Technical Report for the Certej Project, Romania

Effective Date: February 2014

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TECHNICAL REPORT

CERTEJ PROJECT, ROMANIA

Effective Date: February 21, 2014

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Units of Measure

Amphibole)	Amf
Annum (year)	a
Billion	B
Biotite	Bt
Centimetre	cm
Cubic centimetre	cm^3
Cubic metre	m^3
Day	d
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Dollar (American)	US$
Dollar (Canadian)	C$
Feldspar	Fp
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hour	h
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilograms per tonne	kg/t
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilopascal (ambient)	kPa(a)
Kilopascal (gauge)	kPa(g)
Kilotonne	kt
Kilovolt	kV
Kilowatt hour	kWh

Kilowatt hours per year	kWh/a
Kilowatt	kW
Less than	<
Litre	L
Megawatt	MW
Metre	m
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million ounces	Moz
Million tonnes per Annum	Mt/a
Million tonnes	Mt
Million	M
Minute (time)	min
Month	mo
Ounce	oz
Parts per million	ppm
Percent	%
Percent by Weight	wt%
Quartz	Qz
Second (time)	sec
Specific gravity	SG
Square centimetre	cm^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Week	wk
Weight/weight	w/w

Abbreviations and Acronyms

Acid Base Accounting	ABA
Acid Neutralizing Capacity	ANC
Acidity or Alkalinity	pH
Aluminum	Al
Ammonium Nitrate/Fuel Oil	ANFO
Analytical Detection Limit	ADL
Argon	Ar
Arsenic	As

Atonic Adsorption	AA
Backscattered Electron Imaging	BSE
Barium	Ba
Carbon-in-leach	CIL
Canadian Institute of Mining	CIM
Counter-current decantation	CCD
Copper	Cu
Distributed Control System	DCS
East	E
Effective Grinding Length	EGL
Eighty percent (80%) passing particle size	P80
Environmental Impact Assessment	EIA
Environmental Impact Statement	EIS
European Goldfields Limited	EGL
European Union	EU
Flocculant	FLOC
Front End Loader	FEL
Ground-Engaging Tools	GET
Gold	Au
Gold Equivalent	Au_Equiv
Gravity Recoverable Gold	GRG
Hermite Polynomials	Herco
High Density Polyethylene	HDPE
High Grade	HG
Inductively Coupled Plasma	ICP
Internal Rate of Return	IRR
Lead	Pb
Lerchs-Grossman	L-G
Life-of-mine	LOM
Liquefied Propane Gas	LPG
Low Grade	LG
Measured & Indicated	M&I
Million Years Ago	Ma
National Instrument 43-101	NI 43-101
National Technical University of Bucharest	NTUB
Natural Gas	NG
Net Present Value	NPV
Net Smelter Return	NSR
North	N
North East	NE
North West	NW
Potassium	K
Potassium Amyl Xanthate	PAX
Probability Assisted Constrained Kriging	PACK
Programmable Logic Controllers	PLCs

Qualified Person(s)	QP(s)
Quality assurance	QA
Quality control	QC
Reverse Circulation	RC
Run of Mine	ROM
Scanning Electron Microscope	SEM
Selective Mining Unit	SMU
Semi-autogenous Grinding	SAG
Silicon	Si
Silver	Ag
Societate pe Actiuni (Joint Stock Company)	S.A.
Sodium Meta-Bisulphate	SMBS
South	S
South East	SE
South West	SW
Standard Reference Material	SRM
Static Secondary Ion Mass Spectrometry	SSIMS
Strategic Environmental Assessment	SEA
Strontium	Sn
Sulphide	S^{2-}
Sulphuric Acid	H_2SO_4
Tailings Management Facility	TMF
Tellurium	Te
Total dissolved solids	TDS
Underground	UG
Universal Transverse Mercador	UTM
Wardell Armstrong International	WAI
West	W
Yitrium	Y
Ytterbium	Yb
Zinc	Zn
Zonal Urbanisation Plan	PUZ

SECTION • 1 SUMMARY

1.1 TERMS OF REFERENCE

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, owns 80% of the Certej Project through the company's' interest in Deva Gold S.A. (Deva Gold). The remaining 20% of Deva Gold is held by Minvest S.A. (Minvest), a Romanian state-owned mining company. Eldorado's interest in Deva Gold was acquired with the acquisition of European Goldfields Ltd. (EGL), completed in February 2012.

Eldorado has prepared this technical report to support updates to the mineral resources, mineral reserves and the extraction process. The information in the report was obtained from an expanded exploration program carried out in 2012 and 2013. Additional metallurgical testwork completed in 2013 and an updated pre-feasibility study was completed in the first quarter of 2014.

The information and data included in this technical study were prepared in accordance with the requirements as defined in National Instrument 43-101, standards of disclosure for Mineral Projects.

The Qualified Persons responsible for preparing this Technical Report as defined in National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects and in compliance with 43-101F1 (the "Technical Report") are Richard Alexander, P. Eng., Stephen Juras, Ph.D., P.Geo., Richard Miller, P.Eng., and Robin Kalanchey, P.Eng.. All are employees of Eldorado.

1.2 PROJECT DESCRIPTION

The Certej Project is an epithermal gold-silver deposit located in the southern part of the Apuseni Mountains in central Romania lying 400 km west of the Romanian capital city of Bucharest, 163 km south of Cluj-Napoca, 160 km east of the regional center of Timişoara and 12 km Northeast of Deva.

Eldorado Gold's subsidiary Deva Gold holds a mining licence for Certej granted in 1999 for 20 years, with the right to extend for five-year periods until the reserve is exhausted. The reserve associated with the license was updated in 2008, authorizing mining to commence once all other required permits are received.

The region around the Certej site is a historically significant mining area with exploration, mining, and processing, dating back to the 18th century. Mining and processing continued until as recently as 2006, when the state mining enterprise Minvest closed down as part of Romania's accession to the EU. The previous development left an open pit, underground development workings, and waste rock dumps. Under the terms of the mining licence, Deva

Gold is not liable for any previously incurred environmental damage; environmental rehabilitation for previous development will be funded by the Romanian Ministry of Finance.

The following key permits have been secured by Deva Gold for the Project:

- Environmental permit for the Zonal Urbanization Plan;
- Approval of Zonal Urbanization Plan; and
- Environmental permit for the development of the Project.

These permits allow Deva Gold to proceed with obtaining the Construction Permit and Forestry Permit necessary to begin development of the Project, following any updates and addendums to the existing permits based on design revisions.

The Project site is located at approximately 600 meters above sea level in an area that is predominantly hilly with incised valleys. The climate is considered mild temperate- continental with minimum temperatures of -4°C in the winter and maximum temperatures of +28°C in the summer.

The Certej Project is accessed by an all-season paved two-lane road from Deva. The National Highway and Rail System passing through Deva are important transportation routes for goods from major regional centers from within Romania, the Black Sea, and European centers.

In 2006, Minvest's Certej operation shut down as part of Romania's EU accession. As a result, skilled mining labour is readily available in Certej and the surrounding villages. These villages and centres in the county of Hunedoara will benefit with employment opportunities and the flow on effect of investment in the Project.

The Certej Gold Project is ideally located to take advantage of the region's highly developed infrastructure, in close proximity to the site, including:

- Power available within 14 km, from the national grid;
- Fresh water available within 10 km, from the Mures River;
- Fuel and lubrication bulk supply in Deva;
- Limestone available by quarrying within 13 km;
- Processing plant sites immediately east and south of the open pit;
- Waste rock dumps immediately north of the open pit; and
- Tailings facilities within close proximity to the processing plant sites.

The Project is located in a region with a favorable attitude to mining and that is seeking investment. The villages and cities in the county of Hunedoara are a source of skilled mining labour and the region will benefit with employment opportunities and the flow on effect of investment in the project.

1.3 DRILLING, SAMPLING AND ANALYSES

Diamond drill holes have been the principal source of the geological and grade data for the Certej Project. The drill holes, especially the 2012 and 2013 drilling programs, targeted extension of mineralization at depth and laterally and resource delineation of known areas of mineralization. The 2012 and 2013 campaigns had over 70,000 m of diamond drill core drilled in 236 drill holes. Drilling was done by wireline method with mainly HQ size core. The core was photographed before being sampled. Core recovery was excellent, averaging 95% to 100%. Sampling entailed sawing the entire hole by diamond saw and sending half the core to the laboratory. The remaining half core is kept in storage facilities on site.

The core samples are prepared and assayed at the Gura Rosiei ALS laboratory in Romania. The sample batches are arranged to contain regularly inserted control samples. Insertion rate for Standard Reference Material (SRM) samples and duplicate samples are about 1 in 20 submitted samples whereas blank samples are inserted at a rate of 1 in 40 submitted samples. All samples are assayed for gold by 30 g fire assay with an atomic absorption (AA) finish (gravimetric finish re-assay for samples >10 g/t Au) and for multi-element determination using digestion and inductively coupled plasma (ICP) analysis.

Monitoring of the quality control samples showed all data were in control throughout the preparation and analytical processes. In Eldorado's opinion, the QA/QC results demonstrate that the Certej deposit assay database, particularly for new data obtained from 2012 to 2013, is sufficiently accurate and precise for resource estimation.

1.4 MINERAL PROCESSING & GOLD RECOVERY

ROM Ore mined from the Certej deposit is comprised of refractory gold and silver-bearing pyrite mineralization, hosted in an andesite (silicate) lattice. There is very little gravity recoverable, or free milling, gold present in the Certej ores and given the refractory nature of the gold, recovery of the precious metal values by direct treatment with cyanide is largely ineffective. The Certej ores, therefore, first require an oxidative pre-treatment step to liberate the gold values from the pyrite mineralization such that the gold and silver can be recovered by conventional cyanidation. A number of oxidative pre-treatment processes for treatment of refractory gold bearing ores have been developed and in some cases commercialized; for the Certej materials, the pressure oxidation process was demonstrated to be highly effective.

The Certej ores will be treated at a rate of 3.0 Mt/a and comminuted by crushing, followed by a combination of SAG and ball milling. The ground ore will be subjected to simple rougher-scavenger and single cleaner flotation to separate the gold-bearing sulphide fraction (pyritic) of the ore from the host andesite (silicate). Following regrind, flotation concentrate is subjected to pressure oxidation. The oxidized solids will be treated with limestone and lime at elevated temperatures to facilitate silver recovery, prior to conventional precious metal recovery by carbon-in-leach cyanidation, carbon stripping and electrowinning. CIL tails will be subjected to further oxidative treatment to ensure that residual cyanide is destroyed before the tails are impounded.

The Certej project will produce 135 000 oz/a Au and 800 000 oz/a Ag, as a gold-silver doré, assaying roughly 15% Au and 85 % Ag. Overall recovery of the precious metal values from the Certej ore via the pressure oxidation process will be 87.4% and 80% for gold and silver, respectively.

1.5 MINERAL RESOURCE ESTIMATES

Eldorado used significant new drill data from its 2012 and 2013 drill campaigns to update the geologic and structural models. These models were used to guide and validate the mineralized shells created for the grade interpolation. Most significant were a much better understanding of the deposit bounding and, in areas, mineral hosting andesite intrusive units and a better definition between the Cretaceous and Neogene sediments, in particular the respective Breccia units found near or at their contact.

Eldorado created 3D mineralized envelopes, or shells, to constrain gold grade interpolation. These were based on initial outlines derived by a method of probability-assisted constrained kriging (PACK). The threshold value was 0.20 g/t Au and shell selection was done by inspecting contoured probability values. Extreme gold grades were shown to have a measure of risk to the grade estimate. To mitigate this risk, gold grades were capped to 70 g/t in the assay data prior to compositing. In addition, during grade interpolation a further restriction of 45 g/t over 40m was instituted where grades over this threshold were not used when further than 40m from a model block centre.

Block model cell size was 10 m east x 10 m north x 5m high. Modelling consisted of grade interpolation by ordinary kriging (OK). A two-pass approach was instituted for interpolation. The first pass required a minimum of two holes from the same estimation domain whereas the second pass allowed a single hole to place a grade estimate in any uninterpolated block from the first pass. Ag grade was also interpolated into the model. The model was validated by visual inspection, checked for bias and for appropriate grade smoothing.

The mineral resources of the Certej Project were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified into Measured, Indicated, and Inferred mineral resource categories. The Certej mineral resources, as of 31 December 2013, are reported at a 0.7 g/t Au cut-off grade and are shown in Table 1-1.

Table 1-1: Certej Mineral Resources, as of 31 December 2013

Mineral Resource Category	Tonnage (t x 000)	Grade (Au g/t)	In Situ Gold (oz x 000)	Grade (Ag g/t)	In Situ Silver (oz x 000)
Measured	25,680	1.75	1,448	9	7,150
Indicated	85,435	1.23	3,368	9	24,611
Measured+Indicated	111,115	1.35	4,816	9	31,761
Inferred	29,002	1.08	1,010	6	5,268

1.6 MINERAL RESERVE ESTIMATES

1.6.1 Mineral Reserve Estimates

The Mineral Reserve estimate has been calculated and classified in compliance to the CIM definition of Mineral Reserve. Only blocks classified as Measured or Indicated in the December 31, 2013, Block Model prepared by Eldorado Gold were used in the pit optimisation and reserve reporting.

The Mineral Reserves are the Measured and Indicated Resource blocks that are within the reserve pit design and above the ore cut-off grade. A total Proven and Probable Mineral Reserves estimate of 46.984 Mt at a grade of 1.63 g/t Au and 11 g/t Ag was calculated, as shown in Table 1-2.

Table 1-2: Certej Mineral Reserves Estimates

Reserve	Tonnage (t x 000)	Au (g/t)	Au (oz x 000)	Ag (g/t)	Ag (oz x 000)
Proven	20,441	1.91	1,255	10	6,283
Probable	26,543	1.41	1,203	12	9,967
Proven+Probable	46,984	1.63	2,458	11	16,250

1.6.2 Pit Optimization and Design

The open pit optimisation was performed using the Lerchs-Grossman (L-G) algorithm. The most financially beneficial shell from a series of 40 optimised pit shells that were generated formed the basis for the reserve pit, which was created and scheduled using all relevant design criteria. The final pit design incorporates the ramp grade, ramp width, bench heights, bench face angles, bench stack heights, berm width, geotechnical berm width and inter-ramp angle.

The operating plan has an elevated mill feed cut-off grade during the mining phase followed by treating stockpiled lower grade ore later on. The upfront revenue of this more than offset the higher upfront capital and operating costs. During the mining phase, a mill feed cut-off of 1.20 g/t Au_Equiv will be used with proven and probable ore between 0.90 and 1.20 g/t Au_Equiv being stockpiled for later treatment.

The mine plan is based on the reserve pit design and mineral reserves. To support the Technical Study all periods up to and including Year 6 have been detailed to quarterly schedules and thereafter yearly. The open pit mine is scheduled to run for a total of 13 years with the first two years as pre-production. The pre-production period is to gain sufficient ore exposure and build a starter ore stockpile. Suitable waste mined in the pre-production period will also be selectively used for construction needs.

Through direct feed from mining and by stockpile reclaiming, a constant mill feed of 3.0 million tonnes per annum will be maintained for nearly 15.7 years of which 11 years of mill feed is obtained from open pit production and the last 4.7 years of mill feed is from stockpiled low-grade ore. The low-grade ore (between 0.90 and 1.20 g/t Au_Equiv) will be placed in either of two dedicated low-grade stockpiles. The low-grade stockpiles will reach a maximum capacity of close to 15 Mt at the time at which the open pit production ceases.

The reserve pit also has approximately 113.4 Mt of waste and an overall stripping ratio of 2.41:1 (waste tonnes to ore tonnes). Waste rock from the open pit mining will be used in pre-production construction fill and the TMF embankment. All surplus waste rock will be placed into 2 waste dumps north of the open pit.

1.6.3 Mining Methods

All mining at Certej is planned to be performed by owner equipment and personnel using conventional open pit mining methods with drill and blast of the rock followed by load and haul with diesel powered off road shovel and truck equipment. The number of units for the mining fleet has been calculated by the production schedule, equipment capacities, mechanical availability, utilization, scheduled hours and cycle times, which include detailed haulage profiles for various materials, benches, phases and periods. Lube, fuel, labour, tires, wear parts, and maintenance inclusive of major overhauls were calculated for all equipment.

1.7 PROJECT ECONOMICS

Capital and Operating Costs for the Project were developed within the accuracy level of a Pre-Feasibilility study as defined by industry standards. The Capital Cost Estimate included Initial, Sustaining and Closure costs. All estimated costs were defined within the Project Work Breakdown Structure.

Capital cost estimates were developed to a high level of accuracy with mobile and plant equipment budget quotations obtained, labour cost built up from first principles of salaries, burdens and overhead and unit rates verified by Romanian contractor quotations. Contingency was applied to each item based on the source and accuracy of the estimate data resulting in an overall Initial and Sustaining Contingency of 17% and 24% respectively. The Capital Cost Summary is included in Table 1-3.

Table 1-3: Capital Cost Estimates

Area	Description	Initial [US$ x 1000]	Sustaining [US$ x 1000]	Closure [US$ x 1000]
A	Overall Site	15,362	7,618	11,817
B	Mine	84,402	64,111	-
C	Crushing	12,157	-	-
D	Process Plant (Concentrator)	59,031	8,000	-
E	POX	99,750	21,000	-
F	CIL	17,376	-	-
G	Tailings	23,586	73,803	-
H	Infrastructure	10,206	1,170	-
J	Ancillary Facilities	7,166	-	-
K	Off Site Infrastructure	30,501	4,000	-
	Direct	**359,537**	**179,701**	-
	Indirects	**88,374**	**15,228**	-
	Owner's Cost	**11,960**	**-**	-
	Contingency	**79,276**	**45,873**	**5,908**
	Total Installed Cost	**539,147**	**240,802**	**17,725**

Operating costs were developed from first principles, inclusive of labour, consumables, fleet maintenance and repair, and general and administration. The Operating Cost Summary is included in Table 1-4.

Table 1-4: Operating Cost Estimates

Operating Costs	Units	LOM Average [US$]	LOM Expenditure [US$M]
Mining Cost (mined)	US$/t mined	1.87	
Mining Cost (ore)	US$/t ore	6.40	301
Processing Cost	US$/t ore	19.54	918
G&A	US$/t ore	2.79	131
Operating Cost	**US$/t ore**	**28.73**	**1,350**
Oxygen Plant Lease	US$/t ore	3.13	147
Total Operating Cost	**US$/t ore**	**31.86**	**1,497**

The Project economic analysis indicates a positive Net Present Value of US$81M and positive Internal Rate of Return of 8%, at US$1250/oz Au and US$16.50/oz Ag. Economic performance is highly dependent on gold price. There are significant opportunities to improve the economics, and several potential cost optimization strategies that may have a positive impact on the Project economics have been identified and will be subject to further study in the next phase of the project.

1.8 CONCLUSION AND RECOMMENDATIONS

The Certej project is an advanced stage development project that benefits from the following:

- Close proximity to major centers in Romania, important highway and rail transportation routes and a highly developed infrastructure, and local availability of experienced mining labour;
- Critical permits are in place for the project including the Environmental Permit;
- Mineral Resources and Reserves were classified using logic consistent with the CIM definitions referred to in NI 43-101 and are based on a substantial exploration program;
- The economic analysis indicates a positive IRR and NPV; and
- Mineral processing and gold recovery methods selected for the project are based on mature, proven technologies; pilot testing has demonstrated high gold and silver recoveries.
- Significant opportunities have been identified which have potential to increase the value of the Certej Project and enhance the corporation's positive position in the region.

Eldorado plans to advance the Project through a series of optimizations, upon completion of which, a Feasibility Study will be initiated.

SECTION • 2 INTRODUCTION AND TERMS OF REFERENCE

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, owns 80% of the Certej Project through the company's interest in Deva Gold S.A. The remaining 20% of Deva Gold is held by Minvest S.A. (Minvest), a Romanian state-owned mining company. Eldorado's interest in Deva Gold was acquired as part of the acquisition of European Goldfields completed in February 2012.

Eldorado prepared this Technical Report for the Certej Project to support updates to the mineral reserves, mineral resources, and the extraction process.

Information and data contained in, or used in the preparation of this report was obtained from the following sources:

- Information obtained from European Goldfields, reviewed and verified by Eldorado Gold;
- An expanded exploration drilling program undertaken in 2012 and 2013 by Deva Gold and directed by Eldorado Gold;
- Additional metallurgical testwork undertaken in 2013 by Eldorado Gold; and
- An updated pre-feasibility study prepared in the first quarter of 2014 by Eldorado Gold.

The Qualified Persons responsible for preparing this Technical Report as defined in National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects and in compliance with 43-101F1 (the "Technical Report") are Richard Alexander, P. Eng., Stephen Juras, Ph.D., P.Geo., Richard Miller, P.Eng., and Robin Kalanchey, P.Eng.. All are employees of Eldorado.

Mr. Alexander, Project Director, for the Company, was the Project Manager responsible for preparation of the Technical Study. He most recently visited the Certej Project Site on February 3 – 11, 2014.

Dr. Juras, Director, Technical Services, for the Company, was responsible for the preparation of the sections in this report that concerned geological information, sample preparation and analyses and mineral resource estimation. He most recently visited the Certej Project Site on September 15 – 17, 2013.

Mr. Miller, Manager, Mining, for the Company, was responsible for the preparation of the sections in this report that dealt with mineral reserves estimation, and mine operations. He most recently visited the Certej Project Site on October 10 – 11, 2013.

Mr. Kalanchey, Group Metallurgist, for the Company, was responsible for the preparation of the sections in this report that dealt with metallurgy and process operations. As of the Effective Date, he had not visited the project site.

SECTION • 3 RELIANCE ON OTHER EXPERTS

Eldorado prepared the mineral resources and reserves estimates in this report based on expanded exploration drilling, additional metallurgical testwork, and an updated pre-feasibility study.

Eldorado has not relied on third party experts with respect to legal, political, environmental or tax matters for the project.

Preparation of the Pre-Feasibility Study referenced in this report, did incorporate some work or services completed by third party experts, reviewed and incorporated by the authors of this report, including:

Golder Associates

- Tailings Management Facilities, Water Management and Treatment Design; Pit Geotechnical and Hydrogeological Studies.

Norwest Corporation

- Waste Dump and Stockpile Design.

Sherritt International Corporation

- Pressure Oxidation Process Design.

SECTION • 4 PROPERTY LOCATION AND DESCRIPTION

4.1 PROPERTY LOCATION

The Certej Project is located in the southern part of the Apuseni Mountains in central Romania shown in Figure 4-1, lying 400 km northwest of the Romanian capital city of Bucharest, 163 km south of Cluj-Napoca, and 160 km east of the regional center of Timişoara. The Project Site is approximately 12km NE of the closest city, Deva, in Hunedoara County.

Approximate geographic project co-ordinates are:

Latitude: ....45° 59' 47" N

Longitude: ....22° 59' 32" E

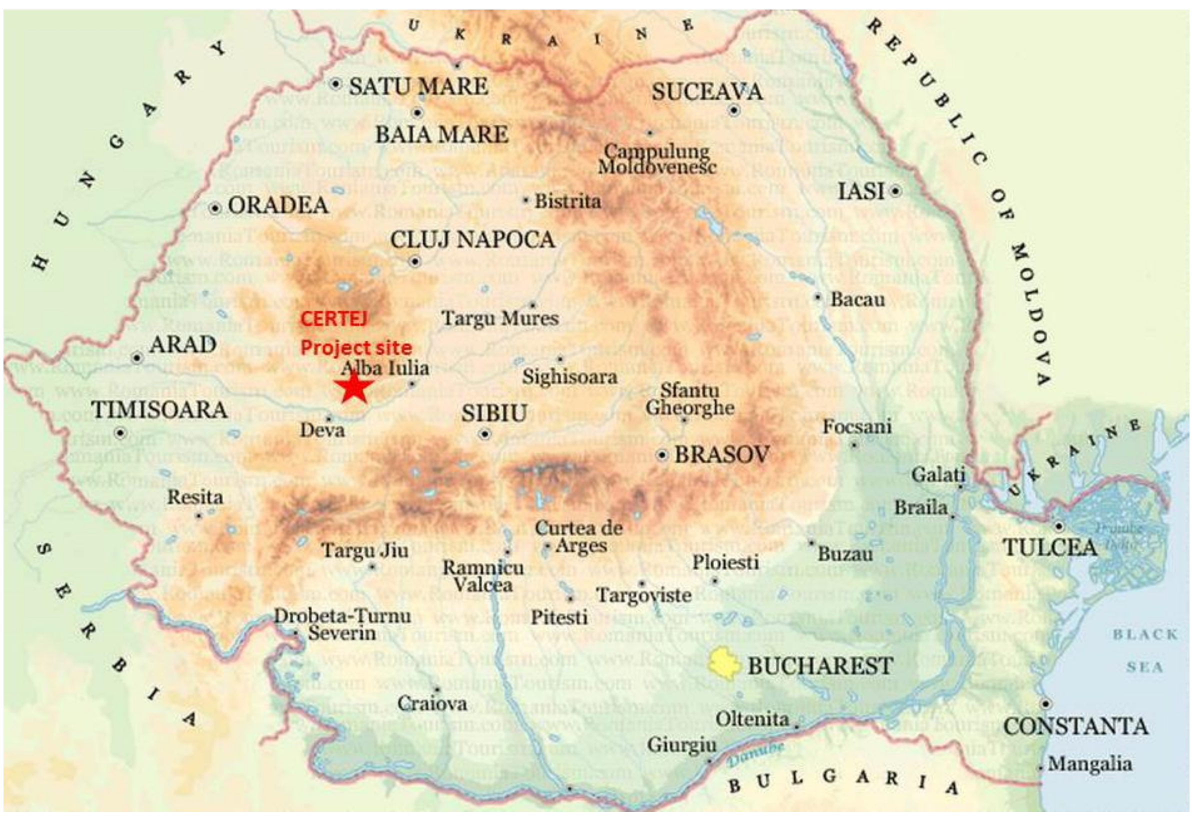


Figure 4-1: Project Location

4.2 LAND TENURE

Eldorado's subsidiary, Deva Gold, holds a mining licence for Certej granted in 1999 for 20 years, under the terms of the Romania Mining Law No. 61/1998. The licence holds the right to extend for five year periods until the reserve is exhausted. The reserve associated with this licence was updated in 2008 when Deva Gold submitted a technical study to the government, which included a mining schedule and resource calculation, in accordance with Romanian technical instructions. This was approved in September 2008, authorizing mining to commence once all other required permits are received.

The licence was granted in accordance with the Mining Law No. 61/1998 by the National Agency for Mineral Resources to Minvest as the “titleholder” and Deva Gold as the ”affiliated company”. This was further transferred in accordance with the provisions of the mining law and its application rules from Minvest to Deva Gold in December 2001. Thus, Deva Gold became “titleholder” and Minvest the “affiliated company”. Currently Minvest owns 20% of the project while Eldorado owns 80% of it, through Deva Gold.

The licence boundaries are shown in Figure 4-2 and defined in UTM and geographic coordinates in Table 4-1.

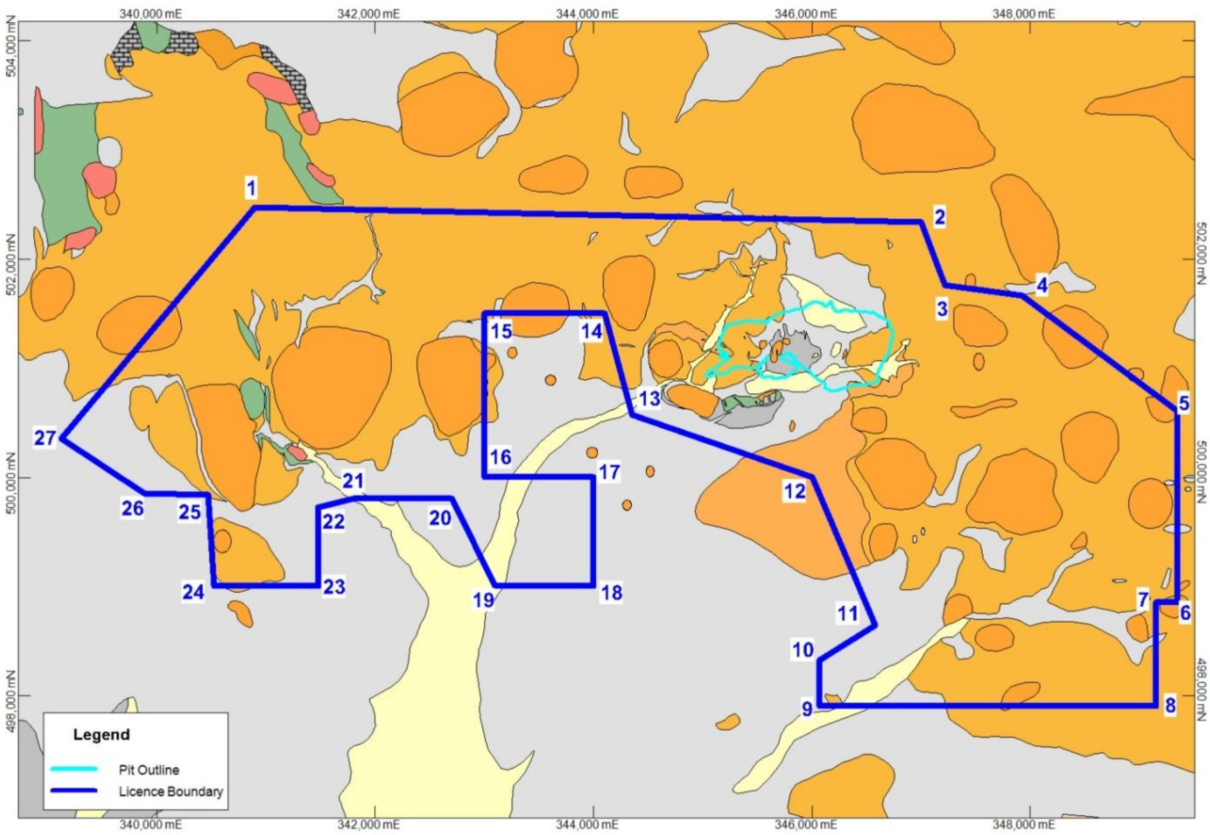


Figure 4-2: Land Position

Table 4-1: UTM and Geographic Coordinates

Number	UTM		Geographical Coordinates	
	X-N	Y-E	Lat-N	Long-E
1	5096267	650503	46.00.12,39	22.56.37,70
2	5096431	656602	46.00.12,79	23.01.21,31
3	5095871	656851	45.59.54,44	23.01.32,19
4	5095810	657557	45.59.51,90	23.02.04,92
5	5094823	659036	45.59.18,71	23.03.12,47
6	5093075	659124	45.58.22,04	23.03.14,46
7	5093065	658924	45.58.21,88	23.03.05,17
8	5092117	658972	45.57.51,12	23.03.06,25
9	5091962	655896	45.57.48,65	23.00.43,23
10	5092385	655875	45.58.02,38	23.00.42,74
11	5092728	656366	45.58.13,06	23.01.05,96
12	5094056	655720	45.58.56,60	23.00.37,54
13	5094540	654045	45.59.13,65	22.59.20,28
14	5095459	653748	45.59.43,62	22.59.07,53
15	5095403	652649	45.59.42,72	22.58.16,42
16	5093905	652724	45.58.54,15	22.58.18,20
17	5093955	653723	45.58.54,97	22.59.04,65
18	5092957	653773	45.58.22,59	22.59.05,82
19	5092912	652874	45.58.21,85	22.58.24,03
20	5093690	652442	45.58.47,40	22.58.04,83
21	5093645	651536	45.58.46,68	22.57.22,70
22	5093551	651218	45.58.43,89	22.57.07,84
23	5092830	651261	45.58.20,51	22.57.09,03
24	5092782	650301	45.58.19,71	22.56.24,35
25	5093612	650208	45.58.46,64	22.56.20,98
26	5093595	649623	45.58.46,57	22.55.53,80
27	5094063	648842	45.59.02,32	22.55.18,02

4.3 ROYALTIES

The project is subject to both mining royalties and mining fees.

Currently the Certej licence stipulates a royalty payment of 4% of net sales revenue, less transport and refining costs. As of January 1, 2014, legislation increased mining royalties to 6%; this increase is also expected to apply to the Certej licence. Royalty payments are applicable only once the project is in production.

Annual mining fees are calculated based on the area of land held under the licence, applicable whether the project is in production or not, in the total amount of 33,600 Lei/km2 (approximately 10,080 US$/$km^2$), categorized as follows:

- Annual fee for prospecting activity = 320 lei/km^2
- Annual fee for exploration activity = 1,280 lei/ km^2
- Annual fee for exploitation activity = 32,000 lei/ km^2

4.4 ENVIRONMENTAL LIABILITIES

Under the terms of the mining licence, the project is liable for future environmental damage arising from the project, however the project is not responsible for any previously incurred environmental damage within or outside of the project area. In accordance with the Romanian government ordinance number 644/2007 regarding the shutdown of Minvest operations, the associated environmental rehabilitation will be funded by the Ministry of Finance.

4.5 PERMITS AND AGREEMENTS

The company started the project permitting process in 2007. Since then the following key permits were secured:

- *Environmental permit for the Zonal Urbanization Plan (PUZ)*. Environmental permitting was conducted in accordance with the provisions of the Government Ordinance Number 1076/2004 regarding environmental permitting for plans and programs. This is also known as the ‘strategic environmental assessment’ (SEA) procedure.
- *Approval of Zonal Urbanization Plan*. Upon receipt of the Environmental Permit, the local council of the Certej village approved the Zonal Urbanization Plan for the industrial area where the project is located.
- *Environmental permit for the development of the Project.* Submission of an Environmental Impact Assessment (EIA) was completed in accordance with the provisions of GUO 195/2005 approved through Law 265/2006, as well as through related directives, ordinances, and laws. Upon completion of all steps required during the EIA process, the environmental authority granted the Environmental Permit for the project. This was a significant milestone in the process of obtaining the permits required in order to obtain the construction permit for the project. Receipt of the project Environmental Permit included receipt of a Water Management Permit, approval of the design for the TMF’s, and approval of the waste management plans.

The next permits required in order to initiate construction activities on site are:

- Deforestation permit
- Construction permit
- Environmental permit for the production phase (following completion of construction).

Permits will be maintained and updated in keeping with the latest understanding of environmental impacts and parameters. These environmental impacts and parameters will be updated in the next stages of project development to reflect the most up-to-date project designs and to ensure compliance with Romanian laws and regulations and with European directives in the field of the environmental protection.

SECTION • 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PHYSIOGRAPHY

Elevations within the project area range from 400 metres to over 1,100 metres above sea level, the hills around the deposit reaching a maximum of approximately 600 metres. Much of the area is hilly, with incised valleys. The hills are forested with beech and oak with occasional conifers, particularly at higher elevations. Scattered small rural settlements, associated pastures, and other agricultural land occur at lower elevations, on gentler slopes, and in the river valleys. The proposed site for the Certej project is comprised of a large area previously used for open pit mining with associated existing dumps, secondary and tertiary forest, and rough pasture.

5.2 CLIMATE

The climate is mild temperate-continental, with mean temperatures of approximately 23°C in summer and -3°C in winter. Snow cover can last until March and may be up to 80cm deep, with frost penetration of approximately 180 cm. Precipitation as rain and snow averages 562 mm annually with a minimum monthly average of 27.0 mm and a maximum monthly average of 72.4 mm.

Exploration and mining activities can continue throughout the year without delays due to heavy weather; there are no seasonal restrictions to site access.

5.3 ACCESSIBILITY

Access to the Certej Project is by National Roadway 761 from the city of Deva to the village of Certej de Sus, extending past the site as shown in Figure 5-1.

The city of Deva is served by the National Highway 76 and a railway line from Bucharest. The National Highway 76 and railway is a major transportation route for goods from the Black Sea to Europe. The National Highway and Rail system also connects Deva to Bucharest and several other regional centers such as Timişoara, Cluj and Arad.

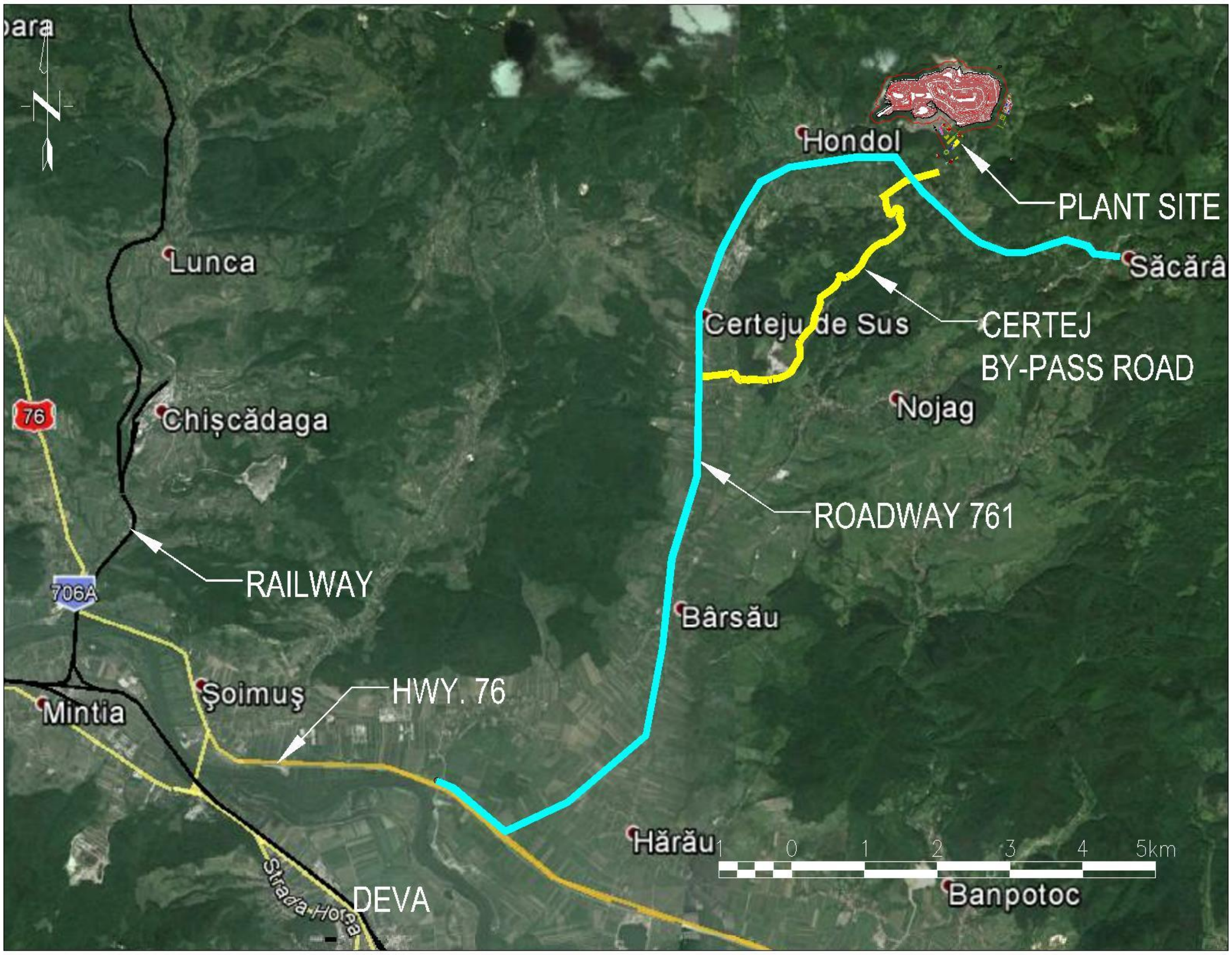


Figure 5-1: Access to the Certej Project

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

In 2006, Minvest's Certej operation was shut down as part of Romania's EU accession. As a result, skilled mining labour is readily available in the surrounding villages, due to a lack of nearby mining operations. In particular, the village of Certej de Sus, with population around 3,500, is located adjacent to the operation.

Deva, with a total population of 69,000, is the closest city to the mining operation and houses the vast majority of the previous mine's national workforce, providing an additional source of skilled labour.

These villages and centres in the county of Hunedoara will benefit with employment opportunities and the flow on effect of investment in the project.

5.5 INFRASTRUCTURE

The Certej Gold Project is ideally located to take advantage of the region's highly developed infrastructure, in close proximity to the site, including:

- Major highway and rail moves through Deva
- Power available within 14 km from the national grid
- Fresh water available within 10 km from the Mures River
- Fuel and lubrication bulk supply in Deva
- Limestone available by quarrying within 13km
- Processing plant sites immediately east and south of the open pit
- Waste rock dumps immediately north of the open pit
- Tailings facilities within close proximity to the processing plant sites

A detailed discussion of the available infrastructure is provided in Section 18.

SECTION • 6 HISTORY

6.1 HISTORICAL MINING AND EXPLORATION ACTIVITIES

Gold mining in the southern Apuseni Mountains dates back more than two millennia. Historical information on the Certej Deposit itself dates back to the 17th century.

6.1.1 Mining

Prior to the 18th century, mining development in the area was performed by artisanal miners and dwellers of the surrounding villages that were involved in small scale surface, underground and alluvial gold mining using rudimentary mining methods.

There is visible evidence of large scale surface and underground mining operations on the Certej deposit beginning in the first half of the 18th century by organized companies and groups, the most notable being the exploitation of the nearby Sacaramb underground mine which started in 1747. For over 100 years the Sacaramb mine was considered the most profitable in Europe, with over 40 tonnes of gold being extracted from the deposit. In 1763, due to the intense mining activity in the area, a gold processing facility was commissioned and was operated until 1882.

Between 1882 and 1917, the Hondol vein system at the Certej Deposit was exploited by a local Sacaramb mining entrepreneur. After the union of Transylvania with Romania in 1918, the local operators in the region worked together with the state-owned companies and continued to develop and operate mines in the area. The mining works during this period resulted in large waste dumps being formed near the portals of the underground drives. In 1935 the Humbold Company built a cableway to Certej and a flotation process plant at Certej in order to process these dumps and produce gold and base metal concentrates.

Between 1975 and 1977 the flotation plant in Certej was expanded by installing a new processing line giving a capacity of approximately 90,000 tonnes per annum (tpa). In 1982 the operation focused on base metals, including lead (Pb) and zinc (Zn) mining, and a new concentrator was constructed at Certej with a throughput of approximately 1,000,000 tpa.

In 1983 the Minvest-owned Certej Mine included the Baiaga-Hondol deposit from IPEG-Deva and the exploration and pre-stripping work of the deposit continued. From 1985 to 1988 the Certej mine (then called the Coranda Mine) produced between 500,000 and 7000,000 tpa of ore. In January 2006, Minvest closed its mining and processing operations at the Coranda open pit and Certej town as part of Romania’s European Union accession requirements.

Although it has not been possible to obtain complete records, it is estimated that the total ore excavated between 1985 to 2006 from the Coranda open pit was 9.2 million tonnes (Mt), with no records being available for 1984 and 1985. Based on this information the total production of ore from the Coranda open pit over its operational life is estimated at 10 Mt.

6.1.2 Exploration

There is no reliable information available on exploration activities on the Certej site prior to the 1960's.

The exploration work conducted on the Certej deposit from the 1960's to 2000 was restricted to underground exploration designed to identify zones of gold-silver and base metal ores. The exploration work during this period was performed by the state owned company I.P.E.G. Hunedoara-Deva and consisted of the following key methods:

- Underground development (adits, drives and cross-cuts).
- Underground diamond and short hole rotary drilling.
- Channel sampling of existing and new underground development.
- Surface rock chip sampling.
- Limited surface diamond drilling.

The Certej deposit includes (from west to east) the old deposits known as: Hondol-Carol, Hondol-Băiaga, Hondol-Coranda and Dealu Grozii, which was part of Certej state owned mine. Figure 6-1 shows the location of these deposits and the extent of historical exploration, in relation to the current project pit and boundaries.

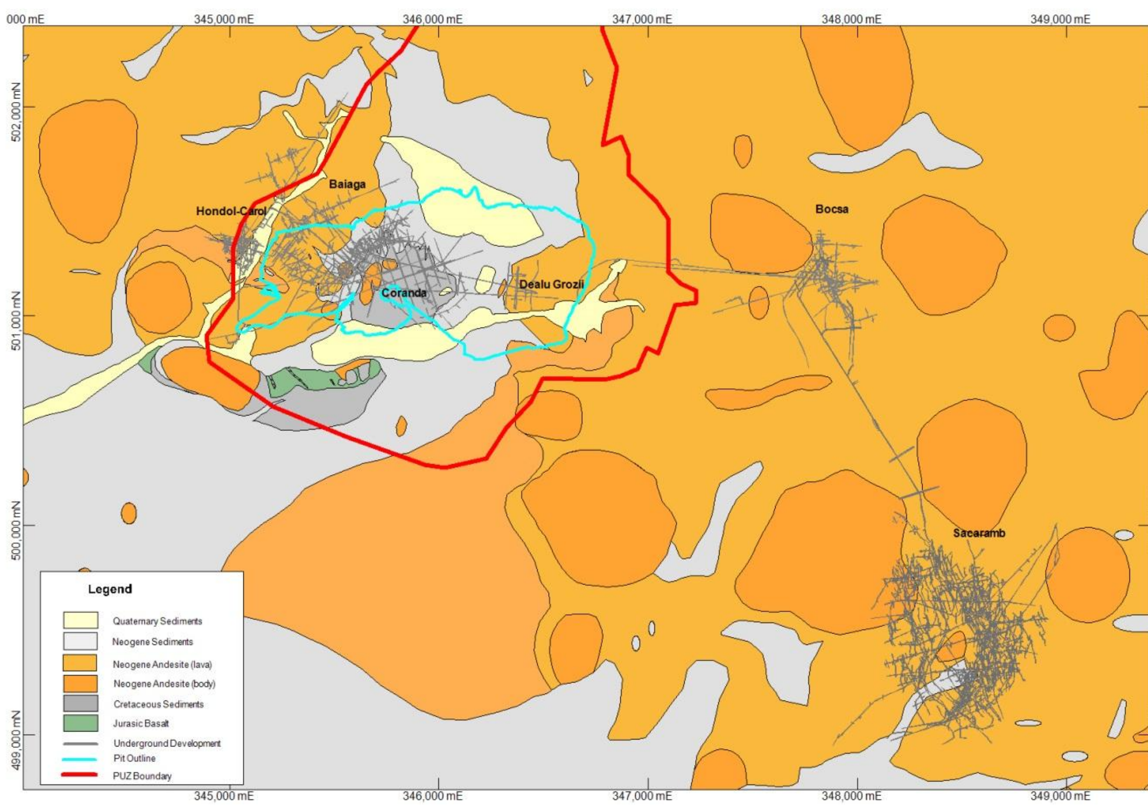


Figure 6-1: Regional Deposits and Historical Exploration

In the 1960's the Hondol-Carol deposit was investigated through the Carol shaft which, after being flooded by Macris creek, was abandoned.

Later, the Coranda-Hondol deposit was investigated through the Hondol shaft with approximately 2770 m of exploration drifts and raises.

From 1963 to 1975, downward exploration of the Băiaga – Hondol deposit was investigated through underground development and diamond drilling; the works included downward development beneath the +403 meter level through the Hondol for 200 meter depth with exploration drifts at 50 meter increments with approximately 7400 meters of exploration development and 2186 meters of underground diamond drilling.

The Coranda-Baiaga-Hondol deposit was investigated through Baiaga intrusive between 1973 and 1983 through underground development through Coranda I, Coranda II, Coranda III, and Coranda IV galleries with approximately 11,100 meters of exploration development and 3200 meters of underground diamond drilling.

Beginning in 1983, upward exploration of the Băiaga – Hondol deposit above the +403 level commenced through the Nicodim, Coranda II, Coranda III and Coranda IV galleries with approximately 3,500 meters of exploration development.

After 1991, underground exploration development and diamond drilling of the Dealu Grozii deposit was undertaken from the Nicodim, Coranda II and Dealu Grozii galleries with approximately 2370 meters of exploration development.

None of the historical exploration information described above has been used in the estimation of resources due to the lack of detailed information. This exploration is described for information only; the current resource and reserve estimates rely entirely on recent exploration programs.

6.2 RECENT MINING AND EXPLORATION ACTIVITIES

6.2.1 Mining

There has been no recent mining activity on the Certej site since Minvest ceased operations in 2006.

6.2.2 Exploration

Deva Gold acquired the Certej project in 1999 and completed a surface and underground channel sampling program. European Goldfields began an exploration program in 2000 comprised of surface and underground channel sampling and reverse circulation and diamond drilling on nominal 80 meter spacing. An independent estimate of resources was made in 2002 based on the European Goldfield exploration program by consultants RSG Global in Perth of 34.7Mt at 2.1g/t Au and 10 g/t Ag at 1.0 g/t cut-off grade.

In November 2004, a drilling campaign by European Goldfields commenced on the deposit and continued into 2005 with an emphasis on tighter drill spacing in higher grade areas. In places where the drill spacing was reduced to 20 meters, results indicated that the SMU model had correctly predicted where larger parent blocks contained high grade sub-blocks and so the approach was applied again in the 2005 estimate. This drilling led to a revised independent estimate of resources in 2005 by consultants RSG Global in Perth of 31.35Mt at 2.1g/t Au and 11 g/t Ag at 1.0 g/t cut-off grade.

In 2007, an infill drilling program was designed to convert inferred material to the indicated category. Drilling was completed and a new resource was calculated by RSG (Coffey Mining) and a new pit optimization completed the revised resource estimate of 41.5 Mt @ 2 g/t Au and 11 g/t Ag at 0.8 g/t cut-off grade.

From 2000 to 2007, European Goldfields completed 22,376 meters of channel sampling and 60,436 meters of reverse circulation and diamond drilling.

From 2008 to 2011, European Goldfields completed 11,400 m of reverse circulation and diamond drilling especially for geotechnical design, waste dump evaluation, and sterilization purposes.

A summary of the recent exploration work is included in Table 6-1.

Table 6-1: Summary of Recent Exploration

Year	Supervision	Exploration Activities
1999	Deva Gold	• Surface Channel Sampling
2000-2003	Deva Gold & RSG	• Surface and UG Channel Sampling and mapping; • Surface drilling, Aeromagnetic data capture; • Completion of Surface and UG sampling 80m; • Spaced surface drilling program; • Surface Geochemical sampling program; • DGPS Control Point Survey, and • Satellite deposit exploration.
2003	Deva Gold	• Infill surface drilling program 40m spacing, and • Satellite deposit exploration.
2004-2006	Deva Gold	• Surface channel sampling to define extensions to mineralization; • Nominal 20m spaced surface drilling program in high grade areas; • Further infill drilling for metallurgy and upgrading of Inferred to Indicated, and • Geotechnical drilling and testwork.
2007	Deva Gold	• Infill drilling of Inferred resources.
2008-2009	Deva Gold	• Geotechnical drilling and testwork for facilities, and • Drilling and Surface Channel Sampling for the Existing Waste dumps exploration.
2010-2011	Deva Gold	• Further infill drilling for upgrading the resources, and • Further drilling for metallurgy test work, geotechnical and sterilization.
2012-2013	Deva Gold	• Further infill drilling for upgrading the resources; • Further drilling for metallurgy and test work; • Geotechnical drilling and testwork for open pit and facilities; • New highly detailed landsat imagery and laser topographic survey, and • Systematic multi-element geochemistry including ICP41 and MS 61.

6.2.3 Exploration of Existing Waste Dumps

Exploration over the Coranda dumps took place between 2004 and 2005, targeting higher-grade oxide material and mineralized Cretaceous sediment. A second phase of systematic surface and RC drilling sampling took place between 2008 and 2011, covering the remaining accessible dump areas. This work included a total of 4439 meters of vertical RC drill holes, and 7,438 meters of surface channeling.

European Goldfields estimated the resources for the existing Coranda waste dumps at 7.02 Mt @ 0.53 g/t Au and 8.87 g/t Ag.

The resources included in the existing waste dumps are not included in the current estimate of resources completed by Eldorado Gold and are discussed here for reference only.

6.2.4 Previous Resource & Reserve Estimates

In February 2009, European Goldfields released a Technical Report based on the Albion process, declaring both mineral resource and mineral reserves estimates of:

- Resource: 41.5 Mt @ 2 g/t Au and 11 g/t Ag at 0.8 g/t cut-off; and
- Reserve: 32.8 Mt at 2g/t Au and 11g/t Ag at $650/oz Au and $7.5/oz Ag.

There have been no other significant mineral resource or mineral reserve estimates published in accordance with Section 2.4 of the National Instrument 43-101.

SECTION • 7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Certej deposit is located in the South Apuseni Mountains of Transylvania in Romania, within the historic ‘Golden Quadrilateral’, an area that has produced between 1,000 and 2,000 t of Au since pre-Roman times as shown in Figure 7-1. Gold was mined primarily from high grade epithermal veins that are associated with Miocene andesitic volcanic and sub-volcanic rocks that were emplaced into three NW-SE aligned belts; Rosia Montana-Bucium in the north, Zlatna-Stanija in the centre and Sacaramb-Certej-Brad, in the south.

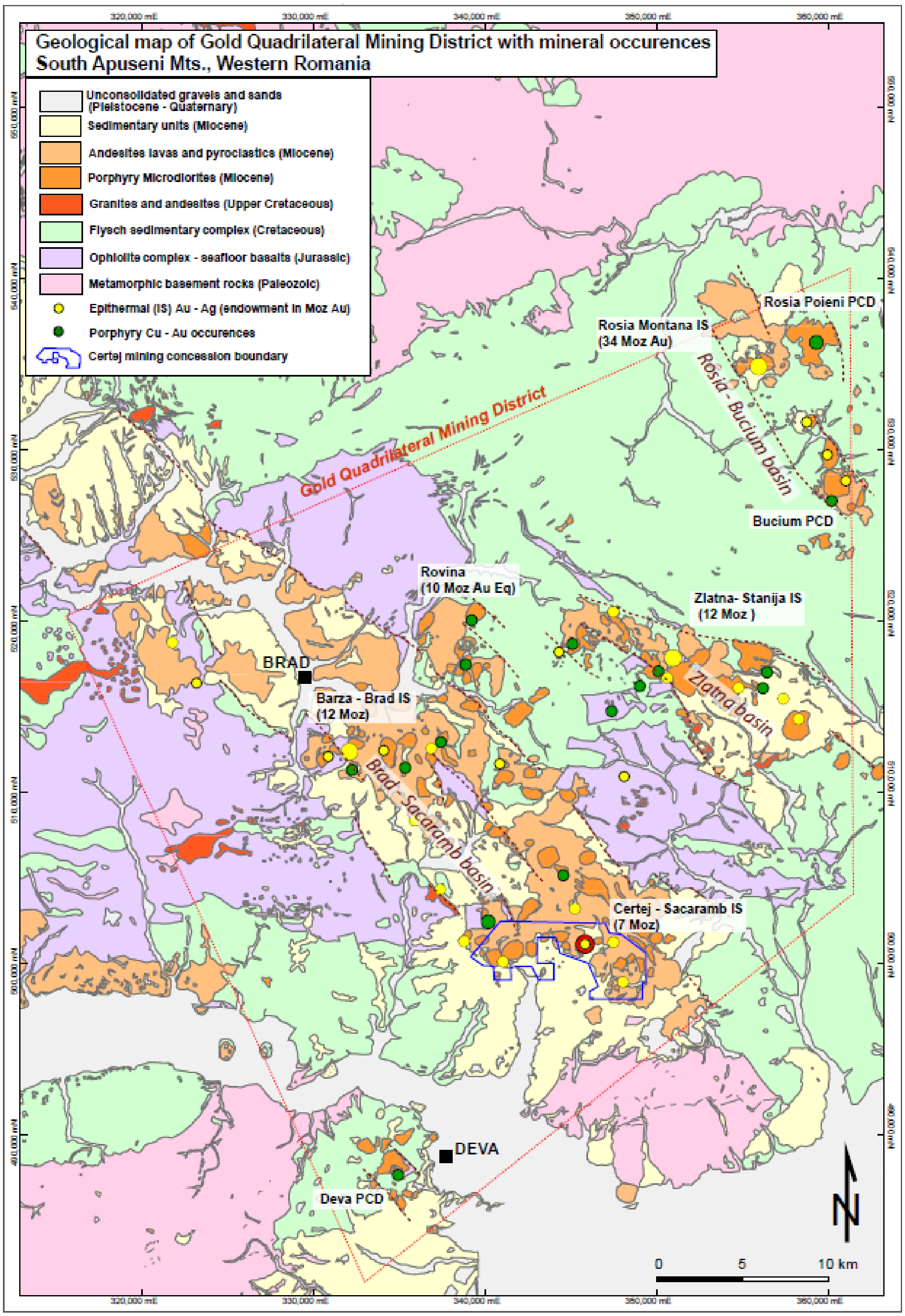


Figure 7-1: Apuseni Camp Map

The geological formations of the South Apuseni Mountains were formed by complex interactions between various microplates squeezed between the African and European tectonic plates. Lithologies include highly deformed pre-Alpine basement metamorphic rocks overlain by late Carboniferous to Palaeogene shallow marine and non-marine sedimentary rocks, which were subsequently covered by Neogene fill of the Pannonian and Transylvanian basins.

During the late Cretaceous north-directed thrust faulting resulted in a series of nappes, and collision continued into the Paleogene driven by north-directed subduction. By the late Oligocene to early Miocene collision had ceased coinciding with slab breakoff of subducted continental lithosphere. Microplate rotation initiated in the late Oligocene and continued through the Neogene driven by north-eastward slab roll-back and retreat of oceanic lithosphere attached to the European continental plate. Seismic tomography appears to show a local break in the lithosphere beneath the Apuseni Mountains that has been interpreted as a slab tear and which may have triggered magmatism. The magmatism occurs at least 200 km behind the main East Carpathian volcanic arc and is interpreted to be controlled by back-arc, graben style extension in the overlying plate resulting in the emplacement of numerous Miocene calc-alkaline volcano-intrusive complexes.

The three NW-SE trending volcanic belts of the South Apuseni are the manifestation of this graben controlled volcanic activity and control the distribution of mineralization in the district. There is a close temporal and spatial relationship between epithermal gold deposits (e.g. Rosia Montana, Baia de Aries, Barza, Certej) and porphyry copper-gold deposits (Rosia Poieni, Rovina) within each of the volcanic belts. Magmatism in this region ranges in age from 14.7 to 7.4 Ma, and although the magmatic rocks are dominantly andesite there is a distinct evolution in their geochemistry that reflects the evolving tectonic setting and magma source. The earliest magmatic rocks are calc-alkaline (non-adakitic) and were emplaced between 14.7 and 12 Ma, followed by magmatic rocks that have a more adakite-like calc-alkaline character emplaced from approximately 12.5 Ma to 10 Ma. The younger andesites (10 Ma to 7.4 Ma) in the region are alkaline. This evolution infers that the magmas were initially sourced from the crust-lithospheric mantle boundary and later sourced from uprising asthenosphere. The lithospheric source was particularly rich in fluids likely due to the earlier subduction processes, and the subsequent rollback, rotation and extension facilitated decompressional melting and generation of significant volumes of fertile, hydrous, oxidized intrusions. The intrusions most closely associated with mineralization (12 to 9 Ma) are mainly porphyritic andesite with favourable geochemical indicators (high Mg number, high Sr/Y ratio, elevated Sr and Ba, and depleted Y and Yb).

7.2 LOCAL AND DEPOSIT GEOLOGY

The Certej deposit is located at the SE extremity of the Brad-Sacaramb volcanic belt that is a NW-SE trending graben approximately 20 kilometres long and 10 kilometres wide. The basement in the Certej area on either side of the graben is basalt of Jurassic age. The basalts contain a spillitic alteration assemblage of carbonate, albite and chlorite. Cretaceous sedimentary rocks within the graben comprise sandstone and siltstone intercalated with black shales and is interpreted to be Barremian-Aptian (130 to 120 Ma) in age. Neogene conglomerates, grits, marls and grey shales overlie the Cretaceous sedimentary rocks. The Certej licence is dominated by extrusive and intrusive andesite. Andesite stocks throughout the area are commonly propylitically altered and surrounded by andesite flows, which blanket the surrounding Neogene sedimentary rocks. The SW quarter of the tenement comprises a thicker sequence of Neogene sedimentary rocks.

In the Certej deposit three main andesite phases are recognised, namely Hondol, Dealu Grozii (part of the larger Sacaramb andesite), and Baiaga and are shown in Figure 7-2. K-Ar dating indicates that they were emplaced at around 11 Ma, however, geological relationships suggest Baiaga is the youngest phase. The three andesites are broadly similar in composition but the Dealu Grozii/Sacaramb andesite typically contains biotite and is coarse grained, whereas the younger Baiaga andesite is fine grained. The Dealu Grozii and Hondol andesite occur respectively on the eastern and western edges of the main Certej deposit and the Baiaga andesite is located in the centre of the deposit and at depth, and is flanked by Cretaceous and Neogene sedimentary rocks shown in Figure 7-3. Silvers of ophiolitic rocks (dominantly altered basalt) are localized along fault and breccia zones within the Cretaceous sedimentary package. The Baiaga andesite is interpreted to have intruded up into the sedimentary and volcanic rocks, possibly controlled by an E-W fault, resulting in upward buckling of the overlying package.

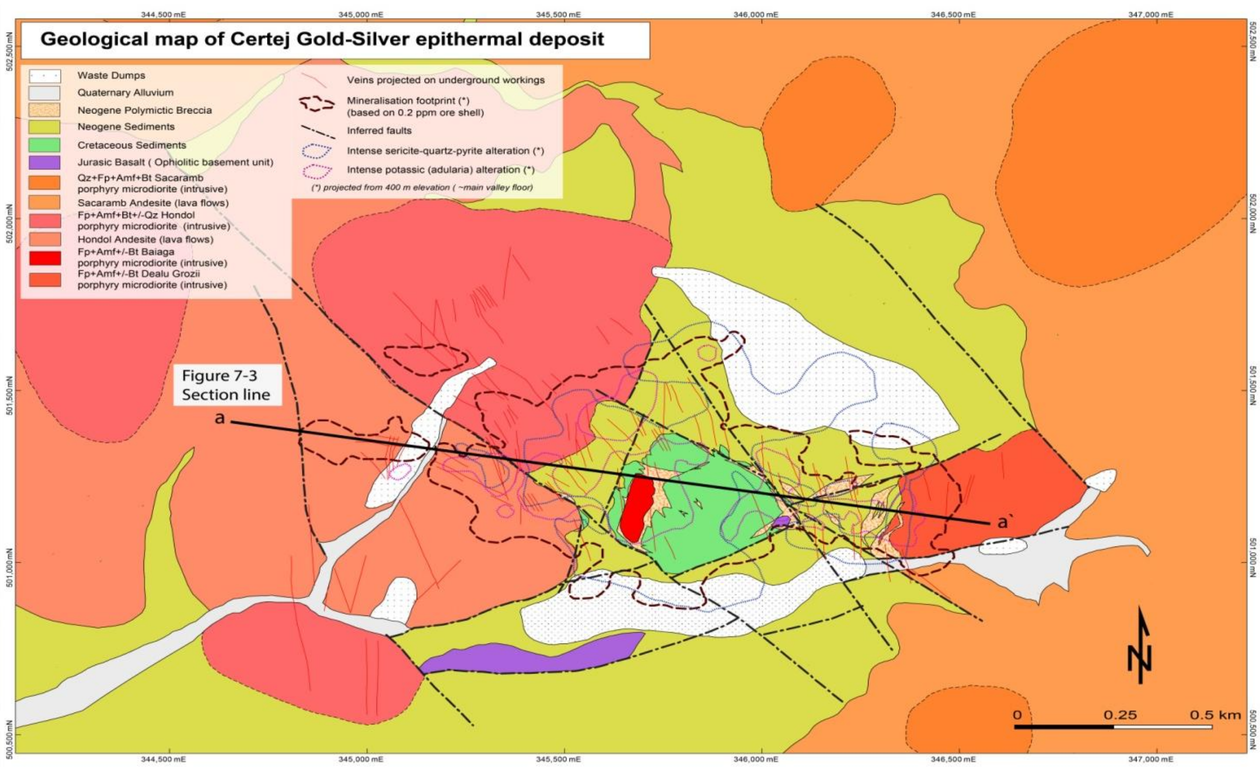


Figure 7-2: Deposit Map

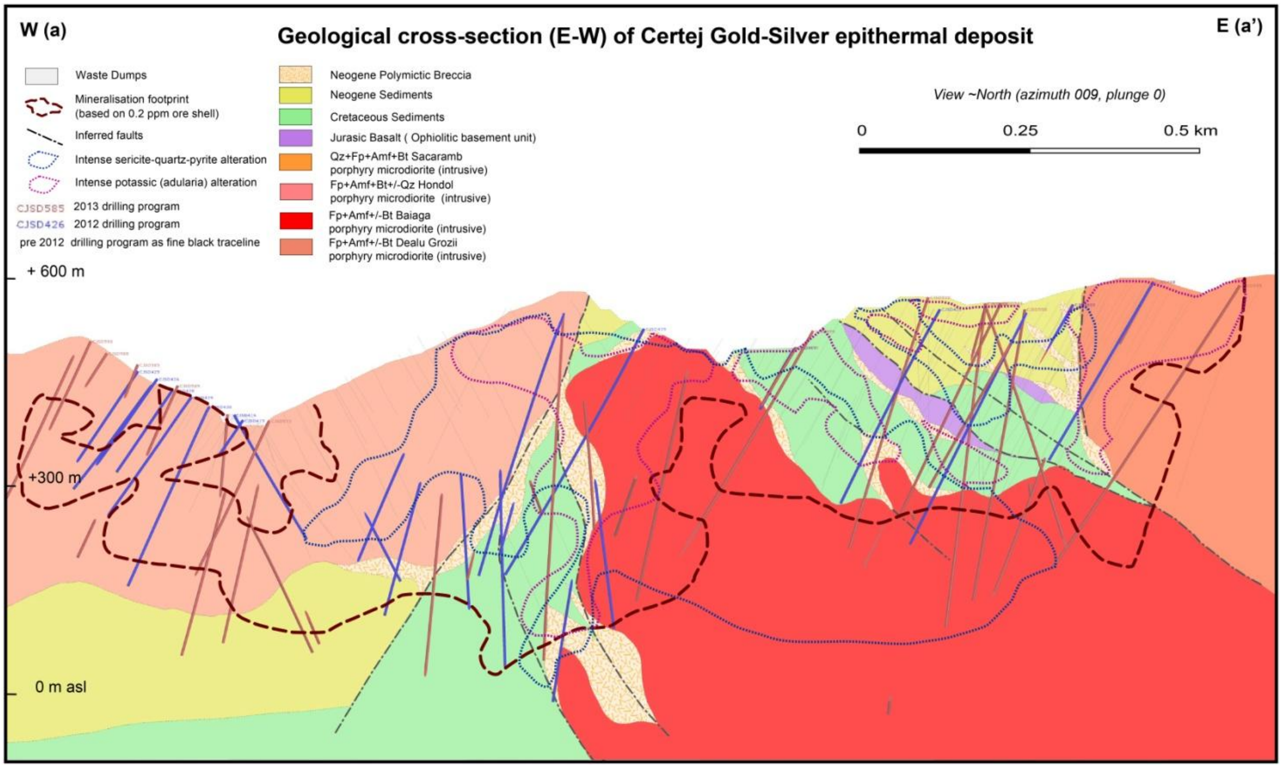


Figure 7-3: Cross Section

The Certej deposit is classified as an intermediate sulfidation epithermal deposit due to the Au being associated with Ag, Pb and Zn and minor Te and As, and the strong quartz-illite-pyrite ±adularia±carbonate alteration assemblage. The mineralization footprint (defined by the 0.2 g/t Au cut-off) has a broad E-W orientation and is constrained between two major NW-SE oriented faults, the East and West faults. In detail the Au distribution exhibits a complex pattern controlled by a combination of structure, proximity to the Baiaga andesite and host rock type. Mineralization occurs at the brecciated contact of the Baiaga and up into the surrounding sedimentary rocks and Hondol and Dealu Grozii andesites. Some of the higher grade mineralization in particular appears to be localized by NW-SE and NE-SW normal faults that may have focussed the mineralization fluids around the Baiaga andesite. Mineralization styles include disseminations in sedimentary units, hydrothermal breccias, stockworks and disseminations at the contacts of intrusive rocks and distal veins. The Au mineralisation is associated with disseminated pyrite (with arsenic and gold-rich rims) occurring together with variable amounts of sphalerite and galena. Native gold is present but restricted to distal veins.

Both hydrothermal alteration and pathfinder geochemistry are zoned about the centrally located Baiaga andesite. White mica crystallinity increases towards the Baiaga andesite suggesting higher temperature conditions, whereas interlayered smectite-illite becomes more abundant at the margins of the deposit. A zoned alteration assemblage is recognised with a sericite-quartz-pyrite alteration forming a central higher temperature alteration proximal to the Baiaga andesite. Within the breccias and higher grade Au-bearing structures an adularia-quartz-sericite-pyrite alteration occurs. More distally to the Baiaga illite-clay-carbonate alteration develops zoning outward, particularly in the west, to a chlorite-carbonate-smectite alteration. In terms of pathfinder elements, the central part of the deposit has elevated Te with lower As whereas the eastern and western margins of the deposit have lower Te and elevated As. This further supports a thermal zonation away from the central Baiaga andesite.

SECTION • 8 DEPOSIT TYPES

The Certej deposit is best classified as an intermediate sulfidation epithermal deposit based on the abundance of base metals (Pb, Zn, including Fe-poor sphalerite) associated with Au and the presence of a near neutral pH alteration assemblage of quartz-illite-pyrite ±adularia±carbonate. The related igneous rock composition and tectonic setting are also consistent with an intermediate sulfidation ore forming environment; specifically the calc-alkaline andesite association and broadly extensional stress state of the magmatic arc, albeit at a distance significantly distal from the arc. Analogous terms in the literature classify similar deposits as sulfide and base metal-rich low-sulfidation epithermal deposits and carbonate-base metal gold deposits. Global analogies include Kelian in Indonesia, and in the Apuseni region Rosia Montana.

Intermediate sulfidation deposits are commonly associated with porphyry copper-gold and low sulfidation epithermal gold deposits. Such deposits form in and around calc-alkaline (both non-adikitic and adikitic-like) volcano-intrusive complexes as a result of shallow level magmatic-hydrothermal processes. The South Apuseni Mountains exhibit these features with known examples of porphyry copper-gold deposits (Rosia Poieni, Rovina) and intermediate to low sulfidation epithermal gold deposits (e.g. Rosia Montana, Baia de Aries, Barza, Certej). Distinguishing intermediate and low sulfidation deposits in the region is challenging due to the commonly overprinting to telescoped nature of the deposits with low sulfidation veins commonly occurring late in the overall paragenesis of the system. Exploration for these deposits is based on well-constrained geological models that utilise predictable patterns in geology, geochemistry, alteration and geophysics.

Exploration criteria for the intermediate sulfidation deposits in the region include:

- Geology: targeting Miocene volcano-intrusive complexes.
- Geophysics: targeting magnetic lows within andesite due to magnetite-destructive alteration, and chargeable IP anomalies due to the abundance of pyrite.
- Geochemistry: identifying favourable lithogeochemical indicators of fertile, hydrous, oxidized intrusions including high Mg number, high Sr/Y ratio, elevated Sr and Ba, and depleted Y and Yb, and pathfinder elements including Au, As, Te, Sb, Zn, Pb, and Ag.
- Alteration: targeting zoned hydrothermal alteration systems with quartz-illite-pyrite-adularia-carbonate alteration assemblages proximal to high grade mineralization, and distal illite-clay-carbonate to chlorite-carbonate-smectite alteration.

SECTION • 9 EXPLORATION

Exploration for additional gold mineralization at the Certej Project is wholly conducted by drilling from surface platforms. These activities are described in subsequent sections of this document.

SECTION • 10 DRILLING

Diamond drill holes have been the principal source of the geological and grade data for the Certej Project. The drill holes, especially the 2012 and 2013 drilling programs, targeted extension of mineralization at depth and laterally and resource delineation of known areas of mineralization. Part of the drilling program also had concomitant purposes for metallurgy sampling, geotechnical information and hydrogeological monitoring. Drilling totals are organized in two groups: those prior to 2011(pre-Eldorado) and those drilled from 2011 to present (2012 and 2013 drilling being directed by Eldorado) presented in Table 10-1.

Table 10-1: Summary of Certej Drilling

Period	DDH Drilling		RC Drilling		RC/DDH Drilling[1]	
	# of holes	Meters	# of holes	Meters	# of holes	Meters
Up to 2011	248	26,828	192	27,342	66	20,156
2012	47	15,995	--	--	--	--
2013	189	55,082	--	--	--	--
Total	**484**	**97,905**	**192**	**27,342**	**66**	**20,156**

[1] RC/DDH denotes holes that started as RC for a predetermined distance then continued as a diamond hole.

Holes drilled during the 2012 - 2013 campaigns ranged in length from 90 m to 723 m long, averaging 361 m. The locations of these holes are shown in Figure 10-1 and Figure 10-2.

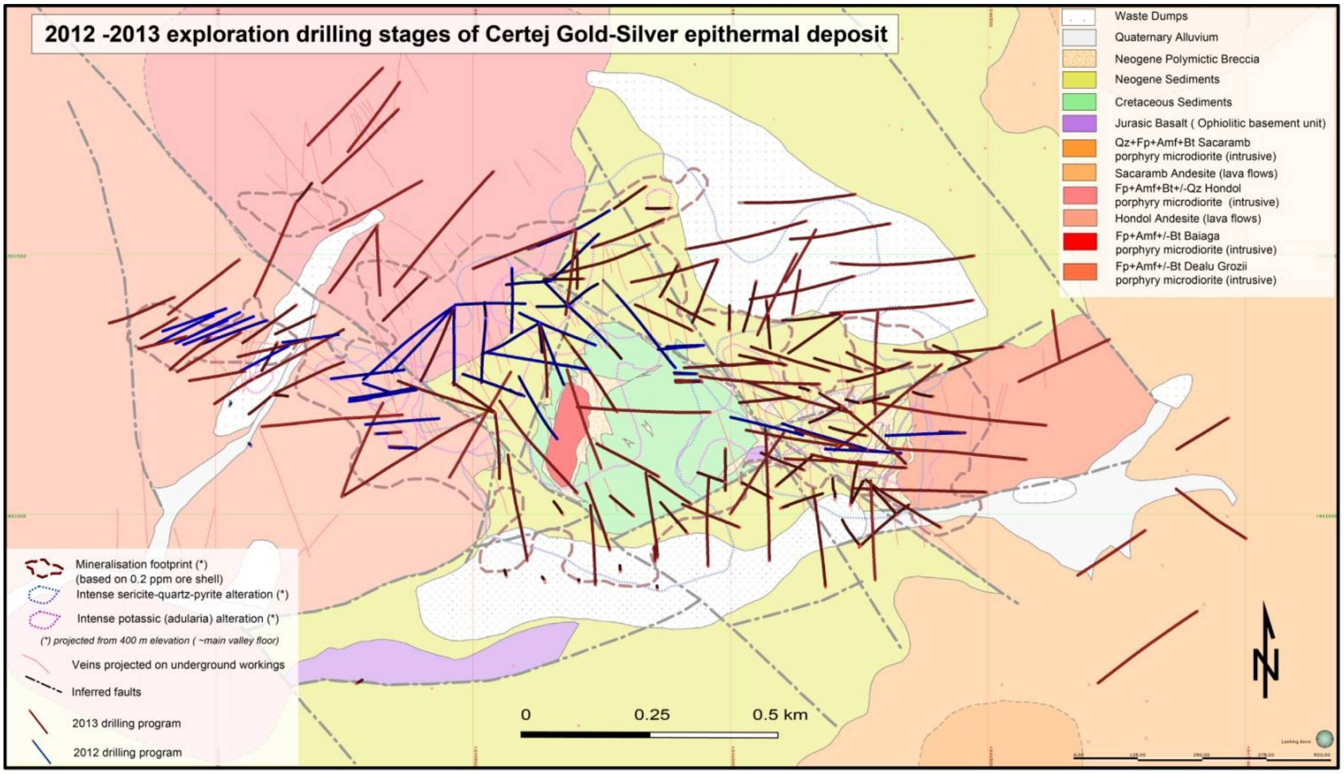


Figure 10-1: Location of Certej Project Drill Holes
Plan view relative to footprint of mineralization

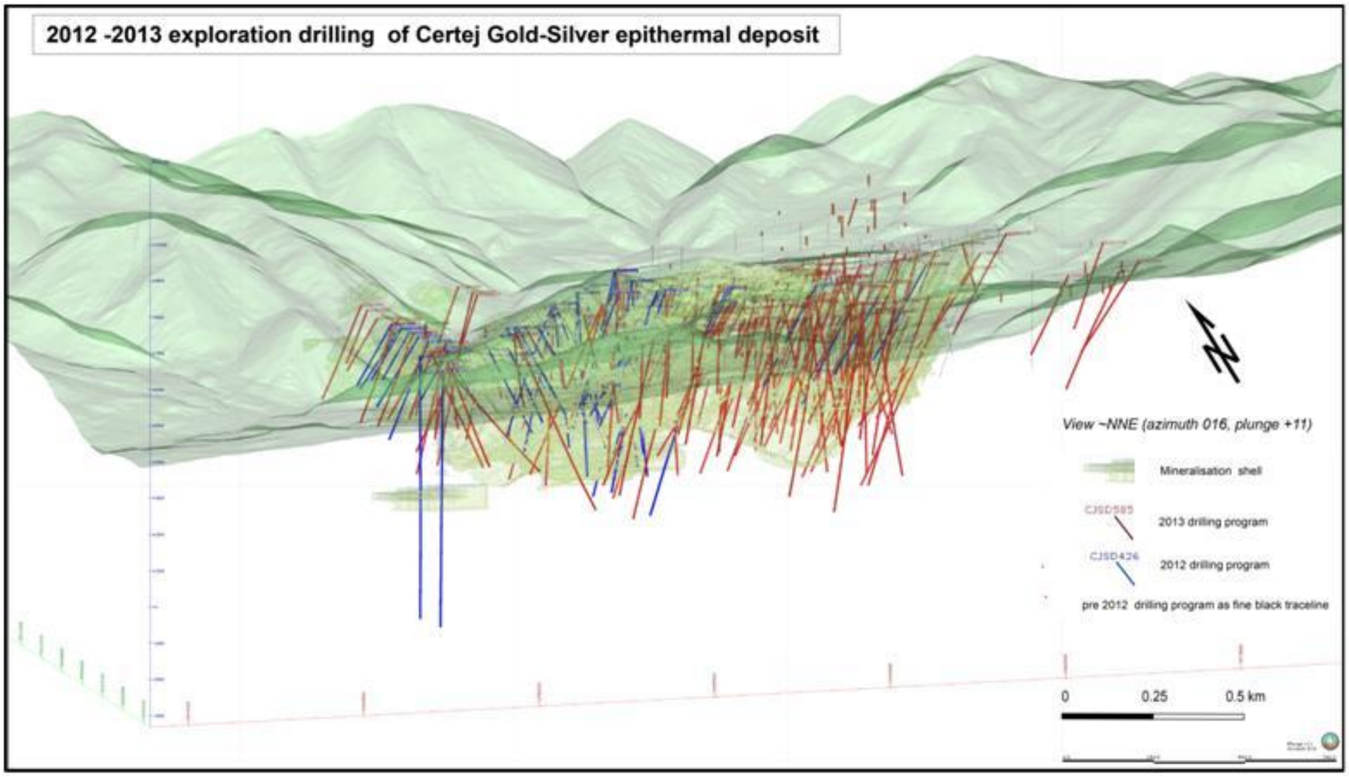


Figure 10-2: Location of Certej Project Drill Holes

Drilling was completed by wireline method with P-size (PQ, 85 mm nominal core diameter, generally on hole starts), H-size (HQ, 63.5 mm nominal core diameter) and, less commonly, N-size (NQ, 47.6 mm nominal core diameter) equipment, using up to four drill rigs. Final completed collars were surveyed using total station.

The drill holes were drilled at an inclination between 50° and 90°, averaging 67°. Holes were drilled along variable azimuths. Down-hole surveys were taken approximately every 50 m using a single-shot measurement system (Reflex survey instrument). Since July 2013, more detailed down hole survey data was captured using a gyro system.

Standard logging and sampling conventions were used to capture information from the drill core. Core was logged using logging templates on portable mini-computers, and the data was then entered into the project database. Data capture included lithology description, alteration types and intensity, sulphide type and content, veining, structural and geotechnical details. Core was photographed before being sampled.

Eldorado reviewed core logging procedures at site, and drill core was found to be well handled and maintained. Material was stored as stacked pallets in an organized core storage facility. Data collection was completed competently. Core recovery in the mineralized units was excellent, usually between 95% and 100%. Overall, the Certej drill program and data capture were found to have been performed in a competent manner.

SECTION • 11 SAMPLE PREPARATON, ANALYSES AND SECURITY

11.1 SAMPLING METHOD AND APPROACH

The principal source of geologic and assay data for the Certej Project (and the sole source since 2011) was diamond drill core. To ensure a high standard of sample and data collection quality, defined control procedures were implemented as shown in Figure 11-1. The diamond core was marked off at 1m sample intervals and cut lengthways using a diamond saw to produce half-core samples for assay. Each meter sample of half HQ drill core was bagged into a separate calico bag for shipment to the laboratory.

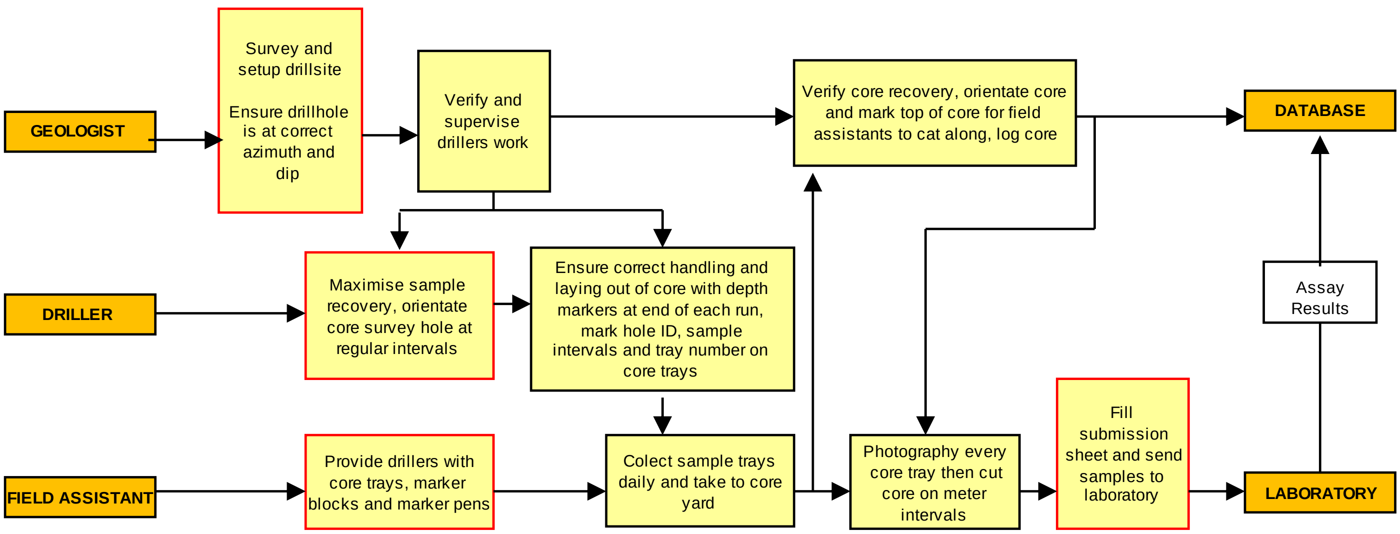


Figure 11-1: Diamond Drill Sampling Control Chart, Certej Project

11.2 SAMPLE PREPARATION AND ASSAYING

Core samples are prepared and assayed at the Gura Rosiei ALS Laboratory in Romania. The core samples are first crushed then pulverized at the Gura Rosiei Laboratory, pulverized to 90% minus 75 µm (200 mesh). A 150g subsample is split off for analysis. All samples are assayed for gold by 30 g fire assay with an atomic absorption (AA) finish (gravimetric finish re-assay for samples >10 g/t Au) and for multi-element determination using aqua regia digestion and inductively coupled plasma (ICP) analysis.

The sample batches are arranged to contain regularly inserted control samples. Insertion rate for Standard Reference Material (SRM) samples and duplicate samples are about 1 in 20 submitted samples whereas blank samples are inserted at a rate of 1 in 40 submitted samples. The SRM's are used to monitor accuracy of the assay results, the duplicates are used to monitor precision and the blank sample can indicate sample contamination or sample mix-ups.

11.3 QA/QC PROGRAM

Assay results are provided to Eldorado in electronic format and as paper certificates. Upon receipt of assay results, values for SRMs and field blanks are tabulated and compared to the establish SRM pass - fail criteria:

- Automatic batch failure if the SRM result is greater than the round-robin limit of three standard deviations.
- Automatic batch failure if two consecutive SRM results are greater than two standard deviations on the same side of the mean.
- Automatic batch failure if the field blank result is over 0.03 g/t Au.

If a batch fails, it is re-assayed until the contained control samples pass. Override allowances are made for samples testing weakly or non-mineralized material. Batch pass/failure data are tabulated on an ongoing basis, and charts of individual reference material values with respect to round-robin tolerance limits are maintained.

11.3.1 Blank Sample Performance

Assay performance of field blanks is presented in Figure 11-2 for gold. The analytical detection limit (ADL) for gold is 0.01 g/t. The rejection threshold was chosen to equal 0.04 g/t. The results show a no evidence of contamination. Rare higher values were investigated and found to be caused by sample mix-ups.

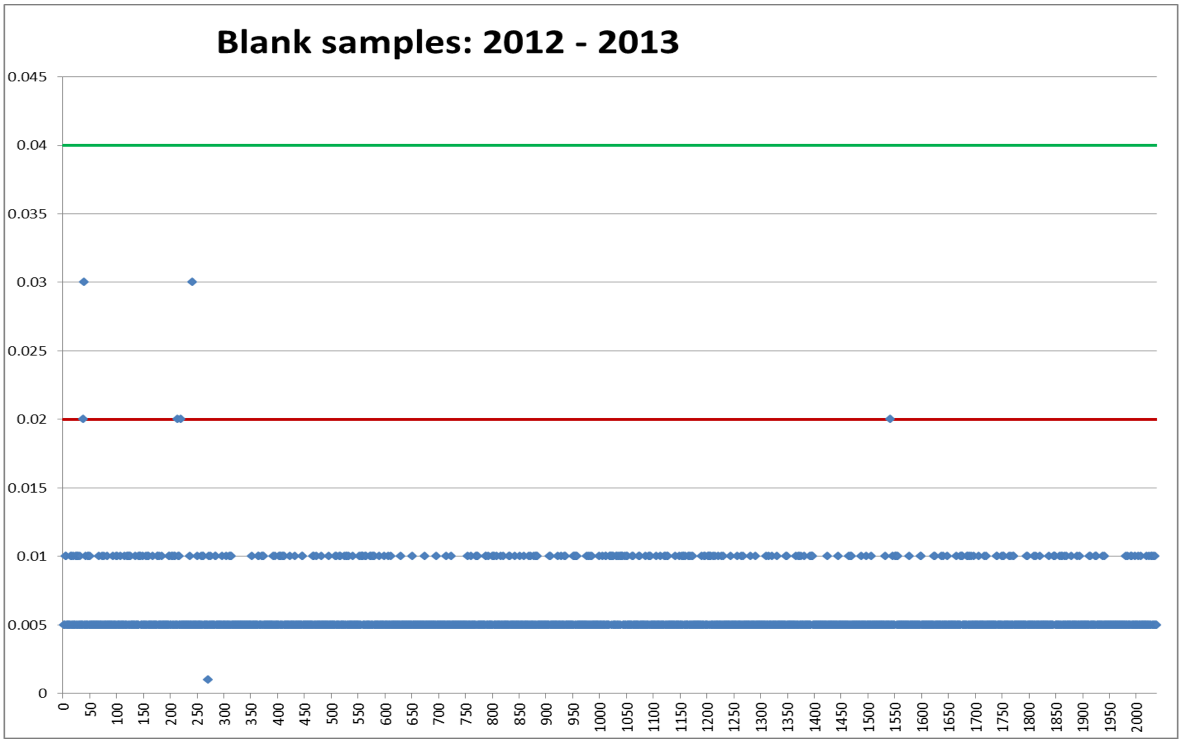


Figure 11-2: Certej Project Blank Sample Data – Jan 2012 to Dec 2013

11.3.2 Standards Performance

Eldorado Gold strictly monitors the performance of the SRM samples as the assay results arrive at site. Numerous SRM samples used are covering a grade range between 0.5 g/t to 13.5 g/t. Charts of SRMs covering the most common grade ranges are shown in Figure 11-3 thru Figure 11-6. All samples are given a “fail” flag as a default entry in the project database. Each sample is re-assigned a date-based “pass” flag when assays have passed acceptance criteria. As of the data cut-off date of December 31, 2013, all samples had passed acceptance criteria.

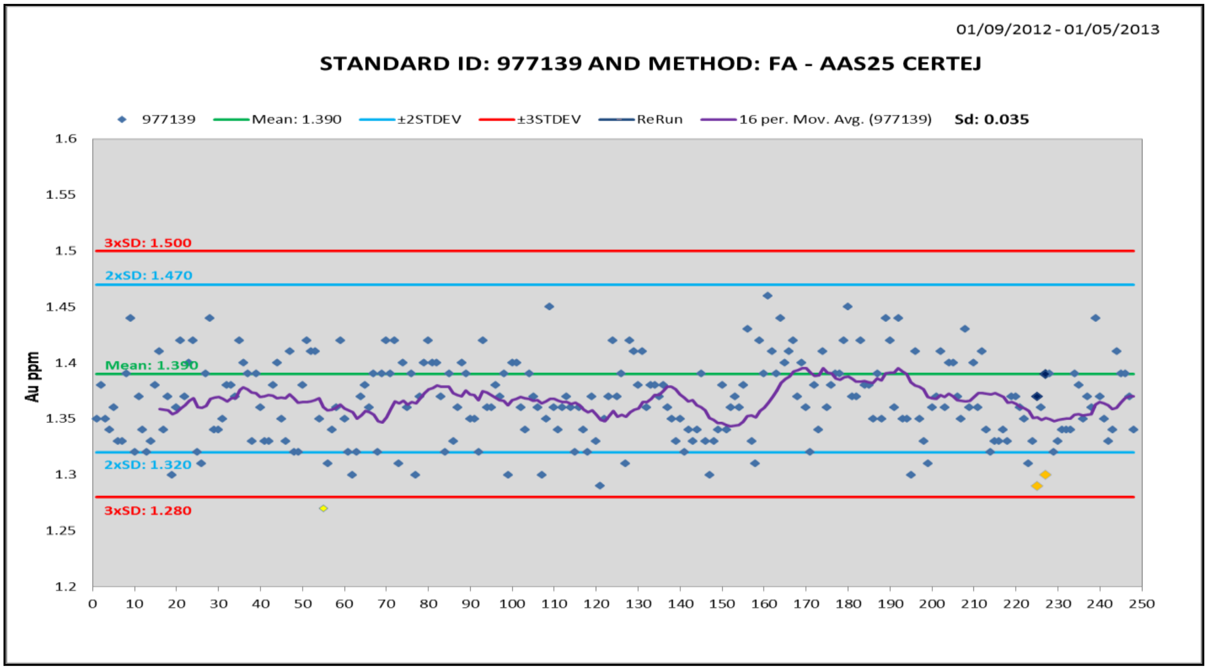


Figure 11-3: Standard Reference Material Charts, Certej Project: 2012 to 2013. Chart representing 0.5 g/t Au

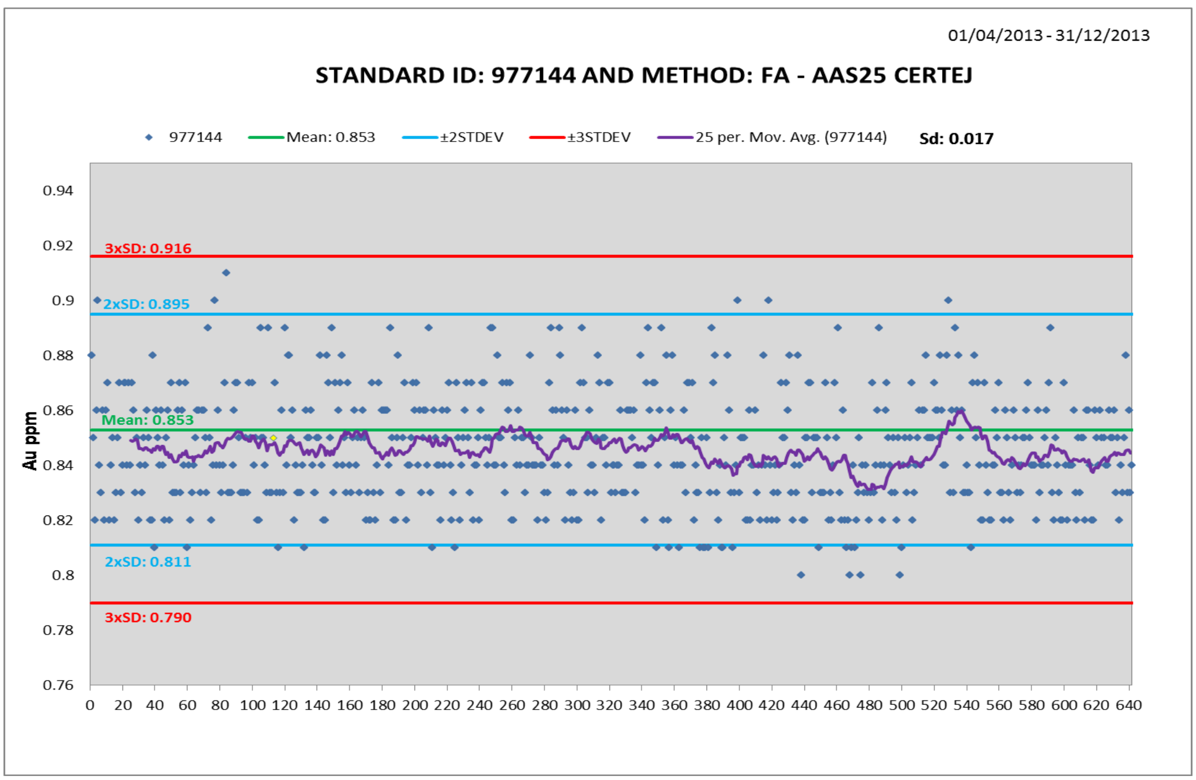


Figure 11-4: Standard Reference Material Charts, Certej Project: 2012 to 2013. Charts representing 0.9 g/t Au

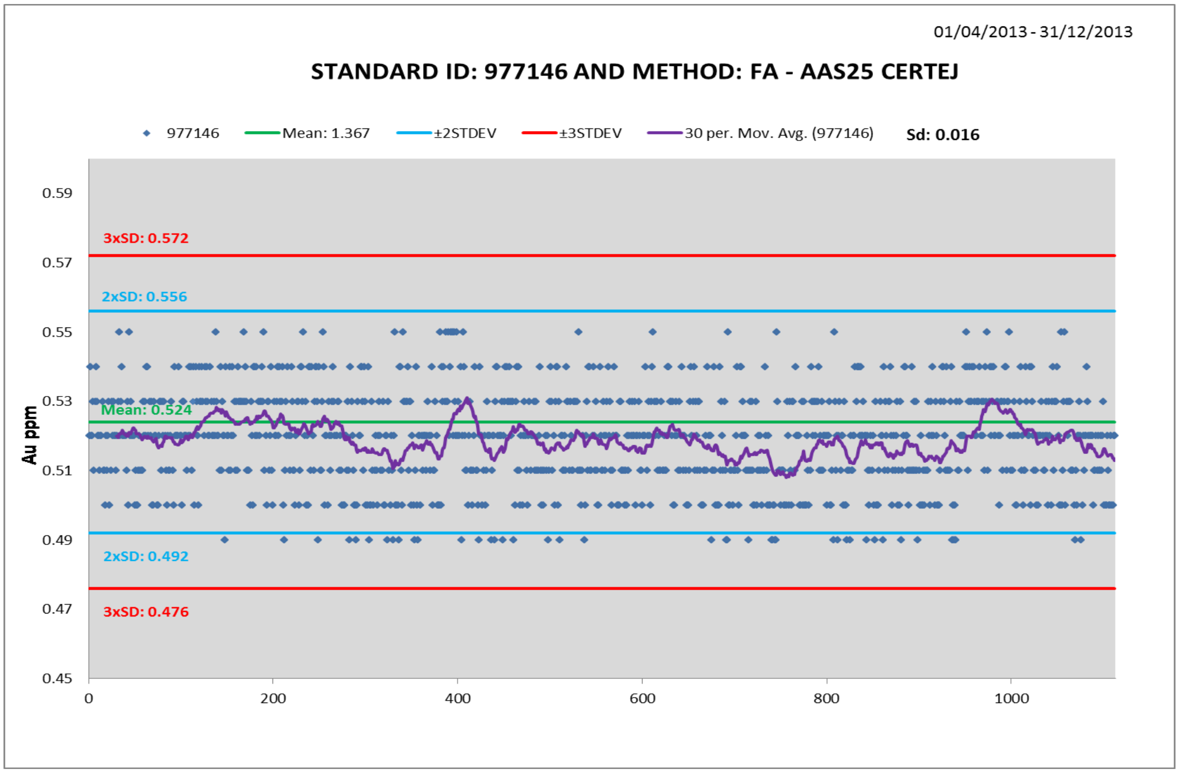


Figure 11-5: Standard Reference Material Charts, Certej Project: 2012 to 2013. Charts representing 1.4 /t Au

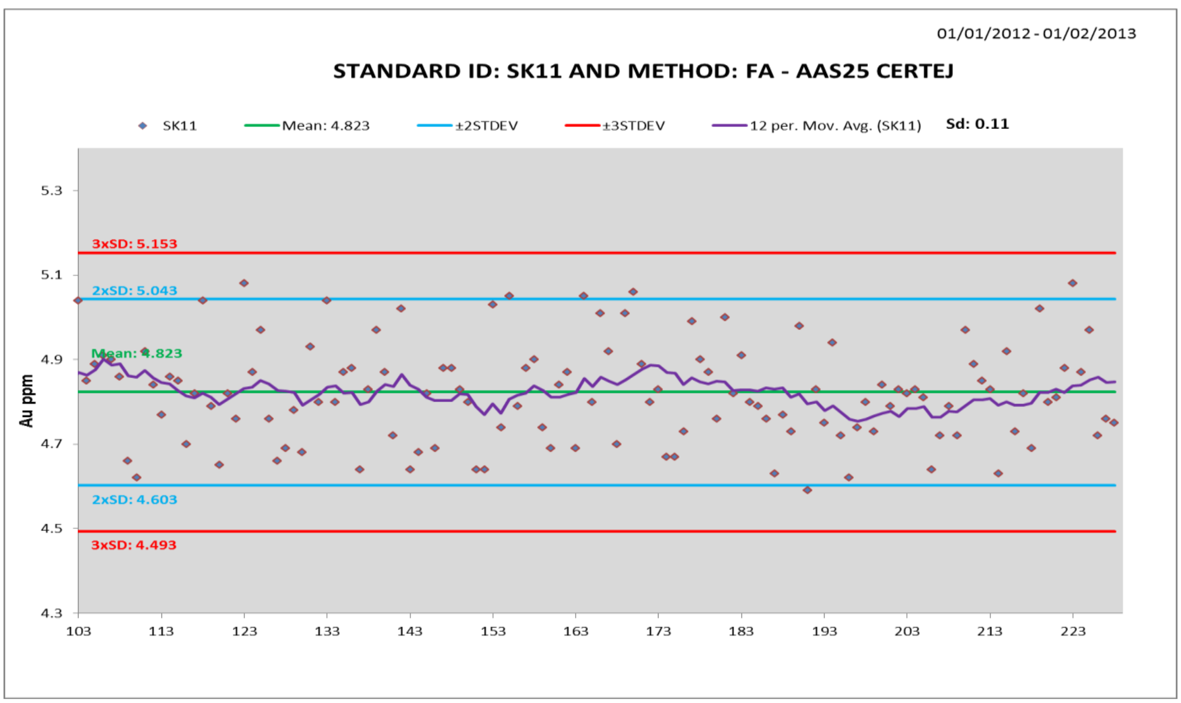


Figure 11-6: Standard Reference Material Charts, Certej Project: 2012 to 2013. Chart representing 4.8 g/t Au

11.3.3 Duplicates Performance

Eldorado implemented and monitored regularly submitted coarse reject duplicates. This data reproduced well. The duplicate data are shown in a relative difference chart in Figure 11-7 and percentile rank chart in Figure 11-8. Patterns observed in the relative difference plot are symmetric about zero suggesting no bias in the assay process. For the 90th percentile of the population as shown on the percentile rank plot, a maximum difference of 20% is recommended for the coarse reject duplicates because these duplicate types can be controlled by the subsampling protocol. The Certej data shows 18% difference in the coarse reject data.

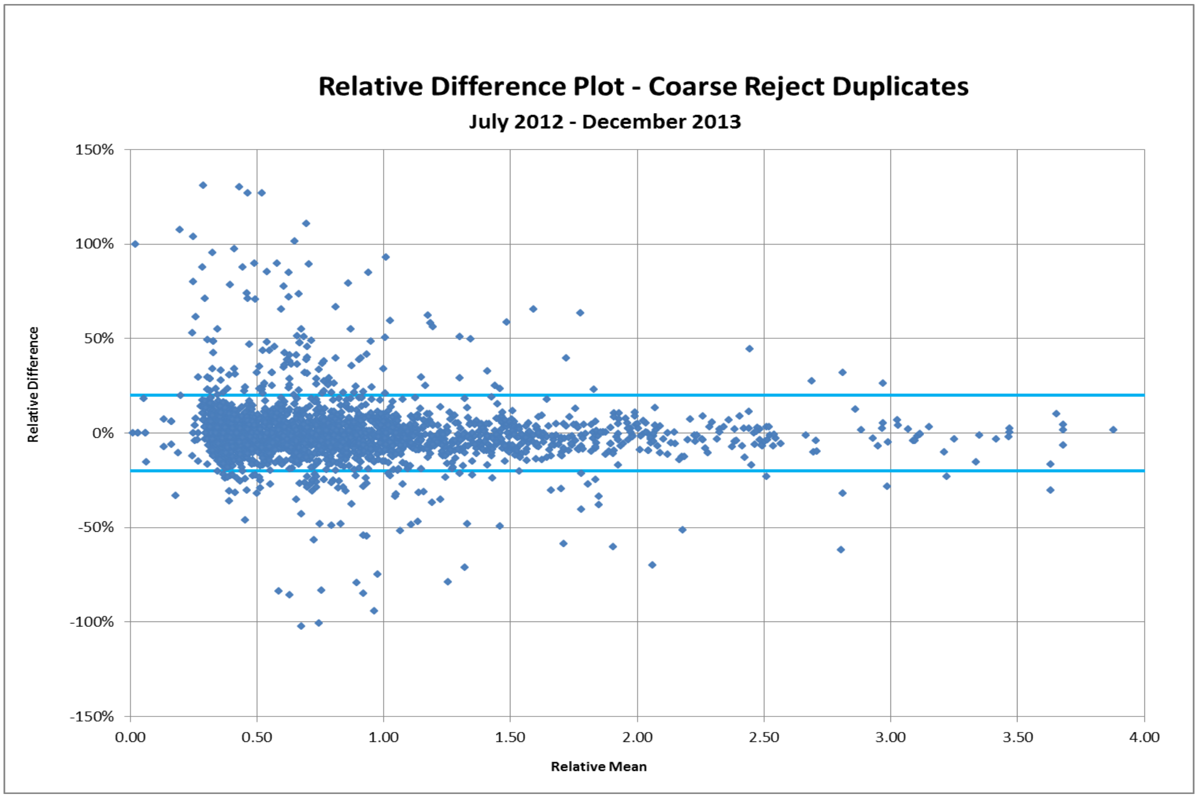


Figure 11-7: Relative Difference Plot of Certej Duplicate Data, 2012 to 2013

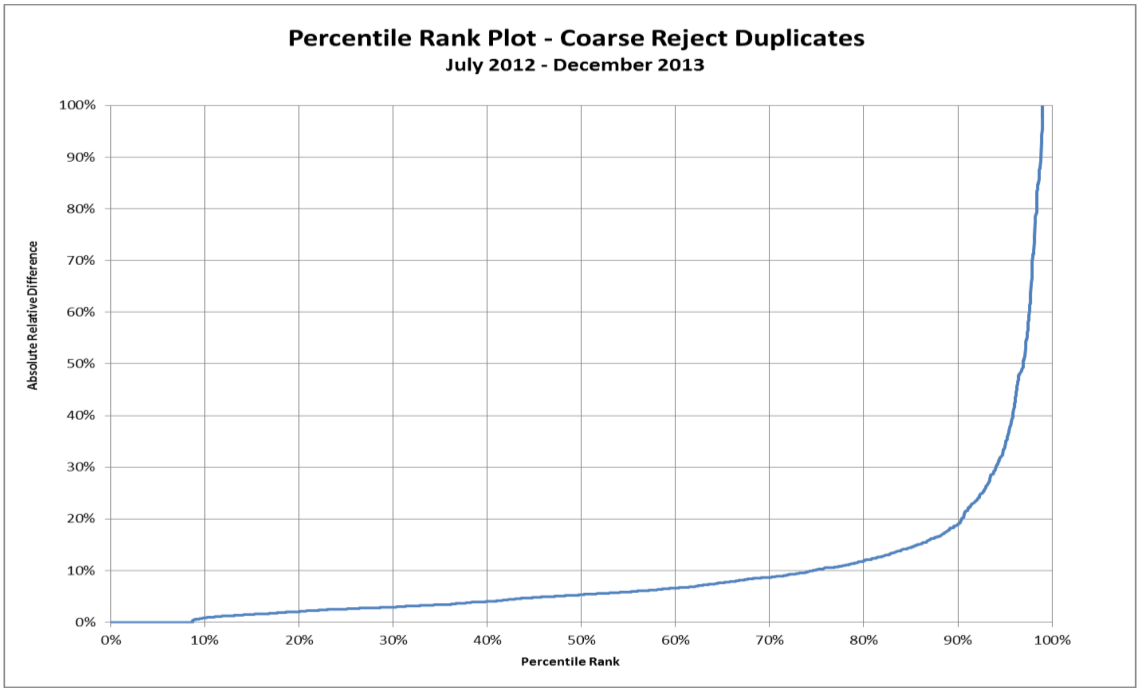


Figure 11-8: Percentile Rank Plot, Certej Duplicates, 2012 to 2013

11.3.4 Pre-2012 Performance

Assay performance of data collected prior to 2012 has been controlled by a system of SRMs, blanks and duplicates controlled the earlier exploration and delineation work. All results showed that the process was in control and that the data was sufficiently reliable to support resource estimation. It was described in detail in a previous Technical Report (Forward et al, 2009).

11.3.5 Specific Gravity Program

Samples taken for assay from core holes are being measured for specific gravity and tabulated by rock type. The specific gravity for non-porous samples (the most common type) is calculated using the weights of representative samples in water (W2) and in air (W1). The bulk density is calculated by W1 / (W1-W2).

11.4 CONCLUDING STATEMENT

In Eldorado's opinion, the QA/QC results demonstrate that the Certej deposit assay database, particularly for new data obtained from 2012 to 2013, is sufficiently accurate and precise for resource estimation.

SECTION • 12 DATA VERIFICATION

Monitoring of the quality control samples showed all data were in control throughout the preparation and analytical processes. In Eldorado's opinion, the QA/QC results demonstrate that the Certej Project assay and geologic database, particularly for new data obtained from the start of 2012, is sufficiently accurate and precise for resource estimation and grade control work.

Also in 2013, checks to the entire drill hole database were undertaken. Checks were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado therefore concluded that the data supporting the Certej Project resource work is sufficiently free of error to be adequate for estimation.

The 2012 and 2013 drill campaigns redrilled volumes of mineralization previously tested by pre-2011 work. These newer holes tested new targets at depth and revised geological interpretations. For the portions tested by both older and newer campaigns, it provided a chance to assess the older diamond and RC drill data. In doing so, Eldorado was able to verify the results obtained before 2011.

Taken all together, these observations demonstrate that the data gathered and measured for the purposes of estimating the gold grades at the Certej Project are verified.

SECTION • 13 MINERAL PROCESSING AND METALLURGICAL TESTWORK

13.1 INTRODUCTION

The Certej deposit contains significant gold and silver values, principally hosted in pyrite mineralization. Previous development of the Certej deposit, completed by then owner European Goldfields Limited (EGL), was predicated on the fact that the bulk of the gold values in the deposit was refractory in nature, that is, not readily recoverable by direct cyanidation of the ore. The EGL work therefore included significant flotation testwork, to produce a high-gold grade, pyrite concentrate from selected ore zones, which demonstrated that high gold and silver recoveries of over 90% could be achieved to the concentrate. The work completed by EGL also highlighted that an oxidation step would be required prior to cyanidation of the flotation concentrate to achieve high overall gold and silver recoveries.

The "Certej Updated Definitive Feasibility Study Summary Technical Report", published by EGL in February 2009 was based on application of the proprietary Albion process, furnished by Xstrata Technology, for oxidation of the high-gold grade, pyrite concentrate generated by flotation as a means to release the gold and silver values such that they could be recovered by conventional cyanidation processes. The Albion process was evaluated with continuous pilot scale testwork to confirm the process selection. The pilot scale testwork program also included a continuous carbon-in-leach (CIL) cyanidation pilot plant operation to recover gold and silver from the concentrate following treatment by the Albion process in an effort to establish overall gold and silver recoveries, reagent requirements, and other operating criteria for commercial design.

Following acquisition of EGL by Eldorado Gold Corporation, the Certej Project was further evaluated in 2012 and 2013. Upon confirmatory testing undertaken by Eldorado, principally to test the response of variability concentrate samples to the Albion process, it was shown that the high overall gold and silver recoveries achieved in earlier studies were not reproducible. In particular the high gold recoveries of 90% to the flotation concentrate, reportedly obtained in producing a high-gold grade concentrate (with associated sulphur grade of about 35% S^{2-}) during the EGL testwork, could not be reproduced during the additional flotation testwork undertaken by Eldorado in 2012. Also, it was shown that organic carbon in the Certej ore reported to the flotation concentrate and adversely affected gold recovery following Albion oxidation, i.e., the organic carbon in the concentrate exhibited 'preg-robbing' properties. Further, technical risks identified in the process design were judged to be significant, including the limited commercial application of the Albion process for gold containing concentrates and the variable residence time requirements of the process. Thus, the Albion process option was abandoned in 2013 in favour of the pressure oxidation process, which was considered to be more robust (less sensitive to the adverse effects of organic carbon) and was known to be relatively widely applied at the commercial scale.

The pressure oxidation process results in effectively complete oxidation of the sulphidic sulphur in the concentrate feed and therefore, concentrate to be treated by pressure oxidation can be of a significantly lower sulphur grade than that required in the Albion process, roughly 12% S^{2-} for pressure oxidation compared with 35% S^{2-} for the Albion process. Thus, evaluation of the pressure oxidation process required a significant amount of flotation testwork to generate low-sulphur grade, pyrite concentrate with the objective of achieving 90% recovery of gold and silver to the flotation concentrate on a consistent basis. Both batch and continuous pilot flotation testwork were completed in 2012 and 2013 and the results generally showed that higher gold and silver recoveries to the flotation concentrate could be achieved when a lower sulphur grade in the concentrate was targeted. The lower sulphur grade requirement also simplified the flotation plant design and eliminated the need for multiple stages of cleaning. Gold recoveries to the flotation concentrate were generally on the order of 91 to 97% (from the ore to the concentrate) for variable ore zones within the Certej deposit when flotation was operated to provide a sulphur grade of 12 to 14% S^{2-} in the concentrate.

Evaluation of the pressure oxidation process also included batch and continuous pilot testing to demonstrate the amenability of the Certej concentrates to the pressure oxidation process, and to generate data for use in detailed design of the commercial plant. The continuous pilot pressure oxidation testing demonstrated that sulphur oxidation was very rapid under typical pressure oxidation conditions and that gold recoveries in the subsequent CIL cyanidation process were of the order of 90 to 96% (from the concentrate to the loaded carbon). The pressure oxidation testing showed that sulphur oxidation was relatively insensitive to changes in process conditions and relatively constant for each of the concentrates produced from the four major ore zones of the deposit. Silver recoveries varied between 72 and 88% (from the concentrate to the loaded carbon), though only after treating the pressure oxidation product solids in a ‘lime boil’ (hot alkaline digestion) process. The lime boil did not adversely affect gold recovery and in most cases, resulted in a modest improvement.

13.2 MINERALOGY

The mineralogy and petrography of the Certej deposit have been extensively studied dating back to 1993. Several reports have been generated by Romanian research institutions and independent world leading laboratories including SGS Lakefield and Amtel.

The Certej deposit contains three main rock types, namely, andesite intrusive, Cretaceous sandstone and Neogene conglomerate and sandstone. Hydrothermal and tectonic brecciation occurs extensively within the deposit.

The Certej resource can be identified broadly by distinct chemical and mineralogical signatures and can be divided into four main and discrete ore zones known as the East, the Intermediate, the Central and the West zones. The gold and silver values are mostly distributed as solid solution and very fine particles within pyrite, in gold bearing minerals locked in pyrite grains, and, for the Central and Intermediate ore zones, associated with complex tellurium minerals. There is a strong compositional association between gold and arsenic, though arsenic content in the deposit is relatively low. The studies identified arsenean pyrite zones within the pyrite grains particularly as rims on the pyrite surface in ores from the East zone and within the pyrite grains in ores from the West zone. Arsenopyrite is only a minor gold carrier, mainly limited to the East and Intermediate ore zones. There is a very strong correlation between gold and silver within the deposit.

13.2.1 Previous Mineralogical Studies

Early mineralogical analysis identified major sulphide minerals as pyrite, followed by sphalerite and marcasite. It was concluded that there was no coarse gold and that the majority of the gold was effectively present in solid solution, mainly in arsenical pyrite rims associated with pyrite grains. It was observed that there was very little or no arsenopyrite mineralization in the Certej deposit.

Some of the associated direct cyanidation leach tests with ore samples gave high gold extractions in excess of 80%, depending on the lithology and the sample source. This indicated that the gold was not entirely locked up and that the ore has some free milling characteristics. However, the location and history of the ore samples, which gave high gold extraction from direct cyanidation, was not well documented. There was speculation that oxidation of the samples may have contributed to the high gold extractions observed.

The gangue minerals were quartz and K-feldspar with minor plagioclase and muscovite.

SGS obtained a total of 197 electron microprobe analyses from pyrite grains in the Mozley concentrate prepared from the Cretaceous ore samples. Approximately 39% of the pyrite analyses reported elevated arsenic contents ranging from 0.99 wt% to 8.23 wt% As, averaging 2.62 wt% As, and they were termed as “arsenical pyrite” in SGS’s report. Backscatter electron imaging of the pyrite revealed distinctive rhythmic compositional banding relating to regions of relatively low and high arsenic content.

Static secondary ion mass spectrometry (SSIMS) analysis was performed on a total of 30 pyrite grains from the Cretaceous Mozley concentrate. Solid solution gold in pyrite was identified in 27 of 32 individual spot analyses. Analyses ranged from less than the instrument detection limit of 0.17 g/t Au to 37.3 g/t Au, with an average of 2.8 g/t Au. Solid solution gold distribution in pyrite was associated with regions of elevated arsenic content, primarily within euhedral (also called zoned), concentric zones on the rims of pyrite grains. In addition, a sub-micrometer-sized inclusion of native gold in pyrite was noted during the SSIMS study, indicating the presence of extremely fine-grained native gold in the sample.

In order to determine how best to characterize the Certej deposit by mineralogy and therefore chemistry, inductively coupled plasma (ICP) spectroscopy data from all the drilling samples was examined in an orebody mapping exercise. This exercise showed that the deposit could be divided into four zones, West, Central Intermediate and East, by a combination of arsenic, silver and tellurium concentrations. Boundaries between the Central and Intermediate zones were not as well defined. Concentrate analyses had revealed the importance of tellurium sulphide minerals as gold carriers in the Central part of the deposit and as a result, tellurium mapping was conducted by ICP on composite drill samples. This enabled hard boundaries to be defined between the Central and Intermediate zones with the boundary modelled using tellurium and arsenic concentrations with gold as a guide. The resulting observations were made following the orebody mapping:

- ICP plots of As vs. Ag define the boundary between the East and the Intermediate metallurgical domains very clearly. This boundary also agrees with the higher Ag in the model for the Intermediate zone.
- Ag grades in the Block Model define a high grade population in the Central zone and the boundary to the West area is defined where the Ag grades fall away. This is very clear in the Block Model.
- Te plots clearly define the Central zone with elevated Te values compared to the rest of the deposit.
- The Intermediate zone is a combination of moderate As and Te values and thus shows some features of the East and some of the Central zones.

13.2.2 Recent Petrographic Studies

Flotation testing of the Certej ores highlighted that the East ore zone gave significantly poorer gold recovery than other ore zones. Similarly, pressure oxidation testing showed that concentrates derived from the East ore zone, which did not contain a high level of organic carbon, gave the poorest gold extraction in cyanidation of the pressure oxidation product solids. As a result of the anomalous behaviour of the East materials, seven mineralized samples from the East ore zone were selected from drill cores for petrographic analysis by Panterra Geoservices Inc. The samples included four volcanic rocks and three polylithic conglomerate samples.

The drill core samples were prepared as polished thin sections and viewed on a standard petrographic microscope in transmitted and reflected light. Each of the samples was also examined, with spot analyses, on a scanning electron microscope (SEM) in the Earth, Ocean and Atmospheric Sciences Department the University of British Columbia. The results out of the petrographic and SEM work, including sample assays and dominant alteration, are summarized in Table 13-1.

Table 13-1: Certej Petrographic Samples with Eldorado Gold Lithcodes and Visually Estimated Mineral Percentages

Drill Hole	Depth	LithoCode	Quartz	K-feldspar	Sericite	Carbonate	Pyrite	Marcasite	Arseonpyrite	Sphalerite	Galena	Rutile	Other
CJSD483	118	2_DGTR_MG_11	86	6	5	trace	3	minor	--	minor	trace	trace	zircon, phosphate
CJSD583	64	3_DG_MGH_10	15	67	10	--	5	minor	trace	3	trace	minor	Pb-sulphosalts, barite
CJSD583	94	4_DG_HGBX_10	40	minor	52	--	4	1	trace	3	minor	--	Pb-sulphosalts, chalcopyrite
CJSD475	277	5_DG_LG_10	20	minor	33	40	5	--	--	trace	trace	--	apatite
CJSD475	303	6_DG_HG_10	30	40	25	--	5	minor	Trace	trace	--	minor	Pb-sulphosalts, apatite
CJSD551	68	7_DGTR_MG_10	84	trace	10	--	4	--	--	2	Minor	--	Pb-sulphosalts, barite, phosphate
CJSD516	128	9_NGCO_LG_9	52	30	15	--	3	--	--	--	--	--	zircon

The samples from the East ore zone represented two main rock types: the tertiary andesitic sequence, and the tertiary coarse-grained sandstone to conglomerates. Adularia-quartz-sericite-pyrite alteration affects two of the volcanic samples, and one of the conglomerate samples. Quartz-sericite-pyrite alteration affects one of the volcanic samples, and two of the conglomerate samples. One of the volcanic samples is carbonate-quartz-sericite-pyrite altered (lowest gold grade). Neither clay nor chlorite alteration was present in these rocks.

Pyrite content of the East ore samples ranged from 3% to 5%. They were dominantly very fine-grained (submicron to 0.5 mm) and exhibited a variety of textures, from framboidal (minor) to spongy or semi-prismatic. There is some late stage marcasite overgrowing pyrite in four of the samples. Based on the SEM analyses of numerous pyrite and marcasite crystals of various textures the crystals are not compositionally zoned, and are either non-arsenical are very weakly arsenical (not quantified). However there is a late stage of extremely fine-grained arsenopyrite that locally overgrows the hackly edges of pyrite and marcasite (where present) in the volcanic samples that correlates to the three higher gold grades. Arsenopyrite was not observed in the conglomerates. Sphalerite is present in all but one conglomerate sample, in veinlets and disseminated, and accounts for the high Zn values. Barite is associated with sphalerite in two samples. Galena is associated with sphalerite and accounts for most of the elevated Pb values, however there are also a variety of Pb+/-Cu sulphosalt minerals (both Sb and As-bearing) present in four samples. These are associated with the galena and sphalerite in veinlets in the volcanics and in the matrix in the conglomerate. No gold or silver bearing phases were observed on the standard microscope, or located with the aid of the SEM. Gold may be related to late stage arsenopyrite on pyrite/marcasite. Silver may be related to galena and possibly Pb-sulphosalts.

13.3 DISTRIBUTION OF GOLD AND SILVER IN CERTEJ ORES

A prior study completed by Amtel titled ‘Deportment of Gold and Silver in Certej Flotation Concentrates and Tails’ and reported to EGL in June 2007 described the mineralogy of flotation concentrates from the four Certej ore zones and the tailings from three ore zones (East, West and Intermediate). This study was considered to be a standard reference text on the mineralogy of Certej ore as the samples analysed were the most representative, although it was based on the production of high sulphur grade concentrate suited to the Albion process. The mineralogical study included optical microscopy, X-Ray Diffraction, SIMS and Backscattered Electron (BSE) imaging.

The materials generated for the Amtel studies were produced from drill cores which had been carefully selected using all the considerable information then known about the deposit. This ensured that the material was representative in terms of ore zone, speciation, grade, lithology and of course spatial location. As much of the same drill cores were used in the preparation of concentrates for pressure oxidation testing, the prior Amtel work on Certej ore is still considered valid.

Amtel found the gold in the Certej deposit in three main forms:

- gold in solid solution in sulphide minerals mainly pyrite but also, in the East ore zone very minor marcasite and arsenopyrite;
- gold minerals such as native gold, electrum and Au-Ag tellurides; and

- gold bearing minerals such as silver tellurides and nagyagite.

A summary of the Certej ore analyses, for drill core samples used in the preparation of flotation concentrates for pressure oxidation testing, is provided in Table 13-2.

Table 13-2: Ore Analysis from the Four Certej Ore Zones

	Ore Analysis						
Ore Zone	Au, g/t	Ag, g/t	Te, g/t	$C_{organic}$, %	S, %	As, %	Pb, %
East	2.00	4.6	7.4	0.21	4.7	0.19	0.18
Intermediate	1.96	11.2	10.6	0.34	5.4	0.11	0.14
Central	1.84	14.7	27.6	0.45	4.8	0.02	0.08
West	1.10	7.13	3.5	0.15	4.5	0.07	0.02

By far the dominant form of gold is in pyrite both as solid solution in the five forms of pyrite identified in the Amtel work and associated with gold minerals. However, provided that the first two mineral types above are exposed to solution (not locked), these gold species will be leachable by direct cyanidation and thus are not refractory. The West ore type contains more solid solution gold in pyrite than the other three ore zones. The Amtel study estimated the amount of refractory gold by ore type, at a grind size of 80% passing 75 to 93 µm, based on mineral species and liberation as shown in Table 13-3.

Table 13-3: Summary of Refractory Gold Content in Certej Ores

Ore Zone	Estimated Refractory Au, %	Estimated Leachable Au, %
East	67 – 84	16 -33
Intermediate	64 - 82	18 - 36
Central	40 – 60	40 - 60
West	80 – 93	7 - 20

The extent of direct cyanide leachable gold was a function of the degree of ore oxidation. This contributed to some of the earlier work being based on the expectation that Certej ores could be considered free milling.

The Central ore zone was considered to be less refractory than the others, likely due to the higher content of liberated tellurides and gold minerals.

Free and liberated gold minerals are only minor contributors to the overall gold balance for each ore zone. Thus, neither gravity recovery nor direct cyanidation of the ore are viable process options.

- the Central is characterised by high Ag, high Te, low As;
- the East by low silver, high arsenic, low Te;
- the Intermediate between these levels, (a reflection of the Intermediate ore zone's physical location between the Central and the East); and,
- the West, which has low Au, but elevated Ag with low Te and As.

A summary of the low-sulphur grade flotation concentrates generated from each of the four-ore zones is shown in Table 13-4 below.

Table 13-4: Flotation Concentrate Composition from the Four Certej Ore Zones

Ore Zone	Concentrate Analysis						
	Au, g/t	Ag, g/t	Te, g/t	$C_{organic}$, %	S, %	As, %	Pb, %
East	5.27	12.0	7	0.25	14.1	0.4	0.3
Intermediate	5.30	23.0	25	0.50	14.6	0.2	0.4
Central	5.43	32.5	79	0.68	13.8	<0.1	0.2
West	3.47	21.7	4	0.09	12.9	0.1	<0.2

Of note, the organic carbon was present in only minor proportions in the flotation concentrates; however, the organic carbon was shown to be problematic and to cause appreciable gold losses in the cyanidation circuit, due to its preg-robbing properties. The preg-robbing properties of the organic carbon were exacerbated in the presence of chloride in process solutions.

Direct cyanidation testing of the flotation concentrates, without first treating by pressure oxidation, showed that gold was highly refractory and thus this confirmed the previous Amtel estimates for the extent of gold recoverable by direct leaching with cyanide. A summary of direct cyanidation results is provided in Table 13-5 below.

Table 13-5: Leachable Gold and Silver in Direct Cyanidation Testing of Concentrates

Ore Zone	Direct Cyanidation Extraction, %	
	Au	Ag
East	16.8	21.4
Intermediate	8.8	36.4
Central	17.7	52.6
West	29.0	27.4

The gold is mostly refractory in nature to varying degrees depending on the ore zone. The Central ore zone is expected to be less refractory because a significant proportion of the gold is contained in free Gold/Tellurium mineral grains, and this also applies, to a lesser degree, to the Intermediate ore zone.

The silver in the Certej deposit is found in three main forms:

- silver minerals, electrum and silver tellurides;
- minerals with silver as an “impurity” in the crystal lattice, tetrahedrite; and,
- colloidal size silver sulphide in pyrite and solid solution silver in sulphide minerals, galena, Pb sulphosalts and pyrite.

By far, the dominant form of silver is colloidal size, less than 0.5 µm silver sulphides in pyrite. Pyriteis a principal carrier of silver accounting for majority of the silver in the concentrates.

13.4 MINERAL PROCESSING TESTWORK

13.4.1 Preliminary Batch Pressure Oxidation Testwork

Eldorado Gold commissioned ALS Metallurgy in Australia to complete a brief program of batch pressure oxidation testwork in 2012 to determine whether the Certej concentrates would be amenable to the pressure oxidation process as a potential alternative to the Albion process. ALS, then known as Ammtec, had completed an amenability study in 2001 with Certej ores and so had some familiarity with the project. Complete results of the preliminary pressure oxidation testwork were included in ‘Pressure Oxidation / Cyanide Leach Testwork conducted upon Two (2) Flotation Concentrates and Two (2) Albion Oxidized Products From the Certej Project’ reported to Eldorado Gold in February 2013.

To expedite the project, selected samples of Certej concentrates representing the Intermediate ore zone, which had been produced for the Albion testing and were therefore of high-sulphur grade, were dispatched to ALS for use in the preliminary pressure oxidation testwork. The Intermediate concentrate was assayed to contain 2.37% organic carbon. Additionally, samples of other materials, which had been treated by the Albion process and yielded poor results, were also sent to ALS for preliminary testing of an oxygenated lime boil treatment.

In summary, gold extraction from the pressure oxidation product solids was high in the ALS testwork, generally in excess of 93%, but the silver recovery was less than 10% due to the likely formation of silver jarosite in the pressure oxidation process. High silver extraction was achieved after subjecting the pressure oxidation product solids to a lime boil treatment, prior to cyanidation.

The encouraging results from the ALS testwork formed the basis for further investigation of the pressure oxidation processing route by Sherritt.

13.4.2 Ore Samples for Pressure Oxidation Testwork

To generate sufficient concentrate for detailed pressure oxidation testwork, ore samples from the same lots as were used to prepare flotation concentrates for the Albion testwork were sent to Wardell Armstrong International (WAI) in the UK. As the ores were from the same original lot as for the Albion testwork, the same four ore zone designations were maintained throughout the pressure oxidation testwork.

Because the material requirements for the pressure oxidation testing exceeded the ore samples remaining from the production of Albion flotation concentrates, additional drilling was completed in

2013 to generate sufficient drill core for completion of the pressure oxidation pilot testwork. Selected drill holes from the previous metallurgical drilling program were twinned to generate comparable ore samples. A summary of the drill core assays from the two campaigns is provided in Table 13-6.

Table 13-6: Comparative Assays for the Two Metallurgical Drilling Campaigns

Ore Zone	Ore Analysis						
	Au, g/t	Ag, g/t	Te, g/t	$C_{organic}$, %	S, %	As, %	Pb, %
East							
Phase 1	1.49	4.77	3.4	0.28	4.58	0.12	0.09
Phase 2	2.51	4.42	1.4	0.14	4.85	0.26	0.14
Intermediate							
Phase 1	1.77	10.7	12.0	0.39	5.03	0.05	0.11
Phase 2	2.14	10.0	8.6	0.29	5.83	0.17	0.16
Central							
Phase 1	1.64	12.3	34.8	0.46	4.85	0.02	0.07
Phase 2	2.03	13.2	30.2	0.43	4.74	0.04	0.08
West							
Phase 1	1.05	9.5	0.68	0.15	4.43	0.04	0.03
Phase 2	1.15	4.8	0.37	0.15	4.48	0.09	0.01

13.4.3 Batch Flotation Testing to Support Pressure Oxidation Testwork

Ore samples labelled as East, West, Central and Intermediate were dispatched to WAI over two separate time periods (referred as Phase One and Phase Two, as shown in Table 13-6). To generate flotation concentrates for the pressure oxidation testwork in a timely manner, Eldorado initially dispatched ore samples (Phase One) that remained from the preparation of bulk concentrate for use in the Albion testwork and reported by WAI in February 2013. Using these ore samples, WAI undertook confirmatory laboratory flotation tests with conditions largely derived from the previous program of detailed flotation testwork. A summary of the batch flotation test results is shown in Table 13-7.

Prior to flotation, the ore samples were comminuted by first jaw crushing to 12 mm, three stages of roll crushing to 1 mm, and rod milling to a P80 of 95 µm.

Table 13-7: Summary of Batch Flotation Test Results for Phase 1 Ores

Sample	Recovery to Concentrate, % (at a target concentrate grade of 13% S^{2-})	
	Au	S^{2-}
East	90.8	98.0
Central	98.2	99.0
Intermediate	94.5	98.1
West	95.8	96.7

The batch flotation tests were conducted under conditions which had largely been developed for the production of concentrate suitable for the Albion process, though operated to generate a lower sulphur grade concentrate, 12 to 14% S^{2-} for pressure oxidation versus 35 to 40% S^{2-} for the Albion process. Despite the less than optimized conditions, gold recoveries to the flotation concentrate were relatively good at better than 94%; the East ore sample responded slightly less favourably to flotation, as had been observed in previous testing.

In comparison to the production of high-sulphur grade concentrates for the Albion process, the production of low-sulphur grade concentrates for the pressure oxidation process resulted in a higher mass pull to the concentrate, which resulted in significantly improved gold and silver recoveries. Reported gold recoveries were of the order of 90% for a composite of the East, Central and Intermediate ores and about 88% for the West ore while producing the high-sulphur grade concentrates for the Albion process (though later determined not to be reproducible). Gold recoveries were 94% on average while producing the low-sulphur grade concentrates for the pressure oxidation process.

13.4.4 Bulk Concentrate Preparation

Following confirmatory batch flotation tests, the Certej ore samples were individually processed through WAI's pilot grinding and flotation plant to generate sufficient quantities of flotation concentrate for subsequent pressure oxidation testwork.

As was the case with the batch flotation testwork, the flotation pilot plant was operated to generate concentrates containing 12-14% S^{2-}. Some concentrates, which did not meet the target sulphur content, were also dispatched for pressure oxidation testwork to ensure sufficient sample was available. A summary of the results obtained from the pilot plant campaign is shown in Table 13-8 (the results are based on the combination of in-spec (12-14% S^{2-}) and out-of spec (<12% S^{2-}) concentrate which comprised the concentrate submitted for pressure oxidation testing). It should be noted that the flotation pilot plant operation was not intended to generate design criteria for the commercial equipment due to intermittent operations and difficulty controlling to the target operating conditions.

Table 13-8: Summary of Pilot Flotation Results

		Head Grade		Mass	Concentrate Grade		Recovery, %	
Phase	**Ore**	**Au, g/t**	**S^{2-}, %**	**Pull, %**	**Au, g/t**	**S^{2-}, %**	**Au**	**S^{2-}**
1	East	1.49	4.33	23.9	3.62	11.7	71.3	82.5
	Central	1.64	4.66	29.0	4.42	13.1	92.3	88.7
	Intermediate	1.77	4.87	29.6	4.66	12.8	88.7	88.8
	West	1.05	4.31	28.3	2.44	13.4	80.7	89.2
2	East	2.51	4.77	22.6	6.81	14.4	78.6	89.1
	Central	2.03	4.63	25.1	5.69	13.8	85.3	92.9
	Intermediate	2.14	5.78	30.3	5.00	14.2	85.8	94.5
	West	1.15	4.42	28.0	3.28	12.2	80.4	89.3

The concentrate samples produced by pilot flotation demonstrated a variable response, with gold recoveries of between 71.3% and 92.3% being achieved. The sulphide sulphur recoveries were relatively high for all samples tested, ranging from 82.5% in the Phase 1 East composite to 94.5% in the Phase 2 East composite. The pilot flotation work successfully generated concentrates in terms of the targeted sulphur grade of 12-14% S^{2-}, though it was not an accurate representation of the recoveries achievable under design conditions.

13.4.5 Gravity Recoverable Gold

Gravity recoverable gold (GRG) tests were undertaken by WAI on samples of the West ore zone from the Phase 1 and 2 lots, principally because the West ore was considered to be the least refractory, as shown in Table 13-5. The GRG content of the West ore samples was 28.6% and 40.7% for the Phase 1 and Phase 2 materials, respectively. The gravity concentrates were subjected to intensive cyanidation leach tests where all samples responded favourably to leaching. The overall recovery of gold to the gravity concentrate leachate was 24.9% and 36.8% for the Phase 1 and 2 samples, respectively. A flotation test was undertaken on the final gravity tailings where flotation recoveries of >88% were obtained. Thus, overall gold recovery to the gravity concentrate leachate and flotation concentrate was 90.4% and 89.3% for the Phase 1 and 2 samples respectively. As flotation of the West ore sample gave as good or better overall gold recovery as the combination of gravity separation and flotation, inclusion of a gravity concentration circuit in the process design was omitted.

13.4.6 Liquid-Solids Separation Testing for Flotation Tails

Solids liquid separation tests were completed by Pocock Industrial Inc. in Salt Lake City, USA, for samples of Central flotation tails, Intermediate flotation tails, East flotation tails and West flotation tails generated in the WAI pilot flotation testwork. Water was used to make the necessary dilutions during testing. The purpose for testing was to evaluate solid-liquid separation characteristics for thickener sizing and determination of thickener underflow density. A brief summary of some of the equipment sizing criteria and recommendations gleaned from the testing program is shown in Table 13-9:

Table 13-9: Summary of Liquid Solids Separation Design Criteria for Flotation Tailings Samples

Sample	Recommended Flocculant	Flocculant Dosage, g/t	Unit Area Requirement, $m^2/(t/d)$	Hydraulic Rate, $(m^3/h)/m^2$	Estimated Underflow Density, %
Central	Hychem AF304	40 – 45	0.47 - 0.52	3.2 - 4.0	52 – 53
Intermediate	Hychem AF304	35 – 45	0.46 - 0.51	4.0 – 4.6	57 – 58
East	Hychem AF304	30 – 35	0.71 - 0.76	3.6 – 4.1	57 – 58
West	Hychem AF304	25 – 30	0.49 - 0.54	3.6 – 4.2	49 - 51

A high density thickener was recommended for flotation tailings service in the commercial plant design.

13.5 METALLURGICAL TESTWORK – PRESSURE OXIDATION

In April 2013, Eldorado Gold commissioned Sherritt Technologies in Canada to complete batch and continuous pilot pressure oxidation testwork in support of the Certej Project development, using the bulk flotation concentrates which had been and were being produced by WAI. Complete results of the pressure oxidation testwork, as of the date of writing, have yet to be formally issued; a partial draft of the report ‘Pressure Oxidation Testwork Report for Certej Project’ was provided to Eldorado Gold in January 2014.

13.5.1 Batch Testwork

A batch test program was completed prior to the continuous pilot plant campaign with the following objectives:

- To generate sufficient data to allow for selection of a preferred flowsheet to treat the Certej concentrates and provide a preliminary definition of the process parameters for the associated unit operations (CCD, solution neutralization and cyanidation) to be included in the Certej project flowsheet.
- To assess gold liberation following the pressure oxidation of the Certej concentrate samples (as received or after regrinding) at variable temperature, oxygen pressure, retention time and chloride level in the quench solution.
- To evaluate the behavior of the concentrates in pressure oxidation, with respect to sulphur oxidation kinetics, gold and silver extraction with and without lime boil pre-treatment, and liquid solid separation.

As noted previously, two lots of the four distinct concentrates were supplied to Sherritt for use in the pressure oxidation testwork. For the purposes of most of the batch testwork, the first lot (Phase 1) was used to expedite the testing program. As the second lot (Phase 2) concentrates were made available, comparative tests were undertaken. Both phases of concentrates were blended together for use in the continuous pilot plant testwork. A summary of the blended concentrates as used in the pilot pressure oxidation testing is provided below. Also included in Table 13-10 is the assay of a 1:1:1:1 blend of the four different concentrates, which was used as the basis for the engineering design of the commercial process plant.

Table 13-10: Summary of Flotation Concentrate Assays as provided for Pilot Plant Pressure Oxidation Testwork

	Central	East	Intermediate	West	1:1:1:1 Blend
Au, g/t	5.43	5.27	5.30	3.47	4.87
Ag	32.50	12.0	23.0	21.7	22.3
Al, wt%	6.55	6.80	5.63	7.57	6.64
Ca	0.40	0.28	0.210	1.17	0.51
C	0.99	0.40	0.57	0.64	0.65
$C_{organic}$	0.77	0.28	0.46	0.09	0.40
Cl^-	0.00	0.00	0.0033	0.00	0.00
F^-	0.05	0.04	0.055	0.03	0.04
Fe	12.2	11.9	12.1	12.0	12.0
K	4.02	4.41	4.22	4.98	4.41
Si	23.0	22.6	22.8	20.9	22.3
S	13.8	14.1	14.6	12.9	13.8
$S(SO_4)$	0.05	0.08	0.08	0.11	0.08
Zn	0.85	1.49	1.96	0.30	1.14

Pressure oxidation tests were conducted on the individual concentrate types, and on blends of concentrates. The 1:1:1:1 blend, as in the table above, consisted of equal parts by dry solids weight of the four separate concentrates.

The batch pressure oxidation tests demonstrated rapid sulphide oxidation kinetics for each of the Certej concentrates under pressure oxidation conditions that have been proven at the commercial scale. Greater than 99% sulphide oxidation was obtained within 40 minutes of pressure oxidation at 220°C, and within 20 minutes at 230°C. Further, the initial batch testwork established that additional size reduction of the concentrate was not beneficial with respect to sulphur oxidation kinetics or gold recovery. The early batch testwork also established that the acidulation (pre-acidification of the concentrate prior to pressure oxidation) step would not be required as the carbonate content of the Certej concentrates was insignificant.

The bulk of the batch testwork was focused on determining the behaviour of organic carbon and soluble chloride in the pressure oxidation process and their effect on subsequent gold recovery in cyanide leaching. Gold recovery in cyanide leaching was significantly influenced by the chloride content of the pressure oxidation solution, regardless of whether the chloride was introduced with the quench (autoclave cooling) water or was dissolved from the concentrate, and the residual organic carbon content in the pressure oxidation product solids. The batch testwork showed that in the presence of chloride and after initial oxidation of the sulphides upon pressure oxidation, the likely formation of gold chloride complexes and subsequent adsorption/precipitation of the gold chloride complexes by organic carbon were detrimental to gold recovery. The batch testwork highlighted that conditions which prevented the dissolution of gold as a chloride complex (such as limited chloride in the pressure oxidation solutions), or which promoted the oxidation (or passivation) of organic carbon (such as extended retention time in the autoclave or higher temperature), mitigated the adverse effects of carbon and chloride on gold recovery.

As was the case in previous testing of the Certej materials, silver extraction from the pressure oxidation product solids was poor, typically less than 20%. However, following lime boil treatment of the oxidized solids, up to 90% of the silver was extracted in the cyanide leach step.

13.5.2 Continuous Pilot Plant Testwork

The amenability of the flotation concentrates to the pressure oxidation process was demonstrated in a 194 h continuous pilot plant campaign, which included pressure oxidation and conditioning, countercurrent decantation (CCD) washing, and solution neutralization unit operations, as well as concurrent batch lime boil and cyanidation testing of pressure oxidation product solids. The pilot plant flowsheet is depicted in Figure 13-1 below.

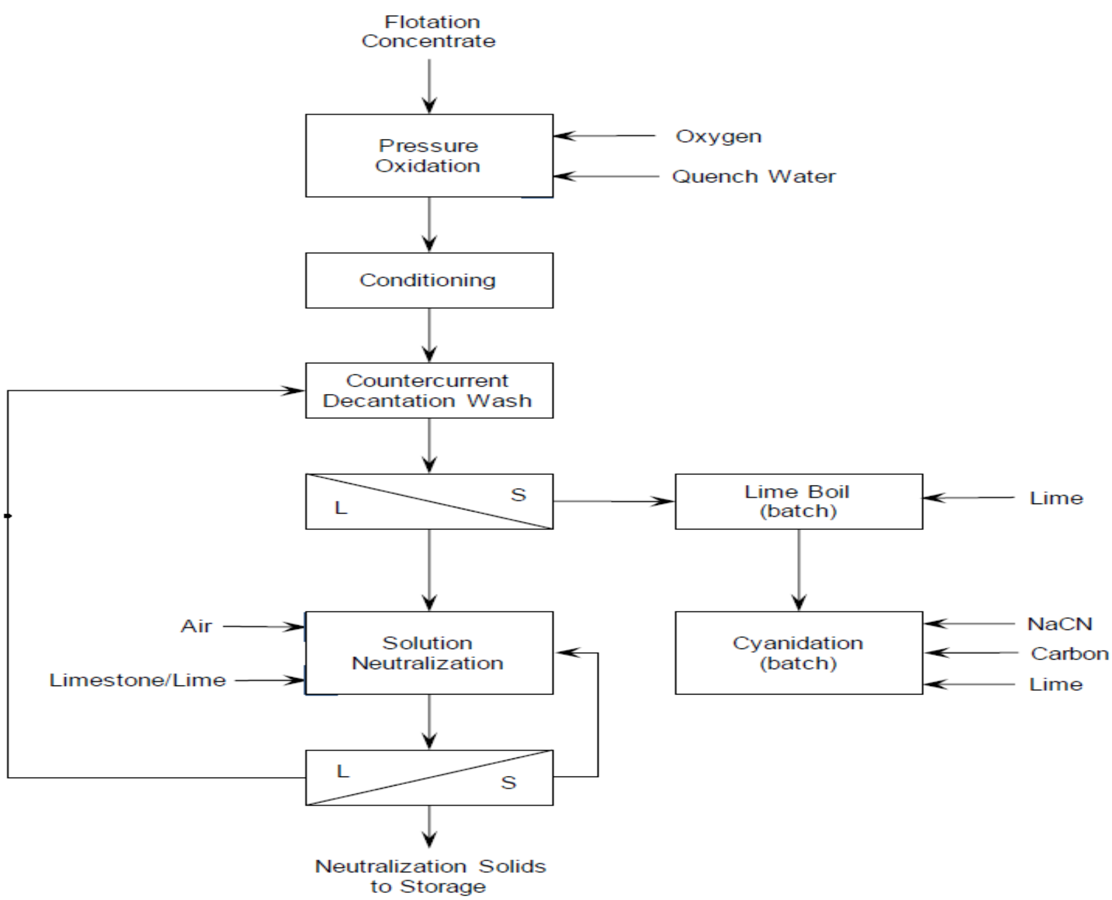


Figure 13-1: Continuous Pressure Oxidation Pilot Plant Flowsheet

Since the concentrates containing relatively high organic carbon content were expected to be most problematic with respect to gold extraction in cyanide leaching, and there was not enough of a single concentrate type to directly compare the effects of numerous pressure oxidation conditions during the pilot plant campaign, two blends of the concentrates were prepared specifically for the pilot plant campaign. These blends contained a 1:1 mixture of Central and Intermediate concentrates (High Carbon blend) and a 14.4% Central; 32.9% East, 52.7% Intermediate concentrate mixture (Medium Carbon blend).

A 1,000 kg shipment of limestone from a quarry within about 12 km of the Certej site was received by Sherritt for use in the solution neutralization step. A summary of the limestone's chemical analysis is provided in Table 13-11. Upon receipt, the limestone was ground in a ball mill and screened to 100% passing 150 µm; P80 of the final product was about 30 µm. The limestone was of good quality and responded well in the process.

Table 13-11: Limestone Analysis

Al, %	Ca, %	C, %	Cl-, g/t	F-, g/t	Fe, %	Mg, %	Si, %
<0.05	39.8	11.9	32	33	0.11	0.16	0.13

Operation of the continuous pressure oxidation pilot plant was divided into eleven operating periods, marked by specific changes to operating parameters or feed materials. A summary of the individual period conditions is provided in Table 13-12.

Table 13-12: Summary of Pilot Plant Pressure Oxidation Operating Parameters

Period	Concentrate	Temp., °C	Retention Time, min	Chloride, mg/L		Oxygen Add'n % of Stoich.
				Cooling Water	Discharge	
1	High C	230	89	15	26	200
2	High C	228	89	50	45	200
3	Intermediate	229	63	15	22	200
4	1:1:1:1	230	60	15	30	200
5	East	230	64	15	24	200
6	West	229	61	15	23	200
7	Central	229	62	50	39	200
8	High C	220	62	50	46	200
9	High C	220	60	15	24	200
10	Med. C	230	61	100	76	200
11	Med. C	230	64	100	76	120

13.5.3 Sulphide Oxidation

Sulphide oxidation was rapid for all feeds tested at both 220 and 230°C with 700 kPa oxygen partial pressure in the pilot plant autoclave, with typically 99.5% oxidation or greater in the final three autoclave compartments, corresponding to essentially complete sulphide oxidation within about 40 minutes in the autoclave. At 230°C with lower oxygen addition (120% stoichiometric) and lower oxygen partial pressure (430 kPa), average sulphide oxidation was somewhat slower.

As shown in Figure 13-2, variations in pressure oxidation conditions had little influence on the overall extent of sulphur oxidation, though it did modestly influence the oxidation kinetics. The following figure illustrates the sulphur oxidation and carbon oxidation reaction kinetics, and the corresponding gold extraction behaviour for Period 4, while treating the proposed design feed material (1:1:1:1 Blend of the four ore zones).

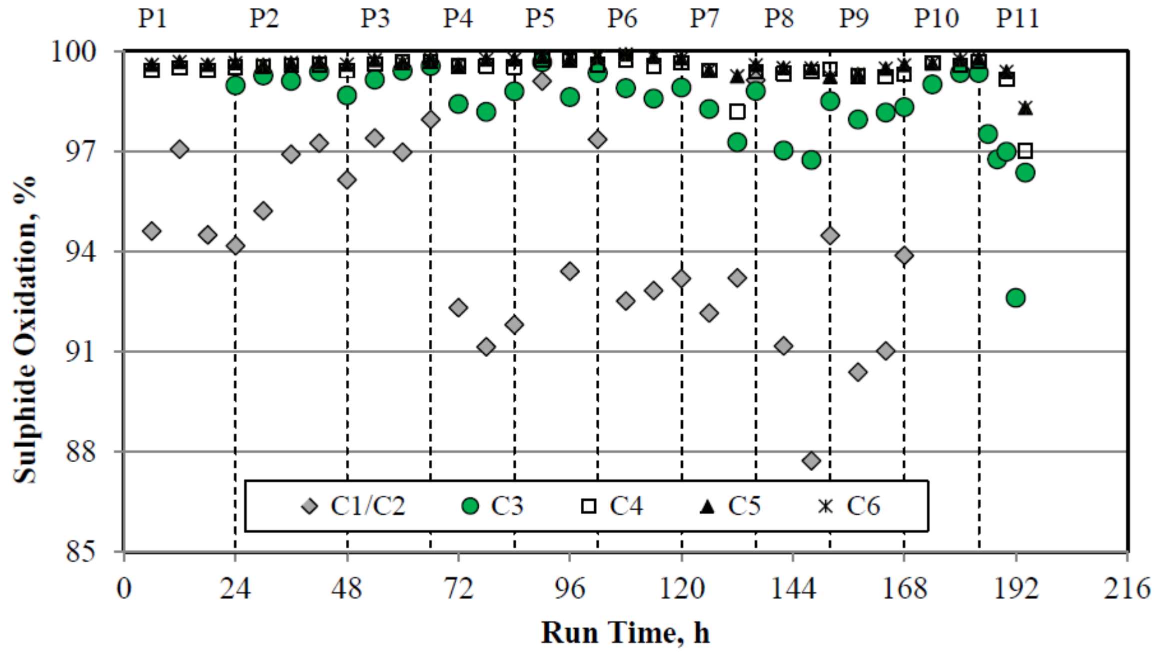


Figure 13-2: Sulphur Oxidation in the Continuous Pressure Oxidation Pilot Plant

The data displayed in Figure 13-3 demonstrate that a residence time of 60 minutes in the pressure oxidation step is sufficient for obtaining near complete sulphur oxidation, high carbon oxidation and accordingly result in high gold extraction in the subsequent cyanidation step.

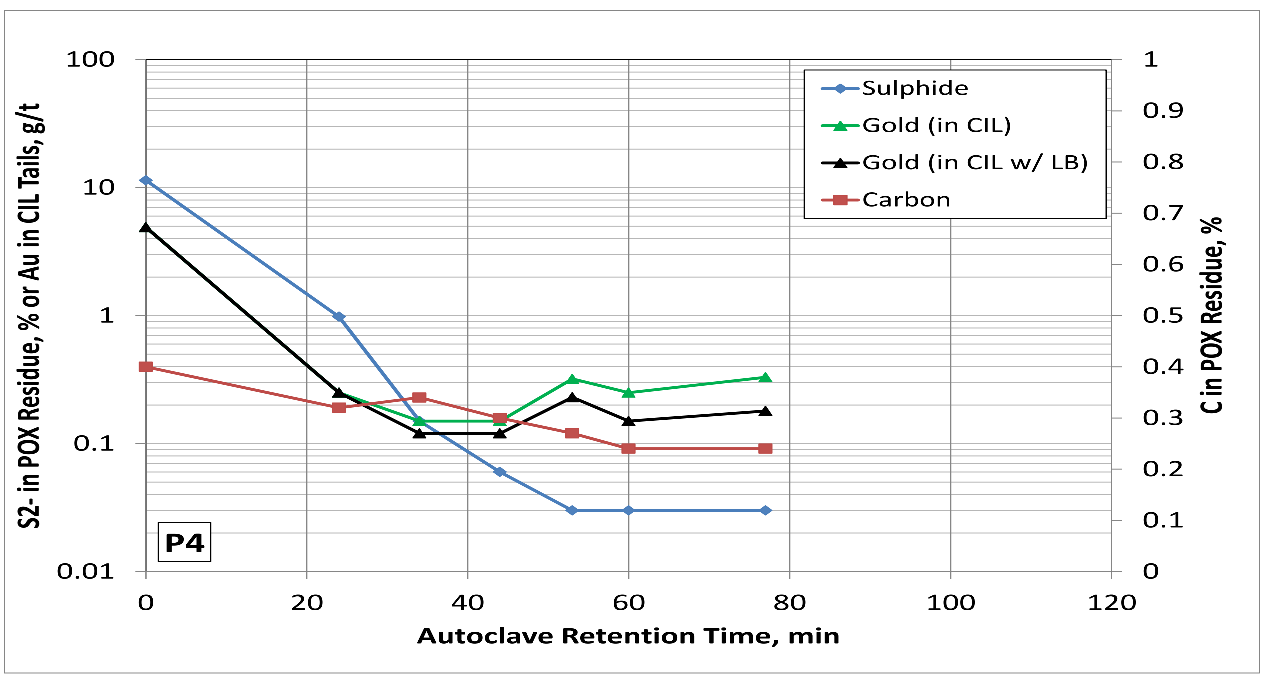


Figure 13-3: Sulphide Sulphur, Residual Organic Carbon and Gold in CIL Tails as a Function of Pressure Oxidation Retention Time

13.5.4 Gold Recovery

Gold recovery from the pressure oxidation product solids was about 95% when quench water containing about 15 mg/L chloride was used, including during treatment of a High Carbon blend at 220°C. Gold recovery from the higher carbon feeds decreased to as low as 90% when quench water containing 50 and 100 mg/L chloride was used, likely due to some gold being solubilized as a gold chloride complex and subsequently collected by the organic carbon in the solids ("preg-robbing"), and which was not recovered to a high degree in cyanide leaching.

Process parameters that likely affect gold recovery from pressure oxidation product solids include:

- concentrate type (mineralogy and organic carbon content);
- chloride concentration in the POX solution (dependent on chloride content in the feed concentrate and quench solution);
- autoclave temperature, oxygen pressure and retention time (affecting extent of sulphide oxidation and gold-chloride-carbon interaction under oxidizing conditions); and
- conditioning time (affecting redissolution or formation of precipitates that may occlude gold).

Despite changes in pressure oxidation parameters, gold recovery from the pressure oxidation product solids was consistently high for all of the feed materials tested. Further, gold recovery from the pressure oxidation product solids after conditioning remained essentially the same as that from the pressure oxidation product solids directly, and therefore conditioning (hot curing) did not have a negative impact on gold recovery. Similarly, the lime boil treatment had little effect on the extent of gold recovery and if anything, it slightly improved gold recovery as shown in Figure 13-4.

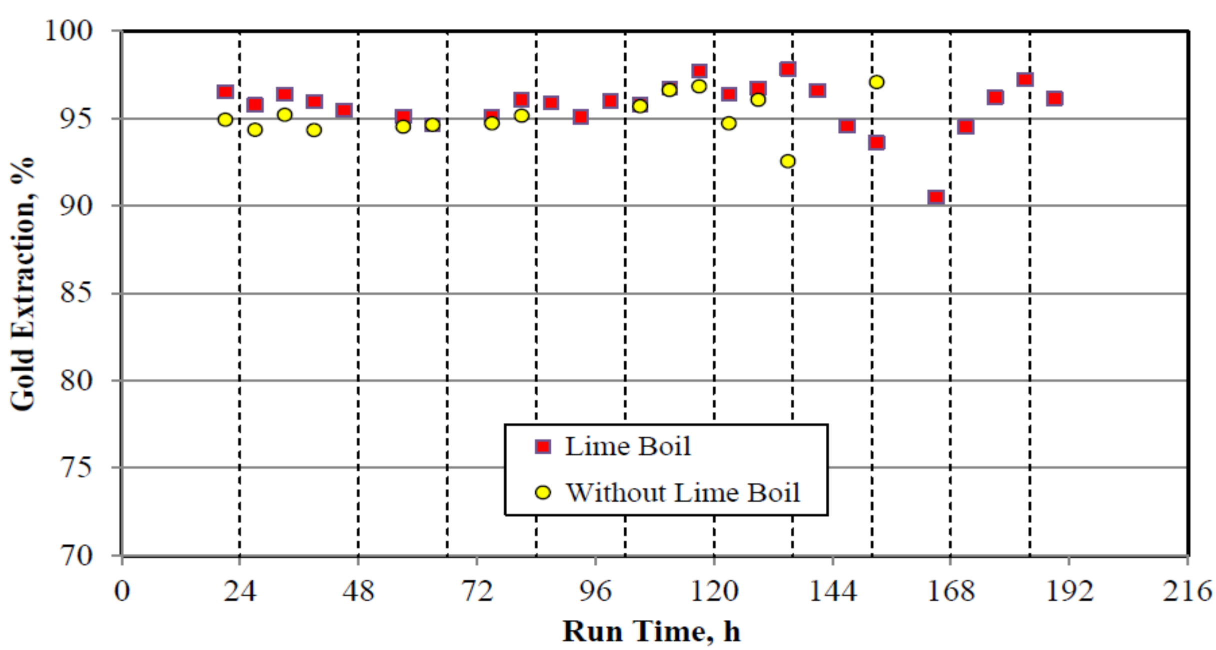


Figure 13-4: CIL Gold Recovery from Pressure Oxidation Product Solids

Regardless of pressure oxidation conditions, the unrecovered gold remaining in the CIL cyanidation residue was clearly dependent on the residual organic carbon content of the pressure oxidation product solids, as shown in Figure 13-5. The figure shows the general trend of increased gold content in the cyanidation tailings with increased carbon content in pressure oxidation product solids for most of the conditions tested. For pressure oxidation completed at lower temperature and chloride level (220°C, 15 mg/L Cl), there was less of an impact of the chloride-carbon "preg-robbing" behavior on the gold recovery.

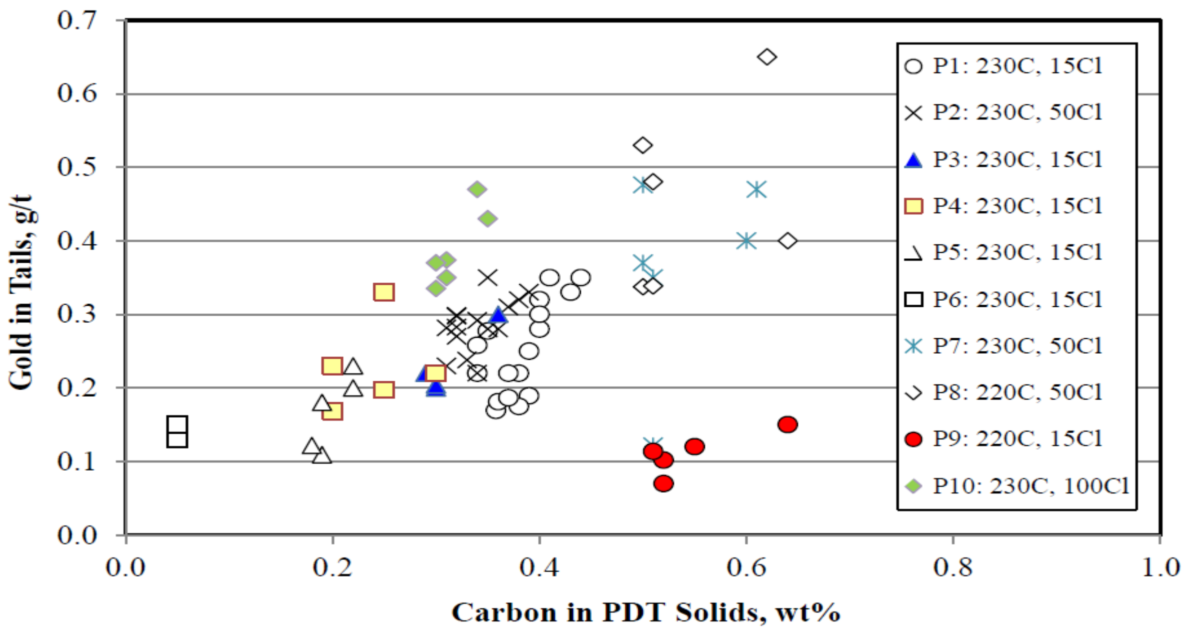


Figure 13-5: Gold in CIL Tailings as a Function of Residual Organic Carbon in Pressure Oxidation Product Solids

13.5.5 Silver Recovery

Silver recovery in cyanide leaching of the pressure oxidation production solids, after conditioning and without prior lime boil treatment, was low at about 25% or less. However, for all of the feeds and conditions tested, silver recovery from the pressure oxidation product solids was significantly increased, to the range of about 80 to 90%, with inclusion of a lime boil treatment prior to cyanide leaching as shown in Figure 13-6. This behaviour is consistent with previous testwork.

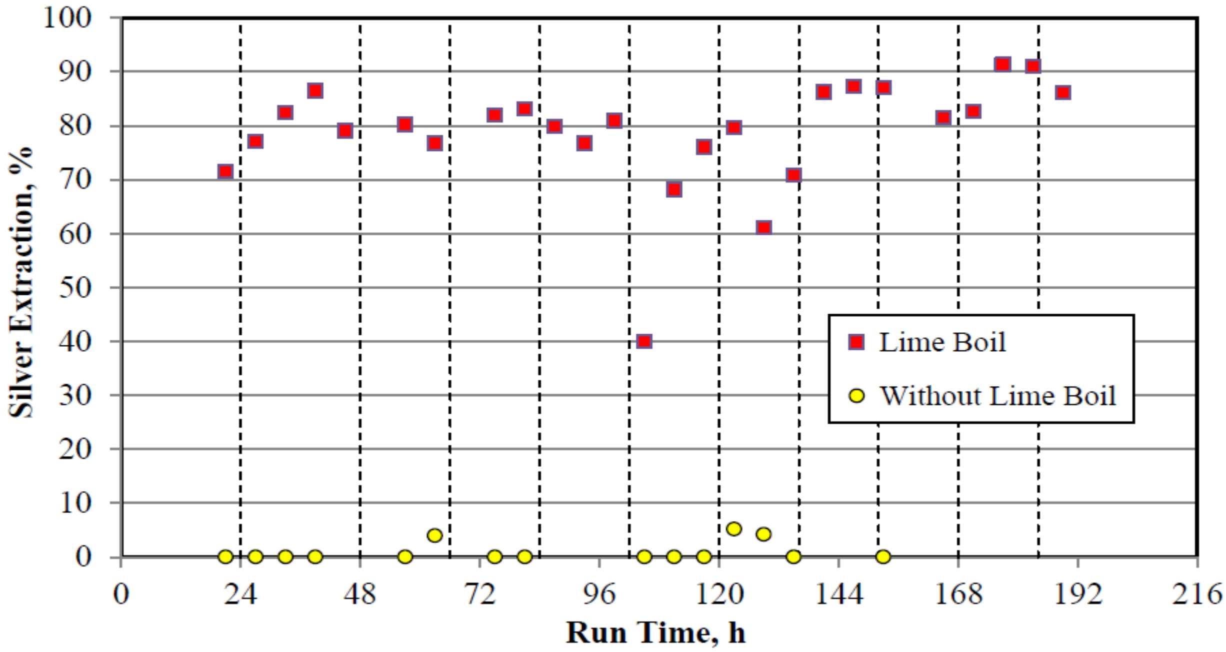


Figure 13-6: CIL Silver Recovery from Pressure Oxidation Product Solids

13.5.6 Conditioning (Hot Curing) and Lime Boil

Conditioning (also known as hot curing) refers to the process step in which pressure oxidation product solids are permitted to react with residual acid for a period, immediately following discharge from the autoclave, typically allowing the basic ferric sulphate in the pressure oxidation product solids to dissolve. This process step is included to minimize lime consumption in the subsequent lime boil and cyanide leaching steps, as shown in Figure 13-7.

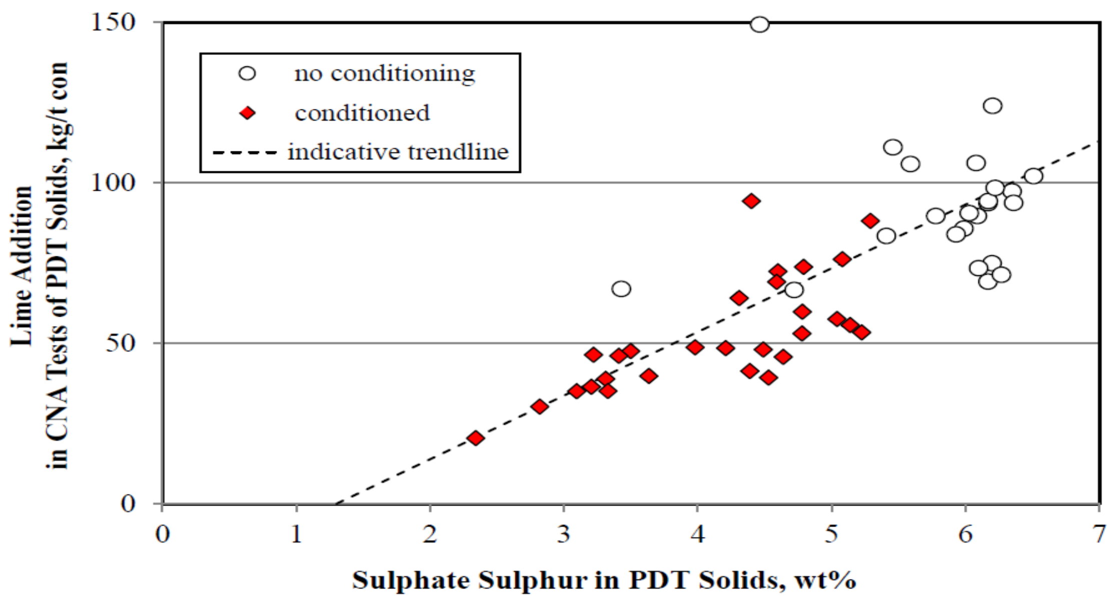


Figure 13-7: Lime Requirements in CIL Cyanidation Testing as a Function of Sulphate Sulphur in the Pressure Oxidation Product Solids

Conditioning was shown to have a negligible influence on subsequent gold recovery. Nevertheless, it did adversely affect silver recovery. Without conditioning, average silver recovery in cyanide leaching was typically about 20% or less. Silver recovery in cyanide leach was typically higher with shorter pressure oxidation retention time, likely due to the increased formation of silver jarosite with increasing retention time or with increasing sulfide oxidation in the autoclave. After batch conditioning of autoclave compartment samples, or continuous conditioning of the final autoclave discharge, almost all of the silver was apparently tied up with silver jarosite, as essentially no silver was extracted in cyanide leaching.

Regardless of silver behavior in the conditioning step, silver recovery was significantly increased after treatment of the conditioned, pressure oxidation product solids in a lime boil, prior to cyanide leaching.

13.5.7 CCD Wash

Pressure oxidation product solids were settled in the CCD wash circuit to about 26 to 39% solids in the thickener underflow. With wash ratios of 2.5 to 4.6 L of wash water per L of entrained solution in the final thickener underflow slurry, the extent of removal of the sulphuric acid and metal salts from the conditioned slurry during the pilot plant operation was between 97 and 99%.

Liquid solid separation tests, under carefully controlled conditions, were carried out by Pocock Industrial on fresh samples of conditioning tank discharge slurry during the pilot plant campaign. Underflow solids contents in conventional thickener tests were in the range of 34 to 38%. The Pocock results are presented in Section 13.7.

13.5.8 Pressure Oxidation Solution Neutralization

In the pressure oxidation solution neutralization circuit, CCD wash circuit overflow solution was reacted with limestone and slaked lime at 70°C to neutralize residual free acid and precipitate dissolved metals. A summary of the Solution Neutralization process results is provided in Table 13-13.

Table 13-13: Summary of Solution Neutralization Operating Results

Operating Period	A	B	C
Temperature, °C	70	70	70
Retention Time, h	6.5	3.9	3.4
Thickener U/F Recycle Solids rate, kg/h	--	3.65	--
Limestone Solids Rate, kg/h	0.92	1.36	2.31
Limestone Consumption, kg/t con	259	271	334
Limestone Utilization, %	94.1	92.7	93.4
Lime CaO Rate, kg/h	0.03	0.14	0.09
Lime Consumption, kg CaO/t con	8.7	21.8	12.7

With limestone additions to a target pH between 5.5 and 6.0 (measured at ambient temperature) and lime addition to a target pH of 8 (measured at ambient temperature), Al, As, Cu, Fe, Si and Zn were all precipitated to their respective detection limits (0.0005, 0.001, 0.005 and 0.01 g/L) in the neutralized product solution. The Ca, Mg and Mn concentrations in the thickener overflow were about 0.55, 0.20 and less than 0.01 g/L, respectively. The solution produced in solution neutralization was of an acceptable quality such that the solution can be recycled to the process, principally as wash water for the CCD wash circuit operation, but potentially as process water should the chloride concentration permit.

Limestone utilization in the first stage of neutralization was between 93 and 94%, resulting in consumption of about 340 kg limestone per tonne of concentrate, without recycle of thickener underflow slurry (Period C). Lime consumption (expressed as CaO) was about 20 kg/t concentrate feed when the 50% thickener underflows recycle was applied, compared with about 10 kg/t without recycle. Considering that the limestone utilization was quite high with or without underflow recycle, the underflow recycle would decrease limestone consumption in commercial practice by about the equivalent of the lime addition rate. That is, 340 kg/t limestone would be decreased to about 305 kg/t limestone if a 50% recycle of thickener underflow slurry were to be applied.

The average lime addition was about twice as much when the 50% recycle of thickener underflow slurry was employed. This is consistent with the species that are precipitated with lime in the second stage of neutralization, redissolved upon recycle to the lower pH of the first stage of neutralization, and then reprecipitated with lime in the second stage. With a 50% recycle, lime addition is therefore doubled.

The settling characteristics of the neutralized precipitate were exceptionally good, with or without recycle of thickener underflow to the first stage neutralization. The thickener underflow was of higher solids content, however, when the recycle was applied (about 55% solids with recycle compared with 45% solids without recycle). Liquid solid separation tests were carried out by Pocock Industrial on fresh samples of the solution neutralization discharge slurry during the pilot plant campaign. Underflow solids contents in conventional thickener tests were about 70% on Period B slurry with underflow recycle, and about 59% on Period C slurry, without underflow recycle. The Pocock results are provided in Section 13.7.

13.5.9 Cyanide Detoxification

For the purposes of generating sufficient solids for completing liquid-solids separation testing, large scale batch lime boil, CIL cyanidation and cyanide detoxification tests were completed using washed pressure oxidation product solids from the continuous pilot plant testwork. Conditions for these tests were based on commercial operating experience and were not optimized.

In the large scale lime boil tests, both limestone and lime were added in similar fashion to the commercial design. Limestone was added at a 50% stoichiometric excess, based on the calculated requirement for acid and acid equivalents (such as Al, Fe and Zn sulphates) in the entrained solution. Similarly, lime additions were made at a 200% stoichiometric excess, based on the sulphate content of the solids, less the sulphate associated with calcium (assuming calcium is present as gypsum only).

Lime boiled solids were then leached with cyanide in the presence of activated carbon (CIL), at a solids concentration of 25%. The cyanide leach was continued for 24 hours, while maintaining a sodium cyanide concentration of 2 g/L throughout.

In the cyanide detoxification step, sodium metabisulphite and sodium sulphite were added to the oxygen-sparged cyanide leach slurry to destroy cyanide. Copper sulphate was added to catalyze the process at a target addition of 0.25 g Cu/g of sodium cyanide.

The resulting slurry was submitted to Pocock for liquid-solids separation testing.

13.6 LIQUID SOLIDS SEPARATION TESTING FOR PRESSURE OXIDATION PRODUCTS

Solids liquid separation tests were completed by Pocock Industrial for selected samples of pressure oxidation product solids, solution neutralization precipitates and detoxified cyanidation tailings generated in the Sherritt pilot plant testwork. Decanted solution was used to make the necessary dilutions during testing. The purpose of testing was to evaluate solids-liquids separation characteristics for thickener sizing and determination of thickener underflow density. Complete test data sheets, figures and correlations have been issued only in draft form, but will be formally included in the 'Pressure Oxidation Testwork Report for Certej Project' to be provided by Sherritt in 2014.

A brief summary of some of the equipment sizing criteria and recommendations for the design cases is provided in Table 13-14:

Table 13-14: Summary of Liquid Solids Separation Design Criteria for Pressure Oxidation Process Samples Stemming from Treatment of the 1:1:1:1 Blend

Sample	Recommended Flocculant	Flocculant Dosage, g/t	Design Net Feed Loading, $(m^3/h)/m^2$	Estimated Suspended Solids, mg/L	Estimated Underflow Density, %
CCD Wash	Hychem CP905H	110 - 130	1.62	150 - 540	33.7
Solution Neutralization	Ciba Rheomax 1030	20 - 25	4.28	150 - 250	59.0
Detoxified CIL Tails	Hychem CP905H	60 - 70	3.38	150 - 250	38.5

The liquid-solids separation testwork resulted in the recommendation of high rate thickeners for the CCD wash and CIL Tails materials for use in the commercial plant design. The liquid solids separation testing also included paste thickening and filtration testing, to provide sufficient data for design should either of those equipment options prove advantageous.

13.7 SUPPLEMENTARY FLOTATION TESTWORK

Given the relatively poor flotation results for the East ore samples, a supplementary batch flotation test program was undertaken by WAI to determine if there was a potential to optimize flotation performance for this specific ore zone. The complete results were reported in 'Flotation Testwork on Samples from the Eastern Domain of the Certej Deposit, Romania' issued to Eldorado Gold in February 2014.

Ore samples used for the test program were those remaining from the previous flotation testwork for production of the bulk pressure oxidation concentrates.

Based on the results of the testwork, refining the flotation conditions led to a modest improvement in gold recovery from the East ore zone. Under conditions identified in earlier work, the East ore typically yielded 90% gold recovery to the concentrate. Under the best conditions, gold recovery was improved by about 2%, although selectivity was noted to decrease and the mass pull was slightly increased.

13.8 ORE VARIABILITY TESTING

As part of the current metallurgical testwork program, variability testing of the Certej ores will be undertaken to test the efficacy of the pressure oxidation process to treatment of a range of concentrates. Thirty one drill core samples were selected to represent the spatial and geological distribution of ore types within the Certej deposit. The variability testing is being undertaken in two parts, and is currently in progress.

13.8.1 Variability Flotation Testwork

The flotation testing portion of the ore variability testwork was completed by WAI in early 2014 and has not yet been formally reported. The flotation testwork was completed entirely in batch mode. For each ore sample, small scale flotation testing was completed to confirm the conditions required for producing a concentrate with the target sulphur grade for subsequent pressure oxidation testing. Following the small scale testing, several, larger bulk flotation tests were completed under identical conditions to produce a concentrate in sufficient quantities to allow pressure oxidation testing.

A summary of the median results for each previously delineated ore zone is provided in Table 13-15.

Table 13-15: Summary of Ore Variability Flotation Testwork Results.

Sample By Location	Number Of Samples	Initial pH	Low Concentrate (nominal 8% S)			High Concentrate (nominal 13%S)		
			Time min	S Grade %	Au Recovery %	Time min	S Grade %	Au Recovery %
Central	11	7.8	35	9.69	94.8	22	12.6	94.8
East	6	5.7	45	9.90	90.9	40	13.2	91.1
Intermediate	7	7.3	65	9.98	96.3	35	12.4	92.6
West	7	6.1	45	8.17	96.7	8	10.2	92.6

Flotation of the variability ore samples gave gold recoveries to the concentrate in the range of 91 to 95% when producing a concentrate containing about 13% S^{2-}. Median recoveries were slightly lower than those achieved in the bulk flotation testwork and some specific sample performed significantly worse.

Importantly, the variability flotation testwork demonstrated that the flotation residence time to produce a suitable concentrate for pressure oxidation and to achieve high gold recoveries was highly variable between samples. As such, the design was adjusted to provide significantly more flotation residence time in the commercial plant.

13.8.2 Pressure Oxidation Testing

The pressure oxidation testing of variability concentrates is planned to start in Q2 2014, results from which will be incorporated into the feasibility study.

13.9 SUMMARY OF RECOVERIES

The flotation and pressure oxidation testwork completed in support of the Certej project development have demonstrated overall recovery of gold and silver achieved from each of the four distinct ore zones and for a blend of each. A summary of the results is provided in Table 13-16.

Table 13-16: Summary of Overall Recovery of Gold and Silver from Ore

Testwork Result		1:1:1:1 Blend	Design Basis
Flotation Recoveries (from Ore), %	Au	94.8	93.0
	Ag*	97.7	93.0
Flotation Concentrate Grades, g/t	Au	4.87	4.7
	Ag	22.3	30.9
Pressure Oxidation Recoveries (from Con), %	Au	96.0	94.0
	Ag	81.7[1]	86.0
Overall Recoveries (form Ore), %	Au	91.0	87.4
	Ag	79.8	80.0

[1] Note that the Ag content of concentrate used in the metallurgical testwork was considerably lower than that of the anticipated commercial plant; 22 g/t versus 32 g/t, giving rise to lower recovery of Ag in the pressure oxidation process than is expected in the operating plant.

Each of the variable ore zone samples responded favourably to flotation, pressure oxidation and CIL cyanidation for recovery of the contained gold. Silver recovery was also high given that an alkaline treatment (lime boil) step was included in the process flowsheet prior to cyanidation. Further, there were apparently no deleterious interactions between ore types.

As demonstrated by the recent metallurgical testwork, the variability of the concentrates to pressure oxidation conditions was relatively insignificant. However, both organic carbon and soluble chloride were shown to inhibit gold recovery. The conditions selected as the basis for commercial design of the process plant have been chosen to specifically mitigate the adverse effects of chloride and organic carbon.

13.10 TECHNICAL UNCERTAINTIES AND FUTURE WORK

Additional mineral processing and metallurgical testwork are required as part of the Feasibility Study and detailed design of the Certej process plant.

13.10.1 Continuous Pilot-Scale Post Pressure Oxidation Process Testwork

Metallurgical testwork to further define the preferred conditions of the precious metal recovery circuits in the flowsheet, including principally the alkaline treatment (lime boil) and cyanidation circuits, and to generate sufficient data for optimal design of those commercial operations will be undertaken as part of the Feasibility Study. At the time of writing, testwork has been planned and is expected to be undertaken in the course of 2014.

13.10.2 Environmental Testing

Geochemical and geotechnical testing of the process plant effluent streams, namely the flotation tailings, the CIL tailing and the neutralization residue are planned for 2014. Limited geochemical testing of selected effluent streams is currently in progress. Likewise, additional environmental testwork to demonstrate the stability of the tailings residues and to generate data for detailed design of the tailings impoundment and effluent water treatment facilities may be required as part of the feasibility study.

SECTION • 14 MINERAL RESOURCE ESTIMATES

The mineral resource estimate for the Certej deposit used data from surface diamond drill and reverse circulation holes, and underground channel sampling. The resource estimate was made from a 3D block model created utilizing commercial mine planning software. The block model cell size was 10 m east by 10 m north by 5 m high.

14.1 GEOLOGIC MODELS

Eldorado used significant new drill data from its 2012 and 2013 drill campaigns to update the geologic and structural models. These models were used to guide and validate the mineralized shells created for the grade interpolation. Most significant was a much better understanding of the deposit bounding and, in areas, mineral hosting Andesite intrusive units and a better definition between the Cretaceous and Neogene sediments, in particular the respective Breccia units found near or at their contact.

Eldorado created 3D mineralized envelopes, or shells, to constrain gold grade interpolation. These were based on initial outlines derived by a method of probability-assisted constrained kriging (PACK). The threshold value of 0.20 g/t Au was determined by inspection of histograms, probability curves, and indicator variography. Shell outline selection was done by inspecting contoured probability values. These shapes were then edited on plan and section views to be consistent with the geology model and drill hole data, such that the boundaries did not violate data or current geologic understanding of mineralization controls. Figure 14-1 and Figure 14-2 show examples of the relationship between the PACK or mineralized shell and the geology model.

All generated 3D shapes were checked for interpretational consistency on section and plan, and found to have been properly constructed. The shapes honoured the drill data and appear well constructed.

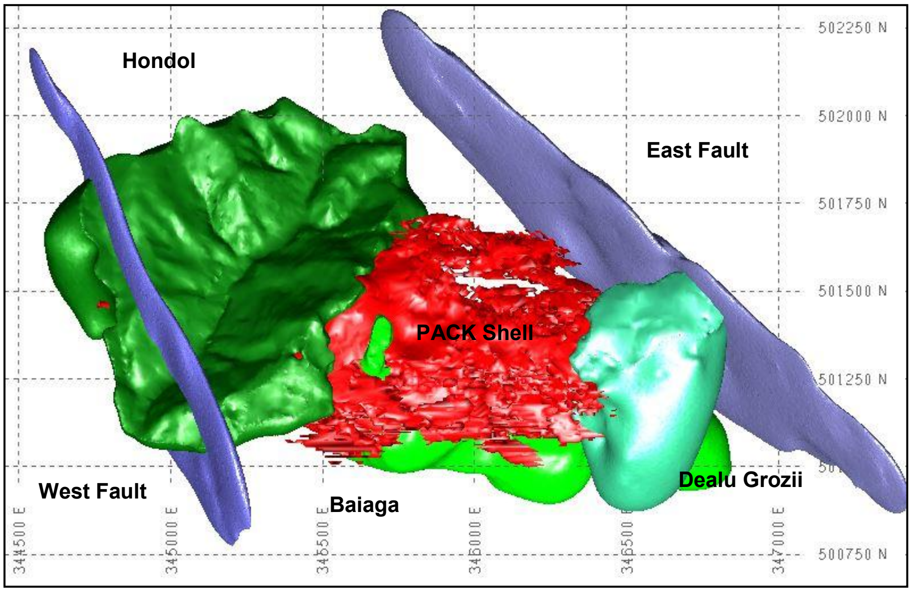


Figure 14-1: Relationship between Mineralized or PACK shell and Main Geologic Units. Shell relative to the intrusive units.

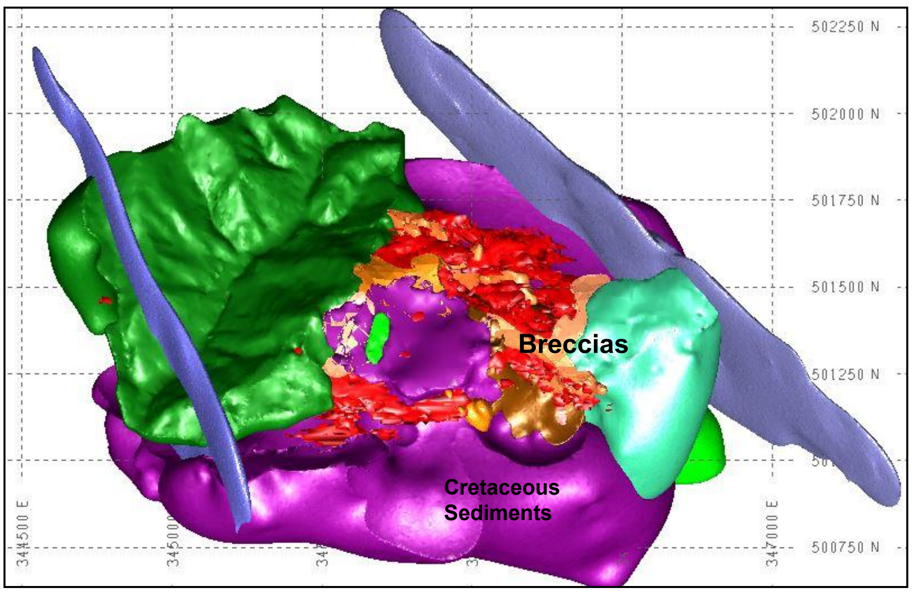
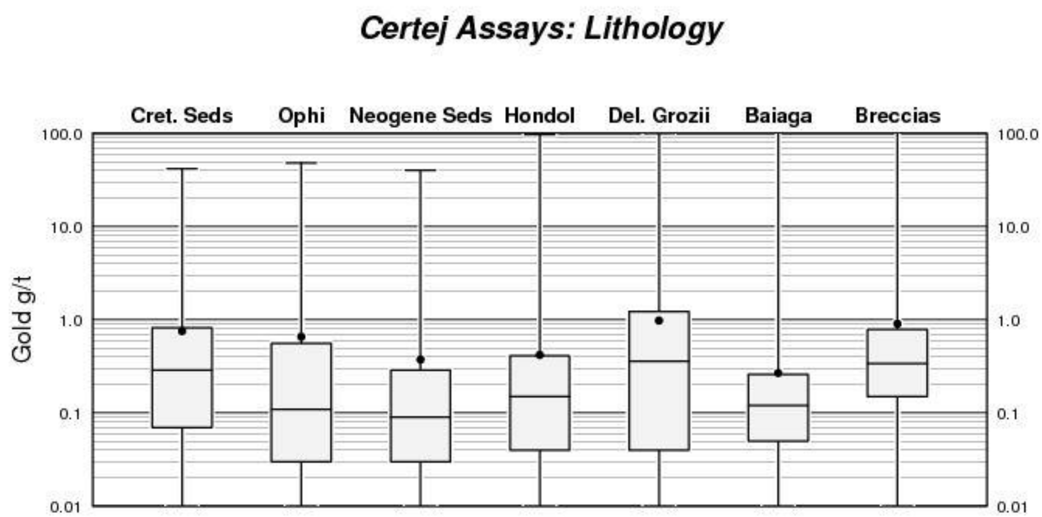


Figure 14-2: Relationship between Mineralized or PACK shell and Main Geologic Units. With Cretaceous sediments and Breccia units superimposed.

14.2 DATA ANALYSIS

The various geology and mineralized domains were reviewed to determine appropriate estimation or grade interpolation parameters. Several different procedures were applied to the data to discover whether statistically distinct domains could be defined using the available geological objects. The domains were statistically analyzed, both independently and relative to the mineralized shell. Results show that in the majority of cases the gold mineralization transcends lithology contacts and that the best way to capture the mineralization is by the mineralized or PACK shell. Statistical properties of the geologic units and Mineralized shell (PACK shell) are shown in Figure 14-3 and summarized in Table 14-1.



	Cret. Seds	Ophi	Neogene Seds	Hondol	Del. Grozii	Baiaga	Breccias
Number of data	10150	1773	9905	9940	3038	5336	1359
Mean	0.758	0.66	0.375	0.421	0.984	0.269	0.907
Variance	2.413	4.434	1.085	3.334	9.016	3.858	24.804
Skewness	9.397	14.12	12.31	32.508	24.765	65.189	25.399
Coef of Variation	2.05	3.189	2.775	4.342	3.052	7.314	5.491
Maximum	41.77	48.52	40.29	97.84	100.6	138.2	152.5
Upper quartile	0.82	0.56	0.29	0.41	1.23	0.26	0.79
Median	0.29	0.11	0.09	0.15	0.36	0.12	0.34
Lower quartile	0.07	0.03	0.03	0.04	0.04	0.05	0.15
Minimum	0.01	0.01	0.01	0.01	0.01	0.01	0.01

Figure 14-3: Certej deposit statistics for drill hole assays, displayed as boxplots, for the main geologic units: Cretaceous Sediments (“Cret. Seds”), Ophiolites (“Ophi”), Neogene Sediments (“Neogene Seds”), Hondol Andesite (“Hondol”), Dealu Grozii Andesite (“Del. Grozii”), Baiaga Andesite (“Baiaga”) and Breccias

Table 14-1: Certej Deposit Statistics for Drill Hole Composites – Au g/t Data

Domain	Mean	CV	Q25	q50	q75	Max	No. of Composites
Within PACK Shell	1.02	2.52	0.26	0.50	1.07	138.4	21,656
Background	0.09	4.42	0.02	0.05	0.10	70.2	21,896

Variography, a continuation of data analysis, is the study of the spatial variability of an attribute. Eldorado prefers to use a correlogram, rather than the traditional variogram, because it is less sensitive to outliers and is normalized to the variance of data used for a given lag. Correlograms were calculated for gold in the PACK shell. Gold in this domain display two structures: a longer-ranged, E-W trending, moderate steeply N-dipping and moderately S-plunging structure, and a sub-horizontal, NW-SE trending, gently SW-plunging, distinctly shorter-ranged structure. The nugget effect is moderate.

14.2.1 Evaluation of Extreme Grades

Extreme grades were examined for gold by histograms and cumulative probability plots, and by a risk-to-production simulation. The examination showed a risk does exist with respect to extreme gold grades at Certej. To mitigate this risk, a hybrid method involving an assay grade cap and a composite grade restriction protocol was implemented. Gold grades were capped to 70 g/t in the assay data prior to compositing. During grade interpolation a further restriction of 45 g/t over 40m was instituted where grades over this threshold were not used when further than 40m from a model block centre. This combination grade cap resulted in a difference of about 5 percent metal loss relative to uncapped / unrestricted data.

14.3 MODEL SETUP

The block size for the Certej model was selected based on mining selectivity considerations (open pit mining and underground drift and fill methods). It was assumed that the smallest block size that could be selectively mined as ore or waste, referred to as the selective mining unit (SMU), was approximately 10 m by 10 m by 5 m. In this case, the SMU grade-tonnage curves predicted by the restricted estimation process adequately represented the likely grade-tonnage distribution.

The assays were composited into 3.0 m fixed-length down-hole composites. The composite data were back-tagged by the mineralized shell on a majority code basis. The compositing process and subsequent back-tagging was reviewed and found to have performed as expected.

Various coding was done on the block model in preparation for grade interpolation. The block model was coded according to the mineralized shell on a majority code basis. Percent below sub-crop surface was also calculated into the model blocks (to negate the effect of overburden and old mine waste rock dumps) as well as percent mined from pre-existing underground exploration development.

14.4 ESTIMATION

Modelling consisted of grade interpolation by ordinary kriging (OK) for all domains. Nearest-neighbour (NN) grades were also interpolated for validation purposes. Blocks and composites were matched on estimation domain.

The search ellipsoids were oriented preferentially to the orientation of the PACK shell. Search ranges were 250m along E-W, 75m N-S and 125m vertically. Block discretization was 3 by 3 by 3.

A two-pass approach was instituted for interpolation. The first pass required a minimum of two holes from the same estimation domain, while the second pass allowed a single hole to place a grade estimate in any uninterpolated block from the first pass. This approach was used to enable most blocks to receive a grade estimate, including the background domain. Blocks received a minimum of 4 composites, within which a maximum of 3 composites were from a single drill hole (for the two-hole minimum pass). Maximum composite limit was 15.

Silver grades were also interpolated into the gold PACK shell: the same approach and parameters were used as for gold. Bulk density was assigned to the model by lithology, based on average values from over 4000 measurements. The values used are shown in Table 14-2.

Table 14-2: Average Bulk Density values for modeled lithologies, Certej

Lithology	Average Bulk Density
Cretaceous Sediments	2.47
Ophiolites	3.00
Neogene Sediments	2.37
Hondol Andesite	2.44
Dealu Grozii Andesite	2.37
Baiaga Andesite	2.44
Breccias	2.35

The model parameters were based on the geological interpretation, data analyses, and variogram analyses. The number of composites ultimately used in estimating grade into a model block followed a philosophy of restricting the number of samples for local estimation. Eldorado has found this to be an effective method of reducing smoothing and producing estimates that match the Discrete Gaussian or Hermitian polynomial change-of-support model, and ultimately the actual recovered grade-tonnage distributions.

14.5 VALIDATION

14.5.1 Visual Inspection

Eldorado completed a detailed visual validation of the Certej resource model. The model was checked for proper coding of drill hole intervals and block model cells in both section and plan. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values. Examples of representative sections containing block model gold grades, drill hole composite gold values, and domain outlines are shown Figure 14-4 through Figure 14-7.

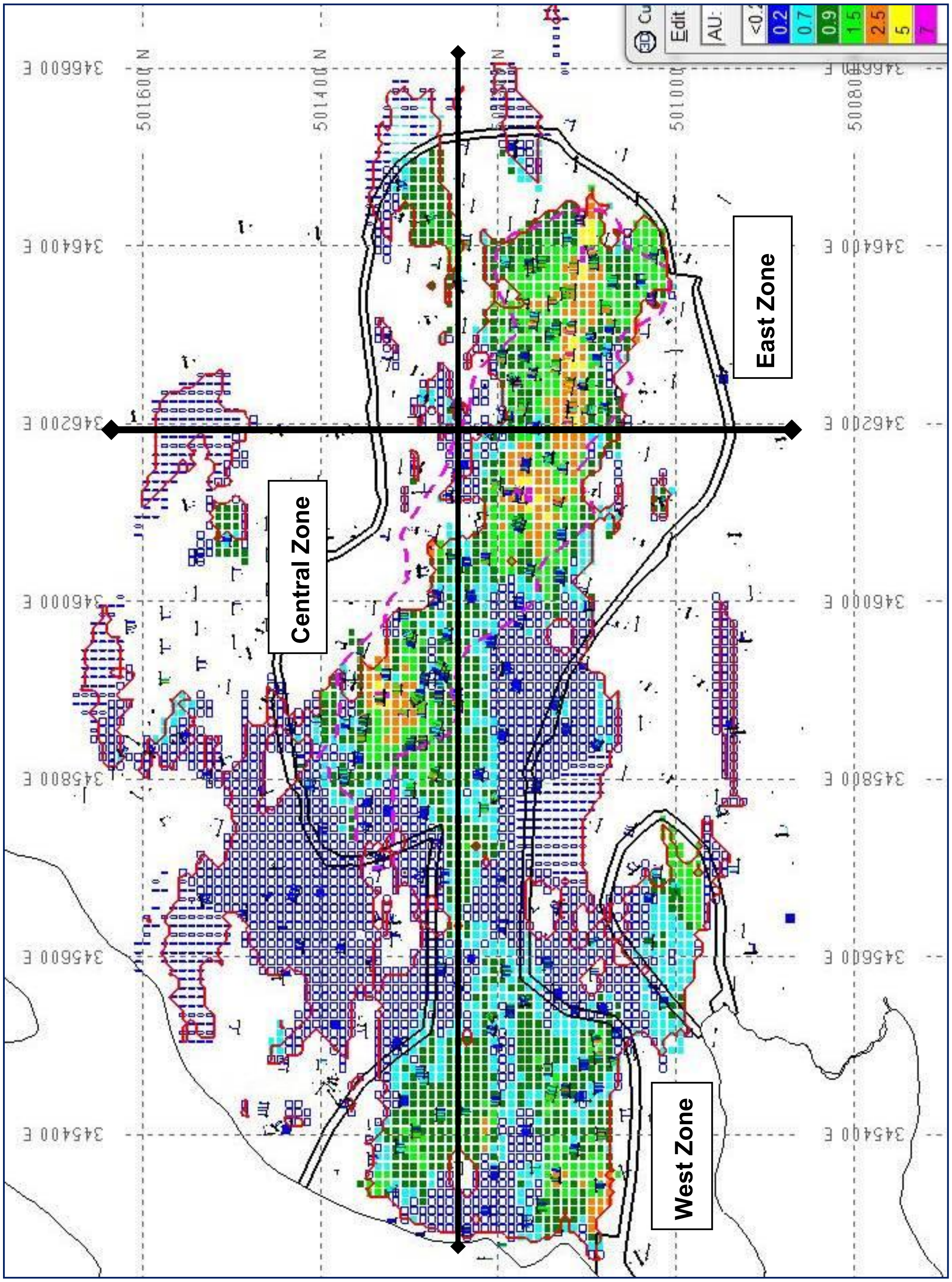


Figure 14-4: Certej Plan 460 rL showing Gold Grade Block Model and Drill Hole Traces (double black lines = 2013 reserve design pit; red outline = PACK shell; narrow blocks = Inferred)

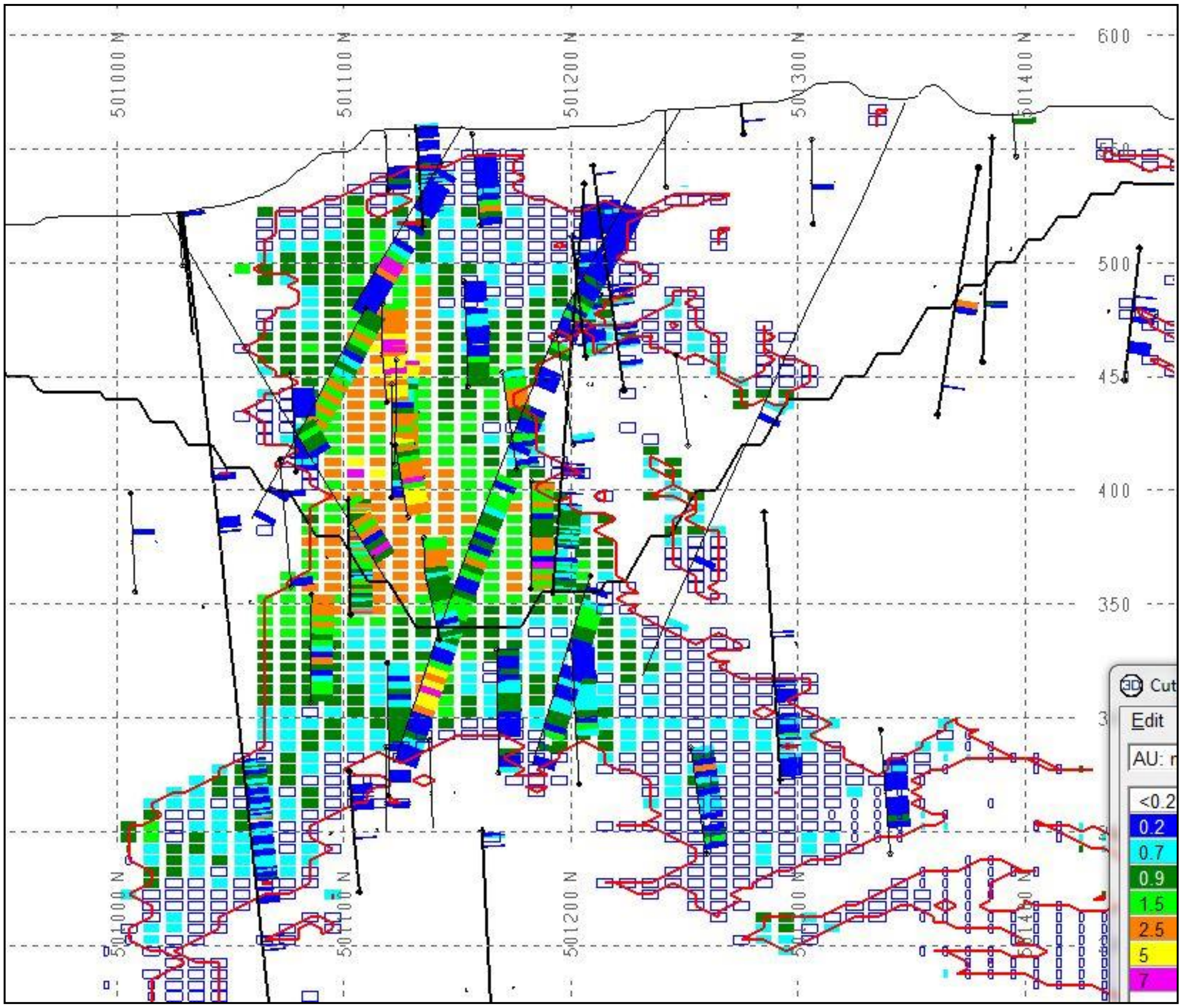


Figure 14-5: Certej N-S section 346200E showing Gold Grade Block Model and Drill Hole Traces (bold black line = 2013 reserve design pit; red outline = PACK shell; narrow blocks = Inferred)

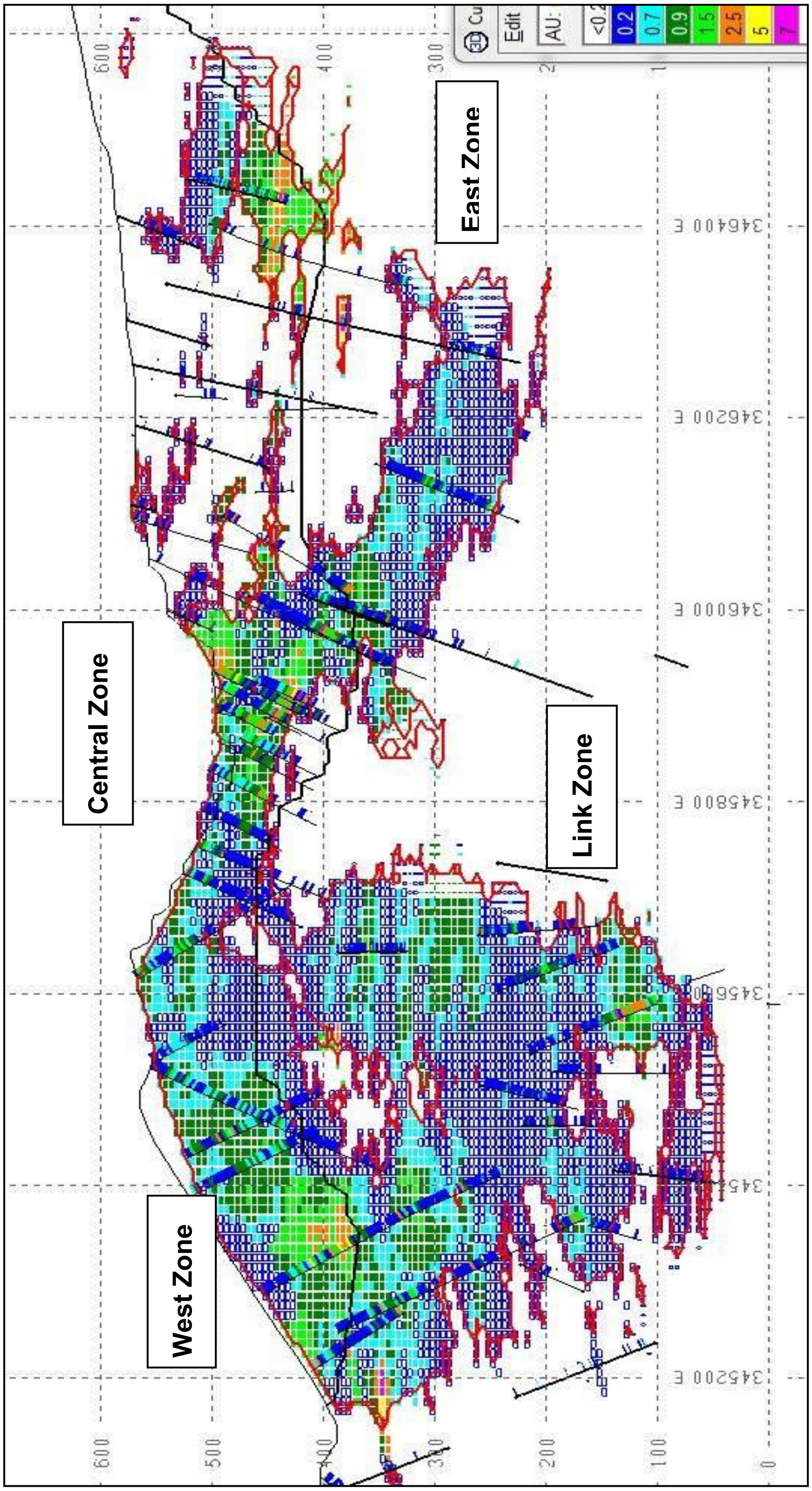


Figure 14-6: Certej W – E section 501300N showing Gold Grade Block and Drill Hole Traces (bold line = 2013 reserve design pit, red outline = PACK shell, narrow blocks = Inferred)

14.5.2 Model Check for Change-of-Support

An independent check on the smoothing in the estimates was made using the Discrete Gaussian or Hermitian polynominal change-of-support method. This method uses the declustered distribution of composite grades from a NN or polygonal model to predict the distribution of grades in blocks. The histogram for the blocks is derived from two calculations:

- the block-to-block, or .
- between-block variance.

The frequency distribution for the composite grades transformed by means of Hermite polynomials (Herco) into a less skewed distribution with the same mean as the declustered grade distribution and with the block-to-block variance of the grades.

The distribution of hypothetical block grades derived by the Herco method is then compared to the estimated grade distribution to be validated by means of grade-tonnage curves.

The grade-tonnage predictions produced for the model show that grade and tonnage estimates are validated by the change-of-support calculations over the range of likely mining grade cut-off values (0.7 to 1.0 g/t Au) shown in Figure 14-7.

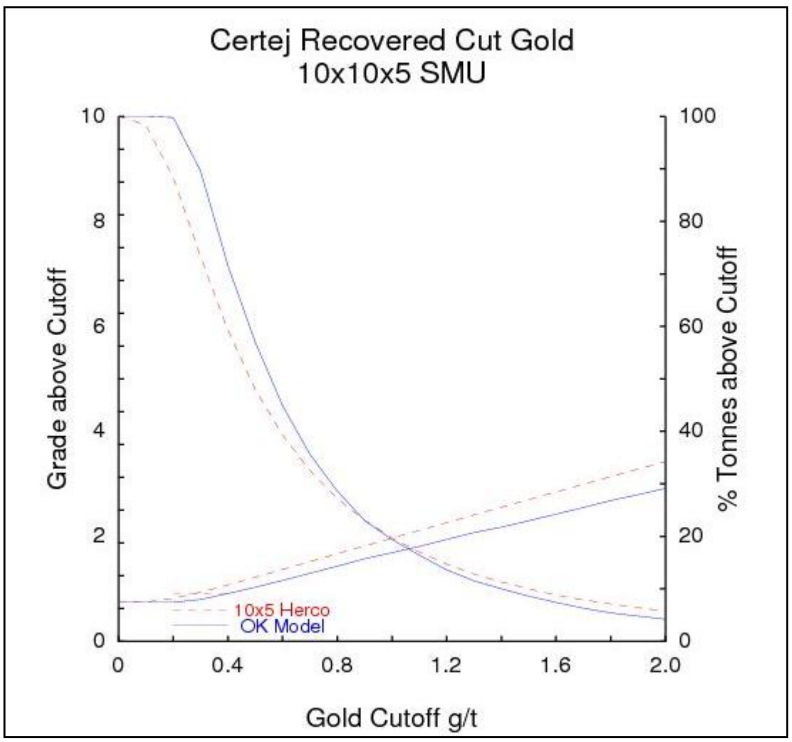


Figure 14-7: Herco Plot for Inside PACK Shell Au Grade Estimate, Certej

14.5.3 Model Checks for Bias

The block model estimates were checked for global bias by comparing the average metal grades (with no cutoff) from the model with mean grades from NN estimates. (The NN estimator declusters the data and produces a theoretically unbiased estimate of the average value when no cutoff grade is imposed and is a good basis for checking the performance of different estimation methods). The results, summarized in Table 14-3, show no problems with global bias in the estimates.

Table 14-3: Global Model Mean Gold Values, inside PACK shell

Domain	NN Estimate	Kriged Estimate	% Difference
Capped estimate	0.82 g/t	0.83 g/t	+1.2 %
Uncapped estimate	0.86 g/t	0.87 g/t	+1.2 %

The model was also checked for local trends in the grade estimates by grade slice or swath checks. This was done by plotting the mean values from the NN estimate against the kriged results on northings, eastings and elevations (all in 30 m swaths). The kriged estimate should be smoother than the NN estimate, which should therefore fluctuate around the kriged estimate on the plots. The observed trends behave as predicted, showing no significant trends of gold in the estimates in the Certej model. The chart showing trends along Northings is shown in Figure 14-8.

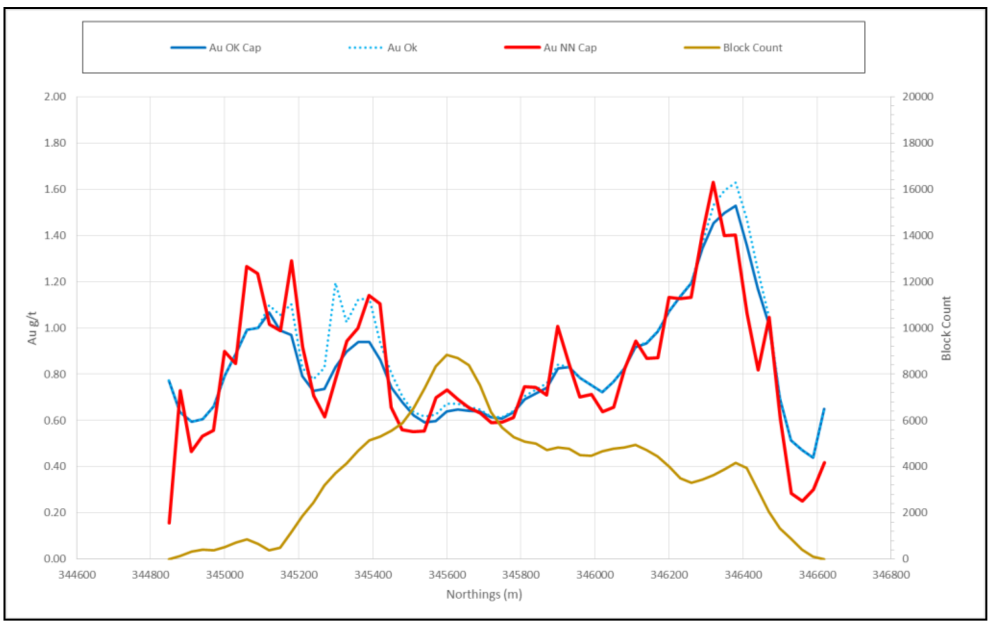


Figure 14-8: Model trend plot showing 30m binned averages along northings for kriged (OK) and nearest neighbour (NN) gold grade estimates, Certej deposit

14.6 MINERAL RESOURCE CLASSIFICATION

The mineral resources of the Certej deposit were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified into Measured, Indicated, and Inferred mineral resource categories.

Inspection of the Certej model and drill hole data on plans and sections, combined with spatial statistical work and investigation of confidence limits in predicting planned annual and quarterly production, contributed to the setup of distance-to-nearest-composite protocols to help guide the assignment of blocks into Measured or Indicated mineral resource categories.

Reasonable grade and geologic continuity is demonstrated over portions of the Certej deposit, which are drilled generally on 30 m to 40 m spaced sections. A multiple-hole rule was used, whereby blocks containing an estimate resulting from samples from two or more drill holes within 45 m from a model block center were classified as Indicated mineral resources. A region in the eastern area of the deposit has undergone a notably more dense level of drilling and as such qualifies to be classified as Measured mineral resources. Blocks in this area were classified as Measured mineral resources where the estimate resulted from samples of three or more drill holes lying within 15 m from the model block center. Both Indicated and Measured mineral resource classifications were input into the block model by constructed 3D shells based on the above criteria.

All remaining model blocks containing a gold grade estimate were assigned as Inferred mineral resources.

A test of reasonableness for the expectation of economic extraction was made on the Certej Indicated and Measured mineral resources by developing a series of open pit designs based on optimal operational parameters and gold and silver price assumptions. Those pit designs enveloped most of the Measured and Indicated mineral resources, thus demonstrating the economic reasonableness test for the estimate and reporting cut-off grade of the Certej mineral resources.

14.7 MINERAL RESOURCE SUMMARY

The Certej mineral resources as of December 31, 2013 are shown in Table 14-4. The Certej mineral resource is reported at a 0.7 g/t Au cut-off.

Table 14-4: Certej Mineral Resources, as of 31 December 2013

Mineral Resource Category	Tonnes (x „000)	Grade (Au g/t)	In Situ Gold (oz x „000)	Grade (Ag g/t)	In Situ Silver (oz x „000)
Measured	25,680	1.75	1,448	9	7,150
Indicated	85,435	1.23	3,368	9	24,611
Measured+Indicated	111,115	1.35	4,816	9	31,761
Inferred	29,002	1.08	1,010	6	5,268

SECTION • 15 MINERAL RESERVE ESTIMATES

The open pit optimisation, pit design work, inclusive of the final pit design, and mine scheduling was completed using MineSight™ software.

The Mineral Reserve estimate has been calculated and classified in compliance to the CIM definition of Mineral Reserve.

The Resource model as referenced in Section 14 of this report was used as the block model input for the Mineral Reserve estimates. The modelling methods, grade models, resource classification, existing underground development void modelling, and density model were reviewed by the QP of Section 15 and found appropriate for reserves estimation. Only blocks classified as Measured or Indicated were used in the pit optimisation and reserve reporting.

The open pit optimisation was performed using the Lerchs-Grossman algorithm as described in Section 15-2. The most financially beneficial shell formed the basis for the reserve pit, which was created and scheduled using all relevant design criteria.

15.1 MINERAL RESERVE CLASSIFICATION AND SUMMARY

The Mineral Reserves are the Measured and Indicated Resource blocks that are within the reserve pit design and above the ore cut-off grade. This amounts to a total Proven and Probable Mineral Reserves estimate of 46.984 Mt at a grade of 1.63 g/t Au and 11 g/t Ag. Table 15-1 provides further details of the Mineral Reserves estimate.

Table 15-1: Certej Mineral Reserves Estimate

	Tonnage (t x 000)	Au (g/t)	Au (oz x 000)	Ag (g/t)	Ag (oz x 000)
Proven	*20,441*	*1.91*	*1,255*	*10*	*6,283*
Probable	*26,543*	*1.41*	*1,203*	*12*	*9,967*
Proven+Probable	*46,984*	*1.63*	*2,458*	*11*	*16,250*

The Mineral Reserves as reported are derived from, and are included within, the Mineral Resources.

No dilution and no ore loss was included in the conversion of Mineral Resources to Mineral Reserves. The block model methodology already accounts for both factors.

The cut-off grade for the Mineral Reserves is 0.90 g/t Au equivalent.

The Mineral Reserves are effective December 31st, 2013.

15.2 PIT OPTIMIZATION PARAMETERS

For the purpose of the Lerchs-Grossman optimisation, the following parameters were used:-

Table 15-2: Pit Optimisation NSR Factors

Metallurgical Recovery		
Gold Recovery	%	87.4%
Silver Recovery	%	80.0%
Metal Prices/Costs		
Gold Price	US$/oz	1,250.00
Silver Price	US$/oz	16.50
Gold Transport & Refining	US$/oz	3.71
Silver Transport and Refining	US$/oz	2.05
Royalty	% of Revenue	4.0%
Material Related Costs		
General & Administration	US$/t (ore)	4.38
Processing	US$/t (ore)	(Note 1)
Sustaining Capital	US$/t (ore)	5.03
Base Mining Cost	US$/t mined	1.61
Incremental Mining Cost	$/t mined / 5m bench (below 515m RL)	0.021

Note 1: Processing costs in US dollars per tonne of ore = 1.50 + 9.77 + (40 x US$/kW.h) + (13.3 x 90% x S% x US$/kW.h) where electricity is US$0.085/kW.h. This therefore simplifies to 14.67 + 1.017 x S%.

No additional dilution was considered for the pit optimisation as the resource block model already included sufficient contact dilution for reserves. Likewise, the resource block model also accounts for expected mining ore loss, so no ore loss was included in the pit optimisation.

The processing cost is sensitive to sulphide sulphur grade, which in almost all cases is nearly equal to the total sulphur grade. To ensure that any high cost blocks are accounted for fully an NSR block model was calculated. It accounts for the variable process cost and revenues for both gold and silver products. The inputs for the NSR calculation are detailed in Table 15-2. The Lerchs-Grossman optimisation used the block NSR values. However, to limit the pit from over expanding into low-grade areas, the optimisation model was restricted by first zeroing the value of blocks with an Au_Equiv grade of less than 1.20 g/t.

If left unconstrained, the optimised pit shell would cross over the Macrisului valley stream. For the purpose of the pre-feasibility study, a constraint was placed in the model to prevent the optimised pit shell in the final run from crossing the stream. The reserve pit was also designed to not interfere with the natural watercourse of the Macrisului valley stream.

The Lerchs-Grossman algorithm used the pit slope recommendations discussed in Section 16.2 of this report. The parameters were integrated into the block model for the L-G run.

15.3 FINAL PIT DESIGN

A series of 40 optimised pit shells were generated for gold price simulations up to US$1,250/oz. Figure 15-1 shows the cumulative value for each shell in the optimisation run. As can be seen the relative change in cumulative value does not change rapidly above shell number 19 except for a minor jump going to shell 25, which was reflective of a rather large increase in pit size as shown in Figure 15-2. The effects of NPV discounting however become sensitive with the larger pit shells and as a result, shell 19 was found as the shell providing the best financial return and selected to be most suitable for further work.

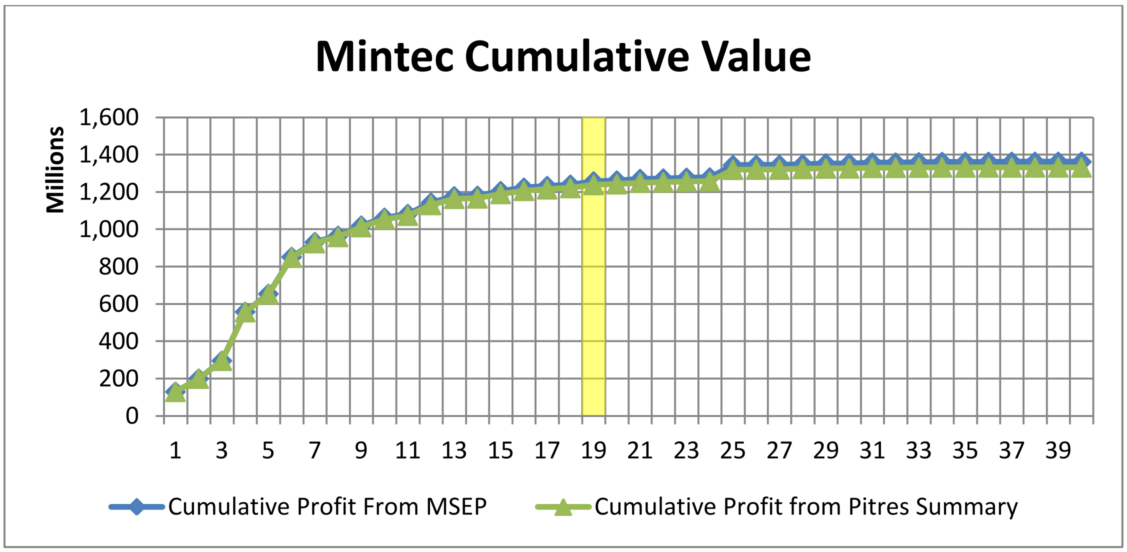


Figure 15-1: Cumulative Value of L-G shells

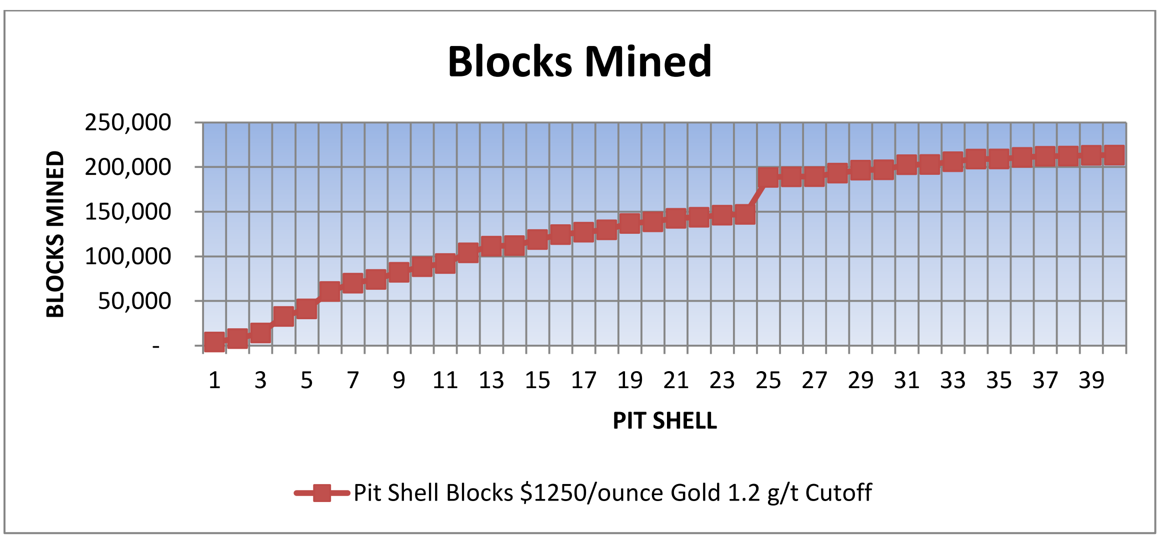


Figure 15-2: Blocks Mined in L-G shells

Shell 19 formed the basis to create the detailed final pit design that followed the detailed recommended geotechnical parameters inclusive of haulage ramps. Figure 15-3 shows the final pit design highlighting the three pit areas, East Final Pit, West Pit and West Satellite Pit. Figure 15-4 shows the same pit but with Au block grades for measured and indicated blocks on the 400m bench. The pink line indicates the bench outline at 400m elevation for the East and West Pit. The blue dashed line (A-A') represents the section line used for Figure 15-5. The long Section displayed in Figure 15-5 shows Au block grades for measured and indicated blocks along the section line A-A'.

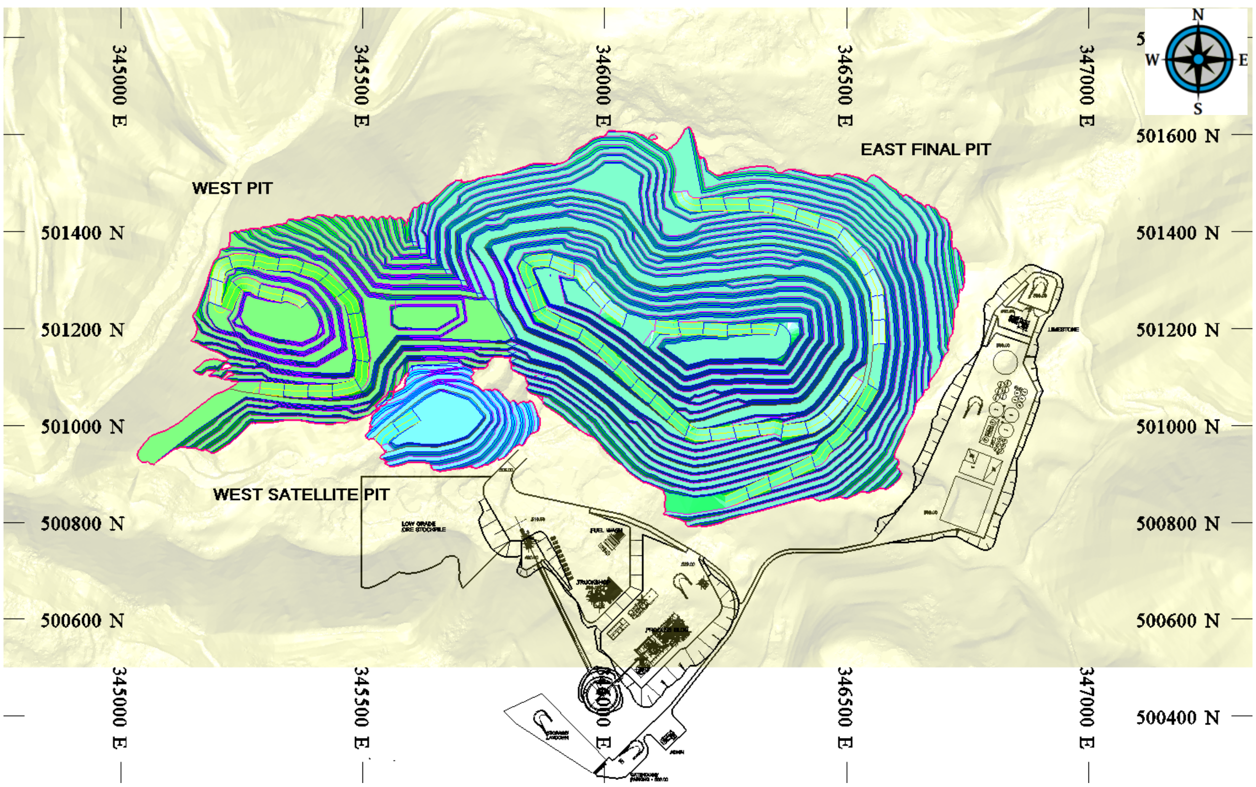


Figure 15-3: General Arrangement with Final Pit Design

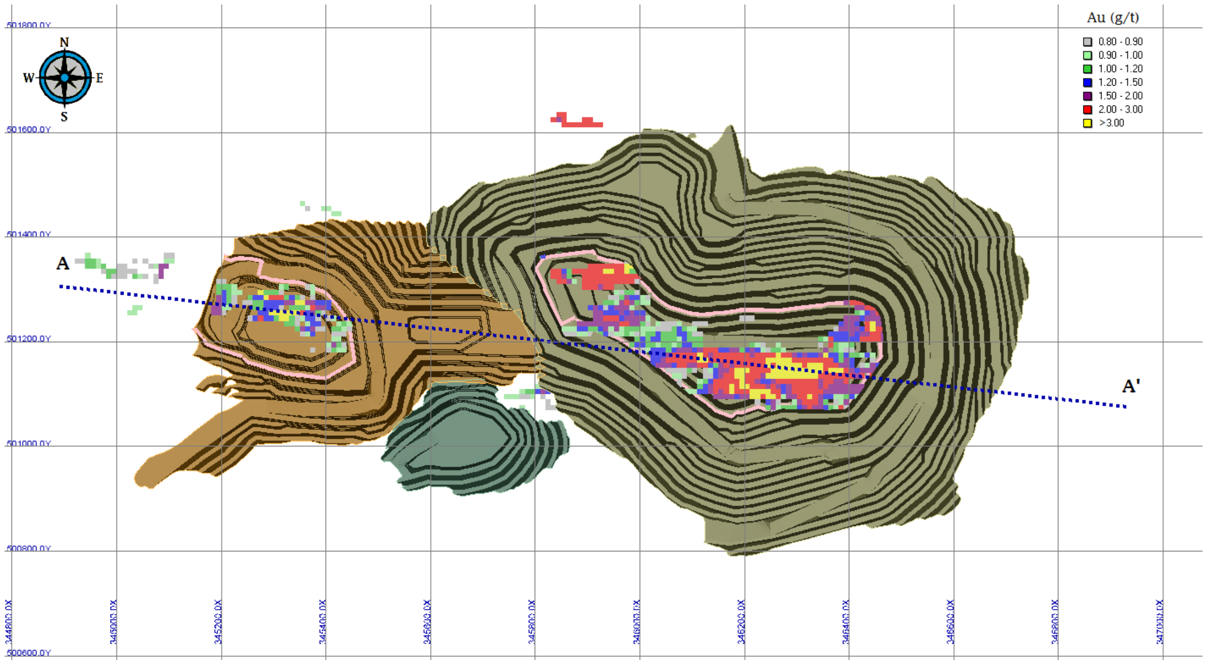


Figure 15-4: Plan view of Final Pit Design, M&I Block Grades for Au (g/t) on 400m Bench

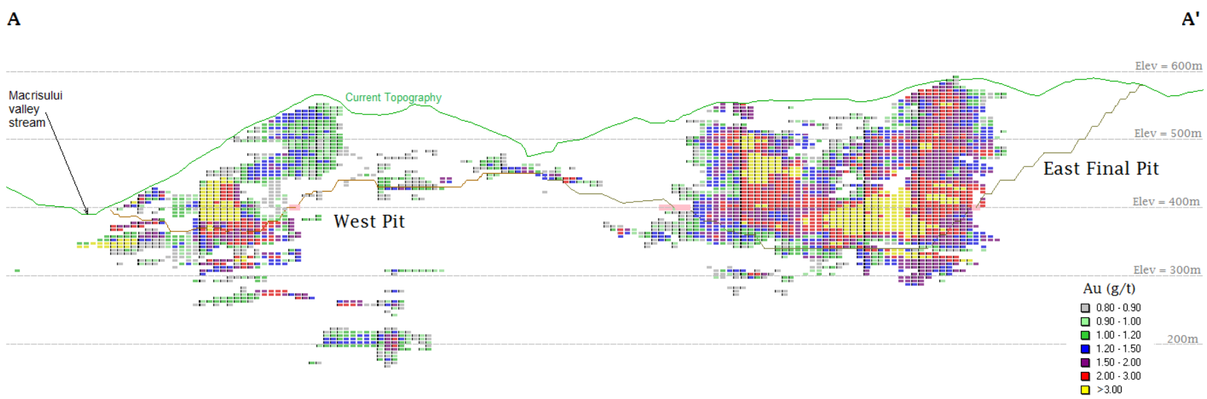


Figure 15-5: Long Section of Final Pit Design Showing M&I Block Grades for Au (g/t)

The final pit design incorporates the bench heights, bench face angles, bench stack heights, berm width, geotechnical berm width and inter-ramp angle as outlined in November 2013 recommendations from Golder Associates as discussed in Section 16.2 of this report. The reserve pit extends from a top elevation of 630m in the most eastern extent to a pit bottom at an elevation of 340m in the East Pit, 365m in the West Pit and 450m in the West Satellite Pit. The pit outline expands 1,600m east to west and 800m north to south at its widest profile.

Each pit has a single haul road access for ore and in the final configuration, only the East Final Pit has a permanent haul road exiting to the waste dumps north of the pit. All in-pit ramps have a maximum gradient of 10% and are designed at a width of 25m with the exception of the bottom bench in the West Pit and the bottom two benches in the East Final Pit, which are designed at 20m wide. The expected trucks are 7.0m wide from mirror to mirror so a 25m wide road should be sufficient for uninterrupted two-way traffic.

15.4 CUT-OFF GRADE

An Au_Equiv model was built for selecting cut-off grades. The calculation for Au_Equiv is:

$$\frac{\text{(Gold Grade * Gold Price * Gold Recovery + Silver Grade * Silver Price * Silver Recovery)}}{\text{(Gold Price * Gold Recovery)}}$$

This works out to be:

(Au_Grade * (1250.00 - 3.71) * 87.4% + Ag_Grade * (16.50 - 2.05) * 80.0%) / (1250 * 87.4%).

The calculation for break-even cut-off grade for all-in sustaining costs when including the differential of waste mining versus ore mining costs and considering low-grade stockpiling is:

$$\frac{\text{(Processing + G\&A + Sustaining + (MCostOre – McostWst) + Rehandle)}}{\text{((Gold Price – Royalty - Transport \& Refining) x Recovery)}}$$

Which on a cost per tonne ore and $ per gram basis is:

(19.54 + 2.79 + 5.13 + (1.78 – 1.94) + 1.00) / ((40.19 – 1.61 – 0.31) * 87.4%)

which works out to be about 0.85 g/t Au_Equiv. The waste mining costs are generally higher than the ore mining costs due to the height of the waste dumps, which increase over time. This does therefore vary with time but on average, the waste to ore cost differential works out to be about 16 cents per tonne. Project value optimisation using a series of higher cut-off grade options identified potential to reduce project risk and improve project NPV / IRR. This resulted in the selection of a slightly higher cut-off grade of 0.90 g/t Au_Equiv for determining the ultimate mill feed and reserve reporting cut-off grade. Using a cut-off grade higher than the break-even cut-off grade ensures a minimum profit for all mill feed.

With a 0.90 g/t Au_Equiv cut-off the reserve pit has a total Proven and Probable Mineral Reserves estimate of 46.984 Mt at a grade of 1.63 g/t Au and 11 g/t Ag. The reserve pit also has approximately 113.4 Mt of waste and an overall stripping ratio of 2.41:1 (waste tonnes to ore tonnes). Figure 15-6 shows the grade-tonnage relationship for material within the final pit design at various Au_Equiv cut-off options.

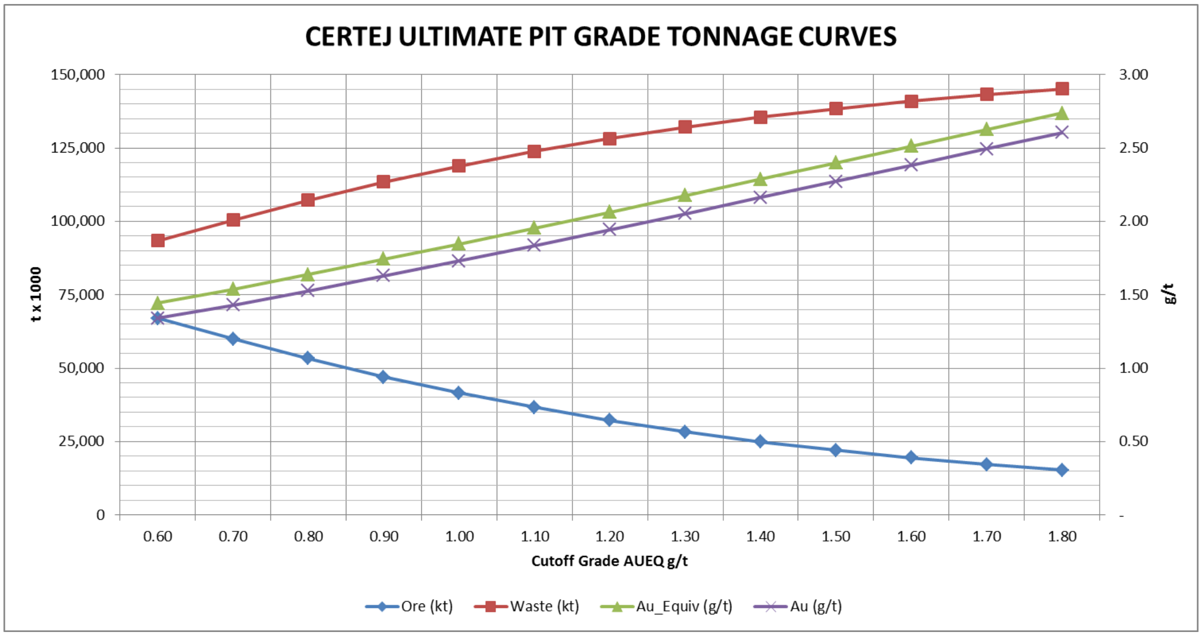


Figure 15-6: Grade-Tonnage Curves within Final Pit Design

The operating plan has an elevated mill feed cut-off grade during the mining phase followed by treating stockpiled lower grade ore later on. The upfront revenue of this more than offset the higher upfront capital and operating costs. During the mining phase a mill feed cut-off of 1.20 g/t Au_Equiv will be used with proven and probable ore between 0.90 and 1.20 g/t Au_Equiv being stockpiled for later treatment.

15.5 FACTORS AFFECTING THE MINERAL RESERVE ESTIMATE

The following factors may affect the Mineral Reserve estimates if base estimates or assumptions used in this study are not realised at the time of implementation:

- Gold and silver price;
- Exchange rates;
- Capital and operating cost estimates for mining, process and general;
- Metallurgical recoveries of gold and silver (test work on-going);
- Permit addendums;
- Process throughput and mine production estimates;
- Ongoing open pit geotechnical and hydrogeological work; and,
- The resource model.

Reasonably conservative estimations and contingencies were used in this pre-feasibility study; therefore, adverse variations with most of the above factors are considered to be somewhat buffered from materially impacting the reserve estimates.

While metallurgical testwork and process optimization are on going, preliminary results indicate improved recoveries (over design basis assumptions carried for the Pre-Feasibility).

Existing laws and procedures in Romania allow for environmental permit addendums. Eldorado Gold feels that the granting of permit addendums and the timing thereof are within reasonable expectations for this Project and as such, pose a very low likelihood of impacting the reserve materially.

The Mineral Reserve estimate is reliant upon the Mineral Resource block model. The accuracy of the block model will have a direct influence on the Project economics and the reserve. The modelling methods, QA/QC and high-grade cutting were reviewed and considered to be in keeping with best practices.

SECTION • 16 MINING METHODS

16.1 INTRODUCTION

All mining at Certej is planned to be by conventional open pit mining methods with drill and blast of the rock followed by load and haul with diesel powered off road equipment. The bulk of the mining will be hard rock excavation, which requires drill and blast. The overburden and existing dumps removal however will not require drill and blast. Planning is based on all mining operations being performed by owner equipment and personnel.

The mine plan is based on the reserve pit design and mineral reserves as outlined in Section 15. The open pit mine is scheduled to run for a total of 13 years with the first two years as pre-production. An elevated cut-off grade of 1.20 g/t Au_Equiv will be used while the open pit is active (HG Ore); with the lower grade ore (LG Ore) being stockpiled for later feed. Through direct feed from mining and by stockpile reclaiming, a constant mill feed of 3.0 million tonnes per annum will be maintained for nearly 15.7 years. The low-grade stockpiles will reach a maximum capacity of close to 15 Mt at the time at which the open pit production ceases. Refer to Section 16.4 for detailed schedules.

16.2 GEOTECH PARAMETERS

Golder Associates (UK) Limited has been involved with the Certej project for work incorporated in the European Goldfields Technical Report of February 2009 and further review in 2011. After the acquisition by Eldorado Gold, additional drilling and initial core, analysis was completed in 2013. This formed the basis for updated recommendations stated in the Technical Memorandum from Golder Associates to Eldorado Gold in November 2013. The Golder technical memorandum incorporated prior work and available data from 2013 drilling at the time of writing. It sufficiently covered the requirements needed for pre-feasibility level design. The 2013 drilling analysis is, however, ongoing. It will culminate with a final report in early 2014 for use in the planned feasibility study. Table 16-1 shows the amount of geotechnical drilling done on this property since 2005. The 2013 drilling was campaigned in two stages and those holes were also used for hydrogeology testing.

Table 16-1: Geotech and Hydrogeology Drilling for Open Pit

Period	Number of holes	Total Drilled (m)
2005	1	208
2006	7	1537
2010	4	975
2011	2	555
Stage 1, 2013[1]	11	3373
Stage 2, 2013[2]	11	2754
Total Hydrogeology	**14**	**3589**
Total Geotech	**29**	**7491**

1 All eleven were used for geotechnical purposes, three were also used for hydrogeology purposes.
2 Four of which were used for geotechnical purposes, all eleven being used for hydrogeology purposes.

The updated recommendations from Golder divided the Certej pit into four main design zones that encompass eleven sectors. Figure 16-1 shows design zones A to D and sectors 1 to 11. The red traces indicate holes drilled in Stage 1 2013, which were designed for geotech. The green traces represent Stage 2 2013 drilling, which was designed for hydrogeology. Some holes in each set were used for both geotech and hydrogeology. The 2013 hydrogeology work is still in progress at the time of writing and will be included in the final feasibility work.

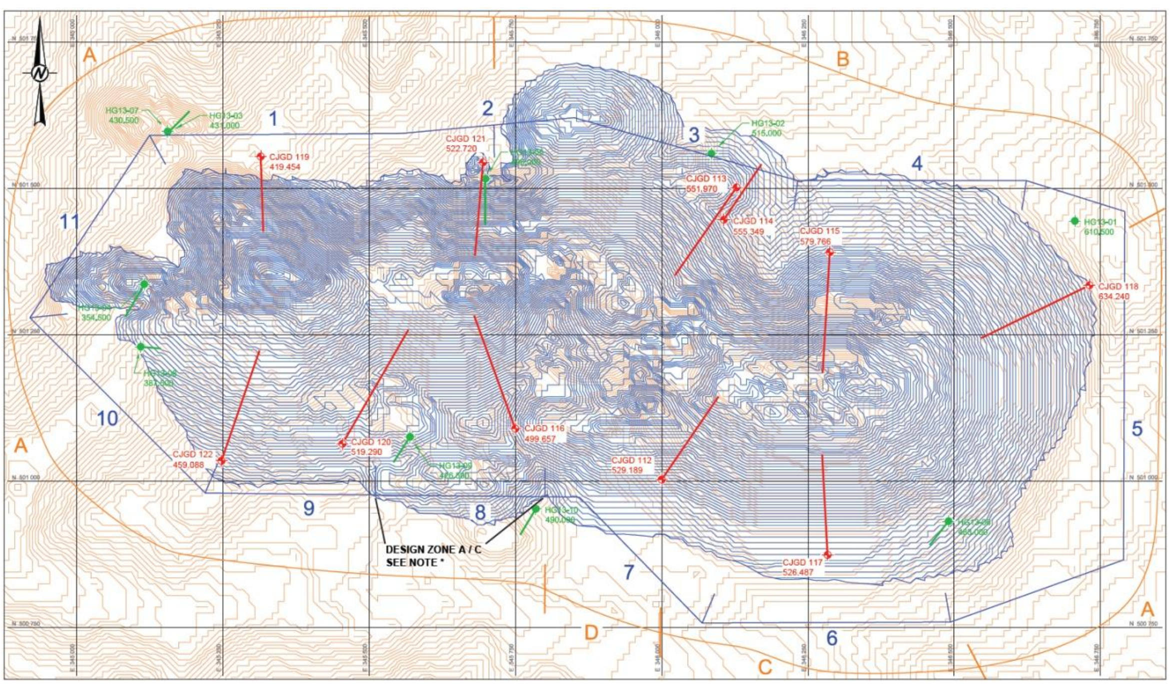


Figure 16-1: Geotechnical Design Zones and Sectors

For each design zone, a table of parameters was given depending on sector and rock type. The face angle varied from 45 degrees in the soil overburden to 70 degrees in the lower andesite. The inter-ramp slope angles ranged from a low of 29.1 degrees in the overburden to a high of 49.2 degrees in the lower andesite.

16.3 MINING PHASES

The pit development at Certej will follow progressive phases, which smooth out the production demand thereby limiting and deferring equipment purchases. The phase designs include the East Starter Pit, The East Expansion Pit, The West Satellite Pit and the West Pit. Figure 16-2 to Figure 16-5 show the various pit phases.

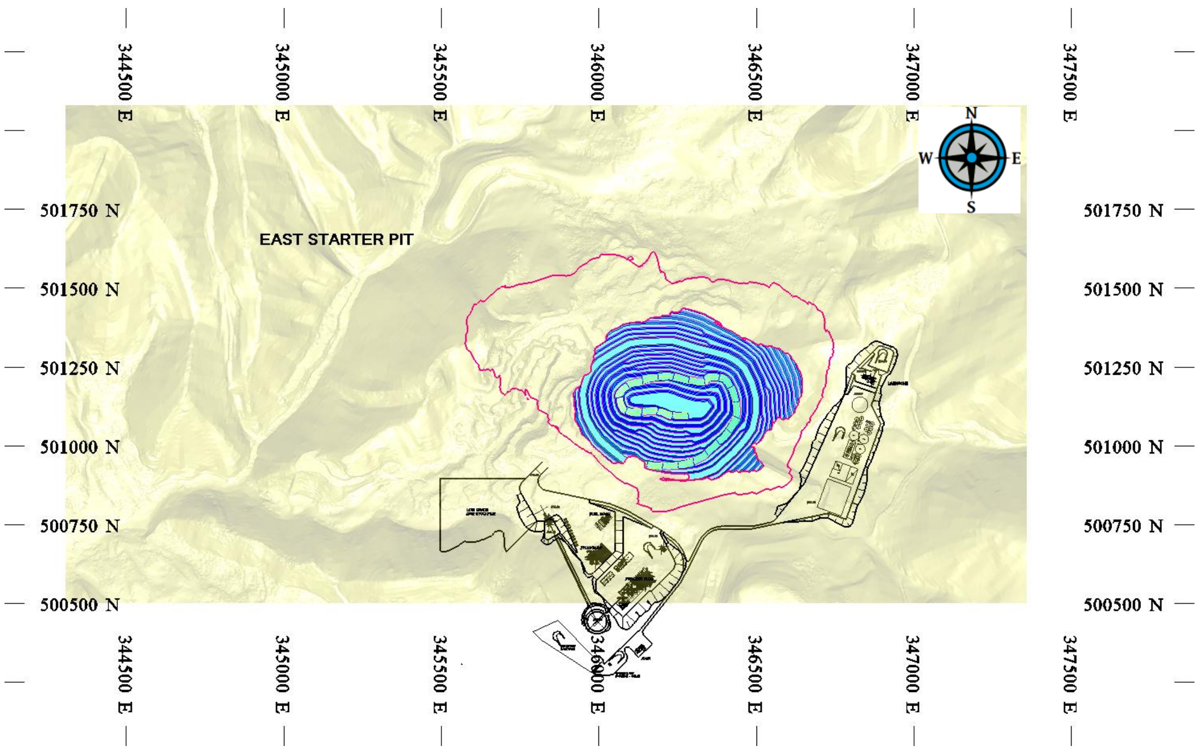


Figure 16-2: East Starter Pit

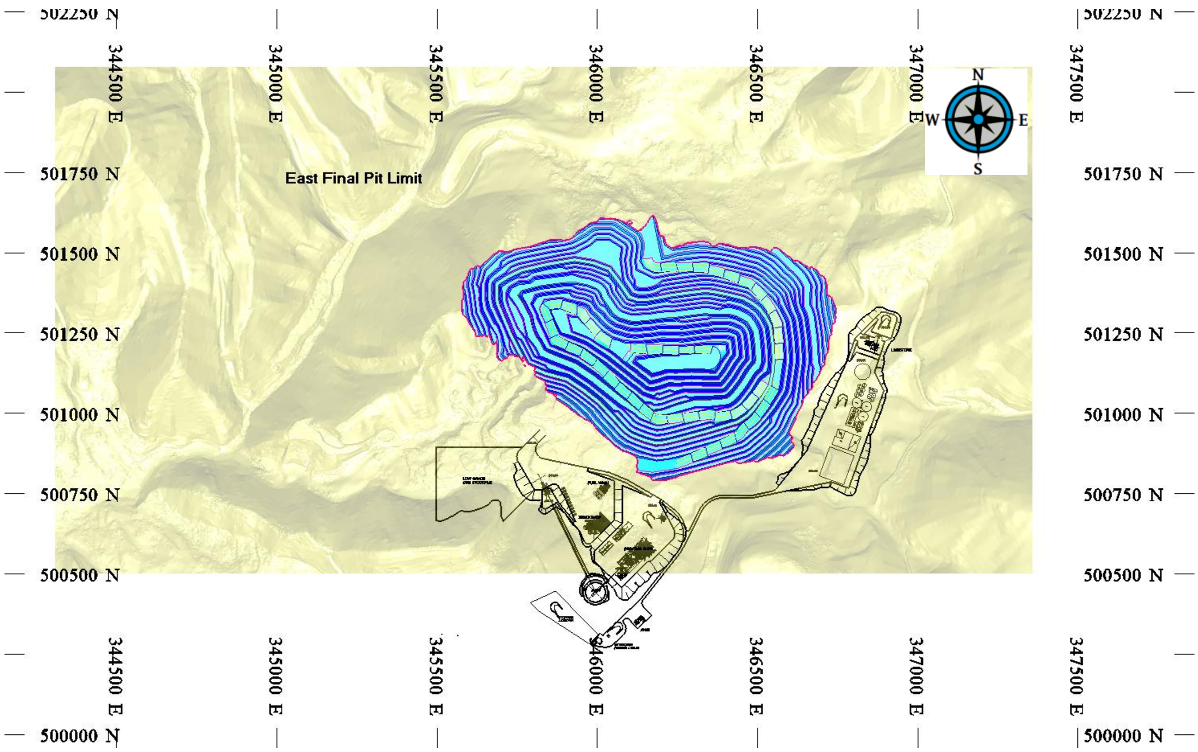


Figure 16-3: East Expansion (or East Final) Pit

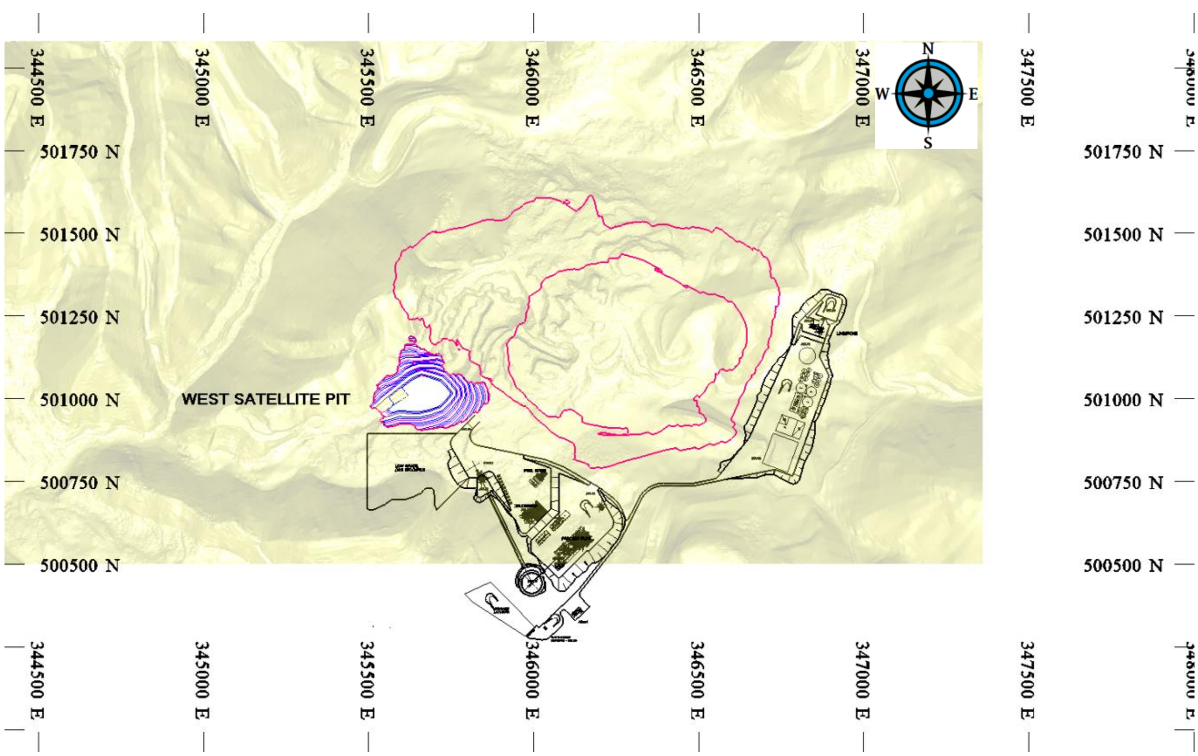


Figure 16-4: West Satellite Pit

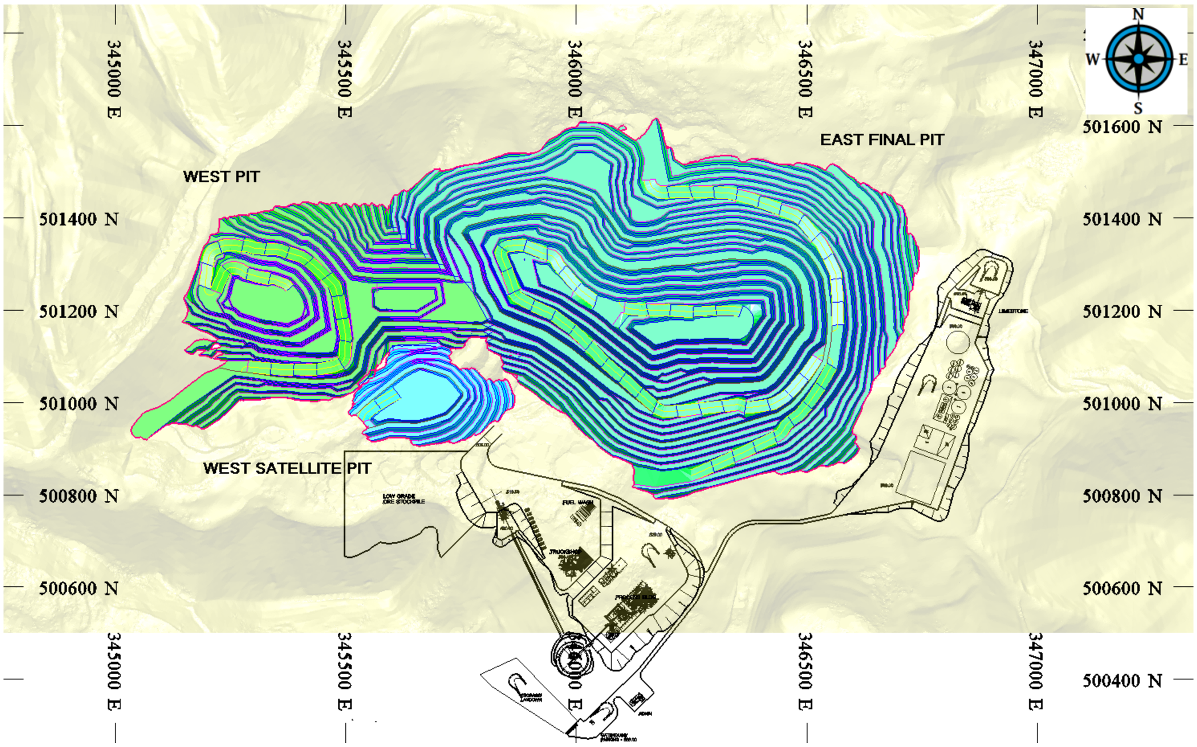


Figure 16-5: Final Pit

Each phase has the same geotechnical parameters as the final pit design and is designed with 5 m bench heights.

The mining schedule draws upon sequential benches of the four phased pits with the phases running concurrent at times. However if mined in isolation, each of the four pit phases would have the following makeup of ore and waste:

Table 16-2: East Starter Pit Ore and Waste

	t x 1,000	Au g/t	Ag g/t
LG Ore	3,151	0.99	5
HG Ore	11,012	2.07	8
Total Ore	14,163	1.83	7
Waste	26,375		
Total Rock	40,538	S.R. =	1.86 : 1

Table 16-3: East Expansion Pit Ore and Waste

	t x 1,000	Au g/t	Ag g/t
LG Ore	6,023	0.94	10
HG Ore	14,026	1.98	12
Total Ore	20,049	1.67	11
Waste	66,196		
Total Rock	80,245	S.R. =	3.30 : 1

Table 16-4: West Satellite Pit Ore and Waste

	t x 1,000	Au g/t	Ag g/t
LG Ore	126	0.89	12
HG Ore	203	1.90	20
Total Ore	329	1.51	17
Waste	2,242		
Total Rock	2,571	S.R. =	6.81: 1

Table 16-5: West Pit Ore and Waste

	t x 1,000	Au g/t	Ag g/t
LG Ore	5,466	0.92	11
HG Ore	6,978	1.67	16
Total Ore	12,444	1.34	14
Waste	18,632		
Total Rock	31,076	S.R. =	1.50 : 1

Table 16-6: Total Reserve Pit Ore and Waste

	t x 1,000	Au g/t	Ag g/t
LG Ore	14,766	0.94	9
HG Ore	32,219	1.94	11
Total Ore	46,984	1.63	11
Waste	113,444		
Total Rock	160,429	S.R. =	2.41 : 1

16.4 MINING SCHEDULE

The mine plan has two years of pre-production activities followed by eleven years of pit production. The pre-production period is designed to gain sufficient ore exposure and build a starter ore stockpile. Suitable waste mined in the pre-production period will also be selectively used for construction needs.

All periods up to and including Year 6 have been detailed to quarterly schedules and thereafter to yearly schedules.

During the pre-production period and the first five years of full pit production, mining will be exclusively in both the East Starter Pit and the East Expansion Pit, with the East Starter Pit being completed at the end of that period. Production in years six and seven will consist of continued mining in the East Expansion Pit whilst starting to mine in the West Satellite Pit and the West Pit, with the West Satellite Pit completing in the seventh year. Production in years eight to eleven will be in the East Expansion Pit and the West Pit, with the West Pit completing in the tenth year and the East Expansion Pit completing in the eleventh year.

The open pit mining will be done on 5 m high benches. The equipment used will be large enough that benches can be sequenced comfortably at nine-5 m benches per year in any working area. Where waste stripping allows 10 m high benching, a maximum of twelve-5 m benches can be scheduled per year. Table 16-7 shows the year-by-year progression of the mining by phase. The bench names reflect the bottom elevation of the bench in metres above mean sea level.

Table 16-7: Annual Bench Sequencing by Pit Phase

	Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11
East Starter Pit	Lowest Bench	570	545	500	475	455	435	405						
	# of Benches	12	5	9	5	4	4	6						
East Expansion Pit	Lowest Bench	585	550	545	525	505	480	455	420	415	395	380	360	340
	# of Benches	9	7	1	4	4	5	5	7	1	4	3	4	4
West Satellite Pit	Lowest Bench								465	450				
	# of Benches								12	3				
West Pit	Lowest Bench								535	485	455	410	365	
	# of Benches								8	10	6	9	9	

The planned ore and waste schedule demonstrates that the peak mine production rate can be held to no higher than 17 Mt per annum whilst maintaining a constant 3 Mt per annum of high-grade ore. Table 16-8 shows a detailed report of annual grades and quantities of mined ore and waste along with stockpile status and mill feed.

Table 16-8: Mine Production Schedule

		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
L.G. Ore	kt	84	399	1,195	837	875	1,165	1,602	1,710	2,005	1,420	1,578	1,280	616	-	-	-	-	-
	Au (g/t)	0.95	0.89	0.98	0.98	0.92	0.93	0.94	0.95	0.95	0.94	0.93	0.93	0.94	-	-	-	-	-
	Ag (g/t)	**6**	**13**	**6**	**6**	**12**	**10**	**10**	**9**	**9**	**9**	**10**	**11**	**10**	**-**	**-**	**-**	**-**	**-**
	S (%)	3.4	3.2	3.7	3.8	3.9	3.9	4.2	3.9	3.6	4.0	4.1	4.0	3.9	-	-	-	-	-
H.G. Ore	kt	69	627	2,335	3,016	2,985	2,857	3,378	3,200	3,181	3,307	3,192	2,567	1,503	-	-	-	-	-
	Au (g/t)	1.76	2.07	1.81	1.82	2.10	2.08	1.98	1.82	1.69	1.95	1.93	2.08	2.24	-	-	-	-	-
	Ag (g/t)	6	8	7	9	13	14	12	11	9	10	11	13	20	-	-	-	-	-
	S (%)	3.3	3.4	3.8	4.0	4.2	4.4	4.6	4.0	3.7	4.2	4.2	4.3	4.4	-	-	-	-	-
Waste	kt	4,848	12,974	13,470	13,146	13,139	11,470	10,513	10,583	7,814	5,273	3,729	4,174	2,310	-	-	-	-	-
Total Rock	kt	5,001	14,000	17,000	17,000	17,000	15,493	15,493	15,493	13,000	10,000	8,500	8,022	4,429	-	-	-	-	-
Mill Feed	kt	-	-	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,008	3,000	3,000	3,000	3,000	3,000	1,978
	Au (g/t)	-	-	1.84	1.80	2.09	2.08	1.97	1.82	1.69	1.95	1.93	2.05	1.89	1.00	1.00	0.96	0.85	0.85
	Ag (g/t)	-	-	7	9	12	14	12	11	9	10	11	13	15	9	9	9	9	9
	S (%)	-	-	3.7	4.0	4.2	4.4	4.6	4.0	3.7	4.2	4.2	4.3	4.2	3.9	3.9	3.9	3.8	3.8
Closing S/P	kt	153	1,179	1,708	2,562	3,423	4,445	6,425	8,335	10,521	12,248	14,019	14,858	13,977	10,978	7,978	4,978	1,978	-
	Au (g/t)	1.32	1.58	1.01	1.03	1.00	0.95	1.01	1.02	1.02	1.03	1.03	1.00	0.94	0.92	0.89	0.85	0.85	-
	Ag (g/t)	6	9	8	7	9	9	9	9	9	9	9	9	9	9	9	9	9	-
	S (%)	3.4	3.3	3.6	3.7	3.7	3.8	3.9	3.9	3.8	3.9	3.9	3.9	3.9	3.9	3.9	3.8	3.8	-

Figure 16-6 to Figure 16-8 graphically show the data from Table 16-8 for material movement, stockpile closing balance and mill feed tonnes and grade.

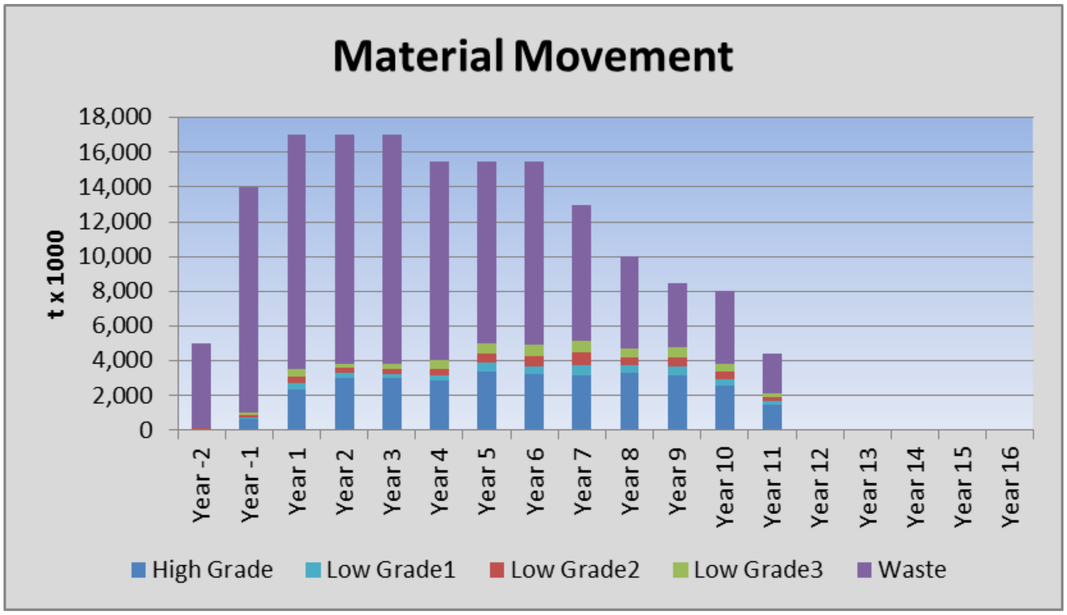


Figure 16-6: Material Movements

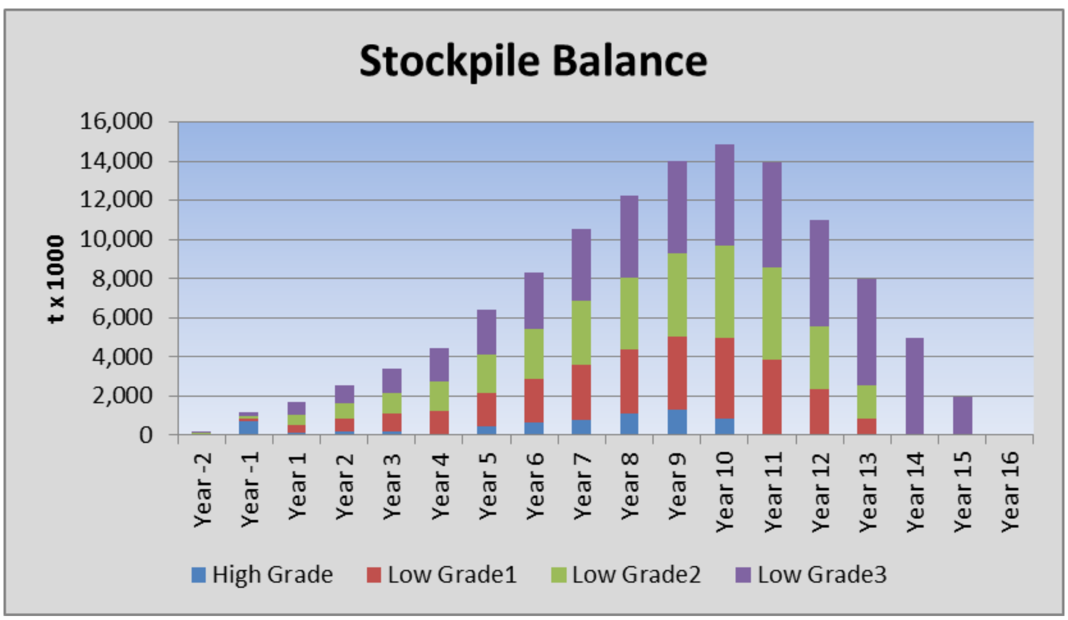


Figure 16-7: Stockpile Balance

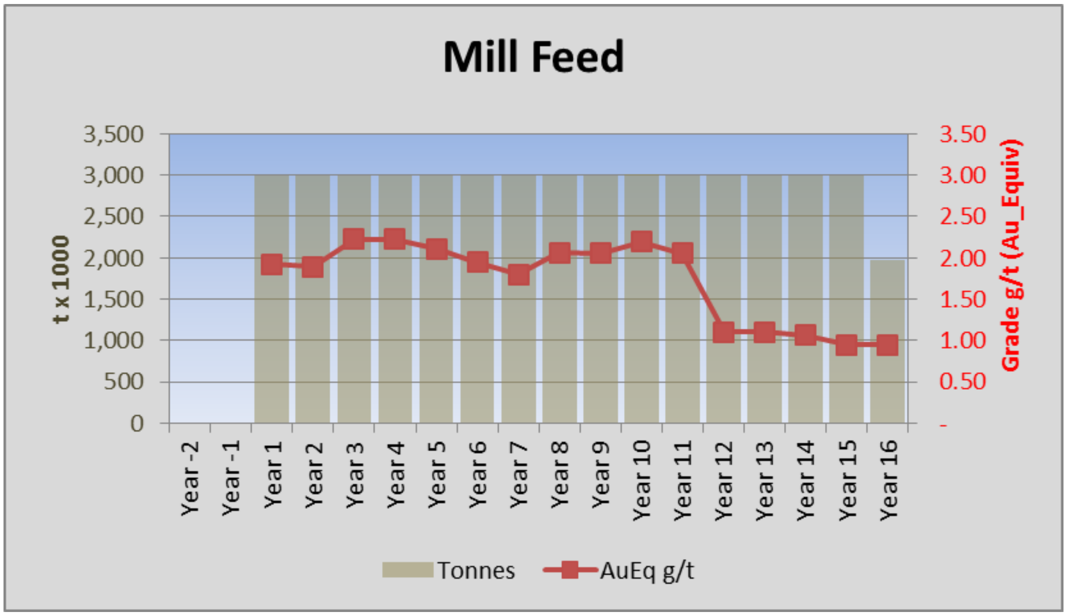


Figure 16-8: Mill Feed

16.5 WASTE ROCK AND LOW GRADE ORE MANAGEMENT

Waste rock from the open pit mining will be used in pre-production construction fill and the TMF embankment. A small amount of waste will also be used for preparing the area for the LG Stockpile 2 adjacent to the Waste Dump. All surplus waste rock will be placed into Waste Dumps north of the open pit as depicted in Figure 16-9.

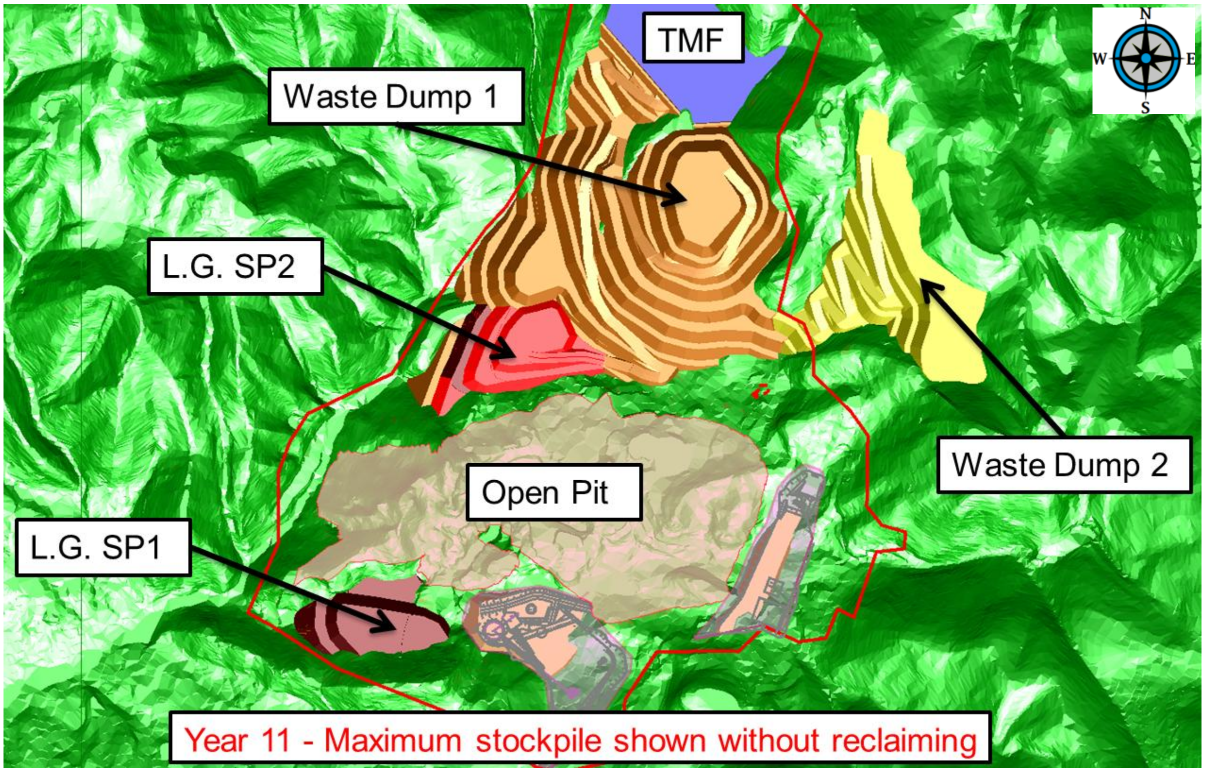


Figure 16-9: Locations for Waste Dumps and Low Grade Stockpiles

Waste Dump 1 will be filled before initiating Waste Dump 2. Ongoing investigation into backfilling one or more phases of the pit with waste rock may reduce the ultimate waste rock footprint and haulage costs.

The low-grade ore (between 0.90 and 1.20 g/t Au_Equiv) will be placed in either LG SP1 or LG SP2 during years -2 to 11. These two stockpiles will be reclaimed for mill feed in years 12 to 16. Figure 16-9 shows the location of these stockpiles with respect to the pit and waste dumps.

The low-grade stockpiles and waste dumps are designed to have 30 m high stacks; each stack is with a batter angle of 37 degrees, and with a 20.1 m wide berm separating successive stacks. The ramps and horizontal roads on the dumps are 35 m wide to accommodate the mining trucks. Overall slope angles are no steeper than 2H:1V and are reduced to 2.5H:1V or flatter in the majority of areas facing southward and westward.

16.6 MINING EQUIPMENT SELECTION

Mining at Certej will be undertaken using a conventional truck and shovel fleet. The number of units required of the various equipment has been calculated by the production schedule, equipment capacities, mechanical availability, utilization, scheduled hours and cycle times, which include detailed haulage profiles for various materials, benches, phases and periods.

The hauling fleet will consist of Belaz 75-135 trucks, which have a capacity of 120 tonnes. At peak demand, eleven of these trucks will be required. Caterpillar 6030 hydraulic face shovels with 16.5 m3 buckets will load the trucks. Two units would be required for all production years and only one required in the two pre-production years. A 12 m3 Caterpillar 993K front end loader will also be required.

Atlas Copco PV-235-D drill rigs, equipped with 165 mm bits, will handle all blasthole drilling. At peak demand, four rigs would be required. For wall control, drilling a single Atlas Copco ROC L6 will be required. A drill rig suited for RC drilling is also required from Year -1.

Other ancillary equipment has been selected to best match the main production equipment. Table 16-9 summarises the annual requirement of all mining equipment.

Table 16-9: Mining Equipment Requirements

	MAKE	MODEL	SIZE	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Drilling																					
Blasthole Drill	Atlas Copco	PV-235-D	165 mm	1	2	3	4	4	4	4	4	3	3	2	2	1					
Wall Control Drill	Atlas Copco	ROC L6	110 mm		1	1	1	1	1	1	1	1	1	1	1	1					
Grade Control RC Drill					1	1	1	1	1	1	1	1	1	1	1	1					
Loading																					
Hydraulic Shovel	Caterpillar	6030 FS	16.5 m3	1	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Wheel Loader	Caterpillar	993K	12 m3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hauling																					
Haul Truck	Belaz	75-135	120 tonne	4	6	6	8	9	10	11	11	8	7	6	6	4	4	4	4	4	4
Roads & Dumps																					
Track Dozer	Caterpillar	D9T	306 kW	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Track Dozer	Caterpillar	D10T	433 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer	Caterpillar	834H	372 kW		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor Grader	Caterpillar	16M	221 kW	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Water Truck	M-Benz	4140B	20,000 l	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Sand Truck	M-Benz	4140B		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator	Caterpillar	330DL	200 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

	MAKE	MODEL	SIZE	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Front End Loader	Caterpillar	980H	6 m3			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Rock Breaker	Caterpillar	330	200 kW			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Support Equipment																					
Low Bed Transporter	M-Benz	3354	60 tonne	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck Crane/Telehandler	TF/Caterpillar	ATS65/514	65t/5t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Handler/Forklift	IMT	TH3565	25t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel/Lube Truck	M-Benz	3340	20,000L	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
HD Mechanic's Field Truck	M-Benz	3340		1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mechanic's Service Truck	M-Benz	3340		1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Welding Truck	Manufacturer	3340		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Pickup	Manufacturer	4x4	3/4 tonne	10	12	12	12	12	12	12	12	12	12	12	12	12	6	6	6	6	6
Light Plant	Manufacturer			2	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2
Crushing Plant				1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Blasting																					
Blasting Crew Flatbed Truck	Manufacturer			1	1	1	1	1	1	1	1	1	1	1	1	1					
ANFO/Emulsion Truck	M-Benz	3340K		1	1	1	1	1	1	1	1	1	1	1	1	1					
Blasters Crew Truck	Manufacturer	4x4		1	1	1	1	1	1	1	1	1	1	1	1	1					
Blasthole Stemmer	M-Benz			1	1	1	1	1	1	1	1	1	1	1	1	1					

Table 16-10 shows the schedule for new purchases of mining equipment. Cells highlighted in yellow reflect purchases that are for replacement of equipment that will have reached the recommended maximum hours of use.

Table 16-10: Mining Equipment Purchase Schedule

	MAKE	MODEL	SIZE	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	TOTAL
Drilling																						
Blasthole Drill	Atlas Copco	PV-235-D	165 mm	1	1	1	1															6
Wall Control Drill	Atlas Copco	ROC L6	110 mm		1																	1
Grade Control RC Drill					1																	1
Loading																						
Hydraulic Shovel	Caterpillar	6030 FS	16.5 m3	1		1																2
Wheel Loader	Caterpillar	993K	12 m3	1																		2
Hauling																						
Haul Truck	Belaz	75-135	120 tonne	4	2		2	1	1	1												11
Roads & Dumps																						
Track Dozer	Caterpillar	D9T	306 kW	1	1								1	1								4
Track Dozer	Caterpillar	D10T	433 kW	1										1								2
Wheel Dozer	Caterpillar	834H	372 kW		1										1							2
Motor Grader	Caterpillar	16M	221 kW	1	1								1	1								4
Water Truck	M-Benz	4140B	20,000 l	1									1									2
Sand Truck	M-Benz	4140B	-	1											1							2
Excavator	Caterpillar	330DL	200 kW	1							1											2
Front End Loader	Caterpillar	980H	6 m3			1																1

	MAKE	MODEL	SIZE	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	TOTAL
Rock Breaker	Caterpillar	330	200 kW			1																1
Support Equipment																						
Low Bed Transporter	M-Benz	3354	60 tonne	1									1									2
Truck Crane/Telehandler	TF/Caterpillar	ATS65/514	65t/5t	1									1									2
Tire Handler/Forklift	IMT	TH3565	25t	1									1									2
Fuel/Lube Truck	M-Benz	3340	20,000L	1									1									2
HD Mechanic's Field Truck	M-Benz	3340		1	1							1	1									4
Mechanic's Service Truck	M-Benz	3340		1	1							1	1									4
Welding Truck	Manufacturer	3340		1						1												2
Service Pickup	Manufacturer	4x4	3/4 tonne	10	2		10	2			10	2			4							40
Light Plant	Manufacturer			2	2		2	2			2	2			2							14
Crushing Plant				1																		1
Blasting																						
Blasting Crew Flatbed Truck	Manufacturer			1						1												2
ANFO/Emulsion Truck	M-Benz	3340K		1								1										2
Blasters Crew Truck	Manufacturer	4x4		1					1													2
Blasthole Stemmer	M-Benz			1					1													2

Haul truck requirements are based on scheduled availabilities, utilization, working times, calculated haul profiles, fixed cycle time elements and performance data for loaded and empty Belaz 75135 trucks. A rolling resistance of 3% was assumed for all cycles. A speed cap of 38 km/h for empty and 36 km/h for loaded hauls is applied. The haul road profiles are calculated for every 5 m bench within each of the four pit phases for each destination (crusher/ROM pad, waste dump, LG Stockpile 1 and LG Stockpile 2). In the case of the low-grade stockpiles and the waste dump, the shapes were sliced into horizontal slabs to simulate construction to determine expected distances reflective of when material is to be placed there. Figure 16-10 is a compilation of calculated cycle times for the various pit phases, benches and destinations. Time indicated is the complete cycle time inclusive of loading, hauling full, dumping, return empty and waiting time.

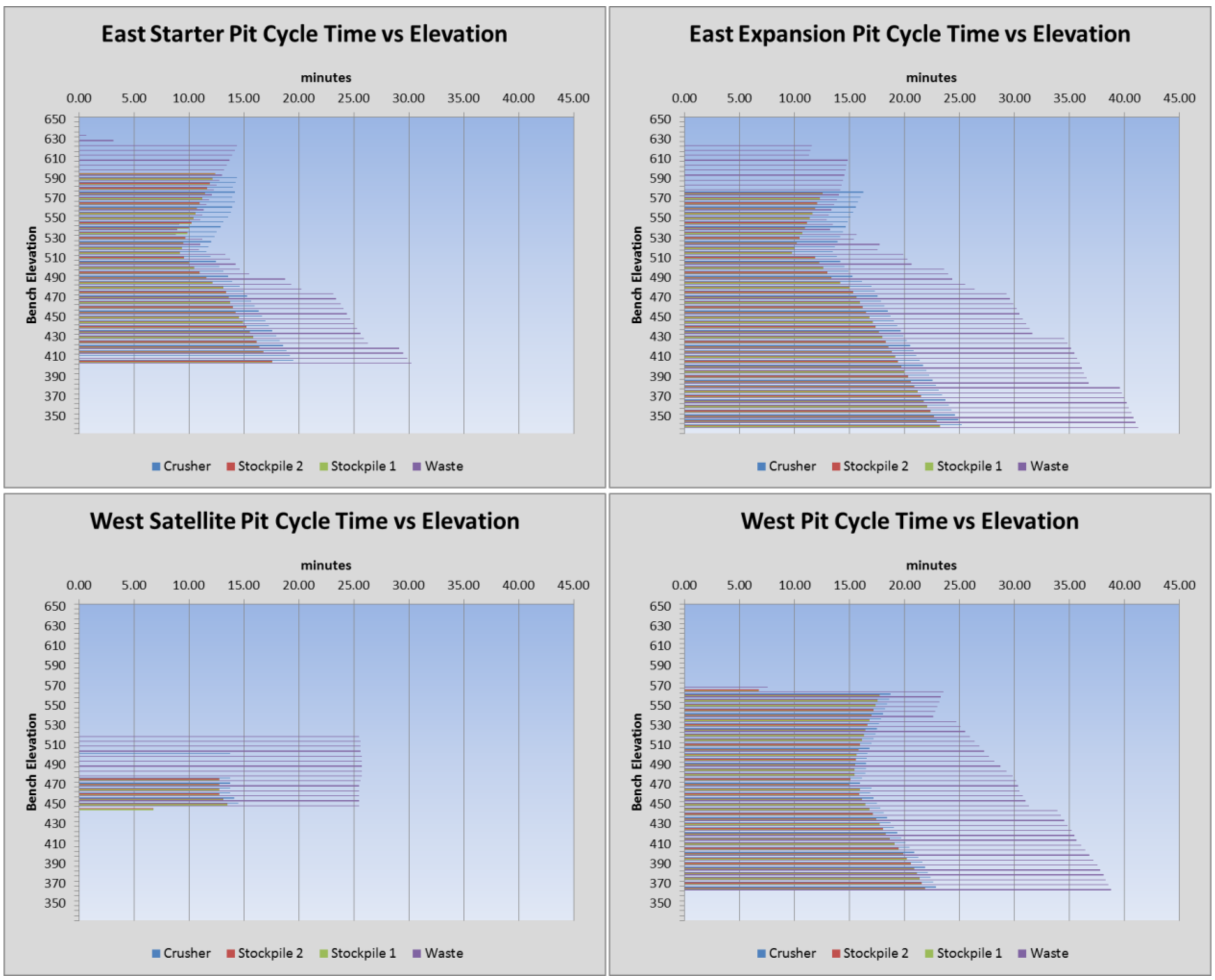


Figure 16-10: Haul Cycle Times

With multiple phase and benches being mined in each period, the weighted average cycle times were calculated by period for the various haul destinations. Figure 16-11 shows graphically the effect this has over time for all open pit mining.

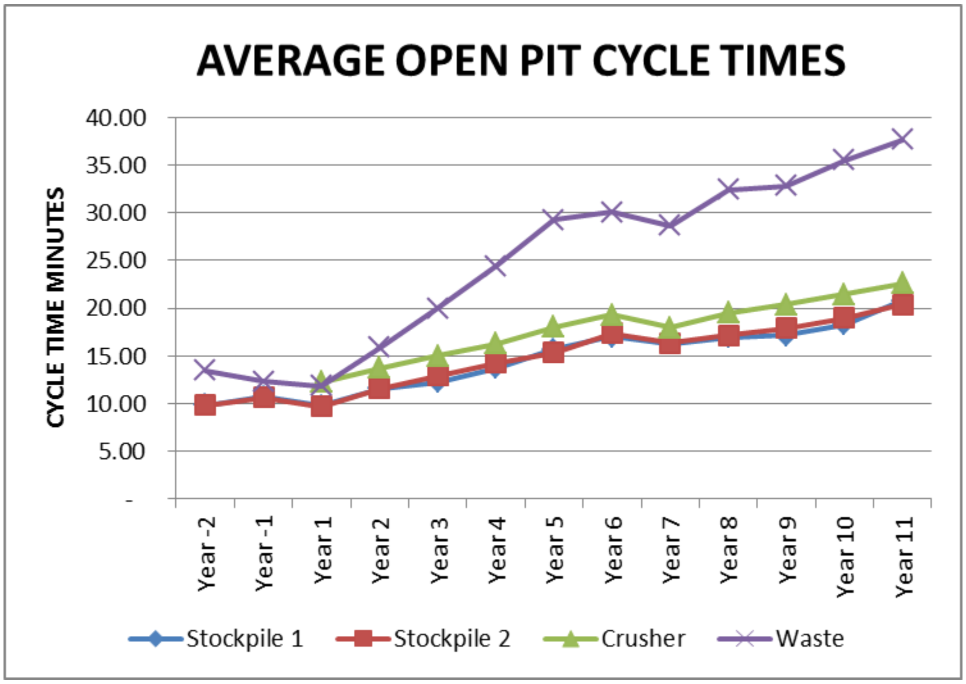


Figure 16-11: Average Open Pit Cycle Times

This data along with the production schedule was used to calculate equipment hours required for the planned haulage. The total scheduled hours required was then used for calculating the number of trucks required in each period and the operating hours accumulation were tracked for replacement timing. The total running hours for all open pit and rehandling haulage comes to 565,000 hours, which for 11 trucks averages under 51,500 hours per truck distributed as shown in Figure 16-12. The workload was scheduled such that no haul truck replacement was required.

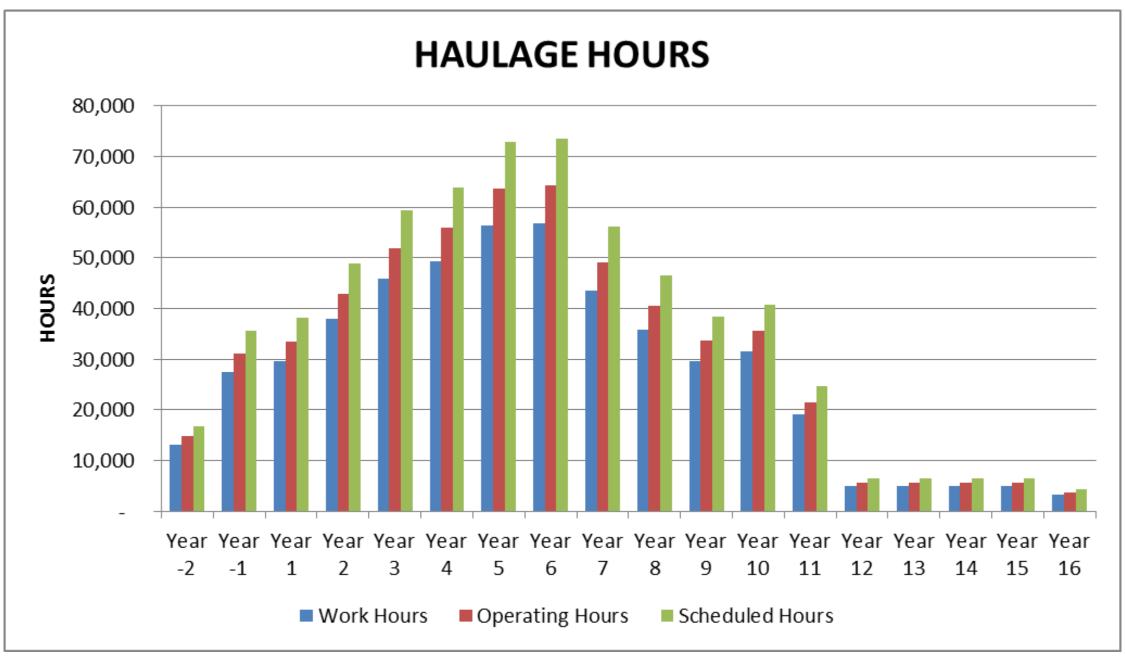


Figure 16-12: Haulage Hour Requirements by Year

The Caterpillar 6030FS with a 16.5 m3 bucket capacity can load the Belaz 75135 trucks with five passes in an average time of 3.67 minutes. The Caterpillar 993K with a 12.0 m3 bucket capacity can load the Belaz 75135 trucks with six passes in an average time of 4.75 minutes. For the production planned, the highest demand years require a calculated 1.4 shovels and 0.2 of a front-end loader. Provision is made for having two Caterpillar 6030FS for all mine production years, with only one Caterpillar 6030FS during the two pre-production years and the five rehandling only years. Additionally, a Caterpillar 993K front-end loader is available for all years, primarily as a backup loader. At the end of the open pit and stockpile reclaiming life, the two shovels will each have less than 50,000 hours and much less than that for the front end loader. No replacement units are planned for shovels and loaders.

Drill and blast will be done on 10 m bench heights with load and haul at 5m benches where ore is expected. Production holes will be 165 mm in diameter and have 0.8 m of subdrill. The drill patterns will be 4.4 m by 3.8 m in ore and 5.1 m by 4.4 m in waste, which will give a powder factor of 0.30 kg/t for ore and 0.22 kg/t for waste. ANFO is used as the main explosive agent. The drill penetration rate in ore is expected to be 32 m per hour and 27 m per hour in waste. A peak requirement of four drill rigs will be required to handle all production drilling in any one year. Over the thirteen years of drill and blast requirement a total of just over 200,000 hours of drilling is needed, which can be managed by the initial four drill rigs without any drill rig replacements. Wall control drilling and RC drilling for grade control will be handled with specialised drill rigs.

16.7 MINING OPERATING COSTS

The basis of this study is that the mining equipment is owned by the mine, operated by the mine and maintained by the mine. Information from suppliers, local data and similar productivities / consumption from other Eldorado operations were used to calculate operating costs from first principles.

The haul profiles and haul cycle times were used to determine fuel burn rates. For all other equipment, equipment hours were accumulated based on production plans and equipment productivities. Lube, fuel, labour, tires, wear parts, and maintenance inclusive of major overhauls were calculated for all equipment.

Labour and staffing for mine operations at maximum production demand includes 25 technical/skilled and 12 unskilled in mine management / supervision, 81 skilled and 27 unskilled as equipment operators, and 27 skilled and 46 unskilled in maintenance for a maximum of 221 mine department employees in Year 5. These numbers include provision for holidays and sick leave.

SECTION • 17 RECOVERY METHODS

17.1 INTRODUCTION

The Certej process plant is designed to produce gold-silver doré from a disseminated and breccia hosted gold-silver mineralisation at a treatment rate of 3 million tonnes of ore per annum. The mill feed grade over the life of mine is expected to be 1.63 g/t Au and 11 g/t Ag.

Ore mined from the Certej deposit is comprised of refractory gold- and silver-bearing pyrite mineralization, hosted in an andesite (silicate) lattice; there is little gravity recoverable or free milling gold in the ore. Given the refractory nature of the gold, recovery of the precious metals by direct cyanide leaching is ineffective. The Certej ore, therefore, first requires an oxidative pre-treatment step to liberate gold and silver by destroying the pyrite mineralization, such that gold and silver can be recovered by conventional cyanide leaching. A number of oxidative pre-treatment processes for treatment of refractory gold ores and concentrates has been developed and in some cases commercialized. For Certej ore, the pressure oxidation process has been demonstrated to be highly effective.

Gold and silver will be recovered from the Certej ore by a series of process steps including crushing, grinding, flotation, pressure oxidation of the resultant flotation concentrate, washing of the pressure oxidation product solids, and cyanide leaching of the washed, pressure oxidation product solids. The ore will first be crushed and ground to a target particle size and then subjected to flotation to produce a gold and silver-bearing sulphide concentrate. The flotation concentrate will be subsequently oxidized by the pressure oxidation process where sulphidic minerals (such as pyrite) are oxidized by reaction with oxygen under elevated temperature and pressure. The pressure oxidation product solids will then be washed to remove acid and dissolved metals. The washed, pressure oxidation product solids will be subjected to lime boil treatment to improve silver recovery. Following washing and lime boil treatment, the pressure oxidation product solids will be leached with cyanide in the presence of activated carbon (CIL). Gold and silver loaded on the activated carbon will be stripped off and recovered by electrowinning and smelting to produce a gold/silver doré. The cyanide leach tailings will be detoxified by cyanide destruction, thickened and then impounded in a tailing storage facility.

A summary of the process flowsheet is provided in Figure 17-1.

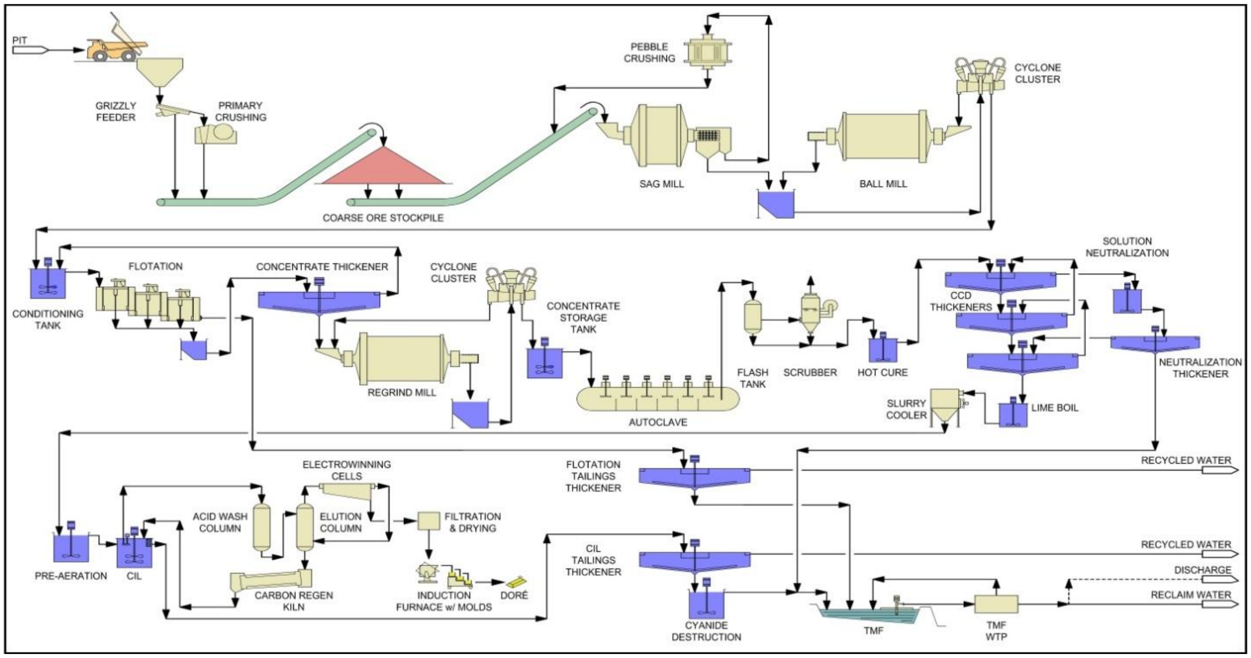


Figure 17-1: Simplified Process Flowsheet for Certej Process Plant

17.2 PREVIOUS RECOVERY METHODS

The previous definitive feasibility study entitled 'Certej Updated Definitive Feasibility Study Summary Technical Report' published by European Goldfields Limited in February 2009 was based on application of the proprietary Albion process for oxidation of the flotation concentrate prior to cyanide leaching for recovery of gold and silver. Subsequent evaluation of the Albion process for treatment of the Certej flotation concentrates in 2012 and 2013 necessitated a change in the process flowsheet in order to achieve consistently high gold recovery over the entire Certej orebody. As such, a pressure oxidation based-process was developed in 2013 and serves as the basis for the current evaluation.

17.3 PRESSURE OXIDATION PROCESS SELECTION

Following extensive testing of the Albion process in 2012 and 2013, it was concluded that large disparities in gold recovery and sulphur oxidation rate were indicative of significant technical risk in applying this technology to the Certej ore.

In June 2006, a desktop economic evaluation was undertaken to evaluate the viability of various process options for treatment of refractory gold ore. The principal inputs to the desktop study are summarized in Table 17-1.

Table 17-1: Inputs to Desktop Evaluation of the Optimum Process Treatment Route of Certej Flotation Concentrate

Oxidation Process	Reference	Metal Recovery		Testwork Source
		Au %	Ag %	
Pressure Oxidation	AKES Olympias study	94	7	AMMTEC report A8025
Bacterial Leaching	Goldfields database	83	53	Cepromin testwork 2005, SGS testwork 2003
Geocoat Process	Geobiotics cost study	75	75	Estimate
Albion Process	Core Resources ref paper	84	93	HRL testwork
Activox Process	Western Mining	83	53	BiOx results
Roasting On-site	Lurgi Roaster/ Acid plant	84	79	AMMTEC A8025 report
Roasting Off-site	Roast at Eti Bor and return	84	79	As above

After completion of the desktop study, the Albion process was selected as the basis of the subsequent 2009 definitive feasibility study. Revisiting the process options after recognizing the technical uncertainties highlighted by recent Albion testwork in 2012 and 2013 led to extensive testing of an alternative process option based on pressure oxidation.

Batch and continuous pressure oxidation testwork completed in 2013, as reported in Section 13, demonstrated that the pressure oxidation process is robust for treatment of Certej concentrate. Excellent gold recovery was achieved from the various ore types, under a wide range of operating conditions. Gold recovery by the pressure oxidation process was show to be sensitive to both organic carbon content in the concentrate and soluble chloride in the process solution. The organic carbon present in the Certej ore was shown to have preg-robbing properties, and the effect of this organic carbon on gold recovery was shown to be more pronounced in the presence of increasing soluble chloride concentrations. As a result, conditions selected as the basis for commercial design of the process plant have been chosen specifically to mitigate the adverse effects of chloride and organic carbon, and were demonstrated to be highly effective in the pilot plant testwork.

The flowsheet as selected for processing of the Certej concentrate includes pressure oxidation, conditioning of the pressure oxidation product solids (hot curing), washing of the conditioned pressure oxidation product solids prior to cyanide leaching, and neutralization of the pressure oxidation solution to neutralize acid and precipitate dissolved metals.

To improve silver recovery, a lime boil (hot alkaline treatment of the washed pressure oxidation product solids) circuit is included prior to cyanide leaching. Silver typically forms argentojarosite during pressure oxidation, and argentojarosite is relatively stable and does not readily release silver during cyanide leaching. In the lime boil treatment, washed pressure oxidation product solids are contacted with lime, at near-boiling conditions, to decompose argentojarosite such that the silver is released and can be effectively recovered in subsequent cyanide leaching.

The Certej concentrate contains very little carbonate, and therefore an acid pre-treatment step prior to pressure oxidation is not required.

Gold and silver recoveries obtained from pilot testing of the pressure oxidation process, completed by Sherritt in 2013, are summarized along with those recoveries used in the current pre-feasibility study and economic analysis in Table 17-2.

Table 17-2: Overall Recoveries of Gold and Silver

		Recovery from Metallurgical Testwork	Recovery for Design Basis
Overall Recoveries (from Ore), %	Au	91.0	87.4
	Ag	79.8	80.0

Pressure oxidation has been widely adopted for the commercial treatment of refractory gold-bearing ores and concentrates. A summary of commercial pressure oxidation plants (excluding alkaline operations) is given in Table 17-3.

Table 17-3: Commercial Pressure Oxidation Plants

Start	End	Company	Project/Mine	Country	Feed	Design Capacity		Feed Grade	
						t/d	t/a	Au, g/t	S, %
1985	2002	Homestake Mining	McLaughlin	USA	Ore	2 700	985 500	5.8	2.9
1986	2007	Gencor	Sao Bento	Brazil	Con.	240	87 600	25.0	18.7
1989	2004	FirstMiss Gold	Getchell	USA	Ore	2 730	996 450	5.7	4.0
1990	--	American Barrick	Goldstrike	USA	Ore	1 350	492 750	7.8	2.5
1991	--	Barrick / Placer Dome	Porgera	Papua New Guinea	Con.	4 050	1 478 250	32.0	9.0
1991	--	Placer Dome Mines	Campbell	Canada	Con.	70	25 550	450	17.7
1992	2003	Nerco	Con	Canada	Mix	90	32 850	43.7	23.2
1993	--	American Barrick	Goldstrike (Expansion)	USA	Ore	17 030	6 215 950	7.8	2.5
1994	2004	Santa Fe	Lone Tree	USA	Ore	2 270	828 550	2.1	4.0
1997	--	Santa Fe / Newmont	Twin Creeks	USA	Ore	3 630	1 324 950	8.0	4.6
1998	--	Lihir Gold Corporation	Lihir	Papua New Guinea	Ore	9 500	3 467 500	5.1	7.2
1999	--	OceanaGold	MacCraes	New Zealand	Con.	528	192 720	18.9	10.3
2008	--	Agnico Eagle	Kittila	Finland	Con.	300	109 500	47.0	19.4
2012	--	Anglogold Ashanti	Corrego do Sitio	Brazil	Con.	194	70 810	50.0	8.2
2012	--	Barrick / Goldcorp	Pueblo Veijo	Dominican Republic	Ore	24 000	8 760 000	5.0	6.7
2012	--	Polymetal	Albazino	Russia	Con.	616	224 840	24.0	6.5
2013	--	Newcrest Mining	Lihir (Expansion)	Papua New Guinea	Ore	9 600	3 504 000	5.1	7.0

The pressure oxidation process designed for treatment of Certej flotation concentrates is very similar to that recently adopted by Barrick Gold at their Pueblo Viejo plant in the Dominican Republic in that a lime boil treatment is included to improve silver recovery. Additionally, several commercial pressure oxidation operations, such as Kittila, Lihir and Corrego do Sitio, are known to successfully manage either organic carbon in the feed materials or chloride in the process solutions (or a combination of both).

Relative to the size of other commercial pressure oxidation plants, the proposed Certej plant is considered to be large for treatment of a flotation concentrate, although it is well within the range of commercial pressure oxidation plants. For comparison, the proposed Certej plant is roughly an order of magnitude larger than the former Sao Bento operation in Brazil. It is worthwhile to note that in 1996, Eldorado Gold acquired the Sao Bento pressure oxidation plant in Brazil from Gencor and successfully operated that plant until exhaustion of the reserves in 2007.

17.4 DESIGN BASIS

The Certej process plant is designed to process 3.0 Mt/a of ore with a mill feed grade of 1.63 g/t Au, 11 g/t Ag and 4.1% S^{2-} over the life of mine (LOM). The Certej deposit is characterized by four ore zones; over the life of the mine, the proportion of each ore zone in the mill feed will be variable. For the purpose of the current Pre-Feasibility study, design of the process plant is based on a 1:1:1:1 blend of the four ore zones. A summary of Certej ore is provided in Table 17-4:

Table 17-4: Certej Ore Grade

Assay	From Metallurgical Testing					Design Basis
	Ore Zone				1:1:1:1	
	East	West	Intermediate	Central	Blend	
Au, g/t	2.00	1.10	1.96	1.84	1.72	1.63
Ag, g/t	6.38	9.92	14.5	17.7	12.1	11
S, %	3.91	3.70	4.51	3.99	4.03	4.1

The process plant has been designed to recover approximately 0.95 Mt/a of flotation concentrate with a maximum sulphide sulphur content of 13.8% S^{2-}. Subsequent pressure oxidation of the concentrate and cyanidation of the pressure oxidation product solids will produce nearly 135,000 oz/a Au and 800,000 oz/a Ag as doré. A summary of the overall recovery of gold and silver from the various Certej ore zones, the design blend and that selected as the basis of the Pre-Feasibility study is reported in Table 17-5.

Table 17-5: Overall Recovery of Gold and Silver from Certej Ores via Pressure Oxidation-based Processing

Metallurgical Testwork Results							
		East	**Central**	**Intermediate**	**West**	**1:1:1:1 Blend**	**Selected Design Basis**
Overall Recoveries (from Ore), %	Au	88.7	91.9	90.8	93.7	91.0	87.4
	Ag	76.7	84.4	80.6	80.2	79.8	80.0

Design of the Certej process plant is based on the following number of working days per annum:

- mining - approximately 290 d/a (based on 80% annualized availability);
- crushing - approximately 237d/a (based on 65% annualized availability); and,
- processing - approximately 310 d/a (based on 85% annualized availability).

17.4.1 Process Inputs

Input streams to the Certej process plant include:

- run-of-mine (ROM) ore;
- make-up water pumped from the Mures River;
- acid rock drainage (ARD) water (also referred to as contact water) from appropriate collection ponds;
- non-contact water from appropriate diversion channels;
- high pressure gaseous oxygen;
- limestone and hydrated lime;
- various reagents and consumables; and,
- electrical power.

17.4.2 Process Outputs

Output streams from the Certej process plant include:

- flotation tailings for impoundment;
- CIL tailings for impoundment;
- neutralization tailings for impoundment;
- effluent water, treated to Romanian discharge standards;
- scrubbed vent gases and steam emitted to the atmosphere; and,
- gold-silver doré.

17.5 PROCESS PLANT DESCRIPTION

17.5.1 Overall Description

The Certej process plant layout is divided into two sections at the project site: the ore preparation and flotation section, and the concentrate oxidation and cyanidation section.

The ore preparation and flotation section comprises the following principal unit operations:

- single stage primary crushing of run-of-mine (ROM) ore using a jaw crusher;
- grinding of ore by semi-autogenous grinding (SAG) followed by ball milling;
- rougher-scavenger and cleaner flotation to produce a gold-silver concentrate;
- concentrate regrinding by ball milling followed by thickening in a conventional thickener; and,
- dewatering of the flotation tailings in a conventional thickener.

From the ore preparation and flotation section, the thickened concentrate slurry will be forwarded to the concentrate oxidation and cyanidation section, while the flotation tailings slurry will be forwarded to a tailings management facility (TMF) for impoundment;

The concentrate oxidation and cyanidation section comprises the following principal unit operations:

- pressure oxidation of the sulphide concentrate in two parallel autoclave trains followed by hot curing of the pressure oxidation product solids;
- counter current decantation (CCD) washing of the pressure oxidation product solids with recycled neutralization solution;
- neutralization of the acidic pressure oxidation liquor with limestone and lime, followed by dewatering of the neutralization tailings in a conventional thickener;
- lime boil treatment of the washed, pressure oxidation product solids to enhance silver recovery;
- cyanide leaching ofthe washed and lime boiled pressure oxidation product solids in a conventional cyanide-in-leach (CIL) circuit, with subsequent thickening of the CIL tailings in a conventional thickener;
- recovery of gold and silver from the loaded carbon by stripping in an two-stage Zadra stripping circuit, with concurrent electrowinning of the gold and silver;
- recovery of the gold-silver cathode sludge and smelting to produce doré bars; and,
- cyanide detoxification of the CIL tails, to less than 3 mg/L weak acid dissociable cyanide (CN_{WAD}), by the Inco SO_2 - air method using sodium metabisulphite as the source of SO_2.

From the concentrate oxidation and cyanidation section, the neutralization tailings and CIL tailings (following cyanide detoxification) slurries will be forwarded to the tailings management facility (TMF) for impoundment.

Services and utilities to each section of the Cetrej process plant will be provided from centralized facilities.

17.5.2 Ore Crushing

The crushing section incorporates a ROM pad, a primary jaw crusher, and a primary crushed stockpile. The ROM ore is generally fed to the primary crusher by direct tipping from the mine haul trucks.

Under normal conditions, the crushing plant will be operated in three shifts per day, with a design capacity of 600 t/h and an availability of 65%. The crushed ore stockpile has a nominal live capacity of 10,000 t crushed ore, which provides16 h surge capacity for the SAG mill.

A provision will be included for the addition of a small amount of limestone to the crusher feed to neutralize any possible acidity generated from the ore and minimize corrosion of equipment.

17.5.3 Ore Grinding

SAG and comminution testwork undertaken by SGS-Lakefield and subsequent mill sizing calculations by Hatch determined the size of mills and motors required for a plant throughput of 3.0 Mt/a and a product size of 80% passing 90 µm. The selected SAG mill will have the dimensions of 8.5 m diameter x 3.0 m Effective Grinding Length (EGL), and the selected ball mill will have the dimensions of 6.1 m diameter x 9.8 m EGL. Both SAG and ball mill motors will be driven by variable speed drives rated at about 5.3 MW, each.

Pebble ports of 70 mm on the SAG mill discharge end will remove pebbles to a trommel screen, with the screen oversize being forwarded by conveyor to a pebble crusher and the crushed pebble being recirculated back to the SAG mill feed.

Ground slurry from the SAG mill is discharged through a grate discharge with 38 mm openings onto a classifying trommel screen. The trommel screen undersize will flow to the discharge pumpbox where the combined SAG and Ball mill products will be pumped to a cluster of 8 x 500 mm hydrocyclones, with four of the cyclones normally being fed, for classification. The primary cyclones are equipped with actuated valves. The circuit is designed to yield cyclone overflow of normally 35 wt% solids with particle size of 80% passing 90 µm. The particle size of the cyclone overflows are measured by an on-stream particle size analyzer.

The SAG mill load will be measured by bearing pressure and sound detection. An expert system may be added at a later stage of the project, subject to costs.

The SAG mill feed dilution water is ratio controlled based on the SAG mill fresh feed rate. The pumpbox dilution water is controlled by the cyclone overflow pulp density. The pumpbox level is controlled by the pump speed.

17.5.4 Flotation

The flotation circuit will comprise rougher and scavenger flotation, as well as a single stage of cleaner flotation Although testwork showed that regrinding of the flotation concentrate was not required to enhance the sulfide oxidation rate during pressure oxidation, the design provides a regrind ball mill for the rougher and cleaner concentrates. This design assures that concentrate will be fed to the pressure oxidation circuit at the design particle size.

Cyclone overflow from the comminution circuit will be fed by gravity flow to the flotation circuit. The slurry will be combined with various reagents and conditioned for 10 minutes in two stages prior to the rougher flotation cells. Rougher tails will be forwarded to the scavenger cells and the scavenger concentrate will be further upgraded in the cleaner cells. The current flotation design allows for recycle of cleaner tails back to the scavenger cells, based on flotation performance. The rougher and cleaner concentrates will be combined and forwarded to the regrind circuit. The scavenger tails will be thickened prior to disposal.

Flotation will be completed in tank cells, each individual air supply and electrically controlled air valves. The cells will be installed in a step-wise fashion providing controlled gravity flow between adjacent grouped cells.

The rougher and scavenger flotation cells will be identically sized with appropriate piping flexibility to allow adjustment in the number of cells in each stage based on flotation response. The rougher flotation will normally comprise 6 x 150 m^3 cells and will have a nominal working volume of 900 m^3, with level control dart valves between alternate cells. The scavenger flotation will normally comprise 4 x 150 m^3 cells providing a nominal working volume of 600 m^3.

The combined rougher and cleaner concentrates will report to the discharge pumpbox of the concentrate regrind mill. The scavenger tailings will report to a thickener for dewatering. All concentrate and tailings pumpboxes are equipped with variable speed pumps and level controls.

There is an on-stream analyzer for sulphur content in the combined rougher and cleaner concentrates.

17.5.5 Concentrate Grinding and Handing

Rougher and cleaner concentrates will be combined and then pumped to a cluster of 8 x 300 mm hydrocyclones, with four normally being fed, for classification. To ensure that the concentrate fed to the pressure oxidation meets the design particle size, the oversize fraction (cyclone underflow) is reground to a target size of 80% passing 63 µm in a conventional ball mill. Regrind mill discharge slurry is returned to the cyclones for classification. The cyclone overflow is gravity fed to a thickener for dewatering.

The selected ball mill will have dimensions of 3.7 m diameter x 7.6 m EGL and will be driven by a fixed speed 1.4 MW motor.

The regrind cyclone feed density controls dilution water flow rate to the cyclone feed sump while the sump level is controlled by pump speed. The cyclone inlet pressure is monitored through the plant distributed control system (DCS).

The flotation concentrate from the flotation circuit will be pumped to a concentrate thickener feed box and then to a 24 m diameter high-rate thickener. The thickener underflow density will be controlled at approximately 60% solids by weight for feed to the pressure oxidation circuit. The design provides for a concentrate surge capacity equivalent to 24 hours of operation at design throughput to decouple the ore preparation and milling operations from the pressure oxidation and cyanidation operations.

17.5.6 Flotation Tailings Handling

The scavenger flotation tailings will be pumped directly to a thickener feed launder. The flotation tailings thickener will be a 38 m diameter high-density thickener, and the underflow density will be operated at as high as possible, but nominally 50% solids by weight.

The thickened flotation tailings will be pumped to the Tailings Management Facility. The overflow from the flotation tailings thickener will be pumped back to the flotation process water tank for recycle within the ore preparation and flotation section of the process plant.

17.5.7 Pressure Oxidation Plant

In the pressure oxidation circuit, the sulphide minerals in the concentrate feed slurry are reacted with oxygen (oxidized) at elevated temperature and pressure in an agitated pressure vessel (autoclave), to liberate gold and silver by decomposing the sulfide minerals. Sulfide minerals, such as pyrite, are largely dissolved and the soluble iron is subsequently precipitated. The liberated gold and silver can be then recovered by cyanide leaching. Without pre-treatment by oxidation, gold and silver will be largely unrecoverable by cyanidation.

A reaction for the oxidation of pyrite, with subsequent formation of basic ferric sulphate, can be described as:

$$2FeS_2 + 7.5\ O_2 + 3H_2O \rightarrow 2Fe\ (OH)SO_4 + 2H_2SO_4$$

The sulfide oxidation reactions are completed inside an autoclave; a horizontal pressure vessel divided into multiple agitated compartments and operated at elevated temperature and pressure.

Because the iron is largely precipitated as a metastable basic ferric sulphate, the autoclave discharge is subsequently conditioned to dissolve much of the basic ferric sulphate, thereby reducing lime consumption in the subsequent lime boil circuit.

The pressure oxidation circuit includes autoclaves and associated feed and discharge systems, the agitator seal system, and the hot cure step.

17.5.8 Pressure Oxidation Autoclave

The pressure oxidation circuit consists of an autoclave feed tank, two autoclaves, two flash discharge tanks, a flash vent gas scrubber, one emergency vent knock-out tank, a high pressure steam blowback tank, an oxygen blowback tank, a scrubber water tank, a quench water system, two seal water systems and associated pumps.

Concentrate feed slurry is received in the autoclave feed tank and is subsequently pumped at a controlled rate to the autoclaves by high pressure positive displacement pumps. The autoclaves are operated at a total pressure of 3,547 kPa(g) and temperature of 230°C. Design of the Certej Process Plant includes two autoclaves, each with nominal dimensions of 5 m diameter by 24 m tangent to tangent.

Oxygen is sparged into the autoclaves to oxidize sulphide minerals contained in the concentrate. Quench water is used to control the temperature in the autoclaves. Quench water, a combination of pressure oxidation plant process water and filtered makeup water (or recycled TMF effluent water), is collected in the autoclave quench feed tank and pumped at a controlled rate to the autoclaves. Quench water is also distributed to the autoclave discharge slurry lines to serve as choke water to assist in controlling the level of slurry in the autoclaves. For start-up, high-pressure steam is injected into each compartment to heat the autoclaves.

A seal water system provides demineralized water to the mechanical seals of the autoclave agitators and the seals of the quench water pumps.

Oxidized slurry is discharged from the autoclaves and is flashed (reduced to roughly atmospheric pressure) to 256 kPa (g) pressure in the flash tanks. Discharge slurry from the flash tanks then proceeds to the corresponding seal tank in the conditioning area. The vent gases from the autoclaves are also directed to the corresponding flash tanks. A portion of the flashed steam from the flash tanks is directed to the lime boil circuit and the rest of the flashed steam is sent to a flash vent scrubber. Vent gases from the seal tanks and conditioning tanks in conditioning area are also directed to the flash vent scrubber. Process water is sprayed inside the vent scrubber to partially condense vent gases reporting to the scrubber. The vent steam condensed in the vent scrubber along with vent scrubber make-up water is collected in the scrubber water tank used in the lime boil circuit as dilution water. A portion of the vent scrubber bottoms stream from the scrubber water tank is recirculated through the scrubber. Scrubbed gases along with some steam from the scrubber are released to the atmosphere.

If required, the autoclaves can be rapidly depressurized through an emergency vent valve, and vent gases are directed to an emergency knockout tank.

17.5.9 Autoclave Seal Water System

The seal water system provides high-pressure water for lubrication and cooling of the autoclave agitator seals. Seal water is pumped from the seal water make-up tank through the seal water make-up filter to the seal water accumulator. The seal water accumulator pressure is maintained with high-pressure nitrogen. High-pressure water from the accumulator is pumped through the seal water filter to the autoclave agitators. Hot seal water from the autoclave agitators is cooled in the air-cooled seal water cooler before returning to the seal water make-up tank. Seal water stored in the accumulator can be transferred to the autoclave agitators with available seal water pressure in the accumulator during a power failure until emergency power is available.

Seal water is also delivered to the quench water pumps, from the corresponding seal water system, to serve as gland water for seal cooling.

Seal water that is lost from the system due to leakage is replaced with a combination of pressure oxidation plant process water delivered to the seal water make-up tank from the pressure oxidation plant process water tank and filtered make-up water delivered from the water collection pond.

17.5.10 Conditioning (Hot Cure)

In the conditioning (hot cure) circuit, basic ferric sulphate in the flashed discharge slurry from the pressure oxidation circuit is provided a sufficiently long residence time to dissolve, thus minimizing the consumption of lime in the lime boil circuit. The hot cure circuit consists of two seal tanks (one for each flash tank), two common conditioning tanks and associated pumps.

The seal tanks and the conditioning tanks operate at a pressure of 94 kPa(a) and a temperature between 95°C and 97°C. Vent gases from the seal tanks and conditioning tanks are directed to the flash vent scrubber in the pressure oxidation area. Oxidized slurry from the conditioning tank is pumped to the counter current decantation (CCD) wash circuit.

17.5.11 Countercurrent Decantation Washing

In the CCD wash circuit, the conditioned pressure oxidation product slurry is thickened and washed to remove entrained acid and metals with the product solids. The CCD wash circuit consists of three wash repulp tanks, three 36 m diameter high rate thickeners, three overflow solution tanks and associated pumps.

All three CCD wash thickeners are equipped with proprietary self-diluting feed well systems to control the solids feed concentration entering the feed well to around 5 wt%.

Oxidized slurry from the conditioning circuit is pumped to the CCD wash circuit where it is repulped to approximately 9 wt% solids in the first wash repulp tank using overflow solution recycled from the first wash thickener. Overflow solution from the second wash thickener is also added to the first wash repulp tank for solids washing. Repulped slurry feeds the first wash thickener, which operates to target 33 wt% solids in the underflow. A portion of the overflow solution collected in the first wash thickener overflow tank is pumped to the neutralization circuit.

First wash thickener underflow is pumped via the first wash thickener underflow slurry pump to the second wash repulp tank and repulped to approximately 11 wt% solids with overflow solution recycled from the second wash thickener. Overflow solution from the third wash thickener is also added to the second wash repulp tank for solids washing. The resulting slurry gravity feeds the second wash thickener, which operates to target 34 wt% solids in the underflow. The second wash thickener overflow solution collected in the second wash thickener overflow tank is pumped to the first and second wash repulp tanks for washing and dilution respectively.

Second wash thickener underflow is pumped via the second wash thickener underflow slurry pump to the third wash repulp tank and repulped to approximately 11 wt% solids with overflow solution recycled from the third wash thickener. Neutralized solution from the neutralization circuit is added to the third wash repulp tank and serves as the net wash water addition for the CCD wash circuit. The resulting slurry gravity feeds the third wash thickener, which operates to target 34 wt% solids in the underflow. The third wash thickener overflow solution collected in third wash thickener overflow tank and is pumped to the second and third wash repulp tanks for washing and dilution respectively.

Washed oxidized slurry from the third thickener underflow is pumped to the lime boil circuit.

Flocculant solution at specified concentration is added to all three of the thickeners to aid in the settling of solids, to optimize the percent solids in the underflow slurries of thickeners and to maintain the overflow clarity. The flocculant solution addition rate is controlled in proportion to the solids feed rate to the thickeners by the speed of flocculant metering pumps set by the operator.

17.5.12 Lime Boil

In the lime boil circuit, the washed pressure oxidation product solids are further treated to enhance silver recovery in the cyanidation circuit. The lime boil circuit consists of a lime boil heat exchanger, four lime boil tanks arranged in series, three lime boiled product coolers (two operating and one standby) and associated pumps.

The washed, pressure oxidation product slurry from the third wash thickener is pumped to the lime boil heat exchanger where the flash tank vent steam from the pressure oxidation circuit is injected directly into the exchanger to heat the slurry to about 96°C.

The discharge slurry from the heat exchanger is directed to the first of four agitated cascading lime boil tanks. Limestone slurry is added to the first lime boil tank to react with soluble metals and free acid. Lime slurry is then added to the second lime boil tank to increase the pH and complete the desired reactions. Provision is made to add lime slurry in the third and fourth lime boil tanks as required. Dilution water consisting of hot scrubber water is added to the first lime boil tank to control the product slurry discharge solids concentration to 25 wt%. The temperature is maintained at about 95°C in all four lime boil tanks by using the flash vent steam from the pressure oxidation autoclave circuit. Provision is also made to inject low-pressure steam directly to the first three lime boil tanks to maintain the temperature.

Treated slurry from the fourth lime boil tank is pumped to the product slurry coolers to cool the product slurry to about 35°C. The cooled product slurry is pumped to the cyanidation circuit.

17.5.13 Solution Neutralization

The objectives of the solution neutralization circuit are to neutralize the free acid and to precipitate the dissolved metals contained in the CCD overflow solution by contact with limestone and lime slurry, and to produce an inert residue containing primarily gypsum ($CaSO_4 \cdot 2H_2O$) and other metal hydroxides. The resultant solution can be appropriately recycled within the overall process plant.

The neutralization circuit consists of four neutralization tanks in series, a thickener, an overflow solution tank, and associated pumps.

CCD first wash thickener overflow solution is fed to the first of four cascading neutralization tanks. Limestone slurry is added to the first neutralization tank to achieve a pH of about 5.5 (measured at ambient temperature) in the discharge of the second tank. Process air is added to the first two tanks to remove carbon dioxide gas and oxidize ferrous iron (Fe^{2+}) to ferric iron (Fe^{3+}), in solution. Lime slurry is added to the third neutralization tank to achieve a pH of about 8.0 (measured at ambient temperature) in the neutralization discharge to the neutralization thickener. Provision of lime slurry addition to the fourth neutralization tank allows the target pH to be maintained as required. The neutralization tanks operate at between 70°C and 75°C.

Neutralized slurry is gravity fed to the neutralization thickener, the thickener feed is targeted to achieve 12 wt% solids with a proprietary self-diluting feed well system. The underflow from the thickener is maintained at 59 wt% solids and is pumped to the Tailings Management Facility.

Overflow solution is collected in a tank. A portion of the overflow solution is recycled back to the CCD wash circuit (as wash water) and the remainder sent to the pressure oxidation process water tank for recycle within the process plant. When neutralization thickener overflow is not available in sufficient quantities to meet the wash water requirement, or is off-spec (high chloride) solution, process water can be added in the first neutralization tank to ensure the quality and quantity of the wash water.

Flocculant solution at specified concentration is added to the neutralization thickener to aid in the settling of solids, to optimize the percent solids in the underflow slurry of the thickener, and to maintain the overflow clarity. The flocculant solution addition rate is controlled in proportion to the solids feed rate to the thickener by the speed of flocculant metering pump set by the operator.

Vent gases from all four neutralization tanks, containing carbon dioxide, water vapour, nitrogen and unused oxygen, are discharged to the atmosphere.

17.5.14 Carbon in Leach (CIL) Cyanidation

Following pressure oxidation and post treatment of the pressure oxidation product solids, gold and silver are extracted in a conventional carbon-in-leach (CIL) cyanidation circuit and recovered by stripping and electrowinning. Because of near complete oxidation of sulphide minerals achieved in pressure oxidation, the cyanidation circuit is very similar to that applied for traditional oxide ores, with the noted exception of having to treat relatively low solids concentration slurry due to potential viscosity limitations.

The cyanidation circuit is designed to provide approximately 30 hours retention time in eight agitated tanks. Air is added through lances discharging below the bottom impeller. The slurry pH will be adjusted, as required, to maintain a target pH between 10 and 11 by the addition of lime slurry from a ring main. Cyanide solution will be fed to the CIL tanks with appropriate dosing pumps. Regenerated carbon and attritioned fresh carbon will be fed to the last CIL tank; carbon concentration will be maintained at 30 g/L in the first two CIL tanks, and 20 g/L thereafter.

The gold and silver loading on activated carbon has been established to be 7,000 g/t. Each CIL tank will be equipped with an interstage screen which will retain carbon while the slurry will flow via pneumatic dart valves and launders from one CIL tank to the next. Carbon will be transferred on an intermittent basis (each 6 h) in counter-current fashion by interstage pumps. Carbon transfer will be offset in that carbon from even and odd numbered CIL tanks will not be transferred at the same time to avoid short-circuiting. Bypass launders will be installed to bypass any of the CIL tanks for maintenance.

Slurry leaving the last CIL tank will flow to a carbon safety screen and then to a CIL tailings thickener. The CIL tailings will be thickened in a single 27 m high-rate thickener. Thickened CIL tails will be forwarded to the cyanide detoxification circuit for destruction of the residual cyanide. Overflow solution will be recycled within the cyanidation circuit (such as for cyanide make up and dilution water) as much as possible to reduce the overall cyanide consumption. Excess CIL tails thickener overflow, which is not required within the cyanidation circuit and which cannot be recycled elsewhere due to the residual cyanide, will be combined with the CIL tails thickener underflow slurry and forwarded to the detoxification circuit.

17.5.15 Carbon Stripping and Gold Electrowinning

Gold/silver loaded carbon will be withdrawn from the first or second CIL tanks by carbon transfer pumps and pumped to a screen where the slurry will be washed off from the carbon. Screen undersize will flow back into the same CIL tank, while the carbon collected on the screen will gravitate into a loaded carbon storage tank.

The stripping section will treat carbon from the CIL circuit to remove gold and silver based on a two-stage Zadra system. A single 26 tonne batch (or potential 2 x 13 t batches, pending operational preference) of carbon will be stripped each day.

Loaded carbon collected on the loaded carbon screen will fall through a loaded carbon sampler and into the loaded carbon storage tank. Carbon is pumped from the surge tank into the acid wash column. Acid washed carbon will be transferred from the acid wash tank to the stripping column by pressurizing the former vessel with transportation water.

The first stage and second stage stripping cycles will be operated at 100 and 130°C, respectively. Spent electrolyte will serve as the stripping solution. Stripping solution will be pumped by circulating pumps through the stripping column. The temperature of the elution solution will be raised to the required temperature using an LPG (or NG) fired thermal fluid stripping heater.

Pregnant strip solution will then flow to the electrowinning and spent electrolyte will be returned to the stripping column in closed loop fashion.

A standard system of electrowinning will be used to recover gold and silver from solution. Pregnant strip solution will flow from the cooler to the flash tank where it will gravitate to two banks of two electrowinning cells. There, the gold and silver will be deposited on stainless steel cathodes. Spent electrolyte will flow back to the strip solution tank. Loaded stainless steel wool cathodes will be stripped manually in the cells using high-pressure water.

17.5.16 Gold Smelting

Sludge from the cathodes and the cells will be collected, filtered and washed in a plate and frame pressure filter. The filter cake will be tried, mixed with fluxes and smelted in an induction furnace. The gold and silver doré will be poured in cascading bar moulds, with displacement of slag to the final moulds. Slag will be collected, coarsely ground and tabled to recover coarse gold/silver. The table tailings will be finely ground and then combined with cyanide leach feed.

Bullion bars, assaying nominally 15 to 20% Au and 80 to 85% Ag, will be stored in a vault prior to dispatch. A bullion balance and sample balance will be provided.

17.5.17 Carbon Regeneration

Stripped carbon will be fed into an electrically (or LPG) heated horizontal regenerating kiln designed to treat nominally 1 t/h. The carbon will be reactivated at a temperature of 750°C.

Reactivated carbon will discharge from the kiln into a quench tank. Water from the transportation water tank will be used for quenching. The regenerated carbon pump will transfer the carbon from the quench tank onto the regenerated carbon screen to remove carbon fines.

17.5.18 Cyanide Detoxification

The anticipated level of weak acid dissociable free cyanide (CN_{WAD}) is 200 mg/L in the CIL tailings stream, and a reduction of the cyanide level to less than 3 mg/L CN_{WAD} is required prior to impoundment. This will be achieved by treatment in an INCO SO_2-air cyanide destruction plant.

The SO_2 -air cyanide destruction entails the oxidation of cyanide ion to less toxic cyanate, OCN^-:

$$SO_2 + O_2 + CN^- + H_2O \rightarrow OCN^- + H_2SO_4$$

This reaction is catalyzed by the presence of copper. Copper sulphate solution will be used to maintain a target copper concentration in the cyanide detoxification solution. Base metals that have previously been complexed by the cyanide, such as zinc, are precipitated as metal hydroxides.

Thickened underflow slurry will be pumped continuously from the CIL thickener underflow at about 39% solids to the cyanide destruction circuit. Any CIL tails thickener overflow solution, which is not recycled within the cyanide leach circuit, will be combined with CIL tailings thickener underflow in the first of two cyanide destruction tanks.

Sodium metabisulphite will be added to the first (and or second) cyanide destruction tank to provide the source of SO_2. Process air will be sparged into each reaction tank to provide the requisite oxygen. The pH will be maintained with the addition of lime as required; a pH probe installed in the slurry will be used to maintain the set point by automatic lime addition.

The detoxified CIL tailings will overflow from the final reaction tank to a pumpbox and then forwarded to the CIL TMF.

17.5.19 Tailings Management and Water Treatment

The Certej process plant will produce flotation tailings, CIL tailings and neutralization tailings as distinct tailings streams, all of which are sent to the TMF for impoundment. The initial production planning for the Certej facility provide for all of the tailings streams to be combined in a single TMF. The provision to maintain the flotation tailings separate from the combined CIL and neutralization tailings has also been considered and may be adopted as part of the feasibility study design.

A TMF facility selection process has identified possible sites within the concession area surrounding the Certej open pit. These were formally assessed with respect to political, social, environmental and technical characteristics.

17.5.20 Transport of Flotation, CIL and Neutralization Tailings Slurries

The flotation tailings thickeners underflow will be pumped at a density of about 50% solids (by weight) by means of four stages of centrifugal slurry pumps or a positive displacement slurry pump. Similarly, the CIL and neutralization tails will also be pumped by dedicated pumps and pipelines from the cyanide detoxification and solution neutralization circuits, respectively, to the TMF.

Water will be reclaimed from the TMF and forwarded to the water treatment plant by two-stage centrifugal pumps.

17.5.21 Water Treatment Plants

A water treatment plant will be located adjacent to the TMF to permit treatment of the reclaimed tailings effluent solution to the Romanian discharge standard, prior to recycling within the process plant or discharge to the environment.

Preliminary design suggests that water reclaimed from the TMF may be treated by a water treatment plant comprising lime neutralization using one agitated tank plus a 15 m diameter clarifier to remove the precipitated sludge with the addition of 17 g/m^3 of flocculant. The resulting residue from the water treatment plant will be returned to the TMF for impoundment.

The water will be treated to meet the process requirements, specifically for residual chloride and cyanide, and as required to meet the Romanian discharge standards. Design of the water treatment plant is subject to change. Preliminary testwork indicates that a treatment plant may include pH adjustment, microfiltration of the water and reverse osmosis. The resulting brine would be recycled to the TMF; the permeate would be collected in a holding tank before being forwarded to the process water system or discharged to the environment. As required to meet cyanide limits for recycle and or nitrite limits for discharge, ozone may be added to the permeate before discharge.

Further testwork and design of the water treatment plant will be completed as part of the Feasibility Study.

17.6 PROCESS CONSUMABLES, REAGENTS AND CHEMICALS

17.6.1 Limestone Preparation

Approximately 550 000 t/a limestone will be required for neutralization of acid and pH adjustment in the process plant.

A small fraction of the crushed limestone, about 30 000 t/a as received, will be fed as required by loader to the ore crusher feed to neutralize any natural acidity in the ore.

The primary crushed limestone will be fed, from a silo or stockpile, by a variable speed apron feeder to the secondary crusher. The limestone will be crushed from nominally 80% passing 120 mm to 80% passing 15 mm. The minus 15 mm product from the secondary crusher will feed directly to the limestone ball mill, which will operate in closed circuit mode with hydrocyclones to yield a product with a P80 of 50 µm. The selected ball mill will have dimensions of 3.2 m diameter x 4.6 m EGL and will be driven by a fixed speed 1.0 MW motor.

Ground limestone slurry from the mill will overflow to the limestone mill discharge sump, where it will be diluted with process water and then pumped to a cluster of 4 x 300 mm hydrocyclones, with two normally being fed, for classification.

Cyclone underflow will flow by gravity back to the limestone ball mill, and the overflow at about 40% solids will be collected and pumped to a 1,100 m^3 limestone storage tank, which will provide a live capacity of about 8 hours. One of two available fixed speed pumps will deliver limestone slurry to selected addition points within the process plant via a ring main, with excess limestone being continuously returned to the stock tank. This continuous pumping of ground limestone slurry from the agitated tank is to prevent settling of the solids in the pipeline.

17.6.2 Lime Preparation

Approximately 100,000 t/a lime (as $Ca(OH)_2$) will be required, principally for addition to the lime boil circuit, and to a lesser extent for the solution neutralization and cyanidation circuits.

From the lime silo, an auger feeder will add lime to a slurry preparation tank where it will be mixed with process water to generate a lime slurry of about 20% solids by weight. The lime slurry will be transferred to a 1,100 m^3 lime slurry storage tank, which supplies 20 hours of surge capacity.

One of two available fixed speed pumps will deliver lime slurry to selected addition points within the process plant via a ring main, with excess slime being continuously returned to the stock tank. This continuous pumping of limes slurry from the agitated tank is to prevent settling of the solids in the pipeline.

17.6.3 Oxygen Plant

The Certej process plant will consume approximately 380 000 t/a gaseous oxygen. Oxygen will be supplied at a minimum of 98% purity and 4 100 kPa(g) supply pressure from a third party supplier. The oxygen plant itself will be located at the Certej site but will be owned and operated by a third party supplier.

From the oxygen plant, gaseous oxygen will be supplied by a pipeline to the oxygen blowback tank in the pressure oxidation circuit. The blowback tank prevents the autoclave slurry from backing up into the oxygen header in the event of a process upset condition.

17.6.4 Cyanide

Cyanide will be supplied as solids sodium cyanide briquettes or prills, with a consumption of approximately 1 500 t/a. A stock sodium cyanide solution will be prepared on site for use in the CIL circuit.

17.6.5 Activated Carbon

Activated carbon will be regenerated on site and recycled internally. Activated carbon make up requirements are anticipated to be about 215 t/a, supplied as coconut husk carbon.

17.6.6 Flotation Chemicals

Several flotation chemicals are required at the Certej process plant. A summary of the specific chemicals and their annual requirements is provided in Table 17-6.

Table 17-6: Flotation Reagent Requirements

Flotation Reagent	Recommended Chemical	Consumption, t/a
Xanthate	Potassium Amyl Xanthate (PAX)	600
Collector	Cytec A3477	210
Frother	Oreprep F-549	195
Sodium Hydrosulphide[1]	NaHS	1070
Copper Sulphate	$CuSO_4 \cdot 5H_2O$	680
Sodium Silicate[1]	Na_2SiO_3	1800

[1] Included in design although not necessarily required for commercial operation.

Flotation reagents will be received in concentrated liquid or solid form; stock solutions will be prepared in batch wise fashion for use in the flotation circuit.

17.6.7 Flocculants

Four flocculants will be supplied to the Certej process plant from a third party supplier(s), including Ciba Magnafloc 351, Hychem AF304, Hychem CP905H, and Ciba Rheomax 1030 (or similar). Flocculants will be supplied in bulk as dry powders and will be mixed with process water to the prescribed solution strength for distribution in the process plant. A summary of flocculant requirements is provided in Table 17-7.

Table 17-7: Flocculant Requirements

Flocculant	Addition Point	Consumption, t/a
Hychem AF304	flotation tails thickener	100
Ciba Magnafloc 351	flotation concentrate thickener	50
Hychem CP905H	CCD Wash / CIL tails thickeners	270
Ciba Rheomax 1030	solution neutralization thickener	25

Individual flocculants will be prepared in designated make-up tanks to generate a stock solution on a periodic basis, typically once per shift. From the make-up tanks, the stock flocculant solution will be transferred to a storage tank, from which it will be metered to the specific addition points by way of a designated metering pump. In all cases, the operator will set the appropriate flocculant solution flowrate to provide the target dosage; fine adjustments will be made by way of needle or globe valves at the addition point.

Ciba Magnafloc 351 stock solution will be delivered to the reground concentrate thickener and Hychem AF304 stock solution is pumped to the flotation tailings thickener. Hychem CP905H stock solution will be distributed to all three CCD thickeners and the CIL tails thickeners. In addition, Ciba Rheomax 1030 stock solution will be pumped to the neutralization thickener.

17.6.8 Grinding Media

Grinding media additions are required for the four grinding mills included in the process plant design; the SAG mill, the primary ball mill, the concentrate regrind ball mill and the limestone ball mill. Nominal annual media requirements for each are provided in Table 17-8.

Table 17-8: Grinding Media Requirements

Grinding Equipment	Media Diameter	Consumption, t/a
SAG Mill	125 mm	3820
Ore Ball Mill	75 mm	5660
Concentrate Regrind Mill	50 mm	280
Limestone Ball Mill	80 mm	90

17.6.9 Hydrochloric Acid

Concentrated hydrochloric acid (32% HCl) will be received in bulk shipments and used to prepare a stock acid solution for acid washing of loaded carbon in the carbon stripping circuit. A consumption of 1750 t/a HCl (at 32% by volume) is expected.

17.6.10 Caustic Soda

Neutralization of the spent acid wash solution in the carbon stripping circuit will be completed with a stock solution of sodium hydroxide (or limestone/lime). Caustic soda is also used for carbon stripping. Solid form caustic soda will be received in bulk and used to prepare the stock solution on a batch wise basis. A consumption of 900 t/a NaOH is expected.

17.7 ENERGY AND WATER REQUIREMENTS

17.7.1 Plant Utilities

Electricity

The Certej process plant will have a nominal electricity requirement of 54 MW.

High Pressure Steam

A package boiler is included in the design to provide high (and low) pressure steam via pipeline to the POX circuit during start-up periods for heating purposes. Provision is made for addition of high pressure steam directly to the autoclaves for preheating purposes. High pressure steam is not required during normal operation.

Demineralized water is supplied to the package boiler feed water tank from the demineralized water plant. Saturated steam at nominally 4 100 kPa(g) is supplied by a pipeline from the package boiler to the high-pressure steam blowback tank in the pressure oxidation circuit. The blowback tank prevents the autoclave slurry from backing up into the steam header in the event of a process upset condition.

Low Pressure Steam

High pressure steam from the package boiler will be depressurized via a letdown station to nominally 260 kPa(g) to supply the low pressure steam header. Low pressure steam is supplied by a pipeline from the let down station to the flash tanks in the POX autoclave circuit for preheating purposes. A provision is also made for addition of low pressure steam to the lime boil circuit to permit preheating as required during start up. Low pressure steam is not required during normal operation.

17.7.2 Process Water

Water is collected from a number of sources for use as process water in the Certej process plant. Separate process water tanks are maintained in the flotation and pressure oxidation areas to ensure chloride distribution does not adversely affect gold recovery.

The total water requirement in the Certej process plant is about 1600 m^3/h, approximately 1,053 m^3/h of which is the fresh water requirement; the balance of the total water requirement is made up by recycling solutions within the plant, such as neutralization thickener overflow recycled as wash solution in the CCD. Under normal operating conditions, the fresh water requirement is made up from roughly 200 m^3/h of contact (ARD and mine water), 400 m^3/h of non-contact water collected from diversion ditches, and 400 m^3/h of water added form the Mures river. The process design allows for a total of 1000 m^3/h of water to be pumped from the Mures River, if either contact or non-contact water is in short supply. Under optimal conditions, approximately 370 m^3/h of water will be reclaimed from the TMF, treated in the water treatment plant, and recycled to the process plant, thereby reducing the Mures river water requirement to almost 0 m^3/h. The exact amount of makeup water to be supplied from the Mures River depends on the annual rainfall, as well as on the ultimate solids compaction achieved in the TMFs.

17.7.3 Ore Preparation and Flotation Process Water

Flotation area process water is generally comprised of recycled process effluents from the flotation concentrate and tailings thickener, with the balance coming from the pressure oxidation process water tank. Flotation area process water is recycled within the ore preparation and flotation circuits as required.

17.7.4 Concentrate Oxidation and Cyanidation Process Water

POX process water is comprised of a mixture of water pumped from the Mures river, water collected or diverted from outside the concession (collectively referred to as non-contact water), and effluent solutions recycled from the neutralization tailings thickener when available. In the event that treated TMF water is reclaimed to the process, it will also be added to the pressure oxidation area process water tank to reduce the requirement for river water. In extreme cases when neither non-contact water nor neutralization thickener overflow is available, the entire requirement for pressure oxidation area process water will be made up from the Mures River.

Pressure oxidation area process water is supplied by a pipeline from the pressure oxidation area process water tank to the pressure oxidation autoclave (principally for vent gas scrubbing), CCD wash, lime boil, cyanidation and neutralization circuits as required, as well as in limestone and lime preparation. Additionally, the pressure oxidation area process water can be used to make up any shortfall in the requirements for reagent preparation when selected recycle solutions are not available. Pressure oxidation area process water is also supplied to all utility stations and as make up water to the cooling water circuit.

17.7.5 Demineralized Water

A stream of pressure oxidation area process water is supplied to the demineralized water plant to generate sufficient demineralized water for the pressure oxidation autoclave requirements. Demineralized water is supplied by a pipeline from the demineralized water plant to the agitator seal water make-up tank and to the package boiler feed water tank.

17.7.6 Gland Water

Gland water is supplied to all the pumps in the process plant from a designated gland water circuit, which will operate in closed loop fashion. Gland water will be supplied from the cooling water supply, after having been cooled, and pumped at pressure to various consumers around the process plant. Gland water return will be collected in a similar collection main and returned to the cooling tower feed to be cooled. The primary consumers of gland water will be the high-pressure centrifugal pumps feeding the primary grinding circuit hydrocyclones and the tailings pumps, but in general, all centrifugal pumps will have some gland water requirement. Gland water will typically be filtered to remove suspended solids prior to distribution throughout the process plant.

17.7.7 Cooling Water

Cooling water is required in the oxygen plant at a nominal rate of 2,000 m^3/h. Cooling water is circulated from the cooling towers to the oxygen plant in a closed loop fashion. Cooling water make up is provided as required from the pressure oxidation area process water tank. The cooling tower will be designed to maintain an average cooling water temperature of less than 20ºC. As described above, a portion of the total cooling water supply will be used as gland water.

17.7.8 Potable Water

A small potable water plant is included in the design to treat pressure oxidation area process water for supply, by pipeline, to all the domestic consumers as well as all the utility safety showers and eye wash stations in the plant.

17.7.9 Process Air

Selected circuits, including the flotation, solution neutralization, cyanidation and cyanide destruction circuits, require process air. As such, process air compressors are included at both the ore preparation and flotation section, and the concentrate oxidation and cyanidation sections of the plant. Process air is supplied to specific addition points from the proximate compressor. Process air is also required for all the utility stations. Process air consumption in the process plant is expected to be of the order of 20,000 m^3/h

17.7.10 Instrument Air

A portion of the process air will be diverted for use as instrument air. Instrument air will be additionally filtered to remove oil and particulate and dried before being distributed throughout the process plant via pipeline. Instrument air is required for operating the control valves, on/off valves and for other instruments within the process plant. Appropriate local storage accumulators will be installed for instrumentation air supply. Instrument air consumption in the process plant is expected to be about 700 Nm^3/h.

SECTION • 18 PROJECT INFRASTRUCTURE

The Certej Project, located in Western Romania is ideally situated with respect to access to infrastructure, being in close proximity to major centers of Western Romania and having close access to the infrastructure required to develop and operate the Project.

The Project is located only 12 km NE of the city of Deva, the capital City of Hunedoara County and a regional center with a population of over 50,000 people.

Deva is situated 400 km NW of Bucharest, the capital of Romania. Cluj-Napoca, commonly known as Cluj, is located 163 km to the North of Deva. Cluj is the second largest city in Romania with a population of over 400,000 people and is considered one of Romania's academic, high technology, and industrial centers. Timisoara, the main social, economic and cultural center in the western part of Romania is located 135 km to the east of Deva; with a population over 300,000 people, it is the third most populous city in the country. Both Timisoara and Cluj have large modern manufacturing capability, international airports, and are major hubs in Romania's intermodal freight network.

After the Revolution, Romania was left with a significant but outdated infrastructure. This, combined with the economic decline that Romania faced in the 1990s due to its transition to a market economy, resulted in a period of relative decline in the country's infrastructure. This situation has been reversed since 2000 with considerable investment in infrastructure.

18.1 TRANSPORTATION AND LOGISTICS

For the construction of the Project, owner-supplied equipment for the processing plant will predominantly be sourced out of Europe, and bulk materials will be sourced within Romania. Proximity to major road and rail service that connect directly to Europe, Eastern Romania, and the Port of Constanţa, as well as access to international airports is an advantage for the Project.

The Project site is accessed by a paved road from the city of Deva. Deva is situated on the Muras River and is located on major highway and rail links between the world class Port in Constanţa on the Black Sea and Europe, as well as international airports in Cluj and Timisoara. Major transportation routes are shown in Figure 18-2: and Figure 18-2.

Figure 18-1: Transportation Routes in Romania

Figure 18-2: Rail Routes in Romania

18.1.1 Road System

Deva is located on National Highway 76, a main east-west connection from the Port of Constanţa with Europe. The highway is predominately a paved two lane highway, however; it is currently undergoing significant upgrades to a four lane highway with several of the upgraded sections already complete and operating.

18.1.2 Constanţa Port

The Black Sea Port of Constanţa is operated by the state-owned Maritime Ports Administration, located 250 km to the east of Bucharest on the Western coast of the Black Sea. The Port of Constanța is the largest on the Black Sea and the 18th largest in Europe. It is the main container hub in the Black Sea and all direct lines between Asia and Black Sea call into Constanţa.

Constanţa Port has a handling capacity of 50 million tonnes per year and depths range between 8 and 19 meters. These characteristics are comparable with those offered by the largest European and international ports making it the ninth busiest cargo port in Europe. The location of the Port is favorable due to proximity to the Danube waterway and access to the road and railway systems.

18.1.3 Danube River Transport

Constanţa Port is both a maritime and river port with river traffic accounting for 23.3% of the total traffic to the port. Daily, more than 200 river vessels are in the port for cargo loading or unloading. Connection of the port with the Danube River is made through the Danube–Black Sea Canal.

The Danube River is the country's most important waterway, and is classified as an international waterway. The Danube River is the longest river in Western Europe, and the second largest in Continental Europe. Cargo on the Danube River is primarily carried by river barges between Constanţa and Central and Eastern European countries.

18.1.4 Rail

The rail network in Romania is connected to the Port of Constanţa and the European rail network. Train freight services carry high volumes of freight to all areas of Romania and Europe.

The railway in Romania is state run by CFR Marfa, one of four autonomous state owned rail companies. CFR Marfa is headquartered in Bucharest with main regional centers in Cluj and Timisoara so is well positioned to service the Project. The Romanian railway network is significantly interconnected with other European railway networks.

18.1.5 Air

There are international airports in Bucharest, Cluj, Timisoara, and Sibiu. Bucharest Airport is the largest airport in Romania and is major international airport and transportation hub. Cluj, Timisoara, and Sibiu have frequent scheduled flights to Europe.

18.2 ACCESS

The Certej Gold Project is presently serviced by a two lane paved roadway (E76) from the city of Deva that passes through Certej Village. An alternate access route will be constructed for the Project that bypasses the congestion of Certej village, as discussed in Section 5.2.

18.3 POWER

The Certej Gold Project is presently serviced by the Mintia power station, located 14 km south-west of the project site, and run by the Romanian state. An existing 110 kV power line from Mintia feeds a substation located in the town of Certej, some 5 kilometres from the proposed plant site substation. A 20 kV line from this substation to the current Certej open pit currently exists. To meet the increased power demand, an additional 110kV overhead line will be run to the plantsite from the Baita Substation, located approximately 11 km north-west of site. Figure 18-3, Power Supply Single Line Diagram shows the concept for the Power Supply to the Project. Figure 18-3, Project Infrastructure shows the Proposed Power supply development for the Project.

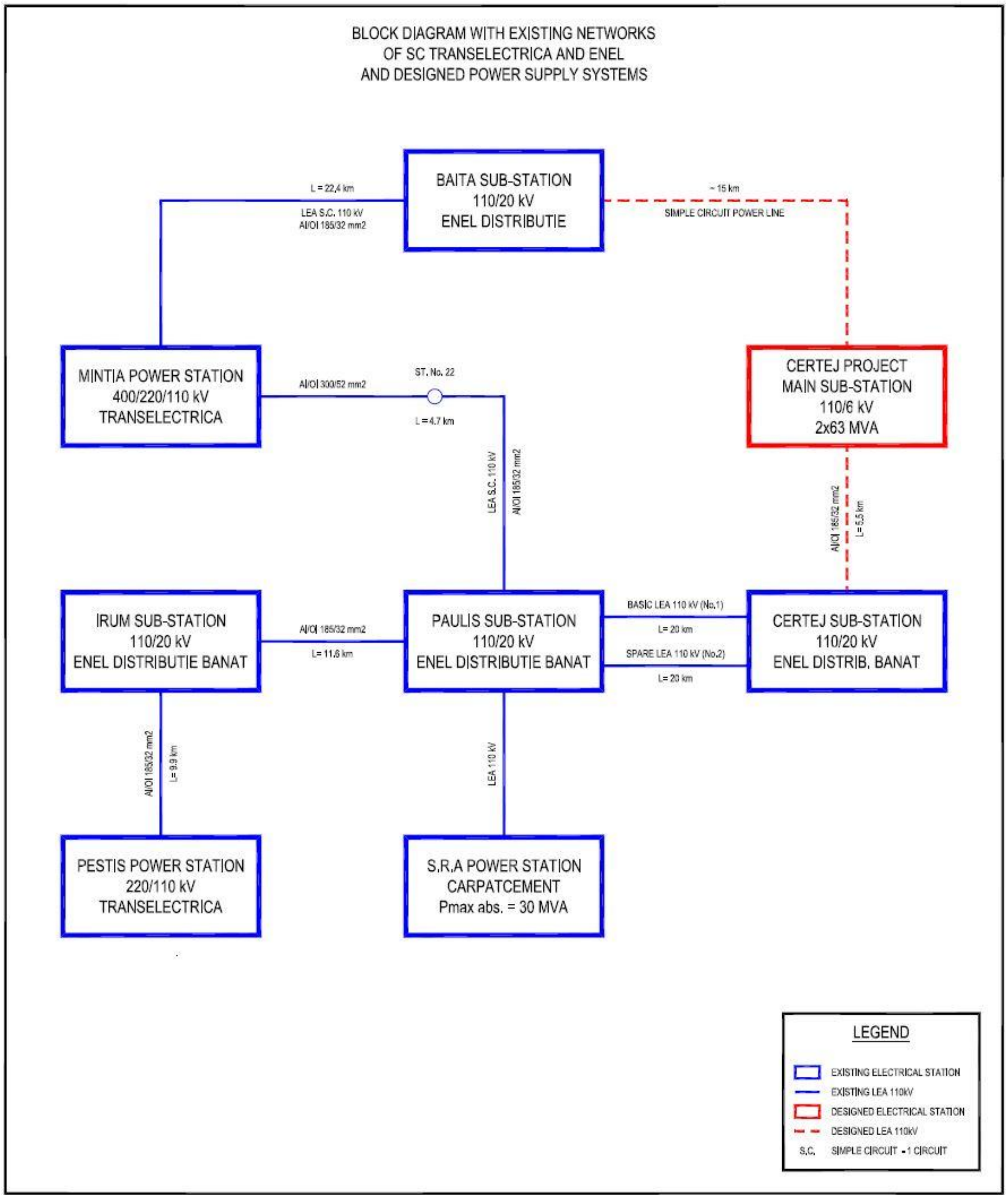


Figure 18-3: Power Block Diagram

18.4 FRESH WATER

Fresh water will be drawn from the Mures River, near Deva, approximately 10km from site. There is an existing Fresh Water Pipeline to Certej but as it is in a state of disrepair, it will be replaced with a new pipeline to site. A new fresh water pumphouse has been constructed at the Mures River for the Project, and an additional booster pumphouse will be constructed on the main supply line. Potable water is sourced from an underground aquifer near the current Certej open pit which was previously developed. Figure 18-4 shows the location of major utilities supply, discussed in Sections 18.3 and 18.4.

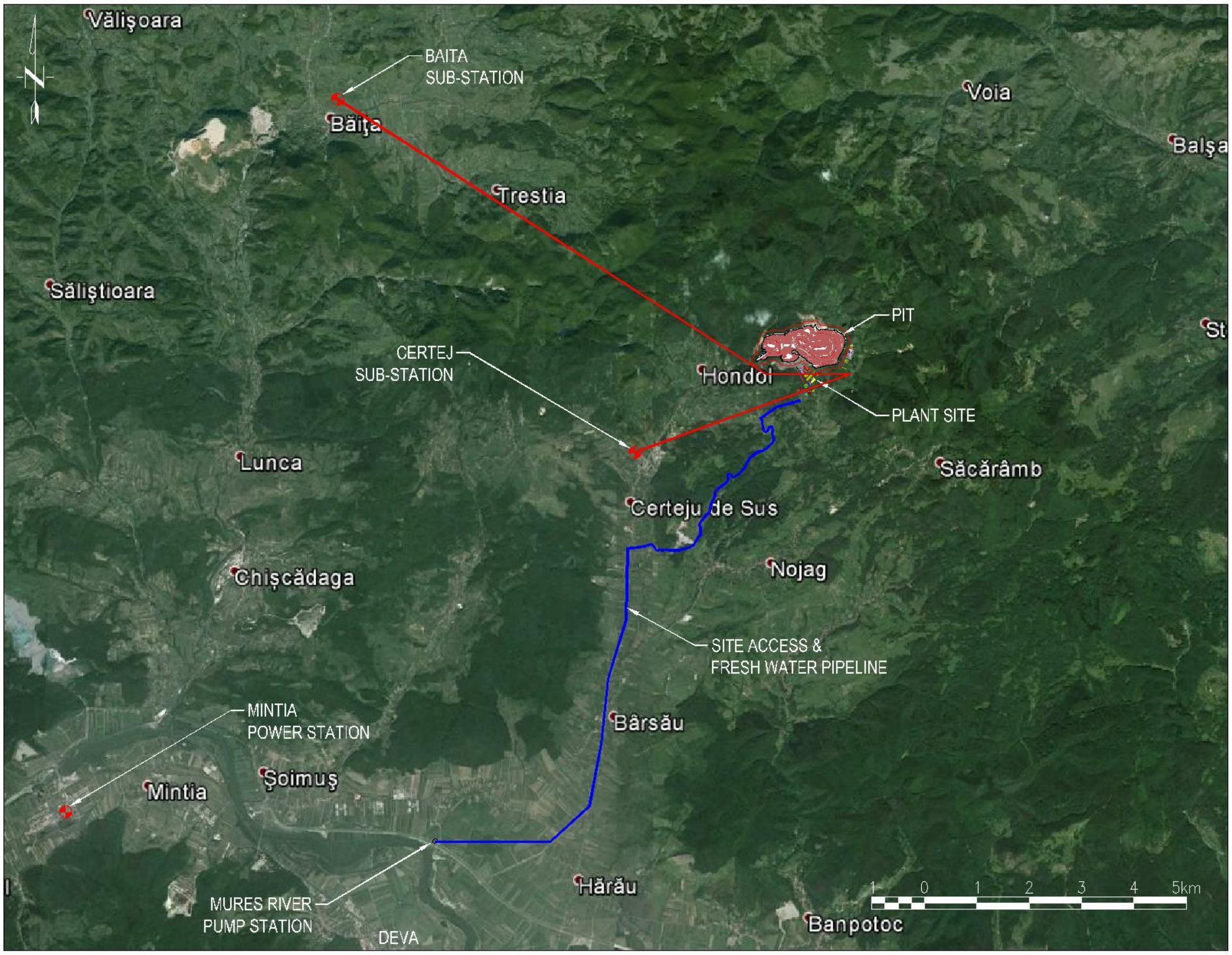


Figure 18-4: Utility Supply

18.5 CONSUMABLES

For the purpose of project construction, concrete can be supplied from a modern 90 cubic meter per hour winterized batch plant located in Deva and operated by Capat Benton. Capat Benton is a subsidiary of Heidelberg Cement, a major international cement company that operates 18 batch plants in Romania. Heidelberg operates 3 additional cement plants, one located 15km from Certej in Chiscadaga, and 14 aggregate plants in Romania.

The national oil company Petrom has a depot in Deva for diesel fuel and oil supplies which supplies the entire Hunedoara County. Bulk fuel and lubricants are available from the depot.

18.6 WASTE ROCK AND LOW GRADE ORE STOCKPILES

Waste rock will be mined from two years in advance of the start of production in order to provide materials for the initial bulk earthworks construction and begin the mine pre-stripping operations. Waste rock will be used to construct the North Tailings starter dam embankment and subsequent raises. Also commencing during this initial waste pre-stripping period, low grade ore will be stockpiled south of the open pit, for re-handling to the process plant in the later years of the mine life. Table 18-1 summarizes the Waste and Low-grade production and rock dump volumes for the Project life.

Table 18-1: Waste Rock and Low Grade Ore Stockpile Volume

	Unit	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Total
Waste Rock Disposal																				
Waste to North Dump	kt	3.52	12.25	13.31	12.98	12.98	11.31	10.35	10.42	7.65	3.47	1.93	2.37	0.51	-	-	-	-	-	105
Waste for TMF Construction	kt	0.23	0.23	0.16	0.16	0.16	0.16	0.16	0.16	0.16	1.80	1.80	1.80	1.80	-	-	-	-	-	9
Waste for Site Construction	kt	- 1.10	- 0.50	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0
Total Annual Waste	kt	4.85	12.97	13.47	13.15	13.14	11.47	10.51	10.58	7.81	5.27	3.73	4.17	2.31	-	-	-	-	-	113
Cumulative Waste to North Dump	kt	3.52	15.77	29.08	42.06	55.04	66.35	76.70	87.12	94.77	98.25	100.18	102.55	103.06						
Low Grade Ore Stockpiles	**Unit**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Stockpile Addition	kt	0.15	1.03	1.31	1.04	0.97	1.17	2.03	2.04	2.19	1.73	1.77	1.28	0.24						17
Stockpile Recovery	kt			0.78	0.18	0.11	0.14	0.05	0.13			-	0.44	1.12	3.00	3.00	3.00	3.00	1.98	17
Stockpile Closing Balance	kt	0.15	1.18	1.71	2.56	3.42	4.45	6.43	8.34	10.52	12.25	14.02	14.86	13.98	10.98	7.98	4.98	1.98	-	

The Waste Dump and Low Grade Stockpiles will be located in close proximity to the open pit to reduce haulage costs. The Proposed Waste Dumps are shown in the staged development plans included as Figure 18-5 thru Figure 18-9 at the end of this section.

Waste pre-stripping during the two year pre-production period as shown in Table 18-1 will also be used as bulk fill for construction of the plantsite, haul roads and site roads. The North Waste Dump construction will commence in Year -1.

Figure 18-6, Year 2, shows Waste Rock mined during operations continuing to be deposited to the North of the open pit with some andesite waste rock being diverted to tailings embankment construction in the early years. Low-grade ore will continue to be stockpiled to the south of the pit.

In Year 6, as shown in Figure 18-7, the North Tailings Management Facility starter embankment will be complete and waste will continue to be deposited on the North Waste Rock dump. Low grade ore storage to the North of the open pit will begin as the south low grade stockpile reaches capacity.

As shown on Figure 18-8, in Year 11, waste rock will begin to be stored to the east, inside the same watershed. In the next phase of the project, with refinement of the mine plan, there will be an opportunity to optimize the waste rock dump design to utilize in-pit storage.

Figure 18-9 shows the status of the waste rock dumps at the end of the project. Approximately 15 million tonnes of low grade ore is to be stockpiled thru the life of mine to be reclaimed at the beginning in year 12 for processing.

18.7 HISTORIC WASTE DUMPS

Historic waste dumps will be relocated where required to accommodate construction of the Process Plant and Low Grade Stockpiles. The balance of the waste dumps will be graded to form a platform for the ROM stockpile and low grade ore storage.

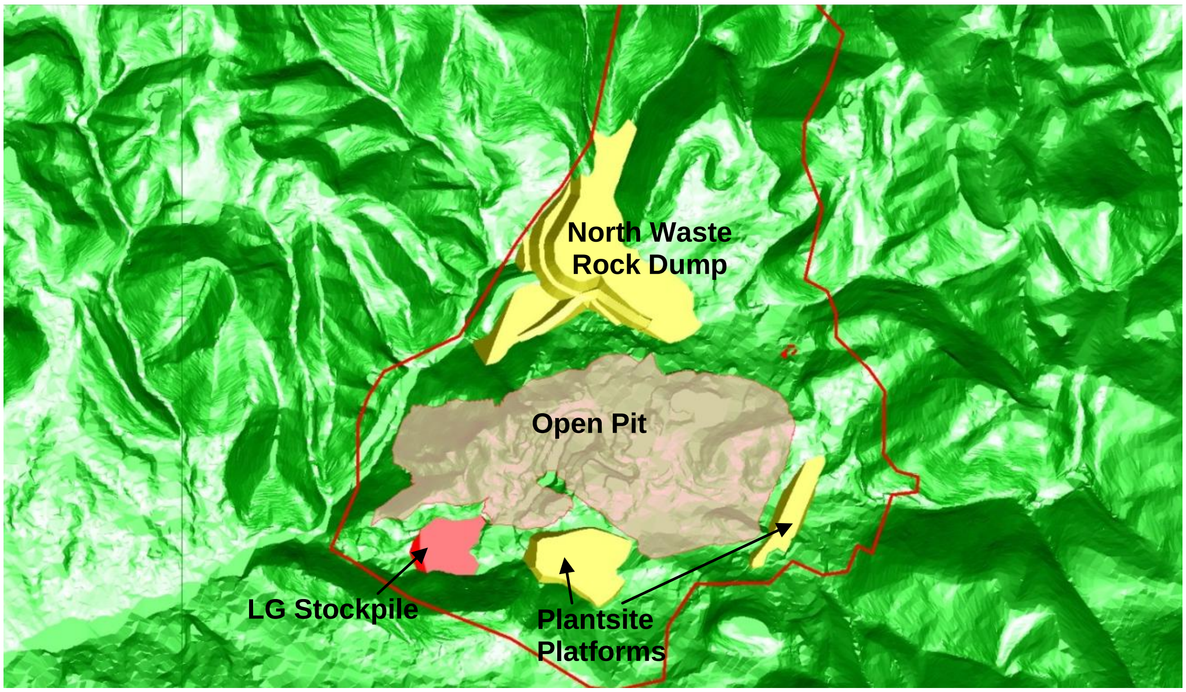


Figure 18-5: Preproduction Waste and Low Grade

Figure 18-6: Year Two - Waste and Low Grade Stockpile

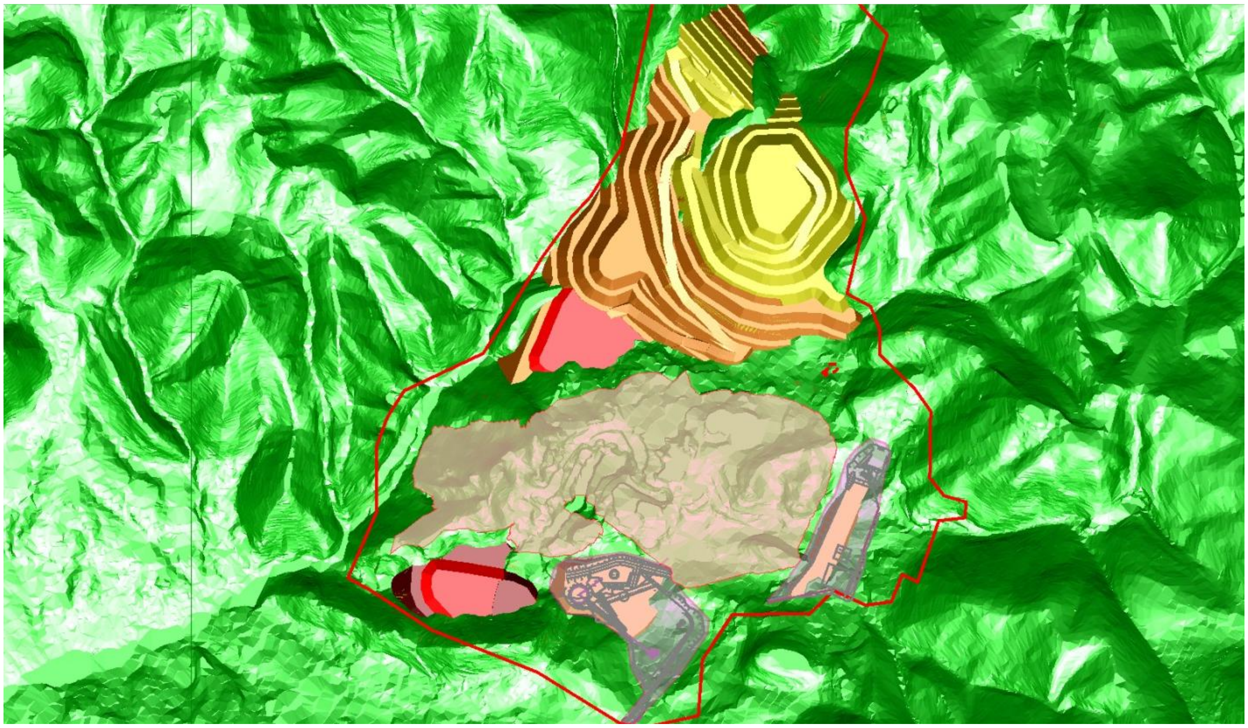


Figure 18-7: Year Six - Waste and Low Grade Stockpile

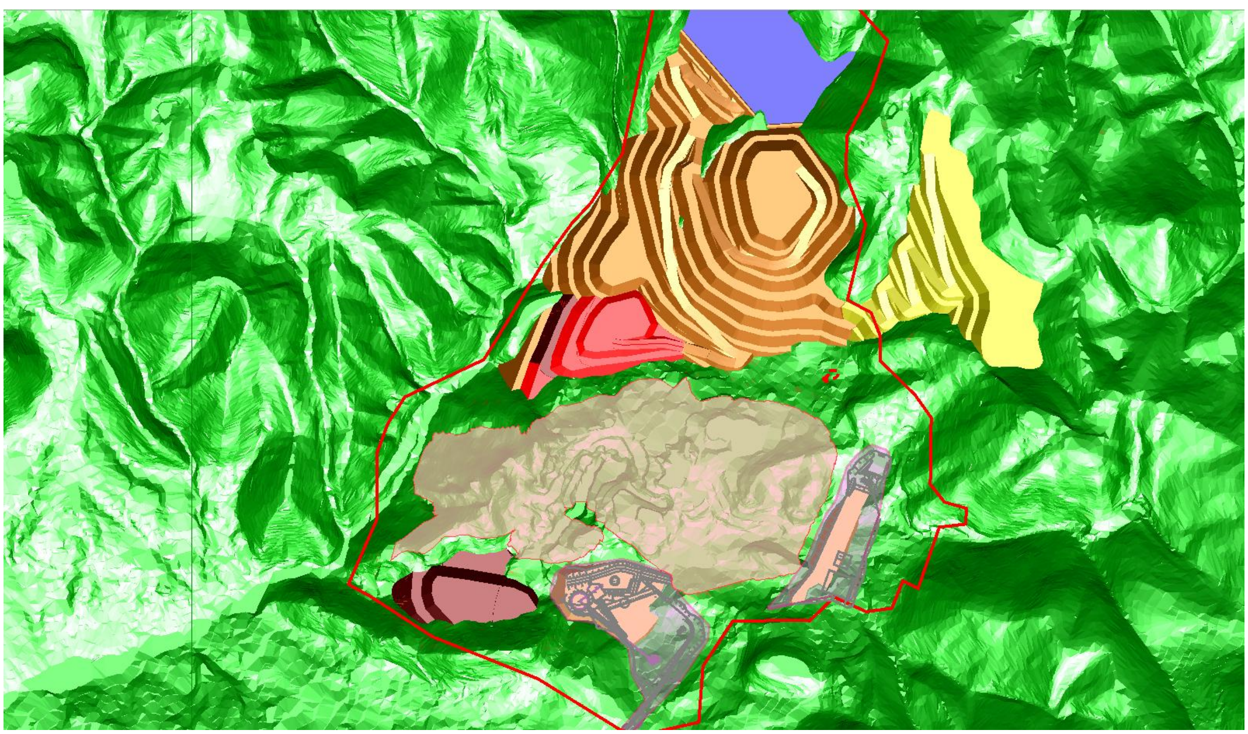


Figure 18-8: Year 11 - Waste and Low Grade Stockpile

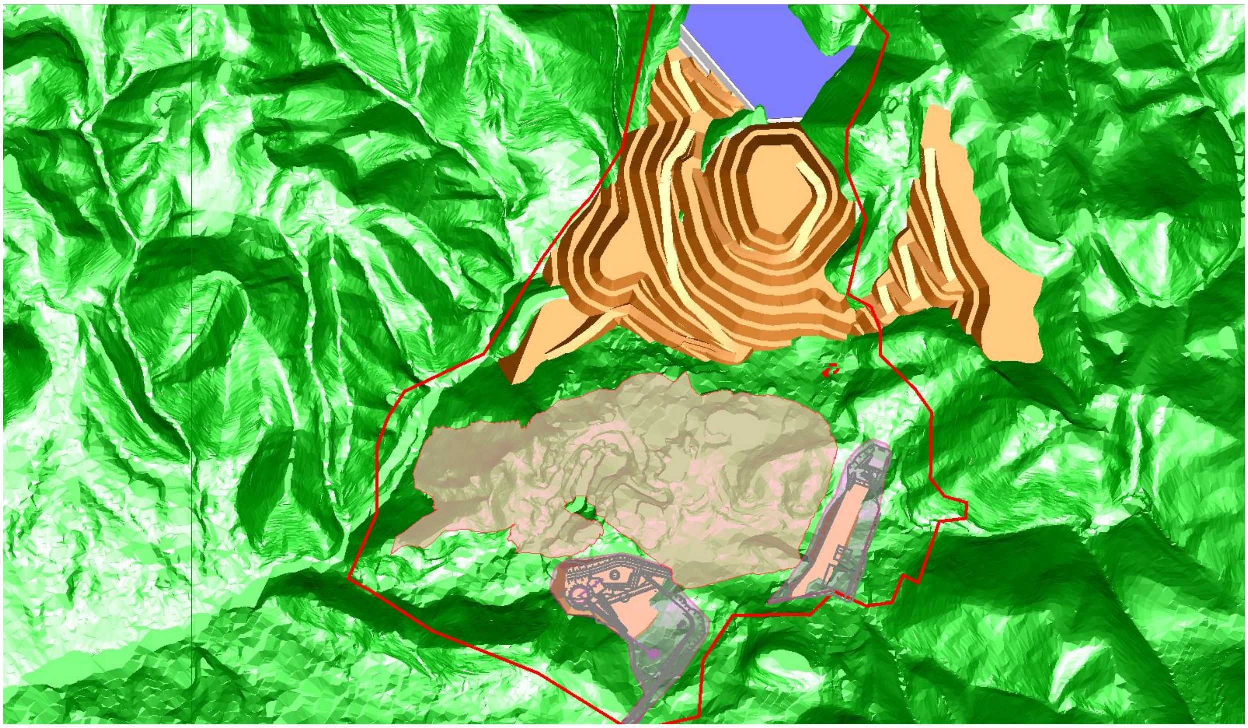


Figure 18-9: End of Mine Life - Waste and Low Grade Stockpile

18.8 TAILINGS MANAGEMENT FACILITIES (TMFS)

For the life of the Project, there will be 56 million cubic meters of tailings produced 30 million cubic meters of Flotation Tailings and 26 million cubic meters of CIL/Neutralization Residue Tailings. Tailings will be pumped from the process plant to the TMF's by conventional pipeline. Several locations for construction of tailings management facilities have been identified and are under various stages of investigation and design development.

The tailings facility in the Bosca Mare valley located immediately north of the minesite is within the watershed of the project and has a total storage capacity of 35 million cubic meters of tailings. The Bosca Mare tailings facility is well defined and has had a high level of site investigation and design completed by the local consultant Cepromin and the National Technical University of Bucharest (NTUB). Climatology, geotechnical, geochemistry, hydrology, hydrogeology, and seismicity have been taken into account in the design of the TMFs.

There is only room for a marginal increase in storage capacity of the Bosca Mare tailings management facility, resulting in the remaining 21 million cubic meters of tailings requiring storage in an alternate TMF. Various options for additional tailings facilities to the west of the mine site have been investigated. The cost to construct tailings storage facilities in these narrow valleys located west of the project site was excessive, as three sites were required to store a relatively small amount of tailings; hence, these sites were rejected. Deva Gold is evaluating several sites outside of the watershed that can contain in excess of the requirement, in a cost effective manner. Field work programs are underway to support engineering of these sites with geotechnical investigations and design scheduled to be completed later in 2014.

The addition of the neutralization residue will result in lower consolidation rate that impacts the utilization of centerline or downstream raises. Initial investigations have indicated there is a potential to improve the consolidation characteristics utilizing co-disposal of the CIL/Neutralization Tailings with a portion of the flotation tailings. A testwork program is being developed to investigate this further in the next phase of the Project.

The flotation tailings will be inert, apart from low levels of residual sulphides not recovered during the flotation process. The CIL tailings will be treated in a detoxification circuit to lower the levels of residual cyanide to below legislated values. A water treatment plant have been included in the study to allow treatment for the process and discharge later if required.

18.9 EMBANKMENT CONSTRUCTION

Initial stages of embankments construction will be for a starter dam. The embankments will subsequently be raised to their final elevation by lifts throughout the life of the project. Figure 18-10 shows typical TMF construction details.

The Bosca Mare TMF will be constructed from Run of Mine Waste Rock and any facilities remote from the site will be constructed of quarried material. Special features such as filters and erosion protection layer will be constructed using specified selected materials.

A 2mm thick HDPE liner will be installed initially on the starter dams to minimise seepage through the upstream face of the Embankments. A tailings beach will form over the embankment face allow the tailings beach to be formed against the embankment, to minimise seepage on future lifts. The HDPE liner will be anchored on the starter dam crest, valley sides and floor, and lie against the fine filter material.

For TMF Main Embankments, a double filter (well graded sand and well graded sand and gravel) will be installed against the rock fill. This arrangement will be carried through for the full height of the starter dam and subsequently for all the raises. The filters will have each a minimum thickness of 1.5m.

Beneath every raise for the Flotation and CIL TMF embankments, a layer of heavy duty geotextile will be placed on top of the tailings and will extend horizontally to the downstream side of the raise.

Seepage will be collected in a seepage collection sump and re-circulated via pumps back to the TMFs.

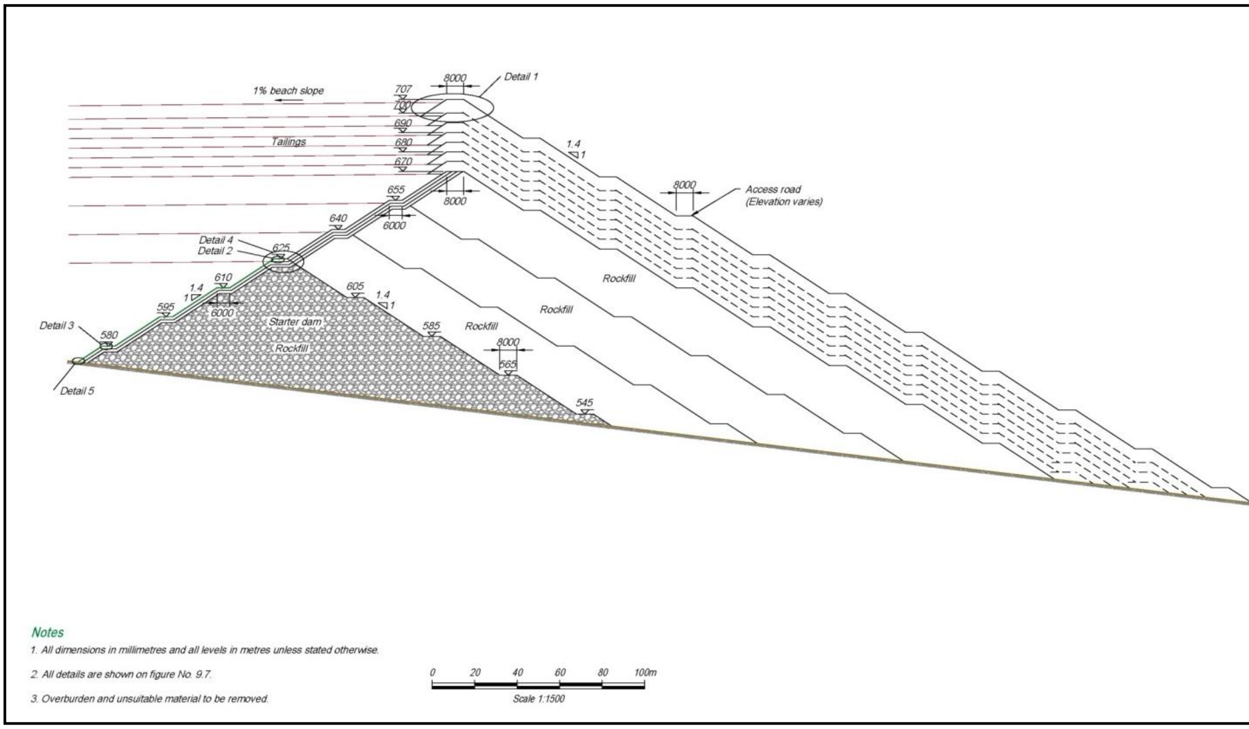


Figure 18-10: Typical TMF Construction Details

18.10 WATER MANAGEMENT

18.10.1 Non–Contact Water

A comprehensive water diversion system consisting of drainage galleries and diversion channels will be constructed around the site to reduce the entrance of clean surface water runoff from the external catchment area to the contact water area. This water may be utilized in the process to minimize Mures River water usage. Clean run off water not used in the process will be diverted around the site and discharged into the Valley Macrisului creek downstream of the site which will flow into the Certej creek and then to the Mures river. This water will not require treatment.

18.10.2 Contact Water

Water that will require treatment includes discharge from the Tailings Management Facility, Site Contact Water from the Plantsite, and Open Pit and Waste Rock runoff and runoff water from the historic mining activities including the existing underground adits and runoff from the existing waste rock dumps. Water is expected to contain elevated concentrations of metals, have a low pH, and contain elevated sulphur. Contact water will be collected and is amenable to be used as make up water in the process with minimal treatment for suspended solids.

18.10.3 Tailings

Water used for the process will exit the plant as part of the tailings stream and be pumped to the tailings storage facility. From screening level analysis, the TMF water is expected to contain high concentrations of metals, calcium and sulphate. Water released to the environment would not require chloride treatment as it is below the Romanian discharge regulations. However, if tailings water is used as recycle water, cycle-up of chloride concentrations may require treatment as high chloride levels would affect gold recovery.

18.10.4 Water Treatment

The treatment process selected for the pre-feasibility study is based on results of geochemical testing of the waste rock, water quality monitoring and process water quality testing. A water balance was prepared to determine the water treatment plant capacity. Treatment will be a combination of chemical precipitation using lime addition and membrane filtration. Lime treatment will be used for precipitation of metals. Solid and Calcium Sulphate removal will be accomplished by microfiltration and nanofiltration respectively, with final eduction of calcium and sulphate to below Romanian discharge criteria. The treatment process selected was chosen to meet the stringent discharge limits of the calcium and sulphate and to keep the process robust and use only proven technologies. Additional treatment capacity can be added by adding additional treatment trains. The solids stream will be dewatered and non-hazardous; determined to be a very small volume, it can be disposed of in the TMF.

SECTION • 19 MARKET STUDIES AND CONTRACTS

19.1 MARKETS

A 2010 study was undertaken for European Goldfields by GFMS Mine Economics, to identify gold refineries, which might provide the best commercial terms to refine doré produced by the Certej project. Nine gold refineries were approached to provide indicative pricing for refinement of Certej doré.

Key findings included:

- The market for gold doré is relatively stable.
- Refiners have specific preferences regarding the gold:silver ratio in doré and differing levels of tolerance for impurity elements; these concerns will be directly reflected in the terms offered.
- Increases in energy costs have a significant impact on refining costs.
- There is a risk that doré transport costs will rise, due to higher aviation fuel and insurance costs.

The study also proposed that there is no generally accepted standard format for refining contracts, however, the main components of the contract are usually comprised of:

- **Assay and handling charges**: usually a charge per lot for assaying, weighing, handling etc. A maximum lot size or value may apply. May also include customs charges.
- **Treatment and refining charges:** usually quoted in terms of $/oz, €/kg, etc. These may be quoted as a single charge, or as separate treatment and refining components.
- **Penalites:** charges imposed for specific impurities. The contract will specify permissible maximum levels of delirious elements, and may set out a scale of charges for impurities in excess of these limits.
- **Payability, or Return Rate**: specifies the percentage of gold and silver contained in the doré for which the miner will be credited.
- **Outturn**: the time period between receipt of doré by the refiner, and payment to the miner.

The costs carried for transport and refining in the economic analysis of the project are quite conservative, relative to the indicative quotations obtained in the 2010 study. These market studies have not yet been updated to reflect the change in extraction process, or recent metal and energy prices. These studies and analyses will be updated as the project progresses into a Feasibility Study.

19.2 MAJOR CONTRACTS & SUPPLY AGREEMENTS

No contracts or formal agreements are currently in place, with a feasibility study still to be completed; however, for input to the Pre-Feasibility cost estimates and for general understanding of supply arrangements, the following preliminary discussions were undertaken:

- **Power Supply** – Discussions with the Romanian authorities were undertaken to confirm availability of power supply and come to agreement on additional infrastructure requirements. A review of power market conditions and influential factors was completed internally, and multiple quotations for power supply were obtained from Romanian energy producers. No contracts or supply agreements have been formalized.

- **Water Supply** – As part of the permitting processes, approval for fresh water supply from the Mures River was received, and an understanding of applicable tariffs was confirmed. No formal agreements are in currently in place.

- **Fuel Supply** – Preliminary discussions were initiated with various domestic suppliers to confirm availability of supply and gain a general understanding of supply conditions. No formal agreements have been made.

- **Oxygen Supply** – For input to the capital and operating cost estimates, discussions were undertaken with potential 'over-the-fence' oxygen suppliers. Preliminary supply requirements were discussed, budget quotations were obtained, but no agreements or contracts have been formalized to date.

SECTION • 20 ENVIRONMENTAL STUDIES, PERMITTNG AND SOCIAL OR COMMUNITY IMPACT

20.1 EXISTING CONDITIONS

The historical workings discussed in Section 6 'History' have resulted in pre-existing conditions on the project site and in the surrounding areas.

As discussed in Section 4, under the terms of the mining license, the project is liable for future environmental damage arising from the project. The project is not responsible for any previously incurred environmental damage within or outside of the project area. In accordance with the Romanian government ordinance number 644/2007 regarding the shutdown of Minvest operations, the associated environmental rehabilitation will be funded by the Ministry of Finance.

20.2 ENVIRONMENTAL STUDIES

A summary of key environmental studies completed in relation to the project site is included in Table 20-1.

Environmental studies and findings will be updated in the next stages of project development to reflect the most up-to-date project parameters and to ensure compliance with Romanian laws and regulations and with European directives the field of the environmental protection.

Of significance, an Environmental Impact Statement (EIS) was produced in 2007 in accordance with the provisions of the Order of the Ministry of Environment and Water Administration No. 863/2002, by a consortium of Romanian agencies and institutes. These studies noted the following significant findings:

- **Water** – Treatment of acid rock drainage would improve the quality of water in the local streams and consequently Mures River water quality. Water discharged from the TMFs to the local river system would be treated and comply with the admissible values laid down in environmental regulations.
- **Air** – An assessment of all airborne pollutions and dispersion patterns during the project life determined that specific management actions would keep the air quality well within admissible levels.
- **Soil** – Existing pollutants are either due to historical activities or due to the natural dispersion of sub-soil mineralistion. As such, project activities will not increase the pollution level but will improve it as mitigating methods are applied.
- **Biodiversity** – It was found that there were minimal risks to flora, fauna and animal life, provided appropriate reclamation measures were undertaken.
- **Social Impact** – The project will impact positively on the communities because of population stabilisation and the economic benefits.

- **Archaeology** – Fifty two archaeological trenches were excavated by the Deva Museum and no pre-historical vestiges have been found in the mine site area.
- **Risks and Hazards** – The use of cyanide, explosives and the TMFs posed the greatest risks but these risks are minimal due to available mitigating measures.

Table 20-1: Environmental Studies

Description	Date Published	Author(s)
Technical memorandum	2009	S.C. CEPROMIN S.A – Mining Designing And Research Institute
Report to the environmental impact assessment study	2010	« Babeş Bolyai » University of Cluj Napoca- Center for Disaster Management Research
Potential trans- boundary impact	2010	« Babeş Bolyai » University of Cluj Napoca- Center for Disaster Management Research
Assessment of the Cumulated Impact of Rosia Montana and Certej Projects and the consequences of a simultaneous accident with possible trans-boundary impacts	2010	AMEC – S.C. *AMEC Earth& Envionmetal S.R.L, România* UTCB – *Technical University of Constructions Bucharest* UBB –*Babes – Bolyai University of , Cluj Napoca*, OCON ECORISC – *S.C. OCON ECORISC S.R.L, Turda* MARILENA PATRASCU – *expert for Environmental Impact Assessment* PAUL WHITEHEAD – *Profesor at Oxford University,* WESTAGEM – *S.C. WESTAGEM S.R.L., Bucuresti*
Safety Report	2012, 2013	S.C. OCON ECORISC S.R.L. Turda (original and revised)
Internal Emergency Plan	2012, 2013	S.C. OCON ECORISC S.R.L. Turda (original and revised)
Technical Design for the environment rehabilitation	2010, 2011	S.C CEPROMIN S.A. Mining Designing And Research Institute (original and revised)
Environmental monitoring programme during the operating and post closure stage	2010 2011	S.C CEPROMIN S.A. Mining Designing And Research Institute (original and revised)

Description	Date Published	Author(s)
Plan of Environmental Rehabilitation	2010, 2011	S.C CEPROMIN S.A. Mining Designing And Research Institute (original and revised)
Initial Plan of Mine Closure	2010, 2011	S.C CEPROMIN S.A. Mining Designing And Research Institute (original and revised)
Waste Management Plan	2012	S.C CEPROMIN S.A. Mining Designing And Research Institute
Plan for the prevention and fight againts Accidental Pollution	2010	S.C. OCON ECORISC S.R.L. Turda
Risk Study for the dams of the tailings management facilities of the gold –silver ore mining of Certej perimeter	2010	Technical University of Constructions Bucharest
Report of the auditing/ approval of the technical documentation concerning the " Dams of the gold-silver ore mining of Certej perimeter, Hunedoar county”"	2010	Alexandru CONSTANTINESCU Expert for dam safety
Adequate assessment study (Biodiversity study)	2013	Wildlife Management Consulting S.R.L. Hunedoara, Romania
Technical water memorandum	2012, 2013	S.C CEPROMIN S.A. Mining Designing And Research Institute (original and revised)

20.3 WATER MANAGEMENT

Existing streams on and around the project site, which collect discharge and seepage water from previous workings, indicate a strong historical pollution caused by hundreds of years of mining operations. Where these flows will be directly impacted by the project, treatment or re-use will reduce impacts on surrounding areas.

Surface water and groundwater at the project site can be classified as ‘non-contact’ or ‘contact’. ‘Non-contact’ water is represented by surface runoff and groundwater seeps that are not impacted by the project operations; conversely, ‘Contact’ water is represented by surface runoff and seepage flows generated from or coming into contact with mining facilities such as waste rock dumps, open pit, TMFs, or processing facilities. Implementation of a detailed Water Management Plan will consider four primary mitigation measures to minimize project impacts:

- **Diversion** – Water will be intercepted and diverted to the maximum extent possible before coming into contact with mining facilities or activities.

- **Collection** – Contact water will be collected in storage facilities for treatment or re-use.
- **Recycling** – A significant amount of process and contact water will be recycled through the process, to reduce discharge and minimize introduction of fresh water.
- **Treatment & Discharge** – Prior to discharge, all water will be treated to meet regulatory water quality requirements, and will be discharged into natural water courses.

20.4 WASTE MANAGEMENT

20.4.1 Mine Waste Rock

Mine waste resulting from stripping and open pit operations will be deposited on Waste Rock Dumps. The selection of the waste dump sites and the design works for the deposit of the waste rock take into account use of best available technologies, best practices, and compliance with regulatory directives and legislation, including :

- stability of the rock layer beneath the site foundation;
- lower permeability of the basement;
- low transportation distance from the mine site;
- good opportunities to use the material in the future;
- geometry of the land / slope geometry;
- inner drainage and drainage around the dump to minimize contact water; and
- diversion of surface water courses that would impact the waste dump.

Dump construction consists of waste unloading, waste spreading, and levelling and consolidation of waste.

20.4.2 Process Tailings

Slurried process tailings transported by pipeline and deposited in Tailings Management Facilities. TMF sites are selected and designed with consideration of:

- Site selection based on geotechnical and geological studies, review of hydrogeological conditions, hydrological conditions site seismicity, zone restrictions, of the distance to the processing plant, of the environmental impact and local community location;
- Embankment construction with rockfill (non-degradable andesites).
- Hydrogeological tests to confirm that the rock mass fracturing and permeability
- Construction of starter dams with two filtering layers; coarse filtering layer of broken stone and fine filter of sand and gravel and geomembrane and geotextile impermeabilization.
- Protection above the starter dam level of the upstream slope with filtering layers to prevent the slurry flowing through the rockfill.
- For vertical dam rises, use of a separation geotextile layer between the rises.

20.5 PERMITTING

Permitting requirements and status are discussed in detail in Section 4 'Property Description and Location'.

An environmental permit for approval of the PUZ by the local authorities was received in 2010.

The EIA and EIS were submitted in 2010 and finalized through the EIA Procedure until 2012; the project was issued an Environmental Permit in July 2012.

This permit was revised in 2013 to include modifications to the original design and EA procedure.

In the next stages of the project, revisions to the environmental permit will be requested as necessary to accommodate progressive design development.

20.6 CONSIDERATION OF SOCIAL & COMMUNITY IMPACTS

As the mining operations have stopped or slowed in the local area, social, economic and cultural activities have slowed down and are in regression. The Certej mining project will have a positive impact on the community and social environment in several areas:

- Infrastructure improvements and upgrades including roads and power, availability of potable water supply and sewage treatment facilities after mine closure.
- Stabilisation and diversification of the population, through creation of new jobs for the indigenous population and migration of new workers.
- Increased family income allowing the houses of the localities and neighbouring areas to be modernized and the building of newer more comfortable dwellings with an architectural style specific for the zone.
- Economic revival of the zone will prompt development of other activities.
- Engagement of local communities is on-going through both formal and informal meetings with the mayoralty and local residents in the area

As project development continues, the company is committed to develop its activities in a transparent manner and to engage the communities in the development of the project.

20.7 MINE CLOSURE

Mine closure activities will primarily consist of:

- **Underground Access** – Closure of disturbed access to previously existing underground workings.
- **Plantsite Decommissioning** – Demolition of structures, removal of equipment foundations, backfilling of voids, clearing of site rubble and total removal of unacceptable materials, land levelling, and recovery of as many re-usable and saleable materials and elements as possible.

- **Infrastructure Decommissioning** – Decommissioning of infrastructure, including: dismantling outer electrical grids, dismantling of tailings pipelines, dismantling of industrial water networks. Some infrastructure may remain for the benefit of the local population.
- **Tailings Facilities Decommissioning** – Surface slope construction, impermeabilization of the facility surface by geomembrane layer placement.
- **Overall Site Rehabilitation** – For the open pit, waste rock dumps, tailings facilities, and other disturbed footprints: Backfilling, removal of oversized surfaces, and levelling or consolidation of slopes. Laying of vegetal soil, grass seeding and reforestation with vegetation specific to the local area. Implementation of a monitoring system.

SECTION • 21 CAPITAL AND OPERATING COST

For all cost estimates, currency exchange rates used are as per Q1 2014 market conditions. All costs are presented in US Dollars (USD).

21.1 CAPITAL COSTS

The total project capital cost estimate is inclusive of the initial, sustaining and closure costs, shown in Table 21-1.

The accuracy of the cost estimates are consistent with the standards outlined by the Association for the Advancement of Cost Engineering (AACE). The Initial Cost Estimate is a pre-feasibility estimate categorized as AACE Class 4. Both Sustaining and Closure Cost Estimates are scoping-study estimates consistent with AACE Class 5.

Direct costs were developed from a combination of budget quotes, material takeoffs, project specific references and historical benchmarks. Indirect and owners' costs were estimated using an amalgamation of calculated project requirements supplemented with historical benchmarks. Finally, contingency for each estimate was applied based on the level of engineering definition. The overall project contingencies for Initial, Sustaining and Closure Capital Cost Estimates are 17%, 24%, 50% respectively.

Table 21-1: Capital Cost Summary

Area	Description	Initial [US$ x 1000]	Sustaining [1] [US$ x 1000]	Closure [2] [US$ x 1000]
A	Overall Site	15,362	7,618	11,817
B	Mine	84,402	64,111	-
C	Crushing	12,157	-	-
D	Process Plant (Concentrator)	59,031	8,000	-
E	POX	99,750	21,000	-
F	CIL	17,376	-	-
G	Tailings	23,586	73,803	-
H	Infrastructure	10,206	1,170	-
J	Ancillary Facilities	7,166	-	-
K	Off Site Infrastructure	30,501	4,000	-
	Direct	**359,537**	**179,701**	-
	Indirects	**88,374**	**15,228**	-
	Owners Cost	**11,960**	**-**	-
	Contingency	**79,276**	**45,873**	**5,908**
	Total Installed Cost	**539,147**	**240,802**	**17,725**

[1] Eleven years of mining and sixteen years of ore processing.
[2] Continuous closure plan starting in year 5.

21.2 OPERATING COSTS

The Operating Cost Estimate was qualitatively assessed to be within a Pre-Feasibility accuracy level (AACE Class 4).

Operating costs include allocations for:

- Open pit mining,
- Processing, and;
- General & Administration

Operating costs were calculated for each year of operation, totalling US$1.5B for an average of US$31.86/t ore milled over life-of-mine, summarized in Table 21-2:

Table 21-2: Operating Costs

Operating Costs	Units	LOM Average [US$]	LOM Expenditure [US$M]
Mining Cost	US$/t mined	1.87	
Mining Cost (ore)	US$/t ore	6.40	301
Processing Cost	US$/t ore	19.54	918
G&A	US$/t ore	2.79	131
Operating Cost	**US$/t ore**	**28.73**	**1,350**
Oxygen Plant Lease	US$/t ore	3.13	147
Total Operating Cost	**US$/t ore**	**31.86**	**1,497**

No contingency was carried on operating cost estimates.

Mining costs were estimated from first principles by unit operation, based on projected fleet requirements for an annual production schedule, historical benchmarks, and associated operation and maintenance labour requirements. Fuel consumption parameters for the major equipment were estimated from manufacturers' data and from experience with similar operating environments and costs were based on a budget quotation for bulk fuel supply.

Processing costs were based on estimated annual consumption of process reagents, major wear parts, and utilities. Budget quotations were obtained for supply of all significant consumables and utilities. Power cost was estimated based on current and projected Romanian market conditions; consumption was calculated based on preliminary electrical load lists.

General and Administrative costs were estimated based on a projected personnel list with salaries indicative of local standards, and annual allowances for general supplies.

SECTION • 22 ECONOMIC ANALYSIS

An economic analysis was prepared in support of the declaration of mineral reserves, to confirm that the mineral reserves could repay life-of-mine capital and operating costs, and generate a positive return. The project was evaluated on an after-tax discounted cash flow basis.

22.1 CASH FLOW & ANNUAL PRODUCTION

For the purpose of cash flow forecasts and economic analysis:

- Gold and silver prices were assumed to be US$1250/oz and US$16.50/oz, respectively, based on the mineral reserves declaration.
- Royalties were calculated at an annual rate of 6% applicable to net sales revenue less transport and refining costs.
- Taxes were considered as payable on earnings before interest, at an annual rate of 16%.
- A cost of US$9.50/oz was assumed for transport and refinement of doré, based on historical project data.
- All costs are presented in US Dollars (US$).
- Currency exchange rates used are as per Q1 2014 market conditions, as provided in Table 22-1:

Table 22-1: Currency Exchange Rates

Currency Code	Currency Name	Exchange Rate
CAD	Canadian Dollar	1.00 CAD = 1.00 USD
EUR	Euro	1.00 EUR = 1.30 USD
RON	Romanian Leu	1.00 RON = 0.30 USD

Cash flow forecasts on an annual basis and an annual production schedule are provided in Table 22-2.

Table 22-2: Cash Flow Forecasts

			YEAR																	
		TOTAL	AVG	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
MINING																					
Ore Mined	kt	46,985	-	153	1,026	3,530	3,853	3,860	4,022	4,980	4,910	5,186	4,727	4,771	3,848	2,119	-	-	-	-	-
Waste Mined	kt	113,444	-	4,848	12,974	13,470	13,146	13,139	11,470	10,513	10,583	7,814	5,273	3,729	4,174	2,310	-	-	-	-	-
Strip Ratio	w:o	-	2.41	31.73	12.64	3.82	3.41	3.40	2.85	2.11	2.16	1.51	1.12	0.78	1.08	1.09	-	-	-	-	-
Total Material Mined	kt	160,429	-	5,001	14,000	17,000	17,000	17,000	15,493	15,493	15,493	13,000	10,000	8,500	8,022	4,429	-	-	-	-	-
PRODUCTION																					
Ore Processed	kt	46,985	2,937	2,937	-	-	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,008	3,000	3,000	3,000	3,000	3,000
Gold Grade	g/t	-	1.63	1.63	-	-	1.84	1.80	2.09	2.08	1.97	1.82	1.69	1.95	1.93	2.05	1.89	1.00	1.00	0.96	0.85
Silver Grade	g/t	-	10.76	10.76	-	-	7.30	8.51	12.50	13.79	12.36	11.45	9.35	10.25	11.31	12.99	15.14	9.47	9.47	9.43	9.10
Overall Gold Recovery	%	-	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%
Overall Silver Recovery	%	-	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Total Gold Production	koz	2,148	134	-	-	155	152	176	175	166	153	143	165	163	173	159	84	84	81	72	47
Total Silver Production	koz	12,823	801	-	-	556	648	951	1,050	941	871	711	780	861	991	1,152	721	721	718	693	457
CASH FLOW																					
Revenue	MUSD	2,902	-	-	-	203	201	236	236	224	206	190	219	218	234	219	117	117	113	101	67
Earnings	MUSD	408	-	-	-12	35	28	52	51	27	13	-7	21	15	30	43	27	28	25	16	16
After-Tax Cash Flow	MUSD	333	-	-270	-270	73	58	89	93	72	70	36	76	85	93	90	14	14	10	1	1
Cumulative After-tax Cash Flow	MUSD	-	-	-270	-539	-467	-409	-320	-227	-155	-85	-49	27	113	205	296	310	324	334	334	333

22.2 DISCOUNTED CASH FLOW ANALYSIS

The following Table 22-3 shows the summary of a discounted cash flow analysis for the base case scenario.

Table 22-3: Summary of Discounted Cash Flow Analysis

Discounted Cash Flow Analysis					
Assumed Gold Price	$/oz	1250	**Equivalent Costs**		
Assumed Silver Price	$/oz	16.50	Operating	$/oz	628
Ore Mined & Processed	kt	46,985	Oxygen Plant	$/oz	68
Waste Mined	kt	113,444	Silver Credit	$/oz	-100
Gold Grade	g/t	1.63	Transport & Refining	$/oz	10
Silver Grade	g/t	11	Royalty	$/oz	80
			Sustaining Capital	$/oz	120
Production			Development Capital	$/oz	251
Mine Life	**yrs**	**16**	All-In Cost	$/oz	1,057
Throughput	Mtpa	3			
Overall Gold Recovery	%	87.4%	**Cash Flow, LOM**		
Overall Silver Recovery	%	80.0%	Revenue	MUSD	2,902
Total Gold Production	koz	2,150	EBITDA	MUSD	1,212
Total Silver Production	koz	12,998	Earnings	MUSD	408
			After-Tax Earnings	MUSD	333
Costs					
Mining Cost (mined)	$/t mined	1.87	**Economic Analysis**		
Mining Cost (ore)	$/t ore	6.40	NPV (5%)	$mln	$84
Processing Cost	$/t ore	19.54	IRR	%	7.6%
G&A Cost	$/t ore	2.79			
Operating Cost	**$/t ore**	**28.73**	Payback Period	yrs	7.6
Oxygen Plant Lease	$/t ore	3.13	NPV/Capex	x	0.16x
Total Operating Cost	**$/t ore**	**31.86**			
Transport & Refining	$/oz	9.50			
Royalty	%	6.0%			
Investment Capital	$mln	539			
Sustaining Capital	$mln	259			

Results of this analysis indicate positive project economics until the end of mine life, and a positive return on investment with a Net Present Value of US$84M and an estimated Internal Rate of Return of 7.6%, paid back within 8 years.

22.3 SENSITIVITY ANALYSIS

Sensitivity analysis was conducted to understand the impact of metals price and project costs on the financial outcome.

For the purpose of the sensitivity analysis, silver price is assumed to increase or decrease proportionately with gold price. Figure 22-1 illustrates the sensitivity of the project considering a discount rate of 5%, based on the values shown in Table 22-4.

Table 22-4: Gold Price (US$/oz)

	Gold Price (US$/oz)			
	1250	**1350**	**1450**	**1550**
NPV (US$M)	84	264	444	624
IRR (%)	8%	12%	16%	20%

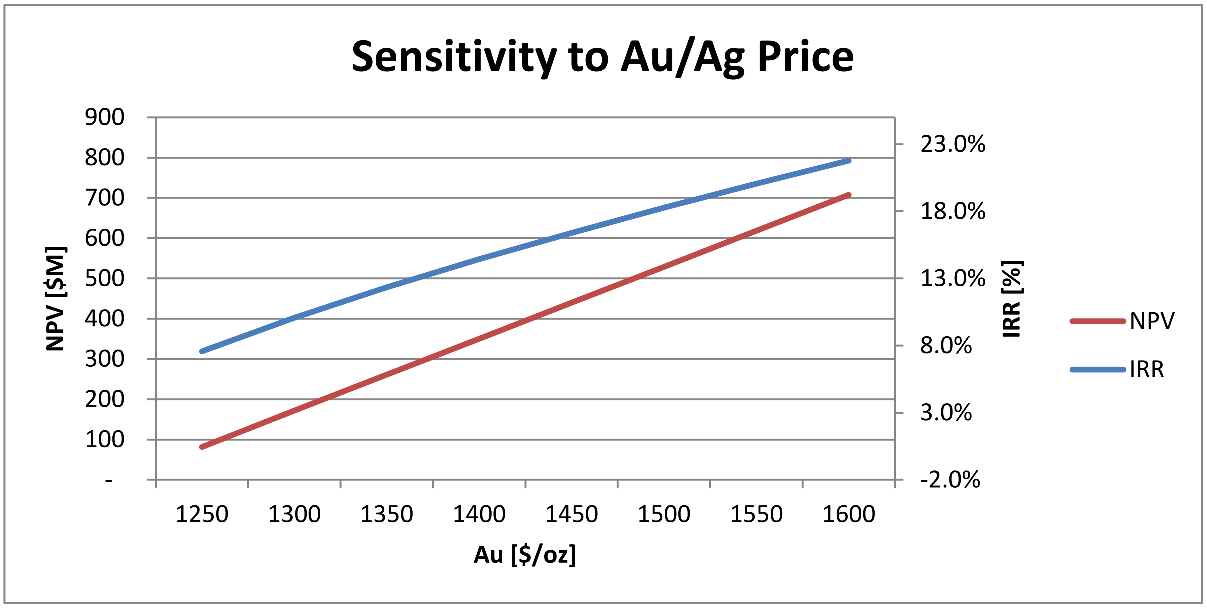


Figure 22-1: Sensitivity to Au/Ag Price

Project economics are highly sensitive to gold price variations and similarly moderately sensitive to silver price variations. Project economics were based on a gold price of US$1250/oz and a silver price of US$16.50/oz

Given the Pre-Feasibility Level of the capital and operating cost estimates, it is also beneficial to understand the impact of increasing or decreasing project costs. Table:22-5 illustrates the bearing of capital and operating costs on the economics of the project.

Table:22-5: Capital and Operating Costs

NPV@5%	CAPEX	431	485	539	593	647
OPEX		80%	90%	100%	110%	120%
25.49	80%	334	289	245	201	156
28.67	90%	254	209	165	120	75
31.86	100%	173	129	84	39	-6
35.05	110%	93	48	3	-42	-88
38.23	120%	11	-34	-79	-126	-172

IRR	CAPEX	431	485	539	593	647
OPEX		80%	90%	100%	110%	120%
25.49	80%	16.2%	13.9%	12.0%	10.3%	8.9%
28.67	90%	13.9%	11.7%	9.9%	8.3%	7.0%
31.86	100%	11.4%	9.4%	7.6%	6.1%	4.8%
35.05	110%	8.7%	6.7%	5.1%	3.7%	2.4%
38.23	120%	5.5%	3.7%	2.1%	0.7%	-0.5%

Further definition and refinement of costs in subsequent project development stages will improve the certainty of the financial outcome.

22.4 DISCUSSION

It is recognized that the economics of the Certej Project warrant careful consideration before taking significant steps to move forward with development of the project.

The sensitivity of the project to both gold price and capital and operating costs is noted and dictates that further definition is required to achieve greater certainty on project costs and consequential cash flows. With respect to commodity prices, as shown in Table 22-4 a gold price of US$1300/oz and a silver price of US$22.00/oz, yields a project NPV of US$178M and IRR of 10%.

Eldorado is pleased that the project can demonstrate a positive return at a Pre-Feasibility level, in current market conditions. Eldorado feels there are significant opportunities to improve the economics of the project and to shape Certej into a position to provide significant value to shareholders. To attain the necessary level of definition and certainty on project costs and projected cash flows, it is necessary to pursue the project to a phase of optimization and trade-off studies, followed by a Feasibility level study.

Eldorado does not intend to subject the project to undue delay, but instead to continue to pursue the opportunity to provide value to shareholders and to offer positive benefits to the local and regional communities in Romania. In keeping with the commitments made to Romanian communities and authorities, Eldorado intends to continue to invest in pre-development activities on the project site throughout the duration of on-going studies and the Feasibility study.

SECTION • 23 ADJACENT PROPERTIES

Not applicable.

SECTION • 24 OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation, which would be relevant to be included in this section. All of the other sections provide the reader with sufficient data and information to make the technical report understandable and not misleading.

SECTION • 25 INTERPRETATION AND CONCLUSIONS

Deva Gold holds a mining licence for Certej granted in 1999 for 20 years, with the right to extend for five year periods until the reserve is exhausted. Possession of this license and key permits for the project, including a crucial Environmental Permit, enable the project to move forward with expectation of minimal or no delays.

Implementation of a comprehensive QA/QC program provides assurance of control throughout the preparation and analytical processes. Monitoring of the quality control samples showed all data were included for gold analyses on drill core and reverse circulation drill samples. These QA/QC results demonstrate that the Certej Project assay database, particularly for new data obtained during 2012 and 2013, is sufficiently accurate and precise for resource and reserve estimations.

Use of conventional open pit methodologies for mine plan development accords well-established understanding of mine plan parameters and considerations. The methodologies employed yield opportunities for optimization upon further development of updated geotechnical and hydrogeological work, in the next phases of the project.

Application of conventional pressure oxidation to the project brings with it a well-established understanding of the recovery process and the factors critical to maintaining or improving metal recovery. This adjustment to the recovery process reduces technological risks and increases the variety of resources available for optimization of the design.

The close proximity of highly developed infrastructure for transport and utilities provide significant opportunities for the project to improve existing infrastructure, minimizing the need for construction of new infrastructure. The availability of space (for process facilities, tailings and waste rock sites) adjacent to the open pit minimizes the need for site development, and reduces the amount of additional infrastructure required. Reliance on existing or upgraded infrastructure and minimal project footprint reduces the need for additional permitting, capital costs, and potential scheduling delays to the project.

The villages and cities in the county of Hunedoara are a source of skilled mining labour and the region will benefit from employment opportunities and the flow-on effect of investment in the project and surrounding communities. The position of the project in this region, which is seeking investment and holds a favorable attitude to mining, promises the opportunity to continue to develop mutually beneficial relationships with the local communities, regional authorities, and the state government.

The cost estimates prepared for this technical study are well defined and within the accuracy level expected of a Pre-Feasibility study. Several potential cost savings opportunities were identified and will be further investigated in the next phases of the project. These opportunities are expected to have a positive impact on the project economics.

Eldorado is confident that the Certej Project Technical Report has been completed to the requirements of National Instrument 43-101 Standards for Disclosure for Mineral Projects, and that there are no significant unidentified risks or uncertainties that could affect the results or conclusions presented herein.

SECTION • 26 RECOMMENDATIONS

The Pre-Feasibility study has provided a technical and economic solution that is a good basis to proceed with further investigations and has identified areas that can be further investigated to optimize the project, prior to proceeding with a Feasibility Study.

The next phases of the project are to complete optimization studies and proceed to a Feasibility Study as follows:

Phase 1 – Prefeasibility Study – Complete

Phase 2 – Optimization Studies

- Field investigations including geotechnical and hydrology studies to support final selection and design of the Tailings Management Facilities;
- Design of infrastructure including Access Road, Fresh Water Supply and Power Supply to allow rapid mobilization upon a project construction decision;
- Equipment Procurement Strategy options including alternate commercial options to the "over the fence" supply of Oxygen as carried in the Prefeasibility Study;
- Initiate a field program to finalize the Limestone Quarry selection;
- Additional metallurgical testwork and waste rock and tailings characterization studies;
- Waste Rock and Mine Plan Optimization;
- Completion of a detailed Logistics Study;
- Develop updated Project Economics, based on the results of the optimization studies; and
- Investigation of potential opportunities for co-processing external resources.

This optimization period is estimated to take 6 months and the work has been budgeted at US$4.1 million to be expended by the end of the third quarter of 2014.

Limited site work to support Optimization of the Project with Water Treatment site investigations, Construction Power distribution and limited site civil work has been budgeted at US$2.6 million to be expended by the end of 2014.

Phase 3 – Feasibility Study

Once the Optimization Studies are complete, the project will proceed to a feasibility study, contingent on the results of the studies having a positive impact on the Project Economics, such that the scope of work can adequately be defined to allow advancement in all areas.

As the design will be significantly advanced at the end of the optimization phase, the Feasibility Study timeframe has been estimated at 10 months, beginning at the end of the fourth quarter of 2014. At this time, a budget for the full feasibility study has not been fully estimated.

SECTION • 27 REFERENCES

Forward, P., Liddell, N., Jackson, T., 2009. '*Certej Updated Definitive Feasibility Study Summary Technical Report*'. National Instrument 43-101 Technical Report, February 2009.

RSG Global Consulting (now Coffey Mining), 2007. '*Certej Gold Silver Project, Romania, Technical Report'.* National Instrument 43-101 Technical Report, November 2007.

SECTION • 28 CERTIFICATES OF AUTHORS

CERTIFICATE OF QUALIFIED PERSON

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I, Stephen J. Juras, am a Professional Geoscientist, employed as Director, Technical Services, of Eldorado Gold Corporation and reside at 9030 161 Street in the City of Surrey in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report for the Certej Project, Romania*, with an effective date of February 21, 2014.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated from the University of Manitoba with a Bachelor of Science (Honours) degree in geology in 1978 and subsequently obtained a Master of Science degree in geology from the University of New Brunswick in 1981 and a Doctor of Philosophy degree in geology from the University of British Columbia in 1987.

I have practiced my profession continuously since 1987 and have been involved in: mineral exploration and mine geology on gold, copper, zinc and silver properties in Canada, United States, Brazil, China, Greece and Turkey; and ore control and resource modelling work on gold, copper, zinc, silver, platinum/palladium and industrial mineral properties in Canada, United States, Mongolia, China, Brazil, Turkey, Greece, Romania, Peru and Australia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Certej Project on numerous occasions with my most recent visit occurring from September 15 to 17, 2013.

I was responsible for reviewing matters related to the geological data and directing the mineral resource estimation and classification work for the Certej Project in Romania. I am responsible for the preparation or supervising the preparation of items 7, 8, 9, 10, 11, 12 and 14 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, *Technical Report for the Certej Project, Romania*, with an effective date of February 21, 2014, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 7th day of April, 2014.

"Stephen Juras"

Stephen J. Juras, Ph.D., P.Geo.

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I, Richard Alexander, am a Professional Engineer, employed as Project Director of Eldorado Gold Corporation and residing at 5922 Boundary Place in the city of Surrey in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report for the Certej Project, Romania*, with an effective date of February 21, 2014.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated from the University of Alberta with a Bachelor of Science degree in mechanical Engineering in 1985.

I have practiced my profession continuously since 1985 and have worked in project management and engineering and construction management of mineral projects in South America, Central America, Canada, United States, Turkey, Greece, Romania, and the former Soviet Union.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Certej Project on numerous occasions with my most recent visit occurring from February 3 thru 11, 2014.

I was responsible for coordinating the preparation of the report entitled *Technical Report for the Certej Project, Romania*, with an effective date of February 21, 2014 , I am responsible for the preparation or supervising the preparation of Items 1, 2, 3, 4, 5, 6, 18, 19, 20, 21, 22, 23, 24, 25, 26, and 27 in the technical report.

I have read National Instrument 43-101 and Form 43-101F1 and the sections for which I am responsible in this report has been prepared in compliance with same.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Vancouver, British Columbia, this 7th day of April, 2014.

“Richard Alexander”

Richard Alexander, P.Eng.

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I, Richard Miller, am a Professional Engineer, employed as Manager, Mine Engineering, of Eldorado Gold Corporation and residing at 832 Victoria Drive in the city of Port Coquitlam in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report for the Certej Project, Romania*, with an effective date of February 21, 2014.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated from the University of British Columbia with a Bachelor of Applied Science degree through the department of Mining and Mineral Process Engineering in 1987.

I have practiced my profession continuously since 1987 and have worked at copper, diamond and gold mines in Canada, South Africa, Namibia, Guinea and Turkey in the capacities of Mining Engineer, Project Manager and Mine Manager covering planning, surveying, production, contract management, department head and global manager covering operations in Turkey, Brazil and China. I have also consulted to mining related companies in Canada, Dominican Republic, Burkina Faso, Serbia and Russia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I visited the project several times with the most recent visit on October 10 to 11, 2013.

I was responsible for directing and reviewing matters related to the mining methods and the mineral reserve estimation work for the Certej Project in Romania. I am responsible for the preparation or supervising the preparation of items 15 and 16 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the sections for which I am responsible in this report entitled, *Technical Report for the Certej Project, Romania,* with an effective date of February 21, 2014, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Vancouver, British Columbia, this 7th day of April, 2014.

"Richard Miller"

Richard Miller, P.Eng.

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I, Robin Kalanchey, am a Professional Engineer, employed as Group Metallurgist, of Eldorado Gold Corporation and residing at 611 Bournemouth Crescent in the city of North Vancouver in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report for the Certej Project, Romania*, with an effective date of February 21, 2014.

I am a member of the Association of Professional Engineers and Geoscientists of Alberta. I graduated from the University of British Columbia with a Bachelor of Applied Science degree in metals and materials engineering in 1996.

I have practiced my profession continuously since 1996 and have been involved in: mineral processing and metallurgical testing, metallurgical process plant design and engineering, and metallurgical project evaluations for gold, nickel, cobalt, copper, zinc and molybdenum projects in Canada, Australia, Bolivia, Bulgaria, Brazil, Chile, China, Colombia, Ecuador, Indonesia, Kazakhstan, Madagascar, Peru, Philippines, South Africa, the United States and Zimbabwe.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I was responsible for reviewing matters related to the mineral processing and metallurgical testing, and preparation of the prefeasibility level engineering design for the process plant for the Certej Project in Romania. Sections 13 and 17 were prepared by me or under my supervision.

As of the Effective Date, I had not visited the project site.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1 and the sections for which I am responsible in this report entitled *Technical Report for the Certej Project, Romania*, with an effective date of February 21, 2014, has been prepared in compliance with same.

As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Vancouver, British Columbia, this 7th day of April, 2014.

"Robin Kalanchey"

Robin Kalanchey, P.Eng.

SECTION • 29 SIGNATURE PAGE AND EFFECTIVE DATE

The effective date of this report entitled "Technical Report for the Certej Project, Romania" is February 21, 2014. It has been prepared by Rick Alexander, P.Eng., Stephen Juras, P.Geo., Richard Miller, P.Eng., and Robin Kalanchey, P.Eng., each of whom are qualified persons as defined by NI43-101.

Signed this 7th day of April, 2014.

SIGNED

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