquisition	8.2
Construction/Project Management	33.8
Contingency (15%)	25.1
Total Indirect Costs	**84.7**
Total Pre-production Expenditures	**220.8**

A closure cost of $27M has been estimated and included in the total capital cost.

OPERATING COSTS

Operating costs were developed from first principles for processing and general and administrative costs. Manpower lists have been developed for all areas, including administrative offices in Ankara. Power and reagent consumptions have been estimated based on test work and engineering work completed to date on the crushing facility, ADR plant, and HLP. Mining costs have been based on discussion with mining contractors in Turkey, with additional costs for contractor oversight, grade control, and mine planning estimated by Centerra. Additional stockpile rehandle costs have also been included. Various royalties have been applied to the Project and have been estimated assuming a gold price of $1,250/oz. Refining charges have also been included. Table 1-5 summarizes the operating costs by area.

TABLE 1-5 OPERATING COST SUMMARY

Area	Unit	Value
Processing	$/t processed	5.17
General & Administrative	$/t processed	2.50
Mining[1]	$/t mined	2.34
Royalties	$/t processed	1.36
Refining	$/t processed	0.11

[1] Mining includes ore re-handling costs and $30M of capitalized stripping.

ECONOMIC ANALYSIS

The material economic assumptions such as operating and capital cost estimates used for the calculations presented in this section are summarized in Tables 1-4 and 1-5.

CASH FLOW FORECAST

Using a price of gold of $1,250 per ounce, as assumed for the Mineral Reserve estimation process, the open pit LOM plan (Table 1-2), the processing schedule (Table 1-3), and the capital and operating cost forecasts (Tables 1-4 and 1-5, respectively), an estimate of the after-tax free cash flow for the Öksüt Project has been made. As shown in Tables 1-6 and 1-7, the total after-tax undiscounted free cash flow is $435.9M whereas the after-tax free cash flow discounted at 8% is $242.0M. The internal rate of return (IRR) of the Project is 42.5% after taxes. The payback period on construction capital and pre-production expenditures is expected to be two and half years after production commences.

The after-tax free cash flow includes, among other costs, all pre-production costs, Environmental and Social Impact Assessment (ESIA) costs, and permitting costs. Additional exploration work including tasks budgeted for 2016 have not been included in the Project financial analysis as it is assumed that any such work would be done to increase the currently defined Project resource. Exploration expenses incurred during 2014 and 2015 have been applied to the financial model for the purposes of taxation.

Capital costs include the construction of the power line and deposits related to forestry and land pasture fees. These deposits will be returned to Centerra over the LOM. In the case of the power line, it has been assumed that the cost will be recovered by a series of five equal payments, paid yearly, beginning in 2017 and ending in 2021. The forestry and land pasture deposits have been assumed to be refunded at the end of closure and reclamation activities in 2025.

TABLE 1-6 PROJECT LIFE OF MINE – CASH FLOW

	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Sales and Revenue													
Gold Price	(US$/oz)	**1,250**	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	-
Gold Produced (Sold)	(oz x 1000)	**895.2**	-	-	90.0	179.2	199.4	151.2	92.5	106.3	69.7	6.9	-
Gold Revenue	($ x 1000)	**1,119,024**	**-**	**-**	**112,494**	**223,974**	**249,295**	**189,034**	**115,620**	**132,885**	**87,112**	**8,610**	**-**
Operating Costs													
Mining	($ x 1000)	**137,075**	-	-	23,462[2]	27,239	18,141	19,530	27,578	16,392	4,733	-	-
Processing	($ x 1000)	**135,133**	-	-	17,530	20,566	20,566	20,566	18,403	20,566	16,936	-	-
Administration	($ x 1000)	**67,379**	-	-	10,715[2]	8,687	8,691	8,695	8,539	8,700	8,352	5,000	-
Royalties	($ x 1000)	**35,626**	-	-	2,616	6,093	8,138	6,882	4,007	4,667	2,946	277	-
Refining	($ x 1000)	**2,990**	-	-	300.6	598.5	666.1	505.1	308.9	355.1	232.8	23.0	-
Direct Costs	($ x 1000)	**378,203**	**-**	**-**	**54,624**	**63,184**	**56,202**	**56,178**	**58,836**	**50,680**	**33,200**	**5,300**	**-**
Capital and Other Costs													
Capitalized Stripping (Cash)	($ x 1000)	**30,260**	-	-	6,321	2,758	10,569	10,612	-	-	-	-	-
Construction Capital	($ x 1000)	**179,172**	19,448	140,529	19,195	-	-	-	-	-	-	-	-
Contingency	($ x 1000)	**25,150**	252	17,605	7,293	-	-	-	-	-	-	-	-
Sustaining Capital[1]	($ x 1000)	**9,685**	-	-	-	3,112	2,913	-	-	3,660	-	-	-
Working Capital	($ x 1000)	**0**	-	817	9,405	8,927	2,333	(4,931)	(6,298)	1,908	(3,581)	(8,580)	**-**
Closure Cost	($ x 1000)	**27,000**	-	-	2,714	5,404	6,015	4,561	2,790	3,206	2,102	208	**-**
Total Capital & Other[2]	($ x 1000)	**271,267**	**19,700**	**158,951**	**44,928**	**20,201**	**21,830**	**10,242**	**(3,508)**	**8,774**	**(1,479)**	**(8,372)**	**-**
Pre-production Expenditures [2]	($ x 1000)	**220,785**	**19,700**	**158,951**	**42,134**								
Cash Flow													
Pre-Tax Cash Flow	($ x 1000)	**469,554**	**(19,700)**	**(158,951)**	**12,942**	**140,590**	**171,263**	**122,614**	**60,292**	**73,431**	**55,391**	**11,682**	-
Income Tax Payable	($ x 1000)	**45,994**	-	-	36	2,617	3,263	13,300	7,009	11,320	8,449	-	-
Gov. Refunds[2]	($ x 1000)	**12,336**	-	-	2,195	2,195	2,195	2,195	2,195	-	-	-	1,361
Free Cash Flow	($ x 1000)	**435,896**	**(19,700)**	**(158,951)**	**15,101**	**140,168**	**170,195**	**111,509**	**55,478**	**62,111**	**46,942**	**11,682**	**1,361**
Cumulative Free Cash Flow	($ x 1000)	**-**	**(19,700)**	**(178,651)**	**(163,550)**	**(23,382)**	**146,813**	**258,322**	**313,800**	**375,911**	**422,855**	**434,535**	**435,896**
All-in Sustaining Cost[1] per oz sold	$/oz	**490**	-	-	638	416	380	472	666	541	507	800	-
All-in Cost[1] per oz sold	$/oz	**725**	-	-	1,002	416	380	472	666	541	507	800	-
All-in Cost[1] per oz produced including taxes	$/oz	**777**	-	-	1,002	430	396	560	742	648	628	800	-

Notes:
1. Non-GAAP measure, see discussion under "Non-GAAP Measures" in Centerra's Management's Discussion and Analysis for three and six months ended June 30, 2015.
2. In 2017, pre-production expenditures include $2.0M of administration costs and $4.3M of mining costs and excludes capitalized stripping and closure costs.

TABLE 1-7 UNDISCOUNTED AND DISCOUNTED AFTER-TAX FREE CASH FLOW

Discount Rate	Net Present Value ($M)
0%	435.9
5%	301.7
8%	242.0
10%	208.8

Note: Gold Price: US$1,250/oz

TAXATION AND ROYALTIES

The corporate income tax rate in Turkey is 20%, however, Investment Incentive Certificates (IIC) are available to provide reduced corporate tax rates for profits derived from investments made in Turkey to promote economic development. The Öksüt Project economic analysis assumes that an IIC will be obtained.

Based on the scope and the amount of the planned investment, it is expected that a Strategic IIC will also be granted to the Project. If the Öksüt Project receives a Strategic IIC, it will be eligible for the additional following benefits:

- Further reduction of corporate income tax rate
- VAT exemption
- Customs duty exemption
- Government support for interest payments on loans which are registered to the investment incentive certificate
- Government support for employer's share of social security premiums
- VAT refund support

For the economic analysis, only the impact of the corporate income tax rate reductions under the IIC and the Strategic IIC has been included.

Gold production from the Project is also subject to three royalties, as follows:

1. A Turkish Government State royalty levied as a percentage of revenue less certain qualifying operating costs. The percentage is determined on a sliding scale that is linked to the price of gold and may be reduced by 50% if the material is further processed in a refinery in Turkey (as assumed in the economic analysis). Based on

the assumed gold price of $1,250/oz, an effective royalty rate of 2% was used for the economic analysis.

2. A 1% NSR royalty payable to Stratex Gold up to a maximum of $20M; and
3. A sliding scale NSR royalty payable to Teck based on the cumulative ounces produced over the LOM. Centerra has estimated this royalty to be 0.6% of total gold revenues.

The total effective royalty rate used for the economic analysis, at an assumed gold price of $1,250/oz, is 3.6%.

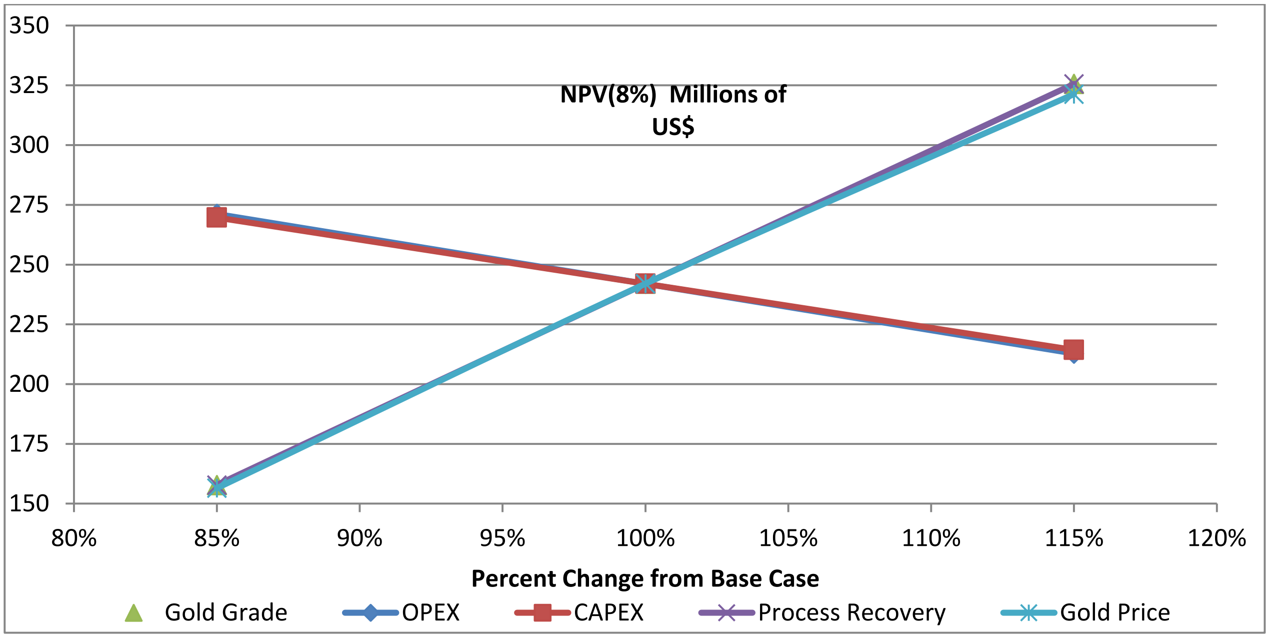
SENSITIVITY ANALYSIS

Table 1-8 provides after-tax free cash flow forecasts for the Öksüt Project based on the current LOM plan and a gold price of $1,250 per ounce. The table also shows the sensitivity of the Project NPV to gold prices ranging from $1,050 to $1,450 per ounce on $100 increments and on discount rates ranging from 0 to 10%. Sensitivities to four other variables at the base case gold price and a discount rate of 8% are also shown on a ±15% basis.

TABLE 1-8 CASH FLOW SENSITIVITIES (MILLIONS OF US$)

Sensitivity to Gold Price				
Discount Rate	**0%**	**5%**	**8%**	**10%**
Gold Price ($/oz)				
$1,050	298.2	195.6	150.5	125.5
$1,150	367.0	249.0	196.7	167.6
$1,250	**435.9**	**301.7**	**242.0**	**208.8**
$1,350	497.0	348.4	282.1	245.2
$1,450	567.1	400.3	326.7	285.5
Sensitivities to other Variables at $1,250 per ounce of gold and an 8% discount rate				
Variable	**Operating Costs**	**Capital Costs**	**Gold Grade**	**Process Recovery**
+15%	212.8	214.3	325.6	325.6
Base Case	**242.0**	**242.0**	**242.0**	**242.0**
-15%	271.2	269.7	157.8	157.8

FIGURE 1-1 CASH FLOW SENSITIVITIES



The LOM $NPV_{8\%}$ is most sensitive to changes in the gold grade and process recovery under an upside scenario (15% increase) followed by the gold price and then by operating and capital costs respectively. An increase of 15% to the gold grade or process recovery increases the Project $NPV_{8\%}$ by $83.6M whereas a gold price increase of 15% results in an additional $79.3M to the Project $NPV_{8\%}$.

When considering the downside scenario, the Project is most sensitive to the gold price followed by gold grade or process recovery and then by operating and capital costs respectively. A 15% reduction in the gold price reduces the Project $NPV_{8\%}$ by $85.5M. Similarly, a reduction in the gold grade or process recovery of 15% reduces the Project $NPV_{8\%}$ by $84.2M.

At a gold price of $767 per ounce and an 8% discount rate, the Project is cash flow neutral, however, it meets all foreseen capital and operating expenses including cash taxes. At gold prices higher than $767 per ounce, the Project has the potential to generate positive free cash flow.

CONCLUSIONS AND RECOMMENDATIONS

CONCLUSIONS

- At June 30, 2015, the Öksüt database contained results of 330 diamond and reverse circulation (RC) drill holes for a total of 82,006 m. Centerra concludes that the data, the data density, and the additional information from the Öksüt database are adequate to form the basis for a Mineral Resource estimate.

- A site visit to the Öksüt Project reviewed property and deposit geology, exploration and drilling methods and results, sampling method and approach, sample and data handling, including chain of custody. A qualified geologist evaluated the compilation of quality assurance/quality control (QA/QC) data from the Öksüt Project and is of the opinion that the sample preparation, security, and analytical procedures used by OMAS and prior companies followed industry-standard procedures and the resulting analytical data are acceptable for use in resource estimation.

- Drilling to date has been completed at an average drill hole spacing of 50 m along and across strike over known mineralized zones, the majority of which has been classified as Measured or Indicated Mineral Resources. Drilling has not completely defined the limits of the mineralization at the Güneytepe Deposit, and the mineralized zones remain open along strike to the north.

- Centerra estimates the current Measured and Indicated Mineral Resources at the Öksüt Project to be 33.0 Mt at an average grade of 1.2 g/t Au containing 1.3 million ounces Au and Inferred Mineral Resources to be 2.4 Mt at an average grade of 0.8 g/t Au containing 64,000 ounces Au.

- In order to comply with the CIM Definitions requirement of "reasonable prospects for eventual economic extraction", a preliminary Whittle pit shell was used to constrain the Mineral Resource estimate using process recovery of 77%, a gold price of $1,450, and other assumed pit parameters. Only blocks located within the pit shell are reported in the Mineral Resource estimate.

- The mineralization at the Öksüt Project that is included in this analysis comes from two separate deposits to be mined by open pit methods. This mineralization is amenable to conventional open pit, loader/truck mining methods that will be carried out by a mining contractor.

- Utilizing the updated Measured and Indicated Mineral Resources, pit optimization was carried out on both the Keltepe and Güneytepe Deposits, using updated geotechnical parameters, updated operating costs, metallurgical recovery estimates, and a gold price of $1,250 per ounce. The resulting pit shells were used as a guide in completing engineered pit designs which include ramps suitable for 36 tonne haul trucks consistent with other contractor mining operations in Turkey. This generated an estimated Probable Mineral Reserve of 26.1 Mt at 1.4 g/t Au containing 1.2 million ounces of gold at a cut-off grade of 0.3 g/t Au.

- Test work carried out on representative samples of the deposits indicates that the Öksüt Project mineralization can be treated with heap leach methods. The HLP is designed to process oxide ore from two open pits.

- Metallurgical testing identified an unexpected variability in ore metallurgical properties which requires follow-up analysis and may impact the overall recovery estimate.

- The proposed Öksüt process plant design will be based on gold heap leach technology, which will consist of crushing, stacking, and heap leaching with cyanide solution, gold adsorption from the pregnant solution onto carbon, carbon elution, and gold smelting.

- Financial analysis of the Öksüt Project has determined that the Project will be economically viable and profitable.

- The FS is based on the open pit mining of the Keltepe and Güneytepe Deposits. The target ore production rate during the life of the mine is 4.0 Mt per year.

- Using a price of gold of $1,250 per ounce, the total after-tax undiscounted free cash flow is $435.9M, whereas the after-tax free cash flow discounted at 8% is $242.0M. The internal rate of return of the Project is 42.5% after taxes and payback is in 2.5 years.

- Centerra concludes that the FS demonstrates the viability of the Öksüt Project as proposed, and that further development is warranted.

RECOMMENDATIONS

The key recommendation is to proceed with the development of the Öksüt Project in 2015 with the target of achieving initial gold production in Q2 2017.

Additional recommendations include:

- Implement a database management system that is capable of supporting production sampling.

- Complete an independent audit of the assay database after implementation of a database management system.

- Perform additional density measurements within barren material outside of the existing mineralization.

- Survey the remaining five GPS surveyed holes with differential global positioning system (DGPS).

- Create an alteration model using inductively coupled plasma (ICP) data as the basis for a geo-metallurgical recovery model.

- Evaluate the impact of using shorter composite lengths for grade estimation.

- Perform a contact plot analysis to assess the need for soft, semi-soft, or hard estimation boundaries for high grade and low grade material.

- Based on the completed geo-metallurgy recovery model, identify any gaps in the samples tested to date. Using either available samples or samples from new drill holes, complete additional leach tests.

- Conduct additional testing utilizing an updated alteration model to provide guidance for sample selection in order to manage the unexpected variability in ore metallurgical properties identified in the KCA Phase 3 testing program.

- Complete a detailed leach plan and schedule utilizing the updated geo-metallurgy recovery model.

- Complete additional geotechnical drilling for final HLP, waste dump, and other surface infrastructure designs, to provide enough data for detailed engineering.

- As mining begins, data from pit wall mapping should be incorporated into the geotechnical model and pit designs adjusted accordingly.

- Confirm, or re-visit, the hydrogeological assumptions after mining commences. If there are material changes to the water table assumptions, the slope stability analyses should be updated.

- Sensitivities to mining losses and dilution should be quantified. As mining begins, these assumptions should be monitored and adjusted, if necessary.

- Due to the climate characteristics at the Project site, the ponds are expected to be empty throughout most of the year, with the majority of inflows concentrated in the months of March and April. Assessment of options for protecting the geomembrane from exposure to natural environment would need to be considered during detailed design (e.g., covering the geomembrane with a protective layer of sand or gravel).

2 INTRODUCTION

Centerra Gold Inc. (Centerra) has prepared a Technical Report on the Öksüt Gold Project (the Öksüt Project or the Project), located in the Kayseri province of south-central Turkey. The Technical Report supports the disclosure of the Project's Mineral Resource and Mineral Reserve estimates at June 30, 2015 and has been prepared in accordance with National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101). All dollar amounts in this Technical Report are stated in United States dollars (US$) unless otherwise noted.

Centerra is a North American-based gold mining and exploration company engaged in the operation, exploration, development, and acquisition of gold properties in Asia, Canada, and other markets worldwide. Centerra is a publicly listed company on the Toronto Stock Exchange.

In August 2009, Centerra formed a joint venture with Stratex International Plc (Stratex), a publicly listed company on the London-based Alternative Investment Market (AIM), for the Öksüt Project through its now wholly-owned Turkish subsidiary, Öksüt Madencilik Sanayi ve Ticaret A.Ş. (OMAS). In October 2012, Centerra increased its interest in the joint venture to 70% after completing total expenditures of $6M on exploration. In January 2013, Centerra purchased Stratex's remaining 30% interest for a cash payment of $20M and a 1% net smelter return (NSR) royalty capped at $20M. In addition to the royalty payable to Stratex's wholly-owned subsidiary Stratex Gold AG (Stratex Gold), the Project is also subject to a sliding scale NSR royalty in favour of Teck Resources Limited (Teck) as well as a sliding scale royalty in favour of the Turkish Government. Closing of the purchase was also conditional on the granting of two Operation Licences by the Turkish authorities, which occurred on January 16, 2013. Centerra now owns 100% of the Project.

The Öksüt Project was first discovered in 2007 when geological staff of Stratex identified gold mineralization in reconnaissance rock chip sampling from outcrops located on what is now referred to as the Güneytepe Deposit area. Prior to this, there is no record of any modern exploration for gold being conducted on the property. In 2008, Stratex followed up these surface sampling results with an initial 13-hole diamond drilling program located within the Güneytepe Deposit area.

Following the formation of the joint venture, several additional drilling programs were completed on the property resulting in the first Centerra publication of a Mineral Resource estimate on the Project in February 2013 (effective at December 31, 2012) subsequently followed by a revised Mineral Resource estimate published in February 2014 (effective at December 31, 2013). On February 19, 2014, Centerra announced the results of a Preliminary Economic Assessment (PEA). An updated Mineral Resource estimate was published in February 2015 (effective at December 31, 2014). On July 28, 2015, Centerra announced the results of a Feasibility Study (FS) on the Öksüt Project.

The Öksüt Project is planned as a conventional truck and shovel open pit mine. A total of approximately 26.1 Mt of ore at a grade of 1.4 g/t Au, containing a total of approximately 1.2 million ounces of gold, is planned to be mined and processed by heap leach method over a mine life of eight years from two open pits, the Keltepe and the smaller Güneytepe pit.

SOURCES OF INFORMATION

This Technical Report was prepared by Centerra, OMAS, and Centerra Madencilik Sanayi ve Ticaret A.Ş. (CMAS) personnel. The dates of personal inspections of the Project by the Qualified Persons are provided in Section 29 of this Technical Report.

The Qualified Persons and their responsibilities for this Technical Report are listed in Table 2-1.

The documents reviewed, and other sources of information, are listed at the end of this report in Section 27 References.

TABLE 2-1 QUALIFIED PERSONS AND RESPONSIBILITIES

QP	Title/Company	Primary Areas of Responsibility	Report Sections Authored
Gordon Reid, P.Eng.	Vice-President and COO, Centerra	Overall Supervision	All sections
Peter Woodhouse, P.Eng.	Director, Projects, Centerra	Conclusions and recommendations, market studies, other relevant data and information	Sections 1, 2, 3, 18, 19, 24, 25, and 26.
Malcolm Stallman, MAIG	Regional Exploration Manager, Western Asia and Eastern Europe, CMAS	Geology, exploration, drilling, sample preparation and analysis, data verification	Sections 4, 5, 6, 7, 8, 9, 23, and parts of Sections 10, 11, and 12.
Mustafa Cihan, MAIG	Chief Geologist, OMAS	Geology, exploration, drilling, sample preparation and analysis, data verification	Sections 4, 5, 6, 7, 8, 9, 23, and parts of Sections 10, 11, and 12.
Pierre Landry, P.Geo.	Corporate Geologist, Evaluation and Development, Centerra	Mineral Resource estimate	Section 14 and parts of Sections 10, 11, and 12.
Tommaso Roberto Raponi, P.Eng.	Director, Metallurgy, Centerra	Metallurgical testing and mineral processing	Sections 13 and 17 and parts of Section 21.
Tyler Hilkewich, P.Eng.	Corporate Mining Engineer, Centerra	Mineral Reserve estimate and mining methods	Sections 15 and parts of Sections 16 and 21.
Kevin D'Souza, CEng	Vice-President, Sustainability & Environment, Centerra	Environmental studies, permitting, and social and community impact	Section 20
Chris Sharpe, P.Eng.	Senior Mining Engineer, Centerra	Economic analysis	Section 22

ABBREVIATIONS AND UNITS OF MEASUREMENT

A list of abbreviations is provided in Table 2-2. The units of measurement used in this report conform to the metric system unless otherwise indicated. The currency used in this report is US dollars (US$) unless otherwise noted. An exchange rate of US$1: 2.5 TL was used to convert US dollars to Turkish lira.

TABLE 2-2 LIST OF ABBREVIATIONS

a	annum	µg	microgram
A	ampere	µS/cm	microsiemens per centimetre
°C	degree Celsius	m	metre
cm	centimetre	M	mega (million); molar
cm/s	centimetres per second	m^2	square metre
d	day	m^3	cubic metre
ft	foot	m^3/h	cubic metres per hour
g	gram	masl	metres above sea level
G	giga (billion)	mg	milligram
g/L	gram per litre	min	minute
g/t	gram per tonne	µm	micrometre
ha	hectare	mm	millimetre
hr, h	hour	Mt	million tonnes
Hz	hertz	MW	megawatt
in.	inch	oz	Troy ounce (31.1035g)
k	kilo (thousand)	ppb	part per billion
kg	kilogram	ppm	part per million
km	kilometre	RL	relative elevation
km^2	square kilometre	s	second
km/h	kilometre per hour	t	metric tonne
kPa	kilopascal	t/m^3	tonnes per cubic metre
kt	thousand tonnes	TL	Turkish lira
kW	kilowatt	tpa	metric tonne per year
kWh	kilowatt-hour	tpd	metric tonne per day
L	litre	US$ or $	United States dollar
L/s	litres per second	V	volt
µ	micron	W	watt
		wt%	weight percent

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by Centerra, OMAS, and CMAS. The information, conclusions, opinions, and estimates are based on:

- Information available at the time of this report, including Centerra's internal Feasibility Study (FS) of the Project, and
- Assumptions, conditions, and qualifications as set forth in this report.

The authors have relied, and believe they have a reasonable basis to rely upon the following individuals who have contributed to the legal, political, environmental, and tax information stated in this report, as noted below:

- Alper Sezener, Director, External Affairs and Sustainability, OMAS, with respect to legal matters in Sections 4 and 20.
- Rajeev Hampole, Director, Taxation, Centerra with respect to taxation matters in Sections 4 and 22.

The date of these contributions is June 30, 2015.

The authors of this Technical Report have reviewed the information provided by the other experts as listed above and, based on the authors' review of this information, believe it to be reasonable and reliable.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

The Öksüt Project is located in south-central Turkey, 295 km to the southeast of the capital city of Ankara and 48 km directly south of the city of Kayseri which has a population of 1.1 million. The nearest administrative centre is at Develi (population 64,000) located approximately 10 km north of the Project. Ankara and Kayseri have international airports and are serviced by international and domestic airlines. The Project's co-ordinates are 715000-722100 Easting and 4236500-4249300 Northing (UTM ED 50 zone 36) (Figures 4-1 and 4-2).

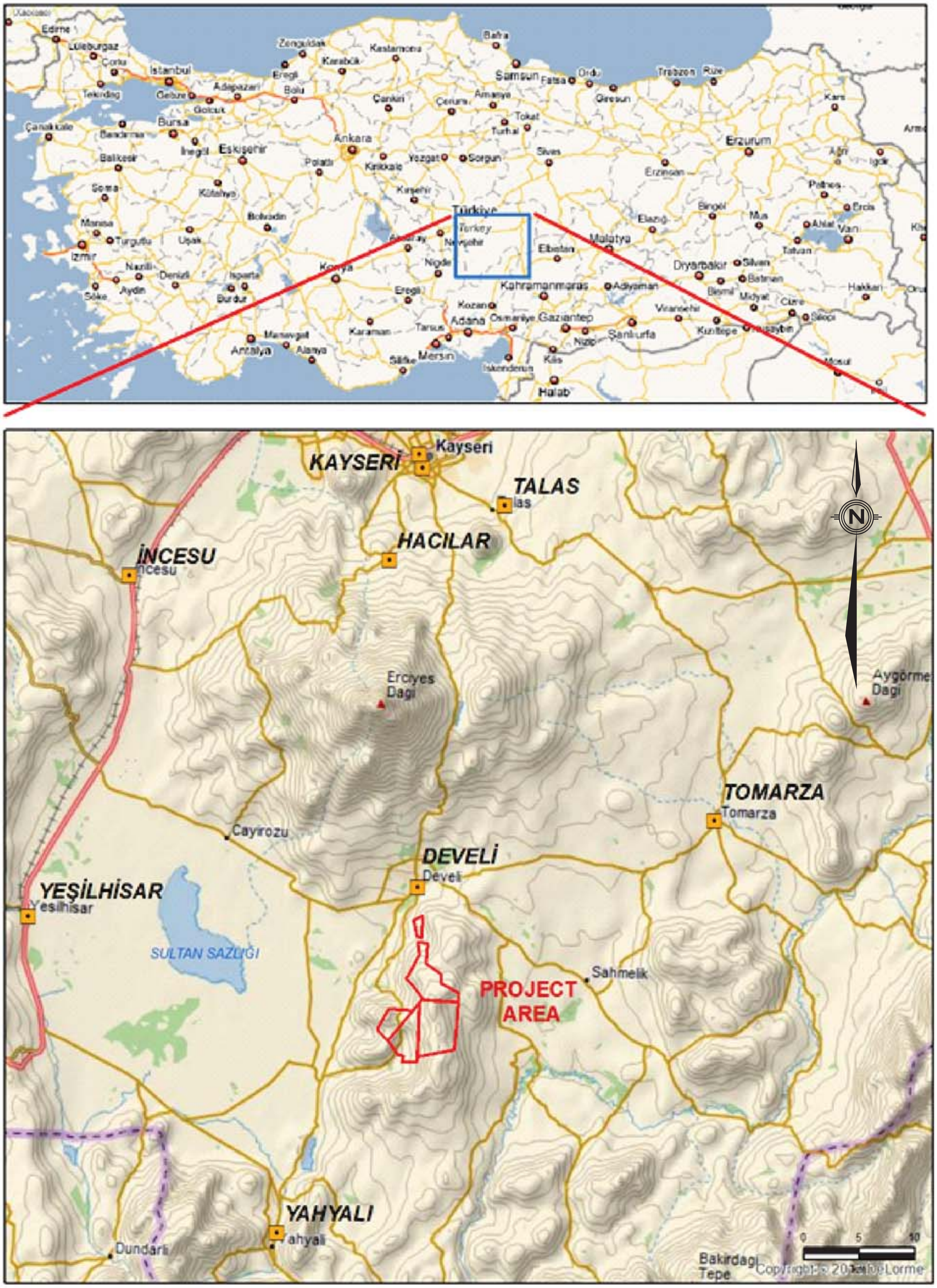
The Project is located in the Develi Mountains on a north-south trending topographic high. The topographic relief comprises steep-sided V-shaped valleys, and locally, cliffs tens of metres high, capped by flat-lying mesas and plateaus. The Project site is located at an elevation of approximately 1,800 m. The valleys are extensively farmed, with the local population living in a number of small villages including the villages of Öksüt and Zile.

LEGAL FRAMEWORK

Mining rights and minerals are exclusively owned by the state. The state delegates rights to explore and operate to Turkish individuals or legal entities through set period licences in return for royalty payments. Local and foreign investors can acquire the same mining rights, however, foreign investors need to establish a Turkish company to do so, which can be 100% foreign owned.

Mining licensing is regulated by the General Directorate of Mining Affairs (MIGEM), a unit of the Ministry of Energy and Natural Resources. Other institutions of importance are central government ministries, the provincial administration, and local government institutions.

Mining is primarily governed by the 1985 Mining Law no. 3213, which was most recently amended in June 2010. This latest amendment aims to provide a more investment friendly environment, particularly for exploration projects, as well as bringing aspects such as environmental protection and health and safety more in line with international practice.



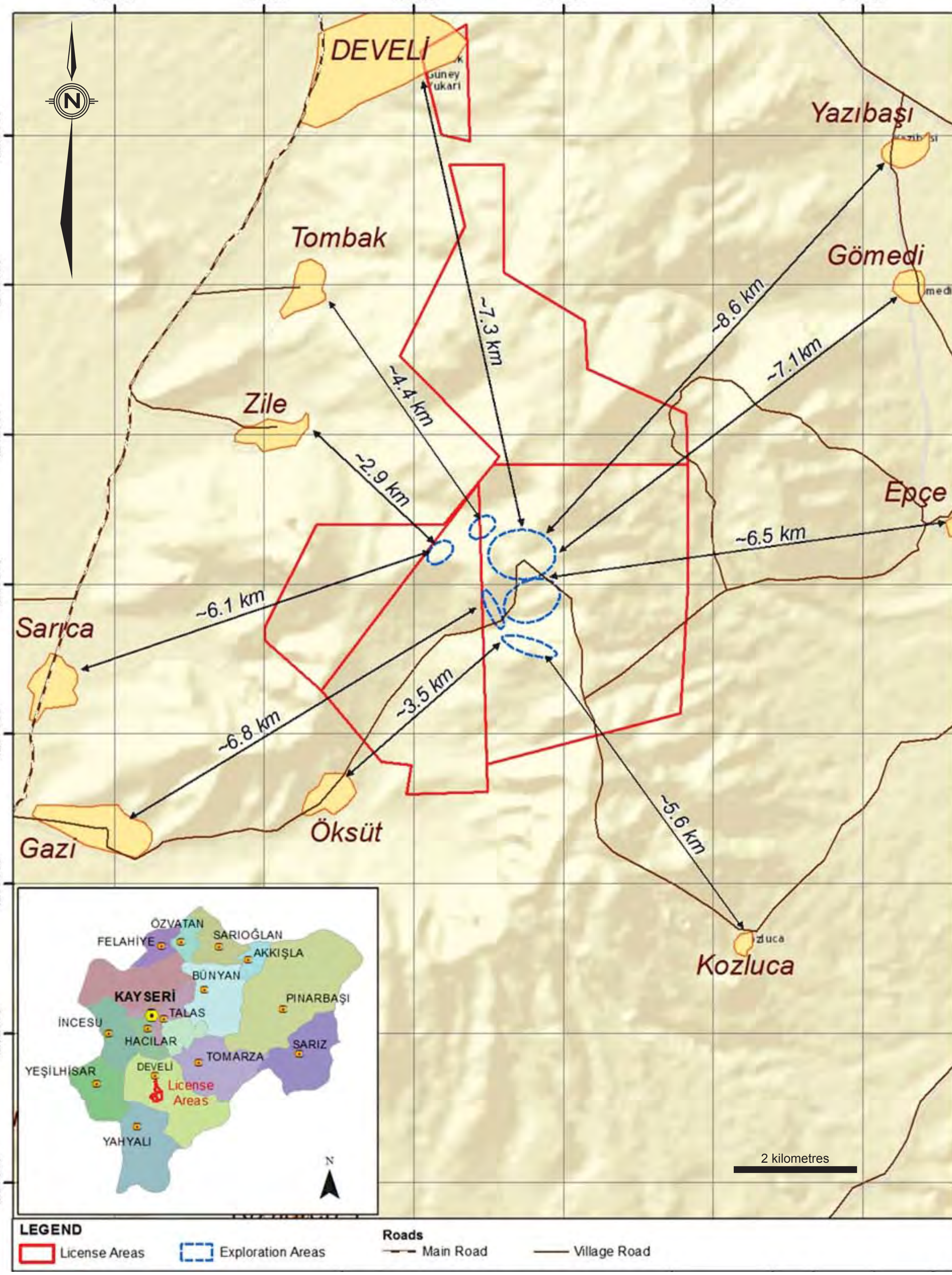
Centerra Gold Inc.

Öksüt Gold Project, Turkey
2015 Technical Report

Project Location (General)

Date: Sept. 2015	File: ProjectLocation.cdr	Figure 4-1

Source: SRK Consulting Ankara, 2014b



Source: SRK Consulting Ankara, 2014b

Centerra Gold Inc.

Öksüt Gold Project, Turkey
2015 Technical Report

Project Location (Detailed)

Date: Sept. 2015	File: ProjectLocationDetailed.cdr	Figure 4-2

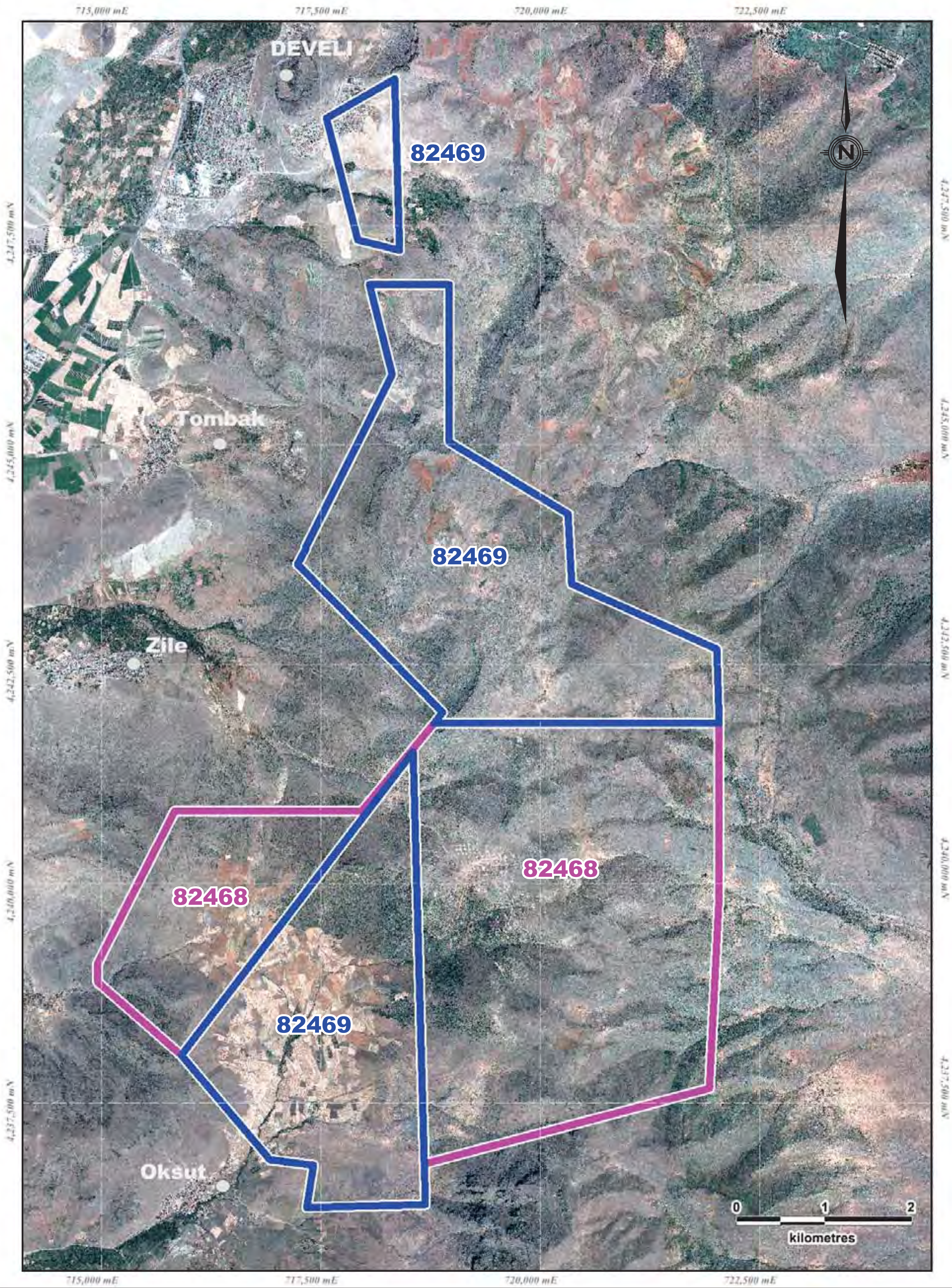
MINERAL TENURE

The Öksüt Project land position consists of two contiguous Operation Licences (numbers IR 82468 and 82469 – the Licences) with a total area of 3,995.81 ha (Table 4-1 and Figure 4-3). Mineral rights under these licences have been granted to OMAS. According to the Mining Law, OMAS has the right to explore and develop any Mineral Resources contained within the Operation Licences, provided fees and taxes are paid in order to keep the licences in good standing. The Licences were issued on January 16, 2013 and are currently set to expire on January 15, 2023.

TABLE 4-1 SUMMARY OF LICENCES

Licence No.	Access No.	Type	Area (ha)	Expiry Date	Owner
82468	3 298 759	Operation	1,999.86	16.01.2023	OMAS
82469	3 298 736	Operation	1,995.95	16.01.2023	OMAS
			3,995.81		

While OMAS has the right to explore and develop within the area covered by the Operation Licences, it must first complete an Environmental Impact Assessment (EIA) Report (as further described in Section 20) before permits for development of the Project (including those that grant surface rights to the Project area) can be obtained.



centerragold

Centerra Gold Inc.

Öksüt Gold Project, Turkey
2015 Technical Report

Mineral Tenure

Date: Sept. 2015	File: LicenseAreas.cdr	Figure 4-3

SURFACE RIGHTS

LAND OWNERSHIP

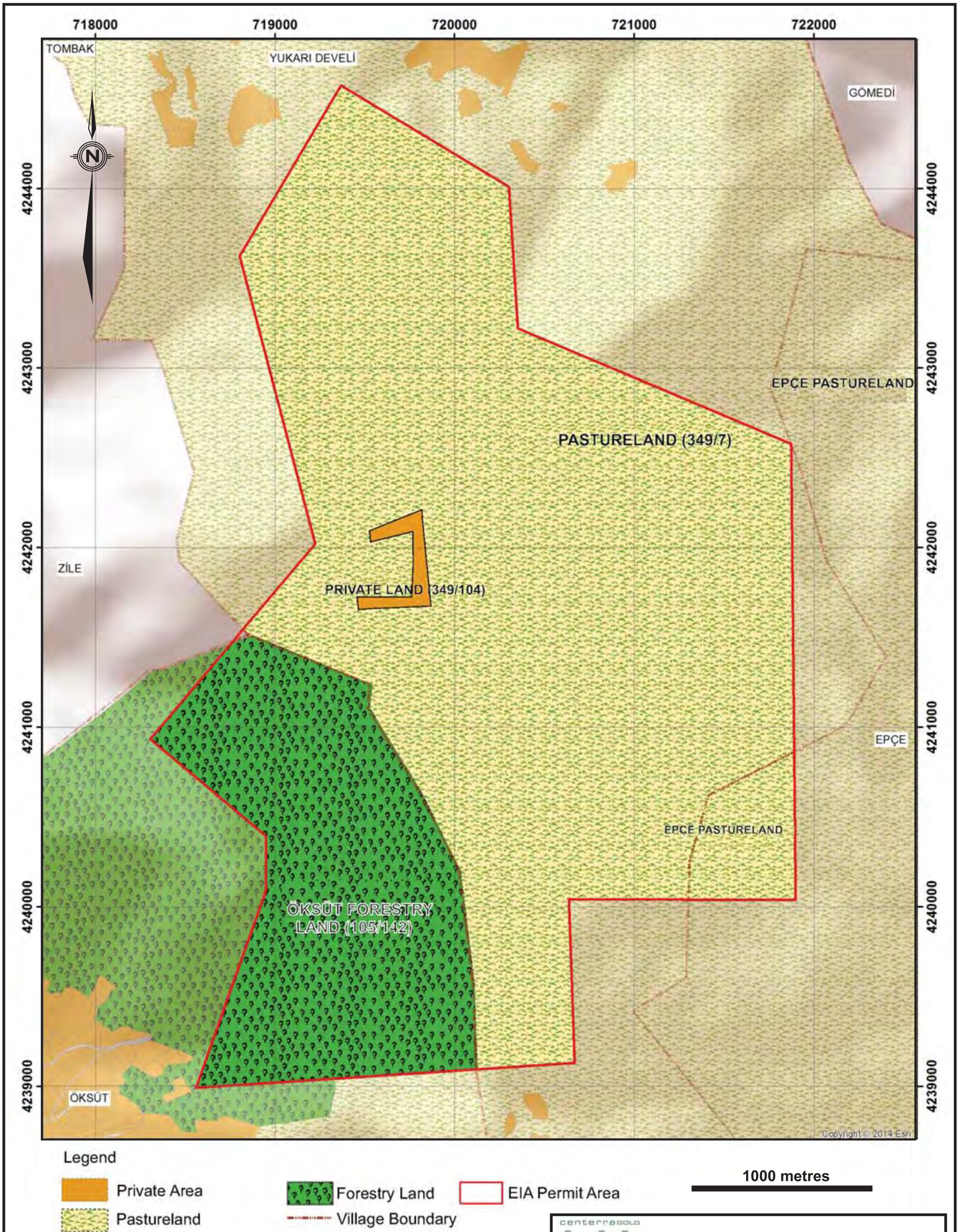
As noted above, OMAS currently holds two Operation Licences relating to the Project. Within the areas covered by the Operation Licences, a fence line zone has been demarcated for the purposes of permitting under the EIA Report (EIA Fence Line Zone). Once OMAS has obtained approval of the EIA Report, it will be allowed to apply for permits to conduct development, operations, and other activities within the EIA Fence Line Zone.

There are three different types of land located within boundaries of the EIA Fence Line Zone: forestry, pasture, and private. Both forestry land and pasture land are considered government lands. According to the Mining Law and Regulations, land title transfers of government lands may be sought following approval of the EIA Report. There is only one parcel of private land covering approximately 80,000 m^2 which is in the middle of the EIA Fence Line Zone (Figure 4-4). No roads or other facilities at the Project are expected to impact these private lands. Negotiations with owners of private land have begun and are continuing.

In addition, to lands contained within the EIA Fence Line Zone, acquisition of additional lands has been considered for the purposes of obtaining water, power and road access to the Project. Based on the results of fresh water studies completed by Golder Associates Ltd. (Golder) between November 2014 and February 2015, two locations (9,200 m^2 and 5,000 m^2), which are private land, were determined to be suitable water resources for mine usage. OMAS has purchased these lands and the water wells have been permitted.

Following a preliminary review of the ownership of lands underlying proposed access roads for the Project, it was determined that there are pasture, government, and private lands on the route. OMAS is currently in the process of negotiating with local private landowners for the acquisition or use of such private lands.

Power line related negotiations have been conducted with the Turkish Electricity Transmission Company (TEIAŞ) and according to applicable laws, the acquisition of lands and other rights on the power line route will be carried out by TEIAŞ.



Centerra Gold Inc.

Öksüt Gold Project, Turkey
2015 Technical Report

Mine Site Land Ownership

Date: Sept. 2015	File: OwnerShip.cdr	Figure 4-4

ROYALTIES

The royalties applied to the Öksüt mine operations are a Turkish Government State royalty, a net smelter return (NSR) royalty payable to Stratex Gold, and a NSR royalty payable to Teck. The royalty to Stratex Gold is a 1% NSR on all precious metals extracted from Öksüt, net of refining charges, up to a maximum of $20M. The royalty to Teck varies depending on the amount of gold ounces sold as shown in Table 4-2.

TABLE 4-2 ROYALTY TO TECK BASED ON GOLD SOLD

Total Gold Sold (koz)	Royalty to Teck
0-250	0%
250-750	0.5%
750-1,250	1.0%
> 1,250	1.5%

The Turkish Government State sliding scale royalty for gold and other metals is shown in Table 4-3.

TABLE 4-3 TURKISH GOVERNMENT STATE SLIDING SCALE ROYALTY

Gold price ($/oz)	Royalty
<800	2%
801-1,250	4%
1,251-1,500	6%
1,501-1,750	8%
1,751-2,000	10%
2,001-2,250	14%
>2,251	16%

ENVIRONMENTAL LIABILITIES

To the QP's knowledge, there are no known environmental liabilities at the Project site.

REQUIRED PERMITS

Applications for all permits required for the operation of the Project will be made following completion and approval of the EIA Report. The process and timeline for the EIA Report are further described in Section 20.

Required permits before the construction and operation phases are listed below:

- **EIA Permit Approval** (General Directorate of EIA Permit and Monitoring under the Ministry of Environment and Urbanization)**:** Reclamation Plan Approval, Domestic Solid Waste Agreement, Medical Waste Agreement, Noise Control Permit, Emission Permit, Discharge Permit, Waste water Treatment Approval, DSI Approval (General Directorate of Government Water Affairs), Meteorology approval (General Directorate of Meteorology), National Parks and Nature Conservation Approval (General Directorate of National Parks and Nature Conservation), Culture and Tourism approval (Provincial Directorate of Culture and Tourism)

- **Land Access Permits:** Forestry Lands Permit, Pasture Lands Permit, Treasury Lands Permit, Private Lands Permit (if required)

- **Non-hygienic Establishments Licence (GSM Licence),** City Governorship and Municipality

- **Water Permits:** Water Usage Permit, Water Well Drilling Permit, Water Pipeline/Land Access Permit (from General Directorate of State Hydraulic Works)

- **Energy and Powerline Permits** (Requirements: Land Access Permits and EIA Approval), in cooperation with TEIAŞ

- **Construction Permit** (Requirements: Land Access Permits and EIA Approval), from Provincial Directorate of Public Works and Municipality

- **Access Road Permit** (Requirements: Land Access Permits and EIA Approval), from Provincial Directorate of Highway

- **Explosive Permits** (Explosive Purchase and Usage Permit, Explosive Magazine Permit), Governorship, Provincial Directorate of Security

- **Operational Security Services** (Security of Camp and Facilities), Governorship

To the QP's knowledge, except as set out in this Technical Report, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

Currently, travel to the Project is via the paved dual lane road from Kayseri southwards to Develi over the eastern flank of Mt Erciyes. From there, the road connects Develi to the town of Gömedi, 33 km to the south-southwest, and then via the minor access roads to the villages of Zile. The Project site is accessed via a narrow agricultural dirt road from the village of Zile and as a result, access to the site in winter is presently limited.

CLIMATE

The climate is continental with cold, snowy winters and warm, dry summers with cool nights due to the area's high elevation. According to the 48 year temperature measurements made by the Develi Meteorological Station, the annual mean temperature in the region is 11˚C. Temperature ranges from 10˚C to 29˚C in summer and -5˚C to 7˚C in winter. The mean temperature is highest at 22.9˚C in July and lowest at -1˚C in January. The annual average total amount of precipitation is approximately 355 mm, generally falling as mixed rain and snow in winter and as rain during spring. The prevailing wind direction in the region is north-northeast. The operations of the Project are expected to occur throughout the year.

LOCAL RESOURCES

The nearest administrative centre is at Develi (population 64,000) located approximately 10 km north of the Project.

The Develi area is well-serviced with electrical power lines. The Project is expected to be supplied from a substation 28 km away, near the town of Sendrimeke. The proposed all weather access to the Project is from Epçe. Water for the Project will be supplied from two wells located just south of the village of Epçe. These wells have been fully tested, and the permits and land rights are now in hand. Preliminary reviews of the region suggest that an abundant level of mining support services and skilled to semi-skilled labour available to support a potential mining operation.

INFRASTRUCTURE

There is currently no existing infrastructure on the lands covered by the Operation Licences due to the fact that the appropriate permits for such development will only be applied for following final approval of the EIA Report (see Section 20).

For a description of potential waste disposal areas, heap leach pad areas and potential processing plant areas, please refer to Figure 16-7.

PHYSIOGRAPHY

The Project is located in the Develi Mountains on a north-south trending topographic high. The topographic relief comprises steep-sided V-shaped valleys, and locally, cliffs tens of metres high, capped by flat-lying mesas and plateaus. The Project site is located at an elevation of approximately 1,800 m. Vegetation at higher elevations is predominantly scrub oak, small shrubs, and grasses.

6 HISTORY

The Öksüt Gold Project was discovered by Stratex in early 2007. Reconnaissance rock chip sampling returned up to 0.113 g/t Au from silica ledges within altered andesitic volcanic rocks at what is now the Güneytepe Deposit. In late 2007, Stratex made applications for tenements to cover the property and obtained a total of nine contiguous exploration licences covering an area of 111.6 km^2.

In 2007 and 2008, Stratex carried out geological mapping, rock chip and channel sampling, soil sampling, a topographical survey, and acquired Quickbird high-resolution satellite imagery for an area of 5.0 km by 4.5 km over the Project. Prior to this, there is no record of any modern exploration for gold conducted on the property. More specific information relating to historical exploration is provided in Section 9 of this Technical Report.

In 2009, Stratex and Teck agreed that Teck would relinquish its rights under a 2004 strategic alliance agreement to acquire interests in projects owned by Stratex. In exchange, Teck received shares of Stratex and a sliding scale royalty on, among others, the Öksüt Project as described above in Section 4. Centerra and Stratex subsequently formed a joint venture in 2009, to explore the Project. Centerra earned an initial 50% equity in the Project by advancing $3M to the joint venture through October 2011 and acquired an additional 20% interest in the Project in October of 2012 with an additional contribution of $3M, which brought its equity interest to 70%. In January 2013, Centerra purchased Stratex's remaining 30% to own 100% of the Öksüt Gold Project in exchange for a cash payment of $20M and a 1% NSR royalty up to a maximum of $20M.

Stratex carried out three historical Mineral Resource estimates for the Project. The most recent estimate, prepared in 2012 to the JORC Code, includes a total of 1,047,872 ounces of gold in the Ortaçam North (now Keltepe) and Ortaçam (now Güneytepe) zones at a cut-off grade of 0.2 g/t Au. This historical Mineral Resource estimate is relevant, however, it has been superseded by the current Mineral Resource estimate in Section 14.

7 GEOLOGICAL SETTING AND MINERALIZATION

GEOTECTONIC AND REGIONAL GEOLOGICAL SETTING

The Öksüt Project is a high-sulphidation epithermal gold system located in Central Anatolia, Turkey. It lies within part of the Neogene-Quaternary Central Anatolian Volcanic Province (CAVP) which is part of the highly mineralized Tethyan Metallogenic Belt (TMB), a continuous belt of Mesozoic and Cenozoic deformation that extends from Spain to Southeast Asia.

The Project is hosted within the Develidağ Volcanic Complex (DVC) of late Miocene age, which is one of the numerous stratovolcanoes situated along the Central Anatolian Fault Zone (CAFZ). The DVC is surrounded by pre-Miocene (probably Paleozoic) basement rocks of the Central Anatolian Crystalline Complex (CACC) and also Quaternary volcanics and sediments. The CACC rocks and the Neogene-Quaternary volcanics in Central Anatolia are considered to be segments of the TMB formed due to the convergence between the Afro-Arabian and Eurasian Plates, i.e., as a result of the closure of the Tethyan Ocean during the Alpine-Himalayan orogeny in the Late Cretaceous/Early Tertiary.

The collision between the Afro-Arabian and Eurasian Plates accreted many micro-continental blocks. This prolonged collision and the consequent subduction and accretion resulted in the formation of three major tectonic units separated by suture zones in Turkey: the Pontide Orogenic Belt to the north, the Anatolide-Tauride Platform in the middle, and the Arabian Platform in the southeast (Ketin, 1966). The Pontides are separated by the İzmir-Ankara-Erzincan Suture Zone from the Anatolide-Tauride Platform. The CACC is regarded either as the northern part of the Anatolide-Tauride Platform or as a distinct terrane, bounded by the Inner Tauride Suture Zone. The Arabian Platform is separated by the Bitlis-Zagros Suture Zone from the Anatolide-Tauride Tectonic Unit (Figure 7-1).

Drawing Modified After, Okay & Tuysuz, 1999 and Ketin, 1966

IntroductoryFigures.Wor/Geotectonic

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Tectonic Map of Turkey
Major Sutures and
Continental Blocks

Date: Sept. 2015	File: IntroductoryFigures.wor	Figure 7-1

After the closure of the northern branch of the Neo-Tethyan Ocean along the İzmir-Ankara-Erzincan Suture Zone, the northward subduction of the Afro-Arabian Plate below the Eurasian Plate began in the Early-Middle Miocene to the south of Cyprus. This subduction zone is considered to have triggered the formation of extensive calc-alkaline Neogene volcanism throughout Anatolia and including the CAVP. Meanwhile, the southern branch of the Neo-Tethyan Ocean, extending from southeastern Turkey to Cyprus, continued its evolution through the late Middle Miocene and then closed entirely during continent-continent collision along the Bitlis-Zagros Suture Zone. Such suturing between the Arabian and Eurasian Plates was followed by continued convergence in eastern Turkey until the Early Pliocene. Continuing northward movement of the Arabian Plate relative to the African Plate resulted in the tectonic escape of the Anatolian Plate to the west along the East Anatolian Fault (EAFZ) and North Anatolian Fault Zones (NATFZ) during the Late Miocene-Early Pliocene (Bozkurt, 2001). This episode is referred to as a neotectonic period in which a new compressional-extensional regime commenced in Turkey.

During this neotectonic period, Central Anatolia underwent complex deformation characterized by the development of intra-continental transtensional fault systems and the formation of tectonic depressions and basins. During this time, many faults and intra-continental basins in this region were either newly formed or were reactivated.

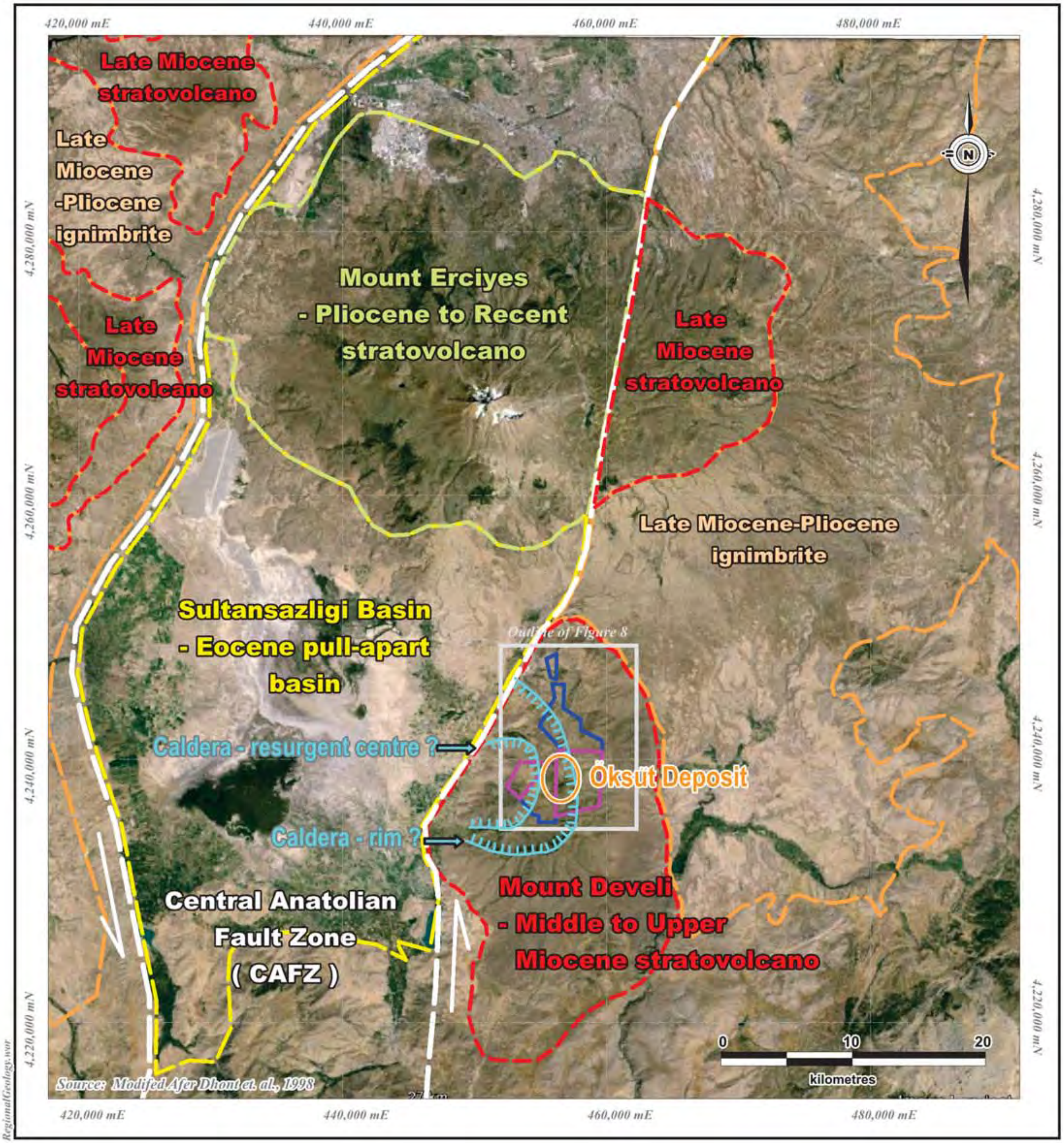
The NE-SW trending CAFZ has been the major structure shaping Central Anatolia's tectonic/geologic setting since the Late Miocene. The CAFZ is a left-lateral transcurrent fault system with about 60 km of offset along its strike. In its central part it is characterized by transtensional depressions formed by left stepping and southward bending of the fault zone. The Sultansazlığı pull-apart basin is one of these and has been active since the Late Pliocene. With its rhomboidal shape, it displays characteristic morphologic features including steep and stepped margins, large alluvial and colluvial fans, and hosts a large stratovolcano of Quaternary age, Erciyes Mountain, located 30 km to the north of Öksüt.

While the CAFZ and its basins were evolving, they were reshaping the development of the CAVP in this part of Central Anatolia. In this region, the CAVP developed within a system of successive tectono-volcanic extensional depressions starting in Middle-Late Miocene. It extends as a belt about 300 km long in a NE-SW direction. It is suggested that the CAVP in this area evolved in three major periods (Pasquare et al., 1988). The first period occurred in the Middle-Late Miocene and is represented by the activity of effusive centres as well as of endogenous domes of andesitic composition as observed at Öksüt. The second period includes deposition of andesitic-basaltic lava and pyroclastics during the Late Miocene-Pliocene. The last period is recorded by the growth of large andesitic-basaltic stratovolcanoes such as the Erciyes and Hasandağ composite volcanoes.

REGIONAL AND LOCAL GEOLOGY AND DEPOSIT EVOLUTION

The Öksüt Project lies on the eastern flank of the Sultansazlığı Basin within the eroded stratovolcano of the DVC that was controlled and truncated by segments of the CAFZ. The eroded DVC is hosted within the Paleozoic metamorphic basement, which was uplifted along the basin shoulders during extension (Figure 7-2). The DVC is composed of Miocene basaltic-andesitic volcanic domes, pyroclastics (lithic tuff and volcanic breccia), and lava flows. Flow-banded Pliocene andesite overlies these sequences to the north and east. The majority of the eroded volcano is covered by the licences of the Öksüt Project (Figure 7-3).

Although there has been no radiometric dating done on the DVC, stratigraphic correlation with similar andesitic to basaltic rocks in the region suggests that the andesitic rocks at the core of the eroded caldera are the oldest volcanic rocks. K/Ar age dating of similar andesitic rocks within the CAVP (Keçikalesi Volcanics) near Niğde, approximately 120 km to the WSW of Öksüt, are dated at 13 million years (Besang et al., 1977). Hence, the andesitic volcanics at the Öksüt Project are considered to have a similar age, i.e., Middle Miocene.



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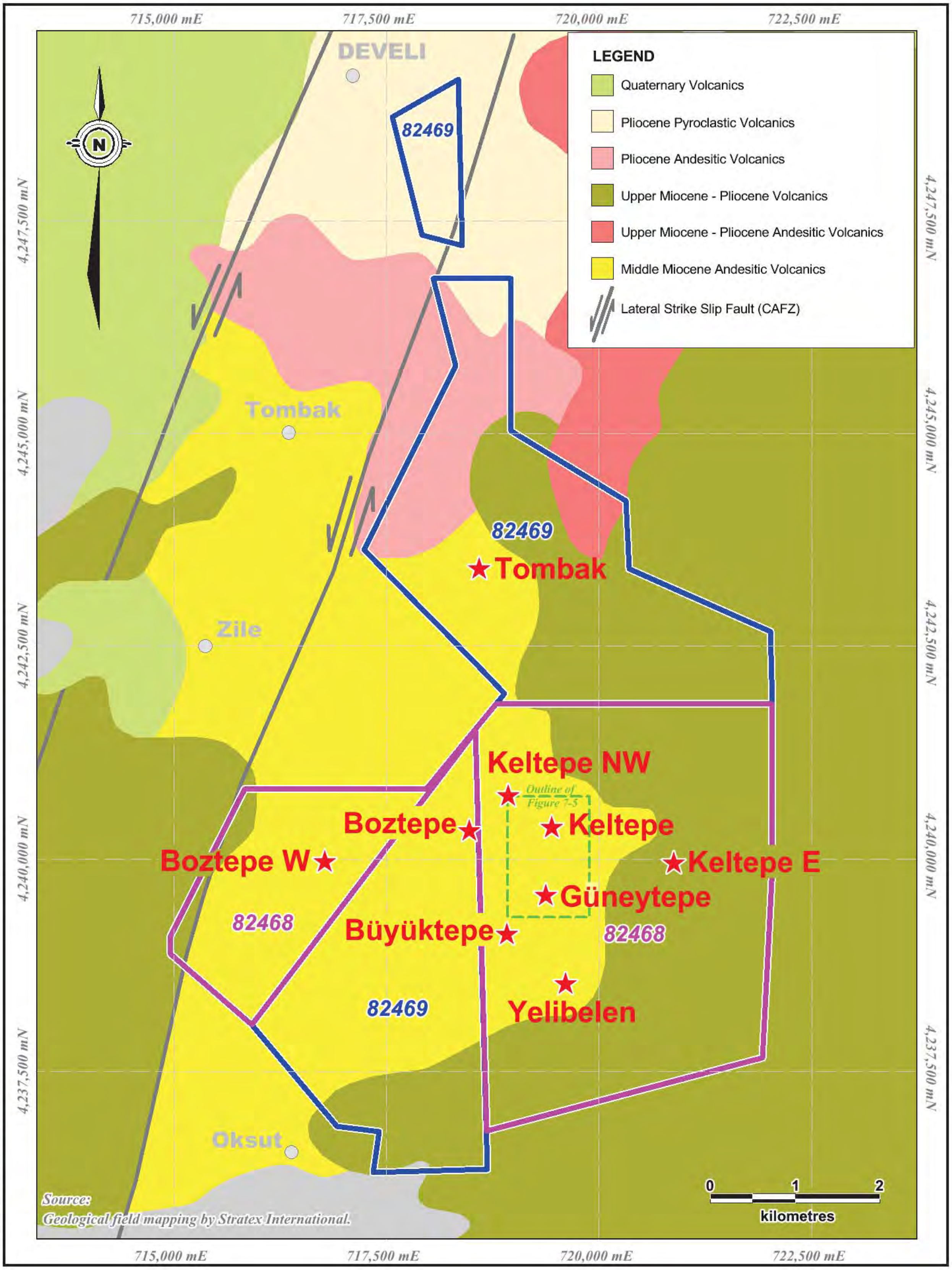
Regional Geology

Date: Sept. 2015	File: RegionalGeology.cdr	Figure 7-2



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Geological Map of the Project Area and Prospect Locations

Date: Sept. 2015	File: ProjectAreaGeology.cdr	Figure 7-3

Middle Miocene-aged andesitic volcanics outcrop at the core of the DVC. These are stratigraphically overlain by Upper Miocene-Lower Pliocene flow-textured andesitic-basaltic volcanics intercalated with pyroclastic units. The overlying andesitic-basaltic lavas and pyroclastics dip away from the core at 10° to 30°, forming a prominent cliff up to 300 m high from the central area. To the east of the Keltepe and Güneytepe Deposits, andesitic-basaltic lavas dip gently to the east and cap the older rocks at an altitude of 2,074 m.

The overlying Late Miocene-Early Pliocene units transition upwards into the Pliocene pyroclastics and undifferentiated volcanics to the north and east of the Öksüt Project. To the west, the DVC is covered by Pliocene to Holocene volcanics and Quaternary alluvial deposits of the Sultansazlığı Basin.

Trace element plots of geochemical analyses indicate a calc-alkaline origin for the andesitic rocks at Öksüt (after Cabanis and Lecolle, 1989). However, previous geochemical studies on fresh Late Miocene-Early Pliocene basaltic rocks of the DVC (Kürkcüoğlu, 2010) indicates a mainly tholeiitic origin derived from partial melting of a depleted mantle source. This suggests partial melting of a subducting plate which has caused emplacement of andesitic domes in the Late Miocene. The reactivation of the CAFZ and the formation of extensional basins occurred synchronously with the second phase of volcanism that followed the earlier phase in the Late Miocene-Early Pliocene. Whether gold was introduced into the system during the emplacement of the andesitic domes or during the latter volcanism is unclear. However, supergene gold enrichment may have occurred during the latter phase when fast uplifting-erosion and deep weathering were occurring along the eastern flank of the Sultansazlığı Basin.

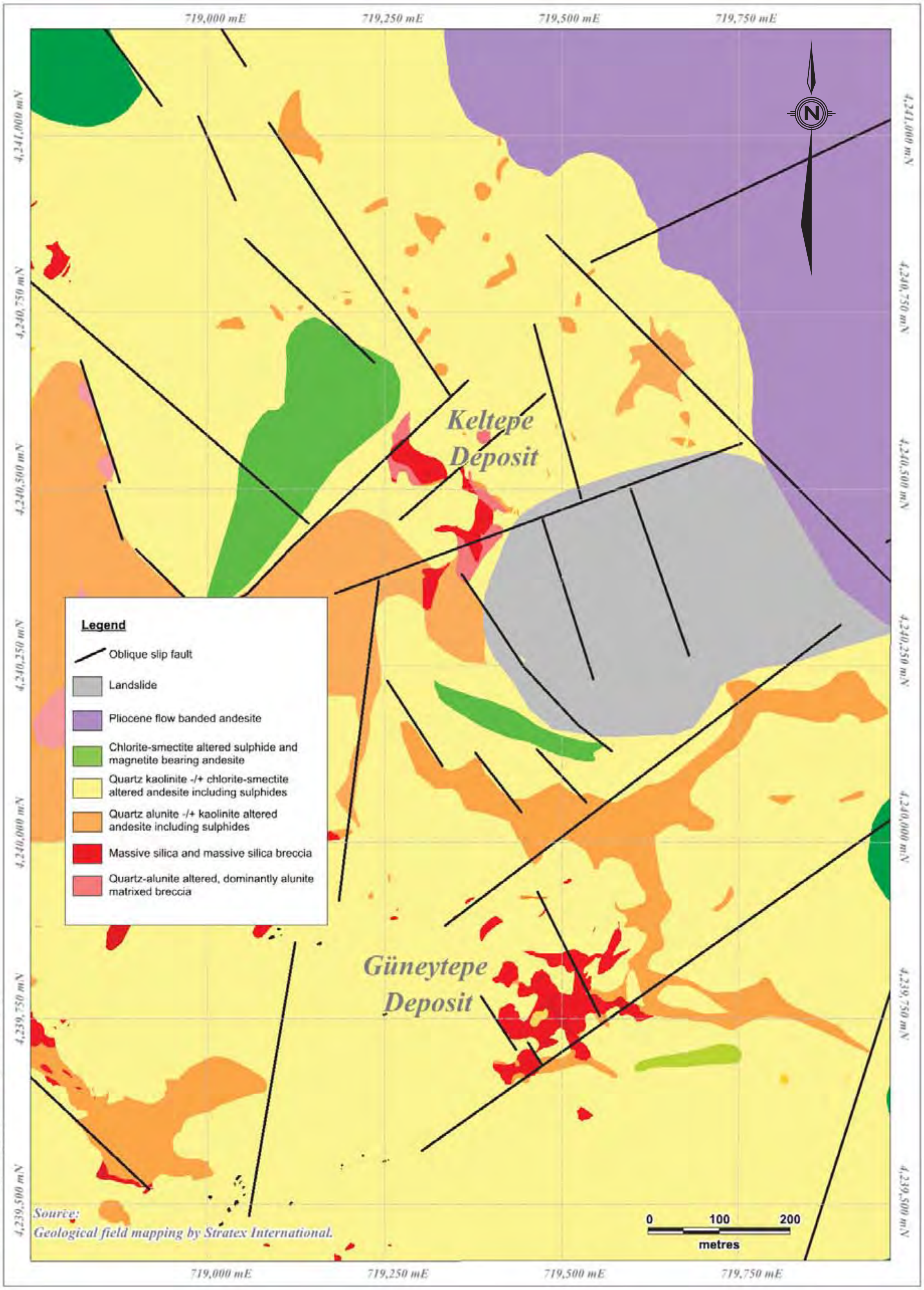
The Miocene to Pliocene units in the vicinity of the Öksüt Project are displaced or distorted by oblique faults that have dominant normal and strike-slip movements. The amount of net-slip or displacement along these faults is difficult to determine as they truncate each other, suggesting repetitive cycles of faulting and/or reactivation. These faults are mainly oriented NE-SW and NW-SE. Structural observations by OMAS staff suggest that NE-SW faults have a dominant strike-slip movement whereas NW-SE faults have a dominant normal displacement. These faults are interpreted to have played an important role in gold deposition as they caused further brecciation and thereby provided multiple pathways for hydrothermal fluids.

Although the andesitic rocks that host the gold mineralization at Öksüt are generally oxidized, brecciated and highly altered, original textures, including porphyritic characteristics and layering, are still preserved in places where weaker alteration or relict breccia clasts are present. In relatively fresh or weakly altered rocks outside of the mineralized breccia bodies, the andesite is porphyritic with phenocrysts of plagioclase feldspar, and rare, altered biotite and pyroxene, within a fine, quartz-plagioclase feldspar-ilmenite groundmass. Apart from these minerals, in most places disseminated fine grained pyrite is also present together with magnetite. The gold mineralization is hosted by a polymictic breccia, which is interpreted to fill a sub-volcanic vent/diatreme rather than being a sub-aerial volcaniclastic sequence, and to be either phreatomagmatic or phreatic in origin.

PROPERTY GEOLOGY AND MINERALIZATION

The Öksüt mineralization is associated with a high-sulphidation epithermal, disseminated gold system hosted within the mid-Miocene age andesitic volcanic dome complex of the DVC. Gold mineralization is closely associated with massive and vuggy silicic alteration and advanced argillic alteration, occurring within large areas of argillic alteration. Silicification occurs in association with multiple phases of hydrothermal breccias with silica/limonite cement and silica/alunite cement, quartz-alunite, quartz-kaolinite, patchy silica and opal/chalcedony, and displays local zonation both vertically and laterally. Gold mineralization is believed to occur as finely disseminated particles in the oxide zone and in association with pyrite, chalcocite, covellite, tennantite-tetrahedrite, and enargite in the sulphide zone (Lashko, 2013).

Nine mineralized zones have been identified at Öksüt (Figure 7-3). Mineral Resources have been estimated for two of these, Keltepe and Güneytepe (Figure 7-4).



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Keltepe and Güneytepe Simplified Geology

Date: Sept. 2015	File: KG_SimpleGeol.cdr	Figure 7-4

KELTEPE DEPOSIT

GEOLOGY AND STRUCTURE

The Keltepe Deposit is elongated NNW-SSE and is approximately 600 m long and 350 m wide with a minimum known vertical extent of 450 m. Two principal rock types are present: a texturally diverse variety of polymictic breccias and a texturally uniform porphyritic andesite (Sillitoe, 2012). The principal host rocks to the gold mineralization are multiple phases of polymictic hydrothermal breccia. The breccia clasts display marked size variability, ranging from ~10 cm through a typical 1-2 cm to highly comminuted rock flour. Locally, monomictic blocks, as large as 25 m, occur within the breccia. The clasts are typically angular to sub-rounded in form and set in a highly altered, fine-grained, clastic matrix. The alteration is too intense and pervasive to determine if a tuff component is present in the matrix. In places, the medium- to fine-grained breccia is laminated and shows size sorting.

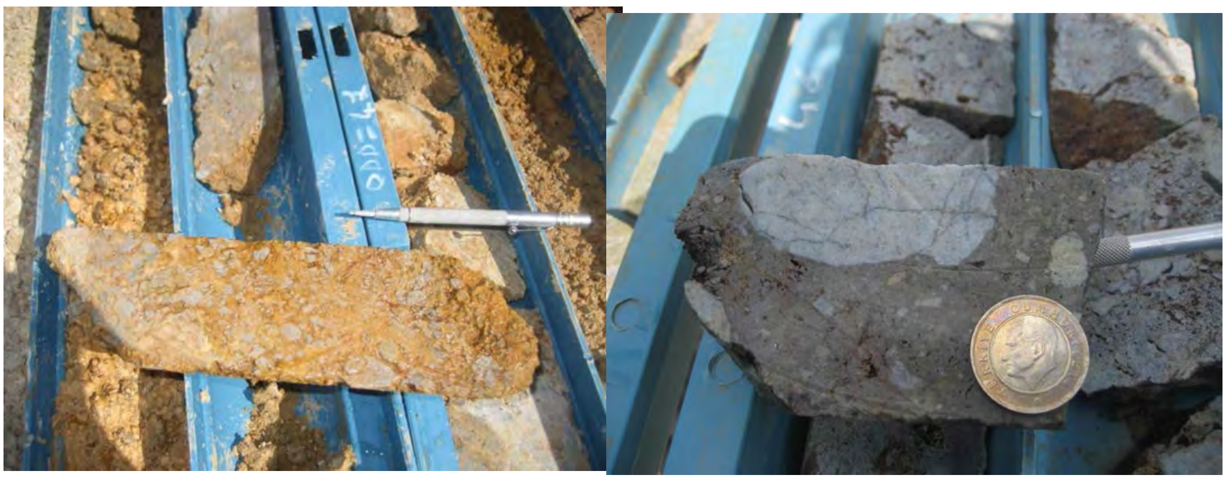
The multiple breccia varieties are erratically distributed and not readily correlated, at least over distances in the order of 100 m. The breccias contain clasts that were either altered or unaltered at the time of incorporation. The latter include andesitic volcanic rocks and uncommon siltstone, locally finely bedded. The altered clasts include pervasively silicified rock, vuggy residual quartz, and quartz-alunite. These siliceous lithologies, which constitute approximately 20% of the breccias, have a variety of white to grey shades and textures that confirm appreciable admixture during the brecciation process (Sillitoe, 2012).

Among the multiple phases of breccias, four main types have been identified based on their cement type (Kurtuluş, 2012). These breccias are: alunite breccia set in alunite-limonite cement; quartz-kaolinite ± alunite breccia set in a quartz-kaolinite ± alunite cement; silica-hematite breccia set in a fine-grained silica-hematite-limonite matrix; and massive silica breccia set in massive silica-pyrite cement.

Alunite breccia is composed of heterolithic clasts set in quartz-alunite-limonite cement (Figure 7-5). Clasts are massive, angular to sub-angular, and their sizes range from a few millimetres up to 10 cm. Alunite breccia is clast-supported to locally matrix-supported. It forms zones from a few centimetres up to 30 m to 40 m in width. Alunite breccia generally transitions laterally and vertically into quartz-kaolinite ± alunite breccia.

Massive silica breccia is composed of monomictic clasts set in a massive silica-limonite-pyrite matrix (Figure 7-5). Clasts are massive silica, angular and their size ranges from few millimetres up to 3 cm. Both clasts and matrix may include disseminated pyrite ± chalcocite. Massive silica breccia texture resembles crackle breccia. It is predominantly matrix supported and forms zones from 50 cm to tens of metres in width.

FIGURE 7-5 ALUNITE BRECCIA (LEFT) AND MASSIVE SILICA BRECCIA (RIGHT)



The silica-hematite-limonite breccia is composed of polymictic clasts set in fine-grained hematite-silica-limonite cement (Figure 7-6). Clasts are massive silica and quartz-alunite altered, sub-angular to sub-rounded, and their size ranges from a few centimetres up to 15 cm. The silica-hematite-limonite breccia is predominantly clast supported and forms zones from 50 cm to 10 m in width.

FIGURE 7-6 SILICA-HEMATITE-LIMONITE BRECCIA AND QUARTZ-KAOLINITE ± ALUNITE BRECCIA

The quartz-kaolinite ± alunite breccia is composed of heterolithic clasts with quartz-kaolinite ± alunite cement (Figure 7-6). Clasts are quartz-kaolinite, quartz-alunite, and massive silica altered, sub-angular to sub-rounded, and their size ranges from 2 cm to 20 cm. The quartz-kaolinite ± alunite breccia is generally coarse grained and predominantly matrix supported. Zones of quartz-kaolinite ± alunite breccia range from a few centimetres up to 15 m in width.

Andesite porphyry, considered to be part of an andesitic flow/dome complex, surrounds and partially overlies the breccias, particularly along the eastern and northern margins of the Keltepe Deposit. The andesite is altered, but most of it more weakly so than the breccias. The overlying andesite is up to ~100 m thick and has a flat to shallowly inclined base. The andesite is porphyritic with phenocrysts of plagioclase feldspar, and rare, altered biotite and pyroxene, within a fine, quartz-plagioclase feldspar-pyrite-ilmenite groundmass.

Texturally identical porphyritic andesite also occurs very locally at depth within the breccia, where it is tentatively interpreted as being blocks of lesser altered material amongst the breccia bodies. A few narrow andesite dykes have been logged at depth at Keltepe and these contain xenoliths of silicified rock, which supports their emplacement during the mineralization event.

Gold mineralization at the Keltepe Deposit is interpreted to be controlled by NW-SE trending dip-slip faults and NE-SW trending dominantly strike-slip faults (Figure 7-4). The latter fault set seems to overprint the NW-SE faults. However, due to reactivation, NW-SE faults are also seen to offset the younger NE-SW faults in places. The outcrop expressions of the faults (or fault zones) may be seen by displacements and/or offsets at the surface and fault geomorphology. The attitude and extent of faults were traced and checked with the aid of drill hole data, ground magnetic geophysical data and, occasionally, with 2D induced polarization (IP)/resistivity geophysical data cross-sections. The contrast across significant fault zones may be clearly seen on 2D IP/resistivity cross-sections. In core samples, faults and/or fault zones are identified by highly broken intercepts, core loss, fault gouge, and fault breccia. Fault breccias may be differentiated from hydrothermal breccias by textural differences. Fault breccia clasts overprint hydrothermal breccias and they have clean angular boundaries, especially when associated with the younger NE-SW trending fault zones.

Gold-bearing hydrothermal fluids are interpreted to have used vertical and/or sub-vertical structures as feeder zones and to have also moved laterally within favourable lithologies before precipitating gold mineralization. Given the multiply brecciated and highly fractured nature of the host rocks, hydrothermal fluid pressures are thought to have been very high.

During subsequent rapid uplift/exhumation and erosion, vertical stress would have been relieved and this is interpreted to have resulted in the formation of horizontal or shallow-dipping (10^{o}-20^{o}) sheeted joints, which are common within some parts of the deposit.

ALTERATION

Four hydrothermal alteration types are recognized from Keltepe: massive silicification and associated vuggy residual quartz, quartz-alunite, quartz-alunite-kaolinite, and quartz-kaolinite. The latter includes patches of hypogene smectite-chlorite alteration as well as supergene kaolinization.

Massive silica and lesser vuggy silicification dominate the core of the mineralization at the Keltepe Deposit. It comprises a porous body of silicified breccia admixed with vuggy residual quartz, the product of hydrothermal leaching of all original rock constituents except silica under low-pH conditions. The porosity is contributed to by hypogene cavities that characterize the vuggy quartz and open space produced by the nearly complete supergene oxidation and removal of sulphide minerals, chiefly pyrite, and associated native sulphur. A ghost-like breccia texture is generally still visible because of the prominence of the silicified and vuggy quartz clasts that were incorporated in the breccia prior to the pervasive silicification and vuggy quartz development. Narrow, massive, silicified, and hematite-stained breccia zones have been observed at depth and these are interpreted as being hydrothermal feeder zones to the shallower siliceous bodies.

Alteration transitions outwards from the siliceous core into pervasive quartz-alunite altered breccia. Quartz-alunite alteration also affects the basal parts of the overlying andesite. The quartz-alunite alteration, like the siliceous core, is highly porous, reflecting both its original granular texture and the nearly complete supergene removal of disseminated sulphide minerals, mainly pyrite. Sugary-textured, almost sandy, quartz-alunite alteration also occurs sporadically throughout the deposit.

The quartz-alunite-kaolinite alteration defines a highly irregular but coherent zone around the known breccia bodies, where it grades both upwards and downwards to quartz-alunite alteration. The resulting rock has a distinct appearance, typically showing patchy alternations of quartz-alunite and quartz-kaolinite besides admixtures of quartz, alunite, and kaolinite.

The quartz-kaolinite alteration is largely confined to the porphyritic andesite, mainly to the near-surface, overlying body and around the margins of the breccia bodies. Further outwards, the porphyritic andesitic rocks that surround and overlie the mineralized breccia bodies are only weakly altered to a low-temperature smectite-chlorite assemblage and, in places, may be little altered and retain fresh hornblende phenocrysts. Supergene kaolinization, a product of the acidic solutions generated by pyrite oxidation, also affects the oxidized parts of the andesite. The quartz-kaolinite-altered andesite is a much less permeable rock than the rest of the altered rock at Keltepe. Hence, the 3% to 4% of fine-grained, disseminated pyrite contained in this andesite has been only partially oxidized.

OXIDATION

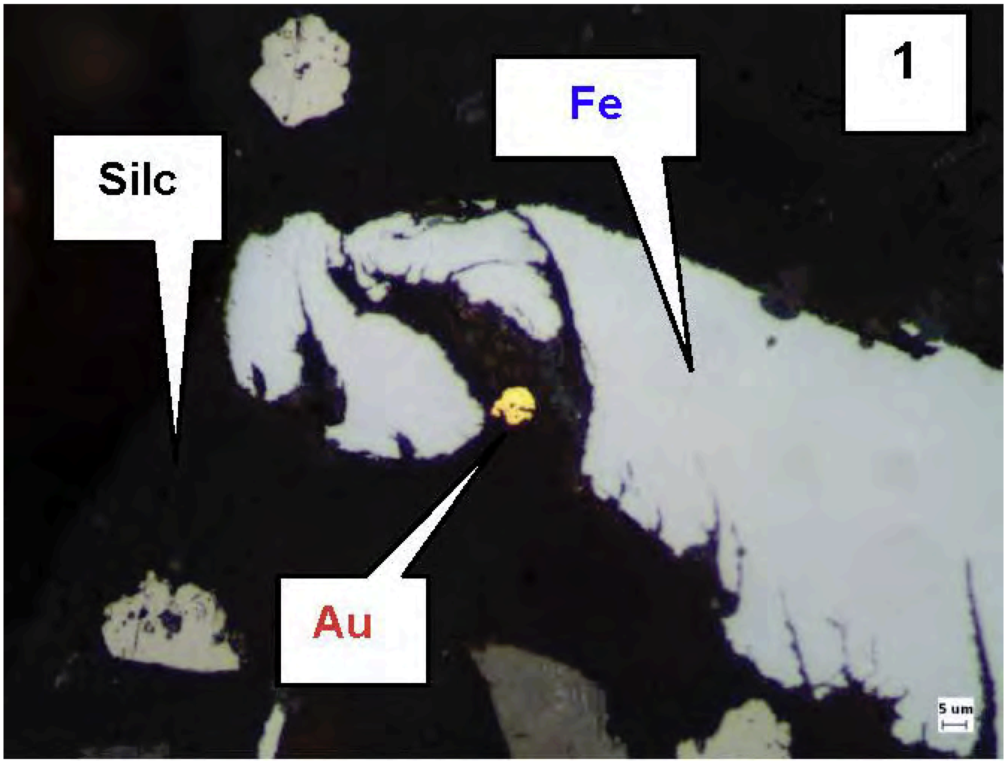
The Keltepe Deposit is strongly oxidized to a maximum known depth of up to 400 m below surface. This unusually deep oxidation is attributed to the porous and permeable nature of the siliceous and quartz-alunite altered breccias and to the presence of a deep groundwater table controlled by the NNW-SSE and NE-SW trending fault zones that drain outwards from the topographic high beneath which the Keltepe Deposit is located. Oxidation is marked by extensive hematitic and jarositic limonite and also occasionally by goethite. The degree of oxidation is higher along or near to the fault zones within the mineralized system. These faults are believed to have acted as conduits for gold-bearing hydrothermal solutions. Away from the faults, gold grades tend to be lower and where they meet another conduit and/or feeder structure they improve again.

Oxidation is not uniformly complete throughout the deposit, with patches of less oxidized or unoxidized rock enclosed by fully oxidized rocks.

MINERALIZATION

Oxide gold mineralization occurs from the surface (~1,800 m RL) to 250 m to 300 m below surface (~1,500 m RL to 1,600 m RL). Mineralization lies in an NW-SE orientation along strike and extends for approximately 950 m. Its width varies along strike, but in the centre of mineralization the width is about 370 m. Gold mineralization is believed to occur as finely disseminated particles as it was not identified during scanning electron microscope (SEM) analysis (Lashko, 2013). This has been confirmed by a gold deportment study that shows that the major gold mineral identified at Keltepe is native gold with an average fineness of 6.9 µm (SGS Canada Inc., 2013, Figure 7-7). This study also indicates that the host minerals for the gold in the sample studied are mainly quartz and other silicates and iron oxide, with minor (2% to 10%) rutile-silicate complexes and trace associations with pyrite.

FIGURE 7-7 PHOTOMICROGRAPH KELTEPE, ORE SAMPLE 68782B



Source: SGS Canada Inc., 2013, Figure 12, page 19. Au = native gold. Gold associated minerals include silicates (Silc), and Fe-oxide (Fe)

Rare "perched" zones of oxide copper mineralization, almost entirely composed of malachite coatings and impregnations in the breccia bodies, occur sporadically within the lower parts of the oxide zone, generally at depths of >200 m. The malachite zones are usually only one metre to two metres in thickness but may attain a thickness of >20 m (e.g., hole ODD0045). Copper grades in these zones, which are suspected to reflect former levels of chalcocite enrichment, may often be >1%.

Chalcocite, covellite, pyrite, and native sulphur are commonly seen in the supergene enrichment zone at depth at the Keltepe Deposit and significant copper ± gold intercepts have been returned from the redox boundary interface (e.g., 38 m grading 3.57% Cu and 0.21 g/t Au in hole ODD0146A). The redox boundary generally occurs some 50 m to 100 m below the base of the oxide gold mineralization, which occurs at a depth of approximately 250 m to 300 m. SEM analysis of sulphide breccia clasts has identified ubiquitous pyrite, enargite as isolated crystals and as inclusions in pyrite and/or rutile, galena as inclusions within pyrite, and tin oxides within vugs in the siliceous clasts (Lashko, 2013). Tennantite-tetrahedrite was also identified in a single sample. The hypogene source of the copper is suspected to be enargite that has been leached from the upper parts of the system.

SYNOPSIS

The Keltepe Deposit is interpreted as being hosted in a large phreatomagmatic vent with subsequent emplacement of one or more andesite domes at or near its top. The breccia that hosts the bulk of the gold mineralization could be a diatreme given the texturally destructive nature of the alteration and the likelihood that basement carbonate rocks would not survive in the acidic high-sulphidation system.

Emplacement of the diatreme breccia that hosts the Keltepe Deposit and the porphyritic andesite appear to have overlapped in both space and time (Sillitoe, 2012). The reconstructed form of the andesite suggests that it is part of a small exogenous volcanic dome(s), the basal parts of which have been preserved above the eastern and northern parts of the mineralized breccia, but with the remainder having been eroded. A thickness of ~100 m, or perhaps even more, seems likely to have been lost to erosion. Multiple episodes of brecciation clearly took place prior to andesite emplacement, but at least minor brecciation of the same type also followed andesite consolidation.

The main alteration event linked to the gold mineralization either accompanied or immediately followed andesite emplacement, as shown by the quartz-alunite alteration of the margins of the porphyritic andesite. The generally weaker, quartz-kaolinite alteration of the andesite reflects its inter-mineral timing and/or lower intrinsic permeability. The andesite that overlies the breccia bodies along their eastern margin may even have acted as a seal during the gold mineralization event.

Gold was preferentially precipitated in the siliceous ledges and their immediate quartz-alunite haloes, and was fed by at least one, and probably several, steeply dipping, NW-SE trending, structures represented by massive silicification. The restricted 200 m to 250 m vertical extent of ledge development and associated gold mineralization likely reflects a key palaeo-temperature interval that existed beneath the overlying andesite and was controlled by approach to the palaeosurface (Sillitoe, 2012).

The Keltepe breccia is considered to be inter-mineral in timing because of the ubiquity of the siliceous clasts. These clasts are presumed to have been derived from pre-existing silicified vuggy quartz and quartz-alunite ledges, like those at the nearby Güneytepe Deposit, and, hence, may be gold-bearing. Thus, a proportion of the gold at Keltepe may be present in the altered breccia clasts and rock flour produced by their comminution (Sillitoe, 2012).

GÜNEYTEPE DEPOSIT

GEOLOGY AND STRUCTURE

The Güneytepe Deposit is located approximately 600 m to the south-southeast of the Keltepe Deposit. Gold mineralization primarily occurs along NW-SE and NE-SW trending ledges of two compositions: 1) massive to vuggy residual quartz with associated silicification, and 2) quartz-alunite plus quartz-kaolinite alteration. The location of the ledges is controlled by the intersection of NW-SE and NE-SW trending structures. The siliceous ledges form prominent outcrops in the higher parts of the prospect area. However, their vertical extent appears to be limited to approximately 100 m. The ledges flare upwards and tend to coalesce, suggesting that only their roots are preserved. In contrast, the quartz-alunite plus quartz-kaolinite ledges occur at lower elevations and are largely concealed beneath talus cover. These comprise steep bands of quartz-alunite and quartz-kaolinite alteration and may be vertically more extensive (>200 m).

As observed at the Keltepe Deposit, gold mineralization at the Güneytepe Deposit is also considered to be controlled by NW-SE and NE-SW trending faults. The deposit is bounded to the north and south by two NE-SW trending fault zones, which confine the gold mineralization into a NE-SW trending corridor. Based on field and drill hole observations, the NE-SW faults are interpreted to be dominantly left-lateral strike-slip faults, whereas the NW-SE trending faults are dominantly dip-slip and they present as a step-like morphology from the upper to the lower parts side of the Güneytepe Deposit.

ALTERATION

Both the siliceous and quartz-alunite ± quartz-kaolinite ledges are enclosed by quartz-kaolinite alteration displaying patchy texture. Typically, such alteration comprises amoeboid patches of fine-grained quartz surrounded by massive kaolinite. The texture is considered likely to be a product of quartz nucleation during kaolinization (Sillitoe, 2011). In South America, patchy silica alteration (Gusano texture) is considered to represent a transitional zone from high-sulphidation epithermal style to porphyry style (Redwood, 2009).

Below the base of oxidation, remnant patches of early, pre-ledge alteration are present in the andesitic host rocks. In peripheral parts of the system, this early alteration is best described as weak potassic and comprises fine-grained, pale brown biotite and variable amounts of magnetite. The magnetite, locally constituting 5% by volume of the altered rock, tends to occur as irregular patches, but is also observed as veinlets and as breccia cement. Variable amounts of chlorite, illite, and smectite overprint the potassic assemblage and hematite partially or wholly replaces the magnetite.

In the central parts of the system, the potassic alteration is focused on a body of polymictic hydrothermal breccia, which is at least 150 m x 100 m in areal extent and extends for >150 m vertically. The breccia is inter-mineral in timing as shown by a variety of altered clasts cemented mainly by pyrite. Individual clasts contain minor amounts of chalcopyrite or molybdenite, and some contain truncated quartz veinlets of uncertain affiliation.

The breccia appears to be consistently mineralized with gold, but lacks appreciable copper. In contrast, the peripheral weak potassic alteration is barren, irrespective of the amount of contained hydrothermal magnetite.

The breccia body appears to be of magmatic-hydrothermal type and, as such, seems likely to be underlain at unknown depth by a porphyry stock, which may contain porphyry-type mineralization. The nature and grade of any such mineralization cannot be predicted, although the overall deficiency of copper in the potassic-altered breccia suggests that it may be a gold-only rather than a copper-gold system (Sillitoe, 2011).

OXIDATION

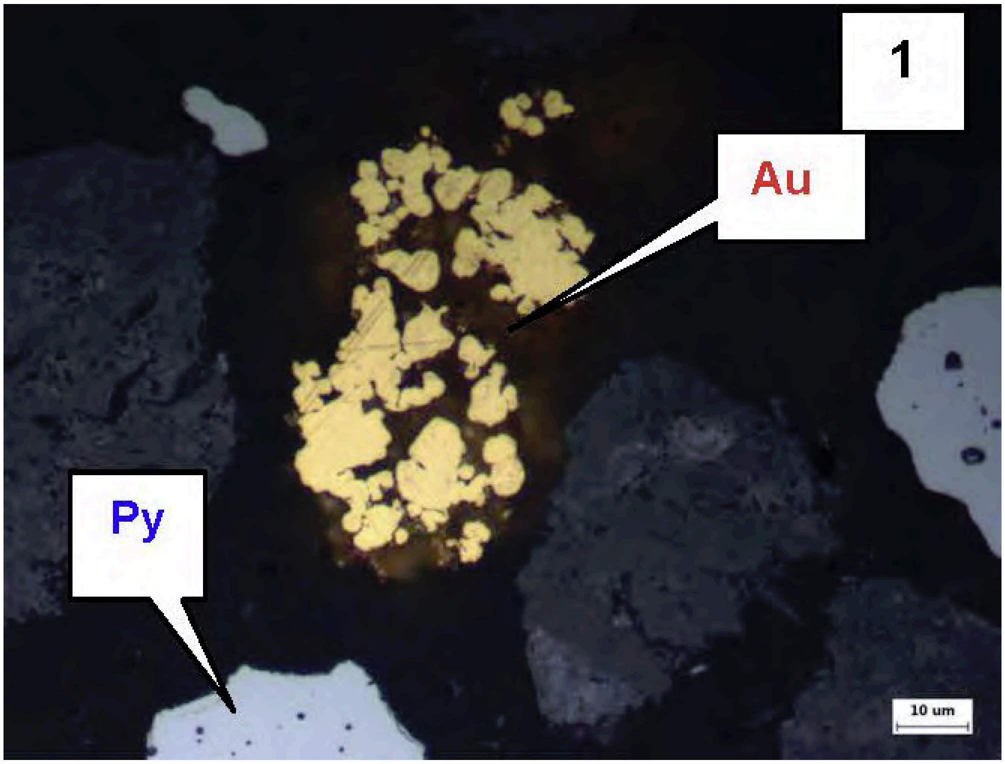
Oxidation in the ledges rarely exceeds 150 m in depth and is usually much less, averaging approximately 50 m to 75 m. Oxidation appears to be deeper in the massive to vuggy quartz and quartz-alunite zones than in those composed mainly of quartz-kaolinite. The deeper oxidation in the siliceous ledges is most likely due the higher porosity and the higher amount of acid-generating pyrite. The main oxidation products are jarosite and hematite. The base of the oxide zone is complex and highly irregular. Beneath the redox boundary, a 10 m to 20 m thick, immature chalcocite enrichment zone has developed. The chalcocite is sooty in texture and occurs as thin replacement films on pyrite. The supergene copper is presumed to have been derived from the oxidation of the minor amounts of enargite and covellite that accompany pyrite in the ledges.

MINERALIZATION

Gold mineralization mainly occurs within massive to vuggy silica and quartz-alunite altered material. The silica bodies are classic ledges in the sense that the best gold values accompany the massive to vuggy quartz, with lower grades being present in the smaller volumes of accompanying quartz-alunite alteration. Gold also occurs in the quartz-kaolinite alteration in which alunite contents are generally lower. Mineralization extends for approximately 335 m in the northwest-southeast direction and is open further to the north. Sulphide mineralization attains depths of 330 m from surface, while oxide mineralization occurs at a depth of not more than 110 m. Gold mineralization at Güneytepe is more variable than at Keltepe in both grade and lateral/vertical distribution. Higher sulphur contents are also recorded in the oxide zone due to sulphides, mostly pyrite, being encapsulated within massive silica and also in patchy silica altered rocks.

A gold deportment study has shown that the major gold mineral at Güneytepe is native gold with an average grain size of 5.4 μm (SGS Canada Inc., 2013, Figure 7-8). The mineral kustelite, an Au-Ag alloy with Ag ≥50 wt%, was also identified. This study also indicates that the host minerals for the gold in the sample studied are alunite with trace (<2%) associations with quartz, silicates, and iron oxide particles.

FIGURE 7-8 PHOTOMICROGRAPH GÜNEYTEPE, ORE SAMPLE 68783



Source: SGS Canada Inc., 2013, Figure 13, page 20. Au = native gold. Gold associated minerals include pyrite (Py). The gold abundance in this photograph is not representative of the Güneytepe mineralization, where gold grains tend to form smaller clusters or individual grains.

OTHER MINERALIZED ZONES

The Öksüt Project includes several other exploration targets in addition to the Keltepe and Güneytepe Deposits. All of these (Keltepe NW, Yelibelen, Büyüktepe, Boztepe, Boztepe W, Keltepe E, and Tombak) have received exploratory work since 2008.

At the Keltepe NW Prospect, located approximately 700 m to the northwest of the Keltepe Deposit, outcrops of quartz-alunite breccia and massive silica returned low-level gold anomalism (maximum 0.6 ppm Au) in rock chip samples. Limited drilling (three holes) beneath these outcrops during 2011 and 2012 returned a number of shallow oxide intercepts with grades up to 0.5 g/t Au over 10.6 m in hole ODD0042. Further drilling (four holes) in 2013, only returned narrow, weakly anomalous gold intercepts in the oxide zone but did intercept 25 m grading 0.89 g/t Au and 1.97% Cu at depth in the sulphide zone (hole ODD0178). This intercept, combined with the intercepts from Keltepe, indicated that the potential exists for a significant sulphide copper-gold resource at depth at the Öksüt Project. However, subsequent deep drilling in 2014 at Keltepe, Keltepe NW and between the two prospects, which was designed to test the potential of the copper-gold sulphides, generally returned narrower, lower grade intercepts up to a maximum of 11.0 m grading 1.44% Cu in hole ODD0237.

At the Tombak and Keltepe E Prospects, limited reverse circulation (RC) percussion and diamond drilling was undertaken in 2014 to test coincident zones of advanced argillic alteration associated with a circular magnetic low at Tombak and with a circular resistivity high at Keltepe E. No significant gold or silver values were returned from drilling at these prospects. However, drill hole ODD0233, at Keltepe E, returned weakly to strongly anomalous assay results for a number of elements including arsenic, barium, bismuth, copper, molybdenum, lead, antimony selenium, tin, tellurium, and uranium. This anomalism reflects a general porphyry-style geochemical signature.

8 DEPOSIT TYPES

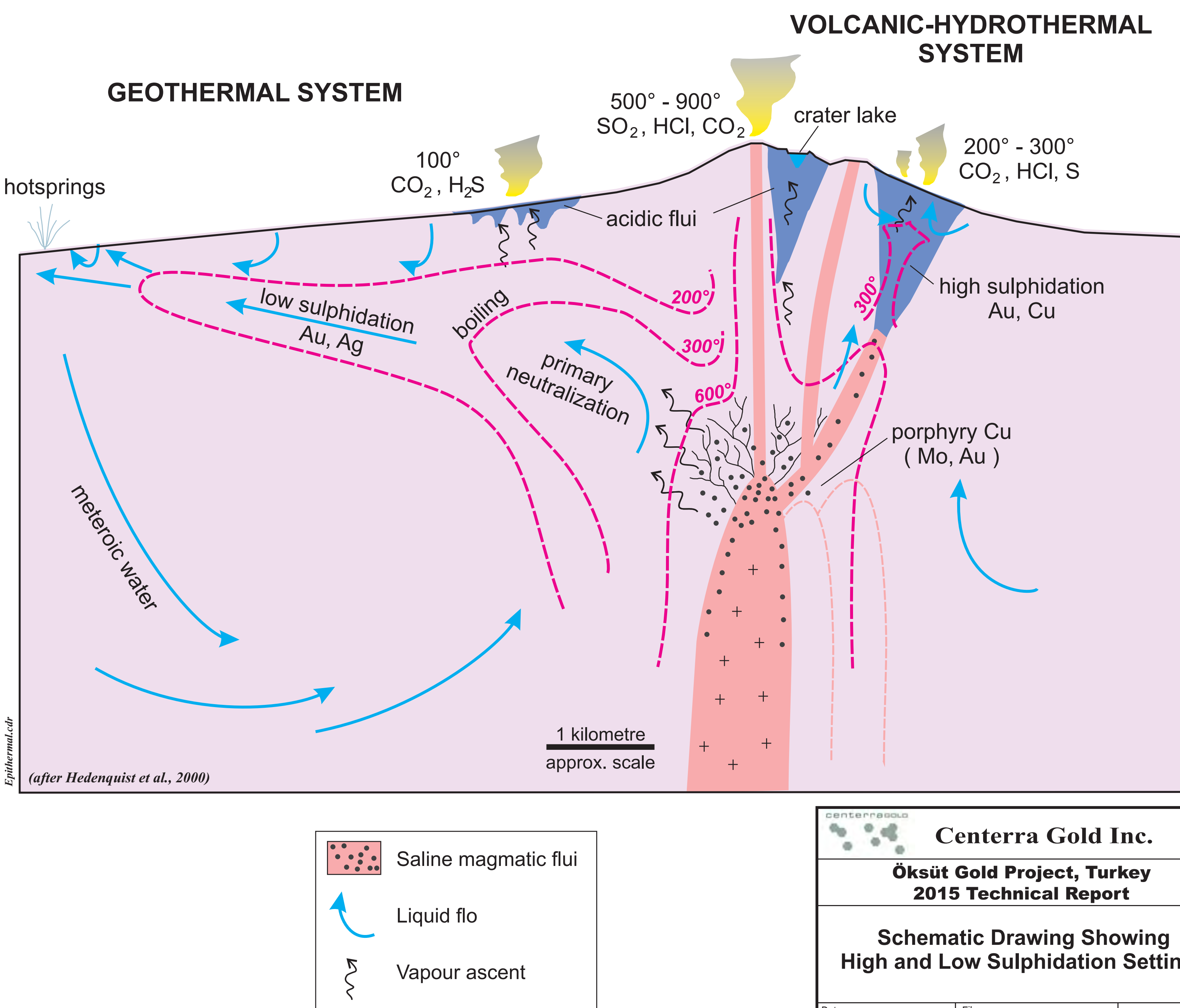
Gold mineralization at the Öksüt Project is associated with intense silicic alteration (mostly massive to lesser vuggy silica), advanced argillic alteration, and the presence of hydrothermal breccias. The original protolith (porphyritic andesitic lavas and coarse-grained volcaniclastics) may have contained in the order of 2% to 5% sulphide as pyrite with trace amounts of enargite and tennantite-tetrahedrite. Gold mineralization at Öksüt occurs primarily within areas of pervasive silicic alteration of the volcanic host rocks as well as within advanced argillic alteration assemblages proximal to silicic alteration. The gold mineralization occurs mainly above the redox boundary, although a number of drill holes have returned anomalous gold intercepts in the supergene sulphide zone.

Based on these observations, the gold mineralization at the Öksüt Project is considered to have formed as part of a high-sulphidation epithermal system, possibly overlying a porphyry system at depth. Epithermal precious metal systems may be classified as high, intermediate, and low-sulphidation styles (Figure 8-1). They are characterized by the sulphidation state of the hypogene sulphide mineral assemblage, and show general relations in volcano-tectonic setting, precious and base metal content, igneous rock association, proximal hypogene alteration, and sulphide abundance (John, 2001; Sillitoe and Hedenquist, 2003).

Occurrences of this type are formed under epizonal conditions, generally within 2 km of the palaeo-surface. High-sulphidation systems have also been referred to as acid-sulphate, enargite-gold, or alunite-kaolinite systems. Most high-sulphidation systems are associated with coeval andesite to dacite volcanic arcs, and are hosted by extensive “pre-mineral” advanced argillic lithocaps. The principal ore host is massive to vuggy residual silica, resulting from intense acid leaching of a pre-existing volcanic host rock. Proximal alteration comprises hypogene dickite, alunite (often crystalline), and/or pyrophyllite (Figure 8-2).

Sulphides include enargite, pyrite, and luzonite. Quartz veining is extremely rare, however, some deposits are overprinted by late barite and quartz veins. Laterally extensive sheets of intensely silicified rocks occur in many districts, and represent zones of lateral outflow of mixed hydrothermal and meteoritic water. Silica is transported in the acidic hydrothermal water, and on intersection with the palaeo-water table, undergoes neutralization and deposition of silica forming cryptocrystalline silica sheets.

Most high-sulphidation deposits are large, low grade bulk-tonnage systems (e.g., Yanacocha and Alto Chicama, Peru), though vein-hosted high-sulphidation deposits also occur (e.g., El Indio, Chile). In contrast, low- and intermediate-sulphidation state systems are typically related to quartz and carbonate veins, near-neutral hydrothermal fluids, and lack proximal advanced argillic alteration and residual vuggy silica. Steam-heated alteration is present above some intermediate- and low-sulphidation state systems' advanced argillic assemblages. However, they usually comprise low-temperature kaolinite, and fine-grained alunite. Sulphides are of a low- to intermediate-sulphidation state.



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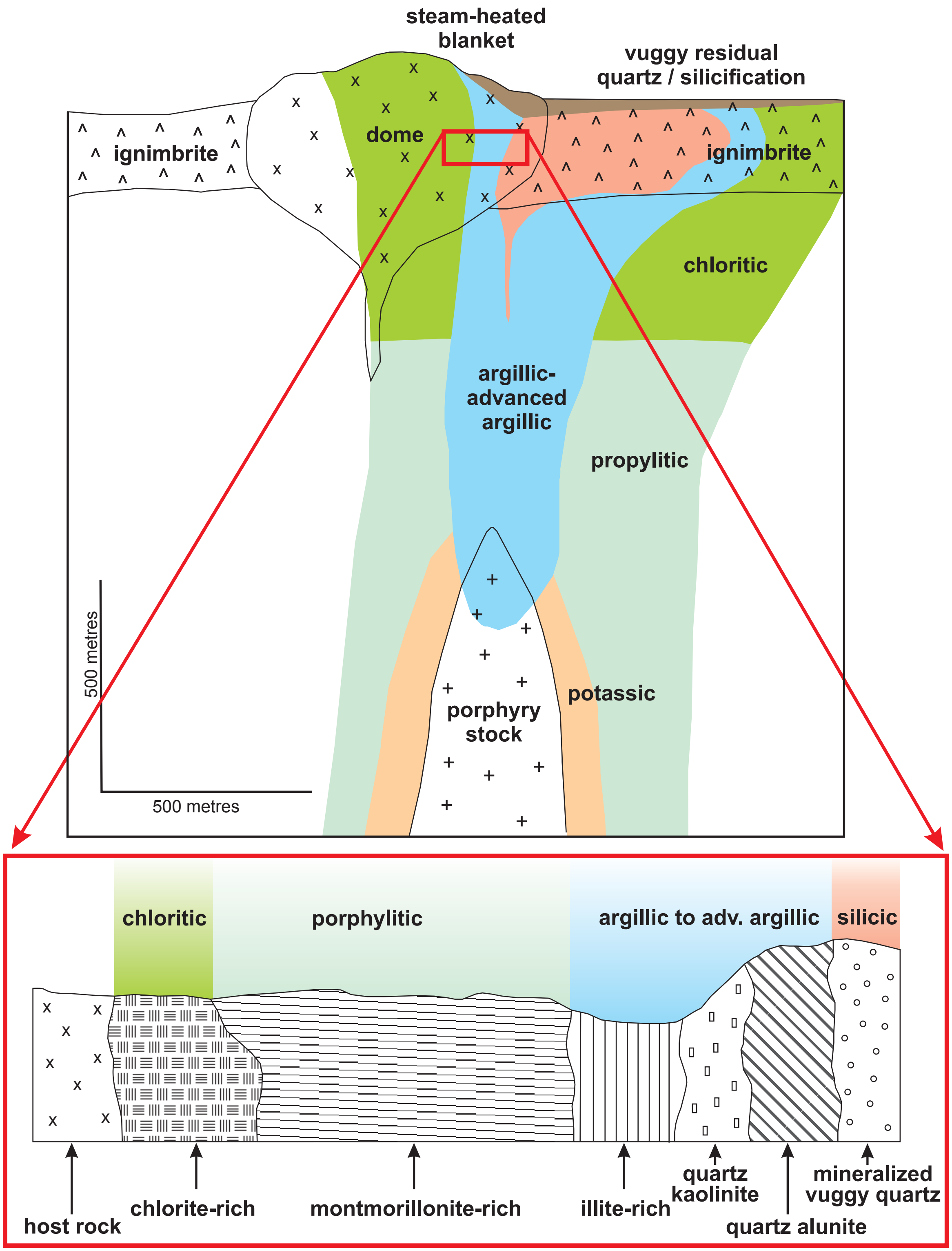
Schematic Drawing Showing High and Low Sulphidation Settings

Date: Sept. 2015	File: Epithermal.cdr	Figure 8-1



Source: Lashko, 2013;
Adapted from Hedenquist et al., 2000,
Sillitoe, 1999 and Sillitoe, 2010.

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Vertical and Lateral Zonation
of High Sulphidation
Epithermal Systems

Date: Sept. 2015 | File: HighSulphidation.cdr | Figure 8-2

9 EXPLORATION

Gold mineralization was discovered at Öksüt in 2007 by Stratex. Prior to this, there is no record of any modern exploration for gold being conducted on the property. Exploration activities had been performed by Stratex staff from 2007 to 2012 (with technical guidance from Centerra from 2009 to 2012) and by OMAS staff from 2013 onwards. In the QP's opinion, the sampling methods used and sample quality are representative of the mineralization and have no significant bias, as described in Sections 11 and 12. A summary of the exploration and some other activities undertaken by both companies is presented in Table 9-1.

Stratex followed up on gold bearing rock chip and channel sampling results in 2007 with additional rock chip and channel sampling and geological mapping in 2008. Based on the continued success of the surface sampling, drilling commenced at Öksüt in August 2008.

The initial drill program was limited to the area of Güneytepe where surface sampling had produced the best results. This program intersected gold mineralization starting at the surface and extending up to 70 m below the surface.

After signing the joint venture agreement with Centerra in 2009, Stratex performed further geological mapping, geochemical sampling, ground geophysics, and trenching. The 2010 drill program confirmed the presence of gold mineralization at Keltepe. The majority of drilling and exploration activities since 2010 have focused on delineating the extents of mineralization at Güneytepe and Keltepe as well as defining additional targets with mineralization potential. Other exploration activities during this period were performed to gather data for environmental and engineering purposes.

TABLE 9-1 SUMMARY OF EXPLORATION ACTIVITIES SINCE 2007

Year	Exploration Activity	Metrics
	Stratex 2007 to 2012	
2007	Rock chip and channel sampling of discovery outcrops	61 samples
2008	Geological mapping 1:25,000 scale	56 km^2
	Geological mapping 1:5,000 scale	20 km^2

Year	Exploration Activity	Metrics
	Geological mapping 1:500 scale at Güneytepe	500 m x 400 m
	Rock chip and channel sampling - prospects	1,430 samples
	Soil sampling: 100 m NS x 50 m EW grid In south part 200 m NS x 100 m EW grid	1,775 samples
	Topographical survey 1:1 000 at Güneytepe and Büyüktepe	0.85 km^2
	Quickbird high-resolution satellite imagery purchased	5 km x 4.5 km
2009	Rock chip sampling - prospects and regional	624 samples
	Infill and extension of soil sampling grids	970 samples
	Topographical survey 1:1,000 scale at Keltepe, Keltepe NW, Büyüktepe, and Yelibelen	2.1 km^2
	Trenching	4 trenches / 740 m / 401 samples
	IP geophysical surveying at Güneytepe, Büyüktepe, Yelibelen and Boztepe West	8.4 line-km
	Ground magnetic surveying at Güneytepe, Büyüktepe, and Yelibelen	5.75 line-km
2010	Rock chip sampling – prospects and regional	93 samples
	Check geological mapping	
	IP/resistivity geophysical surveying at Güneytepe, Büyüktepe, and Yelibelen	10.95 line-km
	Ground magnetic survey at Güneytepe, Büyüktepe, Yelibelen, and Boztepe West	61.9 line-km
2011	Rock chip sampling	19 samples
	Check geological mapping at Büyüktepe	
	Consultant Richard Sillitoe contracted to review and revise geologic interpretations in April 2011.	
	IP/resistivity geophysical surveying at Güneytepe, Keltepe, and Keltepe NW	15.3 line km
	Ground magnetic survey at Güneytepe, Keltepe, and Keltepe NW	7.5 line km
	Bulk density measurements on core from Güneytepe, and Keltepe	137 samples
	Water quality sampling	Quarterly
2012	Geological mapping 1:500 at Keltepe, Keltepe NW, and Büyüktepe	1.5 km x 1.7 km
	Rock chip sampling – prospects and regional	470 samples
	Consultant Richard Sillitoe contracted to review and revise geologic interpretations in April 2012.	
	Purchase of ASTER scene and interpretation	60 km^2
	Regional BLEG stream sediment sampling	52 samples
	Ground magnetic survey at Güneytepe, Keltepe, Keltepe NW and surrounds	102 line km
	Bulk density measurements on core from Güneytepe and Keltepe	218 samples
	Water quality sampling	Quarterly

Year	Exploration Activity	Metrics
	Screening phase archaeological survey	4 km^2
	Metallurgical test work - Heap Leach Amenability Testing, SGS UK	3 composite samples
OMAS 2013-2014		
2013	Geological mapping at 1:2,000 and 1:500 scales at Keltepe E and surrounds	6.75 km^2
	Rock chip sampling – prospects and regional	36 samples
	Soil sampling: 200 m x 50 m on NE-SW grid at Keltepe E	263 samples
	Acquisition of high-resolution satellite imagery and production of detailed DEM (1 m resolution)	131 km^2
	IP/resistivity geophysical surveying at Keltepe, Keltepe NW, Güneytepe and Yelibelen	61.1 line-km
	Ground magnetic survey at Keltepe, Keltepe NW, Güneytepe, Yelibelen and surrounds	251.2 line-km
	Bulk density measurements on core from Keltepe and Güneytepe	404 samples (202 samples repeated at two different labs)
	Water quality sampling	Quarterly
	Metallurgical test work - Heap Leach Amenability Testing, Kappes Cassidy Associates	4 composite samples
	Conceptual Heap Leach Pad Design, SRK Consulting Ankara	
	ESIA – including environmental and social baseline studies	Commenced July 2013
2014	Metallurgical Test Work – Kappes Cassidy Associates	6 composite samples
	Water quality sampling	Quarterly
	Archaeological survey	Site specific
	Environmental sampling (dust and noise)	Monthly
	Bulk density measurements on core from Keltepe and Güneytepe	185 samples
	Detailed topographical surveys of the Keltepe and Güneytepe Deposits and infrastructure sites.	

10 DRILLING

DRILLING DATABASE

The Öksüt database contains a total of 330 drill holes (including re-drills, metallurgical holes, geotechnical holes and water monitoring holes) totaling 82,006 metres of drilling. Of these 330 drill holes, 58 holes are reverse circulation (RC) drill holes and the remaining 272 holes are diamond drill holes (DD).

A subset of 292 holes was used to update Öksüt Mineral Resource estimate. This subset of holes covers all exploration drilling on the Öksüt concessions and includes numerous holes drilled beyond the mineralized extents of Keltepe and Güneytepe. Only 41 holes of the 292 hole subset are RC holes and of these, only four holes are located within the extents of the mineralization and used for grade estimation.

The 38 engineering holes not included within the subset used for the estimate are metallurgical holes, water well holes, or geotechnical holes that were not assayed.

Tables 10-1 and 10-2 summarize the various subsets described above.

TABLE 10-1 SUMMARY OF ÖKSÜT RESOURCE DRILLING

Year	Drill Holes	Length (m)
2008	13	2,678
2009	3	675
2010	21	4,355
2011	21	6,239
2012	38	14,445
2013	91	26,236
2014	105	20,863
Total	292	75,492

TABLE 10-2 SUMMARY OF ÖKSÜT ENGINEERING HOLES

Year	Drill Holes	Length (m)
2008	-	-
2009	-	-
2010	1	89
2011	-	-
2012	1	328
2013	4	1,127
2014	32	4,970
Total	**38**	**6,514**

Industry-standard diamond drilling procedures have been employed at all times. Drilling has been concentrated in two main areas of the Project, Keltepe and Güneytepe. The 2008-2011 drilling programs were carried out by Ankara-based drilling contractor Pozitif Sondaj and were supervised by Stratex personnel. Drilling was carried out using one to three contractor-manufactured drill rigs. The model numbers are PD300, PD400, and PD500 with depth capacities of 650 m, 800 m, and 1,000 m of HQ-size (63.5 mm) core, respectively. Between 2008 and 2011, the drilling was mostly HQ size, which was reduced to NQ-size (47.6 mm) core when ground conditions became difficult. Drill holes with poor recoveries (generally less than 50% to 70%) through mineralized intervals or those that were abandoned before target depth were re-drilled.

The drill hole density varies from prospect to prospect, however, the nominal drill spacing at the Keltepe and Güneytepe Deposits is approximately 50 m. Drill holes were mostly inclined (between -45° and -90°) and oriented predominantly to grid northeast (077º) or grid southwest (257º) at Keltepe and grid northeast (060º) and/or grid southwest (240º) at Güneytepe. Drill hole deviation was measured using Reflex Survey tests taken at nominal 30 m to 50 m intervals down hole to provide control. In general, little hole deviation was observed.

The 2012-2014 drilling programs were carried out by Ankara-based drilling contractor Spektra Jeotek with the 2012 program being supervised by Stratex personnel and the 2013 and 2014 programs being supervised by OMAS personnel. Drilling was carried out using up to five contractor-manufactured drill rigs. The model numbers are D 150 and D 220 with depth capacities of 1,000 m and 1,250 m of HQ-size core, respectively. From 2012 to 2014, mostly PQ- and HQ-size core drilling was carried out, with rare reductions to NQ-size when ground conditions became difficult. Drill holes with poor recoveries (generally less than 50% to 70%) through mineralized intervals or those that were abandoned before target depth were re-drilled.

OMAS commenced drilling triple-tube drill holes with a split tube in late 2013 to aid in determining the orientation of mineralized structures. In 2014, triple-tube drilling was only used for metallurgical and geotechnical drill holes, as a comparison with earlier drilling showed that core recoveries did not improve significantly using the triple-tube drilling technique. Overall, core recoveries are considered to be good and within tolerance to include in a resource estimate.

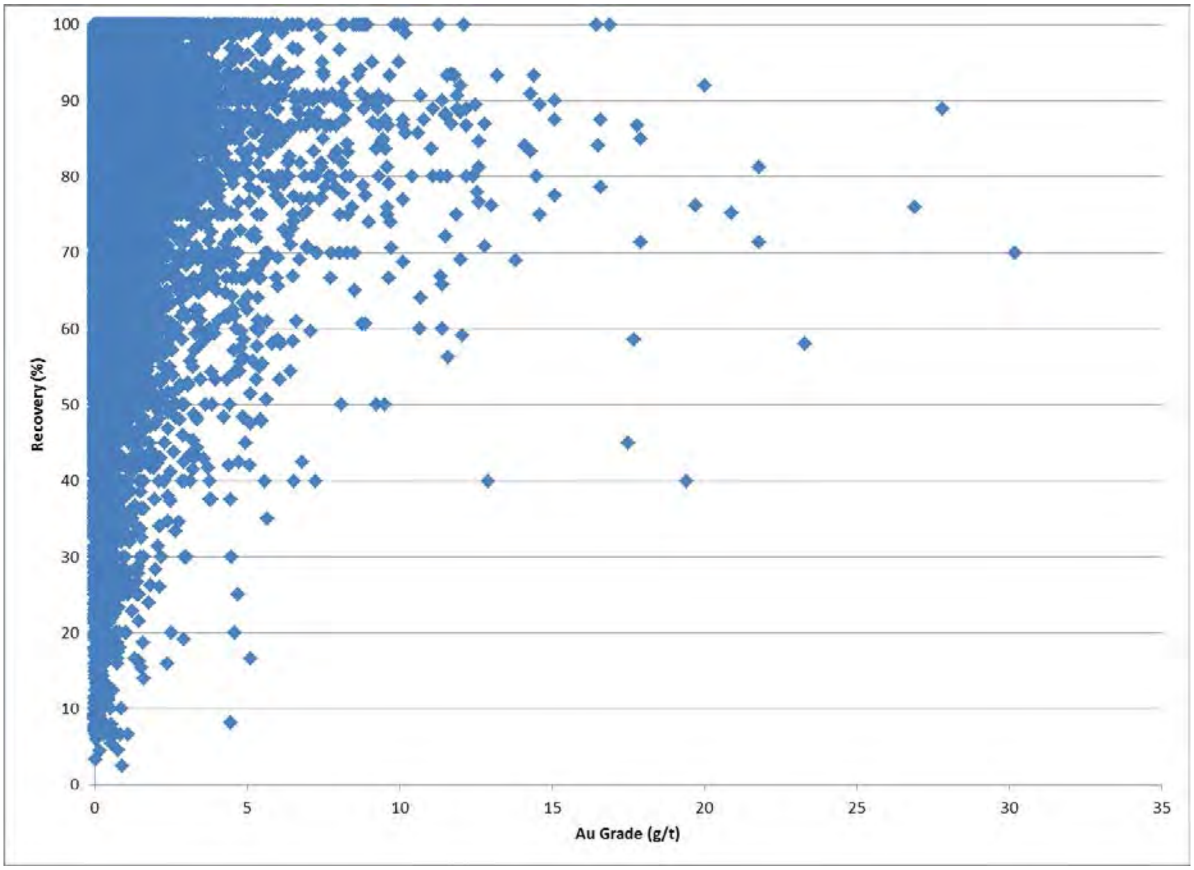
Core recovery measurements were taken on 248 of the 251 diamond drill holes contained within the resource estimation subset of 292 holes. Table 10-3 summarizes the ranges of core recovery for assayed core intervals.

TABLE 10-3 RANGES OF CORE RECOVERY FOR ASSAYED CORE INTERVALS

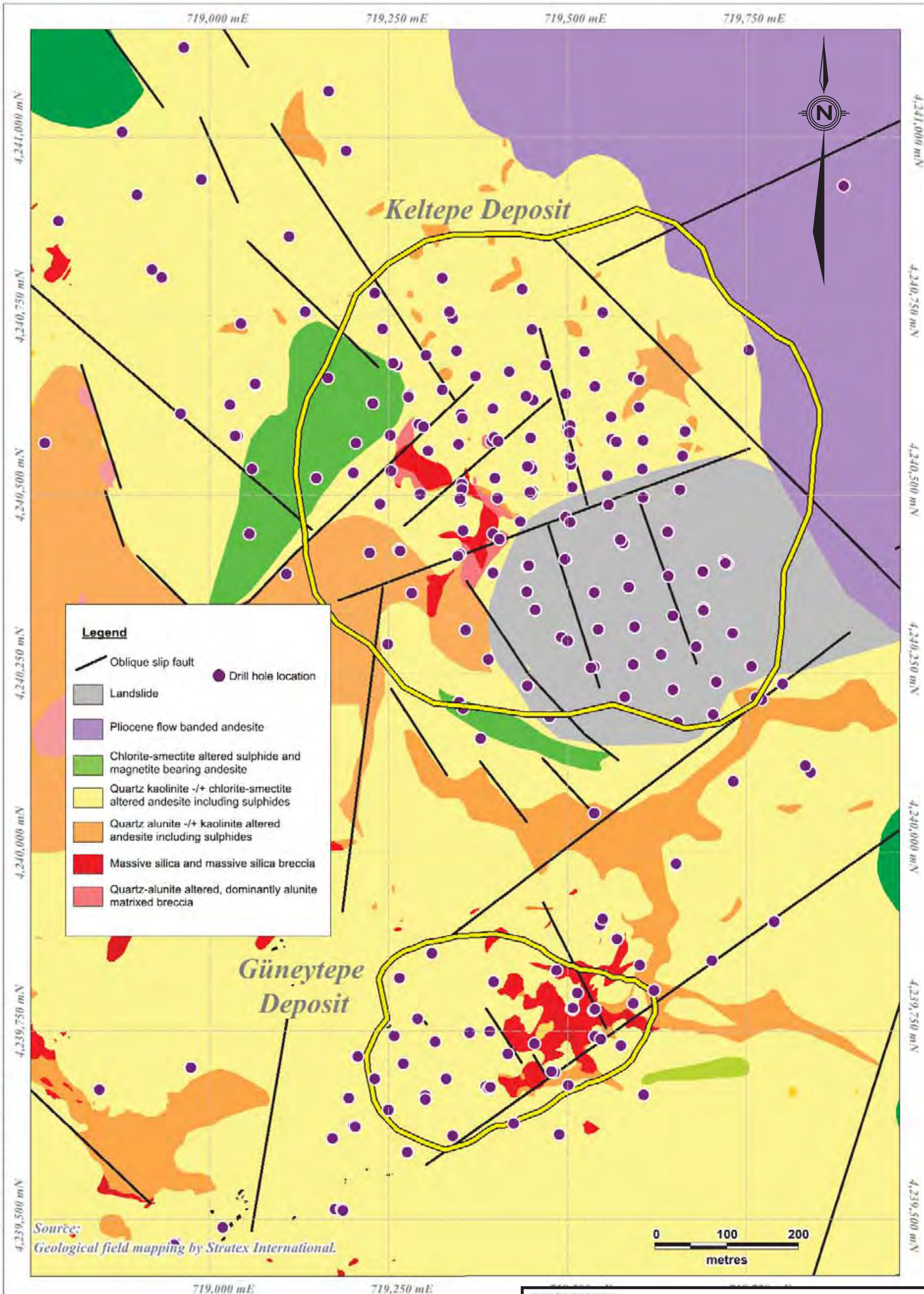
Core Recovery Percentage	Öksüt	
	Number of Samples	Percent of Total
95 to 100%	10,175	22%
90 to 95%	6,824	15%
85 to 90%	8,187	18%
80 to 85 %	4,893	11%
75 to 80%	3,933	9%
70 to 75%	2,839	6%
60 to 70%	4,123	9%
50 to 60%	2,489	5%
< 50%	2,721	6%
Totals	**46,184**	**100%**

The assay data indicate a small statistical decrease of the gold grade with decreasing core recovery, suggesting that lost core tends to be of slightly higher grade than the recovered core. The data show that the magnitude of this trend is not large and Centerra does not anticipate a need to run alternate grade interpolations without intervals with low core recovery. Figure 10-1 plots recovery against gold grade for all samples and shows that samples above 5 g/t Au rarely have recoveries of less than 60%.

FIGURE 10-1 CORE RECOVERY VS. GOLD GRADE (ALL SAMPLES)



Drill collar locations were first determined by site personnel using a hand-held GPS. Subsequent to the completion of the hole, collar locations were surveyed by a registered land surveyor using a differential global positioning system (DGPS) with a horizontal and vertical accuracy of generally ± 20 cm. The positions of all drill collars have been surveyed by this method. Figure 10-2 shows all of the Keltepe and Güneytepe drill hole locations.



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Drill Holes Completed at Keltepe and Güneytepe

Date: Sept. 2015	File: CompletedHoles.cdr	Figure 10-2

Mineralized zones at Öksüt are somewhat irregular in shape and orientation, and true thickness is variable. It is estimated that true widths for mineralized zones are approximately 50% to 90% of the down hole intervals. Geological modelling and mineral resource estimation procedures take into account the intercept angles of drilling versus the interpreted geometry of mineralization.

RC PERCUSSION DRILLING

A total of 58 RC percussion holes (including condemnation holes, exploration holes, and water monitoring holes) were completed by OMAS in the Project area in 2014. The drill hole density varies from area to area, depending upon the purpose of each drill hole. Condemnation and water monitoring holes were generally drilled vertically while exploration holes were drilled at -60° towards varying azimuths.

The RC percussion drilling programs were carried out by Ankara-based drilling contractor Pozitif Sondaj (condemnation holes, exploration holes, and water monitoring holes) and Trabzon-based Kurt Sondaj (water monitoring holes only). The holes drilled by Pozitif Sondaj were drilled with a Gemex MP 1000 HOA rig, capable of drilling to a depth of 400 m with 4 in. outside diameter rods. A Sandvik RE004 RC hammer with a variety of bit sizes was used. The holes drilled by Kurt Sondaj were drilled with a 2004 model AS950 truck-mounted rig with 800 m drilling capacity. An open-hole percussion hammer with 12 in. to 15 in. bits was used.

In the QP’s opinion, there are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

From 2007 to 2012, samples from the Öksüt Project were sent to ALS Chemex in Izmir, Turkey with the actual analyses conducted in the ALS facility in Vancouver, Canada or Roşia Montană, Romania and finally, in Izmir. From September 2012 onwards, preparation and analysis of samples were carried out by SGS Ankara, Turkey. Gold was assayed using standard 50 g fire assay with an atomic absorption (AA) finish, and other elements were determined by multi-acid digestion and inductively coupled plasma (ICP) finish. Both laboratories are independent ISO 9001:2008 registered external commercial assay laboratories. Surface samples are not included in the Öksüt Project Mineral Resource database and are not used for Mineral Resource estimation.

CORE AND RC CHIP SAMPLING METHOD AND APPROACH

At the Öksüt Project, diamond drilling has employed PQ (85.0 mm), HQ (63.5 mm), and NQ (47.6 mm) diameter core tools, with the majority of the holes now being PQ in diameter. Additionally, PQ diameter core has been employed during metallurgical drill programs. All core and RC percussion chip samples are subjected to quality control procedures that ensure best practice in the handling, sampling, analysis, and storage of the drill core and RC percussion chips.

For diamond drilling, diamond drill core is delivered by the drill crew to OMAS geological personnel at the Develi core facilities. To maintain the integrity of the core, the boxes are packed and fastened shut with steel ties in the back of the trucks. At the core facility, the core is photographed and then laid out for logging and geotechnical measurements. The core is measured for core recovery (TCR) and rock quality (RQD) and then logged for geological, structural, and geotechnical features. Sampling intervals are selected on a geological basis but typically vary from one to two metres in length. Core is divided lengthwise into two halves using a diamond saw. Assay samples are collected by trained personnel from one-half of split core and the other half is replaced in the core box for future reference.

For RC drilling, dry chip samples are obtained using a standard cyclone and on-board riffle splitter. A 25:75 split is obtained with the smaller portion being bagged for analysis and the larger portion being retained in the plastic sample bag. A small amount of sample from each bag is washed and placed in a numbered chip tray. A geologist is on site at all times to monitor the sampling and log the chips. Additional chip logging is also carried out in the Develi core facility.

The core and RC chips to be assayed are placed in marked calico bags and sealed. Standard, duplicate, and blank samples are inserted in the sample stream before shipping with pre-assigned sample numbers. Sample weights are not recorded at site but are recorded at the laboratory. The site technicians and one of the site geologists are responsible for sample custody until shipment. The retained core and larger RC chip split are stored in a secure area at the Develi core facility.

Assay samples are shipped by an independent transport company to the laboratory in Turkey. Receipt of sample shipments by the laboratory is confirmed by electronic mail.

SAMPLE PREPARATION AND ANALYTICAL METHODS

Prior to drill hole ODD0070, samples were prepared and analyzed at ALS Chemex laboratories in Roşia Montană, Romania and Izmir, Turkey. For holes drilled after drill hole ODD0070 (inclusive), samples were prepared and analyzed at SGS Ankara, Turkey. Both ALS Chemex and SGS Ankara laboratories have been assessed and certified as meeting the requirements of ISO 9001:2008, and are commercial laboratories independent of Centerra.

Preparation and analytical methods are described in Table 11-1 for samples submitted to ALS Romania or Izmir from 2008 to September 2012.

TABLE 11-1 SAMPLE PREPARATION AND ANALYTICAL METHODS – ALS CHEMEX

Method Code	Item/Element	Description
CRU-31	Sample preparation	Crush to 70% passing 2 mm; riffle split
PLU-32		Pulverize 1,000 g to >85% passing 75 µm
Au-AA24	Gold	50 g fire assay with AAS finish, 5 ppb detection limit
ME-MS41	50 multi-element	Aqua regia digestion, ICP-MS and OES finish
Dec'11 to Sept'12: ME-ICP61	33 multi-element	4-acid (HCl, HNO_3, $HCLO_4$, HF) digestion, ICP-OES finish
MEOG62	Base metals	Over-range assay method using 4-acid digestion, ICP-OES or AAS finish

Preparation and analytical methods are described in Table 11-2 for diamond core and RC percussion chip samples submitted to SGS Ankara after September 2012.

TABLE 11-2 SAMPLE PREPARATION AND ANALYTICAL METHODS – SGS

Method Code	Item/Element	Description
PRP89	Sample preparation	<3 kg dry, crush to 75% passing 2 mm, split to 250 g, pulverize to 85% passing 75 µm
Au-FAA505	Gold	50 g fire assay with AAS finish, 10 ppb detection limit
ICP40B	34 multi-element	4-acid digestion, ICP-OES finish
CSD06V	Sulphur (total)	Carbon-sulphur LECO analyzer
AAS42S	Multi-element	Over-range 4-acid digestion, AAS finish

QUALITY CONTROL PROCEDURES

Geochemical Certified Reference Material (CRMs) has been used in all diamond and RC drilling programs at the Öksüt Project. During the period August 2008 to February 2013, Stratex inserted CRMs and blanks at the rates shown below.

- Insertion of a coarse blank, one in every 100 samples;
- Insertion of a CRM, three in every 100 samples;
- Insertion of a duplicate ¼ core sample, one in every 100 samples.

During the period May 2013 to November 2014, OMAS inserted CRMs, duplicates, and blanks at the rates shown below.

- Insertion of a coarse blank, one in every 50 samples;
- Insertion of a CRM, one in every 30 samples;
- Insertion of a duplicate ¼ core sample, one in every 50 samples;
- Routine duplicate assays of pulps as part of laboratory QC protocols.

Blank control samples are from either barren limestone (Stratex) or barren basalt (OMAS). Blanks weighing from one to two kilograms are inserted in the sampling stream by the site field technician.

CRMs were supplied by Rocklabs Limited (Rocklabs) in New Zealand (Stratex) or Geostats Pty Ltd (Geostats) in Australia (OMAS). Approximately 60 g of the standards are inserted in the sampling stream in small "Kraft" paper envelopes or sealed plastic sachets. The certified gold values of the Geostats standards were selected to approximate the mining cut-off grade (0.2 g/t Au), the deposit average grade (1.2 g/t Au), and three to five times above the average grade.

To date, a total of 2,094 gold CRMs have been submitted along with core samples from 22 different standards (15 – Stratex and 7 – OMAS) with certified assay values varying from 0.201 ppm to 4.075 ppm Au (Rocklabs) and from 0.22 ppm to 4.086 ppm Au (Geostats). This includes a total of 23 copper CRMs that have been submitted from a single CRM with a certified assay value of 1.04 ppm Au and 1.034% Cu. A total of 55 gold CRMs (as described above) have been submitted along with RC percussion chip samples.

QUALITY CONTROL RESULTS

Quality control reviews of each batch of assay results were undertaken before the results were entered into the Mineral Resource database.

CERTIFIED REFERENCE MATERIALS

A total of 2,094 CRMs were analyzed in conjunction with core samples. Results are summarized in Table 11-3 and are presented in graphic form for gold in Figure 11-1. A total of 48 CRM samples are not included in Table 11-3 or on Figure 11-1 due to mis-labelling, which resulted in spurious statistical average values.

FIGURE 11-1 DIAMOND CORE CRM ASSAY RESULTS, ALS CHEMEX AND SGS

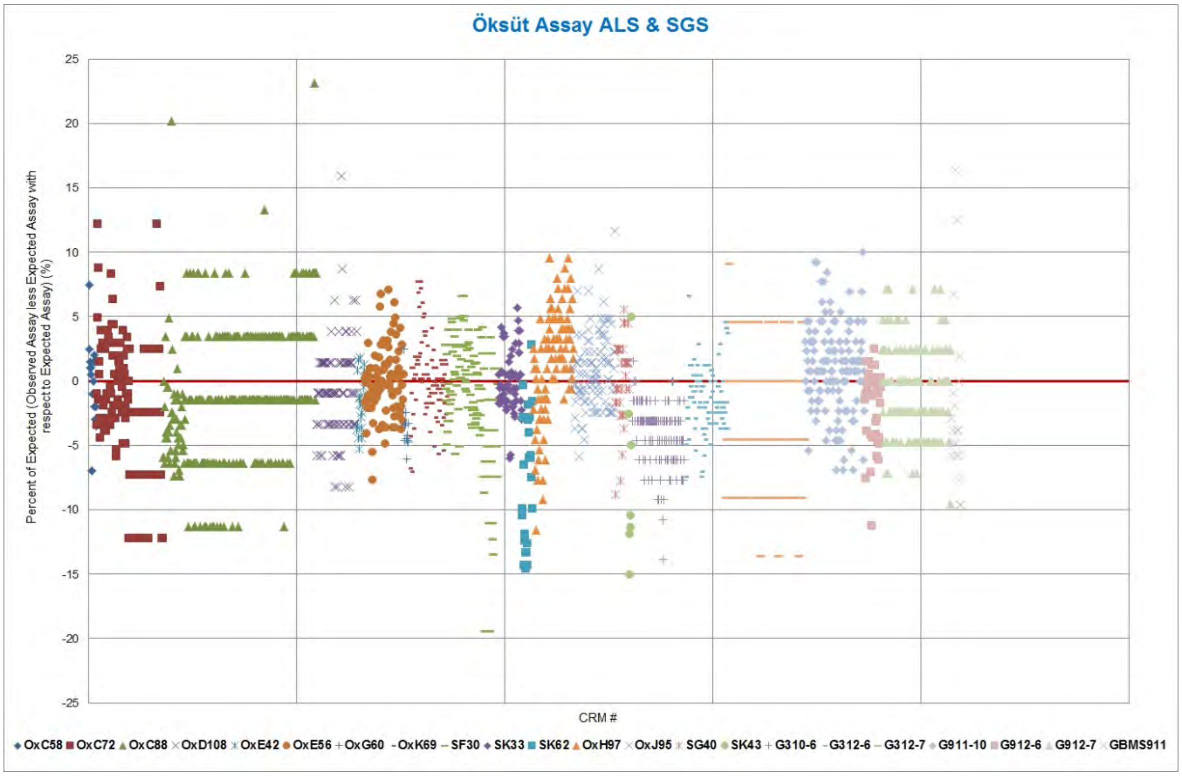


TABLE 11-3 SUMMARY OF DIAMOND CORE CRM ASSAY RESULTS, ALS CHEMEX AND SGS

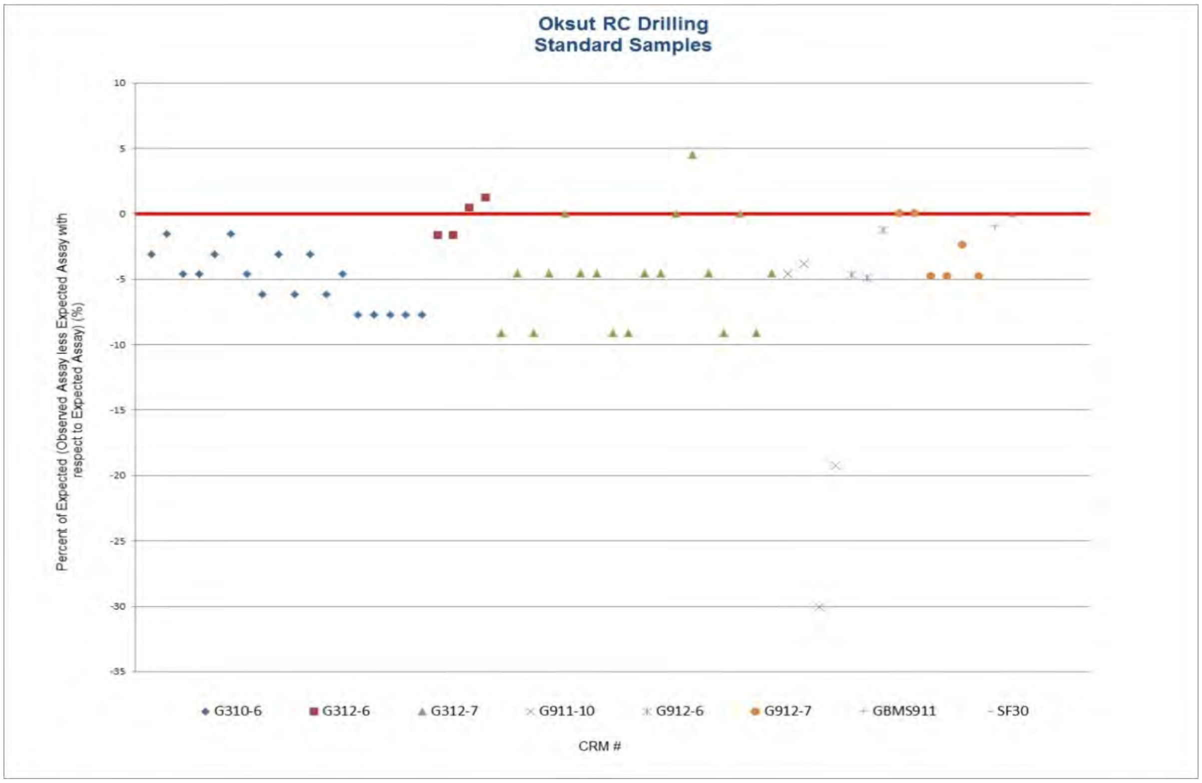
CRM Identification	Certified Value (Au – g/t)	ALS Chemex		SGS	
		N	Mean Value Achieved (Au – g/t)	N	Mean Value Achieved (Au – g/t)
OxC58	0.20	18	0.20		
OxC72	0.21	79	0.21	81	0.20
OxC88	0.20	53	0.20	316	0.20
OxD108	0.41			96	0.41
OxE42	0.61	20	0.60		
OxE56	0.61	94	0.61		
OxG60	1.03	11	0.99		
OxK69	3.58	89	3.59		
SF30	0.83	89	0.83	38	0.79
SK33	4.04	57	4.04		
SK62	4.08			26	3.76
OxH97	1.28			98	1.30
OxJ95	2.34			99	2.36
SG40	0.98			34	0.98
SK43	4.09			7	3.79
G310-6	0.65			126	0.62
G312-6	2.42			101	2.37
G312-7	0.22			188	0.22
G911-10	1.30			145	1.31
G912-6	4.08			39	4.00
G912-7	0.42			167	0.41
GBMS911	1.04			23	1.08
All ALS	**1.44**	**510**	**1.44**		
All SGS	**0.93**			**1,584**	**0.93**

A total of 55 CRMs were analyzed by SGS in conjunction with RC percussion chip samples. Results are summarized in Table 11-4 and are presented in graphic form for gold in Figure 11-2.

TABLE 11-4 SUMMARY OF RC PERCUSSION CHIP CRM ASSAY RESULTS, SGS

CRM Identification	Certified Value (Au – g/t)	SGS	
		N	Mean Value Achieved (Au – g/t)
G310-6	0.65	18	0.62
G312-6	2.42	4	2.41
G312-7	0.22	18	0.21
G911-10	1.30	4	1.11
G912-6	4.08	3	3.93
G912-7	0.42	6	0.41
GBMS911	1.04	1	1.03
SF30	0.83	1	0.83
All SGS		**55**	**0.82**

FIGURE 11-2 RC PERCUSSION CHIP CRM ASSAY RESULTS, SGS



The CRMs generally performed well, with very few failures outside of three standard deviations from the mean. In the instances of a CRM failure, ten samples on either side of the affected CRM were re-assayed and acceptable results were returned in all instances. The SGS gold assays tend to be biased low by 3% to 5% for some of the CRMs in use.

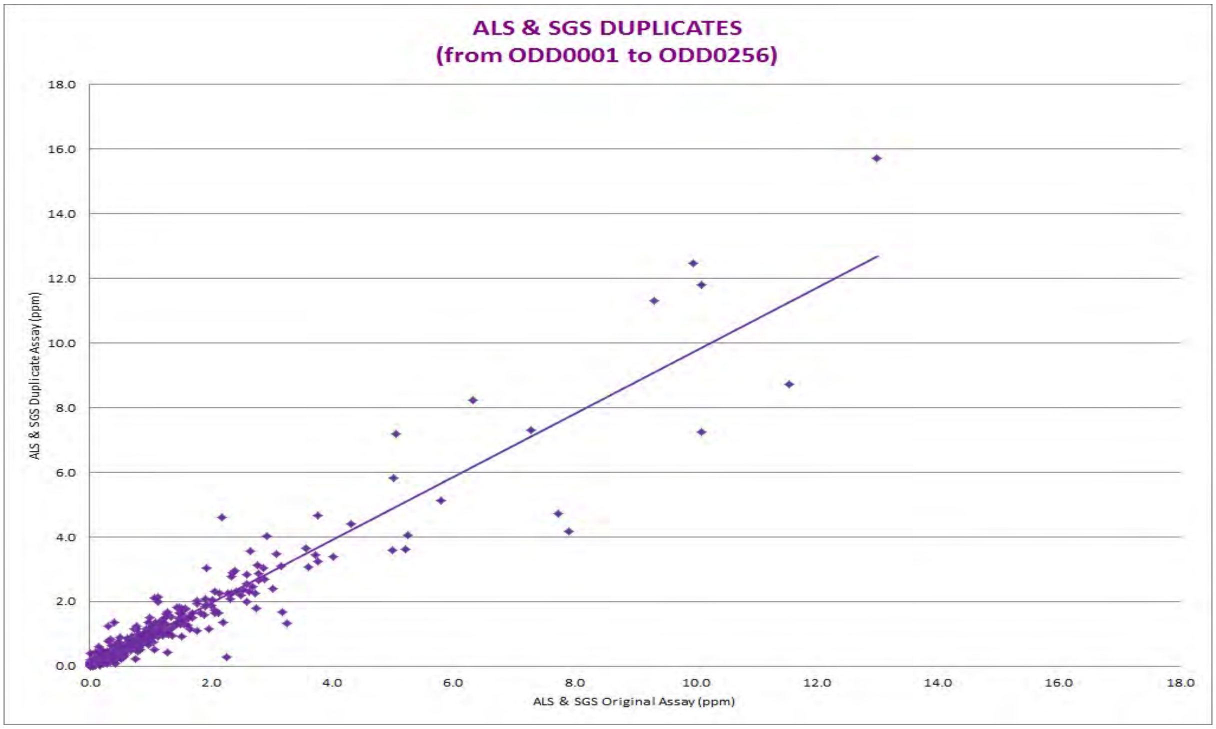
FIELD DUPLICATE DIAMOND CORE AND RC PERCUSSION CHIP SAMPLES

The results of assaying a total of 1,518 one-quarter diamond core field duplicates are summarized in Table 11-5 and shown in Figure 11-3:

TABLE 11-5 SUMMARY OF FIELD DUPLICATE CORE ASSAY RESULTS, ALS CHEMEX AND SGS

	Drill Holes	Number of Pairs	Average Grade Original Sample (Au – g/t)	Average Grade Duplicate Sample (Au – g/t)
ALS Chemex	ODD 001 to 069	488	0.455	0.462
SGS	ODD 070 to 186	752	0.402	0.395
SGS 2014	ODD0193 to 256	178	0.125	0.120

FIGURE 11-3 FIELD DUPLICATE CORE ASSAY RESULTS



The results of assaying of a total of 24 RC percussion chip field duplicates are summarized in Table 11-6:

TABLE 11-6 SUMMARY OF FIELD DUPLICATE RC PERCUSSION ASSAY RESULTS, SGS

	Drill Holes	Number of Pairs	Average Grade Original Sample (Au – g/t)	Average Grade Duplicate Sample (Au – g/t)
SGS 2014	CRC004, 014, 018-020, 028, 032-038; KRC001-004	24	<0.01	<0.01

The results for the quarter diamond core and RC percussion chip duplicate data compare well and do not show any significant bias between the duplicate pair analysis results. The grade dispersion is at a scale that is to be expected for duplicate core samples.

CHECK ASSAYING

A protocol was initiated in 2012 to send 5% of all assayed sample pulps to a second laboratory for analysis. At that time, ALS Chemex was the laboratory used for routine analysis of drill samples and Acme Labs, Ankara, Turkey, was selected to provide external check assays. External check assays are an additional means of data verification and are useful in identifying any substantial biases between laboratories, which may have been introduced during the course of the Project. In all cases, CRMs were included in the sample stream to be check assayed. The results of the check assays are compiled in Table 11-7.

TABLE 11-7 SUMMARY OF CHECK ASSAY RESULTS

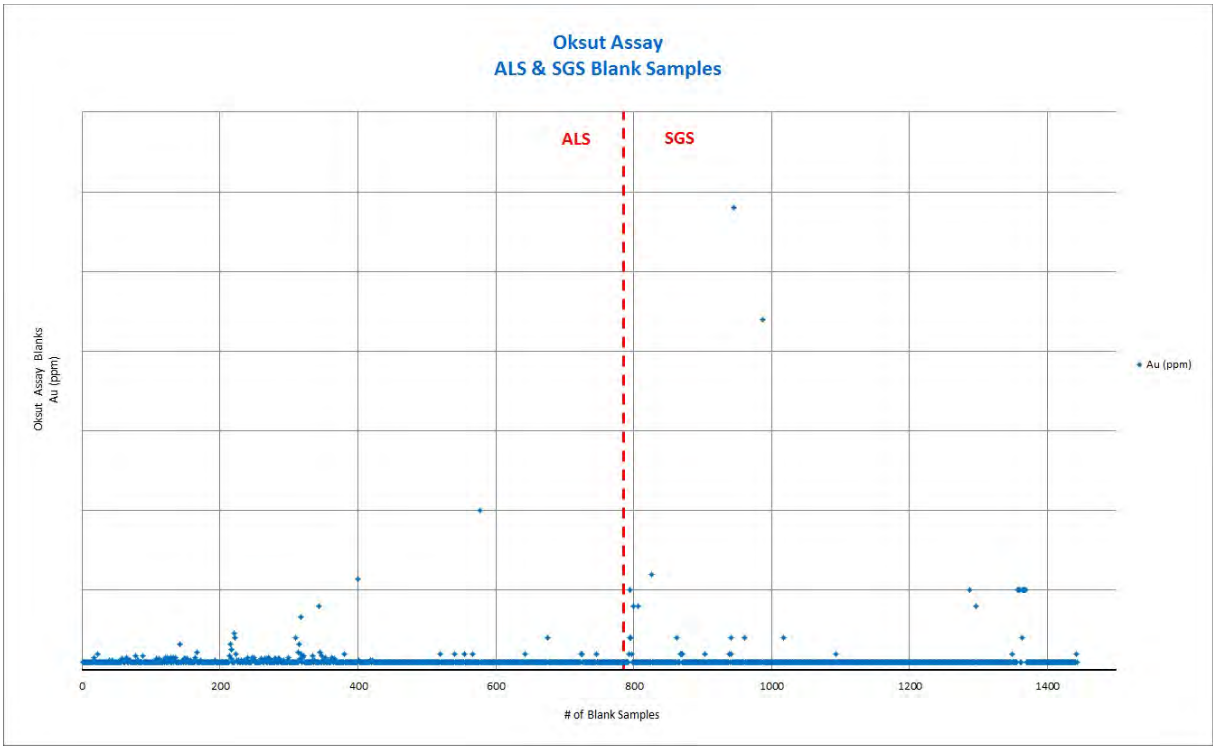
Date	Type of Samples	Number of Samples *(CRMs)*	Primary Laboratory	Au (g/t)	Check Laboratory	Au (g/t)
Early 2012	Pulps	15 *(1)*	ALS Chemex	2.24	Acme Ankara	2.08
	Quarter Core	46 *(1)*	ALS Chemex	3.33	Acme Ankara	3.32
Late 2012	Pulps	159 *(13)*	ALS Chemex	1.92	SGS Ankara	1.83
Late 2013	Pulps	732 *(48)*	SGS Ankara	1.34	Acme Ankara	1.45
Late 2014	Pulps	358 *(23)*	SGS Ankara	1.95	Acme Ankara	2.08

This limited program of check assaying shows reasonable reproducibility of the gold analytical results between the three laboratories and there was no significant bias in the results.

FIELD BLANKS

A total of 1,445 blanks were submitted and analyzed in conjunction with core samples. All blank assay data were graphed for gold (Figure 11-4). Nearly all the blank samples returned gold values less than three times the lower detection limit of 0.005 g/t Au (ALS) or 0.010 g/t Au (SGS). However, 15 field blanks (1.0%) returned unacceptably high values. In the instance of a blank failure, ten samples either side of the affected blank were re-assayed and acceptable results were returned in all instances. The blank results demonstrated that carry-over contamination at the crushing and pulverizing stage has not been a significant issue.

FIGURE 11-4 CORE FIELD BLANK ASSAY RESULTS



Note: Outlier values of 0.64 and 4.45 g/t Au for SGS are not shown on the graph.

A total of 34 blanks were submitted to SGS and analyzed in conjunction with RC percussion chip samples. All the blank samples returned gold values below the lower detection limit (0.010 g/t Au).

AUDITS

A site audit was completed by Lynda Bloom of Analytical Solutions Ltd. (ASL) in May 2013. The purpose of the visit was to:

- Prepare an assay quality control report
- Carry out an audit of the SGS Ankara laboratory
- Undertake a review of drill core sampling and handling
- Conduct assay QC training

Based on the review of QC data and a site visit to the Öksüt Project, ASL considered that “there is no evidence of bias within the current database (at May 2013) which would materially impact a mineral resource estimate”. ASL recommended that the possible small low bias at SGS Ankara be monitored (Bloom, 2013).

In the QP’s opinion, the sample preparation, analysis, and security procedures at the Öksüt Project are adequate for use in the estimation of Mineral Resources.

12 DATA VERIFICATION

HISTORICAL DATABASE

OMAS conducted an internal validation of the databases supplied to it by Stratex and has determined that no significant data entry problems existed. This validation was performed by comparing the gold values from the assay database against gold values from the assay laboratory certificates.

OMAS DATABASE

OMAS conducts comprehensive internal validation of its databases, both in hard copy and electronic format. Drill collars are professionally surveyed. All laboratory results are checked by a qualified geologist before loading into the database and again at regular intervals for long-term trends.

At the end of the 2012, 2013, and 2014 drill seasons, OMAS performed internal database audits on all hard copy and electronic data and determined that no significant data entry problems existed. The analytical data are considered to be sufficiently reliable to support geology and resource modelling. No external audits have been conducted on the Öksüt database, however; a third-party audit is planned for 2016.

In the QPs' opinion, the various steps taken by OMAS to ensure the integrity of analytical data are consistent with standard industry practice. The sampling procedures are appropriate for the style of mineralization and structural controls for the Öksüt Project and are adequate for the estimation of Mineral Resources in this report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

SUMMARY

Metallurgical testing has focused on supporting the development of the Öksüt Project as a heap leach operation. Testing to date has focused on gold recovery at coarse particle sizes. Metallurgical testing was initiated in 2012 using samples from existing exploration diamond drill holes. A second program, completed in 2012, utilized samples from a single large diameter hole to provide the bulk of the sample for this program. The second program included the first column leach tests. In 2013, four large diameter drill holes were drilled (three in the Keltepe Deposit and one in the Güneytepe Deposit) to provide samples for two large scale column leach test programs. A mineralogy program was also completed on the samples from this program. In 2014, a further five large diameter drill holes (one in the Güneytepe Deposit and four in the Keltepe Deposit) were completed to provide samples for additional large scale column leach tests and further mineralogical analysis.

The results from all programs show that samples from the Öksüt Project are amenable to heap leach processing. Leach rates are relatively fast with comparatively high final recoveries. Size by size analysis of the column leach test feed and tails samples shows gold evenly distributed among the size classes, roughly following the mass splits.

Since the Keltepe Deposit contains approximately 90% of the contained gold for the Öksüt Project, the leach characteristics for the Keltepe Deposit will predominate. Güneytepe Deposit leach characteristics are expected to be as good as or better than Keltepe Deposit and are not anticipated to present any issues. Analyzing all the column test results at the estimated Keltepe Deposit average grade of 1.24 g/t Au, the corresponding leach residue grade will be 0.23 g/t Au, with a resulting recovery of 81.4%. For scale-up purposes, this is discounted by 4% and an Adsorption-Desorption-Recovery (ADR) efficiency factor of 99% is applied which results in a design recovery of 77%.

2012 PRELIMINARY LEACH TESTING

A series of five diamond drill hole composite samples from Keltepe were sent to SGS Mineral Services UK Ltd. (Cornwall, UK Laboratory) for a preliminary leach test program (Williams, 2012). The scope of work included:

- Comprehensive head assays;
- Screened metallics gold and silver head assays.
- Leach tests at particle size distributions of 100% passing (D_{100}) 3.35 mm, 1.00 mm, 0.50 mm, 0.15 mm, and 0.075 mm.

Head grades ranged from 0.76 g/t Au to 2.20 g/t Au. The results showed that gold recoveries at the finer particle size distribution are not significantly better than the coarser size distribution. At the coarser size distribution, recoveries ranged from the 58% (transition quartz alunite) to 86% (oxide quartz alunite). At the finest particle size distribution, recoveries ranged from 78% (transition quartz alunite) to 92% (oxide quartz alunite and oxide massive silica).

The results demonstrate that acceptable recoveries can be achieved at coarse particle size distributions.

2012 COLUMN LEACH PROGRAM

Based on the positive results from the 2012 preliminary leach test program, the first column leach test program was initiated. To provide the required sample for the program, a large diameter drill hole (ODD0075) was drilled in the Keltepe Deposit. This hole twinned existing drill hole ODD0048, which was selected as it intersected the three types of mineralization:

- Oxidized massive silica breccia,
- Transition massive silica,
- Oxidized quartz alunite.

The scope of work for this program included:

- Comprehensive head assays, including screened metallics gold and silver assays,
- Coarse bottle rolls tests at two crush sizes: 100% passing 31.5 mm and 12.5 mm,
- Agglomeration tests,
- Column leach tests at two crush sizes: 100% passing 31.5 mm and 12.5 mm.

This program was also conducted at SGS Mineral Services UK Ltd. (Cook, 2013). The work commenced in the fall of 2012 and was concluded in May 2013.

Screened metallic head assays for the three samples are shown in Table 13-1. Samples were again screened at 106 µm to determine the coarse gold content. No significant coarse gold or “nugget effect” was found.

TABLE 13-1 GOLD HEAD ASSAYS FOR 2012 COLUMN LEACH PROGRAM SAMPLES

Sample	Assay (g/t Au)
Oxidized Massive Silica Breccia	2.34
Transition Massive Silica	3.75
Oxidized Quartz Alunite	1.96

The drill hole location was selected to intersect the three predominant types of mineralization. The head grades of the three samples are substantially higher than the overall resource grade.

In general, the following observations can be made on the coarse bottle roll leach test results:

- The higher recoveries were generally achieved at the finer crush size and highest cyanide concentration for the massive silica samples.
- Recoveries were highest for the oxidized quartz alunite sample

The column leach tests were initially scheduled to run for 42 days but extended to 56 days due to the slow leach rates for the two massive silica samples.

The oxidized quartz alunite sample had the highest recoveries, with the coarser crush size producing the highest recovery at 90% gold and 84% gold for the -12.5 mm sample.
The transition massive silica sample had the next highest recoveries at 64% gold for the -31.5 mm sample and the -12.5 mm sample had a recovery of only 50% after 56 days of leaching.

The oxidized massive silica sample had the lowest recoveries with 46% gold for the -31.5 mm sample and 47% for the -12.5 mm sample after 56 days of leaching.

The results demonstrate that the Keltepe samples tested are amenable to leaching at coarse particle sizes that are used in heap leaching. The leach times were typical for gold samples.

2013 COLUMN LEACH TESTS (KCA PHASE 1)

The 2013 column leach program was designed to build on the results from the 2012 program. One of the objectives was to prepare overall composites of the Keltepe and Güneytepe Deposits to further evaluate the effects of crush size and leach time.

Four large diameter diamond drill holes were twinned from existing holes to provide the required samples in sufficient quantities for large scale testing. The drill holes selected resulted in a composite sample that is spatially representative of the deposit with a grade close to the resource grade. The large diameter diamond drill holes and the original holes from which they were twinned are shown in Table 13-2.

TABLE 13-2 DIAMOND DRILL HOLES FOR 2013 COLUMN LEACH TEST PROGRAM

Metallurgical Diamond Drill Hole No.	Original Diamond Drill Hole No.	Deposit
ODD0057	ODD0105	Keltepe
ODD0080	ODD0106	Keltepe
ODD0081	ODD0107	Keltepe
ODD0008	ODD0108	Güneytepe

The drill holes were shipped in their entirety with instructions provided on which intervals to be used in each composite sample.

Kappes Cassiday & Associates (KCA) in Reno, Nevada, was selected to conduct this program due its extensive experience in the testing and development of heap leach projects (KCA, 2013). The scope of work for this program included:

- Comprehensive head analysis.
- Fine pulverized (80% passing 75 µm) bottle roll leach tests.
- Coarse (80% passing 1.7 mm) bottle roll leach tests.
- Preliminary agglomeration testing.

- Compacted permeability testing.
- Coarse column leach test (100% passing 50 mm, 80% passing 37.5 mm) for the Keltepe sample.
- Fine column leach test (100% passing 19 mm, 80% passing 12.5 mm) for the Keltepe and Güneytepe samples.
- Comprehensive head and tails screen analysis of column leach samples, including size by size gold and silver assays.
- Hardness testing for comminution parameters.

Samples from each composite were also sent to SGS Canada Inc. (SGS Canada) for mineralogical and gold deportment studies.

Despite only being coarsely crushed, each sample shows significant content of fines. This is also evident throughout the drill core.

Comprehensive head assays for the two composite samples are shown in Table 13-3.

TABLE 13-3 KELTEPE AND GÜNEYTEPE HEAD ASSAYS; 2013 COLUMN LEACH PROGRAM

Parameter	Keltepe	Güneytepe
Au (g/t)	1.53	1.68
Ag (g/t)	2.91	1.99
Total Carbon (%)	0.03	0.04
Organic Carbon (%)	0.02	0.03
Inorganic Carbon (%)	0.01	0.01
Total Sulphur (%)	0.64	2.74
Sulphide Sulphur (%)	0.41	1.31
Sulphate Sulphur (%)	0.23	1.43
Mercury (g/t)	<0.05	<0.05
Total Copper (g/t)	158	124
Cyanide Soluble Copper (g/t)	54.7	17.1
Cyanide Soluble Copper (% of total)	35	14

Gold assays are slightly higher than the estimated resource grade of 1.4 g/t Au. The Keltepe sample has minimal sulphide content. The higher sulphide content in the Güneytepe sample is in the form of pyrite. Both samples contain very low levels of organic carbon and negligible mercury levels. The copper content of both samples is well below the level at which cyanide consumption becomes an issue, 500 g/t cyanide soluble copper.

Multi-element and whole rock analyses for each sample are found in the KCA report (KCA, 2013).

Summaries of the bottle roll leach test results for gold are presented in Table 13-4. All tests were conducted for 96 hours.

TABLE 13-4 SUMMARY OF BOTTLE ROLL LEACH TEST RESULTS; 2013 PHASE 1 PROGRAM

Sample	Size of Material	Target D_{80} (mm)	Assays (g/t Au)				Gold Extraction (% Au)	Cyanide Consumption (kg/t NaCN)	Lime Addition (kg/t $Ca(OH)_2$)
			Head Grade	Calculated Head Grade	Calculated Extracted Grade	Tails Grade			
Keltepe	-1.70 mm	--	1.53	1.45	1.15	0.30	80%	0.21	1.00
	pulverized	0.075	1.53	1.48	1.39	0.12	92%	0.89	1.75
Güneytepe	-1.70 mm	--	1.68	1.59	1.35	0.24	85%	0.33	4.00
	pulverized	0.075	1.68	1.68	1.53	0.15	91%	0.31	4.00

The recoveries are in the same range as those from the coarse bottle roll leach tests from the 2012 column program, which were higher grade samples. The pulverized samples show very similar recoveries. The coarse Güneytepe sample recovery is unexpectedly higher than the Keltepe sample, which is more intensely oxidized.

Three column leach tests were completed; two for Keltepe and one for Güneytepe. The tests had two set-ups:

- Large diameter column (200 mm x 1.5 m high) for the -50 mm (80% passing -37.5 mm) Keltepe sample.
- Small diameter column (150 mm x 1.9 m high) for the -19 mm (80% passing -12.5 mm) Keltepe and Güneytepe samples.

Key test conditions included:

- Solution application maintained between 0.010 $m^3/h/m^2$ and 0.012 $m^3/h/m^2$ of the column area. This is the typical solution application rate for most full scale heap leach operations.
- Planned leach time of 60 days that was reduced to 56 days.
- Initial cyanide solution addition of 1.0 g/L NaCN, reduced to 0.60 g/L NaCN.
- Hydrated lime used for pH control.

The results for the column leach tests are shown in Table 13-5.

TABLE 13-5 COLUMN LEACH TEST RESULTS; 2013 COLUMN LEACH PROGRAM

Sample	Calculated Head Grade (g/t Au)	Gold Extracted (g/t Au)	Weighted Average Screen Tails (g/t Au)	Gold Extraction (% Au)	Calculated Tail k_{80} Size (mm)	Cyanide Consumption (kg/t NaCN)	Lime Addition (kg/t $Ca(OH)_2$)
Keltepe Composite	1.46	1.16	0.306	80%	36	0.78	1.01
Keltepe Composite	1.42	1.11	0.32	78%	12	0.62	1.50
Güneytepe Composite	1.49	1.22	0.27	82%	13	0.88	4.05

The weighted average screen tails is the weighted average assay obtained from the size by size analysis of the leach tails. Size by size analysis of the leach feed samples showed gold distributed across the size fractions in proportion to the weight in each fraction. The leach tailings show a similar pattern.

While the proportion of each mineralization type in the composite samples has not been determined, the overall recoveries were not impacted by the expected low recoveries from the two massive silica mineralization types.

The presence of pyrite in the Güneytepe sample has not impacted the gold recovery, indicating either minimal gold occurring with pyrite or sufficient pyrite oxidation to expose any contained gold.

Based on KCA's experience, actual cyanide consumptions are approximately 30% of column test results.

Samples from each of the composites were sent to SGS Canada for mineralogical studies, gold deportment analysis and diagnostic leach tests (SGS, 2014).

Key highlights from this program include:

- Gold occurs as native gold in both the Keltepe and Güneytepe samples.
- The observed gold grains were 85% liberated and exposed in the Keltepe sample and 69% in the Güneytepe sample. The amount of free and exposed gold in the Güneytepe sample is anomalous since gold recovery in the leach tests was well above 80%.
- The average gold grain size was 6.9 µm in the Keltepe sample and 5.4 µm in the Güneytepe sample. This confirms the results of the screened metallic head assays with minimal gold in the coarse fraction.
- In the Keltepe sample, the host minerals are mainly quartz and other silicates and iron oxide, with minor (2% to 10%) rutile-silicate complexes and trace associations with pyrite.
- The major host mineral in the Güneytepe sample is alunite, with trace (<2%) associations with quartz, silicates, and iron oxide particles.

Diagnostic leach tests showing the gold associations are listed in Table 13-6.

TABLE 13-6 DIAGNOSTIC LEACH TEST RESULTS; 2013 COLUMN LEACH PROGRAM

Gold Association	Keltepe	Güneytepe
Free Gold	91.1	89.9
Associated with Sulphides	5.5	10.1
Locked in Silicates	3.5	-

2014 COLUMN LEACH TESTS

KCA PHASE 2

The next set of column tests (known as KCA Phase 2) was completed between May and September 2014. An additional five composite samples were assembled from the four 2013 large diameter metallurgical drill hole samples for the program. The five composite samples are shown in Table 13-7.

TABLE 13-7 KCA PHASE 2 COMPOSITE SAMPLE HEAD GRADES

Parameter	Keltepe			Güneytepe	
	Low Grade	Mid Grade	High Grade	Low Grade	High Grade
Au (g/t)	0.82	2.05	2.34	0.77	3.20
Ag (g/t)	0.74	1.77	2.88	0.89	3.11
Total Carbon (%)	0.04	0.03	0.01	0.03	0.05
Organic Carbon (%)	0.04	0.03	0.01	0.03	0.05
Inorganic Carbon (%)	<0.01	<0.01	<0.01	<0.01	<0.01
Total Sulphur (%)	0.44	1.10	0.41	3.67	1.57
Sulphide Sulphur (%)	0.30	0.77	0.35	2.13	0.71
Sulphate Sulphur (%)	0.13	0.33	0.06	1.54	0.86
Mercury (g/t)	0.73	<0.05	<0.05	<0.05	<0.05
Total Copper (g/t)	230	156	123	130	131
Cyanide Soluble Copper (g/t)	18.8	32.2	30.5	17.9	6.0
Cyanide Soluble Copper (% of total)	8	21	25	14	5

The scope of work was identical to the Phase 1 program, however, no hardness and comminution tests and no mineralogical studies were carried out (KCA, 2014a).

Results of the coarse and fine bottle roll leach tests are shown in Table 13-8. All leach tests were conducted for 96 hours. The results follow the trends of previous similar tests with higher recoveries at the finer particle sizes.

TABLE 13-8 PHASE 2 COARSE AND FINE BOTTLE ROLL LEACH TEST RESULTS

Sample	Size of Material	Target D_{80} (mm)	Assays (g/t Au)				Extracted (% Au)	Reagent Consumptions	
			Head Grade	Calc Head Grade	Calc. Extracted Grade	Tails Grade		(kg/t NaCN)	(kg/t $Ca(OH)_2$)
Keltepe Low Grade	-1.70 mm	--	0.85	0.89	0.61	0.28	68	0.10	1.50
	pulverized	0.075	0.85	0.81	0.69	0.12	85	0.45	1.75
Keltepe Mid Grade	-1.70 mm	--	2.00	1.95	1.52	0.43	78	0.39	1.00
	Pulverized	0.075	2.00	1.93	1.73	0.20	90	1.70	1.50
	-1.70 mm	--	2.33	2.14	1.70	0.44	79	0.41	0.50

Sample	Size of Material	Target D_{80} (mm)	Assays (g/t Au)				Extracted (% Au)	Reagent Consumptions	
			Head Grade	Calc Head Grade	Calc. Extracted Grade	Tails Grade		(kg/t NaCN)	(kg/t $Ca(OH)_2$)
Keltepe High Grade	Pulverized	0.075	2.33	2.20	2.01	0.19	91	1.34	1.50
Güneytepe Low Grade	-1.70 mm	--	0.81	0.74	0.58	0.16	78	0.36	4.00
	Pulverized	0.075	0.81	0.74	0.64	0.10	87	0.20	5.50
Güneytepe High Grade	-1.70 mm	--	3.40	3.03	2.51	0.52	83	0.31	4.25
	Pulverized	0.075	3.40	3.16	2.73	0.42	87	0.22	6.00

Column leach tests were completed under the same conditions as the Phase 1 program. All samples were crushed to -50 mm and leached in 200 mm columns for 62 days. Final results are summarized in Table 13-9.

TABLE 13-9 PHASE 2 COLUMN LEACH TEST RESULTS

Sample	Calculated Head Grade (g/t Au)	Gold Extracted (g/t Au)	Weighted Average Screen Tails (g/t Au)	Gold Extraction (% Au)	Calculated Tail k_{80} Size (mm)	Cyanide Consumption (kg/t NaCN)	Lime Addition (kg/t $Ca(OH)_2$)
Keltepe Low Grade	0.81	0.60	0.21	74	34.9	0.79	1.02
Keltepe Mid Grade	1.98	1.53	0.44	78	39.3	0.82	1.01
Keltepe High Grade	2.45	1.91	0.54	78	37.9	0.75	1.00
Güneytepe Low Grade	0.90	0.68	0.22	76	35.1	1.12	3.08
Güneytepe High Grade	2.90	2.57	0.33	89	40.0	1.18	3.04

Compared to the Phase 1 column test results, the Keltepe sample recoveries were slightly lower overall. The Güneytepe recoveries were consistent with the single Phase 1 test result.

KCA PHASE 3

The next set of column tests (known as KCA Phase 3) was completed between July and December 2014. Six large diameter diamond drill holes were twinned from existing holes to provide the required samples in sufficient quantities for large scale testing. The drill holes selected to provide five Keltepe composite samples that are representative of the two major mineralization types (three quartz alunite samples and two massive silica samples) at the lower grade ranges. A single lower grade quartz alunite composite sample was produced for

Güneytepe. The large diameter diamond drill holes and the original holes from which they were twinned are shown in Table 13-10.

TABLE 13-10 DIAMOND DRILL HOLES FOR PHASE 3 COLUMN LEACH TEST PROGRAM

Metallurgical Diamond Drill Hole No.	Original Diamond Drill Hole No.	Deposit
ODD0187	ODD0003	Keltepe
ODD0188	ODD0097	Keltepe
ODD0189	ODD0126	Keltepe
ODD0190	-	Güneytepe
ODD0190A	ODD0164	Güneytepe
ODD0191	ODD0153	Keltepe

The six composite samples are shown in Table 13-11 along with their head assays. All of the actual gold assays came in significantly higher than the expected gold assays.

TABLE 13-11 KCA PHASE 3 COMPOSITE SAMPLE HEAD GRADES

Parameter	Güneytepe Quartz Alunite	Keltepe				
		Massive Silica	Massive Silica	Quartz Alunite	Quartz Alunite	Quartz Alunite
Au (g/t) (expected)	0.55	0.35	0.76	0.35	0.55	0.73
Au (g/t) (weight average actual)	0.67	0.91	1.12	0.49	1.26	1.30
Ag (g/t)	0.52	0.60	1.58	1.41	0.66	0.80
Total Carbon (%)	0.05	0.03	0.10	0.02	0.03	0.02
Organic Carbon (%)	0.05	0.02	0.09	0.01	<0.01	0.02
Inorganic Carbon (%)	<0.01	<0.01	<0.01	<0.01	0.02	<0.01
Total Sulphur (%)	2.51	0.82	0.16	1.79	0.49	0.87
Sulphide Sulphur (%)	1.18	0.44	0.11	0.86	0.38	0.57
Sulphate Sulphur (%)	1.33	0.38	0.06	0.93	0.11	0.30
Mercury (g/t)	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Total Copper (g/t)	84	66	170	25	87	62
Cyanide Soluble Copper (g/t)	9	21.5	56.9	6.8	15.8	13.4
Cyanide Soluble Copper (% of total)	10	33	33	27	18	22

The scope of work for Phase 3 was identical to the Phase 1 program (KCA, 2014b).

Results of the coarse and fine bottle roll leach tests are shown in Table 13-12. All leach tests were conducted for 96 hours. The results follow the trends of previous similar tests with higher recoveries at the finer particle sizes.

TABLE 13-12 PHASE 3 COARSE AND FINE BOTTLE ROLL LEACH TEST RESULTS

Sample	Size of Material	Target D_{80} (mm)	Assays (g/t Au)				Extracted (% Au)	Reagent Consumptions	
			Head Grade	Calc Head Grade	Calc. Extracted Grade	Tails Grade		(kg/t NaCN)	(kg/t $Ca(OH)_2$)
Güneytepe Quartz Alunite	-1.70 mm	--	0.71	0.70	0.61	0.09	87	0.04	3.00
	pulverized	0.075	0.71	0.57	0.50	0.07	88	0.98	3.50
Keltepe Massive Silica	-1.70 mm	--	0.93	0.91	0.73	0.18	80	0.14	2.00
	Pulverized	0.075	0.93	0.94	0.82	0.12	87	1.61	2.00
Keltepe Massive Silica	-1.70 mm	--	1.03	1.07	0.75	0.32	70	0.14	1.00
	Pulverized	0.075	1.03	1.02	0.90	0.12	88	1.29	1.00
Keltepe Quartz Alunite	-1.70 mm	--	0.55	0.50	0.32	0.18	64	0.22	1.50
	Pulverized	0.075	0.55	0.49	0.41	0.08	84	0.22	1.50
Keltepe Quartz Alunite	-1.70 mm	--	1.23	1.20	0.97	0.22	81	0.22	1.50
	Pulverized	0.075	1.23	0.88	0.81	0.07	92	1.74	2.00
Keltepe Quartz Alunite	-1.70 mm	--	1.32	1.26	0.83	0.43	66	0.22	1.50
	Pulverized	0.075	1.32	1.16	0.96	0.20	83	2.36	1.50

Column leach tests were completed under the same conditions as the Phase 1 program. All samples were crushed to -50 mm and leached in 200 mm columns for 62 days. Final results are summarized in Table 13-13.

TABLE 13-13 PHASE 3 COLUMN LEACH TEST RESULTS

Sample	Calculated Head Grade (g/t Au)	Gold Extracted (g/t Au)	Weighted Average Screen Tails (g/t Au)	Gold Extraction (% Au)	Calculated Tail k80 Size (mm)	Reagent Consumptions	
						(kg/t NaCN)	(kg/t Ca(OH)2)
Güneytepe Quartz Alunite	0.78	0.68	0.10	88	38.2	1.02	3.49
Keltepe Massive Silica	0.95	0.74	0.21	78	37.1	1.05	2.01
Keltepe Massive Silica	1.09	0.69	0.40	63	37.6	0.79	1.00
Keltepe Quartz Alunite	0.60	0.35	0.25	59	39.6	0.87	2.62
Keltepe Quartz Alunite	1.19	0.96	0.23	81	35.7	0.91	1.91
Keltepe Quartz Alunite	1.28	0.79	0.49	62	37.1	1.07	1.51

The Keltepe sample recoveries were unexpected. Massive silica mineralization was considered somewhat refractory; one of the two tests had low recovery while the other had good recovery. Two of the quartz alunite samples had low recoveries. Quartz alunite mineralization is not considered problematic. The Güneytepe recoveries were in agreement with the earlier tests.

Size by size analysis of the column leach feed samples and the column leach residue samples follow a consistent trend. Gold is relatively evenly distributed in each size class, with no concentration of gold in the finest size classes as is often commonly seen.

All five samples from the Phase 3 program were submitted to SGS Canada for mineralogical analysis and gold diagnostic leach tests (SGS, 2015).

The predominant mineral in all samples was quartz. The Güneytepe sample had the highest alunite content of all the samples at 15% and the lowest quartz content of the samples at 50%. The Güneytepe sample had an observed pyrite content of only 0.33%, although it had the highest sulphide sulphur content of 1.2%. The Güneytepe sample also had 18% muscovite. The next most abundant minerals in the Keltepe samples are alunite, kaolinite, and muscovite.

In terms of gold deportment, all samples showed very high levels of liberated plus exposed gold (76% to 98%) with the exception of the Keltepe lowest grade quartz alunite sample, which showed a locked gold content of 87%. This sample had the lowest column leach recovery of

59%, despite the very high observed locked gold content. Observed gold particles were quite fine, ranging in size from 6 µm to 20 µm.

These (including Phase 1) results in combination with the size by size analysis for all phases suggest that the oxide nature of the Öksüt Project and the fine nature of the gold allows for good exposure to leach solution and relatively rapid leaching.

AGGLOMERATION AND COMPACTED PERMEABILITY TESTS

Preliminary agglomeration test work as well as compacted permeability test work was conducted on portions of crushed material from each composite. For the test work, the material was agglomerated with various additions of cement. In the preliminary agglomeration testing, the agglomerated material was placed in a column with no compressive load and then tested for permeability. In the compaction testing, the agglomerated material was compacted in a column with a predetermined static load and then tested for permeability.

The agglomeration and compacted permeability tests showed that under all conditions, flow was maintained at close to three orders of magnitude times the typical heap leach solution application rates. This included loads equivalent to 50 m of lift height.

This type of agglomeration test work is very preliminary but does serve to provide an indication of whether or not agglomeration will be required for the processing of the material at the tested crushed sizes. All agglomeration tests passed the criteria put forth by KCA. No agglomeration was required for both composite samples in the planned column tests.

Compacted permeability tests were completed in the Phase 1 and Phase 3 KCA programs. Compacted permeability tests examine the permeability of the crushed material under compaction loading equivalent to overall heap heights of 10 m, 20 m, and 50 m. KCA has developed parameters to predict material performance in a full scale heap leach. Sample slump in the column is measured in the column after pressure is applied to the simulated height. Slump in excess of 10% is generally an indication of failure. Typical heap leach irrigation rates are 0.010 $m^3/h/m^2$ to 0.012 $m^3/h/m^2$. Minimum flows under load of 10 times these irrigation rates are required for successful heap leach operation.

All Öksüt samples tested showed maximum slumps of <5%. Solution flows under all load conditions were typically 400 to 600 times the typical irrigation rates. The visible amounts of fines in most Öksüt samples do not interfere with solution flow through the column, even at the 50 m equivalent loads.

HARDNESS AND COMMUNITION TESTS

Standard hardness and communition tests were performed on the KCA Phase 1 and 3 samples. The tests included:

- Bond crusher impact work index
- Bond rod mill work index
- Bond ball mill work index
- Bond abrasion index

The data generated from these tests can be used for crusher selection and is indicative of potential wear problems in materials handling equipment. Table 13-14 shows a summary of the results. The Bond ball mill work indices would be categorized as moderate to hard. Some of the higher Bond abrasion indices would be classed as highly abrasive, likely from the high silica content.

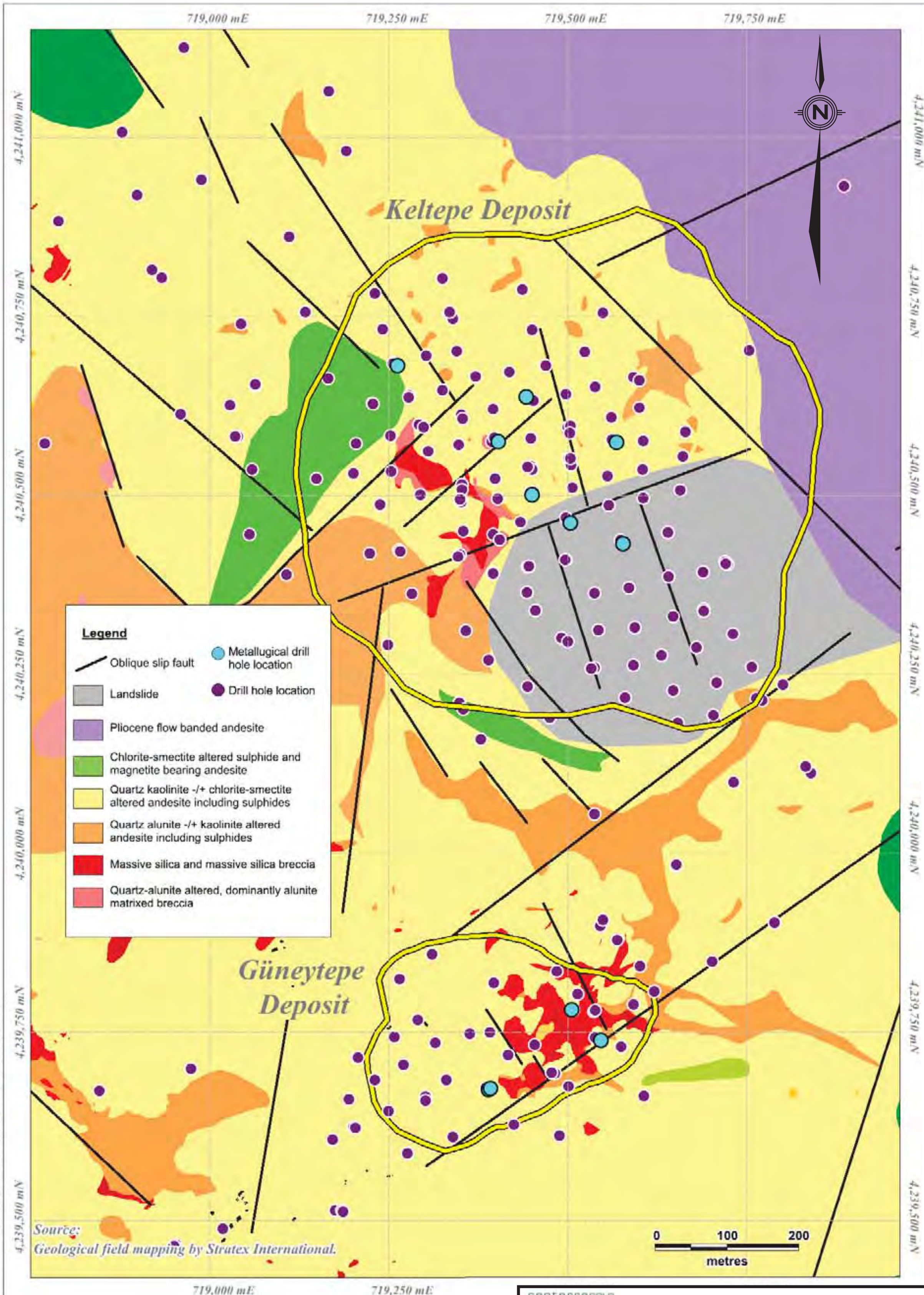
TABLE 13-14 HARDNESS AND COMMINUTION TEST RESULTS

Test	Average	Range
Bond Crusher Impact Work Index	6.51 kWh/t	5.28 – 8.09 kWh/t
Bond Rod Mill Work Index	11.80 kWh/t	8.4 – 13.7 kWh/t
Bond Ball mill Work Index	16.53 kWh/t	13.6 – 18.2 kWh/t
Bond Abrasion Index	0.183	0.085 - 0.318

SAMPLE SELECTION AND REPRESENTATIVENESS

Figure 13-1 shows the metallurgical drill hole locations of the KCA Phase 1, 2, and 3 programs in the Keltepe and Güneytepe Deposits.

The diamond drill holes selected for the KCA metallurgical test programs were selected to spatially represent the Keltepe and Güneytepe deposits. Intervals from the drill holes were selected to provide composite samples that represented both grade ranges and the predominant types of mineralization for metallurgical testing. The samples are considered representative of each deposit.



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Öksüt Gold Project, Turkey
2015 Technical Report

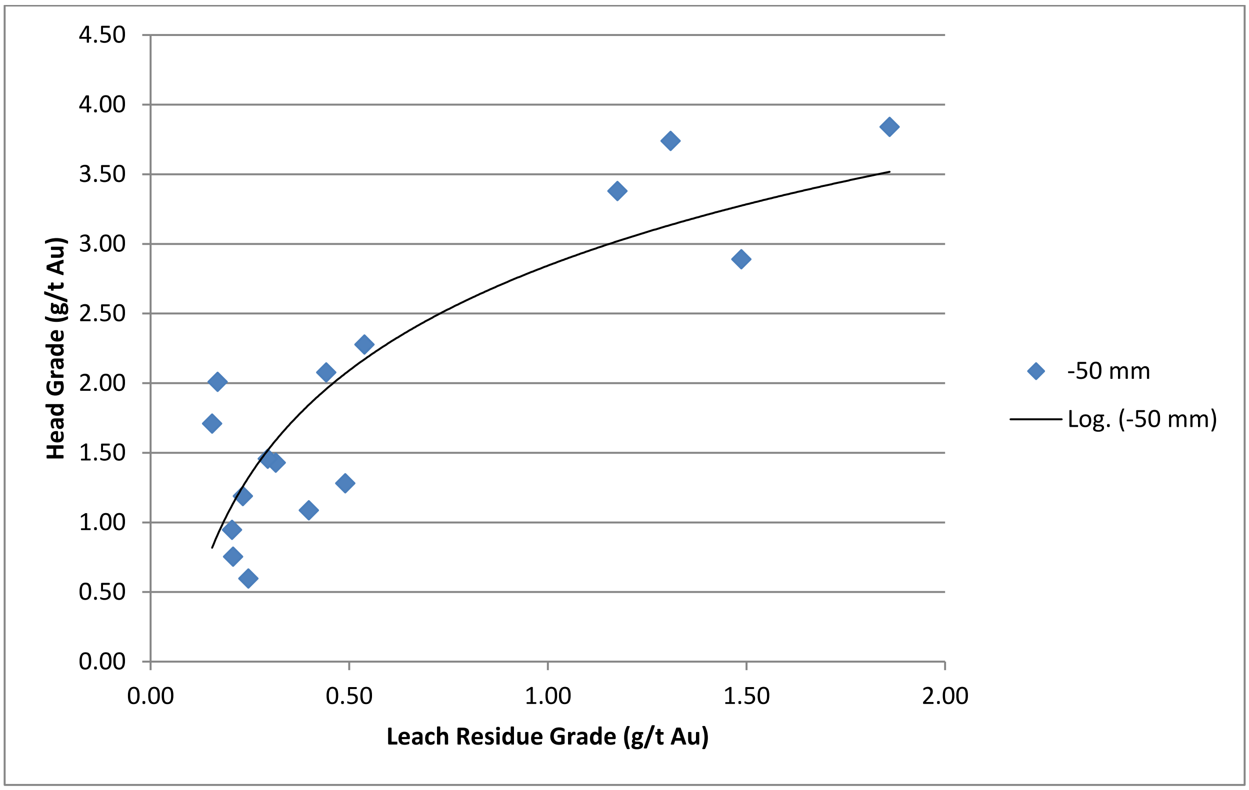
Metallurgical Drill Hole Locations

Date: Sept. 2015	File: MetHoles.cdr	Figure 13-1

LEACH RECOVERY ANALYSIS

Results from all of the column tests were analyzed to estimate recovery for process design, mine scheduling, and economic analysis. Analysis of the results showed that recoveries were not a function of crush size, so tests from both crush sizes were included. Results from Keltepe samples were analyzed separately from results from Güneytepe samples. Figure 13-2 shows the results of Keltepe column tests, with sample head grade plotted as a function of leach residue grades.

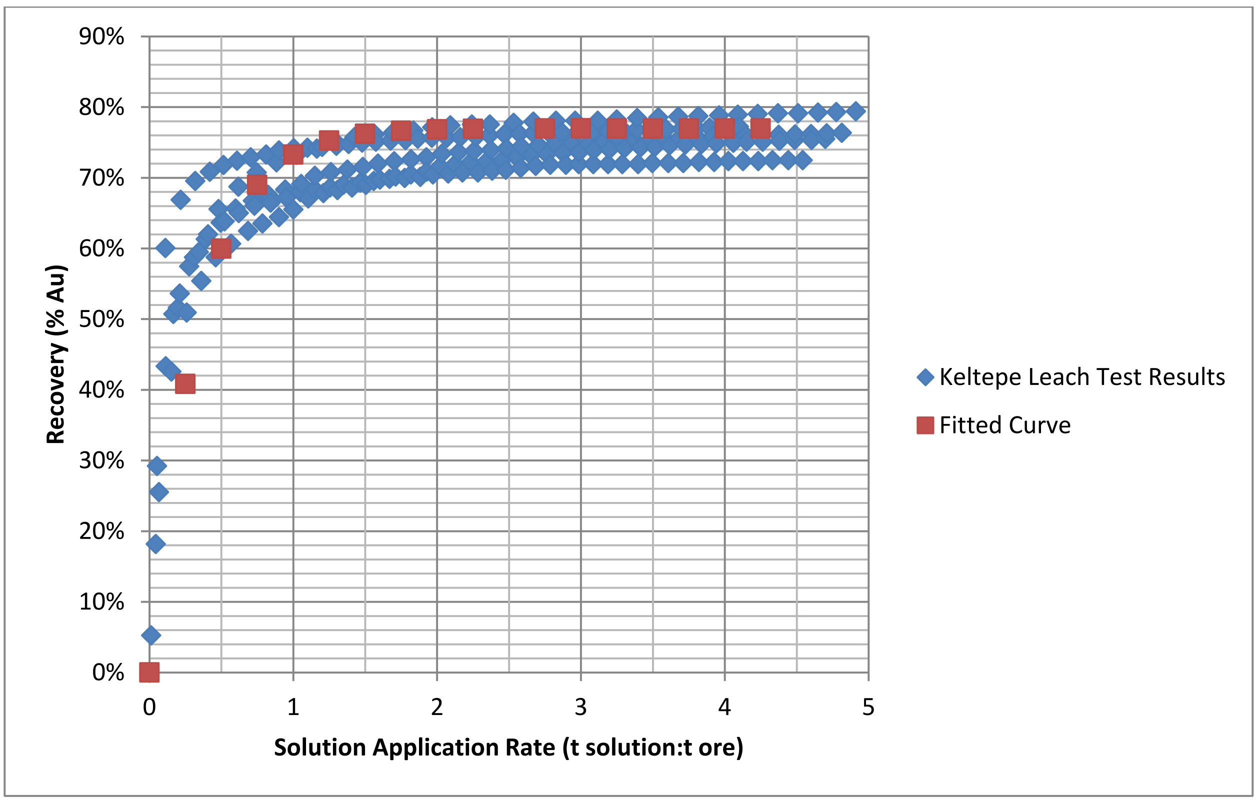
FIGURE 13-2 KELTEPE COLUMN TEST HEAD GRADE VS. LEACH RESIDUE



The Keltepe Deposit contains approximately 90% of the contained gold for the Öksüt Project and, as such, only the leach characteristics for Keltepe have been considered here. Güneytepe leach characteristics are as good as or better than Keltepe and are therefore expected to be consistent with leach characteristics from Keltepe. A best fit curve was fitted to the data. At the estimated Keltepe average grade of 1.24 g/t Au, the corresponding leach residue grade will be 0.23 g/t Au, with a resulting recovery of 81.4%. For scale-up purposes, this is discounted by 4% and an ADR efficiency factor of 99% is applied which results in a design recovery of 77%.

Leach profiles were plotted to determine leach cycle time. Leach cycle times for full scale heap leach operations is typically measured in tonnes of leach solution applied to tonnes of ore under leach. The full leach cycle is not normally completed with a single continuous application of solution. The cycle is usually broken down into the primary leach cycle where solution is directly applied to the ore under leach and a secondary leach cycle where solution flows through an area previously leached from a lift above. Figure 13-3 shows gold recovery as a function of solution application for the Keltepe column tests.

FIGURE 13-3 KELTEPE PHASE 1 AND 2 COLUMN LEACH TEST RESULTS



The fitted curve shows that at a 1:1 solution application rate, gold recovery is 73% or 95.2% of ultimate recovery (77%). Full recovery is achieved between a 3:1 and 4:1 solution application rate. The primary leach cycle includes a solution application rate of 1:1. The remainder of the recovery would be obtained during secondary leaching as ore in subsequent lifts above is leached. The design final solution application rate is 3:1.

There is a correlation between the solution application rate and days of leaching, the latter derived from the heap lift height (10 m), design cell size for each primary leach cycle (500,000 t), and the solution irrigation rate of 0.012 $m^3/h/m^2$. The primary leach cycle will be 46 days and the ultimate leach cycle will be 140 days (secondary leach will be 99 days).

To the QP's knowledge, there are no processing factors or deleterious elements that could have a significant effect on potential economic extraction

14 MINERAL RESOURCE ESTIMATE

Centerra has carried out a Mineral Resource update for the Öksüt Project using a block model constrained with 3D wireframes of the principal mineralized domains. Values for gold were interpolated into blocks using ordinary kriging (OK). The estimate is summarized in Table 14-1.

TABLE 14-1 MINERAL RESOURCE ESTIMATE AT JUNE 30, 2015 (EXCLUSIVE OF MINERAL RESERVES)

Mineral Resources									
Measured and Indicated Mineral Resources									
(tonnes and ounces in thousands)									
	Measured			Indicated			Total Measured and Indicated		
Deposit	**Tonnes**	**Grade (g/t)**	**Contained Gold (oz)**	**Tonnes**	**Grade (g/t)**	**Contained Gold (oz)**	**Tonnes**	**Grade (g/t)**	**Contained Gold (oz)**
Keltepe	2,024	0.7	44	4,450	0.7	106	6,474	0.7	150
Güneytepe	76	0.5	1	248	0.7	5	324	0.6	7
Total	**2,100**	**0.7**	**46**	**4,698**	**0.7**	**111**	**6,798**	**0.7**	**157**

Inferred Mineral Resources			
(tonnes and ounces in thousands)			
Deposit	**Tonnes**	**Grade (g/t)**	**Contained Gold (oz)**
Keltepe	1,705	0.8	44
Güneytepe	675	1.0	21
Total	**2,380**	**0.8**	**64**

Notes:
1. CIM definitions were followed for classification of Mineral Resources.
2. Mineral Resources are in addition to Mineral Reserves. Mineral Resources do not have demonstrated economic viability.
3. Mineral Resources are constrained within an optimized pit shell based on a gold price of $1,450 per ounce.
4. Mineral Resources are estimated based on a cut-off grade of 0.2 g/t Au.
5. Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the Inferred Mineral Resources will ever be upgraded to a higher category.
6. A conversion factor of 31.10348 grams per ounce of gold is used in the reserve and resource estimates.
7. Numbers may not add up due to rounding.

The resource model update for the Öksüt Project was prepared in May 2015 by Centerra using all of the drill holes available as of that date. ARANZ Leapfrog software was used to update the principal mineralized domains at the Keltepe and Güneytepe Deposits and values for gold were interpolated into blocks using OK in GEOVIA GEMS software.

Centerra updated the geological interpretation, and block modelling procedures based on a third-party audit and is of the opinion that the Mineral Resource estimates are appropriate for the style of mineralization and that the resource models are reasonable and acceptable to support the 2015 Mineral Resource and Mineral Reserve estimates.

GEOLOGICAL AND STRUCTURAL MODELS

A detailed geological data compilation was carried out by Centerra geologists to identify major geological contacts, mineralization, weathering, and structural features. These data were used to interpret the primary mineralized domains for the Keltepe and Güneytepe Deposits at the Öksüt Project. Geological logs document rock type, weathering type, alteration type, structural features, and geotechnical characteristics.

WEATHERING MODEL

Using a combination of geological logs, core photos, and the percent of total sulphur (S%), Centerra created a simplified weathering model consisting of oxidized and fresh rock. A small number of intervals that had been logged as “partially oxidized” were assigned to the oxidized or fresh categories based on their core photos and their S% values. The oxidized-fresh rock surface roughly corresponds to values greater than 1.5% total sulphur, however, there are a number of smaller areas where the core photos show complete oxidation despite values >1.5% S. Although, further assay analysis to differentiate between sulphide sulphur and non-sulphide sulphur would help resolve the oxidation state in some specific areas, Centerra considers the current surface to be acceptable to support the current Mineral Resource.

ROCK TYPE MODEL

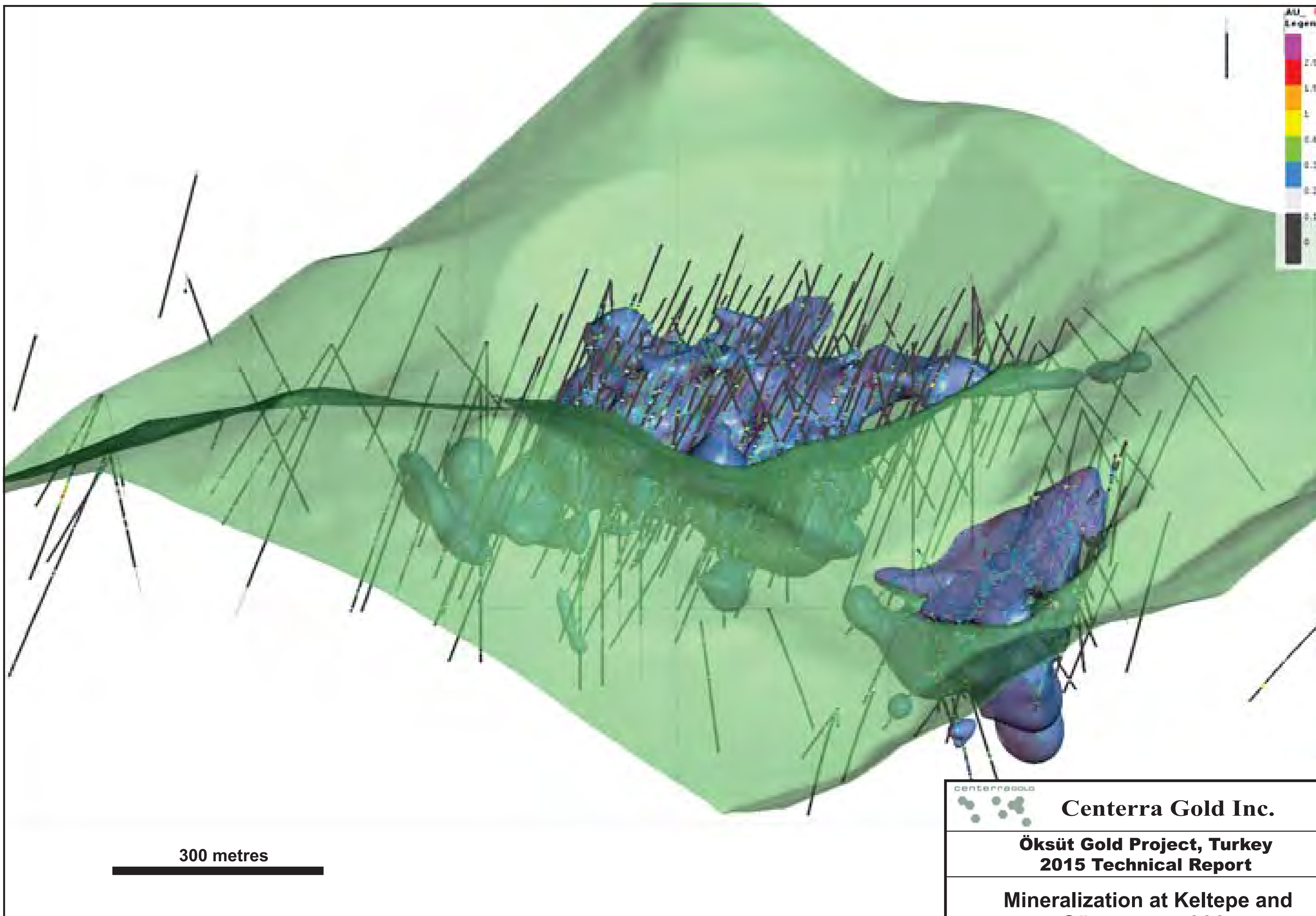
Using a combination of geological logs and core photos, Centerra created a simplified rock type model consisting of four main rock types (breccia, andesite, overburden, and landslide). A number of intervals logged and modelled as overburden are mineralized with similar tenor to the underlying andesite. Visual review of core photos to resolve this issue proved inconclusive and Centerra chose to treat both mineralized overburden and mineralized andesite as a single unit. This methodology was also applied to the unmineralized variety of each rock type.

MINERALIZATION MODEL

Mineralization at Öksüt is believed to be related to the contemporaneous emplacement of a subvolcanic breccia and a porphyritic andesite. A 2012 site visit by Dr. Richard Sillitoe described gold mineralization as being *"preferentially precipitated in the siliceous ledge and its immediate quartz-alunite halo,"* and having been *"fed by at least one, and probably several, steep structures represented by massive silicification."*

Mineralized domains were generated from drill hole intersections with a minimum width of 5 m and a cut-off grade of 0.20 g/t Au. There were a number of instances where lower grade intercepts were included at the edges of the wireframe or within larger intercepts in order to maintain continuity between drilling fences. The interpreted upper and lower boundaries of the mineralized domains were snapped to drill holes.

There are two main mineralized wireframes that represent the mineralization at Keltepe and Güneytepe, respectively (Figure 14-1).



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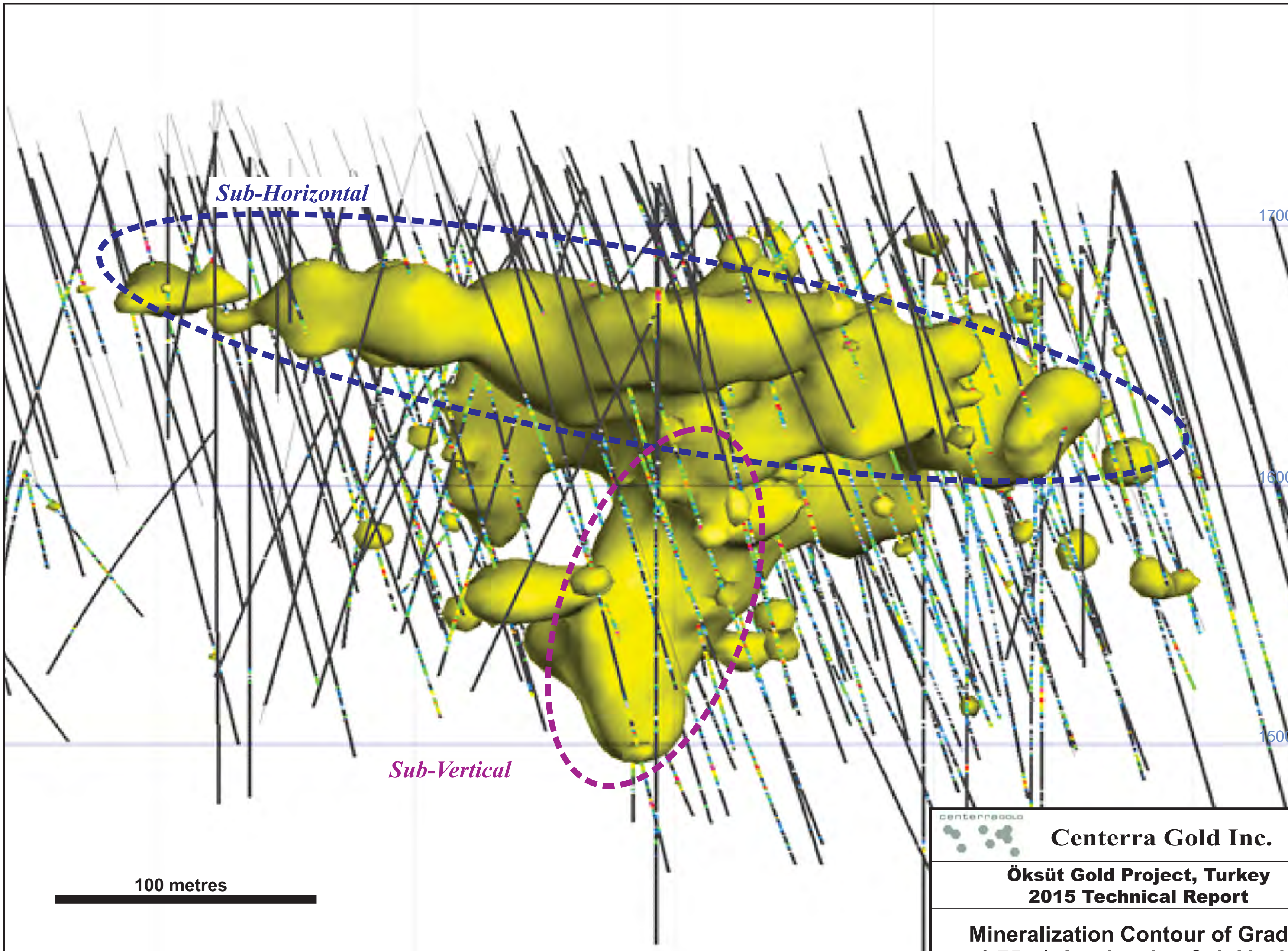
2015 Technical Report

Mineralization at Keltepe and Güneytepe within $1450 Resource Pit Shell

Date: Sept. 2015	File: MinKelGun.cdr	Figure 14-1

The gold oxide mineralization at Keltepe extends approximately 950 m in the north-northwesterly direction and approximately 370 m in the east-northeasterly direction and extends approximately 300 m below surface. The mineralization has two general orientations, a sub-vertical orientation that corresponds to steep feeder structures and a sub-horizontal orientation that corresponds to gold mineralization that was precipitated in the siliceous ledge and its immediate quartz-alunite halo. The sub-horizontal mineralization has a general strike direction of 340º and dips between horizontal and -10º towards the southwest. The sub-vertical mineralization has a general strike direction of 340º and plunges between -50º and -75º towards the southeast.

The gold oxide mineralization at Güneytepe lies approximately 300 m to the southwest of Keltepe. The mineralization extends approximately 370 m in the north-northwesterly direction and approximately 335 m in the east-northeasterly direction and extends approximately 110 m below surface. Similar to Keltepe, the mineralization at Güneytepe is interpreted as having two general orientations, a sub-vertical orientation that corresponding to steep feeder structures and a sub-horizontal orientation that corresponds to gold mineralization that was precipitated in the siliceous ledge and its immediate quartz-alunite halo. The sub-horizontal mineralization has a general strike direction of 085º and dips between horizontal and -25º towards the west. The sub-vertical mineralization has a general strike direction of 265º and dips -75º towards the southeast (Figure 14-2).



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Mineralization Contour of Grades >0.75 g/t Au showing Sub-Vertical and Sub-Horizontal Orientations

Date: Sept. 2015	File: MinContour.cdr	Figure 14-2

Table 14-2 lists the mineralized sub-domains used to code the Rock Type block model attribute.

TABLE 14-2 ROCK TYPE DOMAINS

Deposit	Description	Orientation	Rock Type (Code)
Keltepe	North-northwest trending structures	Sub-Horizontal Mineralization	500
		Sub-Vertical Mineralization	501
Güneytepe	East-northeast trending structures	Sub-Horizontal Mineralization	600
		Sub-Vertical Mineralization	601

ALTERATION MODEL

The alteration model for the Öksüt deposits is currently being updated based on a re-interpretation of the alteration logging. The previous interpretation amalgamated three of the main logged alterations into a single alteration wireframe. Further analysis using logged alteration types to flag multi-element ICP assay data suggests that two of the logged alterations (quartz-kaolinite and quartz-alunite-kaolinite) have a materially different geochemical composition than the third (quartz-alunite). It is Centerra's intention to develop a new model that takes the geochemical composition into consideration.

BULK DENSITY

A total of 742 samples for bulk density measurements have been collected from drill core from the Keltepe and Güneytepe Deposits. Samples for bulk density determination were taken from the main alteration types and from waste rocks. The first three batches of samples (137, 99, and 119 samples, respectively) were sent to the Rock Mechanics Laboratory of the Mining Engineering Department at the Middle East Technical University (METU) in Ankara, Turkey. The measurements were carried out in accordance with the standards suggested by American Society for Testing and Materials (ASTM), Designation: C914-95 "Standard Test Method for Bulk Density and Volume of Solid Refractories by Wax Immersion".

The third batch of samples (202 samples) was sent in 2013 to ALS Chemex in Vancouver, Canada, for bulk density determination using the wax immersion method (ALS method OA-GRA09a).

Following the receipt of a number of seemingly spurious values from the ALS measurements, 200 samples out of 202 were dispatched to SGS Lakefield to check bulk density determinations. The remaining two samples were not repeated as they were broken and hence, not useable for bulk density measurement.

In 2014, a total of 185 samples (108 samples from Keltepe and 77 samples from Güneytepe) were sent to SGS in Ankara, Turkey for bulk density determination using the wax immersion method referred to above.

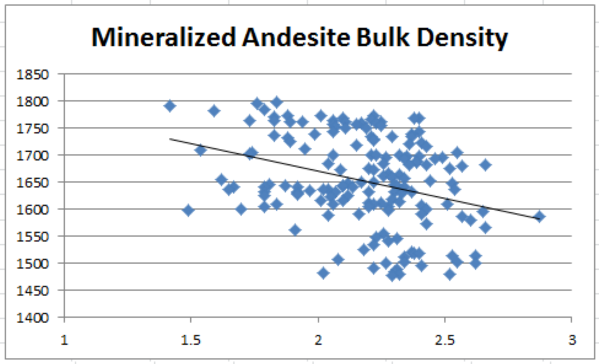
A batch of the METU sample results was inadvertently omitted from analysis due to a clerical error. Accordingly, Centerra's density study only considers the results of 621 bulk density measurements rather than the complete set of 742 samples. Centerra does not believe that the omitted samples will have a material impact on the Mineral Resource estimate, but future Mineral Resource updates will make use of the complete dataset.

Centerra's analysis suggests that a statistically supported argument can be made for sub-division of the density data on the basis of mineralized versus unmineralized material, rock type and weathering type. In line with the weathering model, it appears that elevation has the most significant impact on density. Therefore, Centerra elected to sub-divide the bulk density samples on the basis of mineralized versus unmineralized material and rock type and used a flat-lying search ellipse to accommodate for increasing density with depth. Figure 14-3 shows the descriptive statistics for each sub-divided population as well as their respective elevation vs. bulk density (t/m^3) scatter plot.

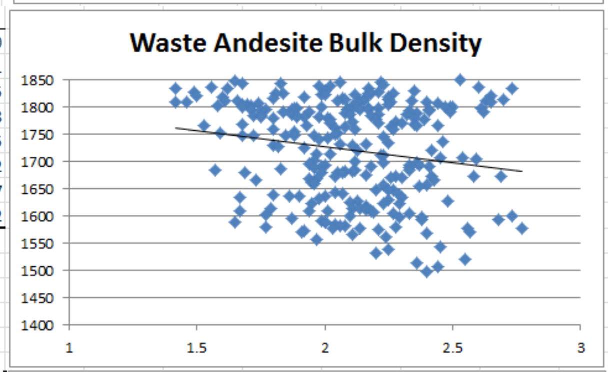
centerraGOLD

FIGURE 14-3 BULK DENSITY DESCRIPTION STATISTICS

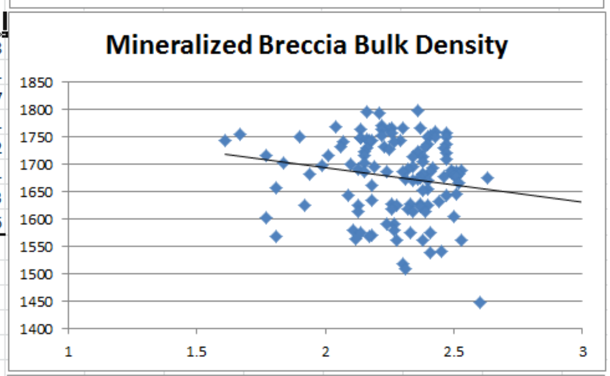
Mineralized Andesite Bulk Density	
Mean	2.18
Median	2.22
Mode	2.22
Standard Deviation	0.26
Range	1.45
Minimum	1.42
Maximum	2.87
Count	164



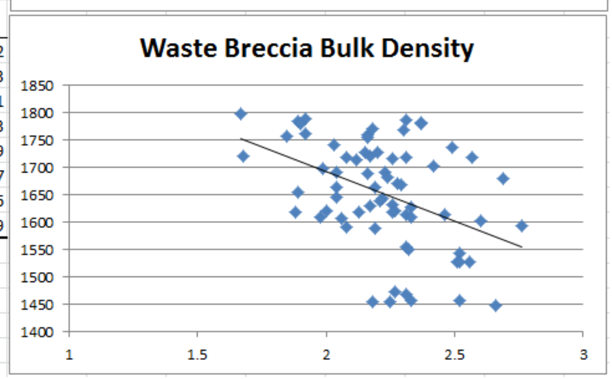
Waste Andesite Bulk Density	
Mean	2.10
Median	2.11
Mode	2.25
Standard Deviation	0.28
Range	1.35
Minimum	1.42
Maximum	2.77
Count	262



Mineralized Breccia Bulk Density	
Mean	2.28
Median	2.31
Mode	2.47
Standard Deviation	0.21
Range	1.52
Minimum	1.61
Maximum	3.13
Count	126



Waste Breccia Bulk Density	
Mean	2.22
Median	2.23
Mode	2.31
Standard Deviation	0.23
Range	1.09
Minimum	1.67
Maximum	2.76
Count	69



Since the waste andesite contains numerous samples between 1800 EL and 1850 EL, the population was also reviewed after removing these samples to validate the statistical basis for sub-dividing the andesite bulk density samples into two separate groups. Even after removing the samples between 1800 EL and 1850 EL from the waste andesite, the population still showed a marked difference from the mineralized andesite.

Density values were interpolated into the block model using inverse distance squared (ID^2). The interpolation used three passes, with progressively more relaxed search requirements. A flat-lying, pancake shaped ellipse was used during interpolation to better honour the changes in density with depth. The first pass honours the sub-divisions discussed above while the second pass only honours the separation of the mineralized and waste groups but allows for interaction between the rock types (soft boundary). This process allows blocks not populated in the first pass because of an insufficient number of samples to have access to more local samples. Centerra observed that the immediate area of the resource estimate was populated using the first two passes, so the third pass was used to populate the remaining blocks with the ID^2 value from all samples contained within a large flat-lying ellipse. Based on the search ellipse orientation and dimensions, some areas of the block model extents were not populated during the interpolation passes. The unpopulated blocks above 1890 EL were assigned a density of 2.03 t/m^3 while the blocks below elevation 1420 EL were assigned a density of 2.44 t/m^3. Table 14-3 contains the search ellipse and interpolation parameters used to populate the block model with bulk density values.

TABLE 14-3 BULK DENSITY INTERPOLATION PARAMETERS AND ELLIPSE ORIENTATIONS

	Density Attribute
Estimation Pass 1: (Hard RT Boundary)	
Samples	
Min. samples used	4
Max. samples used	12
Max. samples per hole	-
Distances	
Range Major (m)	150
Range Semi-Major (m)	150

	Density Attribute
Range Minor (m)	50
Ellipsoid Orientation (GEMS ADA)	
Rotation about ‘Z’ (degrees)	240
Rotation about ‘X’ (degrees)	0
Rotation about ‘Z’ (degrees)	-
Estimation Pass 2: (Soft RT Boundary)	
Samples	
Min. samples used	4
Max. samples used	12
Max. samples per hole	-
Distances	
Range Major (m)	150
Range Semi-Major (m)	150
Range Minor (m)	50
Ellipsoid Orientation (GEMS ADA)	
Rotation about ‘Z’ (degrees)	240
Rotation about ‘X’ (degrees)	0
Rotation about ‘Z’ (degrees)	-
Estimation Pass 3: (Complete Soft Boundary)	
Samples	
Min. samples used	4
Max. samples used	75
Max. samples per hole	-
Distances	
Range Major (m)	1,500
Range Semi-Major (m)	1,500
Range Minor (m)	50
Ellipsoid Orientation (GEMS ADA)	
Rotation about ‘Z’ (degrees)	240
Rotation about ‘X’ (degrees)	0
Rotation about ‘Z’ (degrees)	-

Centerra intends to collect additional density measurements to improve the density estimates at elevations above 1,850 m.

DATABASE

GENERAL DESCRIPTION

The Mineral Resource estimate for Öksüt is based primarily on information from surface diamond drill holes and supplemented by surface mapping. The Öksüt database contains a total of 330 collar records totaling 82,006 m of drilling.

A subset of 292 holes was used to update the Öksüt Mineral Resources. This subset of holes covers all exploration drilling on the Öksüt licence areas and includes numerous holes drilled beyond the mineralized extents of Keltepe and Güneytepe. Only 41 holes of the 292 hole subset are RC holes and only four of the RC holes are located within the extents of the mineralization and used for grade estimation.

The 38 holes not included in the estimate are metallurgical holes, water well holes, or geotechnical holes that were not assayed. Most of the holes at Öksüt are drilled at an inclined orientation between -45˚ and -90˚ (vertical). Drilling at Keltepe covers an approximate area of 1,250 m (north-northwesterly/south-southeasterly) by 850 m (east-northeasterly/west-southwesterly), while drilling at Güneytepe covers an approximate area of 450 m (northwesterly/southeasterly) by 550 m (northeasterly/southwesterly). Hole lengths vary widely, but they are typically in the range of 125 m to 425 m. Drill hole spacing on both deposits is fairly consistent, generally measuring 50 m across strike by 50 m along strike.

DRILL HOLE DATABASE VALIDATION

The following is a list of the data validation checks performed on the drill hole database by Centerra:

- Checked for duplicate drill hole collar locations and hole numbers.
- Checked collar locations for zero/extreme values.
- Checked assays for missing intervals, long intervals, extreme high values, blank/zero values, reasonable minimum/maximum values, etc.

- Ran validity checks for out-of-range values, missing intervals, overlapping intervals, out-of-sequence intervals, etc.
- Carried out visual inspection of drill holes for unusual azimuths, dips, and deviations.
- Digital comparison of gold values from the assay database against gold values from certificates.

The Öksüt exploration databases are actively maintained under the supervision of OMAS site personnel. The data provided for the resource estimate was exported directly from these databases. The QP is of the opinion that the database is acceptable for the purposes of resource estimation. The collar, survey, and assay data was imported into GEMS software for the purposes of resource estimation and no material errors were found during the import process.

ASSAY STATISTICS

Of the 292 drill holes within the Öksüt subset, 170 drill holes intersect the mineralization wireframes. The mineralization wireframes were used to flag drill hole intervals in the database that lie inside the wireframes. The assay statistics for each mineralized domain are presented in Table 14-4.

TABLE 14-4 ÖKSÜT ASSAY STATISTICS BY DEPOSIT

Deposit	Number	Min (g/t Au)	Max (g/t Au)	Average (g/t Au)	Std Dev (g/t Au)	Coefficient of Variation
Keltepe	13,379	0.00	27.80	1.06	1.61	1.52
Güneytepe	2,699	0.00	30.20	0.95	1.64	1.71

ASSAY CAPPING STATISTICS

The Öksüt assays for gold are positively skewed. In positively skewed statistical distributions, high outlier assay values can have a disproportionate effect on average values, which can result in a positive bias to the grade estimate. Centerra has capped high outlier assay values to minimize the potential for a positive grade bias.

A combination of histograms, decile analysis, probability plots, and the visual inspection of higher grade assays were used to determine the capping levels for Keltepe and Güneytepe. Both deposits were evaluated independently, and coincidentally, both were capped at 13 g/t Au (Figures 14-4 to 14-7). All assays were capped prior to compositing.

FIGURE 14-4 KELTEPE PROBABILITY PLOT

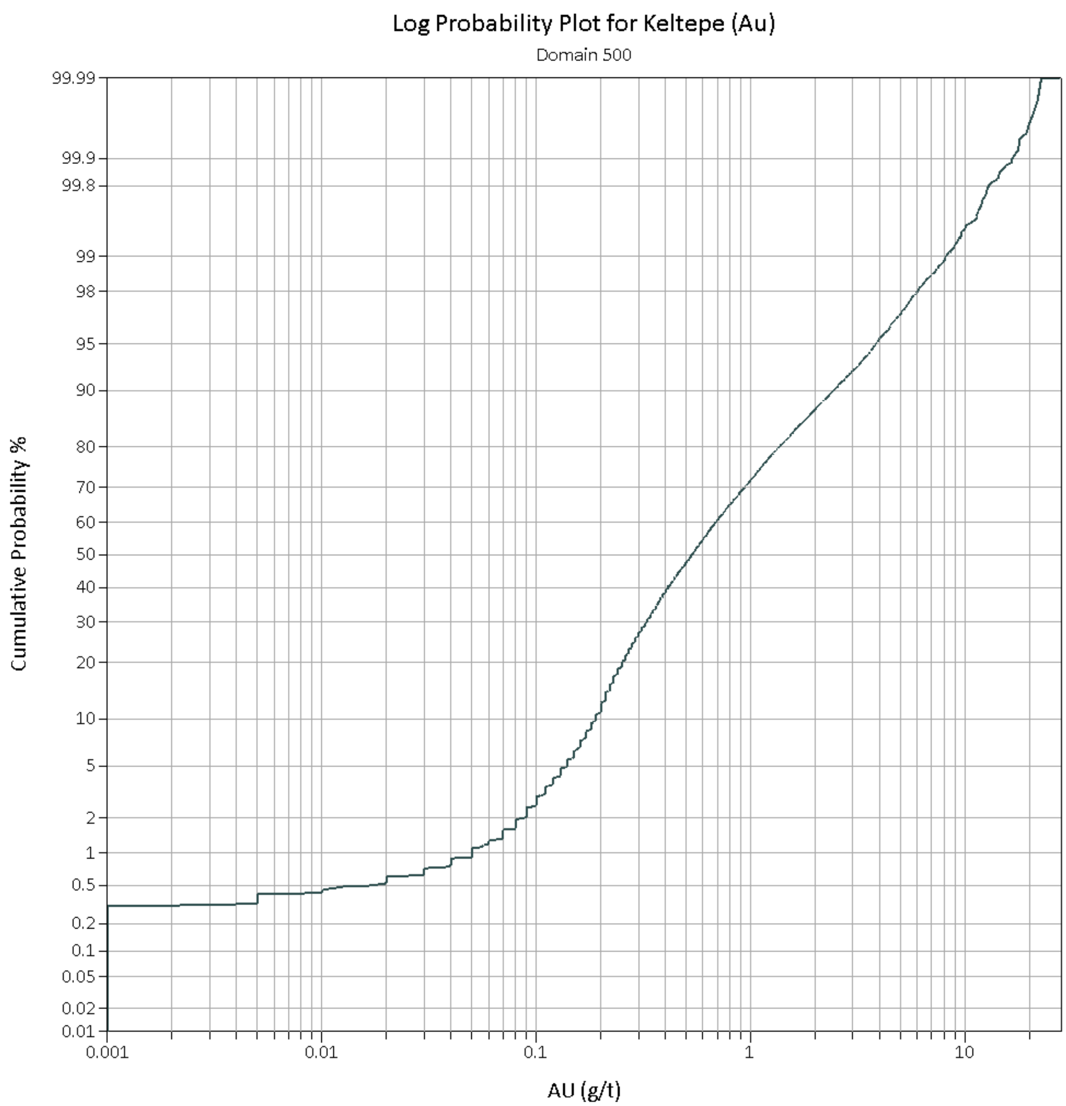


FIGURE 14-5 KELTEPE HISTOGRAM

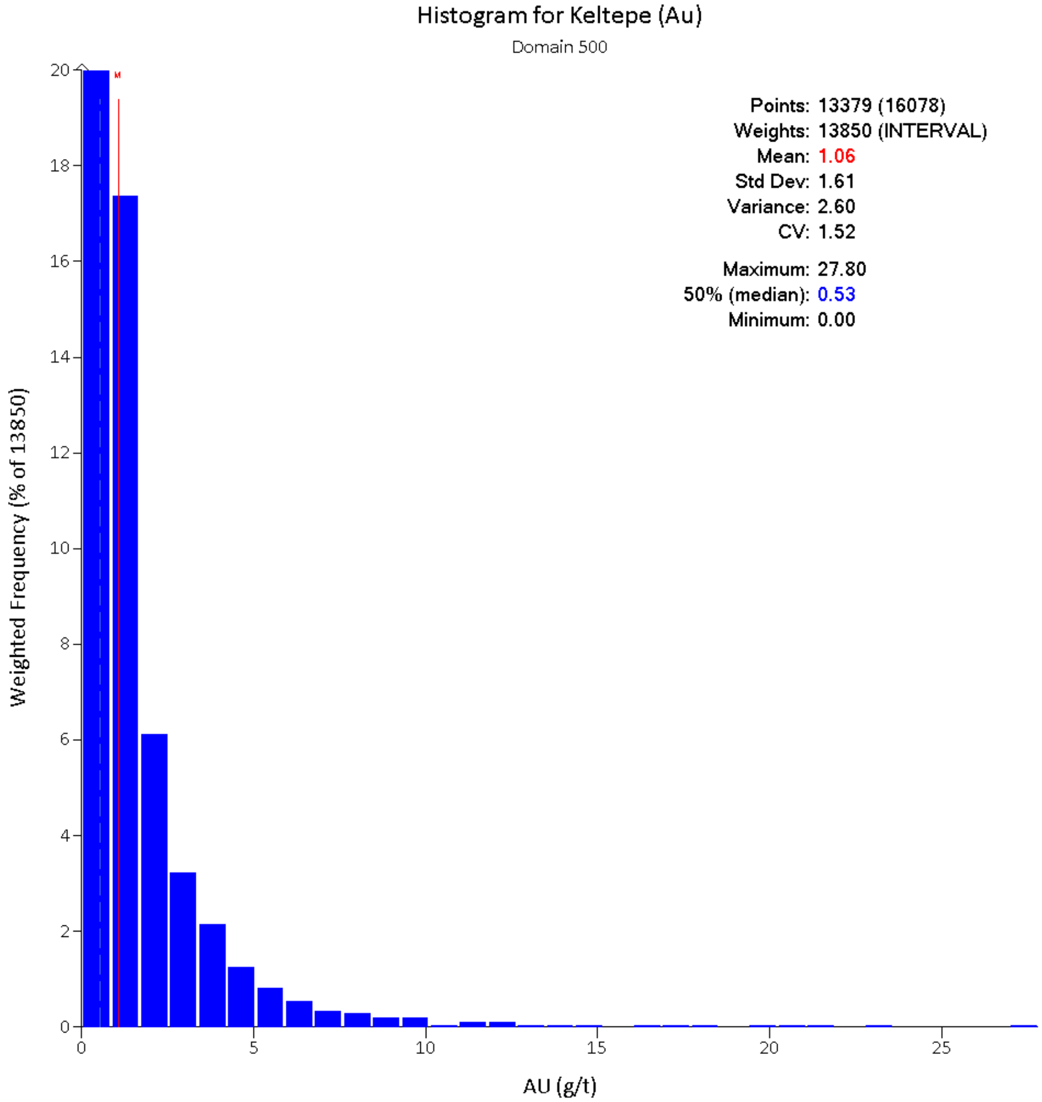


FIGURE 14-6 GÜNEYTEPE PROBABILITY PLOT

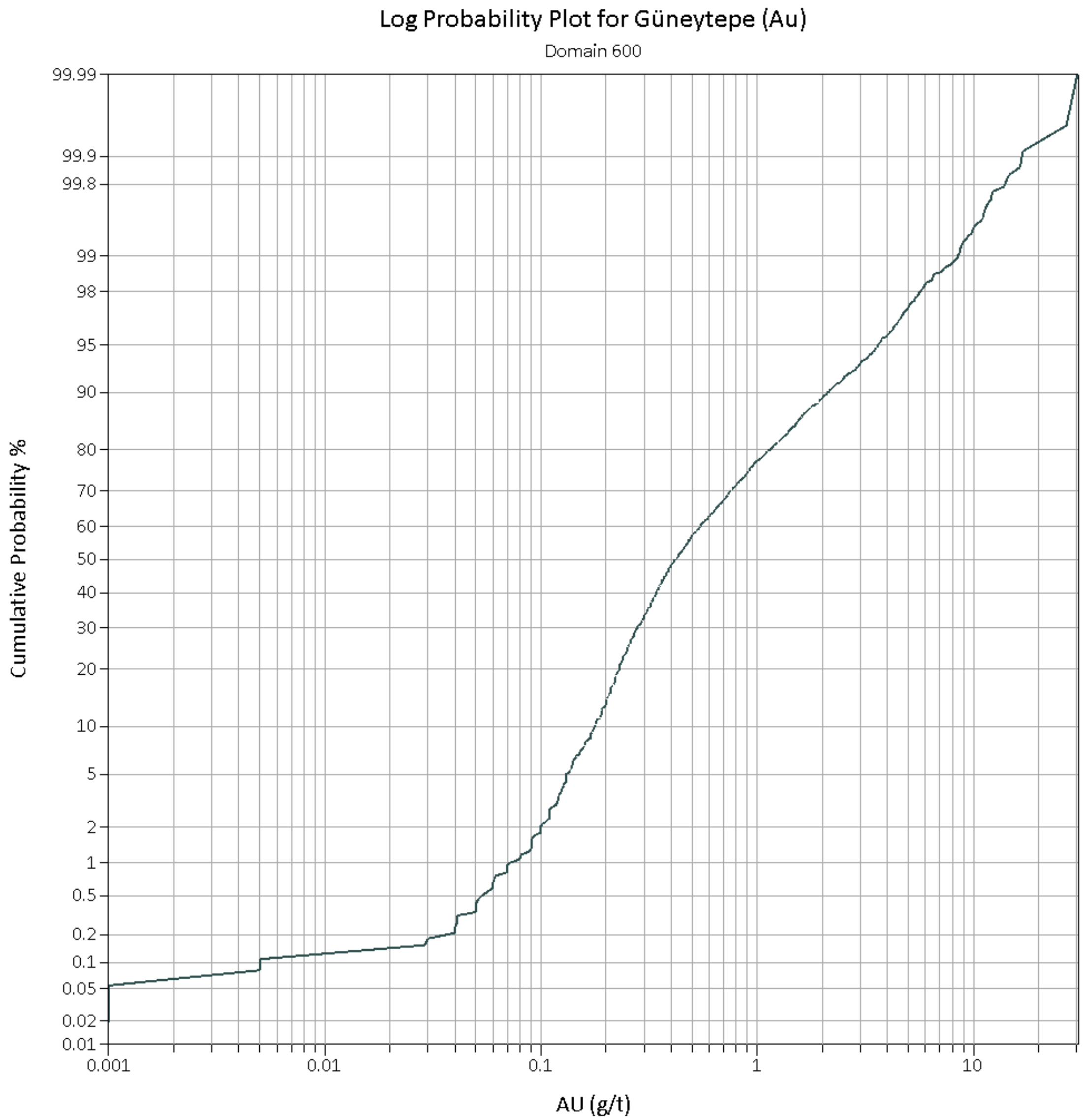


FIGURE 14-7 GÜNEYTEPE HISTOGRAM

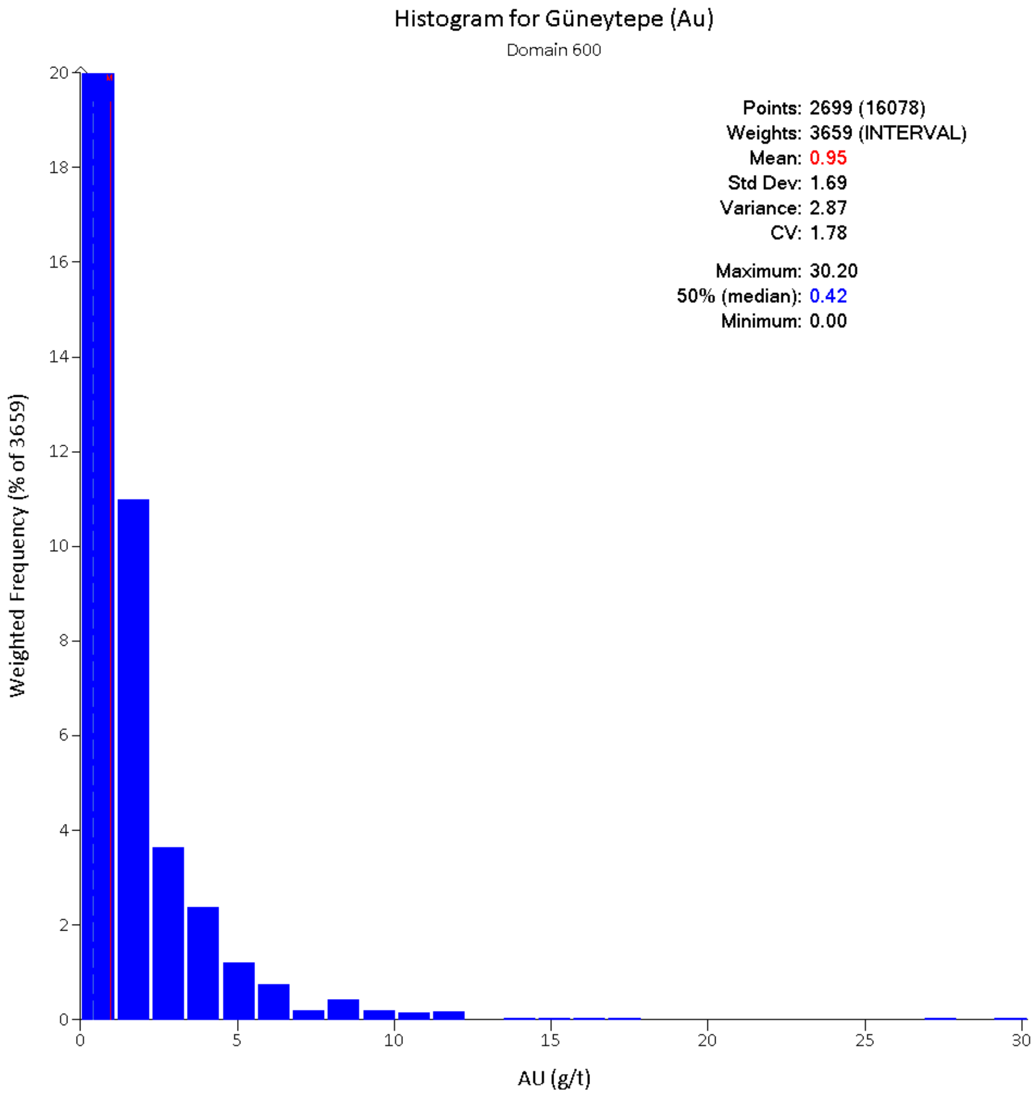


Table 14-5 summarizes statistics on the raw data after capping was applied.

TABLE 14-5 CAPPED ÖKSÜT ASSAY STATISTICS BY DEPOSIT

Deposit	Number	Min (g/t Au)	Max (g/t Au)	Average (g/t Au)	Std Dev (g/t Au)	Coefficient of Variation	Contained Metal Lost (%)	No. Assays Capped
Keltepe	13,379	0.00	13.00	1.05	1.53	1.45	0.8	27
Güneytepe	2,699	0.00	13.00	0.94	1.45	1.55	1.8	6

In general, it can be seen that the application of capping has not had a significant effect on the assay average or coefficient of variation.

COMPOSITE STATISTICS

Sample intervals varied from 15 cm to 6.0 m, with 76% of the samples measuring 1.0 m in length and 17% of the samples measuring from 1.0 m to 2.0 m in length. Only 1% of samples measured more than 2.0 m in length. Centerra elected to composite assays to 5.0 m lengths in order to mitigate the impact of splitting some higher grade samples with lengths greater than 2.0 m. Future estimates will consider the impact associated with using different composite lengths, such as 2.0 m or 2.5 m composites. After capping was applied to the raw assay data, assays were composited to 5.0 m within each mineralized wireframe domain. Samples were composited in downhole intervals of 5.0 m, starting at the wireframe pierce-point for each domain, continuing to the point at which the hole exited the domain. Any unsampled intervals that fell within either domain were assigned a value of zero.

Compositing to 5.0 m intervals occasionally resulted in the last composite in the mineralized domain being less than 5.0 m. For the composites less than 5.0 m, Centerra examined the grade distributions, including mean grades, of composites greater than and less than 2.5 m and concluded that removing these samples would noticeably increase the average composite grade. The zoned nature of the mineralization at Öksüt shows a gradual decrease in grade moving away from the higher grade feeders and massive silicification. Accordingly, the majority of holes that intersect the wireframes exit the wireframes in local areas where the grade of mineralization is below the average grade of the deposit. This, in turn, results in a population of partial composites with markedly lower grades than the full length composite population. Centerra has included all partial composites in the estimate to allow for more data at the margins of the mineralization.

Table 14-6 summarizes statistics of the composite grades.

TABLE 14-6 ÖKSÜT COMPOSITE STATISTICS BY DEPOSIT

Deposit	Number	Min (g/t Au)	Max (g/t Au)	Average (g/t Au)	Std Dev (g/t Au)	Coefficient of Variation
Keltepe	2,884	0.01	12.96	1.03	1.30	1.26
Güneytepe	779	0.08	12.56	0.91	1.27	1.40

BLOCK MODEL

DIMENSIONS AND CODING

A block model framework was created in GEOVIA GEMS software for the Öksüt Mineral Resource update using a block size of 10 m by 10 m by 5 m. The block model was rotated to align with the southeast-northwest trend of the mineralized domains. The block model extents for the "ÖksütMay2015" model are given in Table 14-7.

TABLE 14-7 ÖKSÜT BLOCK MODEL EXTENTS

Number of Blocks	
Columns:	160
Rows:	185
Levels:	174
Origin and rotation	
Min X:	718,700
Min Y:	4,239,183.626
Max Z:	2,070
Rotation:	13º
Block Size	
Column size:	10 m
Row size:	10 m
Level size:	5 m

Before grade estimation, all blocks were assigned mineralization codes. Each block was assigned based on a minimum 50% (majority rules) of each block being located within each mineralized wireframe domain. This method of block coding results in a “whole block model” that does not maintain a percent attribute. The volumes of the mineralized domains in the block model were compared to the volumes from the mineralization wireframes and were within ±0.5%. The relevant attributes used in the block model are given in Table 14-8 with a list of the codes used for each attribute given in Table 14-9. Keltepe and Güneytepe each have two rock type codes.

TABLE 14-8 CONSOLIDATED BLOCK MODEL ATTRIBUTES

Attribute Name	Description
Rock Type	Coded Mineralized Domains (majority rules)
Density	Interpolated Density
AU	Capped OK Gold Attribute Populated During Estimation
Class	Block Classification
Zone	Model coded to separate Güneytepe from Keltepe
Oxidation	Model coded to separate oxide from sulphide material

TABLE 14-9 BLOCK CODING FOR ATTRIBUTES

Attribute Name		Description	Rock Type (Domain Code)
Rock Type	Keltepe	Sub-divided for interpolation	500, 501
	Güneytepe	Sub-divided for interpolation	600, 601
Zone	Keltepe		500
	Güneytepe		600
Oxidation	Keltepe & Güneytepe	Oxide	100
		Sulphide	300
Class	Keltepe & Güneytepe	Measured	1
		Indicated	2
		Inferred	3
		Unclassified	4

VARIOGRAPHY AND TREND ANALYSIS

Variograms for gold were developed in GEOVIA GEMS software using the 5 m composites, flagged by their respective sub-vertical and sub-horizontal domains. Grade shells were generated in Leapfrog at a variety of cut-off grades to evaluate grade trends.

Linear downhole variograms were created to establish the nugget component of the variogram model for each of the four domains (500, 501, 600, and 601). The modelled variograms showed nugget values ranging between zero and 10% of the total sill and ranges between 10 m and 20 m. Based on experience, Centerra used a nugget component of 20% of the total sill for the variogram model used for estimation.

At Keltepe, the sub-horizontal (Domain: 500) mineralization appears to be spatially more continuous than the sub-vertical (Domain: 501), however, visual inspection indicates that parts of Domain 500 may be more related to the sub-vertical component of the mineralization. Hence, for Keltepe, the composites used for variography were further filtered to reflect this observation.

Higher grade composites (above 0.75 g/t Au) were re-flagged with codes 5000 and 5010 to reflect the behaviour of the horizontal and vertical components, respectively, and this data was subjected to variographic analysis. Horizontal trend variograms for high grade Domain 5000 show maximum continuity between 310º and 330º and ranges between 100 m and 160 m (Figure 14-8). The orientation of 320º showed a range of 115 m as well as good continuity across strike (050º) with a range of 100 m (Figure 14-9).

Generally, the best variograms for the sub-vertical high grade trend (Domain 5010) are aligned with the plunge direction of the mineralization. The orientation of 160º dipping -40º showed a range of 80 m (Figure 14-10).

FIGURE 14-8 MAJOR VARIOGRAM (AU) KELTEPE (5000)

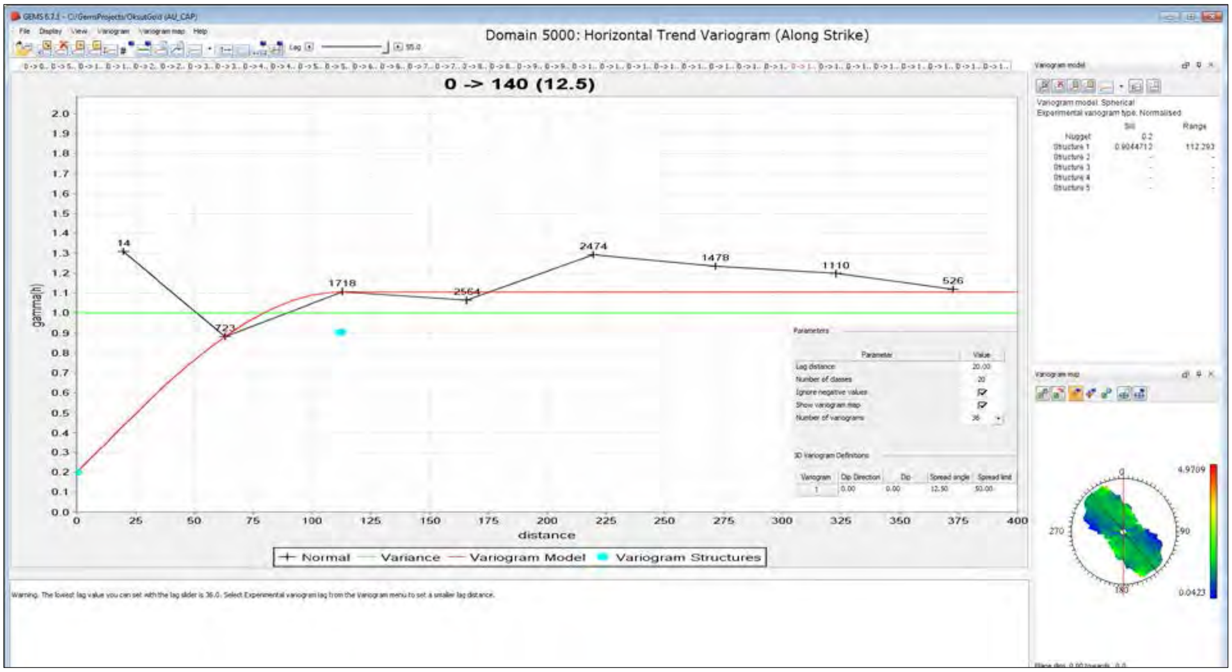


FIGURE 14-9 SEMI-MAJOR VARIOGRAM (AU) KELTEPE (5000)

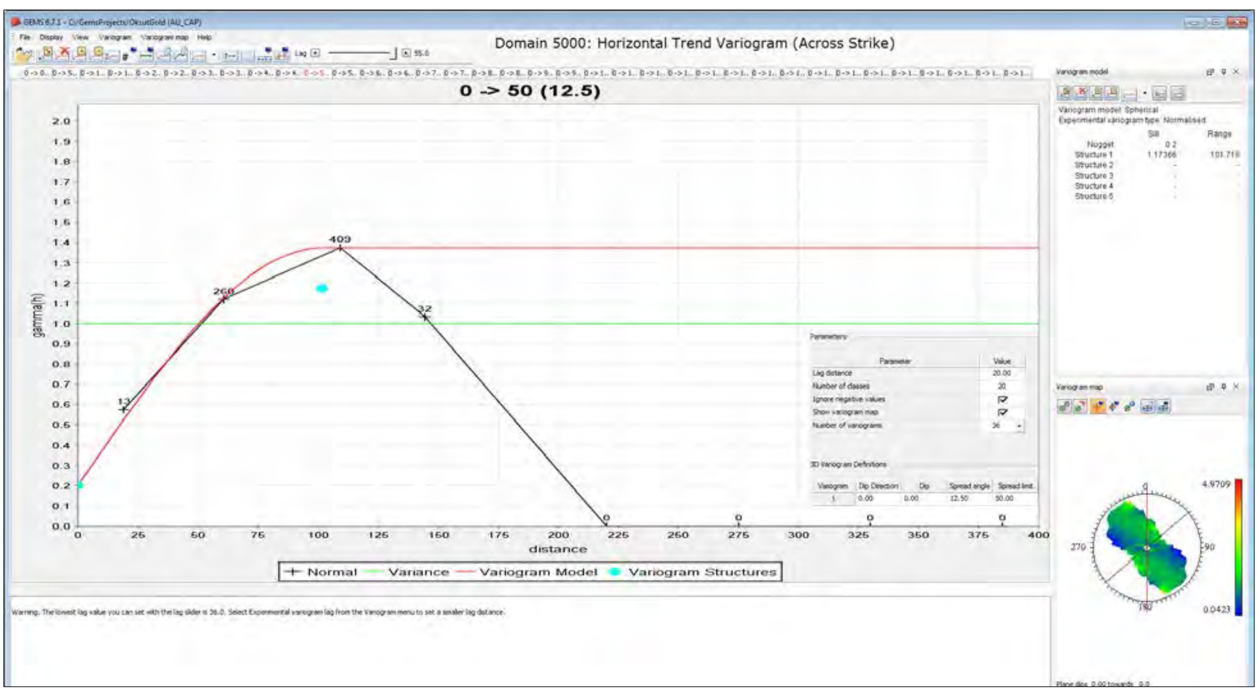
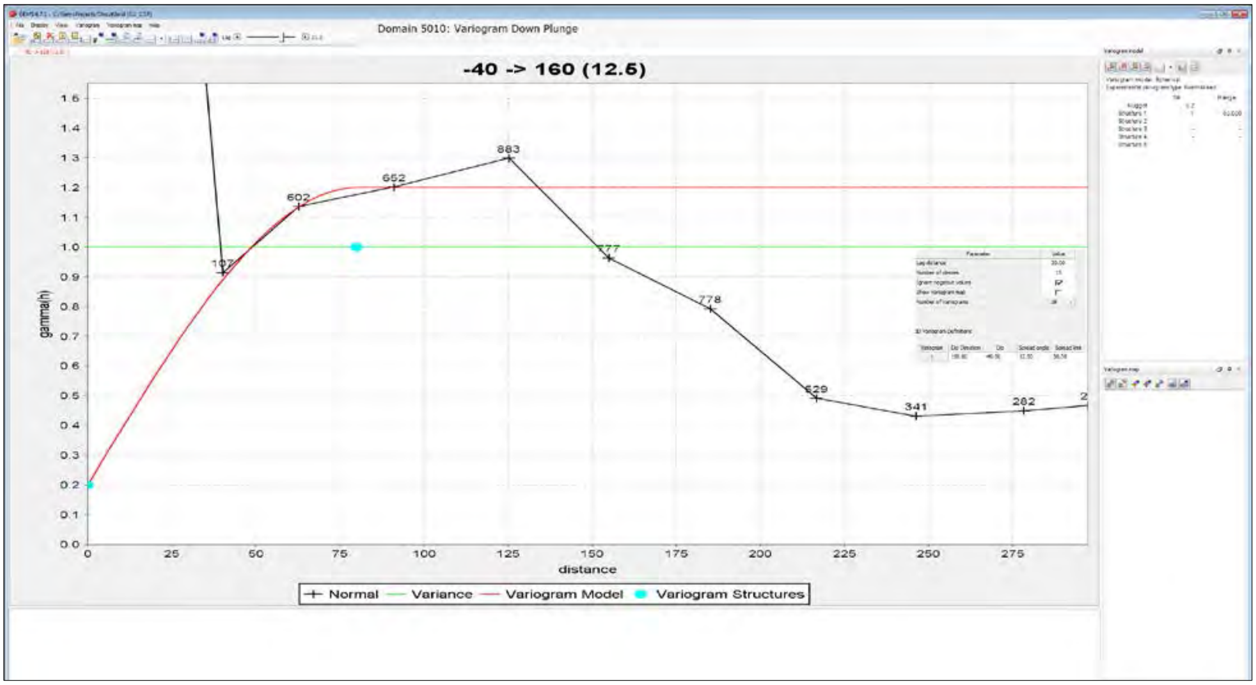


FIGURE 14-10 MAJOR VARIOGRAM (AU) KELTEPE (5010)



Güneytepe variography used sub-horizontal (Domain: 600) and sub-vertical (Domain: 601) coded composites. Horizontal trend variograms for Domain 600 show maximum continuity between 075º and 085º and ranges between 60 m and 85 m (Figure 14-11). The orientation of 080º showed a range of 85 m as well as good continuity across strike (170°) with a range of 70 m (Figure 14-12).

Generally, the best variograms for the sub-vertical trend (Domain 601) are aligned with the plunge direction of the mineralization. The orientation of 075º dipping -80º showed a range of 115 m (Figure 14-13).

FIGURE 14-11 MAJOR VARIOGRAM (AU) GÜNEYTEPE (600)

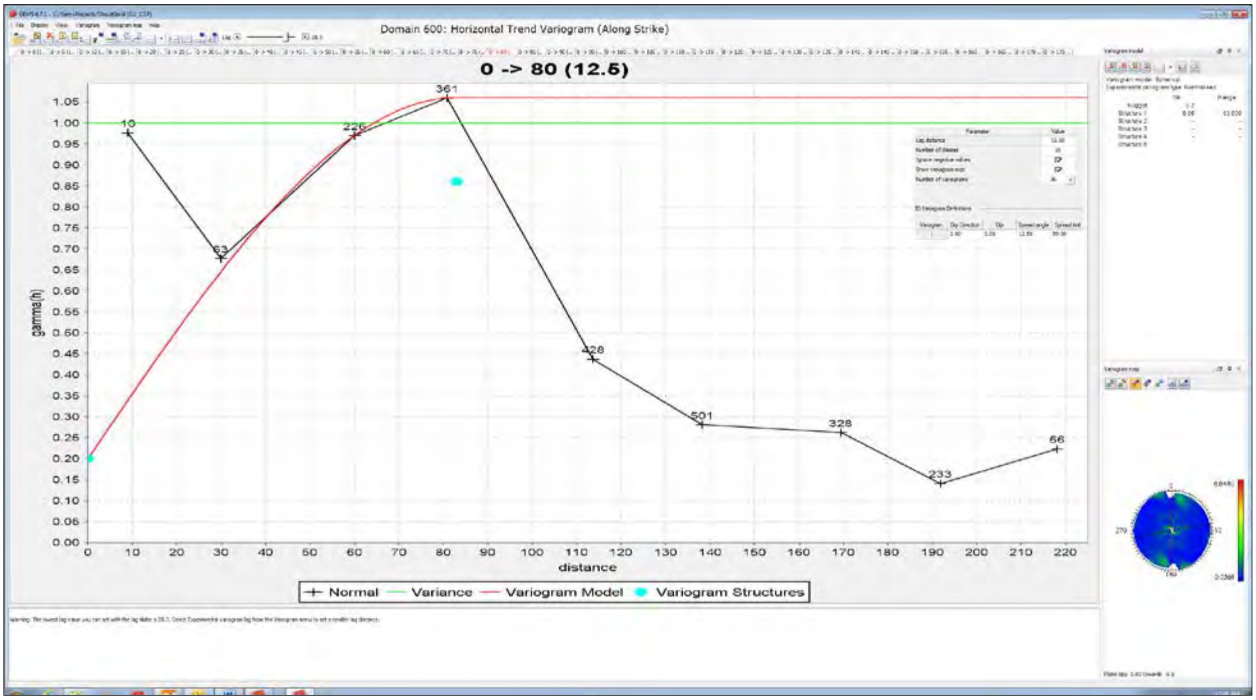


FIGURE 14-12 SEMI-MAJOR VARIOGRAM (AU) GÜNEYTEPE (600)

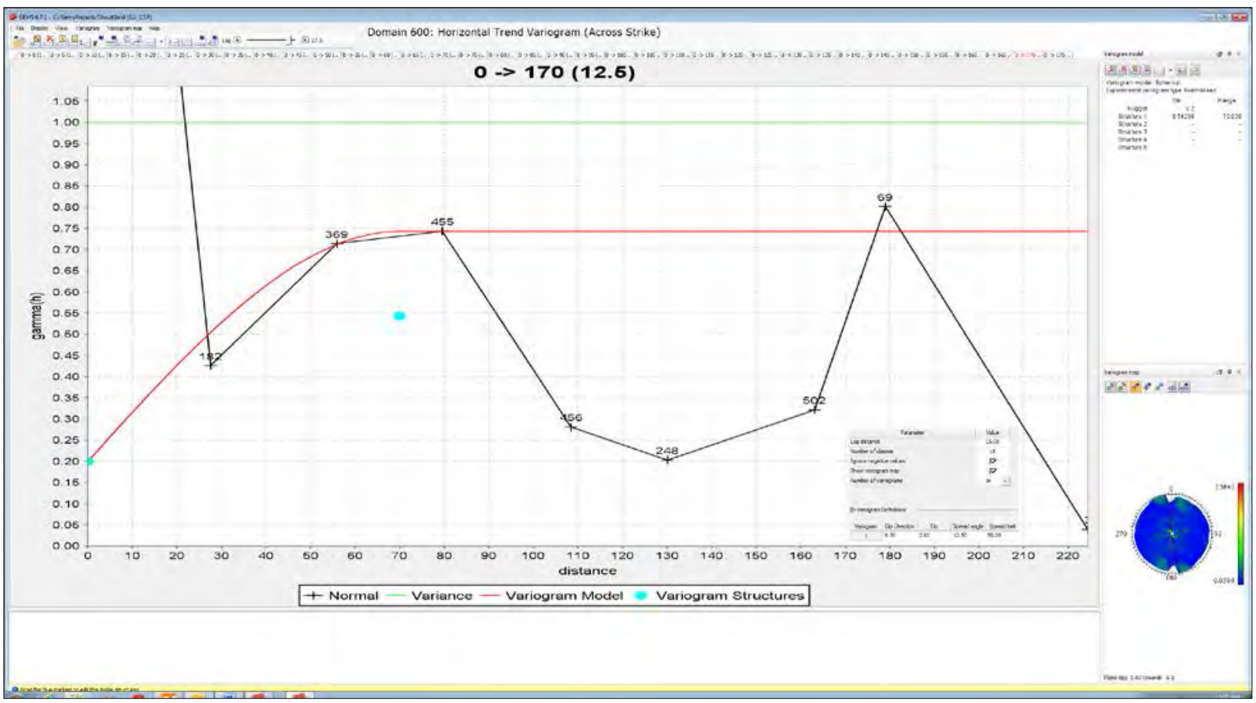
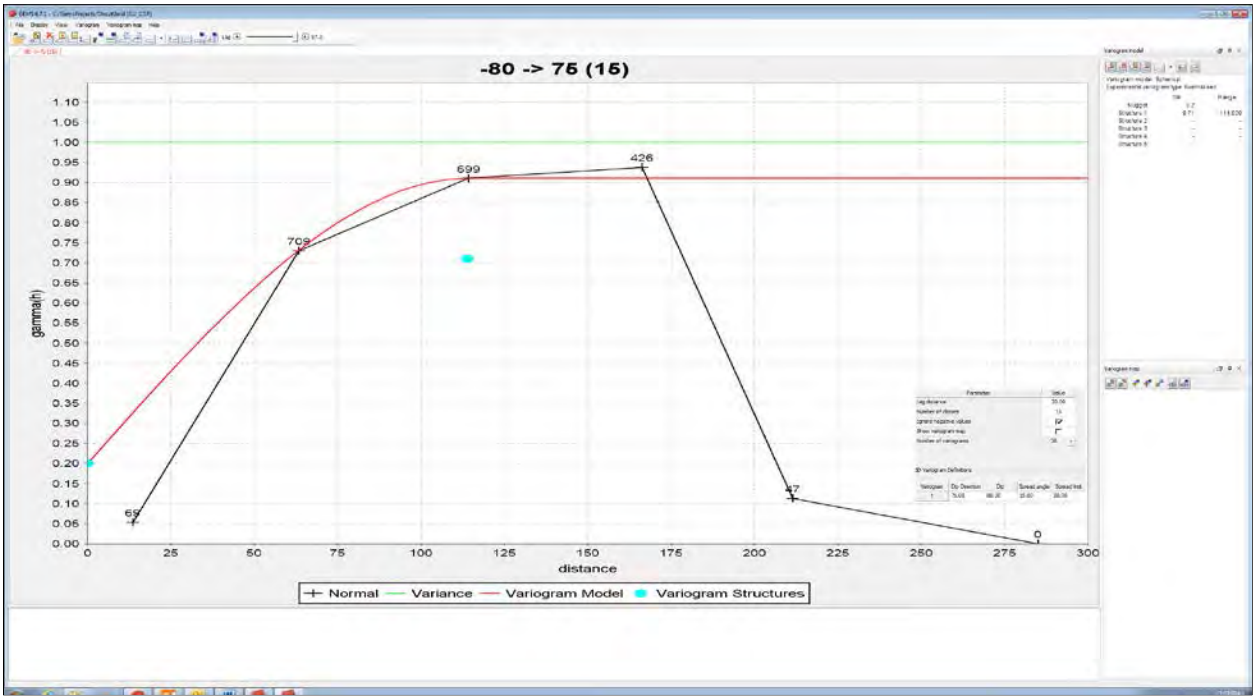


FIGURE 14-13 MAJOR VARIOGRAM (AU) GÜNEYTEPE (601)



Based on experience, Centerra adjusted the search orientations to best fit the shape of the observed mineralization trends. Centerra is of the opinion that the search ranges obtained from the variogram analysis provide sufficient support for the search ellipse dimensions used for the estimation of Mineral Resources.

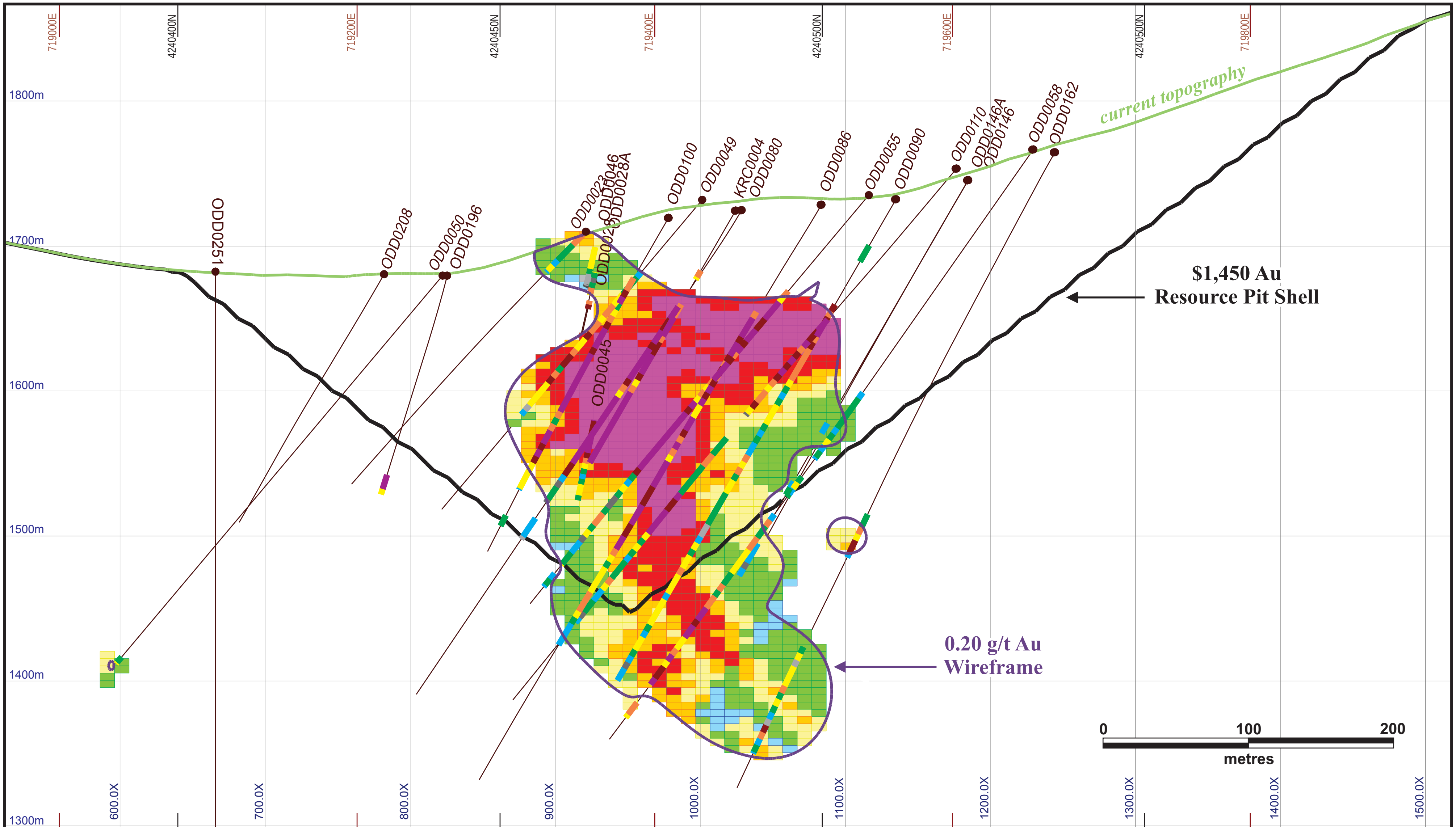
GRADE INTERPOLATION

Estimation of gold grades was carried out using ordinary kriging (OK) constrained within the mineralized domain wireframes. Two passes with increasingly longer search axes were carried out to ensure that all blocks within the mineralized domain wireframes were assigned a grade and to assist with resource classification. The estimation parameters and ellipse orientations used are summarized in Table 14-10. The subdivided areas of the high grade wireframes were treated as soft boundaries during the interpolation. In future resource updates, Centerra intends to perform a contact plot analysis to assess the need for soft, semi-soft, or hard estimation boundaries for high grade and low grade material.

The ellipsoid orientations for all mineralized domains were fixed to reflect the variography as well as the dominant azimuth and dip for each domain or sector. Figures 14-14 and 14-15 show the interpolated gold block grades relative to the composites for Keltepe and Figures 14-16 and 14-17, the interpolated gold block grades relative to the composites for Güneytepe.

TABLE 14-10 ÖKSÜT INTERPOLATION PARAMETERS AND ELLIPSE ORIENTATIONS

Estimation Pass	Keltepe Sub-horizontal (500)	Keltepe Sub-vertical (501)	Güneytepe Sub-horizontal (600)	Güneytepe Sub-vertical (601)
Estimation Pass 1:				
Samples				
Min. samples used	3	3	3	3
Max. samples used	12	12	12	12
Max. samples per hole	3	3	3	3
Distances				
Range Major (m)	70	70	70	70
Range Semi-Major (m)	50	35	50	35
Range Minor (m)	35	35	35	35
Ellipsoid Orientation (GEMS ADA)				
Principal Azimuth (degrees)	*327*	*327*	85	*85*
Principal Dip (degrees)	*0*	*50*	25	*-75*
Intermediate Azimuth (degrees)	-	*0*	-5	*5*
Estimation Pass 2:				
Samples				
Min. samples used	3	3	3	3
Max. samples used	12	12	12	12
Max. samples per hole	3	3	3	3
Distances				
Range Major (m)	140	140	140	140
Range Semi-Major (m)	100	70	100	70
Range Minor (m)	70	70	70	70
Ellipsoid Orientation (GEMS ADA)				
Principal Azimuth (degrees)	*327*	*327*	85	*85*
Principal Dip (degrees)	*0*	*50*	25	*-75*
Intermediate Azimuth (degrees)	-	*0*	-5	*5*



Gold Block Model Au g/t
> 2.50 g/t Au
1.50 to 2.50 g/t Au
1.00 to 1.50 g/t Au
0.60 to 1.00 g/t Au
0.30 to 0.60 g/t Au
0.20 to 0.30 g/t Au
0.15 to 0.20 g/t Au

Drill Hole Intercepts
> 2.50 g/t Au
1.50 to 2.50 g/t Au
1.00 to 1.50 g/t Au
0.60 to 1.00 g/t Au
0.30 to 0.60 g/t Au
0.20 to 0.30 g/t Au
0.15 to 0.20 g/t Au
0.01 to 0.15 g/t Au

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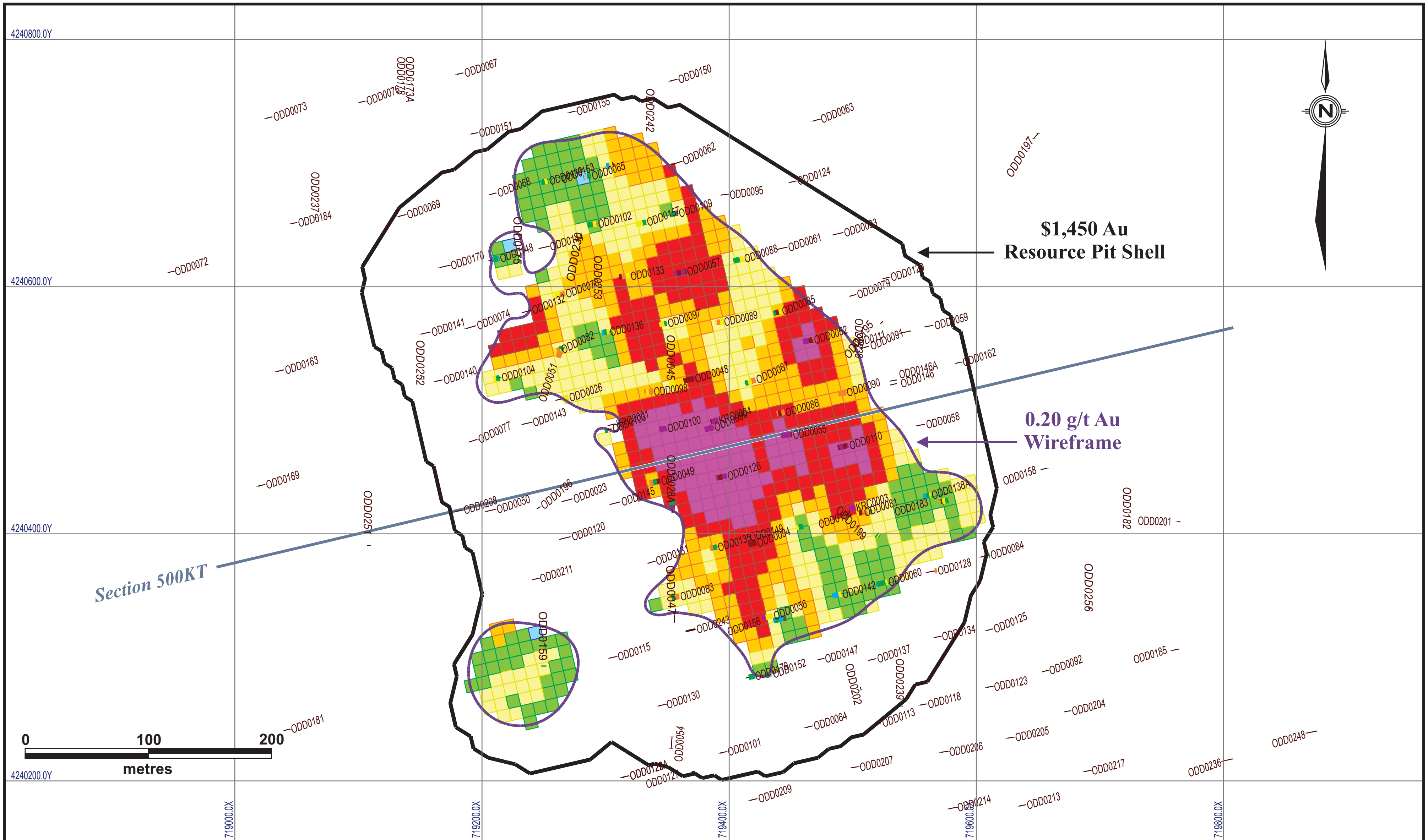
Keltepe Section 500KT
(looking northwest)
Gold Block Model

Date: Sept. 2015 | File: KT500_GRADE.cdr | Figure 14-14



Gold Block Model Au g/t
> 2.50 g/t Au
1.50 to 2.50 g/t Au
1.00 to 1.50 g/t Au
0.60 to 1.00 g/t Au
0.30 to 0.60 g/t Au
0.20 to 0.30 g/t Au
0.15 to 0.20 g/t Au

Drill Hole Intercepts
> 2.50 g/t Au
1.50 to 2.50 g/t Au
1.00 to 1.50 g/t Au
0.60 to 1.00 g/t Au
0.30 to 0.60 g/t Au
0.20 to 0.30 g/t Au
0.15 to 0.20 g/t Au
0.01 to 0.15 g/t Au

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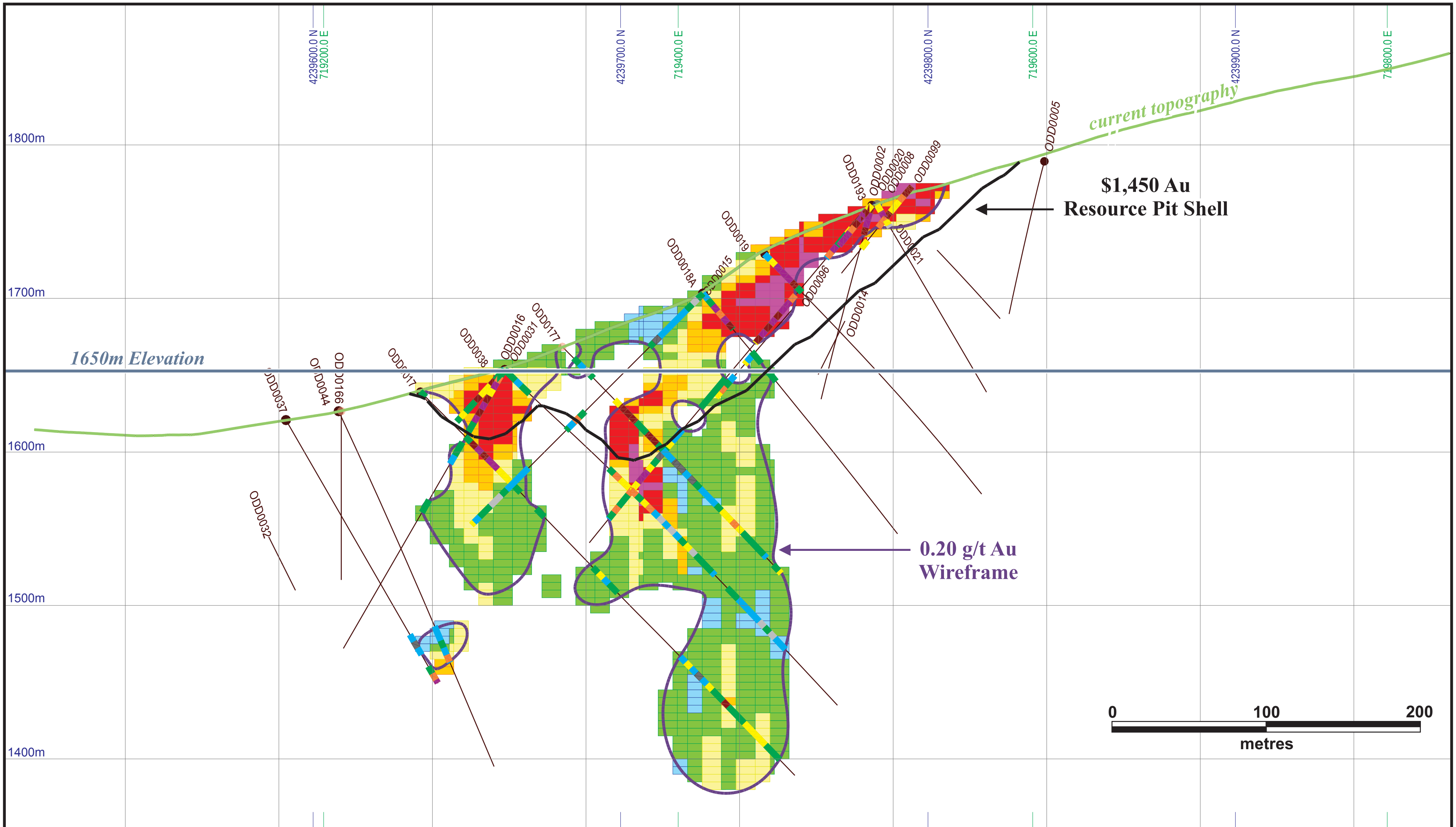
Keltepe
Surface Plan 1710m
Gold Block Model

Date: Sept. 2015	File: KT1710_GRADE.cdr	Figure 14-15



Gold Block Model Au g/t

- > 2.50 g/t Au
- 1.50 to 2.50 g/t Au
- 1.00 to 1.50 g/t Au
- 0.60 to 1.00 g/t Au
- 0.30 to 0.60 g/t Au
- 0.20 to 0.30 g/t Au
- 0.15 to 0.20 g/t Au

Drill Hole Intercepts

- > 2.50 g/t Au
- 1.50 to 2.50 g/t Au
- 1.00 to 1.50 g/t Au
- 0.60 to 1.00 g/t Au
- 0.30 to 0.60 g/t Au
- 0.20 to 0.30 g/t Au
- 0.15 to 0.20 g/t Au
- 0.01 to 0.15 g/t Au

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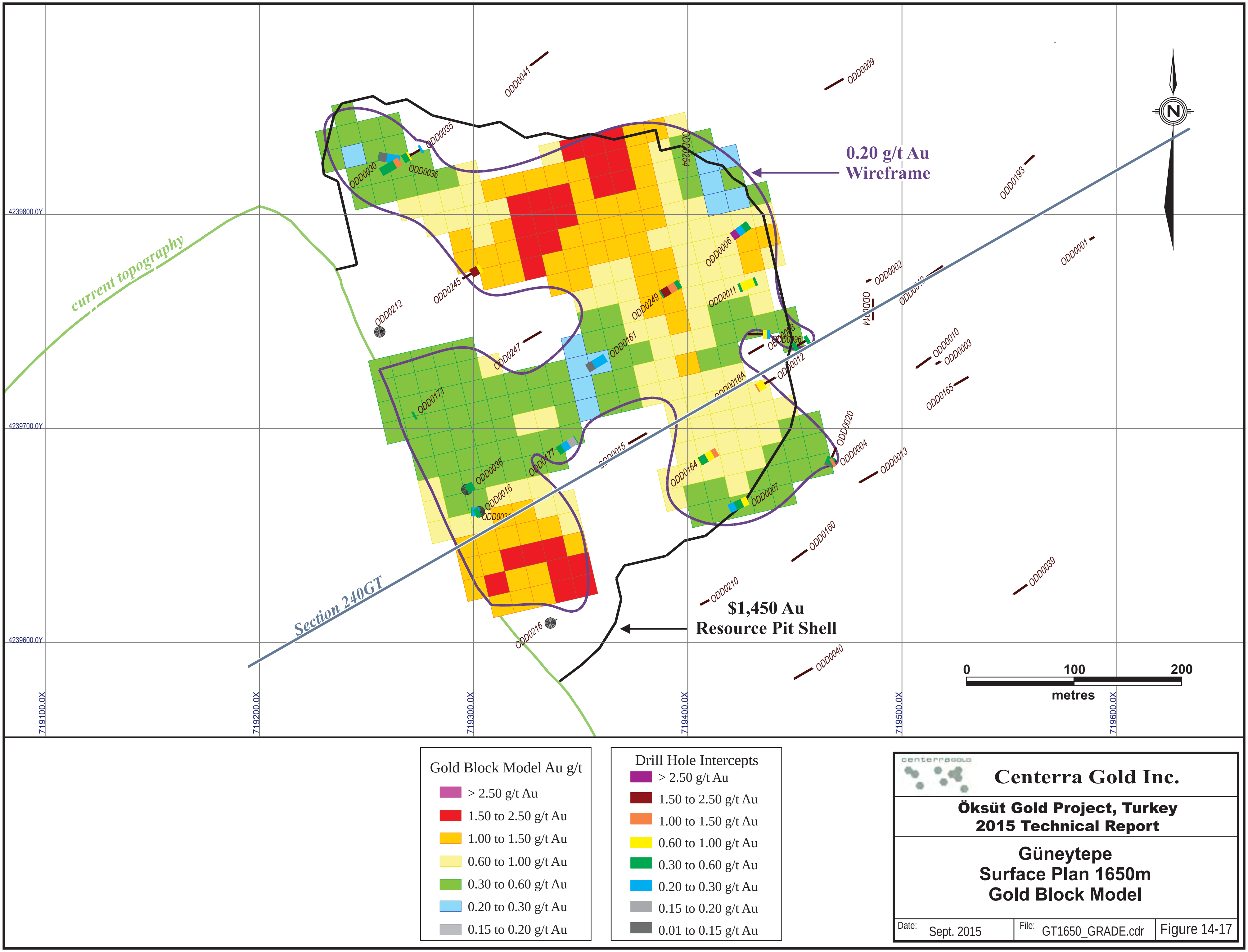
Güneytepe Section 240GT
(looking northwest)
Gold Block Model

Date: Sept. 2015 | File: GT240_GRADE.cdr | Figure 14-16



0.20 g/t Au
Wireframe
current topography
Section 240GT
$1,450 Au
Resource Pit Shell
0
100
200
metres
4239800.0Y
4239700.0Y
4239600.0Y
719100.0X
719200.0X
719300.0X
719400.0X
719500.0X
719600.0X
ODD0041
ODD0009
ODD0035
ODD0030
ODD0036
ODD0254
ODD0193
ODD0006
ODD0001
ODD0245
ODD0002
ODD0249
ODD0011
ODD0014
ODD0212
ODD0161
ODD0010
ODD0003
ODD0247
ODD0012
ODD0165
ODD0171
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ODD0015
ODD0004
ODD0177
ODD0038
ODD0164
ODD0013
ODD0016
ODD0007
ODD0160
ODD0039
ODD0210
ODD0216
ODD0040
Gold Block Model Au g/t
> 2.50 g/t Au
1.50 to 2.50 g/t Au
1.00 to 1.50 g/t Au
0.60 to 1.00 g/t Au
0.30 to 0.60 g/t Au
0.20 to 0.30 g/t Au
0.15 to 0.20 g/t Au
Drill Hole Intercepts
> 2.50 g/t Au
1.50 to 2.50 g/t Au
1.00 to 1.50 g/t Au
0.60 to 1.00 g/t Au
0.30 to 0.60 g/t Au
0.20 to 0.30 g/t Au
0.15 to 0.20 g/t Au
0.01 to 0.15 g/t Au
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Öksüt Gold Project, Turkey
2015 Technical Report
Güneytepe
Surface Plan 1650m
Gold Block Model
Date: Sept. 2015
File: GT1650_GRADE.cdr
Figure 14-17

CUT-OFF GRADE

Based on the parameters outlined in Table 14-11 as well as other considerations, Centerra has reported the Öksüt Mineral Resources at a cut-off grade of 0.20 g/t Au. Only those blocks contained within the preliminary pit shell are reported as a Mineral Resource.

PIT OPTIMIZATION

In order to comply with the CIM definitions of “reasonable prospects for eventual economic extraction”, a preliminary Lerchs-Grossmann pit shell was prepared using GEOVIA Whittle software and the assumed costs and parameters as shown in Table 14-11. Only those blocks contained within the preliminary pit shell are reported as Mineral Resources.

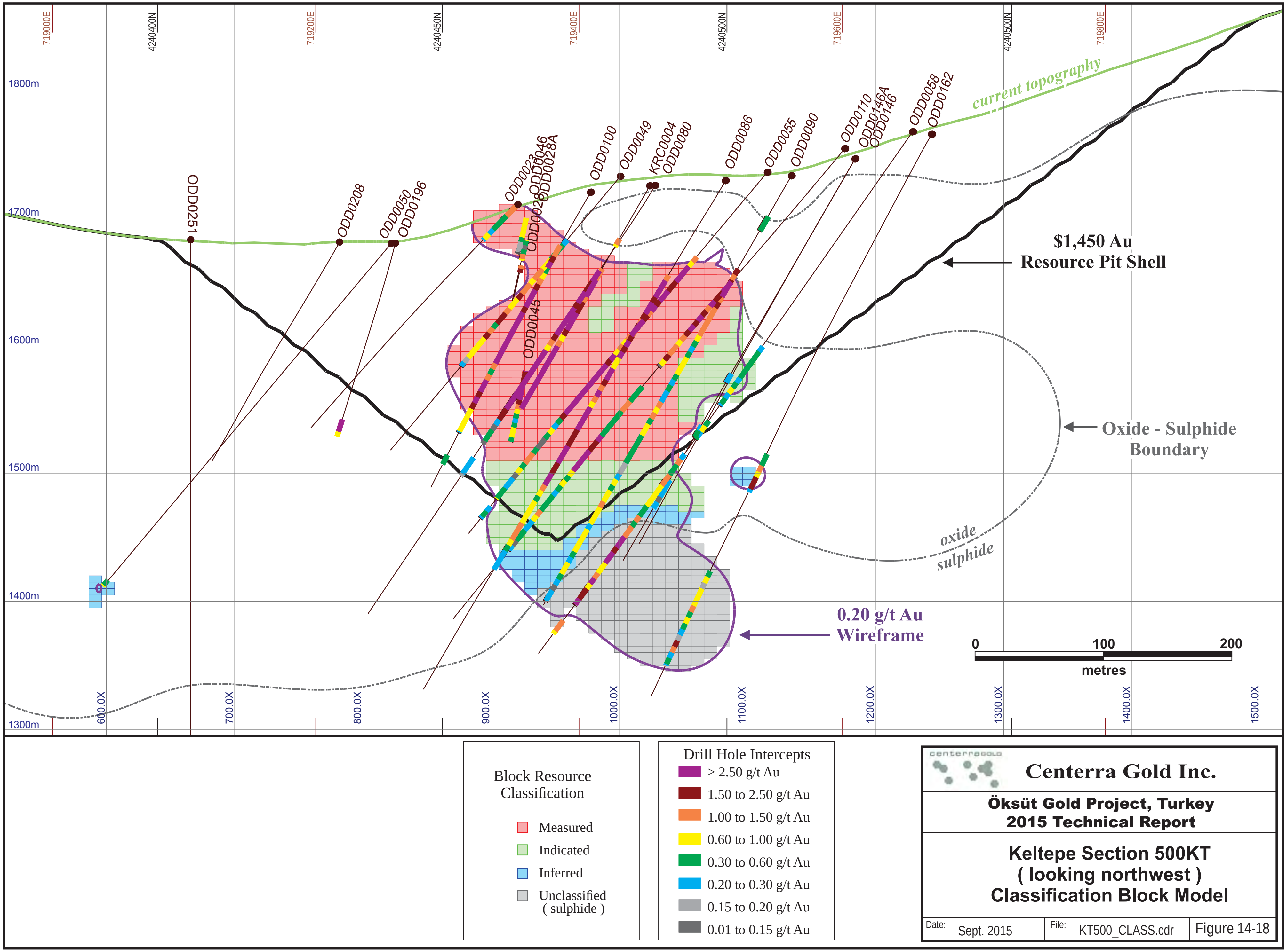
TABLE 14-11 ÖKSÜT PIT OPTIMIZATION PARAMETERS

Input	**Unit**	**Value**
Mining Cost	$/t mined	2.24
Processing Cost	$/t processed	5.25
G&A Cost	$/t processed	2.91
Gold Price	$/oz Au	1,450
Gold Recovery	%	77
Royalty	%	4.6
Refining Cost	$/oz Au	5.00
Mining Recovery	%	100
Mining Dilution	%	0
Pit Slopes	Degrees *(Variable by sector)*	Keltepe 36-40 Güneytepe 40

OPEN PIT CLASSIFICATION

Centerra developed resource classification criteria based on geological continuity, grade continuity, distance of a block to the nearest composite, and distance of a block from the oxide/sulphide surface. Shapes created based on these criteria were used to assign the respective Measured, Indicated, and Inferred classifications.

Figures 14-18 and 14-19 provide examples of vertical cross-sections showing the open pit classification block models.



719000E
4240400N
719200E
4240450N
719400E
4240500N
719600E
719800E
1800m
1700m
1600m
1500m
1400m
1300m
600.0X
700.0X
800.0X
900.0X
1000.0X
1100.0X
1200.0X
1300.0X
1400.0X
1500.0X
current topography
ODD0251
ODD0208
ODD0050
ODD0196
ODD0045
ODD0100
ODD0049
KRC0004
ODD0080
ODD0086
ODD0055
ODD0090
ODD0110
ODD0146A
ODD0146
ODD0058
ODD0162
$1,450 Au
Resource Pit Shell
Oxide - Sulphide
Boundary
oxide
sulphide
0.20 g/t Au
Wireframe
0
100
200
metres
Block Resource
Classification
Measured
Indicated
Inferred
Unclassified
(sulphide)
Drill Hole Intercepts
> 2.50 g/t Au
1.50 to 2.50 g/t Au
1.00 to 1.50 g/t Au
0.60 to 1.00 g/t Au
0.30 to 0.60 g/t Au
0.20 to 0.30 g/t Au
0.15 to 0.20 g/t Au
0.01 to 0.15 g/t Au
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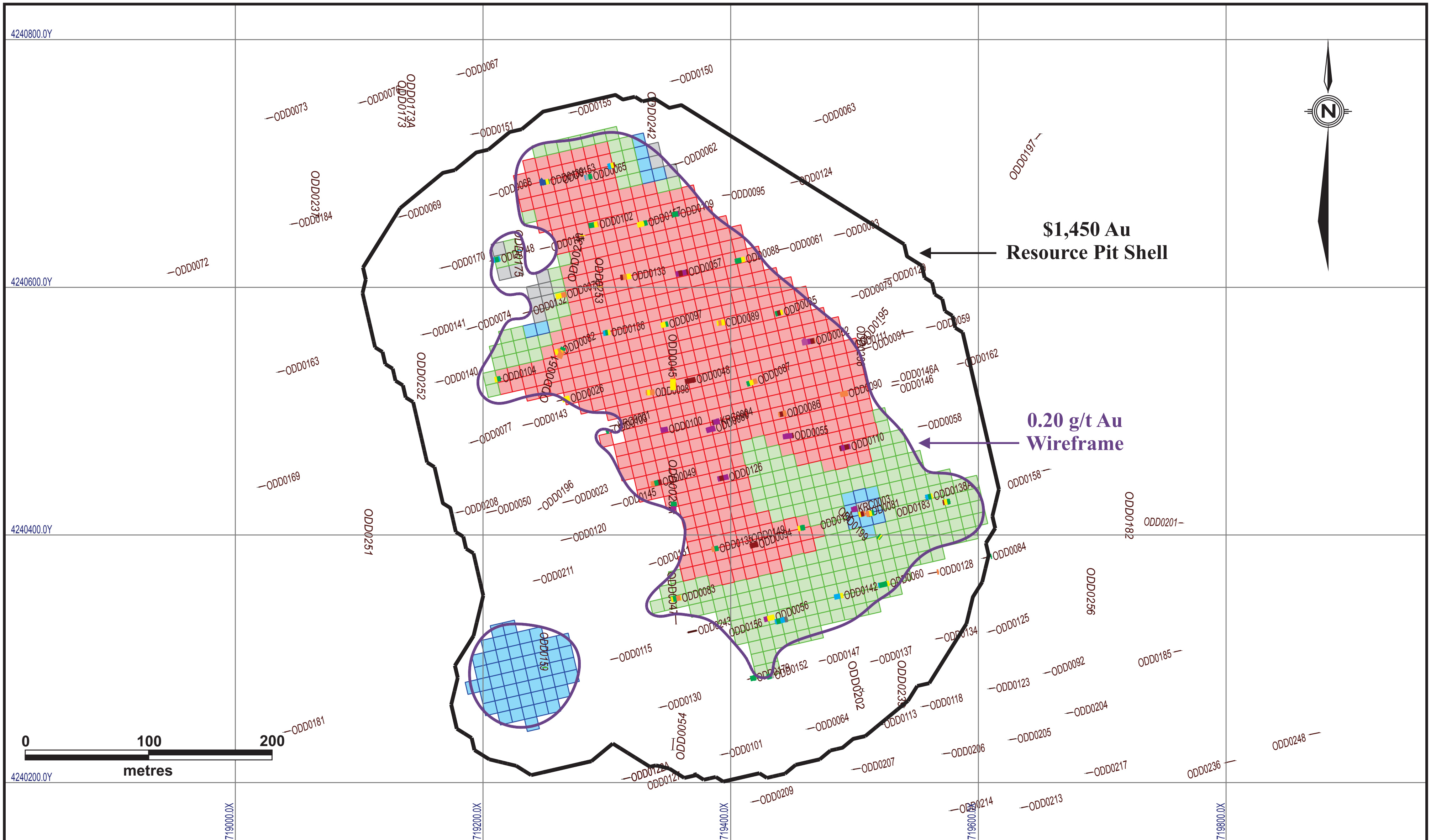
Öksüt Gold Project, Turkey
2015 Technical Report
Keltepe Section 500KT
(looking northwest)
Classification Block Model
Date: Sept. 2015
File: KT500_CLASS.cdr
Figure 14-18



Block Resource Classification

- Measured
- Indicated
- Inferred
- Unclassified (sulphide)

Drill Hole Intercepts

- > 2.50 g/t Au
- 1.50 to 2.50 g/t Au
- 1.00 to 1.50 g/t Au
- 0.60 to 1.00 g/t Au
- 0.30 to 0.60 g/t Au
- 0.20 to 0.30 g/t Au
- 0.15 to 0.20 g/t Au
- 0.01 to 0.15 g/t Au

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Öksüt Gold Project, Turkey
2015 Technical Report

Keltepe
Surface Plan 1710m
Classification Block Model

Date: Sept. 2015	File: KT1710_CLASS.cdr	Figure 14-19

BLOCK MODEL VALIDATION

The mineralization wireframes were checked for overlaps and blocks were checked to ensure consistent rock type coding between the wireframes, composites, and blocks. The mineralized domains were reviewed to confirm that extensions beyond last holes were reasonable and consistent.

Global statistics were tabulated for the assays, composites, and blocks for each domain at Öksüt and are summarized in Table 14-12. The assay, composite and block statistics compare fairly well for Keltepe, but there is a noticeable drop in grade between the composites and blocks at Güneytepe. This drop in grade is caused by the declustering of higher grade composites in the upper portion of Güneytepe when estimating using OK.

TABLE 14-12 ASSAY COMPOSITE AND BLOCK STATISTICS BY DOMAIN

	Deposit	Number	Min (g/t Au)	Max (g/t Au)	Average (g/t Au)	Std Dev (g/t Au)	Coefficient of Variation
Assays Raw	Keltepe	13,379	0.00	27.80	1.06	1.61	1.52
	Güneytepe	2,699	0.00	30.20	0.95	1.64	1.71
Assays Capped	Keltepe	13,379	0.00	13.00	1.05	1.53	1.45
	Güneytepe	2,699	0.00	13.00	0.94	1.45	1.55
Composites	Keltepe	2,884	0.01	12.96	1.03	1.30	1.26
	Güneytepe	779	0.08	12.56	0.91	1.27	1.40
Blocks	Keltepe	46,759	0.09	7.19	0.98	0.87	0.89
	Güneytepe	14,684	0.17	6.83	0.73	0.55	0.75

The block model grades were visually compared with composite grades on plan and section views. The model was also visually checked for grade banding, smearing of high grades, plumes of high grades, on sections and plans. This evaluation indicated an acceptable spatial correlation of block grades with composite grades.

Centerra also prepared a grade and tonnage curve to evaluate the impact of different cut-off grades on the reported resources (Figures 14-20 and 14-21).

FIGURE 14-20 KELTEPE GRADE AND TONNAGE CURVE

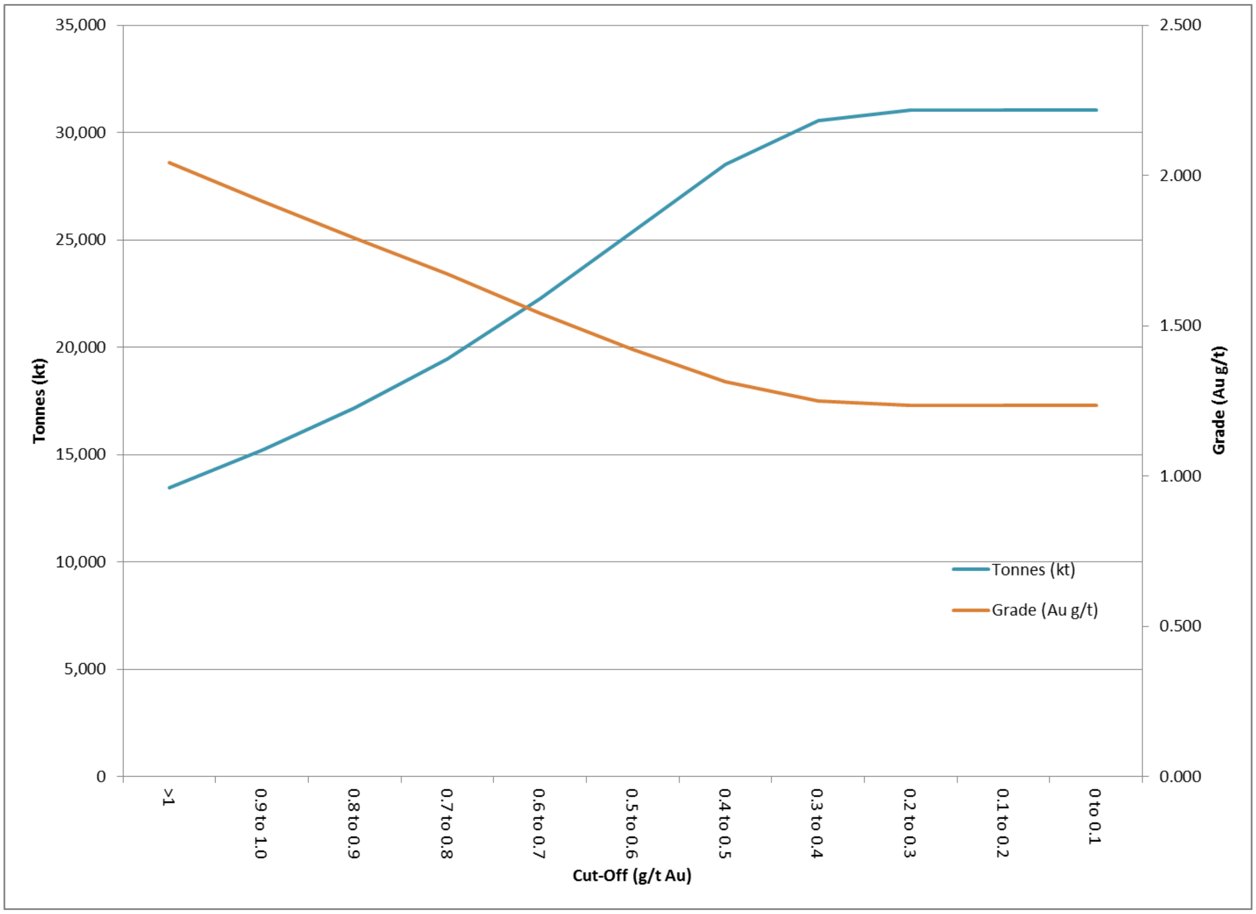
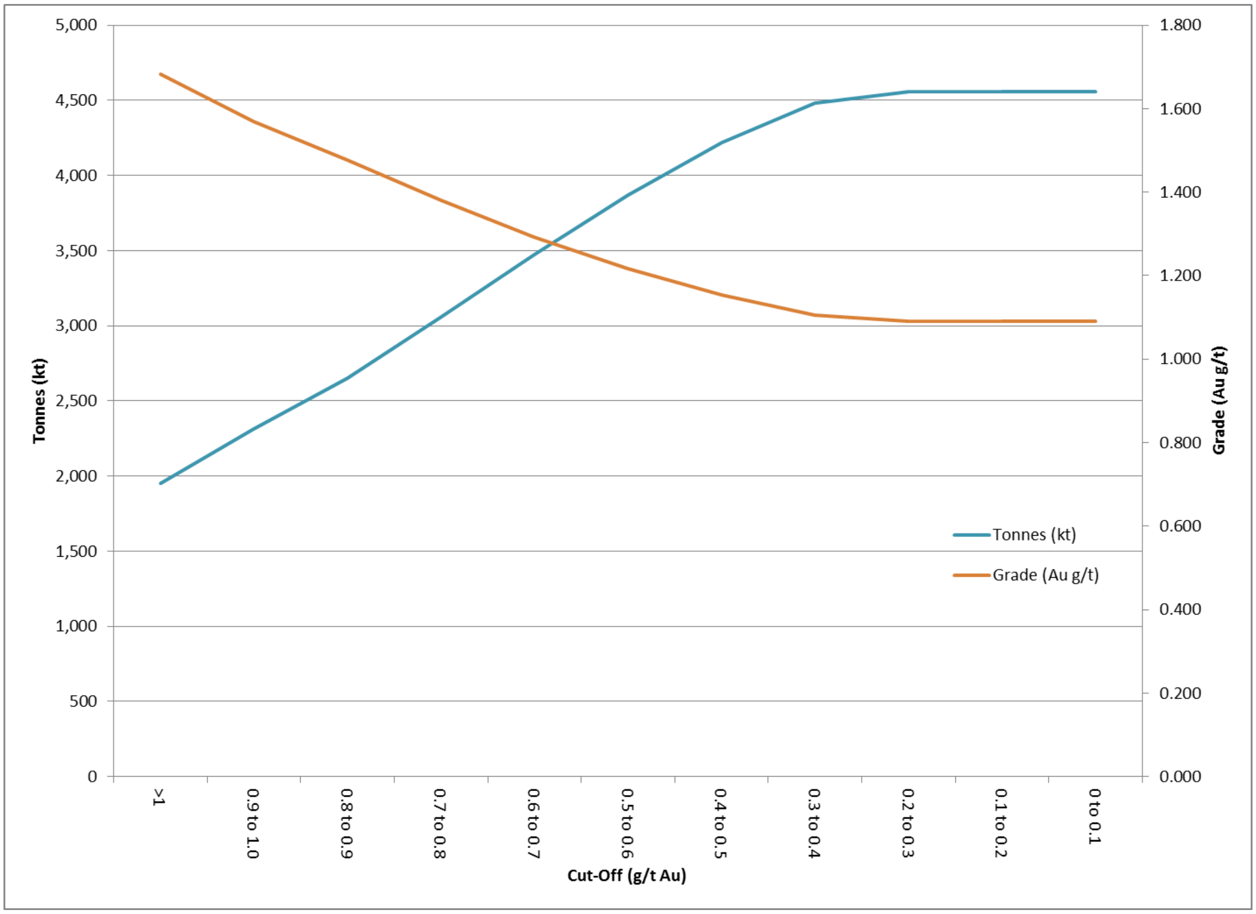
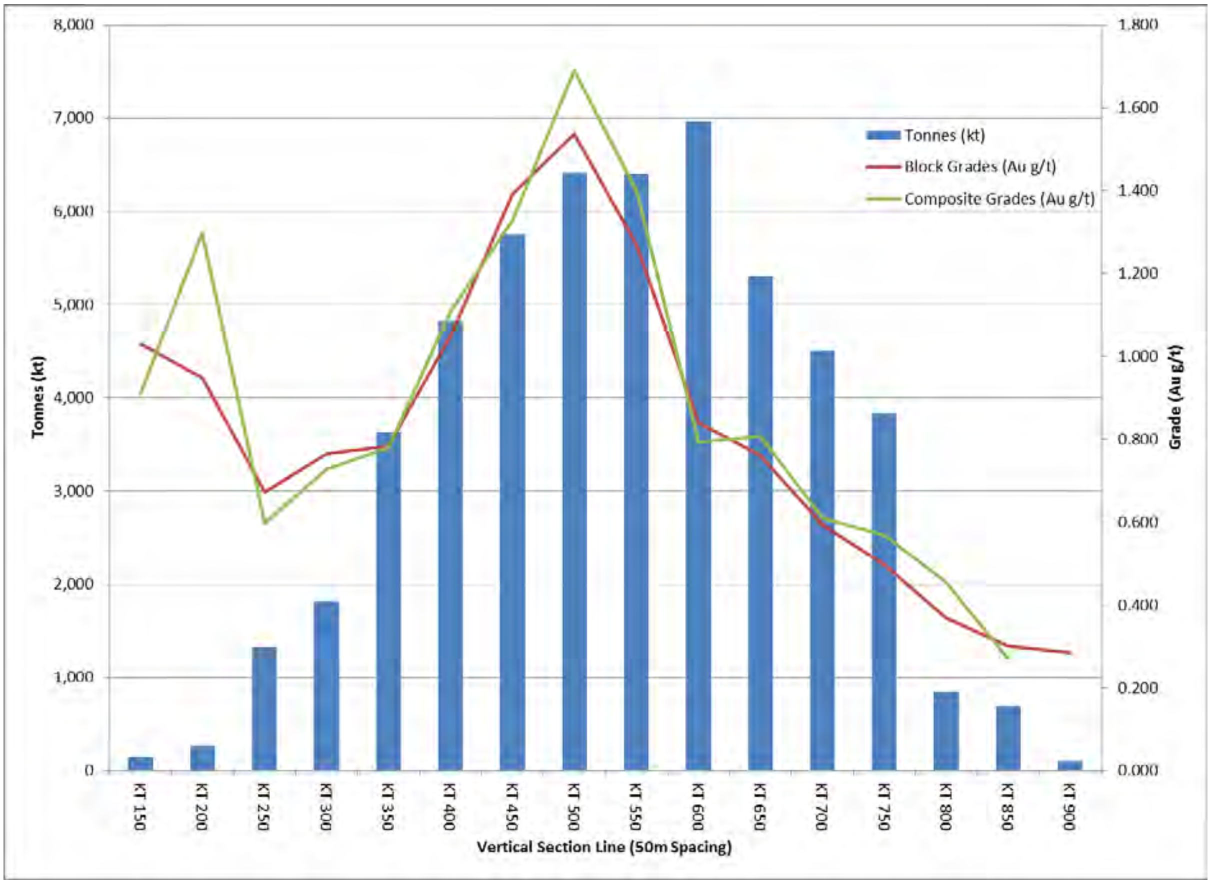


FIGURE 14-21 GÜNEYTEPE GRADE AND TONNAGE CURVE



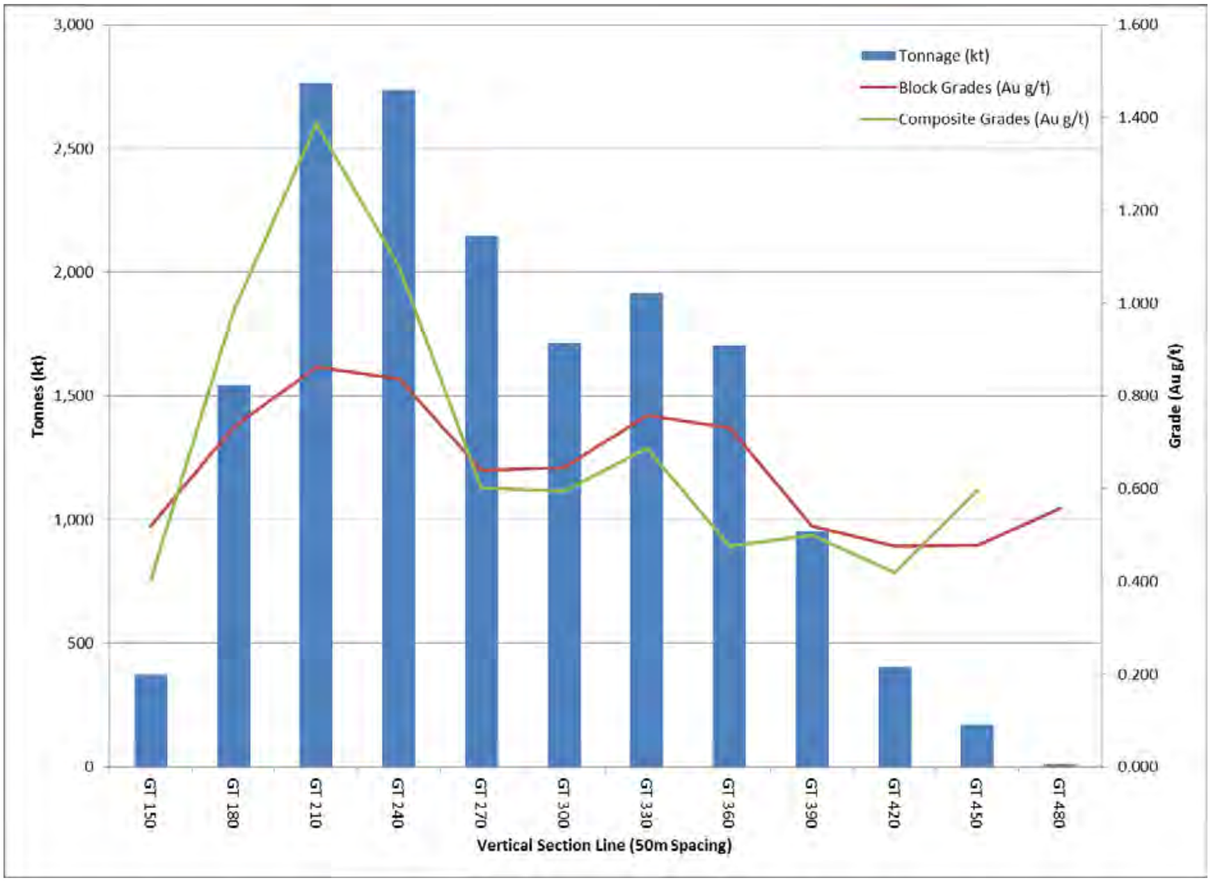
Swath plots were prepared to evaluate and compare trends in composite and block grades throughout the Öksüt Project. These plots were compiled on 50 m swaths at Keltepe and 30 m swaths at Güneytepe to correspond with the spacing of drilling fences through each deposit (southeast/northwest). In Centerra's opinion, the plots at both Keltepe and Güneytepe show a good comparison between the blocks and the composites, particularly in portions of the deposit with the greatest drilling density (Figures 14-22 and 14-23). Again, there is noticeable difference between the composite grades and block grades at Güneytepe between sections GT 180 and GT 240. This drop in block grades is caused by the declustering of higher grade composites in the upper portion of Güneytepe when estimating using OK.

FIGURE 14-22 KELTEPE SWATH PLOT (SOUTHEAST/NORTHWEST)



centerraGOLD

FIGURE 14-23 GÜNEYTEPE SWATH PLOT (SOUTHEAST/NORTHWEST)



Centerra is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the resource estimate.

15 MINERAL RESERVE ESTIMATE

DEFINITION

Mineral Reserves are that part of the Mineral Resource that can be legally, safely, and profitably mined given a specific set of technical and economic parameters. These include the gold price, mine, mill, and administrative operating costs, metallurgical recovery, geotechnical behaviour of the rocks in the future pit walls, and equipment size parameters. Computer software "optimizes" the pit shape by interrogating each block of the block model as to its ability to pay for its removal plus the incremental tonnage of waste that must be removed to mine the block. Detailed mine planning using commercial software then creates a number of intermittent pit designs that test the ability to access sufficient ore to provide adequate mill feed while postponing waste mining as long as possible. This process results in the creation of one or more "pit shells" which recover the economic part of the Mineral Resources and which are then engineered in detail by adding ramps for mining access and by smoothing of the pit walls.

For the Öksüt Project, Mineral Reserves are the "economically mineable part of a Measured and/or Indicated Mineral Resource" as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resource and Mineral Reserves adopted by the CIM Council on May 10, 2014 and incorporated by reference into NI 43-101.

OPEN PIT MINING

The Öksüt Project is planned as a conventional truck and shovel open pit mine. Material will be drilled and blasted, before being loaded and hauled to the waste dump, crusher, or the various ore stockpiles depending on the most profitable way to process the material. At the Öksüt Project, two pits have been planned to be mined simultaneously, the main Keltepe pit and the small satellite Güneytepe pit.

OPTIMIZATION

Pit optimization was carried out using GEOVIA's Whittle v4.5 software on the ÖksütMay2015 block model. Input parameters were developed based on preliminary cost estimations, and nested shells were developed reflecting optimal pits at different gold prices. In all cases, only resources that were considered to be oxide and were classified as either Measured or Indicated were included in the optimization.

Table 15-1 summarizes the parameters used for the final pit optimization.

TABLE 15-1 PIT OPTIMIZATION PARAMETERS

Input	Unit	Value
Mining Cost	$/t mined	2.24
Processing Cost	$/t processed	5.25
G&A Cost	$/t processed	2.91
Gold Price	$/oz Au	1,250
Gold Recovery	%	77
Royalty	%	3.6
Refining Cost	$/oz Au	5.00
Mining Recovery	%	100
Mining Dilution	%	0
Pit Slopes	degrees	Variable by sector Keltepe 36-40 Güneytepe 40

OPERATING COSTS

For optimization purposes, process and general and administrative (G&A) costs were developed from first principles, while the mining cost was based on a contractor mining quote and an estimate of additional owner's costs that will be incurred.

ROYALTIES

The royalties applied to the Öksüt mine operations are a Turkish Government State royalty, a NSR royalty payable to Stratex Gold, and a NSR royalty to Teck each of which are described in Section 4.

Preliminary work suggested that gold production would be slightly less than one million ounces. This results in the average royalty rate being 0.6% payable to Teck, 1% NSR payable to Stratex Gold, and 4% payable to the Turkish Government assuming a $1,250/oz price of gold.

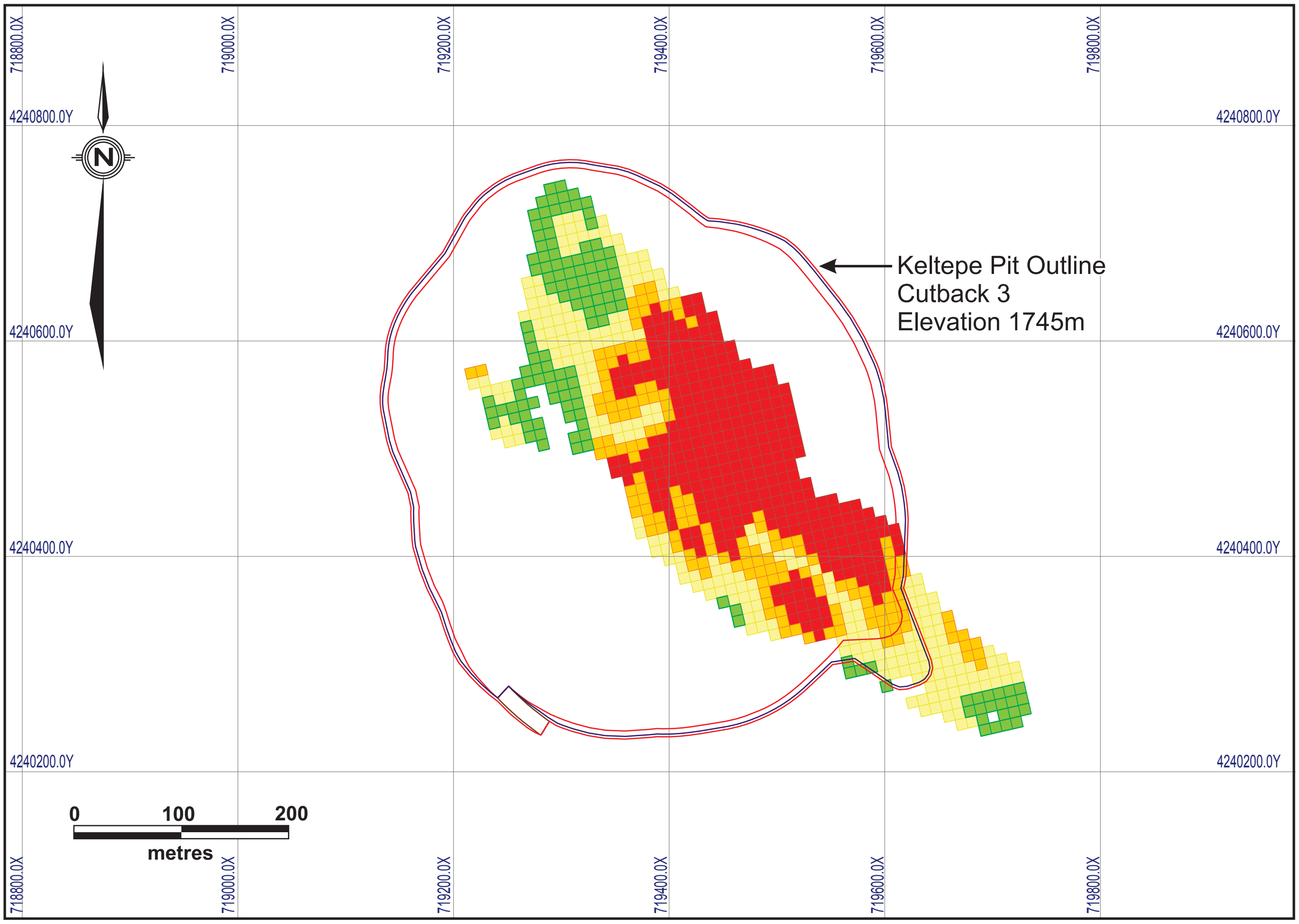
However, Centerra plans to refine the doré from Öksüt using a refinery in Turkey, which would reduce the government royalty by 50% to 2%. The overall royalty rate that was used in the final pit optimization was 3.6%.

METALLURGICAL RECOVERY

Based on current metallurgical test work results, a flat recovery rate of 77% has been used for all oxide material in the pit optimization.

DILUTION AND MINING LOSSES

No external dilution or mining losses have been applied. The Keltepe and Güneytepe Deposits are very continuous and the cut-off grade is low. It is expected that with the small mining fleet chosen, and due to the nature of the deposits and the cut-off grade, the deposits can be mined selectively with minimal ore losses and dilution experienced. The leach pad is designed with additional capacity than is required for the Mineral Reserves, so the main impact from unplanned dilution will be the additional costs to process the material. As ore mining starts, this assumption should be monitored for validity and adjusted if required. Figure 15-1 shows the continuity of the ore body in the Keltepe pit. Only blocks above the cut-off grade of 0.3 g/t Au are shown.



Gold Block Model Au g/t

- > 1.50 g/t Au
- 1.00 to 1.50 g/t Au
- 0.60 to 1.00 g/t Au
- 0.30 to 0.60 g/t Au

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Centerra Gold Inc.

Öksüt Gold Project, Turkey
2015 Technical Report

Keltepe Cutback 3
1745m Elevation
Gold Block Model

Date: Sept. 2015	File: KeltepeBMCB3.cdr	Figure 15-1

SLOPE ANGLES

After a first pass of pit optimization using estimated overall slopes described in Section 16, overall slope angles were re-measured to improve the estimate of the impact of the ramp system on overall slope angles. These slopes were then used in the final pit optimization and are shown in Table 15-2 for the Keltepe pit. Overall slopes of 40º were used in the Güneytepe optimization.

TABLE 15-2 OVERALL SLOPE ANGLES USED FOR KELTEPE PIT OPTIMIZATION

Slope Azimuth (º)	Overall Slope Angle (º)
353	36
43	36
75	36
153	40
250	38
313	40

RESULTS AND SHELL SELECTION

Optimization was completed for the Keltepe and Güneytepe pits separately from each other to ensure that each selected pit shell is the optimal one. Shell selection was made at the point where the operating cash flow curve flattens and the incremental material does not contribute significantly to an overall increase in operating cash flow of the Project. Figure 15-2 shows the results of the optimization for Keltepe. Shell 24 was selected as the optimal shell.

FIGURE 15-2 KELTEPE PIT OPTIMIZATION RESULTS USING A $1,250/OZ GOLD PRICE

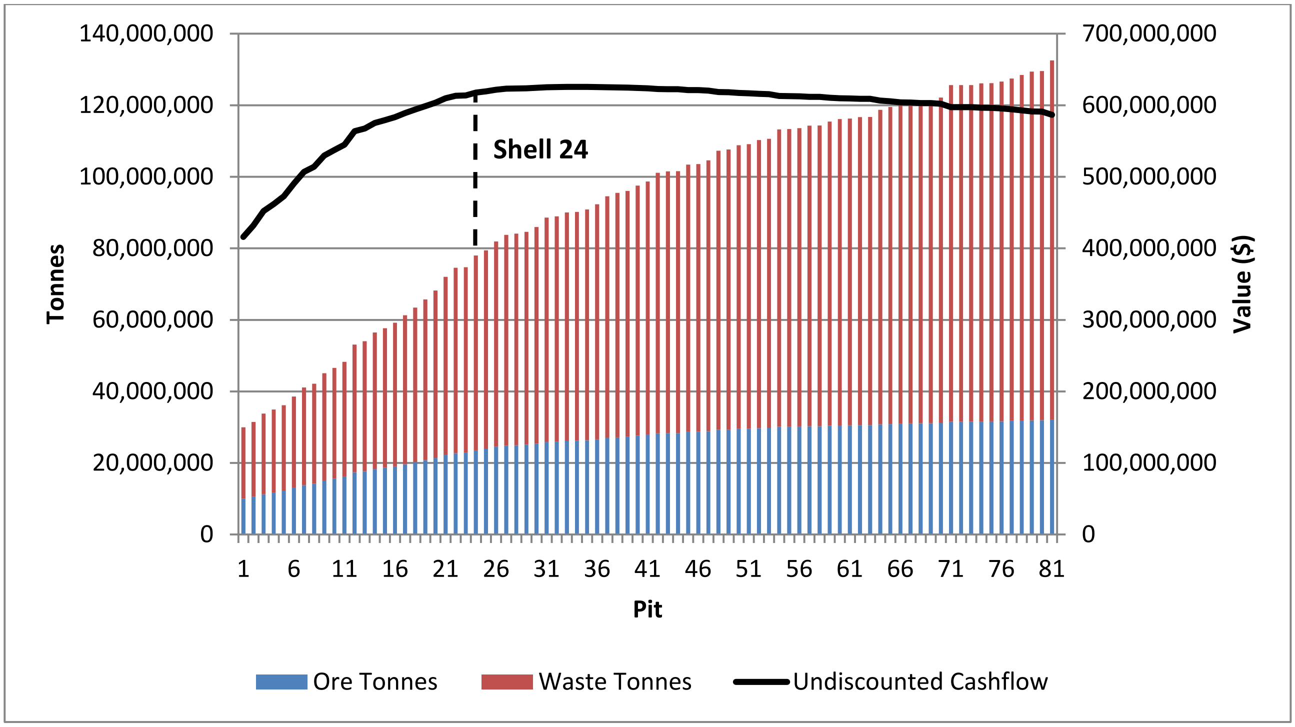
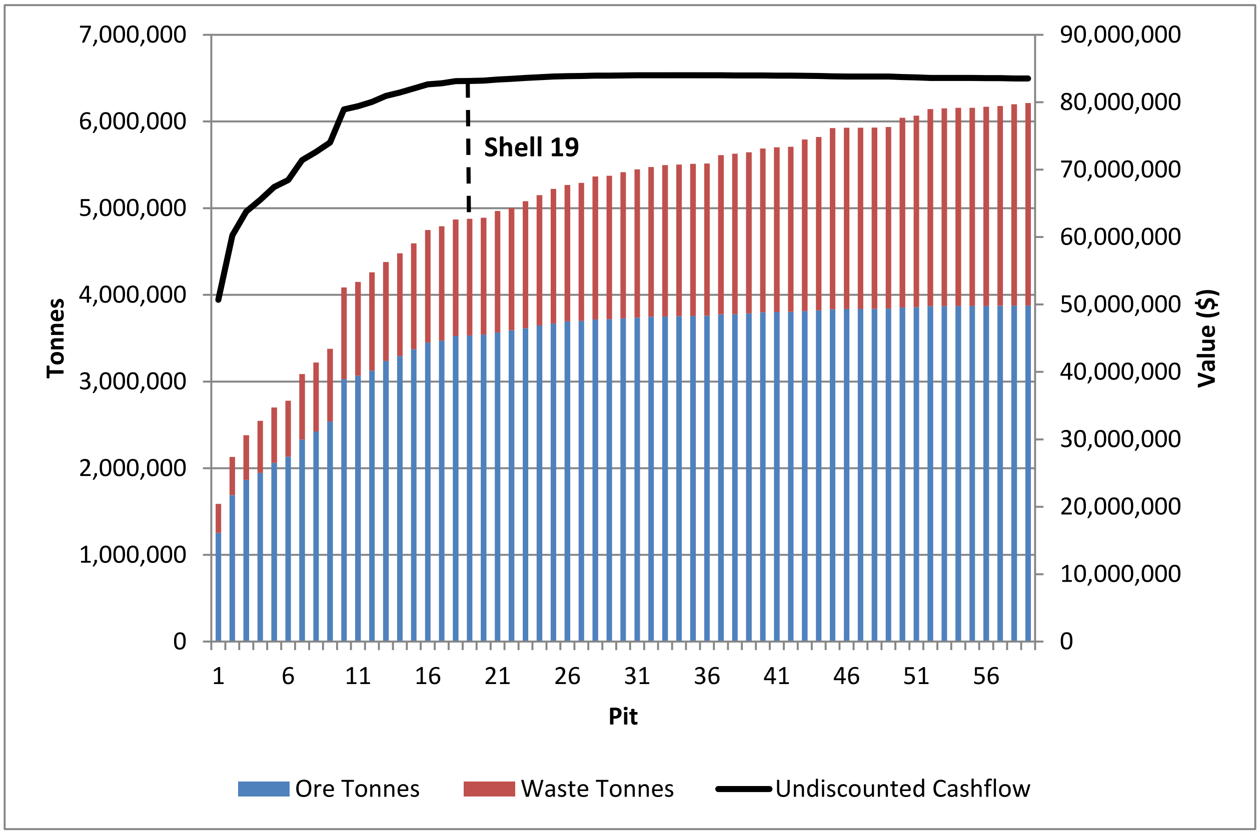


Figure 15-3 shows the results of the optimization for Güneytepe. Shell 19 was selected as the optimal pit shell.

FIGURE 15-3 GÜNEYTEPE PIT OPTIMIZATION RESULTS USING A $1,250/OZ GOLD PRICE



Figures 15-2 and 15-3 both show a relatively flat cash flow curve beyond the chosen pit shells. This shows that, at a gold price of $1,250/ounce, expanding the pits would result in a minimal increase to operating cash flow. By choosing a smaller pit that still captures most of the cash flow, the discounted value of the Project can be increased significantly.

MINE DESIGN

Mining will be done utilizing conventional truck and shovel method, with rock material blasted five times every week. At Keltepe, the open pit will be accessed directly from the haulage road for the upper benches above 1,740 m elevation with the in-pit road constructed for the lower benches. At Güneytepe, all material will be hauled over to join the haul road near the Keltepe pit entrance. The in-pit road width will be 15 m, which is acceptable for 36 tonne trucks running on a two-lane road. Road gradient will be 10%. At the bottom of the pit, the road will be converted to a 10 m wide single-lane road.

Single bench height will be 5 m and double-benching is proposed, which brings the overall height of the final bench to 10 m. Batter angle of the bench slope is 63º. No access to the berms is envisaged in this design. Respective inter-ramp and overall angle constraints are implemented for the corresponding sectors of the walls based on the geotechnical slope design table. Minimal mining width of 30 m was targeted during cut-back design. Mining widths are mostly 50 m or more, but in some places the mining widths narrow to 30 m.

Table 15-3 shows a comparison between Shell 24 and the pit design for Keltepe. The pit design captures 98% of the gold ounces in the shell while mining 94% of the overall tonnes. This is considered acceptable for moving from a pit shell to a design.

TABLE 15-3 KELTEPE PIT DESIGN COMPARISON WITH OPTIMIZED SHELLS

	Keltepe Shell 24	Keltepe Pit Design	Variance
Ore (Mt)	23.3	22.8	-2%
Grade (g/t)	1.4	1.4	0%
Contained Gold (koz)	1,062	1,036	-2%
Recovered Gold (koz)	817	798	-2%
Waste (Mt)	52.7	48.9	-7%
Total Material (Mt)	76.1	71.7	-6%
Strip Ratio (w:o)	2.3	2.1	-5%

A comparison between Shell 19 and the pit design for Güneytepe is shown in Table 15-4. Due to the small size of the Güneytepe pit, additional waste material had to be included in the design in order to maintain minimum working widths. Due to the low strip ratio, this increased the waste tonnes by over 50%, however, this is only an additional 800 kt, which is considered to be acceptable.

TABLE 15-4 GÜNEYTEPE PIT DESIGN COMPARISON WITH OPTIMIZED SHELLS

	Güneytepe Shell 19	Güneytepe Pit Design	Variance
Ore (Mt)	3.2	3.3	4%
Grade (g/t)	1.2	1.2	-2%
Contained Gold (koz)	123	125	2%
Recovered Gold (koz)	95	97	2%
Waste (Mt)	1.4	2.2	56%
Total Material (Mt)	4.6	5.5	20%
Strip Ratio (w:o)	0.4	0.7	51%

CUT-OFF GRADE CALCULATION

The cut-off grade for the Öksüt Mineral Reserves was calculated using the same inputs as those used for the pit optimization. This calculation is summarized in Table 15-5. Based on this, a cut-off grade of 0.3 g/t Au was chosen for the Öksüt Mineral Reserves.

TABLE 15-5 CUT-OFF GRADE CALCULATION

Parameter	Value
Gold Price ($/oz)	1,250
Processing and G&A cost ($/t)	8.16
Recovery (%)	77
Royalty to Turkish Government (%)	2.0
Royalty to Teck (%)	0.6
Royalty to Stratex (%)	1.0
Refining Cost ($/oz)	5.00
Cut-off grade (g/t Au)	**0.3**

Due to the high grade nature of the deposit in comparison to the cut-off grade, small changes to the cut-off grade do not have a significant effect on the Mineral Reserves. Increasing the cut-off grade from 0.3 g/t Au to 0.4 g/t Au decreases the ore tonnage by 4% and the contained gold by only 1%. Decreasing the cut-off grade from 0.3 g/t Au to 0.2 g/t Au only adds 1% more to the ore tonnes and a negligible amount to the contained gold.

MINERAL RESERVES

Table 15-6 summarizes the Mineral Reserves at a 0.3 g/t Au cut-off grade for the Keltepe and Güneytepe pits. All oxide resources classified as either Measured or Indicated inside the reserve pits have been classified as Probable Mineral Reserves.

TABLE 15-6 OXIDE MINERAL RESERVE SUMMARY BY CLASSIFICATION USING A CUT-OFF GRADE OF 0.3 G/T AU (AT JUNE 30, 2015)

	Keltepe			Güneytepe			Combined		
Class	**Tonnes (Mt)**	**Grade (g/t)**	**Contained Gold (koz)**	**Tonnes (Mt)**	**Grade (g/t)**	**Contained Gold (koz)**	**Tonnes (Mt)**	**Grade (g/t)**	**Contained Gold (koz)**
Proven	-	-	-	-	-	-	**-**	**-**	**-**
Probable	22.8	1.4	1,036	3.3	1.2	125	**26.1**	**1.4**	**1,162**
Total	22.8	1.4	1,036	3.3	1.2	125	**26.1**	**1.4**	**1,162**

Notes:
1. CIM definitions were followed for classification of Mineral Reserves.
2. Mineral Reserves have been estimated based on a gold price of $1,250/oz.
3. Mineral Reserves are estimated based on a cut-off grade of 0.3 g/t Au.
4. A conversion factor of 31.10348 grams per ounce of gold is used in the reserves estimates.
5. Numbers may not add up due to rounding.

In the QP's opinion, except as set out in this Technical Report, there are no risks known at this time that could materially affect the Mineral Reserve estimate.

16 MINING METHODS

The Öksüt Project is planned as a conventional truck and shovel open pit mine. Material will be drilled and blasted, before being loaded and hauled to the waste dump, crusher, or the various ore stockpiles depending on the most profitable way to process the material. At the Öksüt Project, two pits have been planned to be mined simultaneously, the main Keltepe pit and the small satellite Güneytepe pit.

GEOTECHNICAL PIT DESIGNS

SLOPE DESIGN

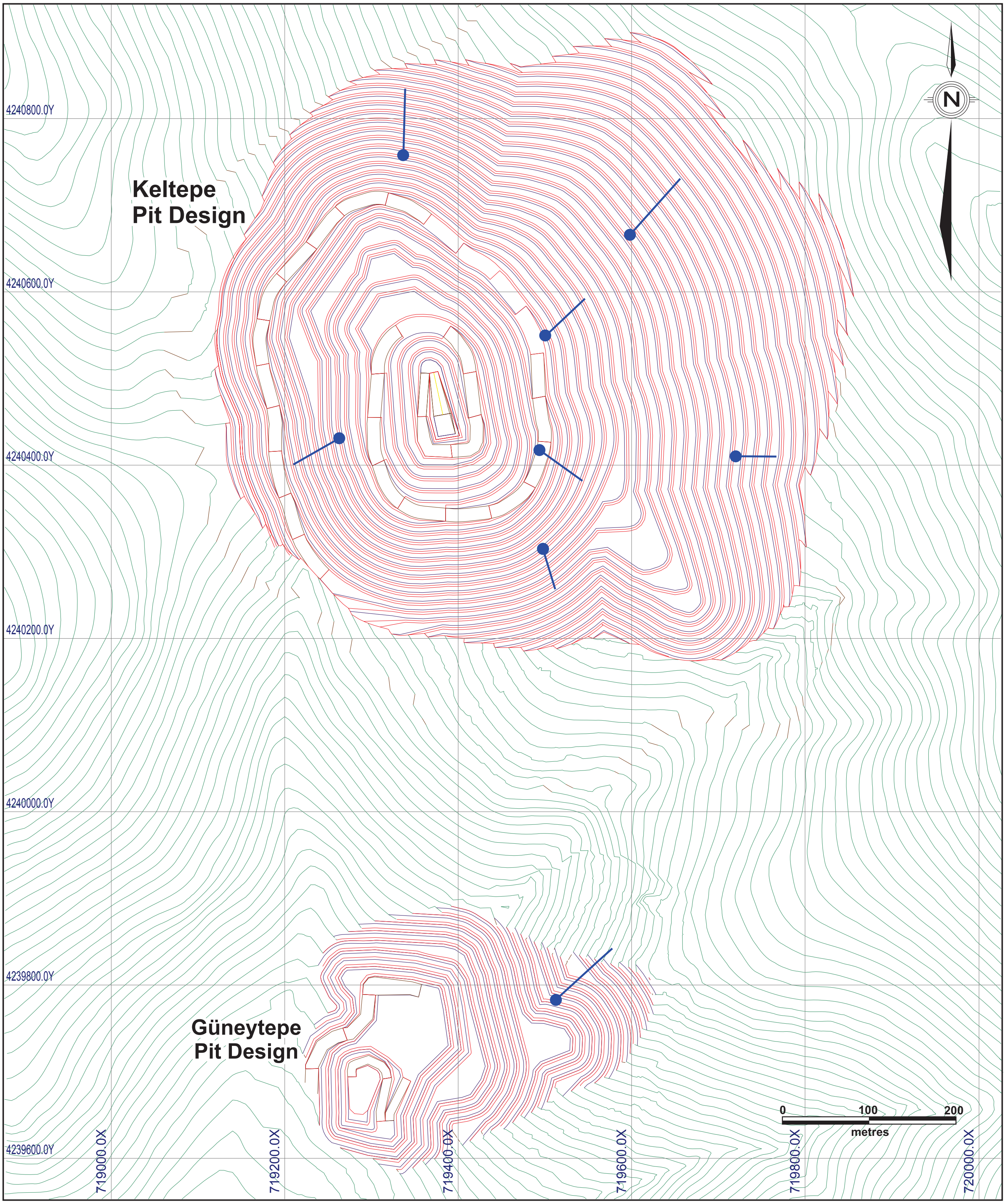
A comprehensive geotechnical field investigation program was carried out by Golder Associates (Golder, 2014d). The program was based upon eight geotechnical drill holes drilled into the main walls of the proposed Keltepe and Güneytepe final pits (Figure 16-1). The drill holes were logged and mapped according to accepted geotechnical standards with oriented core measurements taken using a Reflex ACT II tool as well as the Spear method. The majority of the hole measurements were taken using the ACT tool. Table 16-1 shows the tests that have been completed:

TABLE 16-1 SUMMARY OF THE TESTS COMPLETED DURING FIELD INVESTIGATION PROGRAM

Test Type	Number of Tests Completed
Uniaxial Compressive Strength	24
Indirect Tensile Strength	25
Triaxial Compressive Strength	25
Direct Shear Strength – Rock discontinuities	10
Direct Shear Strength – Fault gauge	6
Soil classification – Fault gauge	6

This field investigation program formed the basis for the slope stability calculation report (Centerra, 2015) that was developed by Centerra's technical team, and was reviewed by an external consultant. Two open pits are planned at the Öksüt site with Keltepe being the main pit and Güneytepe, the smaller pit located to the south of the Keltepe pit.

Structural geology at the Öksüt site is characterized by a series of joints dipping at moderate to shallow angles towards the W-SW. There is a second joint system determined from the geotechnical drill holes that dips at shallow angles towards the N-NE and will be controlling stability of the S-SW wall. Rock mass properties were defined by means of geotechnical drill holes and indicate Fair to Poor rock mass conditions. The rock mass is broken up and significant faulting/shearing of the rock mass is registered. The major fault structures, while difficult to map in the drill holes, have been identified as steeply dipping structures dipping towards the W-SW and sub-parallel to the major high wall of the Keltepe open pit. It was determined by drilling that the high wall of the Güneytepe pit will be controlled by the shear zones of the same orientation (W-SW).



Geotechnical borehole

Centerra Gold Inc.

Öksüt Gold Project, Turkey
2015 Technical Report

Location of Geotechnical Boreholes

Date: Sept. 2015	File: GeoTechDrilling.cdr	Figure 16-1

Every other wall of both the Keltepe and Güneytepe pits will be controlled by sub-horizontal (or shallow dipping) joint sets forming the basal planes combined with the rock mass.

No water was used for the base-case stability calculations as the rock mass is heavily fractured and it is expected that water, if any, will be easily drained by exposed open pits or drill holes. Also, given the current level of understanding of the hydrogeology model as described later in this section of this report the assumption of the “dry” state of the walls is deemed appropriate. Sensitivity to different water tables has been provided for the highest wall of the Keltepe pit Table 16-2.

TABLE 16-2 WALL STABILITY SENSITIVITY ASSESSMENT WITH RESPECT TO VARIOUS WATER TABLE SCENARIOS

Water Table Assumption	Factor of Safety
Dry	1.2
Low	1.2
Medium	1.0
High	0.7

All the work done to date by SRK Consulting Ankara (SRK) has indicated a dry or very low water table behind the ultimate wall. Given that even a “Medium Water Table” assumes a relatively high level of water saturation, it is considered unlikely that the wall will deform due to water related issues. It is still advised that the hydrogeological assumptions be confirmed or revisited after mining commences. In case there are material changes to the water table assumptions, the slope stability analyses should be updated. The pit slope configuration is indicated in Table 16-3. The current pit designs do not make allowances for shallower slopes in the colluvium or highly weathered rock near surface. As mining starts, small adjustments will have to be made to the pit crests depending on the actual depth of this material. It is not expected that this will have a material impact on the Project.

TABLE 16-3 PIT SLOPE SUMMARY FOR ÖKSÜT SITE

Pit	Material	Slope Design Azimuth	Max Inter-ramp Angle	Maximum Overall Angle	Design control	Comments
Keltepe	Colluvium	000-360	36	36	Rock mass strength	Assume first 20 m below Topo will be colluvium or highly oxidized rock, which must be constructed at reduced angles.
	Quartz-alunite or argillic rock types	000-115	42	36	Sub-parallel faulting, shallow joint sets	Final wall height: h=300 m. Design inter-ramp: h=100 m. In case vertical distance between ramps exceeds 100 m, approximate slope angles between the two.
		115-360	45	45	Shallow joint sets, rock mass strength	
Güneytepe	Colluvium	000-360	36	36	Rock mass strength	Assume first 20 m below Topo will be colluvium or highly oxidized rock, which must be constructed at reduced angles.
	Quartz-alunite or argillic rock types	000-115	45	42	Sub-parallel shear zones, shallow joint sets	Final wall height: h=210 m. Design inter-ramp: h=100 m. In case vertical distance between ramps exceeds 100 m, approximate slope angles between the two.
		115-360	45	45	Shallow joint sets, rock mass strength	

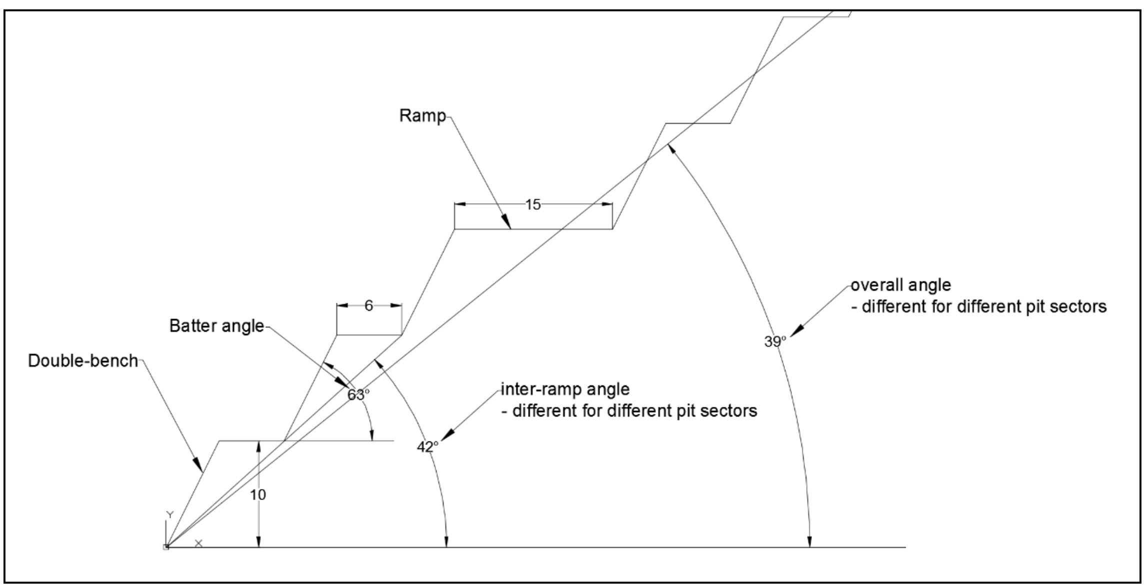
The bench height for both pits will be 5 m, with double benching being employed so that there is 10 m between each safety berm. The bench face angles have been estimated to be 63º. Presplitting will be required in order to reduce damage to the pit walls. The bench widths will be varied in order to achieve the desired inter-ramp angle in each pit sector. These are shown in Table 16-4.

TABLE 16-4 DOUBLE BENCHING DESIGN CRITERIA

Inter-ramp Angle (degrees)	Berm Width (m)
45	4.9
42	6.0
36	8.7

The bench design for an inter-ramp angle of 42º is shown schematically in Figure 16-2.

FIGURE 16-2 SLOPE DESIGN SECTION WITH MAIN BENCH PARAMETERS



GROUNDWATER

SRK has completed a draft hydrological investigation report (SRK Consulting Ankara, 2014d) based on the comprehensive hydrogeological field investigation program that comprised drilling and installing of 30 piezometers/monitoring wells, 10 slug tests, and eight pumping tests in large diameter wells. Information obtained from the tests and subsequent analyses is summarized in Table 16-5.

TABLE 16-5 HYDROGEOLOGICAL SUMMARY

Location	# of Tests	Water Level Below Ground Surface (m)		Water Table Elevation (masl)		Hydraulic Conductivity of the Rock Mass (m/s)	
		Min	Max	Min	Max	Min	Max
Waste Dump	6	116	124	1,713	1,836	2.7e-7	2.1e-6
Keltepe Pit	6	293	360	1,478	1,520	1.9e-9	1.9e-9
Güneytepe Pit	3	84	189	1,531	1,576	4.6e-8	1.5e-10
Heap Leach	8	32	109	1,677	1,867	1.1e-5	2.2e-9

Current findings suggest that during mining the pit walls are expected to be completely dry for Keltepe and Güneytepe pits as their bottoms are designed at 1,585 masl and 1,600 masl, respectively.

WASTE DUMP DESIGN

The waste dump is expected to be located in the Eastern Valley from the proposed Keltepe pit and is secluded from the nearby villages (Figure 16-7). Such a location is thought to be the best available option from an environmental and social standpoint, although at a detriment of longer haulage distance from the pit's rims. Stability calculations were performed by Centerra's technical personnel (Centerra, 2014b), and were reviewed by an external consultant. They determined that the proposed waste dump configuration results in a stable facility for the duration of mine operations. A better understanding of the underlying hydrology and relatively deep-seated water table allowed designing a stable waste rock dump using a top-down dumping approach without a liner.

Dumping will be done using a top-down approach as the intermediate faces of the dump are expected to be stable, although at a marginal factor of safety in some parts during its development. Short-term deformations may happen during these periods, which are considered acceptable. As the waste dump development progresses, the dump stability will improve as subsequent dumping is performed on a shallow foundation. Although further investigation is required, the minimum factor of safety (FOS) for the ultimate waste dump is considered acceptable. Additional field data should be collected, tested, and incorporated into the waste dump model, and the dump design should be adjusted as required.

Appropriate monitoring will be done using prisms and identifying accelerations of the dump. If any acceleration is defined in a particular section of the waste dump, it will be closed off and dumping will not be allowed until an acceptable decrease in the movement rate is observed.

MINE PLANNING

CUTBACK STRATEGY

The Öksüt Project will be a conventional truck and excavator mining operation. The deposit has been separated into three cutbacks for the Keltepe pit and one cutback for the Güneytepe pit. This has been done in order to decrease stripping requirements during start-up of operations, bring higher grade forward by targeting high grade ore in the Keltepe pit, and reduce geotechnical risks.

Keltepe Cutbacks 1 and 2 provide significantly higher grade ore, as well as reducing the stripping requirements when compared to Keltepe Cutback 3. The Güneytepe pit has the lowest strip ratio of all cutbacks, but the ore is lower grade than the first two cutbacks in Keltepe. Table 16-6 summarizes the physicals of all cutbacks.

TABLE 16-6 CUTBACK SUMMARY

	Keltepe Cutback 1	Keltepe Cutback 2	Keltepe Cutback 3	Güneytepe	Total
Ore (Mt)	2.5	9.5	10.8	3.3	26.1
Grade (g/t)	1.7	1.8	1.0	1.2	1.4
Contained Gold (koz)	137	549	351	125	1,162
Recovered Gold (koz)	106	422	270	97	895
Waste (Mt)	5.8	16.0	27.1	2.2	51.1
Total Material (Mt)	8.3	25.6	37.8	5.5	77.3
Strip Ratio (w:o)	2.3	1.7	2.5	0.7	2.0

Figure 16-3 shows that the first two cutbacks in the Keltepe pit are significantly higher grade than either the final cutback of the Keltepe pit or the Güneytepe pit.

FIGURE 16-3 CUTBACK SUMMARY

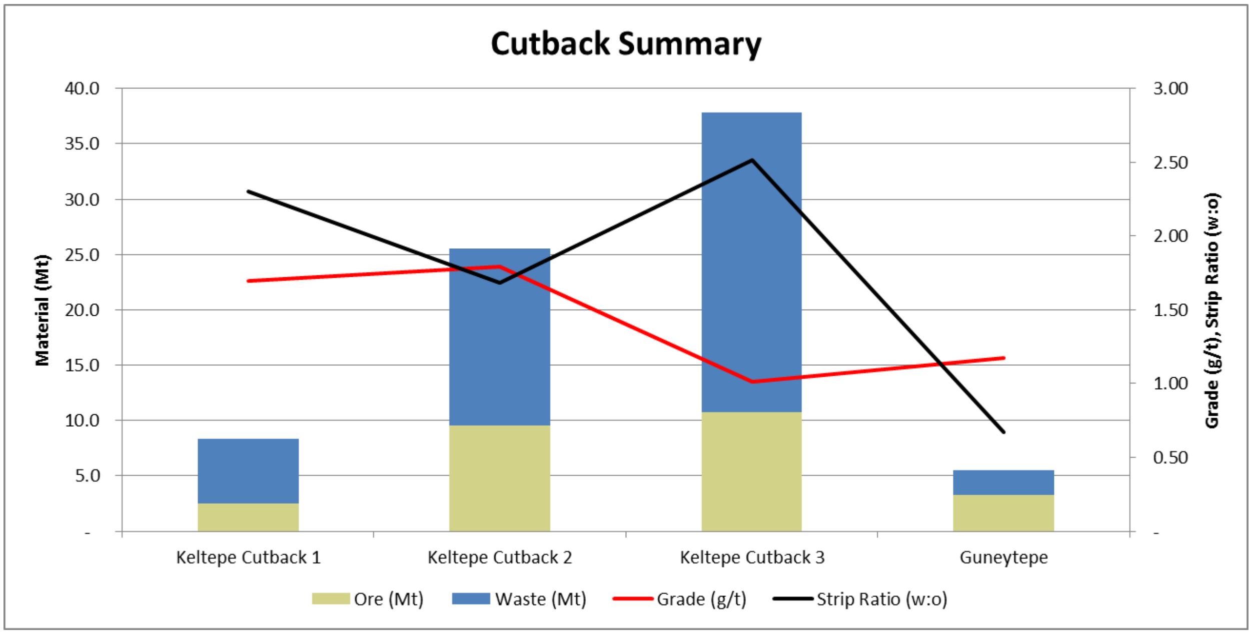
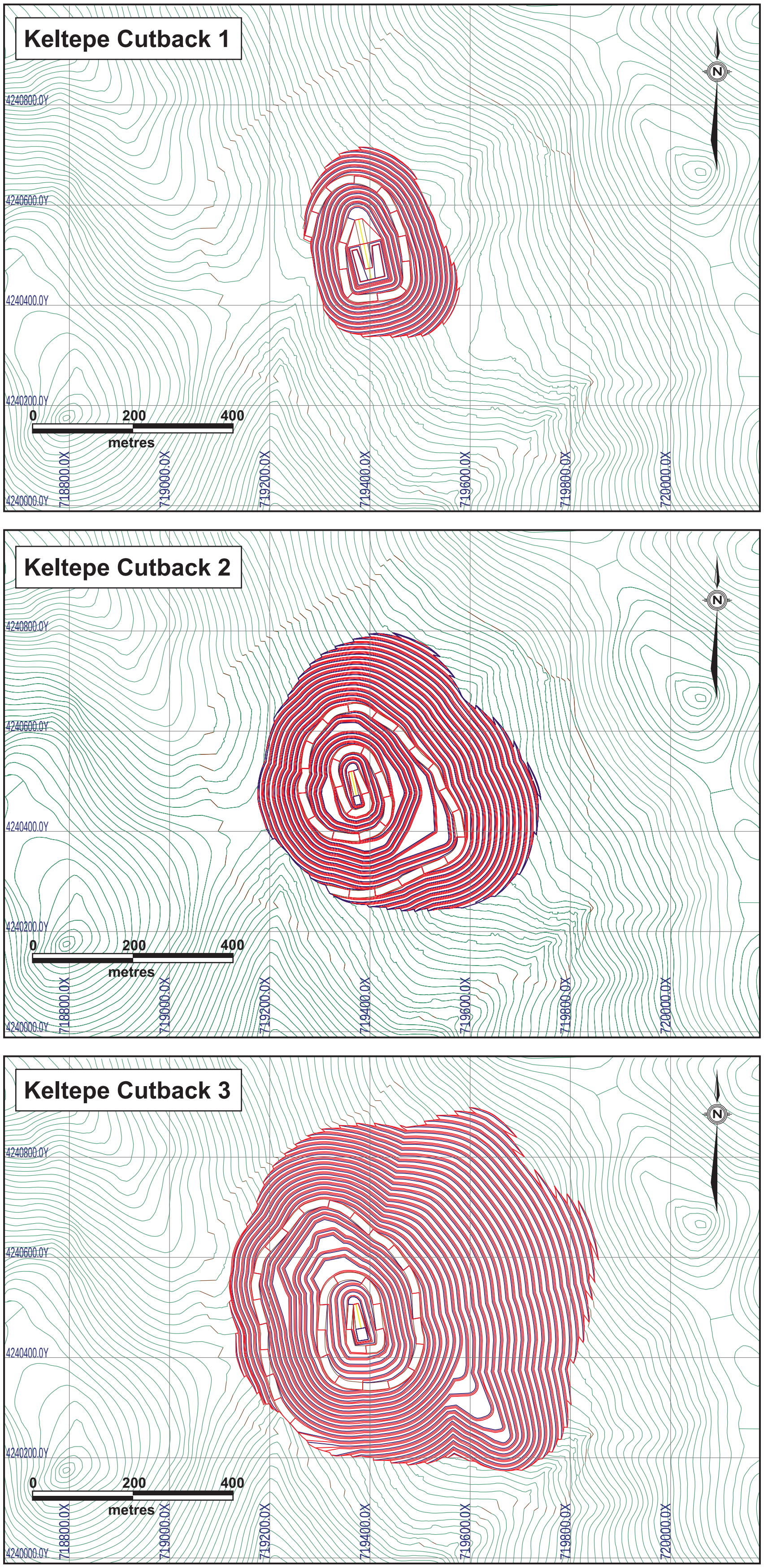


Figure 16-4 shows the three Keltepe cutbacks. Each one involves a single spiral ramp down to the bottom. The ramps have been designed to exit the pit near the lowest point to reduce uphill hauls.



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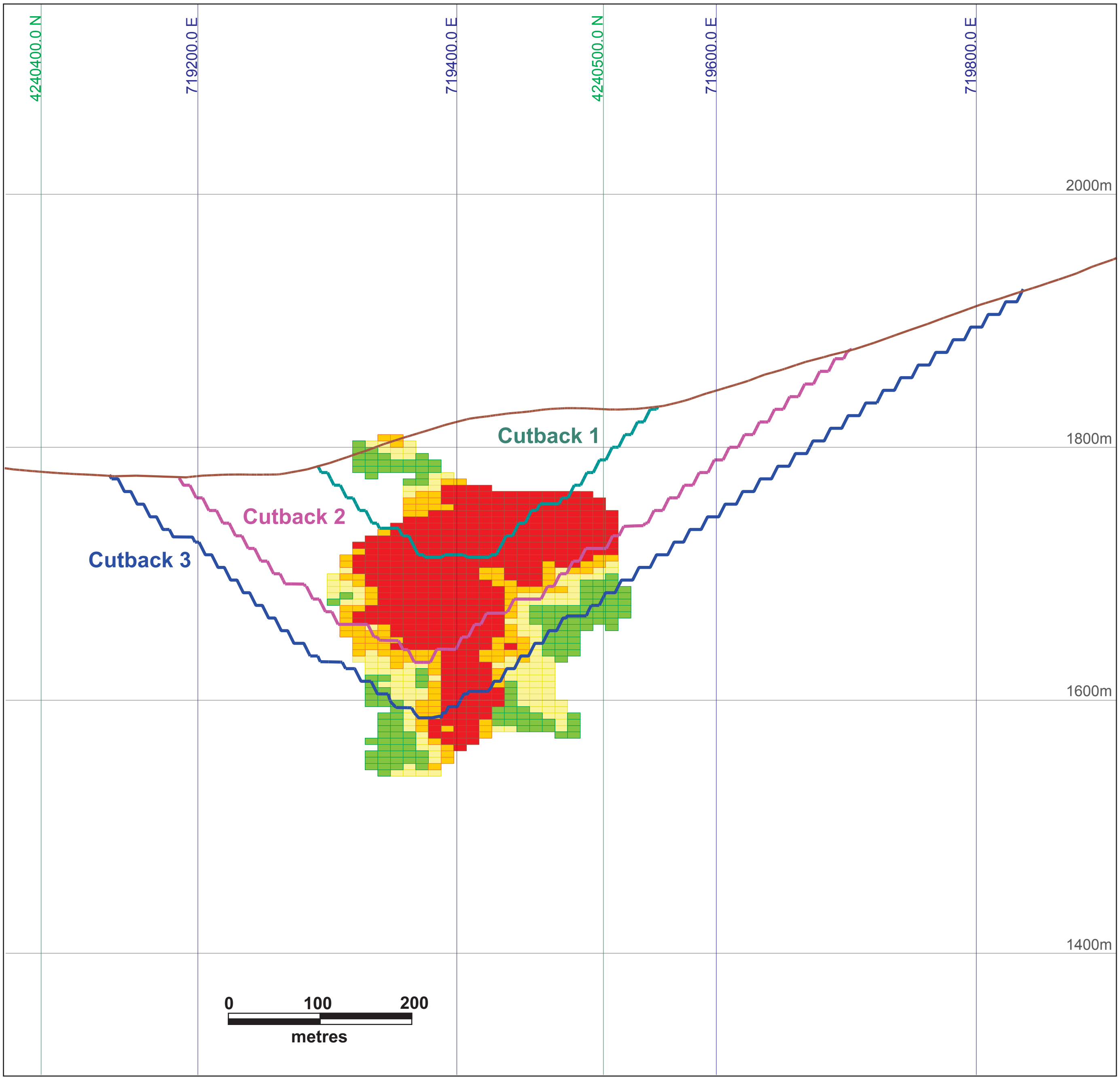
Keltepe Pit Design

Date: Sept. 2015	File: KeltepePits.cdr	Figure 16-4

Figure 16-5 is a cross-section of the Keltepe pit showing the three cutbacks as well as the resource blocks. The first two cutbacks will complete mining of the higher grade, shallow ore. The final cutback requires more stripping to reach the deeper, lower grade portion of the ore body. This strategy increases the production in the first four years of processing significantly, accelerating the payback of the Project.

Figure 16-6 shows the Güneytepe pit design. It will be mined in a single cutback.



Gold Block Model Au g/t

- > 1.50 g/t Au
- 1.00 to 1.50 g/t Au
- 0.60 to 1.00 g/t Au
- 0.30 to 0.60 g/t Au

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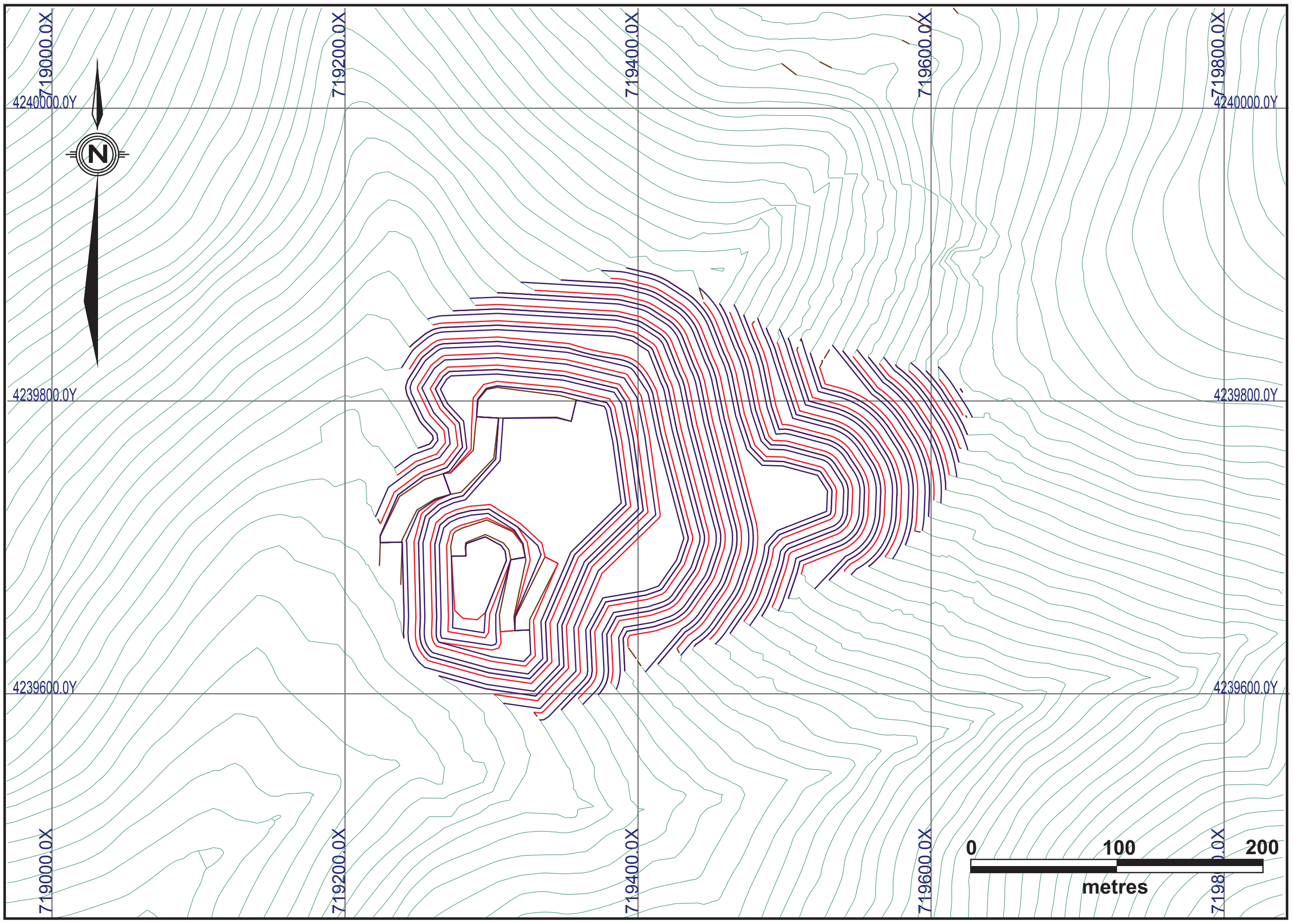
Keltepe Cutbacks and Gold Block Model Section 500KT (looking northwest)

Date: Sept. 2015	File: KeltepeSec500.cdr	Figure 16-5



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Güneytepe Pit Design

Date: Sept. 2015	File: GuneytepePit.cdr	Figure 16-6

HAUL ROAD

The haul road represents an important part of mine infrastructure and will serve as the key element for delivering ore and waste materials to their respective destinations. The haul road was designed to be at least 100 m away from the fence line of the property to allow for some additional flexibility in the detailed stages of the designs for allocation of service roads, dewatering ditches and other infrastructure, if necessary. The haul road width is 25 m. A 15 m wide segment is for the haul trucks, with a separate 10 m wide segment of the road for light vehicles and other traffic. The haul road was mainly designed at a 10% gradient and divided into several segments, which may be constructed in parallel:

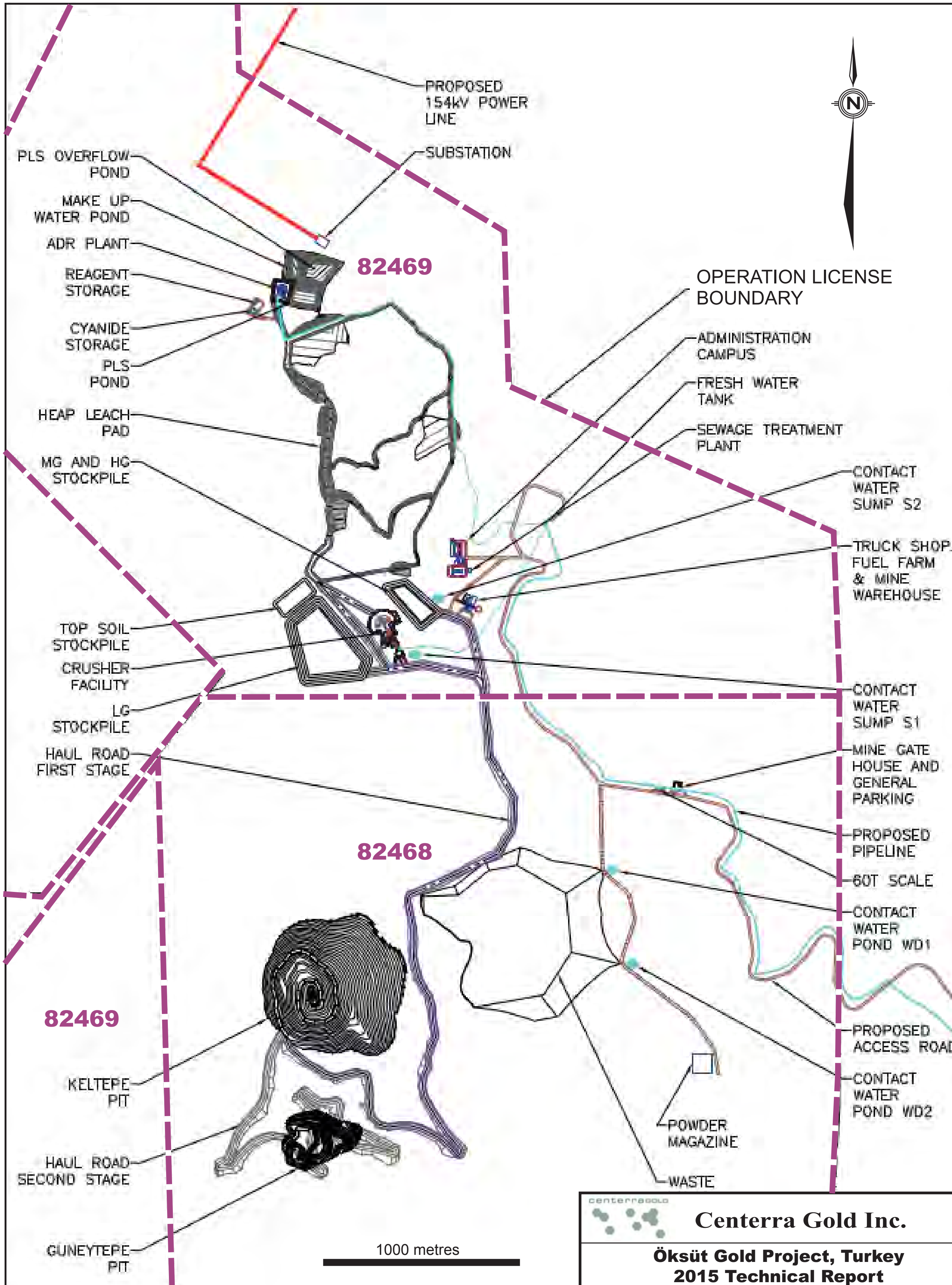
- **Keltepe pit entrance – Waste Dump entrance**. This segment is designed at +10% gradient and is located on steep terrain. This road segment will be constructed first in the sequence, as mining cannot start until access from the pits to the waste dump is available.
- **Waste Dump entrance – start of the Heap Leach Pad.** This segment is designed at neutral/shallow gradients leading to the various infrastructure facilities of the Project, such as stockpiles, truck shop, fuel farm administration building, crusher and heap leach facility. This segment will be constructed second in the sequence, as stripping activities can start prior to its completion. Earlier completion of this road may allow waste rock from the pits to be used in the construction of the heap leach facility, reducing the cost of earthworks.
- **Upper Güneytepe road**. This road segment is designed at 10% gradient and dissects the Güneytepe open pit providing access to the upper benches of the pit down to an elevation of 1,645 masl where the pit entrance is designed. This must be completed in order to begin mining at Güneytepe.
- **Güneytepe entrance – Keltepe entrance.** This segment is designed at +10% gradient and connects the Güneytepe pit with the main haul road. This also must be completed in order to begin mining at the Güneytepe pit.

Road construction will represent both cut and fill in a balanced manner where need for additional material will be eliminated. Table 16-7 summarizes construction requirements for the road segments.

TABLE 16-7 HAUL ROAD DESIGN SUMMARY

Road Segment	Cut (kt)	Fill (kt)	Length (m)
Keltepe (Keltepe pit to Waste Dump)	220	286	2,023
Keltepe (Waste Dump to Heap Leach)	685	696	2,611
Güneytepe Lower (Güneytepe pit entrance to Keltepe pit entrance)	194	203	1,193
Güneytepe Upper	420	191	1,748
Total	**1,520**	**1,376**	**7,575**

It was assumed that the cut material can be used for the road construction as a sub-base material with some gravel material forming the base for the road. Figure 16-7 shows the final pit and waste dump designs, along with the haul road.



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Site Layout

Date: Sept. 2015 | File: InfraStructPlan.cdr | Figure 16-7

IN-PIT RAMP

In-pit ramp width is 15 m for the two-lane access, which is between 3.2 and 3.6 times wider than the operational width of a 36 tonne truck, which is indicative of the likely equipment to be used at Öksüt. This will allow enough width for an internal drainage ditch, a safety berm on the edge of the road, and two running lanes. At the bottom of the pits the ramp width is reduced to 10 m to improve ore recovery. These roads will be reduced to single lane traffic only.

STOCKPILE STRATEGY

In order to increase production early on in the mine life, lower grade ore is stockpiled while higher grade ore is crushed and placed on the pad. Ore will be categorized as Low Grade (0.3 g/t to 0.6 g/t Au), Medium Grade (0.6 g/t to 1.0 g/t Au), High Grade (1.0 g/t to 1.5 g/t Au), and Direct Feed (1.5+ g/t Au). Nearly all direct feed material will be processed without stockpiling, although there will need to be some short term stockpiling when mining in the highest grade areas of the pits. Three stockpile locations have been designed for the rest of the material. One is for top soil that will be stripped and stockpiled, then used for reclamation purposes. The other two are ore stockpiles. The medium and high grade stockpiles will be in the same location. The low grade stockpile will be used only when there is not enough high grade ore to fill the crusher. The rest will be processed at the end of the mine life. The capacities of the designed stockpiles are shown in Table 16-8.

TABLE 16-8 STOCKPILE CAPACITY SUMMARY

Stockpile	Excavation Volume (MBCM)	Fill Volume (MBCM)	Loose Factor	Loose density (t/m^3)	Capacity (Mt)	Max Capacity Required (Mt)
Ore Stockpiles	0.4	2.9	1.3	1.6	4.6	2.3
Topsoil	0.1	0.3	1.3	1.4	0.4	0.3[1]
Total	**0.5**	**3.2**	**1.3**	**1.6**	**5.0**	**2.6**

1 – Surface area of the two pits is 0.5 Mm^2. Thickness of the top soil of 0.3 m is assumed.

MINE SCHEDULE

The cutback designs, block model, haul road designs, and locations for the stockpiles and waste dumps were all brought into RPM's OPMS v.1.2 software. This was then used for mine scheduling.

The following constraints were imposed on the schedule:

- Mining starts in mid-2016 after the road from the pit to the waste dump is constructed.
- Ore placement capacity of 335 kt per month was assigned to the heap leach pad.
- Mining production rate ramp-up was defined such that in months 1 and 2 the mine will operate at 50% of its designed capacity, months 3 and 4 the mine will operate at 75% of its designed capacity until it reaches full capacity in month 5.
- Constraints were placed on the number of excavators permitted in each cutback, safe working areas for each excavator, and vertical advance rates depending on the cutback geometry.
- A stockpile optimizer was used during the scheduling process. Given that other constraints have been honoured, the schedule maximizes the ounce extraction profile by processing the highest grade stockpiles available.

Table 16-9 and Figure 16-8 show the life-of-mine (LOM) mining schedule. Mining begins in mid-2016, with some ore being mined and stockpiled that year, and the final ore being mined in 2023. Table 16-10 and Figure 16-9 show the LOM processing schedule. Crushing begins in 2017 with full production processing operations commencing in Q2 2017. Ore stacking is completed in 2023 while some gold production extends into 2024 as leaching continues. Figure 16-10 shows the annual stockpile balance.

TABLE 16-9 LIFE-OF-MINE MINING SCHEDULE

	Total	2016	2017	2018	2019	2020	2021	2022	2023
Ore Mined (Mt)	**26.1**	0.2	3.8	4.0	5.7	2.3	3.4	5.4	1.4
Grade (g/t)	**1.38**	2.08	1.47	1.41	1.71	1.95	1.00	0.93	1.32
Contained Gold (koz)	**1,162**	15.9	178.4	182.2	310.6	146.1	109.4	161.8	57.3
Ore Rehandled (Mt)	**6.3**	-	0.5	0.7	-	2.2	0.6	0.1	2.2
Waste Mined (Mt)	**51.1**	5.1	8.9	8.8	7.1	10.4	8.6	2.1	0.1
Total Material Mined (Mt)	**77.3**	5.4	12.7	12.8	12.8	12.7	12.0	7.4	1.5

FIGURE 16-8 LIFE-OF-MINE PLAN – MINING SCHEDULE

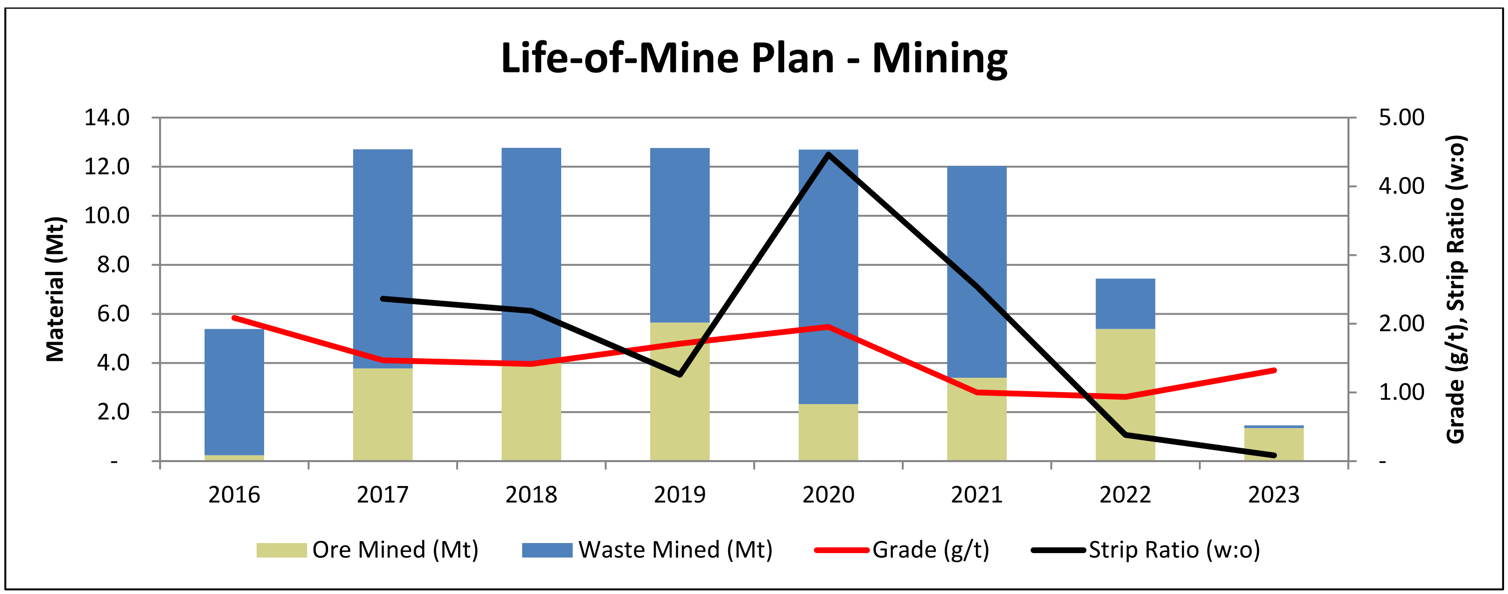


TABLE 16-10 LIFE-OF-MINE PROCESSING SCHEDULE

	Total	2016	2017	2018	2019	2020	2021	2022	2023	2024
Ore Processed (Mt)	**26.1**	-	3.30	4.02	4.02	4.02	3.53	4.02	3.22	-
Grade Processed (g/t)	**1.38**	-	1.69	1.44	2.07	1.47	0.98	1.09	0.83	-
Contained Gold (koz)	**1,162**	-	180	186	267	190	111	141	86	-
Gold Produced (koz)	**895**	-	90	179	199	151	93	106	70	7

FIGURE 16-9 LIFE-OF-MINE PLAN – PROCESSING SCHEDULE

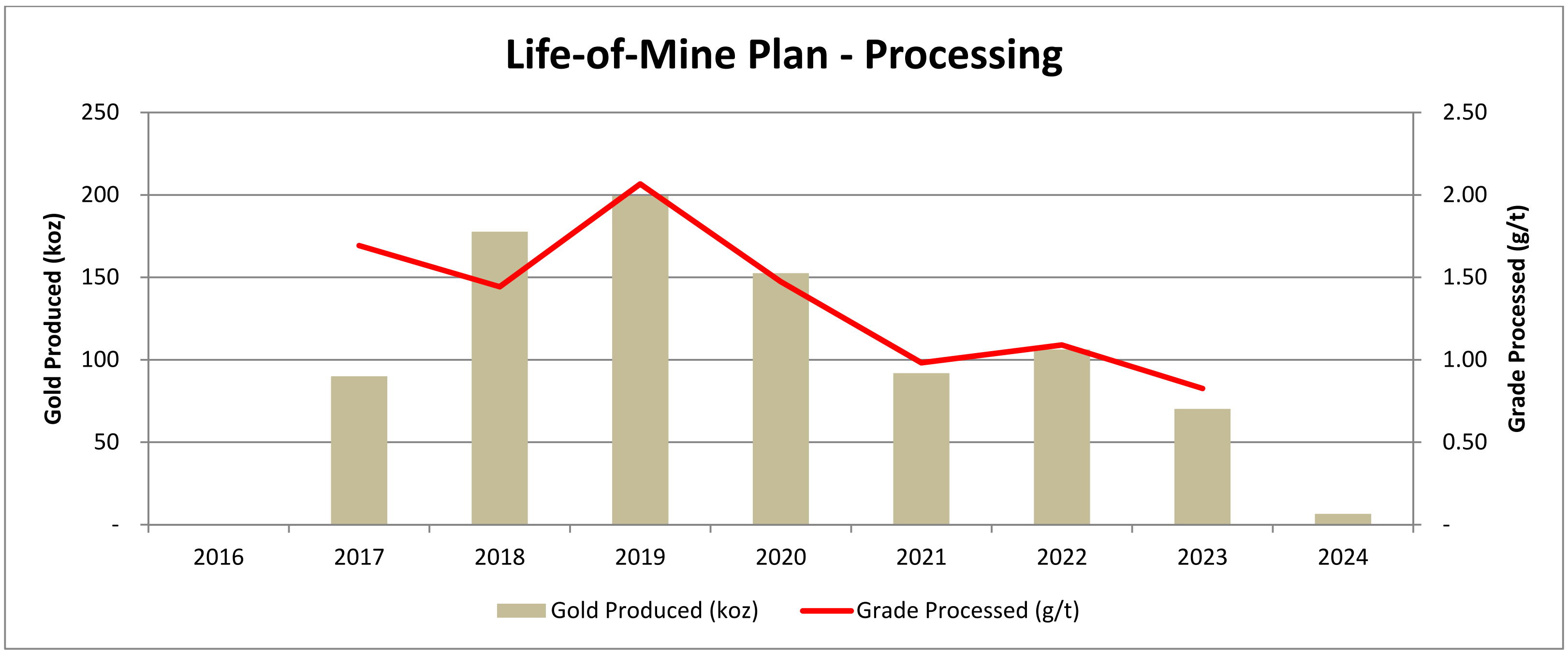


FIGURE 16-10 ANNUAL STOCKPILE BALANCE

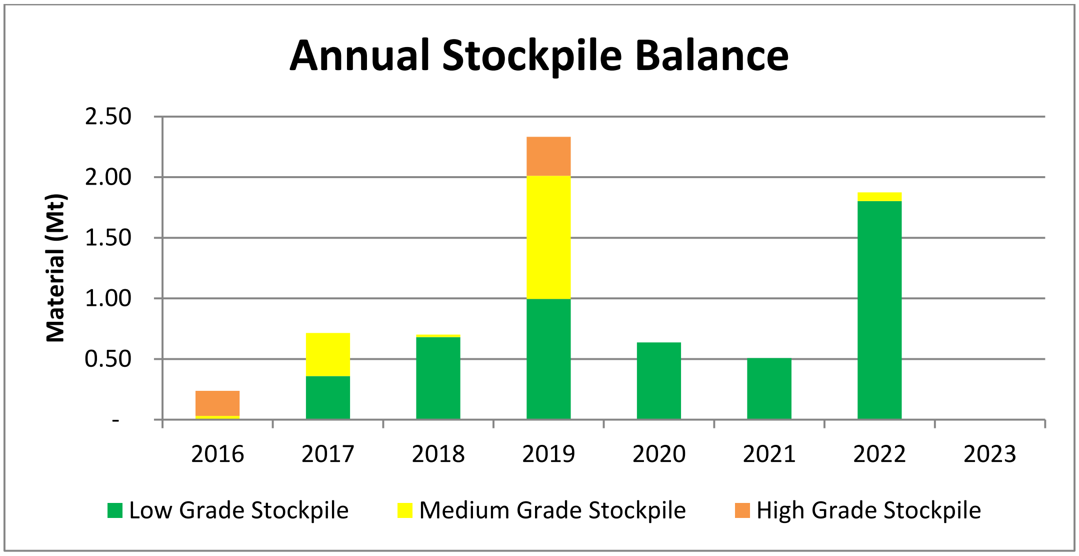
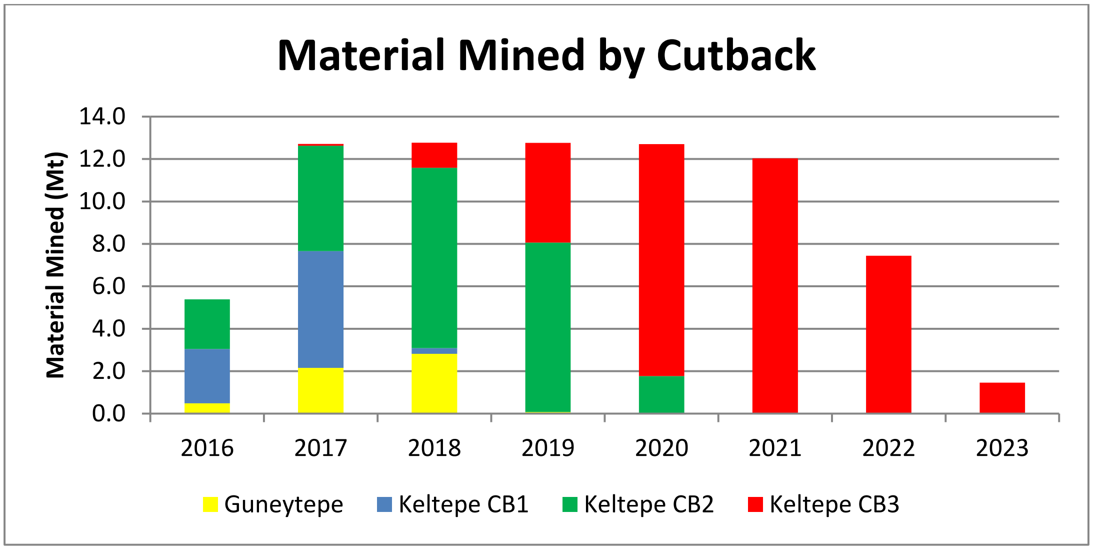


Figure 16-11 shows the material mined by cutback. In 2016, waste stripping of the Keltepe pit begins in cutbacks 1 and 2. Due to the low strip ratio of the Güneytepe pit, it is mined to begin stockpiling ore for processing in 2017. Because of the small size of the Güneytepe pit, it is planned to be mined utilizing only one excavator and is not completed until 2019. As Keltepe Cutback 1 is nearing completion, stripping of Keltepe Cutback 3 begins. By 2020 this will be the primary mining area for the rest of the mine life. This sequence was designed to take advantage of the lower strip ratio, higher grade cutbacks first, helping to increase production in the first four years of the mine life and delaying waste stripping.

FIGURE 16-11 MATERIAL MINED BY CUTBACK



MINING CONTRACTOR AND EXPECTED EQUIPMENT

After discussions with mining contractors in Turkey, it was determined that contract mining would be used at the Öksüt Project. It was determined that a fleet of 3.8 m^3 excavators and 36 t trucks would work well at Öksüt, which is a common size of equipment in Turkey. The mining contractor will be responsible for all activities associated with mining including drill and blast, loading, hauling, and road and dump maintenance. They will also be responsible for supervision of their own personnel. All mine planning, surveying, and ore control activities will be the responsibility of Centerra. The rate paid to the contractor will be based on a $/$m^3$ mined rate agreed upon by the parties. The contractor will be supplied with the mine plan and it will be their responsibility to estimate and supply the equipment required to meet the plan. Equipment fleet estimates were done to ensure the mine plan would be achievable with this size of fleet without excessive congestion in the pit. Table 16-11 shows the availability, utilization, and efficiency assumptions for the production fleet, as well as the peak number of estimated equipment units. Along with this fleet the contractor will need to supply support equipment in the form of bulldozers, graders, explosives trucks, and light vehicles.

TABLE 16-11 EXPECTED PRODUCTION EQUIPMENT FLEET

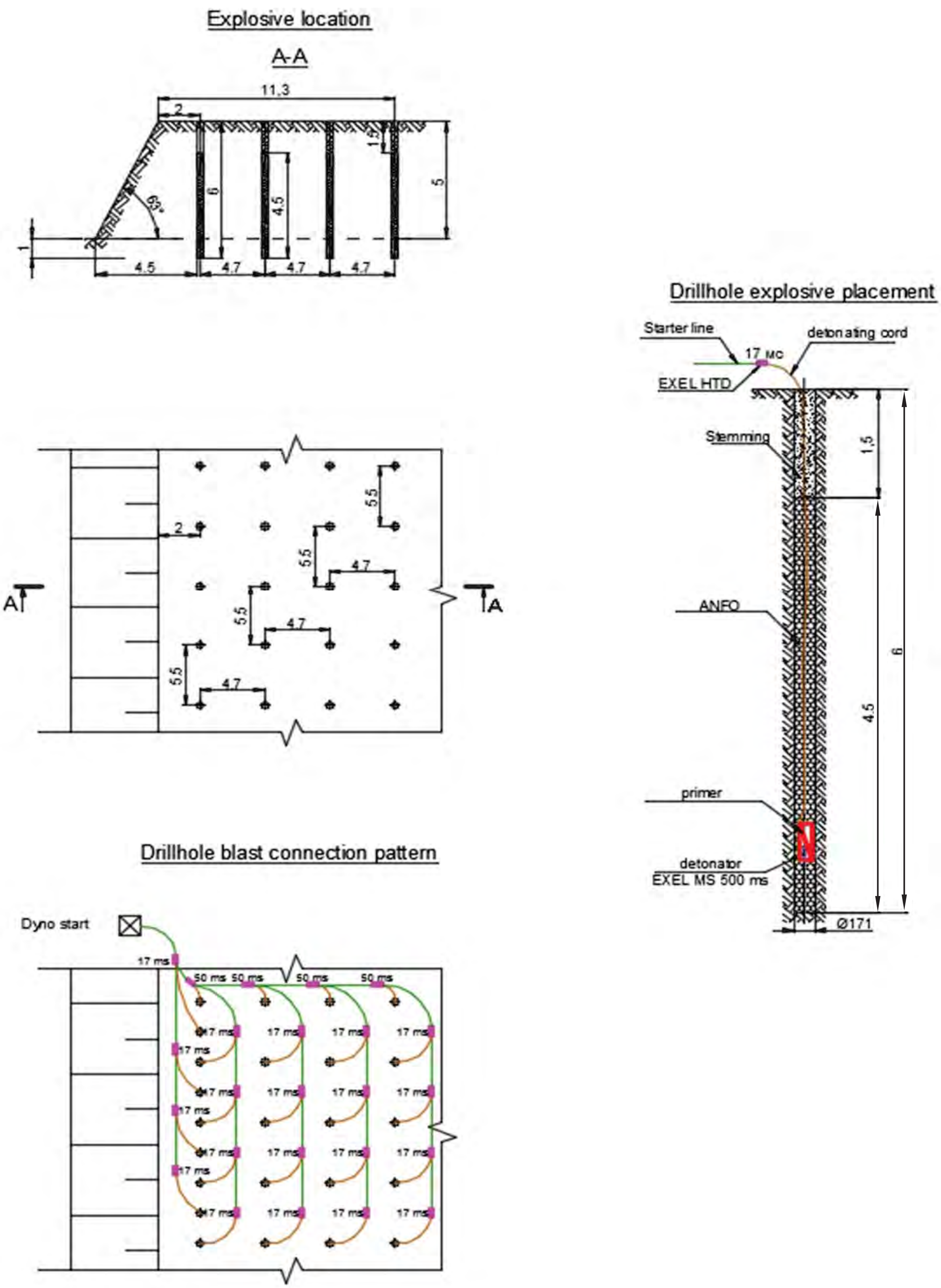
	Excavators	Trucks	Drills
Calendar Hours per Annum	8,760	8,760	8,760
Availability	85%	85%	80%
Utilization	80%	80%	80%
Efficiency	80%	80%	75%
Net Effective Production Hours per Annum	4,765	4,765	4,205
Peak # of units	4	35	4

DRILL AND BLAST

The drill and blast design has been developed by BBA Consulting (BBA Consulting, 2014), where several options were evaluated. The final design parameters are shown in Table 16-12. Figure 16-12 shows a schematic of a typical blast. Pre-splitting will be done on 10 m double benches to reduce damage to the walls.

TABLE 16-12 DRILL AND BLAST SUMMARY

Drill and Blast Design	
Rock density (t/m^3)	2.13
Bench height (m)	5
Hole diameter (mm)	171
Explosive type	ANFO
Explosive density (kg/m^3)	900
Hole length (m)	6
Stemming length (m)	1.5
Burden (m)	4.7
Spacing (m)	5.5
Weight of in-hole explosive (kg)	93.0
Powder factor (kg/m^3)	0.72
Powder factor (kg/t)	0.34



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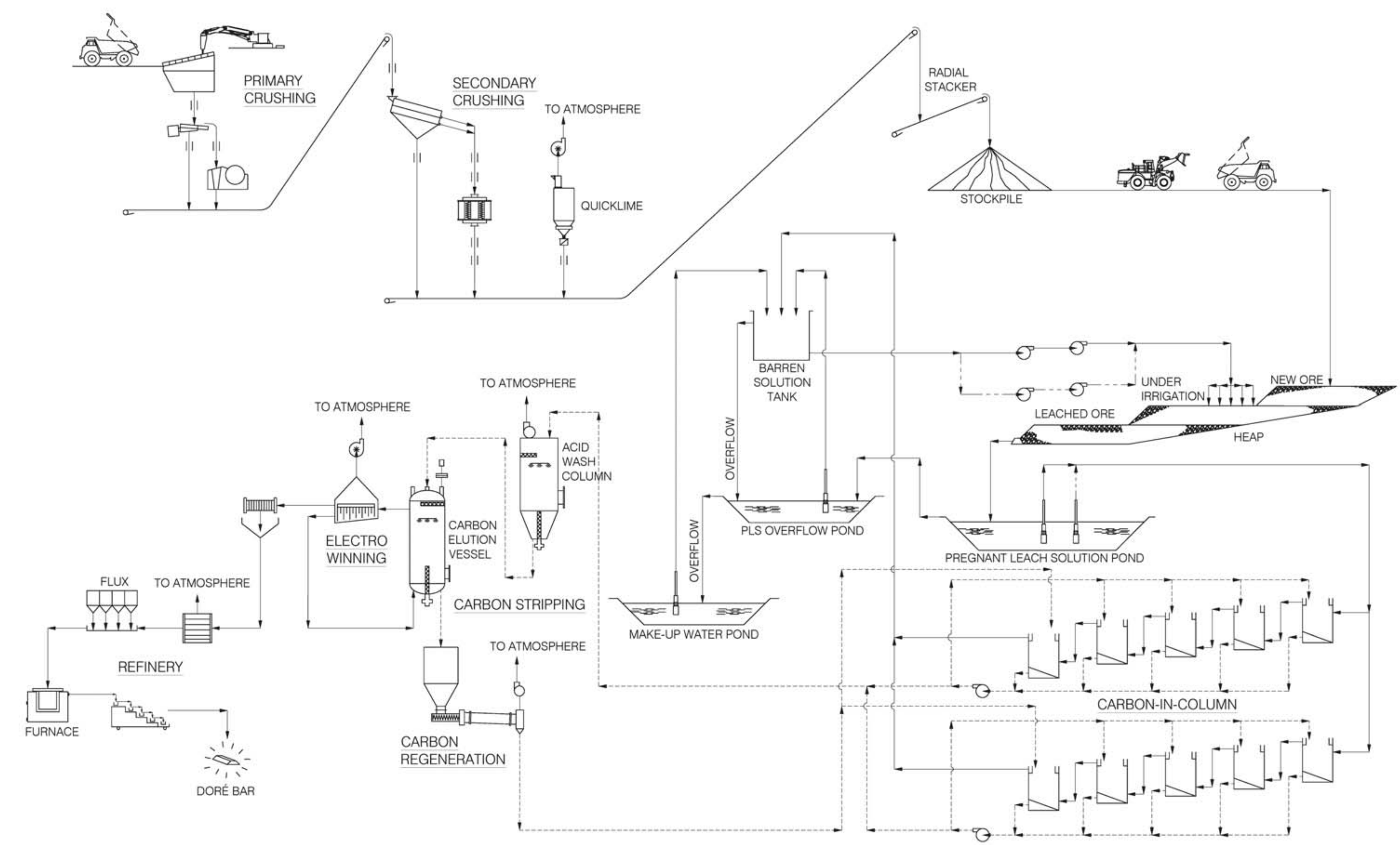
Typical Blast Pattern

Date: Sept. 2015	File: BlastPat.cdr	Figure 16-12

17 RECOVERY METHODS

SUMMARY

The flowsheet for the Öksüt Project was derived from metallurgical test work primarily performed by KCA. The test work results and interpretation are described in Section 13. The flowsheet is based on an 11,000 tpd heap leach operation. It includes primary crushing, screening and secondary crushing, heap stacking and cyanide leaching, carbon adsorption, carbon stripping and regeneration, electrowinning, and refining. The concepts and data presented in this section are taken from the Process Flowsheets and Process Design Criteria developed for the Project. The simplified process flowsheet is presented in Figure 17-1.



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Simplified Process Flowsheet		
Date: Sept. 2015	File: SimplifiedPFlow.cdr	Figure 17-1

ORE PROCESSING

PRIMARY CRUSHING

Run-of-mine ore will be delivered by 36-tonne haul trucks to the primary crusher. The ore will be dumped on the stationary grizzly installed over the 80-tonne truck dump hopper. Oversize rocks will be handled by a rock breaker. The ore will be withdrawn from the dump hopper via a 2.0 m wide x 4.0 m long grizzly feeder. The grizzly oversize will feed the 1.6 m x 2.0 m jaw crusher that will reduce the rock size to minus 150 mm prior to being conveyed by a 1.4 m wide x 95.5 m long belt conveyor to the secondary crushing circuit, along with the grizzly feeder undersize. A self-cleaning belt magnet will be installed over the conveyor belt feeding the secondary crusher building. A metal detector installed after the belt magnet will identify any remaining piece of metal and the conveyor will be stopped to allow manual removal by an operator.

SECONDARY CRUSHING

The product from the primary crushing circuit will feed a 2.4 m wide x 6.1 m long double-deck screen. The screen oversize will feed a 600 kW cone crusher while the screen undersize will report with the cone crusher product and will be transported by a 1.1 m wide x 50.7 m long belt conveyor to a radial stacker after quicklime has been added to the crushing circuit product. A 10,000 t capacity stockpile will be formed by the 1.1 mm wide x 39 m long stacker installation.

Dust collection units will be provided at the crusher discharge and transfer points in both crushing buildings and a dry fog system will be installed at the truck dump. If required, additional dust control measures could be implemented. A compressor will provide compressed air for process and instrumentation applications.

Key crushing circuit design criteria are presented in Table 17-1.

TABLE 17-1 KEY CRUSHING CIRCUIT DESIGN CRITERIA

Parameter	Units	Value
Throughput (average)	tpd	11,000
Availability	%	75
Throughput	t/h	611
Primary Crushing		
Crusher type	-	Jaw
Crusher size	m x m	1.5 x 2.0
F_{80}	mm	392
CSS	mm	175
P_{80}	mm	137
Secondary Crushing		
Arrangement	-	Open circuit with screened feed
Crusher type	-	Cone
Motor size	kW	600
Availability	%	75
CSS	mm	45
P_{80}	mm	38

HEAP STACKING

The crushed ore will be trucked from the crushing facility to the HLP. The HLP will be developed in three phases for an ultimate ore capacity of 40 Mt. Key heap stacking design criteria are presented in Table 17-2.

TABLE 17-2 KEY HEAP STACKING DESIGN CRITERIA

Parameter	Units	Value
Lift height	m	10
Ore under leach	t	547,945
Ore density	t/m^3	1.45
Area under leach	m^2	37,790

HEAP LEACHING

The heap will be irrigated with a diluted cyanide solution recirculated from the gold recovery circuit, via a network of piping covering the surface area under leach. The barren leach solution will be pumped from the barren tank at the ADR plant to the heap. The cyanide concentration will be adjusted and the pH will be controlled so that HCN gas formation is inhibited. The solution will be filtered to remove carbon fines prior to distribution over the heap to minimize emitter plugging. The solution will be pumped by means of two centrifugal pumps installed in series. The first pump will cover operation for the first three years of operation (end of phase 1) while the second pump will be required from year four and beyond.

The irrigation distribution piping will consist of a 300 mm diameter main header made of carbon steel from the barren pumps discharge to the heap perimeter followed by a high-density polyethylene (HDPE) header ending at the ore panels to be irrigated. Drip emitters will be used to provide irrigation. A typical panel piping arrangement will include a 300 mm diameter HDPE header starting from the main header and running for 190 m along the 250 m side of the panel. It is proposed to use two lengths of 300 mm diameter HDPE header in rotation for irrigation, one in operation and the other one available for installation on the next section to be irrigated. Four lateral pipes spaced at every 62.5 m will be branched from the header. Each lateral pipe will include a 150 mm butterfly valve, a pressure gauge and 75 m of 150 mm diameter HDPE pipe followed by 75 m of 100 mm diameter HDPE pipe. Emitter lines will be branched at every 500 mm on the pipes and emitters will be spaced at every 762 mm on the emitter lines.

Key heap leaching design criteria are presented in Table 17-3.

TABLE 17-3 KEY HEAP LEACHING DESIGN CRITERIA

Parameter	Units	Value
Irrigation method	-	Drip emitters
Solution application rate	$L/h/m^2$	12
Irrigation rate	m^3/h	453
Primary leach cycle	Days	50
Total leach cycle	Days	151
Irrigation solution pH	-	10.5 - 11.0
Irrigation solution concentration	mg NaCN/L	500
PLS grade – Au (average)	g/t Au	0.79
PLS grade – Ag (average)	g/t Ag	0.36
Recovery (life-of-mine) - Au	%	77
Recovery (life-of-mine) - Ag	%	14.0

The pregnant leach solution (PLS) will flow by gravity through a network of collection pipes at the base of the heap to the PLS pond prior to being pumped to the ADR plant for precious metals recovery.

ADSORPTION

The PLS will be reclaimed from the pregnant solution pond by a submersible pump at a rate varying between 300 and 500 m^3/h. The PLS flow rate is dependent on the footprint development of the heap and on climatic conditions. Since the flow rate will vary throughout the mine life, a two-line adsorption circuit was selected to add flexibility for the operation.

The PLS will be sampled and will pass through a trash screen prior to being distributed to the carbon-in-column (CIC) circuit. The solution will flow by gravity from the first to the fifth contactor of each line. Discharge from the last contactor will report to the carbon safety screen and will be sampled again prior to being pumped to the barren solution tank for recirculation on the heap. Oversize from the carbon safety screen will be recovered in a tote bin.

Each contactor will hold three tonnes of carbon. Periodically, based on the precious metals loading, carbon from the first contactor will be transferred to the elution circuit using the carbon transfer pump, prior to the content of the following contactors being pumped upstream to the previous unit. It is planned to transfer three tonnes of carbon on a daily basis, alternating between both lines.

Key adsorption circuit design criteria are presented in Table 17-4.

TABLE 17-4 KEY ADSORPTION CIRCUIT DESIGN CRITERIA

Parameter	Units	Value
Number of lines	Qty	2
Number of contactors per line	Qty	5
Flow rate (design)	m^3/h	500
Carbon hold-up per column	t	3
Carbon loading – Au (Average)	g Au/t	2,400
Carbon loading – Ag (Average)	g Ag/t	640

RECOVERY PLANT

ACID WASH

Carbon transferred from the first contactor of each adsorption line will be pumped to the acid wash vessel where, under contact with hydrochloric acid, scale build-up will be removed. The acid wash vessel will have a 3-tonne capacity and will be constructed of fiberglass reinforced plastic (FRP).

DESORPTION AND ELECTROWINNING

Once the acid wash step has been completed, carbon will be transferred to the elution column where it will be stripped from its precious metals content using the pressure Zadra process. The circuit comprises a 3-tonne capacity carbon elution column and a barren strip solution tank with pumps. The solution exiting the elution column is pumped to electrowinning. Barren solution from electrowinning will be returned to the barren strip solution tank for recycle to the elution column.

The electrowinning circuit will include two electrowinning cells.

Once the elution cycle is complete, part of the barren eluate solution (bleed) from the stripping circuit will be pumped to the barren solution tank. Fresh water, sodium hydroxide (NaOH), and sodium cyanide (NaCN) will be added to the barren strip solution tank to make-up for the lost barren strip solution.

CARBON REGENERATION

Once the elution cycle is complete, carbon will be pumped to the stripped carbon dewatering screen located in the carbon reactivation circuit. The screen undersize will flow by gravity to the carbon fines collection tank while the screen oversize will report to the reactivation kiln feed hopper. From there, the carbon will be fed by a screw feeder to the reactivation kiln. Reactivated carbon will leave the kiln and will be quenched using fresh water prior to being pumped to the reactivated carbon sizing screen. Screen oversize will be returned to carbon contactors No. 5 or 10, while screen undersize will report to the carbon fines collection tank.

Fresh carbon will enter the circuit via the agitated carbon pre-attrition tank, from bulk bags. Fresh water will be added to the tank, and the resulting slurry will be pumped to the reactivated carbon sizing screen.

The agitated carbon fines collection tank will receive undersize carbon. Carbon fines slurry from the tank will be pumped to the carbon fines filter. Filter cake from the carbon fines filter will be collected in the carbon fines bin and shipped off-site for further processing.

REFINING

Precious metals will be washed from the cathodes and the resulting sludge pumped to the gold sludge filter. The filter cake will be dried in an oven prior to being mixed with fluxes. The mixture will then be processed in the gold melting furnace, which will produce liquid metal and liquid slag; the liquid metal will be poured into doré bars while the solidified slag will be collected and shipped off-site for subsequent treatment.

Once in operation, dust and gas emissions from the ADR plant will be monitored for locally regulated metals and chemical compounds.

Key design criteria for acid wash, elution, carbon regeneration, and refining are presented Table 17-5.

TABLE 17-5 KEY CARBON ACID WASH, ELUTION AND REGENERATION CIRCUIT DESIGN CRITERIA

Parameter	Units	Value
Carbon plant capacity	t	3
Elution circuit type	-	High Pressure Zadra
No. Acid wash vessels	Qty	1
Acid wash vessel capacity	t	3
Acid wash cycle duration	h	3 - 4
No. Strip vessels	Qty	1
Strip vessel capacity	t	3
Strip cycle duration	h	8 - 12
Carbon kiln type	-	Horizontal, Electric
Carbon kiln capacity	kg/h	125
No. of electrowinning cells	Qty	2
Furnace capacity	L	57
Production – Au (maximum)	oz Au/day	485
Production – Au (average)	oz Au/day	233
Production – Ag (average)	oz Ag/day	97

HEAP LEACH FACILITIES

The heap leach process consists of stacking crushed ore on the leach pad in lifts and leaching each individual lift to extract the gold content. Barren leach solution (BLS) containing dilute sodium cyanide will be applied to the ore heap surface using drip emitters at a design application rate of 12 L/hr/m^2. Ten metre height (nominal) lifts will be leached in single 50 day primary leach cycles.

The applied solution will percolate through the ore to the drainage system above the pad liner, where it will be collected in a network of perforated drain pipes embedded within a 0.6 m thick (minimum) granular drain cover fill layer above the liner. The solutions will flow to the pond via gravity. PLS collected in the pregnant solution pond will be pumped to the ADR plant for processing to extract the gold.

SITE GRADING

The HLP site will be prepared for construction by first clearing, grubbing, and stripping of growth medium, which will be stockpiled at a location designated by OMAS. Grading will involve local cuts and fills of native soil and bedrock. Waste rock or other construction borrow materials may be required as a thin lift above local bedrock where exposed at the surface to create a uniform subgrade surface for placement of the low-permeable soil liner and this has been accounted for within the subgrade preparation. The ultimate foundation grading and phasing plan was developed to minimize cuts and fills while maintaining foundation/pad slopes suitable for a stable heap and allowing gravity drainage of surface water around the perimeter of Phase 1, Phase 2, and the ultimate Phase 3 configuration of the HLP. The foundation will be graded to provide a shallower slope along the soil-geomembrane interface below the toe area of the heap to promote stability. Slopes in the upper portion of the pad generally range from 5% to 25%, with some localized areas having slopes up to approximately 50%, or 2H:1V. These areas will be re-graded to 40%, or 2.5H:1V, which is the maximum slope that is constructible without employing modified design details and specifications for steeper slopes.

LEACH PAD

The leach pad will be constructed in three phases with approximate areas of 578,000 m^2, 212,000 m^2 and 155,000 m^2 for Phases 1, 2 and 3, respectively. The total cumulative pad area including Phases 1 through 3 will be 945,000 m^2.

The leach pad will have a composite liner system consisting of a 2 mm smooth linear low-density polyethylene (LLDPE) geomembrane underlain by a 0.5 m thick (minimum) compacted low-permeability soil layer with a maximum permeability of 1 x 10^{-9} m/s.

A drain pipe network will be constructed above the pad liner and will be embedded within a 0.6 m thick (minimum) drain cover fill layer, which will consist of free-draining, hard, and durable granular material. Solution and storm/snowmelt infiltration flows collected by the drain pipe network will gravity-drain to the process ponds.

ORE HEAP

The fully stacked (i.e., end of Phase 3) leach pad will have a nominal ore capacity of 40 Mt and a nominal maximum heap height of 80 m above the liner.

The ore is planned to be stacked in 10 m thick, horizontal lifts in three phases. A summary of the heap capacity by phase, number of lifts, elevations, and stacking schedules are provided in the following paragraphs.

Phase 1 was sized to contain nominally three years of ore production (i.e., 12 Mt) while maintaining a relatively flat configuration (i.e., ore shall be stacked against the foundation forming a wedge with a larger slope on the down gradient side and nearly no elevation difference between the top of ore and foundation grades on the upgradient side).

Phase 2 was also sized to contain nominally three years of additional ore production while maintaining a relatively flat slope and up to 24 Mt. For the case where the Project mine life is extended, a capacity of 40 Mt is required. Phase 2 was also sized to contain approximately 32 Mt of ore in the event additional reserves are not realized up to the 40 Mt design basis. For this contingency, ore may be stacked up to the maximum allowable heights. Sufficient top surface areas shall be maintained to facilitate leaching of the uppermost lifts.

Phase 3 has been sized for the total HLP storage capacity of 40 Mt, with ore stacked up to the maximum allowable heights. Sufficient top surface areas shall be maintained to facilitate leaching of the uppermost lifts.

SLOPE STABILITY

With the preferred liner system, the limit equilibrium stability analyses results showed a static FOS of 1.6, and a pseudo-static FOS of 1.0 for the Operating Basis Earthquake (OBE) considering a peak ground acceleration (PGA_{OBE}) of 0.20g, both of which meet or exceed the minimums established in the Project design criteria. The pseudo-static FOS for the Maximum Design Earthquake (MDE) was 0.8. A displacement analysis was performed to develop estimates of expected deflections during the MDE (PGA_{MDE}: 0.40 g) in accordance with standard industry practice.

For the alternate liner systems, the ore outslopes will need to be constructed at an overall slope of 3.5H:1V rather than 3.0H:1V in order to maintain adequate FOS. With the reduced slope, the static FOS for the GCL and HDPE alternatives were calculated to be 1.7 and 1.5, respectively. Both alternatives showed FOS below 1.0 for the OBE and MDE events.

Therefore, additional displacement analyses were performed to estimate deflections in accordance with standard industry practice.

Golder used the Makdisi and Seed (1978) dynamic deformation method to evaluate deformations in those cases where the FOS was less than unity. For the preferred liner system, the analysis shows that the MDE will result in approximately 24 cm of displacement at the base of the HLP, where the ore contacts the liner. For the alternative liner systems, the analysis shows the displacement at the base of the liner system will be approximately 10 cm to 35 cm for both alternatives during the OBE. The selected geomembranes have sufficient tensile properties to accommodate this level of displacement and are therefore considered acceptable.

At closure, the MDE is expected to induce displacements of up to approximately 1m, which is considered acceptable for a closed facility. However, in order to maintain acceptable deformations to the basal liner system, a buttress should be constructed along the toe of the HLP by pushing some ore from the crest down towards the toe.

SOLUTION COLLECTION SYSTEM DESIGN

The drainpipe network should maintain an order of magnitude of higher permeability in the drain cover fill compared to the overlying ore heap. The network is designed to drain the planned operational solution flow plus the additional flow from meteorological sources (determined from the probabilistic water balance), while maintaining acceptably low hydraulic heads on the pad's composite liner system (i.e., less than 0.6 m).

The drainpipe network system consists of tertiary, secondary, and primary pipes. Tertiary pipes will consist of 100 mm diameter perforated corrugated polyethylene (PCPE) tertiary pipes, which will be placed across the leach pad at 10 m spacing and laid to drain at slopes between 2% and 5%. Secondary pipes will consist of 200 mm to 450 mm PCPE pipes. Primary pipes will consist of 450 mm PCPE (within the leach pad) and 450 mm solid wall HDPE pipe (DR-21) outside of the leach pad limits (i.e., between the pad and the process ponds). The drainage system and flow capacity calculations are based on ultimate ore load and consider potential of crushing of the pipes.

The drainpipe network on the HLP will be covered with a 0.6 m thick (minimum) drain cover fill layer. The drain cover fill will have a maximum particle size of 38 mm, and will contain less than 5% passing a No. 200 US standard sieve. The drain cover fill will consist of crushed and/or screened low-grade ore, mine waste, and/or natural borrow material. The drain fill permeability requirement is $1x10^{-3}$ m/sec or greater under the 80 m maximum ore heap load to ensure drained heap conditions.

DRAINAGE CONTROLS

Surface water drainage will be controlled with 0.5 m depth "v" shaped diversion channels (3H:1V side slopes) to convey the design storm event (100-year frequency, 24-hour duration storm event) around the HLP and ADR plant area. Additionally, 1.5 m high temporary storm water management berms (2.5H:1V side slopes), which will be constructed upgradient (i.e., along the southern side) of the Phase 1 and Phase 2 HLP footprints to prevent surface water run-on to the HLP during expansion of the facility.

The HLP site is located on a natural plateau, and there are no active springs or seeps within the planned facility footprint. Therefore, no underdrain system is included in the HLP design. Additionally, currently available data indicates there are no active springs or seeps within the pond area footprint. However, one of the test pits excavated in the pond area footprint during Golder's field exploration program (TP-02) identified wet to saturated soil, indicative of perched groundwater conditions, and one active spring was present nearby flowing at low rates during the time of the initial site investigation, indicating the presence of shallow perched groundwater. An underdrain system may be required in the event that the excess perched groundwater does not quickly dissipate during initial excavation for the ponds.

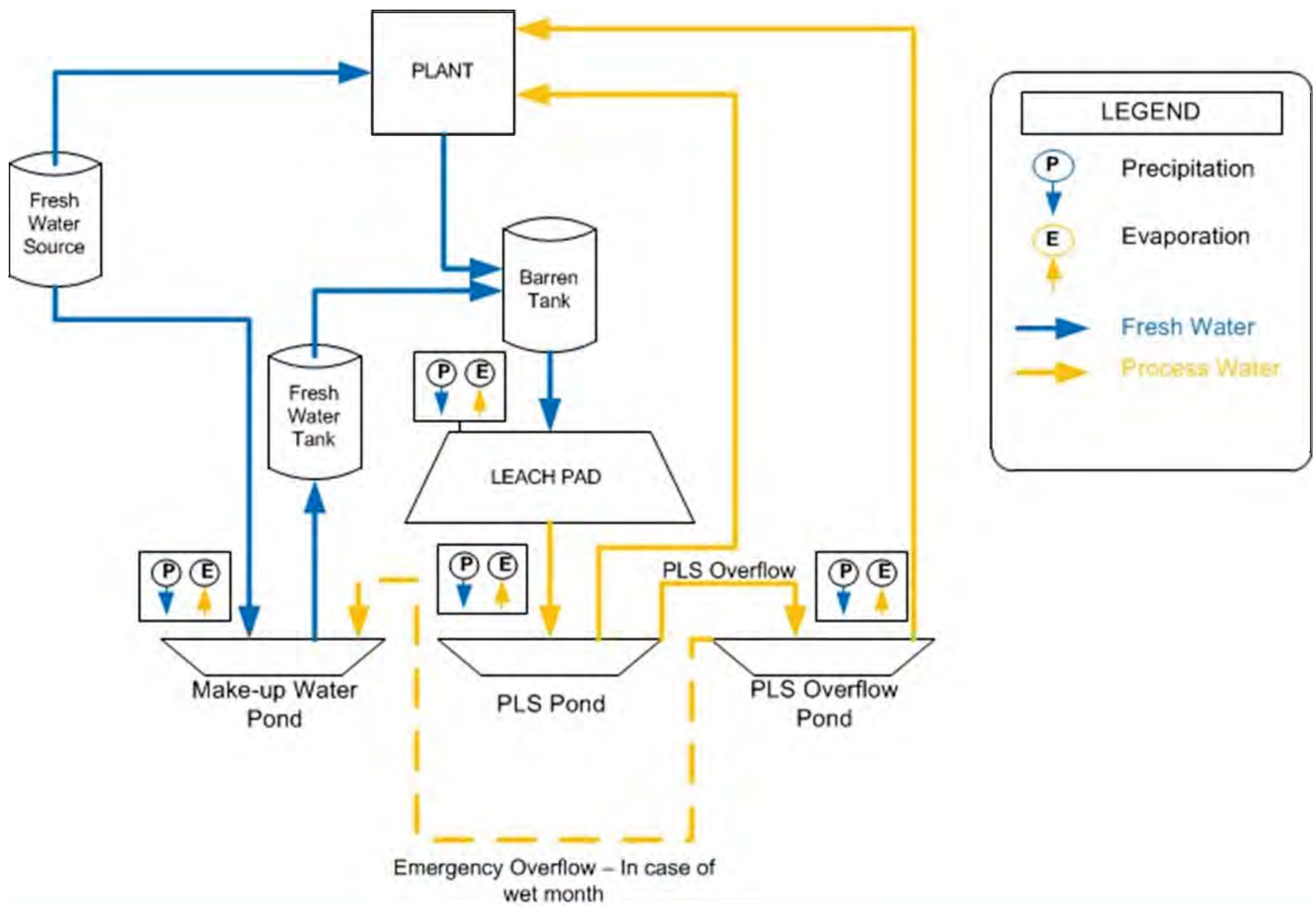
The leach pad will have 1.5 m high (minimum) perimeter berms to prevent applied solution and rainfall/snowmelt water within the pad from overflowing the pad. The solution and storm flows will be collected by a drain pipe network constructed above the pad liner and routed by gravity to the process pond.

WATER BALANCE AND SOLUTION MANAGEMENT

A probabilistic water balance model was developed for the HLP to simulate the performance of the facility. The objectives of the analysis were to evaluate the demand for makeup water from external sources and the volume of excess water generated during HLP operation (i.e., for use in event pond sizing).

The water balance model was developed using GoldSim software. The generalized flow chart is shown in Figure 17-2.

The water balance model was developed and applied to three different scenarios based on varying hydraulic conductivity (K_{sat}) of the ore. K_{sat} values of 0.1 cm/s, 1.0 cm/s, and 8.0 cm/s were assigned as scenarios 1, 2, and 3, respectively. Laboratory testing of the sample of post-column leached ore received from KCA under the design normal loads indicated a permeability of 8.0 cm/s. However, to account for field conditions, Golder believes that a more representative K_{sat} value may be within a range of 0.5 cm/s to 1.0 cm/s. Factors that may result in lowering the effective vertical hydraulic conductivity of the ore heap include the following: 1) surface trafficking (compaction) on top of previously placed ore, 2) breakdown of ore particles over time, 3) migration of fines particles, or 4) one lift (or a portion of one lift) placed with incrementally higher fines content.



Centerra Gold Inc.

Öksüt Gold Project, Turkey
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Generalized Water Balance Flow Chart

Date: Sept. 2015	File: WBalance.cdr	Figure 17-2

The model applies precipitation based on a stochastic climate model calibrated to local weather stations. A Monte-Carlo type analysis was used to calculate the required makeup water and pond storage capacity for the 10^{th}, 50^{th}, and 95^{th} percentile climatic conditions (i.e., dry, average, and wet). Based on the results, the PLS overflow pond was designed with a capacity of 83,800 m^3 (plus one metre freeboard), and the make-up water pond was designed with a capacity of 26,250 m^3 (plus 1m freeboard). These capacities will provide enough capacity to prevent discharges from the facility for scenarios 1 and 2. For scenario 3, the currently proposed ponds provide enough capacity for dry year conditions. For average and wet year conditions, the PLS overflow pond is predicted to overflow late in Phase 2 and Phase 3, respectively (but not during Phase 1). To provide contingency for this scenario in the planning and design, Golder has allowed for expansion of the PLS overflow pond during Phase 2, if the operational experience during Phase 1 shows that this is necessary.

The model assumes that the make-up water demand is supplied by water taken from the PLS overflow pond, fresh water source, and make-up water pond, respectively. Water balance results show that after the first year of operations, for the first two scenarios, a fresh water source with 125 m^3/hr flow rate will provide sufficient supply for the required barren flow rate (453 m^3/h). The model predicts that for scenario 3 (K_{sat} = 8 cm/sec), at certain times during the operational life of the HLP some of the barren flow rate (453 m^3/hr) may need to be supplied by an additional outside fresh water source. Alternately, fresh water could be pumped into an additional fresh water storage pond for use during times when fresh water demand exceeds 125 m^3/hr.

POND DESIGN

The location and dimensions of the ponds are shown on Drawing 18 of the Öksüt Mine Heap Leach Facility Design Report submitted by Golder Associates (Turkey) Ltd. Co. on February 11, 2015. The internal pond side slopes will be graded to 3H:1V, and the external slopes will be graded to 2.5H:1V. The PLS pond and PLS overflow pond will be double-lined, with a primary liner consisting of a 1.5 mm thick single sided textured (textured side up) HDPE geomembrane, and a secondary composite liner consisting of a 1.5 mm thick smooth HDPE geomembrane overlying a 500 mm thick low permeability soil layer. The make-up water pond will be single composite lined with a one 1.5 mm thick single sided textured (textured side up) HDPE geomembrane overlying a 500 mm thick low permeability soil layer. The double lined ponds will have a LCRS system located between the two liners. The LCRS will be provided by a highly transmissive HDPE geonet material that will drain to a gravel filled LCRS sump (one per pond). A smooth-walled, solid HDPE riser pipe will be installed within the LCRS sump and up the pond sideslopes to accommodate a submersible pump to remove solution that may report to the LCRS sump. No penetrations in the liner systems will be made since the riser pipe will be booted through the liners in the anchor trench, outside of the containment area. An overflow spillway will be constructed to allow transference of solutions from the PLS pond to the PLS overflow pond, and from the PLS overflow pond to the make-up water pond to provide capacity for surplus solutions after significant storm events or under up-set conditions.

REAGENTS

Reagents will be delivered to the site by transport truck and stored on-site, in the vicinity of the ADR plant. Quicklime and hydrochloric acid will be delivered in bulk and other reagents will be delivered in super bags or tote bins. Reagents will be stored adjacent to the ADR plant, on the west side of the building. This approach will be refined in the next Project phase, once discussions with suppliers have been initiated.

Cyanide will be stored in a closed building to which access will be restricted. Cyanide storage will meet the International Cyanide Management Code guidelines. Other reagents will be kept under a shelter that will include containment for liquid products.

Reagent packaging and consumptions are presented in Table 17-6.

TABLE 17-6 REAGENTS PACKAGING AND CONSUMPTION

Parameter	Packaging	Consumption (t/a)
Quicklime	30 t truck	7,600
Cyanide	1,000 kg bulk bag	1,600
Hydrated lime	850 kg bulk bag	1,000
Carbon	500 kg bulk bag	80
Sodium hydroxide	1,000 kg bulk bag	430
Hydrochloric acid	18 t truck	440
Anti-scalant	1,000 kg tote bin	60
Refining fluxes	25 kg bag	1.6

WATER

Fresh water will be pumped from wells located in the vicinity of the town of Epçe to the fresh water tank. The tank location was selected so that water can be distributed by gravity to the crushing plant, the ADR building, the administration complex, and the truck shop. The bottom part of the tank (565 m^3) will serve as a reserve for fire protection. This will allow a water flow of 280 m^3/h during two hours, in accordance with the National Fire Protection Association (NFPA) regulation. The upper part of the tank (145 m^3) will be dedicated to the ADR plant usage, mainly for carbon handling and for reagent preparation.

A make-up water pond, located in the vicinity of the ADR plant, will be filled by trucks from other surface ponds. Make-up water to the barren solution tank will be pumped from the make-up water pond. In the event that the water inventory in the pond is insufficient, fresh water make-up will be used. The fresh water consumption at the ADR plant is estimated at approximately 100 m^3/day (1.2 L/s).

POWER REQUIREMENTS

The estimated total connected load is 7.84 MW with an average power draw of 4.73 MW.

GENERAL ARRANGEMENTS

CRUSHING FACILITIES

The primary and the secondary crushers will be located in two distinct buildings distanced by approximately 70 m. Both crushing buildings were located so as to take advantage of the contours of the local terrain and minimize cut and fill requirements. A 35 t crane will be installed in the primary crushing building and a 20 t crane in the secondary crushing building.

A radial stacker will transport the crushed ore and form a kidney-shaped stockpile. This will allow safe reclaiming of the ore on the side opposite to where the ore is discharged.

ADR BUILDING

The ADR building will house the CIC, carbon elution-regeneration and electrowinning circuits as well as reagent preparation and distribution installations. Also included in the building are a refinery, a mechanical shop, a control room and offices for plant personnel. The CIC circuit and the barren solution tank will be located outdoors while the remaining facilities will be located inside a cladded and heated building. The layout will allow solution from each circuit to be recovered in a dedicated containment area so as to avoid mixing fluids that are not compatible. Each containment area will be serviced by a sump pump.

The ADR plant will be serviced by three cranes located in the carbon elution-regeneration and reagents area (2 t), in the refinery (2 t), and in the mechanical shop (10 t). Two hoist beams will service the CIC circuit.

The barren solution tank will be located so as to allow expansion of the CIC circuit to the south, should it be required over the mine life. The plant is located 15 m away from the solution ponds so as to allow traffic around the building. Three sides of the building will be accessed on a regular basis (mechanical shop, reagents area, and refinery).

A security closed-circuit television (CCTV) system will allow general surveillance for the process buildings and the gold recovery area. The electrowinning and refinery installations will be located in a high security sector of the plant to which access will be restricted. Process areas such as the grizzly feeder, the crushers, and the vibrating screen will be monitored via a process CCTV system.

18 PROJECT INFRASTRUCTURE

SUMMARY

The Öksüt Project requires the support of a well-established and maintained infrastructure especially considering not only the volume of reagent used and transported over the public roads, but also climatic challenges.

The proposed Project site layout is shown in Figure 16-7.

Waste dump details are found in Section 16 (Waste Dump Design and Stockpile Strategy) and 17 (Heap Leach Pad). The Project is not expected to generate any tailings.

ACCESS ROADS

The Project area is located at an elevation of approximately 1,800 m, in an area of high relief with mountains and valleys. The Project area is surrounded by three urban areas, the village of Öksüt four kilometres south-southeast, the city of Develi approximately 10 km north-northwest, and the village of Epçe approximately 10 km east.

The existing roads in the region are located in the adjacent valleys and are generally paved and well maintained. Existing roads are expected to be utilized as much as possible and extended to the Project area with a cut-off from the national road network. The main access route selected is the “east access” via Epçe. This road will be approximately 8.18 km long. The total length of site roads will be approximately 6 km.

A bypass road will be constructed around the towns of Gömedi and Yazıbaşı. Its route will be on the west side of the towns, and will be approximately 6.7 km long. It will be constructed using the same standards as the existing public roads. It may also be necessary for the Project to upgrade the existing road that connects the towns of Gömedi and Epçe. This will be further investigated during the detailed engineering phase of the Project.

POWER SUPPLY

Power to the site will be supplied from a 31.5 kV electrical network through a dedicated 28.5 km overhead line coming from the Sendrimeke substation. The overhead line will consist of two segments, from Sendrimeke to Develi (18 km) and from Develi to the Project site (10.5 km), and will terminate at the site substation where the power provider will have its metering station. The distribution voltage will be regulated using a transformer with an under-load tap changer (ULTC) at the site substation and then distributed throughout the site using 31.5 kV overhead lines. A capacitor bank will also be installed at the site substation to maintain a unity power factor.

At each consumer location, voltage will be stepped down to 400 V using outdoor transformers, either pole mounted or on the ground. The 400 V will then be distributed to the process and auxiliary loads using indoor switchgear units located in the different electrical rooms.

For buildings, power will be supplied from three substations, two of which were sized for loads of buildings only. For the reagent storage area, a substation located in the ADR area will be used.

WATER HANDLING

WATER SUPPLY

Water supply investigations were based on a design value of 35 L/s provided by OMAS as the Project fresh water requirement. The Epçe agglomerate aquifer was chosen as the groundwater source for the Project.

The field investigations conducted between November 2014 and February 2015 at two selected locations in the Epçe area included well drilling, instrumentation, and aquifer testing. One test well and two observation wells were completed at each location. Based on the results of the two long-term tests, the two pumping wells (E1TW1 and E2TW1) are expected to produce a combined volume of over 60 L/s.

The fresh water supply pipeline will deliver water from the two wells located near the village of Epçe to the fresh water supply tank at the mine site. The two wells have been specified to supply 22 L/s (E1TW1) and 35 L/s (E2TW1), respectively, to a sump located at the main pump station. Submersible pumps will pump water from the wells through 150 mm DR17 HDPE pipelines running from the well to the main pump station. At the main pump station, two vertical turbine pumps (one operating and one standby) will be mounted in a concrete sump, and used to deliver the water to the mine site. The fresh water supply pipeline from the main pumping station to the mine site will be approximately 9.3 km long, and constructed of 150 mm standard weight carbon steel.

Operation of the fresh water supply pipeline is assumed to be constant, shut down only for regular scheduled mine maintenance. There are several areas of optimization, including the optimal configuration for the well pump systems, proposed pipeline routes, and sequencing of the pipeline and road construction, that should be investigated during the next level of engineering design.

For Project buildings, raw water will be supplied from the raw water tank which is located at an elevated hill. Prior to distribution to the buildings, raw water will be treated in a package treatment unit via cartridge filters and UV light. Potable water will be supplied to the kitchenette, showers, and washbasins. For toilets, raw water will be distributed. A 20 m^3 treated (potable) water tank is allocated near the treatment plant in order to provide reservoir for buildings as well as to provide sufficient head and reservoir to pump treated water to a mobile tanker for distribution to remote areas.

Small capacity tanks are designed for buildings in remote areas for short term storage.

SITE WATER MANAGEMENT

The water at the Project site can be broadly classified into “non-contact” and “contact” water. Non-contact water is surface runoff and groundwater which is not impacted by mining operations. Non-contact water is typically diverted around the mine facilities. Contact water is surface runoff and groundwater that is generated, or has been in contact with, mine facilities. Contact water typically requires management, sedimentation, and/or treatment prior to discharge to the environment. Golder has prepared a site-wide water management plan (Golder, 2014c). All contact water was considered by Golder to be Potentially Acid Generating

(PAG) based on the current understanding of the geochemical characteristics of the rocks at the Project site.

The proposed site-wide water management plan includes a non-contact water diversion system and a contact water collection system.

The proposed non-contact diversion system would be composed of a network of diversion channels around the waste dump and the pits collecting water from undisturbed catchments located upstream of the mine facilities. The proposed contact water collection system would be composed of a network of channels, underdrains, and ponds to collect contact water runoff and seepage from the waste dump, stockpiles, and pits.

The water management plan and water infrastructure design were based on the climate design parameters described in Section 5, integrated as required (Golder, 2014a). The general approach is to design the contact and non-contact water diversion system to safely accommodate the 1-in-100 year peak flow, including snowmelt contribution. Contact water pond storage will be sized to contain runoff from the 1-in-100 year, 24 hour storm event and 24 hour average seepage.

SEWAGE WATER TREATMENT

Due to dispersed settlement of facilities, a hybrid sewage network system has been designed to provide the most efficient sewage treatment system. In this system, an underground sewage network is installed to serve the most populated Project areas and connected to a nearby sewage treatment plant.

For remote locations such as ADR building and gatehouse/public relations (PR) building where underground sewage system is not feasible to be extended, septic tanks are designed for short term storage of the liquid waste. These tanks will be emptied out via mobile trucks and taken to the main sewage treatment plant, on regular intervals. The sewage treatment plant is designed with an equalization (holding) tank to handle both continuous liquid waste coming from the sewage system and also intermittent bulk loads of liquid waste coming through mobile trucks.

Sludge generated in the sewage treatment plant will be taken to designated landfill areas, at regular intervals. The treated effluent from the sewage treatment plant will be stored in a storage tank and pumped out to the point of consumption. Chemical wastewater produced in the chemical and metallurgical analysis laboratories will be treated by neutralization and disposed into the main sewage network.

PROJECT BUILDINGS

Project buildings will include:

- Process buildings, including
  - Primary and secondary crusher buildings
  - Crushing area electrical room
  - ADR Plant

- Support buildings, including
  - Administration building
  - Dry building
  - Dining hall
  - Laboratory
  - Heating centre and Liquefied Natural Gas storage
  - Gatehouse/PR building
  - Weigh scale house
  - Warehouse
  - Fuel storage
  - Firewater pump house
  - Cyanide storage
  - Reagent storage

FUEL DELIVERY AND STORAGE SYSTEMS

Diesel fuel will be required for process plant and mining operations. The most significant diesel users are as follows:

- Mining equipment including haul trucks;
- Light vehicles;

- Generators.

Monthly diesel consumption is expected to be approximately 500,000 L.

Diesel will be supplied from one of the major oil companies in Turkey, most probably with a long-term purchase and dealership agreement.

Diesel will be delivered to the plant in trucks and stored in a 250 m^3 tank equipped with pumps for unloading diesel from trucks. This main storage tank has the storage capacity for two weeks of consumption. A 20 m^3 day tank will be located next to the storage tank, equipped with pumps for loading diesel service trucks and also a dispenser to service light vehicles.

Vehicles that routinely pass the haul road during operations will refuel at that facility as required. Machinery operating only in the mining areas will be regularly refuelled by service and refuelling trucks.

SITE SERVICES

Site services will include security personnel and equipment, a first aid station, and telephone and internet communications. Employees are expected to live in the city of Develi or nearby villages. Special shuttle buses with adequate seating capacity will handle the transportation between the residential areas and the Project site.

19 MARKET STUDIES AND CONTRACTS

Applicable Turkish laws provide that the Turkish Government State royalty is reduced by 50% on doré refined within Turkey. Accordingly, Centerra expects to make arrangements to refine doré in Turkey and to sell bullion on the open market.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

INTRODUCTION

The environmental and social studies for the Öksüt Project were initiated in June 2013. The Environmental Impact Assessment (EIA) for the Öksüt Project was commenced in accordance with the Turkish EIA regulation and practices on April 14, 2014. A draft EIA Report was submitted to the Ministry of Environment and Urban Planning (MEUP) on July 7, 2015 (SRK Danişmanlik ve Mühendislik A.Ş., 2015). The EIA permitting process was continuing at the date of this Technical Report.

PERMITTING

The Turkish EIA regulation is similar to the European Union (EU) EIA Directive in structure and procedures. The Turkish EIA permitting process involves a scoping phase inherent in the official permitting process. An EIA application describing the Öksüt Project's environment and potential environmental impacts was submitted to the MEUP in June 2014 to initiate the EIA permitting process. Based on the EIA application and the feedback from the official public hearing, the MEUP in coordination with other regulatory agencies provided a mandatory Terms of Reference (ToR) for the Öksüt EIA studies and EIA Report. The EIA application was submitted to the Ministry of Environment and Urbanization on July 7, 2014 by SRK and a public hearing meeting was held on August 7, 2014. The draft EIA Report was submitted on April 14, 2015 in accordance with the ToR. Following their review, the Review and Assessment Committee provided comments and requested certain revisions to the EIA Report in June 2015. OMAS expects that further review by the Review and Assessment Committee will be undertaken and public hearing and consultations will be held prior to final approval of the EIA Report.

Following the formal approval of the EIA Report, OMAS will apply for all necessary construction and operation permits as described in Section 4.

ENVIRONMENTAL STUDIES

BASELINE STUDIES

During the early exploration program (2008-2012) conducted by Stratex, the environmental baseline data collection was carried out by Golder. These baseline studies were preliminary and limited in nature as the Project features and the Project site were not yet defined. During the feasibility phase and the EIA permitting process (2012-2015), comprehensive baseline environmental data was collected by SRK and involved the following:

- Air quality monitoring (gaseous and particulates) which determined that overall air quality in the Project area is at typical rural levels and better than established air quality limits.

- Noise measurement which determined that noise levels in the Project area are at typical rural levels.

- On-site meteorological monitoring.

- Ground and surface water quality monitoring.

- Hydrogeological (groundwater) measurements and monitoring which found five different hydrogeological units, including three such units where groundwater wells were drilled for water supply (near Epçe village). The oldest one, which has low permeability, is mid-Miocene age andesites, which are a member of the DVC volcanites and are in contact with a fault system with a north-south direction, and with upper Miocene agglomerate (a member of the Sarıca volcanics) which has high hydraulic conductivity and is considered as the main aquifer unit.

- Hydrological (surface water) monitoring which began in July 2013 and continued for four quarters involved sampling from groundwater wells at a total of 13 surface water points and 20 groundwater points. Water quality assessments for groundwater and surface water were carried out with regard to Turkish and EU water quality regulations.

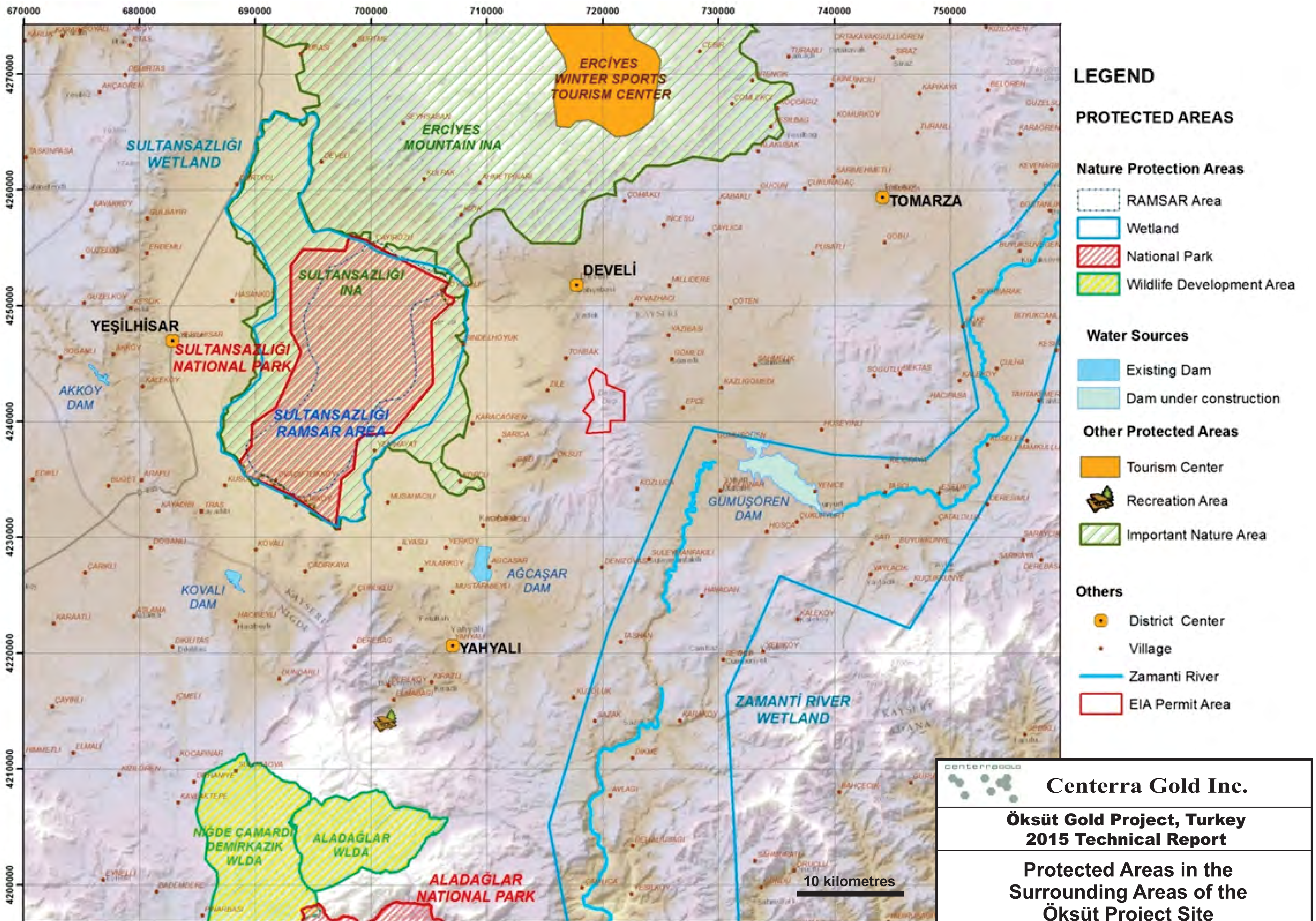
- Soil and sediment quality measurements which found that approximately 40% of the land within the Öksüt Project site consists of lime-free brown forest soil group, and 60% of bare rocky lands.

- Field flora and fauna surveys which identified no protected flora species but found a number of protected reptiles, amphibians, and birds in the Project area.

- Protected areas within the Project area include part of the Sultan Sazlığı Wetland Area (Ramsar Site) and the Zamantı River Wetland (Figure 20-1) as well as the Erciyes Winter Sports Tourism Centre. Other than these, the Project area consists of bare rock, pasture land, and shrubbery.

- Ore and waste geochemical characterization tests (Acid Rock Drainage/Metal Leaching, or ARD/ML) were carried out to determine ARD/ML characteristics of the

Öksüt mine lithologies, assess the potential ARD/ML related impacts in the post-closure phase, and identify mitigation measures relevant to open pits, waste rock dump, and heap leach facility. For this purpose, mineralogical analysis, acid-base accounting, total rock and trace element analysis, contact leach testing, and kinetic analyses were carried out on samples from both the Keltepe and Güneytepe deposit areas. For the closure phase, predictive geochemical modelling studies were carried out for the waste rock dump area and the heap leach pad. The risk of occurrence of ARD was identified for the waste rock dump and measures were determined that will minimize contact with the surface water and ensure collection of the leachates in this area.



Centerra Gold Inc.

Öksüt Gold Project, Turkey
2015 Technical Report

Protected Areas in the Surrounding Areas of the Öksüt Project Site

Date: Sept. 2015	File: Protected.cdr	Figure 20-1

SOCIAL STUDIES

A Preliminary Social Baseline and Impact Assessment was prepared in January 2014 and this is being updated by a comprehensive social baseline survey (undertaken in December 2014) which is being used to prepare an international-standard Environmental and Social Impact Assessment (ESIA) for the Project. Detailed information on households and communities within the vicinity of the Project is under preparation as part of this process.

In addition, the EIA included an assessment of economic and social issues at a regional and provincial level, and a summary of public participation undertaken, to meet EIA requirements.

The social impact assessment for the Project is focused on the neighbourhoods adjacent to the Project area and also includes the regional centre of Develi.

The social baseline survey that will provide the key socio-economic input for the ESIA covers the following topics:

- Demography
- Social Structures
- Economy
- Employment & Livelihoods
- Social Infrastructure
- Land Use and Natural Resources
- Community Health, Safety and Security
- Cultural Heritage

As part of the Turkish EIA and for the ESIA, OMAS has identified key stakeholders including:

- Direct stakeholders – local residents and community representatives within the immediate vicinity of the Project;

- Indirect stakeholders – provincial and regional authorities and other interested parties.

Ten local communities were identified in the Develi District that were considered likely to be directly influenced by OMAS activities. These include Develi, Yukari Develi, Gazi, Öksüt, Sarıca, Tombak, Zile, Epçe, Gömedi, and Yazıbaşı.

In the course of the EIA, a community participation meeting was held at the Old Municipal Wedding Hall in Zile on August 7, 2014 following national and local advertisement and was attended by approximately 60 participants, including representatives of the Ministry of Environment and Urban Planning, the Provincial Environment and Urban Planning Directorate of the Office of Governor of Kayseri, local business representatives, and local residents. As part of the socio-economic baseline survey, and in support of ongoing community engagement, a series of household surveys were undertaken in Öksüt and Zile in December 2014 using detailed registers of all residents.

A community relations team has been established by OMAS and an information office has been opened in Develi. A stakeholder engagement plan is in place to ensure that there is regular ongoing engagement with direct and indirect stakeholders, to inform them of Project plans and developments on an ongoing basis and gather any complaints or feedback. A stakeholder engagement register records all OMAS interactions with stakeholders.

OMAS recognizes the short mine life of the Project and is working to ensure that the Project will leave behind enduring and self-sustaining benefits for local communities. For this purpose, OMAS has developed a Social Policy.

To support implementation of the Social Policy and ensure that Project benefits are maximized and impacts minimized, OMAS is developing an environmental and social management system. In relation to community relations issues, this includes management plans and strategies covering:

- Community Development
- Stakeholder Engagement
- Pastureland Management and Livelihood
- Cultural Heritage
- Land Acquisition and Involuntary Resettlement
- Social Closure
- Local Procurement and Contracting
- Local Employment
- Community Health, Safety and Security
- Contractor Management

- Grievance Management

CONCEPTUAL CLOSURE PLAN

Mine closure and rehabilitation in Turkey is mainly regulated through the Regulation on Reclamation of Mine Sites. The regulation requires preparation of a mine closure report as part of the EIA permit. Further to this, the Landfill Regulation is applicable to mining processed wastes (i.e. excluding waste rock dumps) regarding the cover requirements during mine closure. There is also a draft regulation on Mining Waste Management that is expected to be similar to the European Union Directives on the same topic and is likely to have requirements on waste rock dumps. A conceptual mine closure discussion is provided below for the main mine units.

Closure costs have been estimated at $27M for the Project.

OPEN PITS

Hydrogeology investigations have indicated that the groundwater table would be below the open pit bottoms. Therefore, pit lake formation at closure is considered unlikely. Any surface water run-off into the open pits would report to the Acısu spring, which has already low pH levels and high dissolved metals. Therefore, no special measures would need to be taken at closure to mitigate the ARD/ML generation from the open pit shells.

At closure, the open pits are expected to be bunded off with inert waste rock. The bunding would provide protection against people, animal, vehicles etc. falling into the pit by accident. Around the rock bunding there would be a chain fence with warning signs. The open pits are located within the forestry land. After closure, the open pit area would be returned to the Department of Forestry.

HEAP LEACH PAD

Following waste acceptance criteria testing, waste characterization determined that the Öksüt heap leach spent ore is classified as Category-1 (under the Turkish Landfill Regulation) due to elevated As levels in the test leachate. According to the Landfill Regulation, Category-1 processed mining waste facilities must be sealed with a closure cover comprising the following features:

- The mineral surface sealing should be at least 25 cm thick applied in a double layer;
- Geosynthetics may be used to reinforce the mineral sealing;
- The drainage layer should be at least 0.5 m thick with K ≥ 1.0 x 10^{-4} m/s permeability; and
- Top soil cover should be at least 0.5 m thick and suitable for the growth of chosen vegetation.

Any drainage emanating from the HLP during or after closure may also require management, depending on the chemistry of the solutions. In similar climates in the United States and Australia, both passive and active management measures have proven effective in the management of post-closure heap leach drainage. These range from passive evaporation or evapotranspiration systems, to active water treatment. In the closure phase, the heap leach pad is planned to be rinsed. The rinsate will be sent to the processing plant. Since the lime and hence alkalinity will be washed out from the heap during the rinsing process, acid generation may occur due to sulphur content of the ore, as observed in the lab tests. For this reason, uncontrolled discharge from the heap leaching after the closure should be prevented. The rinsate during the closure period should be treated as process water and not discharged.

The HLP slopes will be adjusted to 3:1 at closure for long-term stability.

WASTE ROCK DUMP

Geochemical testing indicated that most, if not all, of the waste rock is PAG and that development of ARD could occur when the rock is exposed to weathering. It is expected that the leachate at the waste rock dump area will have acidic pH. Furthermore, there is risk that Cd, Cr, Cu, Fe and Zn concentrations in the leachate would exceed the discharge limit values if no measure is taken. For this reason, measures that will minimize contact with the surface water and ensure collection of the leachates must be taken at the waste rock dump area during the closure phase. In order to minimize the effects of ARD/ML on the downstream water quality during the post-closure period, the following closure methods will need to be taken:

- The temporary diversion channels in the upstream of the waste rock dump should be preserved until the end of the rehabilitation period to minimize the entry of water into the waste rock dump.

- A closure cover layer should be formed on the waste rock dump throughout the rehabilitation period. The closure cover would divert the precipitation and subsequent surface flow away from the dump preventing contamination, as well as minimize percolation of precipitation into the dump.

- The upper layer of the cover should consist of top soil to allow rehabilitation and plantation. Below the soil layer, other mineral and artificial cover layers (drainage layer, mineral layer, and artificial layer) should be placed to minimize infiltration. A closure cover similar to the closure cover of the heap leach pad (Class I) should be applied at the waste rock dump.

- Following the completion of the closure cover, the leachate collection pond in the downstream should be re-purposed. The leachate collection pond should be turned into an evaporation cell with impermeable bottom so that any seepage from the dump after the closure would be collected and evaporated.

The WRD slopes will be adjusted to 3:1 at closure for long-term stability.

GENERAL CONSIDERATIONS

With the exception of the open pits, waste rock dump, and heap leach pad, all mining structures will be removed at closure. Any salvageable equipment will be sold as second hand equipment. Those that cannot be sold, will be recycled as scrap. The footprints of the mining units will be scraped and leveled. Where possible, the land surface will be re-vegetated. However, it should be noted that the site consists of either rocky areas (no top soil) or very thin top soil (0 – 20 cm). Any top soil required for reclamation beyond that which is available on site would need to be imported.

The current land use of the Project area is pastureland used for animal husbandry and grazing. After closure, the land will have to return to its original use. This will need to be established in discussions with the regulatory agencies.

21 CAPITAL AND OPERATING COSTS

ASSUMPTIONS

The following material assumptions have been used in the life of mine plans, estimates of operating and capital costs, and Mineral Reserve and Mineral Resource estimates:

- A gold price of $1,250 per ounce;
- An exchange rate: US$1: 2.5 TL; and
- Electricity cost of $0.14 per kilowatt hour.

CAPITAL COST ESTIMATE

All pre-production expenditures and construction capital have been assumed to occur in Years -2 and -1 (2015-2016) and Q1 2017, with mining commencing in mid-2016 and, subsequent to commissioning, full production processing operations commencing in Q2 2017. Total initial direct and indirect capital costs and pre-production operating costs are estimated to be $220.8M. This total includes all applicable Owner's costs, HLP, ADR, infrastructure, engineering, mining, working capital, and project management costs, with a contingency of 15% upon commencement of construction. The estimates are found in Table 21-1.

TABLE 21-1 PRE-PRODUCTION EXPENDITURES

Item	Total ($ x 1000)
Direct Costs	
Mining, including HLP Construction	51,196
Utilities	16,526
Infrastructure	21,990
Processing including Crushing and ADR	27,879
Water Handling	8,254
Working Capital	10,222
Total Direct Costs	**136,067**
Indirect Costs	
Owner's Costs	17,592
Permitting/Land Acquisition	8,154
Construction/Project Management	33,822
Contingency (15%)	25,150
Total Indirect Costs	**84,718**
Total Pre-production Expenditures	**220,785**

DIRECT COSTS

MINING AND MINE CONSTRUCTION COSTS

The costs associated with mining and mine construction include mining costs through to production and the construction of the HLP the Pregnant Solution Pond, the Overflow Pond, as well as the Make-Up Water pond. Also included within this section is the Barren Solution irrigation and distribution piping with the associated costs. These costs are further detailed in Table 21-2.

TABLE 21-2 MINING AND MINE CONSTRUCTION CAPITAL COST

Item	Total ($ x 1000)
Mining	17,727
Explosives Magazine	250
Stockpiles Site Preparation	225
Pits and Shops Site Preparation	1,000
Heap Leach Pad Construction	20,487
Heap Leach Pad PLS Collection	7,265
PLS Pond	866
Barren Solution Distribution	2,425
Overflow/Makeup Water Ponds	951
Total Mine Construction	**51,196**

UTILITIES AND INFRASTRUCTURE CAPITAL COSTS

The utilities section includes all of the capital required to deliver electrical power to the site. There is a provision for the tie-in to the Sendrimeke substation, as well as the power line to the site. This line is estimated to run 28 km from the tie point to the site's main substation. The costs for the tie point hardware and installation as well as the construction of the power line will be recovered from TEIAŞ, the Turkish Power Transmission Company, during the first years of mine production, through a reduction in the usage charges incurred during operation. The final contract has not been negotiated, although OMAS has received the official approval for proceeding with the construction.

The infrastructure portion includes all of the roads, ancillary buildings, water and sewage systems, etc. The capital allocation for the roads include the bypass, estimated at 6.8 km, around the towns of Gömedi and Yasi Basi, as well as the upgrade of the Gömedi -Epçe public road, to bring it up to the standards required for heavy transports, if necessary.

The site haul roads are included in the Roads-On site section of Table 21-3. The cost is based on cutting and filling approximately 1.5M tonnes of material, thus precluding the need for using waste rock as haul road construction material. This will allow for the construction of the haul roads before the mine pre-stripping is concluded.

TABLE 21-3 UTILITIES AND INFRASTRUCTURE CAPITAL COST SUMMARY

Item	Total ($ x 1000)
Electrical Substation	1,896
Sendrimeke Tie and Power Line	10,997
Site Distribution Lines	1,199
Back Up Generators	200
Site-wide Fibre Optic Line/IT	2,234
Subtotal Utilities	**16,526**
Site Preparation	684
Roads – Off site	3,450
Roads – On site	13,873
Main Administration Campus	2,635
Security Gate/Fencing	425
Truck Shop/Warehouse/Fuel Storage	399
Potable/Sewage Water Handling	524
Subtotal Infrastructure	**21,990**
TOTAL	**38,516**

PROCESSING AND WATER HANDLING

The processing section includes the primary and secondary crushers, as well as the reagent storage facilities. However, the main component of this section is the complete construction of the ADR plant. The intention at this time is to issue a contract for the complete designing and building of this plant. Preliminary discussions were held with vendors who have the capability to perform this level of project, including the attaining of functional specification targets relating to the production of doré bars. The processing capital cost summary is shown in Table 21-4.

The water handling section includes the construction of the raw water pumping and piping system from the wells in Epçe to the fresh water tank located east of the HLP. The tank will be located at an elevation that will allow for the gravity distribution of both raw water and firewater. The water handling capital cost summary is shown in Table 21-5.

TABLE 21-4 PROCESSING CAPITAL COST SUMMARY

Item	Total ($ x 1000)
Primary and Secondary Crushing	12,185
Ore Handling and Stacking	2,385
ADR Process	5,673
Cyanide Handling	556
Reagent Handling	1,191
ADR Building and Infrastructure	5,889
TOTAL	**27,879**

TABLE 21-5 WATER HANDLING CAPITAL COST SUMMARY

Item	Total ($ x 1000)
Raw Water Pumping and Pipeline	2,622
Raw Water Storage and Handling	1,446
Surface Water Collection and Storage	3,461
Fire/Sewage Water Handling	725
TOTAL	**8,254**

INDIRECT COSTS

OWNER'S COSTS

The Owner's Cost portion of the capital cost primarily includes the operation of the Ankara and Develi offices for the duration of the construction phase of the Project. The Ankara office costs for OMAS will become an OMAS corporate operating expense once Project operations commence.

There is an allowance in this section for the construction and pre-commissioning insurance, as well as for light vehicles, including the firetruck and the ambulance. The owner's capital cost summary is shown in Table 21-6.

TABLE 21-6 OWNER'S CAPITAL COST SUMMARY

Item	Total ($ x 1000)
OMAS offices –(2015-2017)	15,794
Surveying	500
Ground Transportation	1,298
TOTAL	**17,592**

PERMITTING AND LAND ACQUISITION CAPITAL COSTS

The Permitting section includes the mine operating licence, the rental fees on the forestry land, and the fees for the pasture land for all of 2016, as well as the Q1 2017. These fees are ongoing, and will be accounted for in the Operating Expenses for the remainder of the mine life.

There is a one time deposit of $1.4M in 2016 for the use of the pasture land. This is returnable after the mine closure.

These costs are being further investigated to determine if there are any savings potential.

The capital set aside for land acquisition is an estimate based on the amount of private land that may need to be acquired, and is based on current market rates for similar properties adjacent to the site.

The Permitting and land acquisition cost summary is shown in Table 21-7.

TABLE 21-7 PERMITTING AND LAND ACQUISITION COST SUMMARY

Item	Total ($ x 1000)
Operating Licence	44
Forestry Land Permits	959
Pasture Land Permits, with deposit	1,721
Water/Road permits	180
Land Acquisition	5,250
TOTAL	**8,154**

CONSTRUCTION AND PROJECT MANAGEMENT

The construction and project management section includes all of the capital costs associated with engineering and managing the construction and commissioning of the Project. The engineering component is based on proposals received by the engineering who have presented detailed engineering.

Geotechnical includes all of the subsurface analysis required before construction can commence, to confirm the load bearing characteristics of the supporting surface. Some geotechnical drilling and analysis was performed, but several locations of major sites have shifted, thus requiring additional work.

The construction and project management capital cost summary is shown in Table 21-8.

TABLE 21-8 CONSTRUCTION AND PROJECT MANAGEMENT CAPITAL COST SUMMARY

Item	Total ($ x 1000)
Detailed Engineering	16,355
Construction Management	5,507
Project/Commissioning Management	1,500
Temporary Facilities	1,260
Environmental Monitoring	500
Geotechnical Work	1,700
Freight	4,000
Vendor’s Representation	500
Construction/Critical Spares	1,300
First Fill	1,200
TOTAL	**33,822**

PROJECT SUSTAINING CAPITAL

Sustaining capital requirements for the Project are minimal, primarily due to the contracting out of the mining tasks, obviating the need for allocating sustaining capital for mobile mining equipment, and for haul road maintenance, which is part of the mining contractor’s costs. The estimates are found in Table 21-9.

The major sustaining capital requirements are for completing the Phase 2 and Phase 3 construction of the HLP.

The other area requiring additional sustaining capital is the replacement of light vehicles during the life of the mine.

TABLE 21-9 PROJECT SUSTAINING CAPITAL COST SUMMARY

Item	Total ($ x 1000)
HLF	9,485
Other	200
TOTAL	**9,685**

CLOSURE COST

Closure costs have been estimated at $27M, or $1.03/t processed, for the Project. This value has been based on mine closure industry experience combined with knowledge of current gold operations in Turkey. Prior to construction, a more detailed Conceptual Closure Plan (CCP) will be developed that will further expand on the aforementioned closure concepts, and use a systematic approach for more accurately estimating the closure costs such as Standardized Reclamation Cost Estimator (SRCE).

OPERATING COSTS

Operating costs were developed from first principles for processing and general and administrative costs. Mining costs have been based upon discussion with mining contractors in Turkey with additional estimates for engineering, grade control, and contractor management. These costs have been estimated using typical staffing requirements and wage rates in Turkey, as estimated by Centerra. Unit rates of consumption for materials and supplies are based on test work and relevant industry standards. A summary of unit operating costs is shown in Table 21-10 and total annual operating costs can be seen in Table 21-18 at the end of this section.

TABLE 21-10 OPERATING COST SUMMARY

Area	Unit	Value
Processing	US$/t processed	5.17
General & Administrative	US$/t processed	2.50
Mining[1]	US$/t mined	2.34
Royalties	US$/t processed	1.36
Refining	US$/t processed	0.11

[1] Mining includes ore re-handling costs and US$30M of capitalized stripping.

PROCESS

LABOUR

Labour costs (including overtime, social security payments and bonuses) were estimated from a staffing list generated from knowledge of similar heap leach operations, both in Turkey and other countries. The staffing list is based on the assumption that all employees would live locally near the Öksüt mine and would be bussed to the site. No camp is included in the operation. ADR and heap leach operations would operate on a continuous basis while other aspects of the operation would work Monday to Friday. Table 21-11 shows the ADR and Heap Leach Staffing List and Table 21-12 shows the Maintenance Staffing List. Maintenance includes crushing, heap leach, ADR, and other surface facilities but excludes mining.

TABLE 21-11 HEAP LEACH AND ADR STAFFING LIST

Process	
Process Manager	1
Process Department Office Administrator	1
Process-ADR Solution Handling Coordinator	1
Crushing Plant Coordinator	1
ADR Shift Supervisor	4
ADR Foreman	4
ADR Operator	12
Crusher Shift Supervisor	4
Crusher Foreman	4
Crusher Operator	12
Stacking Supervisor	4
Refinery Supervisor	1
Refinery Foreman	1
Refinery Operators	4
Leach Pad Operator	16
Plant Metallurgist	1
Metallurgical Technicians	4
Laboratory Coordinator	4
Instrumentation Specialist	1
Process Engineer	1
Process - Total	**81**

TABLE 21-12 MAINTENANCE STAFFING LIST

Maintenance	
Maintenance Manager	1
Superintendent of Maintenance Planning	1
Maintenance Manager	1
Preventive Maintenance Coordinator	1
Electrical Engineer	1
Mechanical Engineer	1
Maintenance Planning Coordinator	1
Crusher Plant Mechanical Foreman	4
Crusher Plant Mechanics	4
Crusher Plant Electrician	4
ADR Plant Maintenance Foremen	4
ADR Plant Mechanics	4
ADR Plant Electrician	4
ADR Maintenance E & I Technician	1
Maintenance – Total	**30**

PROCESS OPERATING AND MAINTENANCE SUPPLIES, POWER COSTS

Costs for operating supplies and power consumption were estimated from first principles. Reagent consumptions were estimated from metallurgical testing or based on industrial practice. Cyanide consumption was estimated at 30% of test work data which was 0.30 kg/t. Budget quotations for reagents were obtained. Prices are based on a delivered price to the mine site. The electrical load list was assembled from the engineering completed for the heap leach, ADR facility and the water supply line to the site. Estimates of loads for buildings and heating were used pending further engineering. Demand and utilization factors were applied to each installed electrical load. The total installed electrical load is 7.8 MW and the net power draw is 4.7 MW. Consumable costs for crushing and screening were estimated through the Metso Bruno crushing simulator. The estimates are based on the equipment selected, throughput and hardness data.

Maintenance supplies costs were estimated as a percentage of capital costs. This includes costs for building maintenance, mechanical and piping and electrical and instrumentation.

The leach pad haulage cost to load, haul, and place ore on the heap leach pad is estimated at $0.64/t from first principles. Life of mine average process operating costs are shown in in Table 21-13.

TABLE 21-13 PROCESS OPERATING COSTS SUMMARY

Area	$/t Ore Leached
Labour	1.06
Operating Supplies	1.84
Maintenance Supplies	0.19
Power	1.43
Sub-Total	**4.53**
Leach Pad Haulage	0.64
Total	**5.17**

GENERAL AND ADMINISTRATIVE COSTS

General and administrative (G&A) costs were estimated from staffing lists for the mine site and the Ankara OMAS office. Labour costs were estimated from surveys of other Turkish mining operations and are all-in costs, including overtime, social security, and bonuses. Tables 21-14 and 21-15 show the labour estimates for both the site G&A and the Ankara office.

TABLE 21-14 MINE SITE GENERAL AND ADMINISTRATIVE STAFFING LIST

Mine Site General and Administrative	
Senior Accountant	1
Accountant	2
A/P Accountant	1
Purchasing Coordinator	1
Stock Controller	4
Warehouse Foreman	4
Warehouse Operator	4
Logistics Specialist	1
IT Specialist	1
Administrative Affairs Chief	1

Mine Site General and Administrative	
Human Resources Specialist	2
Office Administrator	1
Office Assistant	1
Human Resources Coordinator	1
Training Specialist	1
Training Coordinator	1
CR Coordinator	1
CR Specialist	1
CSR Specialist	1
Social Performance Specialist	1
Kitchen Staff	20
Administration Building Cleaning Staff	8
DCC Specialist	1
Driver	1
Receptionist	1
Doctor	1
Safety Coordinator	1
Environment Coordinator	1
Security Guards	12
Environmental Technician	2
Health Officer	4
Nurse	1
Safety Officer	4
Security Manager	1
Site General and Administrative Total	**89**

TABLE 21-15 ANKARA OFFICE STAFFING LIST

Ankara Office	
Country Manager/General Manager	1
Director, External Affairs and Sustainability	1
Corporate Lawyer	1
H.S.E. & T Manager	1
Director, Finance	1
Tax Manager	1
Manager Financial Accounting	1
Human Resources Specialist	1
Manager, Procurement and Supply Chain	1
IT Supervisor	1
Corporate Communication Specialist	1
Land and Permits Coordinator	1
Land and Permits Specialist	1
Office Administrator	1
Service Officer	1
Driver	1
Receptionist	1
Ankara Office Total	**17**

General and administrative expenses were estimated based on a list of items. Some of these include:

- Access road maintenance.
- Site kitchen and meal expenses.
- Employee ground transportation to site.
- Communications.
- IT and software costs.
- First aid costs.
- Forestry land permits costs.

In addition, costs also include labour costs for the Ankara office, site accounting and purchasing, site HR and administration, and site health and safety.

The total annual general and administrative expenses are estimated to be $8.7M.

MINING

It has been assumed that mining will be done by a mining contractor who will be responsible for all drilling and blasting, loading, hauling, and road and dump maintenance. The mining contractor will also be responsible for performing maintenance on all mobile equipment. Costs will be based on an all-in rate per cubic metre mined. Centerra will provide engineering and survey support, and will perform all grade control activities. Costs for blasthole assaying, software and supplies, and personnel have been included in the mining costs. Table 21-16 shows the mining labour breakdown excluding the mining contractor.

TABLE 21-16 MINING AREA HEADCOUNT

Mining	
Mine Manager	1
Mine Planning Coordinator	1
Mine Operations Coordinator	1
Surveyors	2
Mine Planning Engineer	2
Mine Shift Engineer	4
Chief Geologist	1
Senior Geologist	1
Geologist	1
Senior Database Geologist	1
Executive Assistant	1
Mining Total	**16**

The average mining cost over the life of the Project is $2.34/tonne mined. This includes mining contractor costs of $2.11/t and $0.19/t of overhead costs. An additional $0.50/tonne of material rehandled ($0.04/t mined) from the stockpile is added to arrive at the total mining cost in Table 21-18.

LABOUR

Table 21-17 shows the peak headcount during operations by area. The mining contractor will supply all operators, supervisors, and maintenance personnel related to mining, which are excluded from Table 21-17. Costs per person were estimated by Centerra as fully loaded costs including all wages, insurance, taxes, and applicable bonuses.

TABLE 21-17 HEADCOUNT SUMMARY BY AREA

‘

Area	Headcount
Mining Total (excl. Contractors)	16
Process Total	81
Maintenance Total	30
Site G&A	89
Site Total (excl. Contractors)	**216**
Ankara Office	17
Project Total (excl. Contractors)	**233**

ROYALTIES

As discussed in Section 4, the royalties applied to the Öksüt mine operations are a Turkish Government State royalty, a NSR royalty payable to Stratex Gold, and a royalty payable to Teck. Subsequent to the pit optimization it was determined that if gold was refined in Turkey the royalty owed to the Turkish government would be reduced from 6% to 3%, which is reflected in the operating cost estimate. The royalty to Stratex Gold is a 1% NSR on all precious metals extracted from Öksüt, net of refining charges, that is subject to a cap of a maximum of $20M.

The total royalty cost is $35.6M, or $1.36/t processed.

REFINING

Refining costs of $3.34/oz of gold sold, or $0.11/t processed, were included in the operating costs. It is assumed all gold will be refined in Turkey.

TABLE 21-18 ANNUAL OPERATING COST SUMMARY

Annual Operating Costs														
	Units	**Total**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**
Mining	($ x 1000)	**137,075**	-	-	23,462[1]	27,239	18,141	19,530	27,578	16,392	4,733	-	-	-
Processing	($ x 1000)	**135,133**	-	-	17,530	20,566	20,566	20,566	18,403	20,566	16,936	-	-	-
Administration	($ x 1000)	**67,379**	-	-	10,715[1]	8,687	8,691	8,695	8,539	8,700	8,352	5,000	-	-
Royalties	($ x 1000)	**35,626**	-	-	2,616	6,093	8,138	6,882	4,007	4,667	2,946	277	-	-
Refining	($ x 1000)	**2,990**	-	-	300.6	598.5	666.1	505.1	308.9	355.1	232.8	23.0	-	-
Direct Costs	($ x 1000)	**378,203**	**-**	**-**	**54,624**	**63,184**	**56,202**	**56,178**	**58,836**	**50,680**	**33,200**	**5,300**	**-**	**-**

[1] In 2017, pre-production expenditures include $2.0M of administration costs and $4.3M of mining costs and excludes capitalized stripping and closure costs.

22 ECONOMIC ANALYSIS

The material economic assumptions such as operating and capital cost estimates used for the calculations presented in this section and the justifications for such assumptions have been given in Section 21. The estimates provided in Section 21 have been summarized in Tables 21-1 and 21-18.

CASH FLOW FORECAST

Using a price of gold of $1,250 per ounce, as assumed for the Mineral Reserve estimation process, the open pit LOM plan, the processing schedule, and the operating and capital cost forecasts, an estimate of the after-tax free cash flow for the Öksüt Project has been made (Table 22-1). As shown in Tables 22-1 and 22-2, the total after-tax undiscounted free cash flow is $435.9M whereas the after-tax free cash flow discounted at 8% is $242.0M. The internal rate of return (IRR) of the Project is 42.5% after taxes. The payback period on construction capital and pre-production expenditures is expected to be two and half years after production commences.

The after-tax free cash flow includes all pre-production costs, ESIA permitting, and other related costs. Additional exploration work including tasks budgeted for 2016 have not been included in the Project financial analysis as it is assumed that any such work would be done to increase the currently defined Project resource. Exploration expenses incurred during 2014 and 2015 have been applied to the financial model for the purposes of taxation.

Capital costs include the construction of the power line and deposits related to forestry and land pasture fees, which will be returned to Centerra over the LOM. In the case of the power line, it has been assumed that the cost will be recovered by a series of five equal payments, paid yearly, beginning in 2017 and finishing in 2021. The forestry and land pasture deposits have been assumed to be refunded at the end of closure and reclamation activities in 2025.

TABLE 22-1 PROJECT LIFE OF MINE – CASH FLOW

	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Sales and Revenue													
Gold Price	(US$/oz)	**1,250**	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	-
Gold Produced (Sold)	(oz x 1000)	**895.2**	-	-	90.0	179.2	199.4	151.2	92.5	106.3	69.7	6.9	-
Gold Revenue	($ x 1000)	**1,119,024**	**-**	**-**	**112,494**	**223,974**	**249,295**	**189,034**	**115,620**	**132,885**	**87,112**	**8,610**	**-**
Operating Costs													
Mining	($ x 1000)	**137,075**	-	-	23,462[2]	27,239	18,141	19,530	27,578	16,392	4,733	-	-
Processing	($ x 1000)	**135,133**	-	-	17,530	20,566	20,566	20,566	18,403	20,566	16,936	-	-
Administration	($ x 1000)	**67,379**	-	-	10,715[2]	8,687	8,691	8,695	8,539	8,700	8,352	5,000	-
Royalties	($ x 1000)	**35,626**	-	-	2,616	6,093	8,138	6,882	4,007	4,667	2,946	277	-
Refining	($ x 1000)	**2,990**	-	-	300.6	598.5	666.1	505.1	308.9	355.1	232.8	23.0	-
Direct Costs	($ x 1000)	**378,203**	**-**	**-**	**54,624**	**63,184**	**56,202**	**56,178**	**58,836**	**50,680**	**33,200**	**5,300**	**-**
Capital and Other Costs													
Capitalized Stripping (Cash)	($ x 1000)	**30,260**	-	-	6,321	2,758	10,569	10,612	-	-	-	-	-
Construction Capital	($ x 1000)	**179,172**	19,448	140,529	19,195	-	-	-	-	-	-	-	-
Contingency	($ x 1000)	**25,150**	252	17,605	7,293	-	-	-	-	-	-	-	-
Sustaining Capital[1]	($ x 1000)	**9,685**	-	-	-	3,112	2,913	-	-	3,660	-	-	-
Working Capital	($ x 1000)	**0**	-	817	9,405	8,927	2,333	(4,931)	(6,298)	1,908	(3,581)	(8,580)	**-**
Closure Cost	($ x 1000)	**27,000**	-	-	2,714	5,404	6,015	4,561	2,790	3,206	2,102	208	**-**
Total Capital & Other[2]	($ x 1000)	**271,267**	**19,700**	**158,951**	**44,928**	**20,201**	**21,830**	**10,242**	**(3,508)**	**8,774**	**(1,479)**	**(8,372)**	**-**
Pre-production Expenditures [2]	($ x 1000)	**220,785**	**19,700**	**158,951**	**42,134**								
Cash Flow													
Pre-Tax Cash Flow	($ x 1000)	**469,554**	**(19,700)**	**(158,951)**	**12,942**	**140,590**	**171,263**	**122,614**	**60,292**	**73,431**	**55,391**	**11,682**	-
Income Tax Payable	($ x 1000)	**45,994**	-	-	36	2,617	3,263	13,300	7,009	11,320	8,449	-	-
Gov. Refunds[2]	($ x 1000)	**12,336**	-	-	2,195	2,195	2,195	2,195	2,195	-	-	-	1,361
Free Cash Flow	($ x 1000)	**435,896**	**(19,700)**	**(158,951)**	**15,101**	**140,168**	**170,195**	**111,509**	**55,478**	**62,111**	**46,942**	**11,682**	**1,361**
Cumulative Free Cash Flow	($ x 1000)	**-**	**(19,700)**	**(178,651)**	**(163,550)**	**(23,382)**	**146,813**	**258,322**	**313,800**	**375,911**	**422,855**	**434,535**	**435,896**
All-in Sustaining Cost[1] per oz sold	$/oz	**490**	-	-	638	416	380	472	666	541	507	800	-
All-in Cost[1] per oz sold	$/oz	**725**	-	-	1,002	416	380	472	666	541	507	800	-
All-in Cost[1] per oz produced including taxes	$/oz	**777**	-	-	1,002	430	396	560	742	648	628	800	-

Notes:

1. Non-GAAP measure, see discussion under "Non-GAAP Measures" in Centerra's Management's Discussion and Analysis for three and six months ended June 30, 2015.
2. In 2017, pre-production expenditures include $2.0M of administration costs and $4.3M of mining costs and excludes capitalized stripping and closure costs.

TABLE 22-2 UNDISCOUNTED AND DISCOUNTED AFTER-TAX FREE CASH FLOW

Discount Rate	Net Present Value ($M)
0%	435.9
5%	301.7
8%	242.0
10%	208.8

Note: Gold Price: US$1,250/oz

TAXATION AND ROYALTIES

The corporate income tax rate in Turkey is 20%. However, Investment Incentive Certificates (IIC) are available to provide reduced corporate tax rates for profits derived from investments made in Turkey to promote economic development. The Öksüt Project economic analysis assumes that an IIC will be obtained.

Based on the scope and the amount of the planned investment, it is expected that a Strategic IIC will be granted to the Project. If the Öksüt Project receives a Strategic IIC, it will be eligible for the additional following benefits:

- Further reduction of corporate income tax rate
- VAT exemption
- Customs duty exemption
- Government support for interest payments on loans which are registered to the IIC
- Government support for employer's share of social security premiums
- VAT refund support

For the economic analysis, only the impact of the corporate income tax rate reductions under the IIC and the Strategic IIC have been included. The corporate income tax rate reduction for the Öksüt Project becomes available through the opportunity to receive an income tax credit against income tax expense that results from taxable income. The amount of the tax credit depends on the amount of eligible capital expenditures qualifying under the IIC. For the Öksüt Project, a strategic IIC would permit the receipt of an income tax credit equal to 50% of the amount of capital expenditures qualifying as eligible under the IIC.

Assets that are used for more than one year and that are subject to wear and tear are depreciated at legislated rates that are based on the useful lives of various categories of fixed assets. In general, the rate determined for fixed assets specific to the mining sector is 10%. Taxpayers in Turkey have the option of selecting either the straight line method or the double declining balance method to calculate depreciation for fixed assets, although some intangible fixed asset categories are not eligible for the double declining method.

Some capitalized expenditures, such as exploration expenses, development expenses, and mine land expenses, are subject to depletion at a rate which is determined by the Ministry of Finance and that takes into consideration the total ore reserve and the amount of ore extracted.

Gold production from the Project is also subject to three royalties, as follows:

1. A Turkish Government State royalty levied as a percentage of revenue less certain qualifying operating costs. The percentage is determined on a sliding scale that is linked to the price of gold and may be reduced by 50% if the ore is further processed in a refinery in Turkey (as is assumed in the economic analysis). Based on the assumed gold price of $1,250/oz, an effective royalty rate of 2% was used in the economic analysis.

2. A 1% NSR royalty payable to Stratex Gold up to a maximum of $20M; and

3. A sliding scale NSR royalty payable to Teck based on the cumulative ounces produced over the LOM. Centerra has estimated this royalty to be 0.6% of total gold revenues.

The base case considers that only the Turkish Government State royalty will be deductible as a taxable expense. The Stratex Gold and Teck royalties have been considered to be non-deductible expenses. The total effective royalty rate used for the economic analysis, at an assumed gold price of $1,250/oz, is 3.6%.

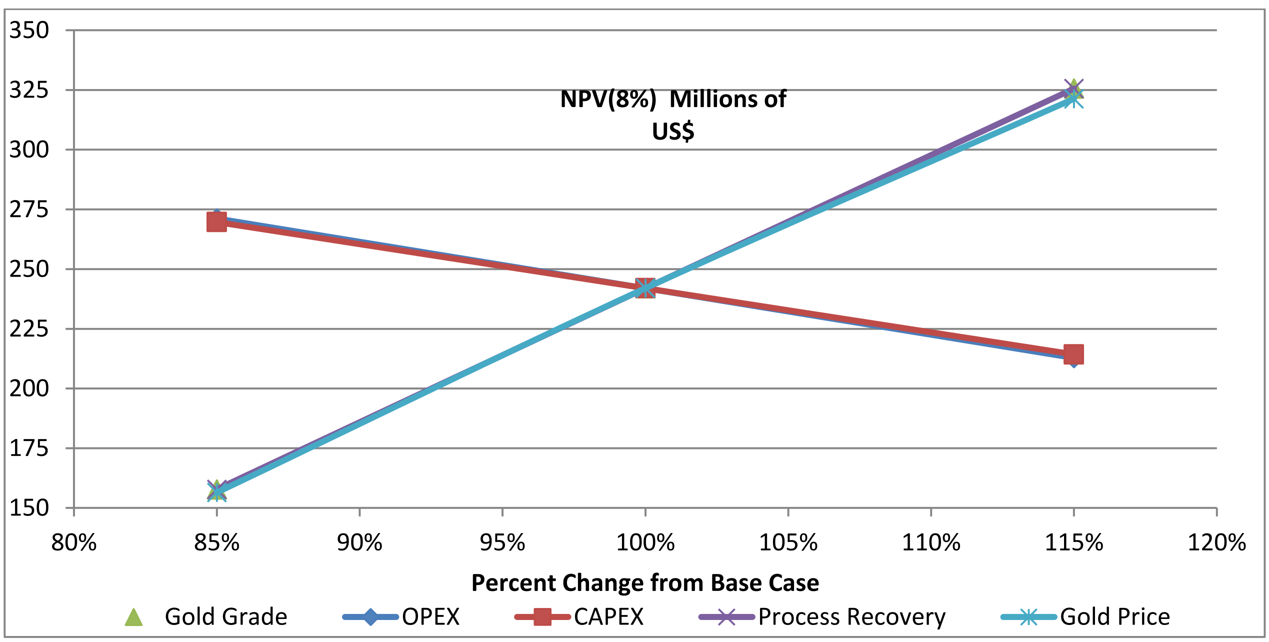
SENSITIVITY ANALYSIS

Table 22-3 provides after-tax free cash flow forecasts for the Öksüt Project based on the current LOM plan and a gold price of $1,250 per ounce. The table also shows the sensitivity of the Project NPV to gold prices ranging from $1,050 to $1,450 on $100 increments and on discount rates ranging from 0% to 10%. Sensitivities to four other variables at the base case gold price and a discount rate of 8% are also shown on a ±15% basis. Figure 22-1 is a graphical illustration of the cash flow sensitivities.

TABLE 22-3 CASH FLOW SENSITIVITIES (MILLIONS OF US$)

Sensitivity to Gold Price				
Discount Rate	**0%**	**5%**	**8%**	**10%**
Gold Price ($/oz)				
$1,050	298.2	195.6	150.5	125.5
$1,150	367.0	249.0	196.7	167.6
$1,250	**435.9**	**301.7**	**242.0**	**208.8**
$1,350	497.0	348.4	282.1	245.2
$1,450	567.1	400.3	326.7	285.5
Sensitivities to other Variables at $1,250 per ounce of gold and an 8% discount rate				
Variable	**Operating Costs**	**Capital Costs**	**Gold Grade**	**Process Recovery**
+15%	212.8	214.3	325.6	325.6
Base Case	**242.0**	**242.0**	**242.0**	**242.0**
-15%	271.2	269.7	157.8	157.8

FIGURE 22-1 CASH FLOW SENSITIVITIES



The LOM $NPV_{(8\%)}$ is most sensitive to changes in the gold grade and process recovery under an upside scenario (15% increase) followed by the gold price and then by operating and capital costs respectively. An additional 15% to the gold grade or process recovery increases the

Project $NPV_{(8\%)}$ by \$83.6M whereas a gold price increase of 15% results in an additional \$79.3M to the Project NPV(8%).

When considering the downside scenario, the Project is most sensitive to the gold price followed by gold grade or process recovery and then by operating and capital costs respectively. A 15% reduction in the gold price reduces the Project $NPV_{(8\%)}$ by \$85.5M. Similarly, a reduction in the gold grade or process recovery of 15% reduces the Project $NPV_{(8\%)}$ by \$84.2M.

At a gold price of \$767 per ounce and an 8% discount rate, the Project is cash flow neutral, but meets all foreseen capital and operating expenses including cash taxes. At gold prices higher than \$767 per ounce, the Project has the potential to generate positive free cash flow.

23 ADJACENT PROPERTIES

There are no adjacent properties to be discussed in this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

GEOLOGY

At June 30, 2015, the Öksüt database contained results of 330 diamond and RC drill holes for a total of 82,006 m. Centerra concludes that the data, the data density, and the additional information from the Öksüt database are adequate to form the basis for a Mineral Resource estimate.

A site visit to the Öksüt Project reviewed property and deposit geology, exploration and drilling methods and results, sampling method and approach, sample and data handling, including chain of custody. A qualified geologist evaluated the compilation of QA/QC data from the Öksüt Project and is of the opinion that the sample preparation, security, and analytical procedures used by OMAS and Stratex followed industry-standard procedures and the resulting analytical data are acceptable for use in resource estimation.

Drilling to date has been completed at an average drill hole spacing of 50 m along and across strike over known mineralized zones, the majority of which has been classified as Measured or Indicated Mineral Resources. Drilling has not completely defined the limits of the mineralization at the Güneytepe Deposit, and the mineralized zones remain open along strike to the north.

Centerra estimates the current Measured and Indicated Mineral Resources at the Öksüt Project to be 33.0 Mt at an average grade of 1.2 g/t Au containing 1.3 million ounces Au and Inferred Mineral Resources to be 2.4 Mt at an average grade of 0.8 g/t Au containing 64,000 ounces Au.

In order to comply with the CIM Definitions requirement of “reasonable prospects for eventual economic extraction”, a preliminary Whittle pit shell was used to constrain the Mineral Resource estimate using process recovery of 77%, a gold price of $1,450, and other assumed pit parameters. Only blocks located within the pit shell are reported in the Mineral Resource estimate.

OPEN PIT MINING

The mineralization at the Öksüt Project that is included in this analysis comes from two separate deposits to be mined by open pit methods. This mineralization is amenable to conventional open pit, loader/truck mining methods that will be carried out by a mining contractor.

Utilizing the updated Measured and Indicated Mineral Resources, pit optimization was carried out on both the Keltepe and Güneytepe Deposits, using updated geotechnical parameters, updated operating costs, metallurgical recovery estimates, and a gold price of $1,250 per ounce. The resulting pit shells were used as a guide in completing engineered pit designs which include ramps suitable for 36 tonne haul trucks consistent with other contractor mining operations in Turkey. This generated an estimated Probable Mineral Reserve of 26.1 Mt at 1.4 g/t Au containing 1.2 million ounces of gold at a cut-off grade of 0.3 g/t Au.

MINERAL PROCESSING

Test work carried out on representative samples of the deposits indicates that the Öksüt Project mineralization can be treated with heap leach methods. The HLP is designed to process oxide ore from two open pits.

The proposed Öksüt process plant design will be based on gold heap leach technology, which will consist of crushing, stacking, and heap leaching with cyanide solution, adsorption of the pregnant solution, elution, and gold smelting.

FEASIBILITY STUDY RESULTS

Financial analysis of the Öksüt Project has determined that the Project will be economically viable and profitable.

The FS was based on open pit mining of the Keltepe and Güneytepe Deposits. The target ore production rate during the life of the mine is 4.0 Mt per year.

The results of the FS included the following:

- Using a price of gold of $1,250 per ounce, the total after-tax undiscounted free cash flow is $435.9M, whereas the after-tax free cash flow discounted at 8% is $242.0M. The internal rate of return of the Project is 42.5% after taxes and payback is in 2.5 years.

Centerra concluded that the FS demonstrated the viability of the Öksüt Project as proposed, and that further development is warranted.

RISKS

Centerra has identified risks in the areas of Mineral Reserve estimation, recovery, geotechnical, mine plan, permitting, cost estimation, and schedule and has a mitigation plan for these items.

MINERAL RESERVE ESTIMATION RISKS

The OksutMay2015 Mineral Resource model was used as the basis for the June 30, 2015 Mineral Reserve estimate. As with any mineral resource estimate, there is no guarantee that the model will accurately predict future gold production. As production commences, the resource model will be reconciled to production and adjusted as required.

RECOVERY RISKS

The KCA Phase 3 testing program identified an unexpected variability in ore metallurgical properties which requires follow-up analysis and may impact the overall recovery estimate. Additional testing will be performed in 2016 utilizing an updated alteration model to provide guidance for sample selection in order to manage the unexpected variability in ore metallurgical properties identified in the KCA Phase 3 testing program.

GEOTECHNICAL RISKS

Geotechnical designs for the pits, heap leach pad, and waste dump are based on the available data. If actual conditions differ from those currently assumed, changes to designs may be required, which could have an adverse effect on Project economics. Additional data collection is planned throughout the construction period and the operations phase of the Project.

MINE PLAN RISKS

The current mine plan assumes that a mining contractor will be able to achieve the planned mining rates for the planned costs. It also assumes that this contractor will be able to mine selectively to deliver the desired head grade to the heap leach pad, and will utilize proper blasting and mining techniques in order to achieve the geotechnical designs for the pit. Although this is common practice in Turkey, no contract has yet been signed and if a suitable contractor cannot be found, or the chosen contractor does not perform adequately, there could be an adverse effect on the project economics. Centerra expects to work closely with the contractor to ensure that Project schedule cost and head grade will be maintained.

PERMITTING RISKS

Although the EIA Report has been submitted, other factors, including ongoing political uncertainty, may delay the EIA Report's approval. This would result in overall Project delays including an inability to secure necessary permits and start construction.

COST ESTIMATION RISKS

Not all of the cost items are based on firm quotations or detailed engineering. Several of the key costs items, including earthworks, road construction, electrical power distribution, and water handling, are based on estimates using projected unit costs. Detailed engineering, as well as finalization of the procurement process, will aid in the management of the cost estimation risks.

SCHEDULE RISKS

The major schedule risks are due to delays caused by a number of factors including variations in actual equipment delivery, construction schedules, weather delays, and permitting delays. Detailed engineering, as well as finalization of the procurement process, contract awarding, and project management controls will aid in maintaining the Project schedule.

26 RECOMMENDATIONS

The key recommendation is to proceed with the development of the Öksüt Project in 2015 with the target of achieving initial gold production in Q2 2017.

Additional recommendations include:

- Implement a database management system that is capable of supporting production sampling.
- Complete an independent audit of the assay database after implementation of a database management system.
- Perform additional density measurements within barren material outside of the existing mineralization.
- Survey the remaining five GPS surveyed holes with differential global positioning system (DGPS).
- Create an alteration model using ICP data as the basis for a geo-metallurgical recovery model.
- Evaluate the impact of using shorter composite lengths for grade estimation.
- Perform a contact plot analysis to assess the need for soft, semi-soft, or hard estimation boundaries for high grade and low grade material.
- Based on the completed geo-metallurgy recovery model, identify any gaps in the samples tested to date. Using either available samples or samples from new drill holes, complete additional leach tests.
- Conduct additional testing utilizing an updated alteration model to provide guidance for sample selection in order to manage the unexpected variability in ore metallurgical properties identified in the KCA Phase 3 testing program.
- Complete a detailed leach plan and schedule utilizing the updated geo-metallurgy recovery model.
- Complete additional geotechnical drilling for final HLP, waste dump, and other surface infrastructure designs, to provide enough data for detailed engineering.
- As mining begins, data from pit wall mapping should be incorporated into the geotechnical model and pit designs adjusted accordingly.

- Confirm, or re-visit, the hydrogeological assumptions after mining commences. In case there are material changes to the water table assumptions, the slope stability analyses should be updated.

- Sensitivities to mining losses and dilution should be quantified. As mining begins, these assumptions should be monitored and adjusted, if necessary.

- Due to the climate characteristics at the Project site, the ponds are expected to be empty throughout most of the year, with the majority of inflows concentrated in the months of March and April. Assessment of options for protecting the geomembrane from exposure to natural environment would need to be considered during detailed design (e.g., covering the geomembrane with a protective layer of sand or gravel).

27 REFERENCES

AACE. 2005. *Cost Estimate Classification System – As Applied in Engineering, Procurement, and Construction for the Process Industries*. AACE International Recommended Practice No. 18R-97, 2005.

Arpacıoğlu, B., 2014: Environmental Impact Assessment (EIA) Application Document. Unpublished memorandum from SRK Danişmanlık, Ankara, to Kevin D'Souza, Alper Sezener, Malcolm Stallman dated February 5, 2014, 4 p.

Baytop, T., 1997. *Türkçe bitki adları sözlüğü.* Ankara : Türk Dil Kurumu.

BBA Consulting. 2014. Cost evaluation for Drill & Blast scenarios, Öksüt Gold Mine project – Turkey. December 12, 2014.

Bener Law Office, 2013. Turkey: Mining Sector and Mining Law. Information Memorandum, January, 2013.

Bertelsmann Stiftung Transformation Index (BTI) 2010: Turkey Country Report, Bertelsmann Foundation 2010

Besang, C., Eckhardt, F.J., Harre, W., Kreuzer, H., and Müller, P., 1977. Radiometrische Alterbestimmungen an neogenen Eruptivgesteinen der Türkei. *Geologisches Jahrbuch*, **25**, 3-36.

Biçer, A.E., 2013: Merger of Operation Licenses and Exemption Provided for Beneficiation Facilities under the Mining Law. Unpublished memorandum from Çakmak Avukatlık Bürosu, to Malcolm Stallman, Centerra Madencilik A.Ş. dated December 4, 2013, 5 p.

Bloom, L., 2013. Assay Quality Control Annual Report (August 2008 – January 2013) Öksüt Project. Unpublished report for Öksüt Madencilik A.Ş. May 2013, 20 p.

Bozkurt, E., 2001. Neotectonics of Turkey - a synthesis. *Geodinamica Acta,* **14**, 3-30.

Burgan Securities, 2013: Economic Review – Turkey, November 22, 2013

Cabanis, B. and Lecolle, M., 1989: The La/10-Y/15-Nb/8 diagram; a tool for distinguishing volcanic series and discovering crustal mixing and/or contamination*; Comptes Rendus de l'Academie des Sciences*, **309**, pages 2023-2029 (in French with English abstract)

Centerra, 2014: Öksüt Gold Project Feasibility Study CAPEX template (Draft). Received from Centerra Gold Inc. January 16, 2015.

Centerra, 2014: Öksüt Gold Project Pre-Feasibility Study, February 2014.

Centerra, 2014b: Stability of the waste rock dump. Öksüt project. November 26, 2014

Centerra, 2015: Pit Slope Stability Assessment. Report submitted January 9, 2015.

Centerra, 2015: Öksüt Gold Project Feasibility Study, July 2015.

Cook, M. R., 2013: Heap Leach Amenability Testing of Öksüt Madencilik Gold Ore Samples. Prepared for Mr. Bahri Yildiz, Centerra Gold Inc. Unpublished report by SGS London, UK, Project Number 10866 – 374, 10th May 2013

Davis, P.H., Mill, R.R., and Tan, K, 1988. *Flora of Turkey and the East Aegean Island*s. Edinburgh: Edinburgh University Press.

Dhont, D., Chorowicz, J., Yürür, T., Froger, J. L., Köse, O., and Gündoğdu, N., 1998. Emplacement of volcanic vents and geodynamics of Central Anatolia, Turkey. *Journal of Volcanology and Geothermal Research*, **85**, 33-54.

Dirik, K., 2001. Neotectonic evolution of the northwestward arched segment of the Central Anatolian Fault Zone, Central Anatolia, Turkey. *Geodinamica Acta*, **14**, 147-158.

Ekim, T., et al., 1989. *Türkiye Bitkileri Kırmızı Kitabı* [The Red Book of Turkish Plants], Türkiye Tabiatini Koruma Derneği, Ankara.

Esen, E., 2014: Seismicity Memo. Unpublished Memorandum by SRK Danişmanlık, Ankara, dated February 6, 2014, 3 p.

European Bank for Reconstruction and Development, 2013: Turkey Country Strategy, March 2013

FHWA (Federal Highway Administration). 2014. HY8 Culvert Hydraulic Computation. Version 7.30.

Fitch Ratings Ltd., 2013: Sovereigns – Turkey – Full Rating Report, November 13, 2013

Giardini, D. Grünthal, G. Shedlock, K., and P. Zhang, 1999: Global Seismic Hazard Assessment Program, Global Seismic Hazard Map

Golder (Golder Associates Ltd.) 2014a. *Climate and Hydrology Design Parameters – Öksüt Project Site Wide Water Management Plan and Water Balance*. Technical Memorandum submitted November 10, 2014.

Golder. 2014b. *Design Report for the Öksüt Heap Leach Facility* (Draft). Report submitted November 26, 2014.

Golder. 2014c. Surface Water Management design criteria and conceptual flow diagram – Öksüt Project, Turkey (Draft). Report submitted November 26, 2014.

Golder. 2014d. Geotechnical Pit Slope Investigation – Field Data Report. September 22, 2014.

Golder. 2015a. Site-Wide Surface Water Management Plan and Water Balance – Öksüt Project, Turkey (Draft). Report submitted February 6, 2015.

Golder. 2015b. *Design Report for the Öksüt Heap Leach Facility* (Final). Report submitted February 11, 2015.

Hedenquist J.W., Arribas A., Jr., and Gonzalez-Urien E., 2000. Exploration for epithermal gold deposits: *Reviews in Economic Geology*, **13**, p. 245-277.

John, D.A., 2001, Miocene and early Pliocene epithermal gold-silver deposits in the northern Great Basin, western USA: Characteristics, distribution, and relationship to magmatism: *Economic Geology*, **96**, 1827–1853.

Kappes, Cassiday & Associates (KCA), 2013: Öksüt Project. Report of Metallurgical Test Work. Unpublished report for Centerra Gold Inc., December 2013, 146 p.

Kappes, Cassiday & Associates (KCA), 2014a: Öksüt Project. Report of Metallurgical Test Work. Unpublished report for Centerra Gold Inc., September 2014, 95 p.

Kappes, Cassiday & Associates (KCA), 2014b: Öksüt Project. Report of Metallurgical Test Work. Unpublished report for Centerra Gold Inc., December 2014, 144 p.

Ketin, İ., 1966. Tectonic units of Anatolia. *MTA Bulletin*, **66**, 23-34.

Kiziroğlu, I., 2008, Türkiye kuşları kırmızı listesi [Red data book for birds of Türkiye]. Ankara, Türkiye: Center for Environmental Education, Avian Studies and Bird Ringing, Hacettepe University.

Kürkcüoğlu, B., 2010. Geochemistry and petrogenesis of basaltic rocks from the Develidağ volcanic complex, Central Anatolia, Turkey. *Journal of Asian Earth Sciences*, **37**, 42-51.

Kurtuluş, O., 2012. Geology, alteration and gold mineralisation of the Ortaçam North Prospect. Unpublished report for Stratex International Plc. March, 2012, 7 p.

Lashko, E., 2013. The Ortaçam North high-sulphidation epithermal gold deposit: A description of the breccias and associated mineralisation. Unpublished MSc thesis, University of Southampton, England, 140 p.

Mandziak, T., 2013: Öksüt Heap Leach Pad Conceptual Design, Kayseri Province, Turkey. Unpublished report by SRK Consulting (U.S.), Inc. (Denver) dated December 2013.

Moody's Investors Service, 2013: QE Tapering & Turkey – Impact on Various Sectors of Turkish Economy Will Likely be Limited and Short-Lived Given Existing Buffers, December 2, 2013

Okay, A. I. and Tüysüz, O., 1999. Tethyan sutures of northern Turkey. *In*: Durand. B., Jolivet, L., Horvath, E and Seranne, M. (eds) *The Mediterranean Basins: Tertiary Extension within the Alpine Orogen*. Geological Society, London, Special Publications, **156**, 475-515.

Öksüt Madencilik (Öksüt Madencilik Sanayi ve Ticaret A.Ş. 2014a. Digital layout of Mine Facilities (pits, ponds, stockpiles, waste dump and administrative buildings) and Existing Ground Surfaces dated November 21, 2014. Digital File: TUOK-01-501-O-M-002.dwg.

Öksüt Madencilik (Öksüt Madencilik Sanayi ve Ticaret A.Ş. 2014b. *Preliminary Economic Assessment – Öksüt Gold Project, Turkey*. Effective Date June 19, 2014.

Pasquare, G., Poli, S., Vezzoli, L., and Zanchi, A., 1988. Continental arc volcanism and tectonic setting in central Anatolia, Turkey, *Tectonophysics* **146**, 217–230.

Phillipson, P.B., and Ayaş, Z., 2014. Öksüt Gold Mine Project Biodiversity Review, January 2014, 8 p.

Redwood, S.D., 2009. Exploration review of the Öksüt high sulphidation epithermal gold deposit, and porphyry gold potential, Develi District, Kayseri Province, Turkey. Unpublished report prepared for Stratex International Plc and Centerra Gold Inc. JV, November, 2009, 68 p.

SGS Canada Inc., 2013: An Investigation into Gold Deportment and Metallurgical Diagnostic Leach Tests on two Samples. Prepared for Öksüt Madencilik Saniye ve Ticaret A.Ş. Project 14318-001 Final Report, February 19, 2014

SGS Canada Inc., 2014: Gold Deportment and Metallurgical Diagnostic Leach Tests on Six Samples. Prepared for Öksüt Madencilik Saniye ve Ticaret AS. Project 14318-002 Final Report, February 2, 2015

Sillitoe, R. 1999. Styles of High-Sulphidation Gold, Silver and Copper Mineralisation in Porphyry and Epithermal Environments. *Proceeding of the Australasian Institute of Mining and Metallurgy*, **305**, 19-34.

Sillitoe, R. 2010. Porphyry Copper Systems. *Economic Geology*, **105**, 3-41.

Sillitoe, R., 2011. Geology and exploration of the Öksüt epithermal gold prospect, Turkey. Unpublished report for Stratex International Plc. April 2011, 7 p.

Sillitoe, R., 2012. Geology and potential of the Ortaçam North gold prospect, Öksüt Project, Turkey. Unpublished report for Stratex International Plc. April 2012, 12 p.

Sillitoe, R.H., and Hedenquist, J.W., 2003, Linkages between volcanotectonic settings, ore-fluid compositions and epithermal precious metal deposits, in Simmons, S.F., and Graham, I., eds., *Volcanic, geothermal and ore-forming fluids; rulers and witnesses of processes within the Earth*: Society of Economic Geologists Special Publication **10**, 315–343.

SRK Consulting Ankara, 2014a. Geochemical Baseline and High Level Impact Assessment Study Report for Öksüt Project (Phase-1), January 2014, 182 p.

SRK Consulting Ankara, 2014b. Öksüt Preliminary Social Baseline and Impact Identification, January 2014, 98 p.

SRK Consulting Ankara, 2014c. Phase-1 Environmental Baseline Report for Öksüt Project, January 2014, 404 p.

SRK Consulting Ankara, 2014d. Update on Hydrological Field Investigations. September 2014

SRK Danişmanlik ve Mühendislik A.Ş., 2015. Öksüt Gold Mine (Open Pits, Heap Leach and Process Plant) Project EIA Report - Environmental Impact Assessment, Ankara, April 2015.

Standard & Poor's Ratings Services, 2013: Ratings Direct – Has Turkey's External Adjustment Entered a New Phase?, November 22, 2013

Standard & Poor's Ratings Services, 2013: Ratings Direct – Research Update: Turkey Foreign Currency Ratings, November 22, 2013

Thalenhorst, H., 2013: Öksüt Project – Note on Bulk Density (Update). Unpublished Letter Report by Strathcona Mineral Services Limited, November 8, 2013

The World Bank, 2013: Doing Business 2014, Economy Profile: Turkey, October 29, 2013

The World Bank, 2014: Turkey Regular Economic Brief, 2014-I

Toprak, V., 1998. Vent distribution and its relation to regional tectonics, Cappadocian Volcanics, Turkey. *Journal of Volcanology and Geothermal Research*, 85, 55-67.

Transparency International, 2013: Corruption Perceptions Index 2013

Transparency International, 2013: Transparency International Turkey Worried About Rising Violence, Moves Against Freedom of Expression, June 11, 2013

Transparency International, 2013: Transparency International Turkey Statement on Latest Corruption Scandal, December 19, 2013

USDA-SCS (United States Department of Agriculture- Soil Conservation Service). 1986. *Urban Hydrology for Small Watershed*. TR-55, 164 p.

Wardell Armstrong International, 2011: Stratex International Plc. Öksüt Resource Modelling. Unpublished report prepared for Stratex International Plc, March, 2011, 31 p.

Wardell Armstrong International, 2012: Stratex International Plc. Öksüt Resource Modelling. Unpublished report prepared for Stratex International Plc, February, 2012, 41 p.

Wilkins, J.K., 1956. *Flow of Water Through Rockfill and its Application to the Design of Dams*. Proceedings of 2nd Australia – New Zealand Conference on Soil Mechanics and Foundation Engineering, January, Christchurch, New Zealand, pp. 141 -149.

Williams, G.L., 2012: Report on Preliminary Leach Testing of Öksüt Gold Ore, Turkey. Prepared for Mr Bahri Yildiz, Öksüt Madencilik. Unpublished report by SGS London, UK, Project Number 10866 – 344, 19th July 2012

USACE (US Army Corps of Engineers). 2010. Hydrologic Modeling System HEC-HMS. Version 3.5, released August 2010.

USBR (United States Bureau of Reclamation). 1984. *Hydraulic Design of Stilling Basins and Energy Dissipators*. Engineering Monograph 25.

28 DATE AND SIGNATURE PAGE

This report titled "Technical Report on the Öksüt Gold Project, Turkey" dated September 3, 2015 was prepared and signed by the following authors:

	(Signed) "*Gordon D. Reid*"
Dated at Toronto, ON September 3, 2015	Gordon D. Reid, P.Eng. Vice President, and Chief Operating Officer Centerra Gold Inc.
	(Signed) "*Peter Woodhouse*"
Dated at Toronto, ON September 3, 2015	Peter Woodhouse, P.Eng. Director, Projects Centerra Gold Inc.
	(Signed) "*Malcolm Stallman*"
Dated at Ankara, Turkey September 3, 2015	Malcolm Stallman, MAIG Regional Exploration Manager, Western Asia and Eastern Europe CMAS
	(Signed) "*Mustafa Cihan*"
Dated at Ankara, Turkey September 3, 2015	Mustafa Cihan, MAIG Chief Geologist OMAS
	(Signed & Sealed) "*Pierre Landry*"
Dated at Toronto, ON September 3, 2015	Pierre Landry, P.Geo. Corporate Geologist Centerra Gold Inc.
	(Signed & Sealed) "*Tyler Hilkewich*"
Dated at Toronto, ON September 3, 2015	Tyler Hilkewich, P.Eng. Corporate Mining Engineer Centerra Gold Inc.

(Signed & Sealed) "*Tommaso Roberto Raponi*"

Dated at Toronto, ON
September 3, 2015

Tommaso Roberto Raponi, P.Eng.
Director, Metallurgy
Centerra Gold Inc.

(Signed) "*Kevin P.C.J. D'Souza*"

Dated at Toronto, ON
September 3, 2015

Kevin P.C.J. D'Souza, MEng, ARSM, CEng, FIMMM, FRGS
Vice-President of Sustainability & Environment
Centerra Gold Inc.

(Signed & Sealed) "*Chris Sharpe*"

Dated at Toronto, ON
September 3, 2015

Chris Sharpe, P.Eng.
Senior Mining Engineer
Centerra Gold Inc.

29 CERTIFICATE OF QUALIFIED PERSON

GORDON D. REID

I, Gordon D. Reid, P.Eng., as a co-author of this report entitled “Technical Report on the Öksüt Gold Project, Turkey”, dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am Gordon D. Reid, Vice President, and Chief Operating Officer of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of the University of Manitoba in 1994 with a Master’s degree in Business Administration, and a graduate of Michigan Technological University in 1981 with a Bachelor of Science degree in Mining Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #38536504). I have worked as a mining engineer for a total of 30 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Since 2004, employed by Centerra Gold Inc. in various positions including VP & COO, Corporate VP Operations, President Kumtor Operating Company, and VP Business Development, involved in preparation of and adherence to annual mine plans and budgets, review and approval of capital programs, mine optimization studies, and other engineering and management related tasks.
   - From 1998 to 2002 was the Director – Technical Services responsible for the development of a feasibility study and Environmental Impact Assessment for a mine development project in Wisconsin.
   - From 1986 to 1992 was Chief Mine Engineer and Mine Superintendent at an operating mine in Ontario responsible for planning and implementing long term and short term mine development and production strategies to achieve budgeted production and cost targets.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I last visited the Öksüt Gold Project in June 2015 for a period of 2 days. I have visited the Öksüt property periodically since 2013.

6. I am responsible for overall supervision of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

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

Dated this 3rd day of September, 2015

(Signed) "*Gordon D. Reid*"

Gordon D. Reid, P.Eng.

PETER WOODHOUSE

I, Peter Woodhouse, P.Eng., as a co-author of this report entitled “Technical Report on the Öksüt Gold Project, Turkey”, dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am Director, Projects of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I graduated from McGill University, in Montreal, Quebec in 1980 with a degree in Mechanical Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (#51032506. I have worked as a mechanical or project engineer for a total of 35 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - I have worked at Centerra Gold as a project manager, and the director of projects since May 2014. I was responsible for managing the Feasibility Study of this project.
   - I have managed feasibility studies, detailed engineering, construction, and commissioning projects for the mining industry since 1997.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Öksüt Gold Project on June 1, 2015 for a duration of 2 days. Prior to this, I visited the Öksüt property in May 2014 for 2 days.

6. I am responsible for Sections 1, 2, 3, 18, 24, 25, 26, and parts of Section 21 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed) “*Peter Woodhouse*”

Peter Woodhouse, P.Eng.

MALCOLM STALLMAN

I, Malcolm Stallman, MAIG, as a co-author of this report entitled "Technical Report on the Öksüt Gold Project, Turkey", dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am Regional Exploration Manager, Western Asia and Eastern Europe of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I graduated from the Queensland Institute of Technology, Brisbane, Australia in 1986 where I obtained a BAppSc in Applied Geology.

3. I am a Member of the Australian Institute of Geoscientists (#2468). I have practiced my profession continuously since 1986 and have been involved in gold and base metal projects in Australia, Fiji, Turkey, Armenia, Saudi Arabia, Cyprus and Portugal, ranging from grassroots exploration through to advanced resource evaluation and mine planning activities. I have held positions ranging from Exploration Geologist to Regional Exploration Manager throughout my 30 years of industry experience.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I last visited the Öksüt Gold Project on June 22, 2015 for a duration of two days. Prior to this, I have completed numerous inspections of the Öksüt property since 2011.

6. I am responsible for Sections 4, 5, 6, 7, 8, 9 and 23, and parts of Sections 10, 11, and 12 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed) "*Malcolm Stallman*"

Malcolm Stallman, MAIG

MUSTAFA CIHAN

I, Mustafa Cihan, MAIG, as a co-author of this report entitled "Technical Report on the Öksüt Gold Project, Turkey", dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am Chief Geologist for Öksüt Madencilik A.Ş. (OMAS), subsidiary of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of Middle East Technical University, Geological Engineering Department, Ankara, Turkey, with a B.Sc. degree in 1998. I am also graduate of James Cook University, School of Earth Sciences, Australia with a PhD degree in 2005.

3. I am registered as a Member of Australian Institute of Geoscientist. I have worked as an academic and geologist for a total of seventeen years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Chief Geologist at Öksüt Gold Project, 2013 – Present,
   - Exploration Manager, Asia Minor Mining, Ankara, Turkey 2010 – 2013,
   - Project Manager/Senior Geologist, Axiom Mining, Vietnam, 2006-2009
   - Contract Mine/Exploration Geologist, Placer Dome, Australia, 2005-2006

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have been visiting the Öksüt Gold Project since May 2013 on a regular basis.

6. I am responsible for Sections 7, 8, 9 and parts of Sections 10, 11, and 12 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with OMAS, Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed) "*Mustafa Cihan*"

Mustafa Cihan, MAIG

PIERRE LANDRY

I, Pierre Landry, P.Geo., as a co-author of this report entitled "Technical Report on the Öksüt Gold Project, Turkey", dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am a Corporate Geologist with Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of Queen's University, Kingston, Ontario, in 2006 with a B.Sc.H degree in Geological Science (Major) and Economics (Minor).

3. I am registered as a Professional Geologist in the Province of Ontario (Reg. #2319). I have worked as a geologist for a total of seven years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and creation of block models as part of NI 43-101 Mineral Resource estimates, audits, and due diligence reports.
   - Mine Exploration Geologist at operations and mine development projects in Canada, Africa, and South America.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Öksüt Gold Project on July 13 to 15, 2015 for a duration of two days.

6. I am responsible for preparation of Section 14 and contributed to Sections 10 to 12 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those Sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed & Sealed) "*Pierre Landry*"

Pierre Landry, P.Geo.

TYLER HILKEWICH

I, Tyler Hilkewich, P.Eng., as a co-author of this report entitled "Technical Report on the Öksüt Gold Project, Turkey", dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am a Corporate Mining Engineer of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of Queen's University, Kingston, Ontario in 2007 with a BScE in Mining Engineering and a graduate of the University of Toronto, Toronto, Ontario in 2011 with a MEng in Civil Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #100158813). I have worked as a mining engineer for a total of eight years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Mine design, planning, and operations work on multiple open pit, heap leach operations in the United States.
   - Project evaluations and contributions to feasibility studies on mining projects in North America, South America, Africa, Australia, and Europe.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Öksüt Gold Project on July 15, 2015 for a duration of one day.

6. I am responsible for Sections 15 and parts of Sections 16 and 21 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed & Sealed) "*Tyler Hilkewich*"

Tyler Hilkewich, P.Eng.

TOMMASO ROBERTO RAPONI

I, Tommaso Roberto Raponi, P.Eng., as a co-author of this report entitled "Technical Report on the Öksüt Gold Project, Turkey", dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am Tommaso Roberto Raponi, Director, Metallurgy of Centerra Gold Inc. (the "Corporation"), a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of the University of Toronto in 1984 with a Bachelor of Applied Science in Geological Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #90225970) and the Association of Professional Engineers and Geoscientists of BC (Reg. #23536). I have worked as a mineral processing engineer for a total of 31 of years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - design, commissioning and operation of mineral processing plants in Canada, United States, Mexico, Brazil, Venezuela, Surinam, Chile, Kyrgyzstan, Mongolia, Turkey, and Saudi Arabia.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Öksüt Gold Project on June 3, 2014 for a duration of two days. Prior to this, I have completed several inspections of the Öksüt property since 2012.

6. I am responsible for Sections 13 and 17 and parts of Section 21 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed & Sealed) "*Tommaso Roberto Raponi*"

Tommaso Roberto Raponi, P.Eng.

KEVIN P.C.J. D'SOUZA

I, Kevin P.C.J. D'Souza, MEng, ARSM, CEng, FIMMM, FRGS, as a co-author of this report entitled "Technical Report on the Öksüt Gold Project, Turkey", dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am Kevin D'Souza, Vice President Sustainability & Environment of Centerra Gold Inc., a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of Royal School of Mines, Imperial College of Science Technology and Medicine University of London in 1993 with a Master's (MEng) degree in Mining Engineering.

3. I am registered as a Charted Engineer (1997, Reg.500601) through the British Institute of Materials, Minerals and Mining. I have worked as a sustainability and environment professional in the mining industry for a total of 22 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Worked in a range of mining environmental, sustainability, and Corporate Social Responsibility (CSR) issues, from remote exploration camps and operational mines to Corporate head offices.
   - Managed macro-level environmental and sustainable mining programmes for international non-governmental organizations (NGOs) and with the public sector (UN agencies, World Bank and the UK's Department for International Development).
   - Worked as a consultant with many junior exploration and mining companies and many of the industry's majors including Barrick, AngloGold Ashanti, Gold Fields, Rio Tinto, BHP Billiton, Kinross and De Beers.
   - Direct mining experience in around fifty countries worldwide largely in Asia, Latin America and sub-Saharan Africa.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Öksüt Gold Project on 11 May 2015 for a duration of 3 days. Prior to this, I have completed numerous inspections of the Öksüt property since 2013.

6. I am responsible for Section 20 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed) "*Kevin P.C.J. D'Souza*"

Kevin P.C.J. D'Souza, MEng, ARSM, CEng, FIMMM, FRGS

CHRIS SHARPE

I, Chris Sharpe, P.Eng., as a co-author of this report entitled "Technical Report on the Öksüt Gold Project, Turkey", dated September 3, 2015, with an effective date of June 30, 2015 do hereby certify that:

1. I am a Senior Mining Engineer of Centerra Gold Inc. (the "Corporation"), a corporation with a business address of 1 University Avenue, Suite 1500, Toronto, Ontario, Canada M5J 2P1.

2. I am a graduate of Dalhousie University with a Bachelor of Mining Engineering (2001).

3. I am registered as a Professional Engineer in the Province of Ontario. I have worked as a mining engineer for a total of 14 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Project evaluations including due diligence reviews and cash flow analysis for a number of international projects while working as a consultant and directly for operating companies.
   - Worked in the financial services industry as part of a mining evaluations group.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Öksüt Gold Project.

6. I am responsible for Section 22 of the Technical Report.

7. I am not independent of the Issuer, Centerra Gold Inc., applying the test set out in Section 1.5 of NI 43-101, as a result of my employment with Centerra Gold Inc.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, those Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2015

(Signed & Sealed) "*Chris Sharpe*"

Chris Sharpe, P.Eng.