Technical Report
Olympias Mine
Greece

Centered on Latitude E 40° 36' and Longitude N 23° 50' approximately

Effective Date: December 31, 2019

Prepared by:
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GLOSSARY

Units of Measure

Annum (year)	a
Billion	B
Centimeter	cm
Cubic centimeter	cm^3
Cubic meter	m^3
Day	d
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Dollar (American)	US$
Dollar (Canadian)	CAN$
Euro	€
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hour	h
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic meter	kg/m^3
Kilograms per hour	kg/h
Kilograms per square meter	kg/m^2
Kilometer	km
Kilometers per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilowatt hour	kWh
Kilowatt hours per tonne	kWh/t
Kilowatt hours per year	kWh/a
Kilowatt	kW
Less than	<
Litre	L
Megavolt Ampere	MVA
Megawatt	MW

Meter	m
Meter above Sea Level	masl
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimeter	mm
Million cubic meters	Mm^3
Million ounces	Moz
Million tonnes per Annum	Mtpa
Million tonnes	Mt
Million	M
Million Years	Ma
Newton	N
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%
Percent by Weight	wt%
Pound	lb
Square centimeter	cm^2
Square kilometer	km^2
Square meter	m^2
Thousand tonnes	kt
Three Dimensional	3D
Tonnes per day	t/d or tpd
Tonnes per hour	tph
Tonnes per year	tpa
Volt	V
Watt	W
Weight/volume	w/v
Weight/weight	w/w

Abbreviations and Acronyms

Acidity or Alkalinity	pH
Acid Base Accounting	ABA
Aluminum	Al
ALS	ALS
Antimony	Sb

Argillic	ARG
Argon	Ar
Arsenic	As
Association for the Advancement of Cost Engineering	AACE
Atomic Adsorption	AA
Barium	Ba
Block Model	BM
Bond Abrasion Index	Ai
Bottle Roll	BR
Bed Volumes	BV
Business Opening and Operations Permit	GSM
Cadmium	Cd
Calcium Hydroxide	$Ca(OH)_2$
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Cemented Aggregate Fill	CAF
Certified Reference Material	CRM
Cobalt	Co
Coefficient of Variance	CV
Concentric Cylinder Rotational Viscometry	CCRV
Construction Management	CM
Copper	Cu
Copper Sulphate	$CuSO_4$
Critical Solids Density	CSD
Cyanide	CN
Drift and Fill	DAF
Semi pure gold alloy	Doré
East	E
Eldorado Gold Corporation	Eldorado
Engineering, Procurement, Construction Management	EPCM
Environmental Impact Assessment	EIA
Environmental Impact Statement	EIS
Environmental Management Plan	EMP
European Goldfiels	EGL
European Union	EU
Factor of Safety	FOS
Feasibility Study	FS
Flocculant	FLOC
General and Administration	G&A
Geological Strength Index	GSI
Geosynthetic Clay Liner	GCL
Gold	Au
Gold Equivalent	Au Equiv
Hellas Gold SA	Hellas Gold

Hellenic Fertilizer Company	HFC
High Density Polyethylene	HDPE
High Grade	HG
Hydrochloric Acid	HCl
Hydrogen Oxide	H_2O
In Stream Analysis System	ISA
Induced Polarization	IP
Inductively Coupled Plasma	ICP
Inductively Coupled Plasma Emission Spectroscopy	ICP-ES
Inductively Coupled Plasma Mass Spectrometry	ICP-MS
Inner Diameter	ID
Internal Rate of Return	IRR
International Financial Reporting Standards	IFRS
International Organization for Standardization	ISO
Iron	Fe
Lead	Pb
Life-of-mine	LOM
Locked Cycle Test	LCT
Manganese	Mn
Mechanical, Piping, Electrical, Instrumentation	MPEI
Measured & Indicated	M&I
Mercury	Hg
Methyl Isibutyl Carbinol	MIBC
Ministry of Environment	MOE
National Instrument 43-101	NI 43-101
Nearest Neighbour	NN
Nearest Neighbour Kriging	NNK
Net Acid Generation	NAG
Net Present Value	NPV
Net Smelter Return	NSR
Nickel	Ni
North	N
North East	NE
North West	NW
Operator Control Station	OCS
Ordinary Kriging	OK
Outer Diameter	OD
Paste Fill	PF
Polyvinyl Chloride	PVC
Potassic	POT
Potassium	K
Potential of Hydrogen	pH
Prefeasibility Study	PFS

Probability Assisted Constrained Kriging	PACK
Public Power Corporation	PPC
Quarter	Q
Qualified Person(s)	QP(s)
Quality assurance	QA
Quality control	QC
Quantitative Evaluation of Minerlas by Scanning Electron Microscopy	QEMSCAN
Quartz	Qz
Request for Quotations	RFQ
RL=Reduced Level	masl
Rock Quality Designation	RQD
Run of Mine	ROM
Selenium	Se
Silicon	Si
Silver	Ag
Sodium Cyanide	NaCN
Sodium Hydroxide	NaOH
Sodium Metabisulphite	$Na_2S_2O_5$
Sodium Metabisulphite	SMBS
South	S
South East	SE
South West	SW
Specific Gravity	SG
Spherical	SPH
Standard Reference Material	SRM
Strontium	Sn
Suub-level Caving	SLC
Sulfur	S
Sulfur Dioxide	SO_2
Sulphide	S^{2-}
Sulphuric Acid	H_2SO_4
Tailings Management Facility	TMF
Technical Study	TS
Transportable Moisture Limit	TML
Transverse Longhole Open Stoping	TLHOS
TVX Gold Inc.	TVX
Unconfirmed Compressive Strength	UCS
Underflow	U/F
Underground	UG
Universal Transverse Mercador	UTM
Uranium	U
Value Added Tax	VAT
West	W

Work Breakdown Structure	WBS
Xanthate Collector	SIPX
Zinc	Zn

SECTION • 1 SUMMARY

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, owns and operates the Olympias mine in Greece through its 95% owned subsidiary Hellas Gold SA (Hellas Gold). Eldorado has prepared this technical report of the Olympias mine to provide an updated description of the geology and mineralization, mineral resources and mineral reserves, and mine and mill operations in light of significant additional technical studies as well as operating experience since the prior technical report, which had an effective date of July 14, 2011 (Technical Report on the Olympias Project Au Pb Zn Ag Deposit, Northern Greece, 2011).

Information and data for this report were obtained from the Olympias mine and Eldorado's corporate office. The work entailed review of pertinent geological, mining, process and metallurgical data in sufficient detail to support the preparation of this technical report.

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 David Sutherland, P.Eng., Paul Skayman, FAusIMM, Ertan Uludag, P.Geo., Colm Keogh, P.Eng. and Sean McKinley, P.Geo., whom are all employees of Eldorado. All qualified persons have visited the Olympias gold mine.

1.1 PROPERTY DESCRIPTION AND LOCATION

The Property is located within the Kassandra Mines complex located on the Halkidiki Peninsula of Northern Greece. The Olympias mine lies 9 km north-northwest of the Stratoni port and loading facility, accessed by a paved road along the coast. The terrain is characterized by hills rising to about 600 m above sea level, with steeply incised valleys.

The Property consists of mining concession numbers F13 and F14, which have a combined area of 47.27 km^2. Hellas Gold has been granted mining rights over these concessions until 7 April 2024. The concessions are conditionally renewable for a further two consecutive periods of 25 years each. Hellas Gold has ownership of a small portion of private land within the concessions.

In July 2011, the Ministry of Environment (MOE) formally approved the Environmental Impact Statement (EIS) submitted by Hellas Gold for the three Kassandra mines mine sites, being Olympias, Skouries, and Stratoni. This EIS is valid for 10 years and subject to renewal in 2021.

For production to commence, the MOE required the submission of a technical study. A study was submitted to the MOE and approved in early 2012. The installation permit for what was termed the Phase II process plant was issued on 22 March 2016. The Company received the operating permit for the Phase II plant in September 2017, allowing commencement of commercial production operations. Also, in September 2017 the Company received an extension of the installation permit and an interim operating permit for the Kokkinolakkas TMF, as well as the delayed installation permit for the paste backfill plant.

Notifications for the Operation of the Olympias Paste Plant and Kokkinolakkas Tailings Management Facility (TMF) were formally submitted in 2018 and remain in force in line with new legislation that replaced previous operating permit issuance procedures.

1.2 GEOLOGY

The Western Tethyan orogenic belt in southeast Europe contains several major metallogenic provinces including the Serbo-Macedonian Metallogenic Province that hosts the Kassandra mining district. Crystalline basement within the district includes the upper Serbo Macedonian Vertiskos Unit and the lower Kerdilion Unit exposed within the southern Rhodope metamorphic core complex.

The Olympias deposit is located 6 km north of the Stratoni fault within the Kerdilion unit. Replacement-style sulfide orebodies are hosted by marble interlayered within a sequence of quartzo-feldspathic biotite gneiss, amphibolite and plagioclase microcline orthogneiss. The massive sulfide orebodies plunge shallowly to the southeast for over 1.8 km, subparallel to the orientation of F2 fold hinges and a locally developed L2 intersection lineation. The locations of the sulfide lenses, however, are largely controlled by strands of the ductile-brittle Kassandra fault and East fault and sub-horizontal shear zones that occur between the two faults.

Sulfide mineralogy of the Olympias deposit consists of coarse-grained, massive and banded lenses dominated by variable amounts of sphalerite, galena, pyrite, arsenopyrite, chalcopyrite and boulangerite. Gold occurs primarily in solid solution with arsenopyrite and pyrite.

Olympias is an example of a polymetallic carbonate replacement deposit. However it is somewhat unusual due to the high Au content of the deposit. Key characteristics of this class of deposit include carbonate host rocks, massive sulfide mineralization, spatial and temporal relationship with magmatism and zoned metal distribution.

1.3 DRILLING, SAMPLING AND VERIFICATION

Diamond drillholes continue to be the prime source of subsurface geologic and grade data for the Olympias deposit. The previous operator, TVX, drilled 764 drillholes for a total of 93,246 m. These are becoming less important as new information is acquired. Currently holes are drilled by Eldorado using contractors drilling HQ or NQ-size (63.5 mm or 47.6 mm nominal core diameter). The average drillhole depth is 100 m, as the holes are drilled from locations underground giving good intersection angles with the zones. There are currently 1,855 drillholes for 210,893 m in the database.

Core is delivered to secure core logging areas, and the core is logged in detail straight into a database using computer tablets. Lithology, alteration, structure and mineralization data is collected; core recovery data is also measured. Core photos are routinely taken of all the core, both wet and dry, using a camera stand to ensure consistent photographs. Collar and downhole survey data is collected. Downhole surveys are taken using a Reflex Gyro^{TM} or a Devico Deviflex. Both of these multishot instruments are calibrated annually.

A dataset of measured bulk densities from over 900 mineralized samples is used to inform the resource block model.

Sampling of the core is carried out on 1 m intervals or to geological contacts. The core is sawn using an automated core saw and half is bagged for dispatch, with the remainder being placed in the core box for storage. Drill core samples are routinely sent to the ALS facility in Romania. They are bagged and packed in large sealed wooden bins before being trucked to ALS. The sample rejects are returned to the mine site in the same bins. The samples are prepared for assaying at the ALS facility.

All samples were assayed for gold by 30 g fire assay with an AAS finish, with Au values above 10 ppm determined by a gravimetric finish. Multi-element determination was carried out by inductively coupled plasma mass spectrometry (ICP-MS) analysis and / or inductively coupled plasma emission spectroscopy (ICP-ES) analysis.

Eldorado employs a comprehensive QA/QC program as part of the assaying procedure, involving regular insertion of Certified Reference Materials (CRMs), duplicates and blank samples. Site geologists regularly monitor the performance of CRMs, blanks and duplicates as the assay results arrive on site. In Eldorado's opinion, the QA/QC results demonstrate that the Olympias mine's assay database is sufficiently accurate and precise for the resource estimation.

An important measure of performance at any producing mine is reconciliation of the block model to the final mill production figures, adjusted for stockpiles as necessary. The reconciliation conducted at Olympias is detailed and thorough. It is currently providing a quarterly snapshot and demonstrating that the block model, and thus the mineral resources, are valid and robust. This validates the data underpinning the model and is, by association, a good verification of the work done.

1.4 METALLURGICAL TESTWORK

Historical and independent metallurgical test reports on samples from the Olympias deposit were reported to be commissioned by previous mine owners. The 2011 Technical Report concluded that, in general, the testwork confirmed the anecdotal evidence from historic operations that the ore responds well to flotation with high recoveries.

In 2015, further metallurgical testwork and mineralogical investigations were carried out on Fresh Ore and Old Ore samples from the Olympias deposit. The main sulphide minerals in these samples were galena, sphalerite, pyrite, and arsenopyrite. Other than quartz, all other minerals can be considered minor. The liberation of all sulphide minerals was excellent at a P_{80} of 120 µm. Flotation of galena, sphalerite and pyrite / arsenopyrite in a sequential flowsheet was found to be effective at producing lead, zinc and gold concentrates.

Lead flotation was found to work well with Aerophine 3418 A, with lime and sodium cyanide the only depressants needed. Copper activation and zinc flotation were found to be optimum at a pH of 11.8 with $CuSO_4$ and SIPX. Flotation of pyrite and arsenopyrite was effective at a pH of 6.0 controlled with sulphuric acid and SIPX.

The rejection of arsenic and antimony were found to be problematic in the Pb concentrate in different samples. It was found to be extremely difficult to reduce the content of arsenic in the Pb concentrate from Fresh Ore to less than approximately 3%. Antimony was not an issue with the Fresh Ore, but was identified with the Old Ore as being present in the Pb concentrate at a grade of approximately 3%, although arsenic was not an issue.

Tests confirmed good settling-thickening characteristics for the concentrates and tailings samples. Vacuum filtration of the Tails underflow sample showed high throughput and low residual cake moisture, which ranged from 13.6% to 22.3%. Pressure filtration on concentrate underflow samples indicated high throughput and low residual moisture.

Acid Base Accounting (ABA) testing indicated uncertain acid generation potential for the Olympias tailings; NAG testing reported no net acidity generated. Analysis of the NAG liquor reported arsenic at an environmentally significant concentration.

1.5 MINERAL RESOURCES

Mineral resource estimates for Olympias mine were made from a 3D block model utilizing MineSight 3D software. Project limits, in UTM coordinates, are 478105 to 479700 East, 4491165 to 4493480 North and -800 to +60m elevation. Cell size for the project was 5m east x 5m west X 5m high.

A grade based discriminant was developed to allow for more consistent interpretations to be made. This was accomplished by creating a simplistic value formulae based on the logic of a Net Smelter Return (NSR) formula that used a combination of metal prices and metal recoveries to act as weighting factors against each metal. This metric, a dollar value, proved to be an excellent surrogate for a comprehensive equivalent grade. Inspection of these resource defining values (RDV) showed that for the parameters used, a value of \$50 best defined what one would classify as likely economically mineralized zones.

For the Olympias modeling, the deposit was divided into three zones: East, West, and Flats. Within each of these zones, modeling domains were created using the \$50 RDV. Assays and composite samples were tagged by these domain shapes ahead of data analysis and grade interpolation. The assays were top-capped prior to compositing and were composited into 1 m composites within the wireframes.

Grade estimates for Au, Ag, As, Pb, Zn and Fe were interpolated using an inverse distance to the power of 4 (ID^4) method. Nearest–neighbour (NN) grades were also interpolated as a declustered distribution to validate the estimation method. A multi-pass approach was instituted for interpolation. The first pass required a grade estimate to include composites from 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. The metal models were validated by visual inspection, checks for global bias and local trends and for appropriate levels of smoothing (change-of-support checks).

The mineral resources of the Olympias mine were classified using logic consistent with the CIM definitions referred to in the National Instrument 43-101-*Standards of Disclosure for Mineral Projects* (NI 43-101). The mineralization of the mine satisfies sufficient criteria to be classified into measured, indicated, and inferred mineral resource categories.

Olympias mine mineral resources, as of September 30, 2019, are shown in Table 1-1. The Olympias mine mineral resources are reported at a zero cut-off within wireframes based on $50 RDV.

Table 1-1: Olympias Mine Mineral Resources as of September 30, 2019

Classification	Tonnes (Kt)	Au (g/t)	Au (Koz)	Ag (g/t)	Ag (Koz)	Pb (%)	Pb (Kt)	Zn (%)	Zn (Kt)
Measured	2,702	10.93	950	156	13,552	5.0	135	6.0	162
Indicated	11,779	7.52	2,848	134	50,746	4.6	542	6.2	730
Measured and Indicated	**14,481**	**8.16**	**3,798**	**138**	**64,298**	**4.7**	**677**	**6.2**	**892**
Inferred	3,720	7.98	954	137	16,385	3.9	145	4.0	149

1.6 MINERAL RESERVES AND MINING

The mineral reserve estimates conform to CIM Definition Standards (2014). All design and scheduling has been completed using the mineral resource model and estimate described in Section 14. Only measured and indicated resources have been used for mineral reserves estimation. The estimation assumes that the mining methods employed at the mine will be drift and fill (DAF) and transverse longhole open stoping (TLHOS).

The cut-off values supporting the estimation of underground mineral reserves were developed in 2018 and based on future projected operating costs at a steady-state production rate of 650,000 tonnes per annum. The operating cost assessment indicated that NSR values of $133/t for DAF mining and $116/t for TLHOS mining would adequately cover all site operating costs on a breakeven basis. The weight averaged operating cost can be estimated at $125/t considering the balance between TLHOS and DAF. These DAF and TLHOS cut-offs were used to create potentially mineable stope shapes from the NSR block model (NSR BM). The NSR BM was created by Eldorado and is based on metallurgical recovery experience and historical sales and includes recognition of transport and refining costs.

In the evaluation of underground mineral reserves, modifying factors were applied to the tonnages and grades of all mining shapes to account for dilution and ore losses. In the DAF stopes, a mining dilution factor of 13% and a mining recovery of 98% were estimated. In the TLHOS stopes a mining dilution factor of 14% and a mining recovery of 95% were estimated.

The mineral reserve estimate is summarized in Table 1-2 and has an effective date of September 30, 2019.

Table 1-2: Olympias Mineral Reserves as of September 30, 2019

Class	Tonnes (kt)	Au (g/t)	Au (Moz)	Ag (g/t)	Ag (Moz)	Pb (%)	Pb (kt)	Zn (%)	Zn (kt)
Proven	2,601	9.19	0.77	133	11.1	4.3	112	5.1	133
Probable	10,324	6.47	2.15	115	38.2	4.0	413	5.3	547
Total	12,925	7.02	2.92	119	49.3	4.1	525	5.3	680

Notes:

- Mineral reserves are included in measured and indicated mineral resources.
- Figures in the tables may not compute due to rounding.
- The mineral reserves are based on a planning cut-off grade of $133/t for DAF and $116/t for TLHOS, and $50/t for development ore.
- Cut-off grades are based on a gold metal price of $1,250/oz, silver metal price of $16/oz, zinc metal price of $2,400/t, and lead metal price of $2,000/t.
- Metallurgical recoveries are based on feed grade and metallurgical algorithms.
- Exchange rate used is €1.20 = US$1.00.
- Average mining dilution and mining recovery factors of 14% and 95%, respectively, for LHOS, and 13% and 98%, respectively, for DAF are assumed.
- Mineral reserves are reported on a 100% ownership basis; Eldorado owns 95% of Hellas Gold.

The Olympias mine is a 100% underground (UG) mining operation extracting ore from three zones: East, West and Flats and two sub-zones: Remnants, and North. These are shown in Figure 1-1 as an isometric view of the Olympias underground zones looking north-west.

Mining is currently at a rate of 360 ktpa. There is a production increase planned to take place between 2019 and 2023 to a steady-state value of 650 ktpa.

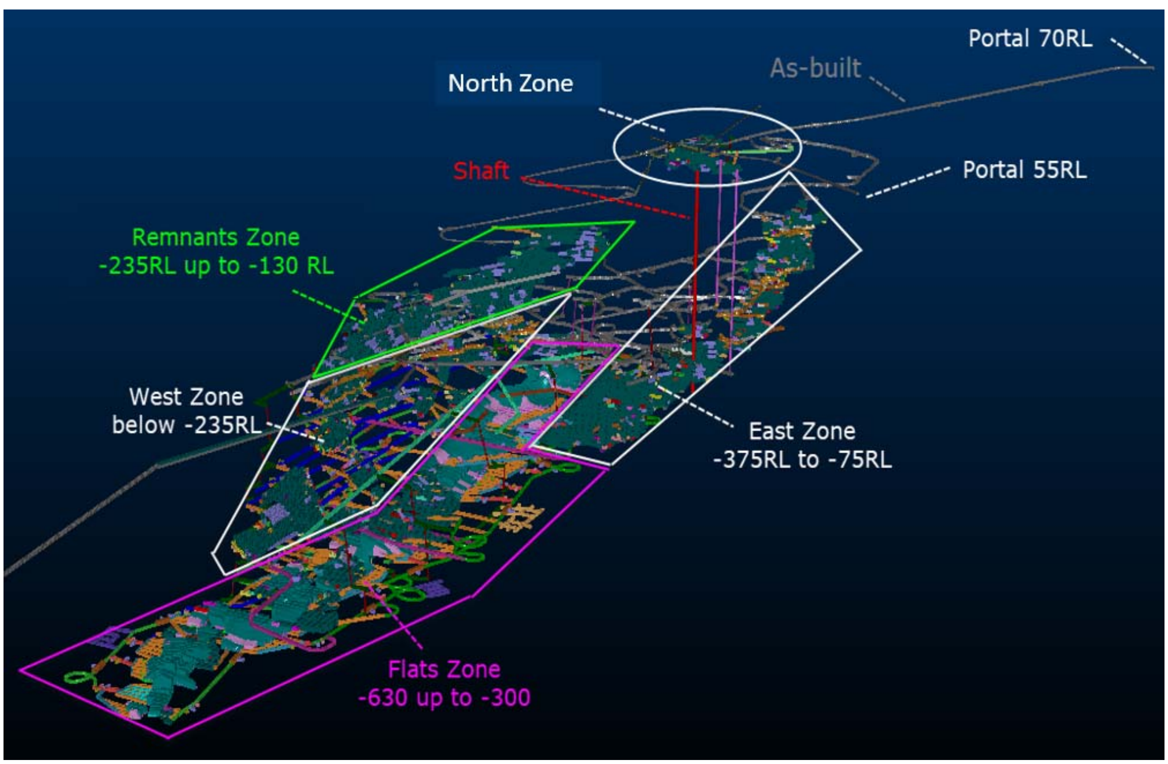


Figure 1-1: Isometric View of Olympias Underground Zones

Ground support is a combination of shotcrete, split sets and swellex bolts of varying lengths. All mined out areas are backfilled either with paste fill or cemented aggregate fill.

Mining at Olympias will be a combination of drift and fill (DAF) and transverse longhole open stoping (TLHOS). TLHOS will be confined to areas of the Flats where geometry and ground conditions support the use of the higher productive method. The remainder of the mine will use DAF.

TLHOS will be excavated 10 m wide and 30 m high maximum. The maximum length varies depending on the height and average rock quality. Blind uppers will be used for drill and blast with no top accesses.

DAF mining utilizes the overhand mining method. Stopes are accessed on the foot wall side from the main ramp starting at the bottom of each 20 m high stoping block. Each lift is mined 5 m high, with each panel limited to 5 m wide. There are two methods of DAF stoping, longitudinal and multi-pass.

Longitudinal is utilized where the stopes are less than 10 m wide. A single pass is mined down the length of the stope and any remaining ore is slashed off the HW on retreat. The stope is then filled prior to taking down the back in the access for the next lift.

Multi-pass is utilized where the stopes are greater than 10 m wide. Mining starts by cross cutting the ore body from footwall to HW. Then a single drift is developed in the ore to the extents of the stope in both directions along the footwall. Transverse cuts are then made from footwall to HW at multiple locations such that multiple working faces are established. After each cut is made it is backfilled, and then a new cut is mined beside the previous one. Once the lift is completely mined out, the back of the access is taken down to start the next lift.

All stopes are filled with backfill after excavation. Currently a combination of cemented aggregate fill (CAF) and paste fill is used. The paste fill system has been designed to produce 42 m^3/hr of paste, which will meet all future backfill requirements at 650 ktpa production with 70% utilization. CAF is delivered to stopes by truck and pushed into place with loaders. Paste is delivered with positive displacement pumps via drill holes and pipes.

There are two declines currently in use, one accessing the West Zone down to the Flats and one accessing the East Zone down to the Flats. There are multiple cross-over drifts between the two declines. Both declines are currently being extended into the Flats and to the bottom of the mine.

Both ore and waste are hauled to surface utilizing 40 tonne haul trucks on the existing and expanding declines. This will continue to be the case after the production increase to a steady-state value of 650 ktpa.

The total of personnel currently working at the mine is 562. The mine operates 3 shifts a day, 20 shifts per week.

There is currently 23 large pieces of mobile mining equipment on site: four jumbos, two bolters, five trucks, six loaders, four transmixers and two shotcrete sprayers. To achieve the production increase to 650 ktpa, funding has been allocated to increase this fleet number to 33. The increase will consist of one jumbo, two bolters, two longhole drills, two trucks, and three loaders.

The ventilation system consists of a single exhaust raise with fan. Air intake is via the two declines, the shaft and the old workings. Two means of egress are provided by the two declines. Current flow is 115 m^3/s; this will increase to 360 m^3/s for the 650 ktpa production rate.

Currently packaged explosives are being used for all blasting. There are no magazines on site and explosives are brought to site daily by the supplier. The use of bulk explosives is being investigated as an opportunity. The construction of a new underground magazine is planned for 2020. Steady-state full production explosives consumption is estimated at 53 tonnes per month.

As an operating mine, infrastructure is well developed with existing process water, compressed air, electrical distribution, and dewatering systems. For the 650 ktpa expansion, a new compressor, dewatering station and underground shop is being installed. These activities are currently in progress.

1.7 PROCESSING

The Olympias lead-zinc-gold-silver process plant has been developed in phases:

- Phase I involved the recommissioning of the plant, after a prolonged inactivity, and processing of the existing tailings. Phase I commenced in 2013 and was completed on commissioning of Phase II.
- Phase II was commissioned in late 2017 and is currently in operation. It included refurbishment and upgrading of the process facilities to process 430 ktpa of ore. The Phase II process facility consists of comminution, flotation and filtering to produce three saleable concentrates: lead / silver (lead), zinc and arsenopyrite / pyrite gold (gold). All concentrates are sold to worldwide markets. Tailings are used for underground backfill via the on-surface paste plant. Any tailings not used for underground mine backfill are filtered and trucked from the Olympias processing facility to the Kokkinolakkas tailings management facility (TMF) over public roads. Phase II will be in operation until commissioning of the Olympias Phase III expansion.
- The expansion Project involves upgrading of the existing Olympias process plant to handle a mine feed rate of 650 ktpa of ore, and upgrades to the port facilities at Stratoni. A permit is required from the Government of Greece to extend the mine life of the existing Phase II facilities and to increase the processing rate to 650,000 tpa. A further expansion may be required. Studies are currently underway to extract more value from the gold concentrate that is currently being sold to traders and smelters around the world. If a viable extraction method is developed, then construction of a new metallurgical facility would follow.

The treatment facility incorporates the following unit process operations:

- Three-stage crushing to produce 80% material passing 13 mm ore.
- Ore storage between crushing and grinding in a fine ore bin with a 1,155 t live capacity.
- Single-stage ball milling in closed circuit with hydrocyclones to produce 80% material passing 120 µm.
- Flash flotation to remove high grade lead from the recirculating load.
- Lead flotation employing the following circuits:

    - Roughing and scavenging.
    - Regrind of flash flotation and rougher / scavenger concentrate to 80% passing 15 µm size.
    - Three stages of cleaning and one stage of cleaner scavenging (in open circuit).
- Zinc flotation employing the following circuits:
    - Roughing and scavenging.
    - Regrind of rougher / scavenger concentrate to 80% passing 15 µm size of rougher / scavenger concentrate.
    - Three stages of cleaning and one stage of cleaner scavenging (in open circuit).
- Gold-pyrite flotation utilizing roughing and scavenging and a single stage of cleaning.
- Concentrate thickening, filtration, packaging, and storage prior to dispatch from the mine site by road.
- Tailings thickening and filtration prior to direct or reclaim addition to a cemented backfill plant.
- Tailings paste backfill.
- Reagent mixing, storage and distribution.
- Water and air services.

1.8 INFRASTRUCTURE

As an operating mine, current infrastructure is robust and complete. The mine has access to the main highway system in Greece via paved roads to the mine site. Local services are provided via the towns of Olympiada and Stratoni, with additional services available through Thessaloniki.

Zinc concentrate is shipped via the port facility at Stratoni (owned by Hellas Gold). Lead and arsenopyrite concentrates are shipped via Thessaloniki. There is a plan in place to rehabilitate and upgrade the Stratoni port facility over the period from 2020 to 2023; this will allow for the shipping of all concentrates by bulk out of Stratoni with the associated cost savings.

Water for the mine is obtained from underground dewatering, after treatment. Excess water from underground is discharged into the Mavrolakkas stream after settling and treatment to meet discharge standards. Currently, the capacity to handle 400 m^3/hr is available; this is being increased to 650 m^3/hr, which will be sufficient for the mine life. Service water is supplied via a local borehole in the regional aquifer.

Waste rock is either recycled underground for fill or is disposed of in the existing waste disposal facility. Tailings not used for pastefill are dewatered to 13% moisture content and transported by truck to the new tailings management facility at Kokkinolakkas near the Stratoni facilities, about 23 km by public road from the mine.

Existing surface facilities consist of a surface workshop, administration building, dry, shaft, and fuel storage (60,000 litres capacity). The workshop and fuel storage will be adequate for the production increase. The shaft is used for inspection of a legacy pump station only and there are no plans to rehabilitate the shaft further. Construction of a new geology preparation laboratory and technical services building, and expansion to the existing administration building are to be completed as part of the expansion.

Current power to site consists of a 20 kV 10 mVA pole line from the PPC grid. To facilitate the production increase, a new pole line at 150 kV 25 mVA, along with a new substation, will be constructed over the years 2020 and 2021. Backup power consists of 4,920 kW of diesel generation in multiple distributed generators. An additional 2,500 kW of generated power will be added for the production increase.

1.9 MARKET STUDIES AND CONTRACTS

The Olympias plant produces three concentrates: a pyrite concentrate containing gold, a lead concentrate that also contains silver and a zinc concentrate. Hellas Gold has negotiated multiple concentrate sales contracts with commodity traders.

Hellas Gold sells gold, lead and zinc concentrates to various customers from its Olympias operation. Currently contracts are used for short term revenue assumptions and long term averages are used over the life of mine.

1.10 ENVIRONMENTAL

The EIS for the Kassandra mines mineral deposits project (Kassandra Project) includes an area of 26,400 ha in north-eastern Halkidiki (Macedonia Region). The Kassandra Project includes the Skouries, Olympias and Stratoni sites. No significant impact is expected on the landscape, geological environment, atmosphere, or water resources in the area. The overall impacts to date have been positive to the environment, as legacy tailings and concentrate storage are in the process of being removed to the new TMF facility at Kokkinolakkas, and the associated areas rehabilitated. The Kassandra Project provides significant economic and social impacts for the Halkidiki Prefecture, including:

- A significant contribution is made to the national economy.
- Significant infrastructure is constructed and equipped by local companies.
- Service industries in the local economy expand.
- New jobs are progressively created.

After the completion of all Kassandra operations, the project areas will be rehabilitated according to appropriate and approved land uses. All structures are to be removed or left in a state that they do not pose a risk to the environment or public. The environment will be returned to a state of a self-sustaining ecosystem and safe and stable biological conditions will be re-created.

Hellas Gold has provided a letter of guarantee for €50M in favour of the Greek State as an assurance that the funds necessary for rehabilitation works in relation to the mining and metallurgical facilities of the Kassandra mines project and the removal, cleaning and rehabilitation of the old disturbed areas from the historic mining activity in the wider area of the project. Additionally, a Letter of Guarantee to the MOE, in the amount of €7.5M, has been provided as security for the due and proper performance of the Kokkinolakkas TMF.

1.11 CAPITAL AND OPERATING COSTS

The total capital cost estimate includes the expansion to 650 ktpa in the process plant, as well as capital development of the underground mine and sustaining capital, which is primarily for development, mine, processing infrastructure, and maintenance. Expansion capital specifically includes process plant upgrades, water management facilities, Phase 1 and Phase 2 of the Stratoni port facility upgrade as allocated amongst the Kassandra assets, related indirect costs, EPCM, owner's and contingency costs. Capital and sustaining capital costs are summarized in Table 1-3.

Table 1-3: Capital Cost Summary

Area	Growth ($ x 1,000)	Sustaining ($ x 1,000)	Total ($ x 1,000)
Mining Dev't and Equipment	9,616	89,471	99,087
Mine Infrastructure	3,876	49,161	53,037
Process Plant	18,892	14,356	33,248
Tailings + Water	9,457	17,630	27,087
Surface Supports	5,739	2,056	7,795
Stratoni Port Facility	15,478	5,700	21,178
Others	1,566	2,829	4,395
Total Direct Cost	**64,624**	**181,203**	**245,827**
Indirect Cost	6,894	10,990	17,884
Spares and Fills	3,153	-	3,153
EPCM	6,306	3,060	9,366
Owners Costs	3,153	-	3,153
Contingency	15,765	-	15,765
Total Installed Costs	**99,894**	**195,253**	**295,147**

Operating costs include allocations for underground mining, processing, tailings filtration, concentrate transport, tailings filtrations, materials handling, and stacking, water management, and general and administration (G&A).

Operating costs were estimated for each year of operation and are summarized in Table 1-4.

Table 1-4: Operating Costs Summary

Category	LOM ($M)	LOM average ($/t ore)	Steady State: 2024 to 2038 ($/t ore)
Mining costs	908	69.6	48.9
Tailings and water management costs	56	4.4	4.4
Processing costs	652	50.4	50.4
G&A costs	310	23.9	21.3
Total operating cost	**1,925**	**148.3**	**125.0**

Current labour productivities are considered very low when compared to other international operations. Eldorado believes that the envisaged improvement initiatives are reasonable and achievable and has allowed for effective work hours per shift to gradually reach 5.5 hrs. This will be assisted by the increase in the number of available faces for DAF mining through design and sequence changes. The introduction of more cost-effective THLOS mining for approximately one third of the steady-state tonnage is also a major factor in the future operating costs schedules. All of these initiatives should be in place as steady state by 2024. To illustrate the positive impact of these initiatives on LOM operating costs, a separate column was created in Table 1-4 showing estimated costs from 2024 to 2038.

1.12 ECONOMICS

The economic model has been prepared on an annual life of mine basis. The effective date of the estimate is assumed to be December 31, 2019. The LOM for the current mineral reserves is projected to be 21 years.

The after-tax cash flow analysis shows that Olympias is a robust Project once the additional capital is applied and the project reaches a steady state level of production. The net present value (NPV) of the Project is estimated to be $1,067M, using a discount rate of 5%, with positive after-tax net cash flows projected in all years other than 2019, the initial year of expansion capital expenditure.

Key results for the project economics are provided in Table 1-5.

Table 1-5: Key Economic Results

Olympias	Unit	Value
Total ore milled	kt	12,925
Net revenue	$M	4,501
Total Capital costs	$M	295
Operating costs (total)	$M	1,925
Transport, treatment and refining costs	$M	408
Royalties	$M	67
Corporate tax	$M	423
Net post-tax cash flow	$M	1,866
Post-tax NPV @ 5% discount rate	$M	1,061
Post-tax NPV @ 8% discount rate	$M	783

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices, and of operating and capital cost assumptions on the projected financial returns. The results of the sensitivity analysis are provided in Table 1-6 and Table 1-7

The test of economic extraction for Olympias is demonstrated by means of this sensitivity analysis. At the mineral reserve metals prices stated, Olympias shows strong positive economics. The sensitivity analysis also shows that the Olympias economics are robust when evaluated using lower metal price assumptions, or higher operating and capital costs.

Table 1-6: Metal Price Sensitivity Analysis

Sensitivity Ranges						
Parameters	Units	-15%	-7.5%	Project case	+7.5%	+15%
Gold price	$/oz	1,200	1,300	1,400	1,500	1,600
Silver price	$/oz	15.0	17.0	18.0	19.0	21.0
Lead price	$/t	1,800	1,900	2,100	2,300	2,400
Zinc Price	$/t	2,000	2,200	2,400	2,600	2,800
Results (after tax)						
NPV 0%	$M	1,327	1,604	1,866	2,164	2,415
NPV 5%	$M	724	897	1,062	1,241	1,395
NPV 8%	$M	519	655	783	925	1,046
Taxation	$M	248	336	423	512	591
Royalties	$M	42	52	66	76	120

Table 1-7: Capital and Operating Cost Sensitivity Analysis

Sensitivity Ranges				
Parameter	**Units**	**-15%**	**Project case**	**15%**
LOM Capex	$M	**256**	**295**	**347**
NPV 0% (after tax)	$M	1,927	1,866	1,836
NPV 5% (after tax)	$M	1,105	1,062	1,030
NPV 8% (after tax)	$M	823	783	755
LOM operating costs	$/t ore	**126**	**148**	**170**
NPV 0% (after tax)	$M	2,100	1,866	1,663
NPV 5% (after tax)	$M	1,205	1,062	929
NPV 8% (after tax)	$M	899	783	678

1.13 CONCLUSIONS AND RECOMMENDATIONS

Olympias is a mine with a long history that has only been recently put back into production by Eldorado. Commercial production was achieved in December 2017. Project economics are robust. There are sufficient mineral reserves for a mine life of 21 years at a steady-state production rate of 650 ktpa. The mine has embarked on a series of initiatives to increase both productivity and production rates.

Key recommendations are as follows:

- Expand extents of mineralization in upper levels of the mine area through conversion of Inferred resources and exploration. Any additional ore in this region would have synergies with the existing infrastructure.
- Use of ore sorting or dense media technology to pre-concentrate the ore coming from underground before it is processed in the grinding and flotation circuits with the intention of maximizing the annual metal production from the Olympias mine
- Ongoing research into an alternative processing method to extract additional value from the gold concentrate currently being sold directly to smelters and traders worldwide. Technical solutions do exist for extraction of much of the contained gold but these need to be able to demonstrate acceptable rates of return and to comply with environmental and safety requirements. If a viable extraction method is developed, then construction of the new metallurgical facility would follow.

SECTION • 2 INTRODUCTION

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, owns and operates the Olympias mine in Greece through its 95% owned subsidiary Hellas Gold SA (Hellas Gold). Eldorado has prepared this technical report of the Olympias mine to provide an updated description of the geology and mineralization, mineral resources and mineral reserves, and mine and mill operations in light of the long shelf life of the existing technical report, whose effective date was July 14, 2011 (Technical Report on the Olympias Project Au Pb Zn Ag Deposit, Northern Greece, 2011).

Information and data for this report were obtained from the Olympias mine and Eldorado's corporate office. The work entailed review of pertinent geological, mining, process and metallurgical data in sufficient detail to support the preparation of this technical report.

When preparing reserves for any of its projects, Eldorado uses a consistent prevailing metal price methodology that is in line with the CIM Guidance on Commodity Pricing used in Resource and Reserve Estimation and Reporting. These were set for gold, silver, lead and zinc, as of September 2019 for Eldorado's current mineral reserve work, at US$ 1,250/oz Au. US$ 16.00/oz Ag, US$2,000/t Pb and US$2,400/t Zn. All cut-off grade determinations, mine designs and economic tests of extraction used these prices in this technical report. In order to demonstrate the potential economics of a project, Eldorado may elect to use metal pricing closer to the current prevailing spot price and then provide some sensitivity around this price. For the Olympias gold mine, metal prices used for the potential economics evaluation were US$1,400/oz Au, US$18.00/oz Ag, US$2,100/t Pb and US$2,400/t Zn. This analysis (in Section 22 of this report) generally provides a better 'snapshot' of the project value at prevailing prices rather than limiting it to reserve prices, that might vary somewhat from prevailing spot prices. Eldorado stresses that only material that satisfies the mineral reserve criteria is subjected to further economic assessments at varied metal pricing.

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 David Sutherland, P.Eng., Paul Skayman, FAusIMM, Ertan Uludag, P.Geo., Colm Keogh, P.Eng. and Sean McKinley, P.Geo., whom are all employees of Eldorado.

David Sutherland, Project Manager, was responsible for overall preparation of the technical study and sections related to infrastructure and environment (report sections 1, 2, 3, 4, 5, 6, 18, 20, 24, 25, 26 and 27). He most recently visited the Olympias mine on October 2, 2019.

Colm Keogh, Operations Manager, Olympias Mine, was responsible for the mineral reserves and the preparation of related sections on mineral reserves calculation, mining methods and sections related to costs (report sections 15, 16, 21 and 22). He has worked at the Olympias mine since July 2019.

Sean McKinley, Senior Geologist Resource Development was responsible for the preparation of the sections in this report concerned with geological information, exploration and drilling (report sections 7, 8, 9, 10 and 23). He most recently visited the Olympias mine on July 2 to 5, 2019.

Ertan Uludag, Resource Geologist, was responsible for the mineral resources and the preparation of related sections on sample preparation and analyses, data verification and mineral resource estimation (report sections 11, 12 and 14). He most recently visited the Olympias mine on August 21 to 27, 2019.

Paul Skayman, Special Advisor to the COO, was responsible for the preparation of the sections in this report that dealt with metallurgy and process operations and related costs and payability (report sections 13, 17 and 19). He most recently visited the Olympias mine on November 11, 2019.

This document presents a summary of the current and forecast operations at the mine.

Currency used is US$ throughout, unless stated.

SECTION • 3 RELIANCE ON OTHER EXPERTS

The Qualified Persons have relied, in respect of legal aspects, upon the work of the experts listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant sections of the technical report.

The following disclosure is made in respect of this Expert:

- Public Notary Ioanna V. Gabrieli-Anagnostalakis, on behalf of the Minister of Finance and the Minister of Development of the Government of Greece

Report, opinion, or statement relied upon:

- Transfer Contract no. 22.138/12-12-2003 between the Greek State and Hellas Gold, specifically Appendix III which consists of Table 1, list of concessions.

Extent of reliance:

- Full reliance

Portion of Technical Report to which disclaimer applies:

- Section 4.2 Land Tenure

The following disclosure is made in respect of this Expert:

- ENVECO S.A., Environmental Protection, Management and Economy S.A

Report, opinion, or statement relied upon:

- ENVECO S.A., 2010, Environmental Impact Assessment (EIA) of the Mining-Metallurgical Facilities of Company Hellas Gold in Halkidiki

Extent of reliance:

- Full reliance

Portion of Report to which disclaimer applies:

- Section 20

SECTION • 4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The Property is located within the Kassandra mines complex found within the Halkidiki Peninsula of the Central Macedonia Province in Northern Greece. The complex is comprised of a group of mining and exploration concessions, covering 317 km^2, located approximately 100 km east of Thessaloniki. The concessions include the Olympias Mine, the Madem Lakkos and Mavres Petres mines (collectively known as Stratoni), and the Skouries copper-gold porphyry deposit. The Olympias and Mavres Petres mines are currently in production and the Skouries deposit is under development.

Olympias lies 9 km north-northwest of the Stratoni port and loading facility, accessed by a paved road along the coast. The area is centered on co-ordinates 474,000 E and 4,488,000 N of the Hellenic Geodetic Reference System HGRS ‘80, Ellipsoid GRS80 (approximately Latitude 40° 36’ E and Longitude 23°50’ N). Figure 4-1 shows all the concessions of the Kassandra mines complex in beige, with the Olympias property highlighted in red.

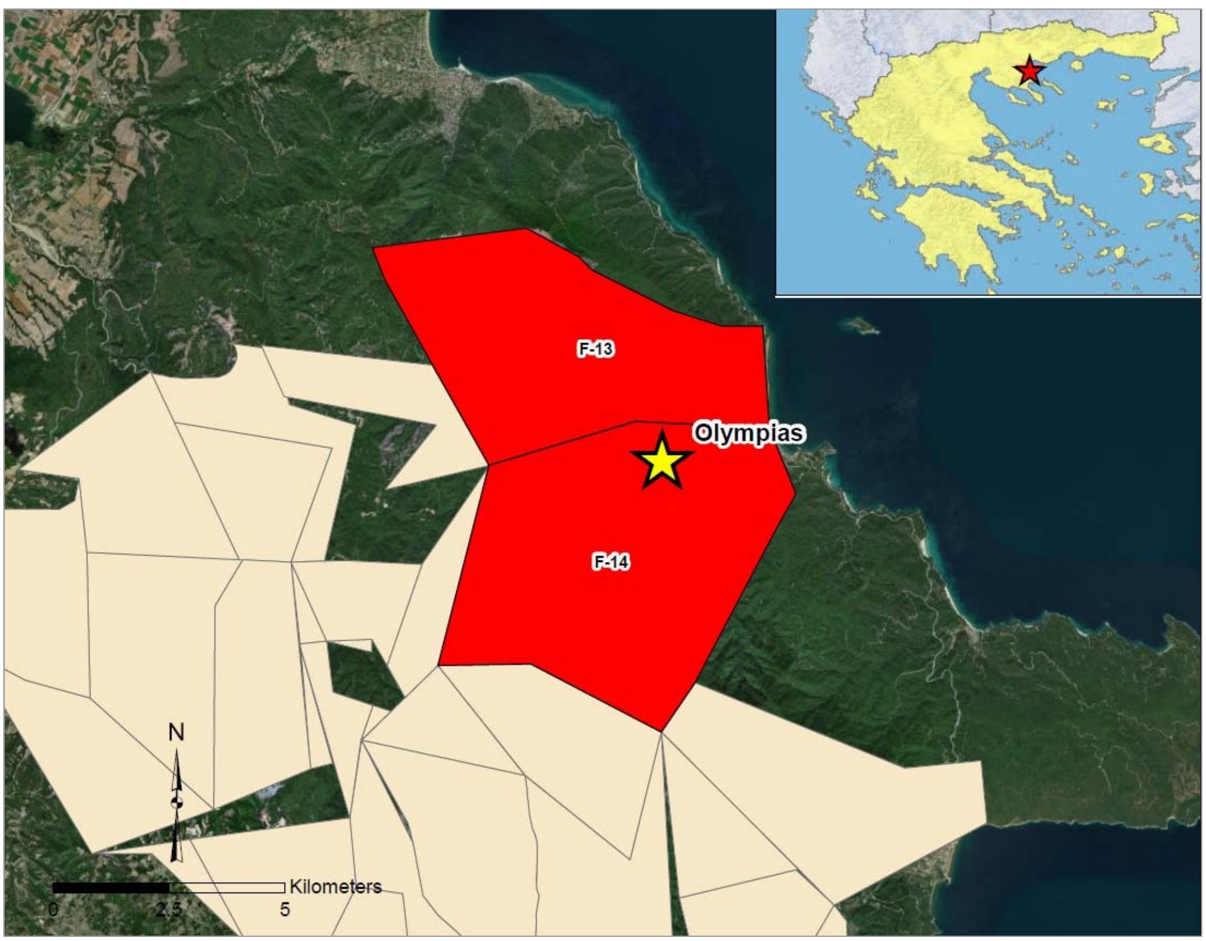


Figure 4-1: Property Concessions

The Property is owned by Hellas Gold, with Eldorado holding a controlling interest of 95% in Hellas Gold.

4.2 LAND TENURE

The Project is located within concession numbers F13 and F14, which have a combined area of 47.27 km^2 (Table 4-1). Hellas Gold has been granted mining rights over these concessions until April 7, 2024. The concessions are conditionally renewable for a further two consecutive periods of 25 years each. Hellas Gold has ownership of a small portion of private land within the concessions.

Table 4-1: List of Concessions

Concession #	Title holder	Tenement type	Area (km^2)	Expiry date
F13	Hellas Gold	Mining Concession	19.47	April 7,2024
F14	Hellas Gold	Mining Concession	27.80	April 7, 2024
Total			47.27	

4.3 PERMITTING

In July 2011, the Ministry of Environment (MOE) formally approved the Environmental Impact Statement (EIS) submitted by Hellas Gold for the three Kassandra Mines mine sites, being Olympias, Skouries and Stratoni, which involves an area of 26,400 ha, in north-eastern Halkidiki (Macedonia Region). This EIS is valid for 10 years and subject to renewal in 2021.

For production to commence, the MOE required the submission of a technical study. A study was submitted to the MOE and approved in early 2012. The installation permit for what was termed the Phase II process plant was issued on 22 March 2016. Installation work was completed in May 2017, at which time commissioning and trial production commenced. The Company received the operating permit for the Phase II plant in September 2017, allowing commencement of commercial production operations. In September 2017, the Company also received an extension of the installation permit and an interim operating permit for the Kokkinolakkas tailings management facility (TMF), as well as the delayed installation permit for the paste backfill plant.

Hellas Gold has provided a €50.0 M Letter of Guarantee to the MOE as security for the due and proper performance of rehabilitation works in relation to the mining and metallurgical facilities of the Kassandra mines project, and the removal, cleaning and rehabilitation of the old disturbed areas from the historic mining activity in the wider area of the project. Additionally, a Letter of Guarantee to the MOE, in the amount of €7.5 M, has been provided as security for the due and proper performance of the Kokkinolakkas TMF.

Notifications for the Operation of the Olympias paste plant and Kokkinolakkas TMF were formally submitted in 2018 and remain in force in line with new legislation that replaced previous operating permit issuance procedures.

4.4 ROYALTIES AND TAXES

Based on current Greek legislation, royalties are applicable on active mining titles. The royalty is calculated on a sliding scale tied to metal prices. At the price index selected for the Project evaluation, Hellas Gold would pay a royalty of approximately 2% on Au, 1% on Ag, 0.5% on Pb, and 1% on Zn revenues.

The corporate income tax rate for Greek companies in 2018 was 29%. This is legislated to drop by 1% per annum until it reaches 25% in fiscal year 2022.

4.5 ENVIRONMENTAL LIABILITIES

The closure and environmental rehabilitation activities for Kassandra Mine Mineral Deposits Project relate to the following facilities:

- Open pit and underground mine
- Integrated waste management facility (IWMF)
- Process facilities and infrastructure

To meet the requirements of the reclamation program, decommissioning, closure, and reclamation of the affected areas must be undertaken.

Hellas Gold has provided €57.5 M in Letters of Guarantee to the MOE as security for the due and proper performance of rehabilitation works in relation to the mining and metallurgical facilities of the Kassandra Mines project and the removal, cleaning and rehabilitation of the old disturbed areas from the historic mining activity in the wider area of the project.

Hellas Gold has also provided insurance coverage in accordance with Presidential Decree 148/2009 (Government Gazette 190/A/29.9.2009) for environmental liability.

4.6 OTHER

To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property. A permit is required from the Greek Government to extend the permitted operating life of the current Olympias facility as well as a permit to increase annual throughput to 650,000 tpa.

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

5.1 LOCATION AND ACCESSIBILITY

The Property is located in the Halkidiki peninsula in the Northern Macedonia region about 100 km by road from Thessaloniki, the second largest city in Greece. Thessaloniki has one of the largest ports in Greece and an international airport (Figure 5-1).

The Property is readily accessible year-round by the national road network with paved secondary roads within 1 km of site. The national road network in the area is among the best in northern Greece, with a major highway (E90), extending east from Thessaloniki to approximately 15 km north of the property. Olympias lies 9 km north-northwest of the Stratoni port and loading facility, accessed by a paved road along the coast, and paved mine road to site.

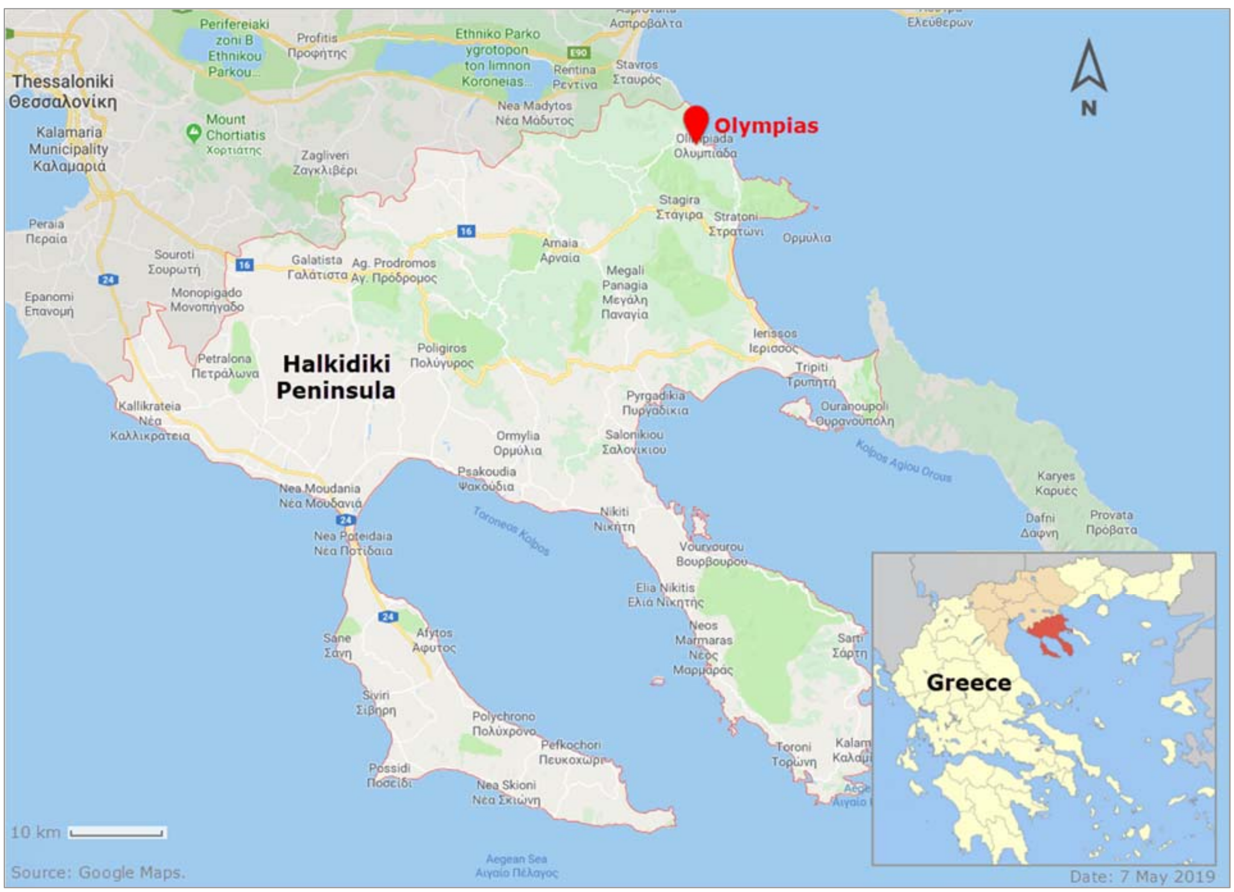


Figure 5-1: Location of the Olympias Property

5.2 INFRASTRUCTURE AND LOCAL RESOURCES

The area is well served by main 115 kV power from the Public Power Corporation (PPC), with power supply right to the mine site. Communications are good. Telephone and broadband are available in the area. Hellas Gold has the existing microwave link at Stratoni, giving additional access to broadband data communications.

Process water is reclaimed from the tailings thickening and filtration circuit and backfill clarifier overflow. A minimum amount of make-up water is supplied from the raw water tank. Potable water is produced from a borehole located in the regional aquifer.

Skilled labour, some of whom worked on the mine historically is available in the village of Olympiada, which is located about 1 km from the mine. Other nearby villages also have a pool of skilled and semi-skilled labour available. The company owns the all the land on which existing and proposed mine infrastructure is located, and this is sufficient to allow for future operation of the mine as designed, without further land acquisition.

The mine site comprises mine offices and change rooms, trackless decline, shaft, underground development, and processing consisting of crushing plant, mill, flotation plant, and backfill plant.

Figure 5-2 shows the location of the Olympias operations and the Kokkinolakkas TMF, which is on the Stratoni property and common to both operations.

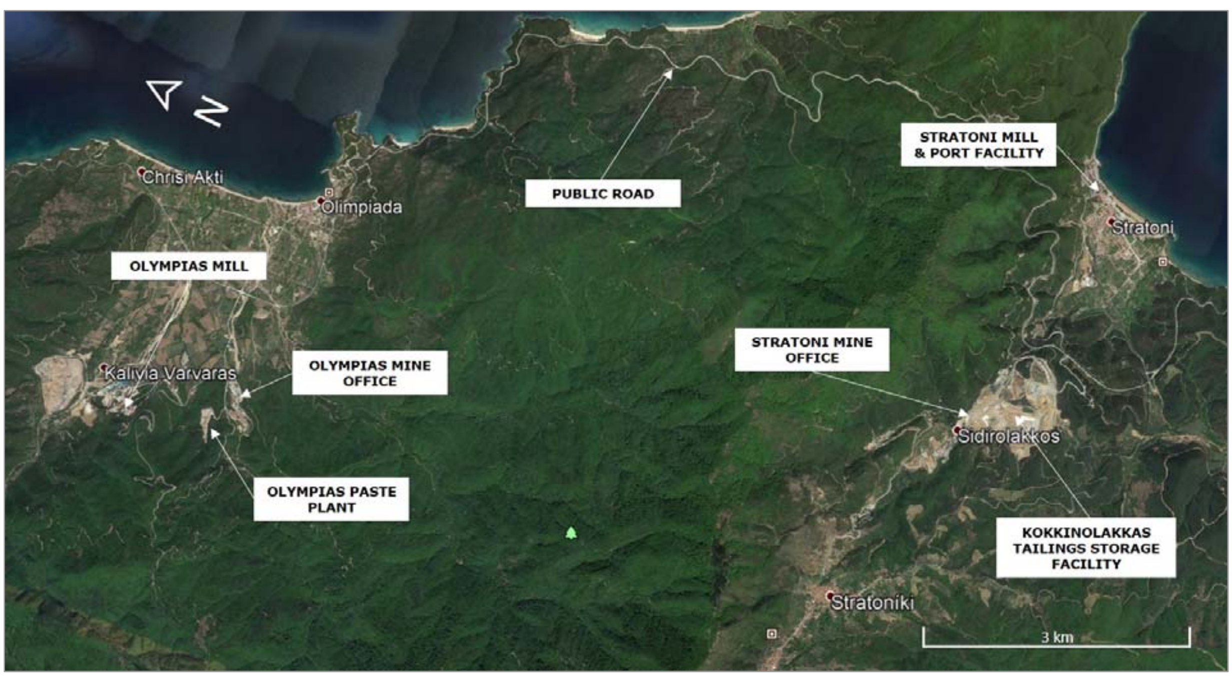


Figure 5-2: Olympias Site Layout

5.3 CLIMATE AND PHYSIOGRAPHY

The Halkidiki Peninsula climate is generally mild with limited rainfall. Over 300 days or around 3,000 hours of sunshine are recorded annually. Average temperatures have limited fluctuations during the year. The lowest temperatures occur during December to February, ranging between 3.5°C to 19°C, while the highest temperatures occur during summer months, ranging between 23°C and 34°C. Temperatures below 0°C are limited to the mountainous areas. Rainfall varies from 19 mm to 40 mm in a month, with October to December the wettest months. Operations can continue all year round.

The terrain is characterized by hills rising to about 600 metres above sea level (masl) with steeply incised valleys.

The area is wooded with oak, beech and pine being the principal species, while inland there are vineyards and farmlands. The main farming products are grapes, honey, and olives.

5.4 SURFACE RIGHTS

Hellas Gold owns the land containing the entire surface infrastructure associated with the Olympias Project. This is sufficient to allow the future operation of the mine without further land acquisition.

SECTION • 6 HISTORY

6.1 INTRODUCTION

There is a long history of mining in the area. Ancient mining reached a peak during the time of Philip II of Macedon and Alexander the Great, in the period 350 to 300 BC. The lead-rich ores from the Madem Lakkos mine at Stratoni were smelted for silver and the Olympias ores were processed for their high gold content. It has been estimated, from the volume of ancient slags that about 1.0 Mt of ore were extracted from each locality during this period. It is believed that by 300 BC, the bulk of the ores above the water table at Olympias had been exploited, though the Stratoni mine continued in production through the Roman, Byzantine, and Ottoman periods.

6.2 OWNERSHIP AND WORK CARRIED OUT

Milestones in the history of the Property are shown in Table 6-1.

Table 6-1: Summary of the history of the Property

Year	Commentary
Historic times	Bulk of ores at Olympias above water table were extracted by 300 BC.
1933	Shaft sunk to 74 m depth with some drifting.
1954	Owners commenced exploration; thin, discontinuous sulphide lenses encountered (and many ancient workings).
1965-66	Further drilling intersected 10 m of lead-zinc mineralization 20 m below the 1933 shaft.
1970	Ownership transferred to Hellenic Fertilizer Company; ramp was started, and production commenced in West Orebody.
1974-84	Mine was developed to mine lead and zinc. Shaft was sunk to the -312 m level; high grade mineralization of East orebody intersected; highly profitable mining using sub-level caving; eventual transition to less profitable drift-and-fill mining due to excessive dilution, ground subsidence and water problems.
1991	Hellenic Fertilizer Company went into receivership; mine continued production under subsidy from Greek government.
1995	Ownership transferred to TVX Gold Inc. (TVX); production suspended to allow for drilling to define mineral resources.
1998-99	TVX completed drilling campaign (760 holes, 91,319 m) and issued a Mineral Resource estimation; initial feasibility study completed.
2004	Aktor acquired mining concessions holding 317 km^2, including the Olympias and Skouries, deposits together with Stratoni (the Kassandra Mines) through its subsidiary Hellas Gold. The Hellas Gold acquisition of the Kassandra Mines was ratified by parliament and passed into law in January 2004 (National Law no. 3220/2004).
	European Goldfields (EGL) acquired its initial ownership percentage interest in Hellas Gold from Aktor through its wholly owned subsidiary European Goldfields Mining (Netherlands) B.V.

Year	Commentary
2007	EGL increased share ownership of Hellas Gold to 95% (with 5% held by Aktor).
2011	EIS approved by Greek government.
2012	Eldorado acquired the project via the acquisition of EGL. Commenced tailings re-treatment and rehabilitation of the underground mine shortly thereafter.

6.2.1 Work Pre 1970

In 1933, a shaft was sunk to 74 m depth at Olympias by the then owners, with lateral development to intersect a marble / gneiss contact. The drift encountered ancient workings, which were largely exhausted.

In 1954, the owners commenced exploration in the Olympias area. Their drilling encountered numerous discontinuous lenses of sulphide mineralization 10 cm to 20 cm in width, and many old workings were intersected. In 1965, further drilling commenced and, in 1966, a 10 m intersection of lead-zinc mineralization was encountered 20 m below the 1933 shaft, which proved to be exploitable.

6.2.2 Hellenic Fertilizer Company 1970 – 1991

By the early 1970s ownership of the mines had transferred to the Hellenic Fertilizer Company, (HFC). A ramp was started in 1970 and full production commenced in the West orebody. From 1974 to 1984 the shaft was sunk to 312 m below mean sea level (msl) and high-grade mineralization, termed the East orebody, was intersected at 254 m below msl. The first ten years of production proved highly profitable when sub-level caving (SLC) could be used, but excessive dilution, ground subsidence and problems with water influx resulted in a change to more expensive drift and fill (DAF) mining. During this period lead and zinc concentrates were sold and gold-bearing pyrite-arsenopyrite concentrate was stockpiled. The stockpile of gold bearing pyrite-arsenopyrite concentrate was sold to off-takers by EGL.

HFC carried out extensive programs of surface and underground drilling in order to define orebody dimensions and to explore the surrounding area. Partial logs are available for this work but none of the original cores are available in labelled boxes; none of the holes were surveyed and no assays, certifiable or otherwise, have been found. It is believed that ore was identified solely by visual assessment of the core. Where available, the partial logs of this work were entered by EGL into the database but only in order to guide exploration work and for use in the modelling of major geological units.

HFC went into receivership in 1991 and the Government subsidized the operation until development and production fell behind schedule. Following three separate international auctions, TVX successfully won the bidding for the Property in 1995.

6.2.3 TVX 1995 – 1999

Shortly after TVX obtained control of the Property, production at the Olympias Project was suspended so that drilling could be conducted to confirm and expand the Mineral Resources.

TVX commenced an intense program of drilling in 1996. An interim Mineral Resource assessment was completed in June 1998. By February 1999, TVX had completed a drill program comprising 760 holes totalling 91,319 m. The Mineral Resource was updated in December 1998 to incorporate all drilling completed to that date. In 1998 and 1999, Kvaerner Metals produced a feasibility study on the Olympias Project. During 1999 - 2000, SNC Lavalin of Toronto was commissioned to conduct a basic engineering study on the Olympias Project.

6.2.4 European Goldfields 2004 – 2014

The Property was acquired by EGL (through its 95% owned subsidiary Hellas Gold) in 2004. No exploration was carried out by EGL; however, sufficient reviews of existing data allowed EGL to file the pre-existing NI43-101 Technical Report with an effective date of July 14, 2011 (Technical Report on the Olympias Project Au Pb Zn Ag Deposit, Northern Greece, 2011).

SECTION • 7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Western Tethyan orogenic belt in southeast Europe contains several major metallogenic provinces including the Serbo-Macedonian Metallogenic Province (SMMP) that hosts the Kassandra mining district in northeast Greece (Janković, 1997; Baker 2019). The Western Tethyan orogen comprises a series of magmatic belts encompassing Cretaceous to Paleogene subduction related arc magmatism through to post-collisional Neogene magmatism (Richards, 2015; Moritz and Baker, 2019). In northeast Greece the orogeny developed during the Late Cretaceous to early Eocene as the Serbo-Macedonian Apulian and Pelagonian microcontinents converged on the previously accreted Rhodope continental fragments at the Eurasian margin (Pe-Piper and Piper, 2006). Crystalline basement within the Kassandra mining district includes the upper litho-tectonic Serbo-Macedonian Vertiskos unit and the lower litho-tectonic Kerdilion unit exposed within the southern Rhodope metamorphic core complex (Figure 7-1).

The SMMP forms a northwest-trending zone of base and precious metal deposits including a large Au-endowment (~25 Moz; Baker, 2019) that is associated with Oligocene to Miocene magmatic complexes including porphyry (Skouries, Greece; Illovitza, Bucim, North Macedonia; and Tulare, Serbia) and carbonate replacement deposits (Olympias, Mavres Petres, Madem Lakkos and Piavitsa in Greece), as well as the Plavica high sulfidation epithermal deposit (North Macedonia). The mineral deposits formed during post-collision extension and emplacement of intermediate to felsic magmas with high K calc-alkalic to shoshonitic composition and localized ultra-potassic mafic magmas (Borojevic Sostaric et al., 2012; Siron et al., 2016). The heterogeneity of the Cenozoic magmas likely resulted from crystal fractionation, assimilation and mixing of melted depleted mantle metasomatized by Cretaceous subduction processes, and partial melting of lower crustal rocks.

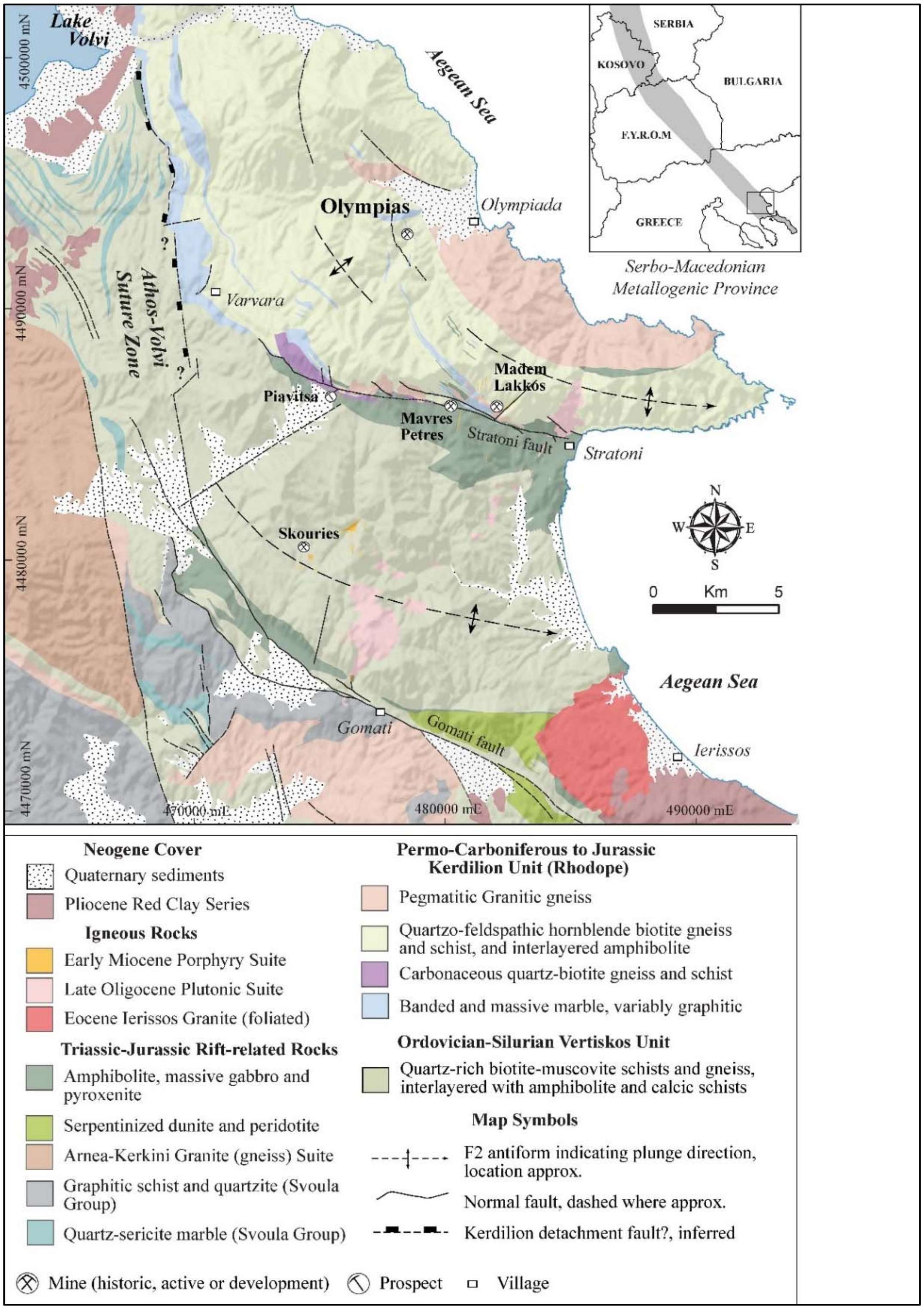


Figure 7-1: Regional Geology of the Kassandra Mining District (modified from Siron et al, 2018)

7.2 LOCAL GEOLOGY

The Kassandra mining district contains a geological endowment of > 8 Moz of gold in measured and indicated resources at the Olympias Mine (14.4 Mt @ 8.2 g/t Au, 138 g/t Ag, 4.7% Pb and 6.2% Zn; this report) and the Skouries deposit (289.3 Mt @ 0.58 g/t Au and 0.43 % Cu; Eldorado Gold Corp. Annual Information Form (AIF), March 29 2019, SEDAR website). Additional measured and indicated resources are reported for the Mavres Petres (Stratoni) Mine (0.8 Mt @ 185 g/t Ag, 7.2% Pb and 10.1% Zn; Eldorado Press Release November 15, 2019, SEDAR). The Stratoni fault is a major tectonic feature and important mineralizing corridor located in the center of the Kassandra mining district (Siron et al., 2016, 2018; Figure 7-1). The east-west oriented, ductile to brittle fault zone extends >12 km from the coast at Stratoni to the village of Varvara in the west. It is a moderately south-dipping normal fault that separates gneiss and marble of the Kerdilion unit to the north from amphibolite and schist of the Vertiskos unit to the south (Figure 7-1). The historically mined Madem Lakkos deposit, the currently mined Mavres Petres deposit, and the undeveloped Piavitsa deposits are hosted dominantly in marble within and adjacent to the fault. The fault zone crosscuts the lower portion of the late Oligocene (25.4 ± 0.2 Ma) Stratoni granodiorite stock but is cut by a Miocene glomerophyric monzonite porphyry dike at Piavitsa (20.62 ± 0.13 Ma) constraining latest major fault movement and related hydrothermal mineralization to the late Oligocene to early Miocene (Siron et al., 2016). The Olympias deposit occurs approximately 6km north of the Stratoni fault with sulfide ore occurring within marble layers of the Kerdilion unit adjacent to two major normal faults, the Kassandra fault and the East fault. Crosscutting structural relationships and 40Ar/39Ar geochronology on ore-related muscovite alteration (22.6 ± 0.3 Ma) indicate that carbonate replacement mineralization at Olympias also occurred during the latest Oligocene, coincident with the early stage of post-deformation magmatism in the region (Siron et al., 2018).

Metamorphic rocks of the Kerdilion unit consist of quartzo-feldspathic hornblende-biotite gneiss, marble, amphibolite, localized bodies of megacrystic plagioclase-microcline orthogneiss, and granite gneiss (Kalogeropoulos et al., 1989; Gilg and Frei, 1994). The lithologies have an arcuate geometry, striking in a north-south direction in the north and becoming east-west near the Stratoni fault (Siron et al., 2016). Middle Jurassic to Early Cretaceous zircon U-Pb and Pb-Pb ages from granitic gneisses of the Kerdilion unit range from 164-134 Ma and are interpreted as primary igneous ages (Himmerkus et al., 2011). The host rock sequence, however, is likely Carboniferous to Permian based on inherited zircon cores derived from the orthogneisses (Himmerkus et al., 2011). Pegmatitic dikes and sills occur throughout the Kerdilion unit and represent anatectic partial melting of the metamorphic rock from about the middle Paleocene to middle Eocene (Wawrzenitz and Krohe, 1998; Kalogeropoulos et al., 1989). The pegmatites are largely absent south of the Stratoni fault.

Graphitic garnet-bearing quartz-biotite gneiss and schist are spatially associated with the Stratoni fault zone and amphibolite occurs within the hanging wall with variably serpentinized pyroxenite. The Vertiskos unit occurs south of the Stratoni fault and hosts the Skouries porphyry deposit. The unit is a monotonous sequence of quartz-rich feldspathic to muscovite-biotite-bearing gneiss and schist. Minor calcareous schist, marble, and amphibolite are also thinly interlayered within the metamorphic sequence. Zircon U-Pb ages show that the micaceous schist ranges from Neoproterozoic (686-576

Ma) to Ordovician (464-450 Ma), which is consistent with the age of the Pan-African Pirgadikia and Vertiskos units of the Serbo-Macedonian terrane (Himmerkus et al., 2007).

Cretaceous to mid-Eocene ductile deformation accompanied by lower amphibolite-grade metamorphism and overprinting retrograde greenschist metamorphism affected the Kerdilion and Vertiskos units. A regionally prominent penetrative shallow dipping S1 foliation is defined by alignment of peak metamorphic minerals (e.g., biotite or amphibole). Subsequent high-strain transposition resulted in tight to isoclinal F2 folds locally accompanied by axial planar S2 cleavage. A later lower-strain deformation event superimposed a spaced steeply dipping S3 foliation on the pre-existing fabrics. This event is associated with km-scale upright and open east-plunging F3 folds evident as district-scale antiforms in the footwall of the Stratoni fault (Siron et al., 2016).

A series of discrete magmatic events are recognized in the region including the Triassic Arnea granite suite (228 ± 5.6 Ma) within the Vertiskos unit, and Late Cretaceous to early Eocene (68 ± 1 Ma to 53 ± 4 Ma) subduction-related calc-alkaline granites at Ierissos, Ouranoupolis, and Grigoriou on the Athos Peninsula. These latter granitic intrusions exhibit a weak tectonic fabric suggesting emplacement during the waning stages of regional deformation. Post-collision Oligocene-Miocene magmatism coincided with the main mineralizing events in the Kassandra mining district. Late Oligocene magmatism ranges from early monzogabbro to monzodiorite to later-stage granodiorite (Siron et al., 2016). These intrusions typically display medium-grained equigranular textures through to porphyritic phases with crowded textures dominated by feldspar phenocrysts. Most unaltered late Oligocene intrusions are high-K calcalkaline, and the intrusions occur along a north-northeast trend defined by the alignment of igneous centers and orientation of dikes. A suite of early Miocene intrusions, including Skouries (~20 Ma; Hahn et al. 2012), have porphyritic textures and are quartz monzonite to syenite in composition. Phenocrysts are prismatic consisting of plagioclase and megacrystic K-feldspar, fine-grained euhedral biotite, and relict amphibole. Rounded quartz phenocrysts occur in minor abundance and K-feldspar and quartz comprise the groundmass with accessory zircon, magnetite, and pyrite. The quartz monzonite and syenite intrusions belong to the high-K calc-alkaline to weakly shoshonitic magma series, and their emplacement was controlled by pre-existing structures such as fold axial surfaces and faults.

SECTION • 8 DEPOSIT TYPES

8.1 DEPOSIT GEOLOGY

The Olympias deposit is located 6 km north of the Stratoni fault within the Kerdilion unit (Figure 8-1). Replacement-style sulphide orebodies are hosted by marble interlayered within a sequence of quartzo-feldspathic biotite-gneiss, amphibolite and plagioclase microcline orthogneiss (Kalogeropoulos et al., 1989; Siron et al., 2016; Figure 8-2). The massive sulphide orebodies plunge shallowly to the southeast for over 1.8 km (Figure 8-3), subparallel to the orientation of F2 fold hinges and a locally developed L2 intersection lineation (Siron et al., 2018). The locations of the sulphide lenses, however, are largely controlled by strands of the ductile-brittle Kassandra fault and East fault and sub-horizontal shear zones that occur between the two faults (Figure 8-2).

The sulphide ore bodies are grouped into three major domains: East Zone, West Zone, and Flats Zone (Figure 8-3). Two smaller sub-zones, the Remnants and North zones, are considered as part of the West Zone for the purpose of resource estimation. The East Zone ore occurs dominantly in the footwall to the steeply northeast-dipping East fault with the sulphide lenses hosted by marble at or below the upper gneiss-marble contact. The stratabound ore lenses dip shallow to moderately to the northeast and strike for ~ 1.1 km. Ore lenses typically pinch and swell with thickness ranging up to 10 m and widths up to 130 m. The West Zone ore bodies are controlled by the Kassandra fault and have a steep (~ 60 °) NE dip that shallows at depth as it merges into the Flats. The West Zone ore has a strike extent of ~ 1.2 km and is up to 25 m thick with a down dip extent up to 200 m. The Flats ore body extends to the east from the West Zone ore and dips shallowly to the NE, however, in detail it comprises a series of stacked lenses that also extend to the west into the footwall of the Kassandra fault. The Flats ore body has a strike extent of over 1 km and is locally up to 15 m thick, although more commonly ranges from 3 to 10m thick, and ore lenses are tens of metres wide to up to 100 m in places.

Sulphide mineralogy of the Olympias deposit consists of coarse-grained, massive and banded lenses dominated by variable amounts of sphalerite, galena, pyrite, arsenopyrite, chalcopyrite and boulangerite. Accessory minerals include pyrrhotite, marcasite, stibnite, mackinawite, enargite, freibergite, bournonite, geocronite, cubanite, bornite and covellite (Nicolaou and Kokonis, 1980; Kalogeropoulos and Economou, 1987; Kalogeropoulos et al., 1989; Siron et al. 2016). Secondary ion mass spectroscopy studies indicate that Ag and Au primarily occur in solid solution with their respective host minerals; Ag in galena and Au in both arsenopyrite and pyrite.

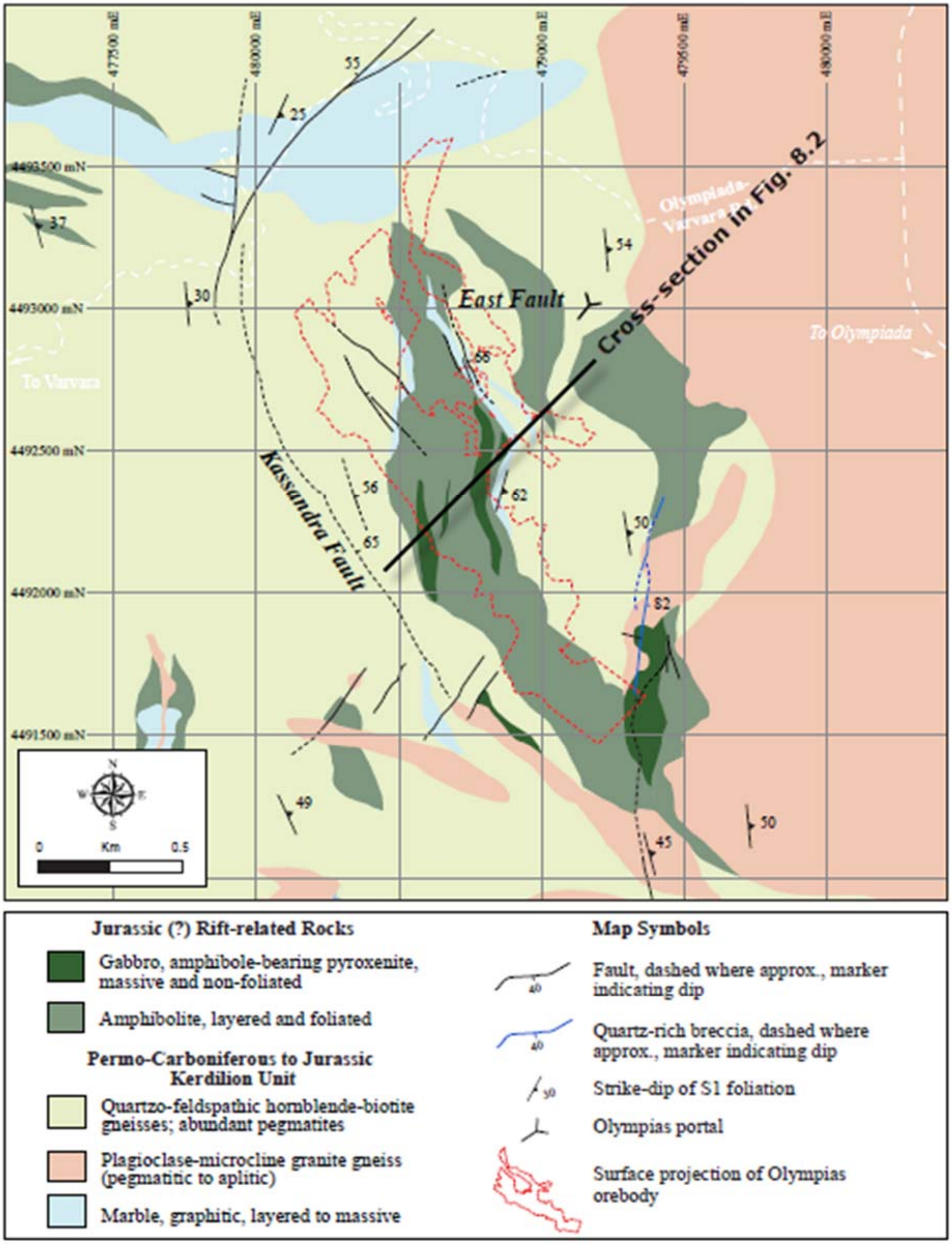


Figure 8-1: Geology of the Olympias mine area (modified from Siron et al., 2018)

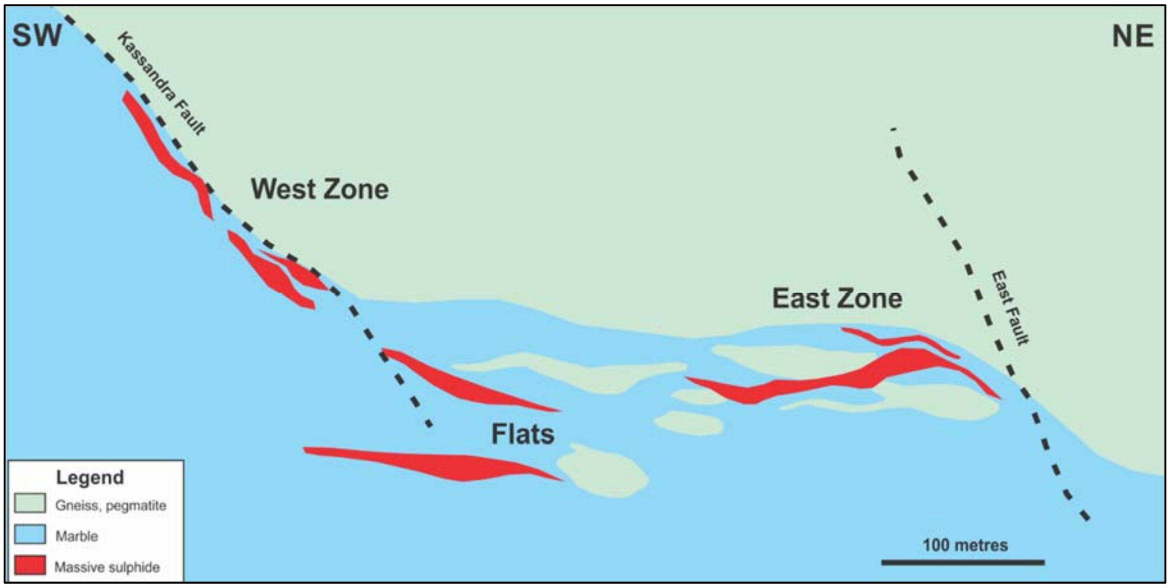


Figure 8-2: Simplified Geological Cross Section through the Olympias Deposit (facing NW)

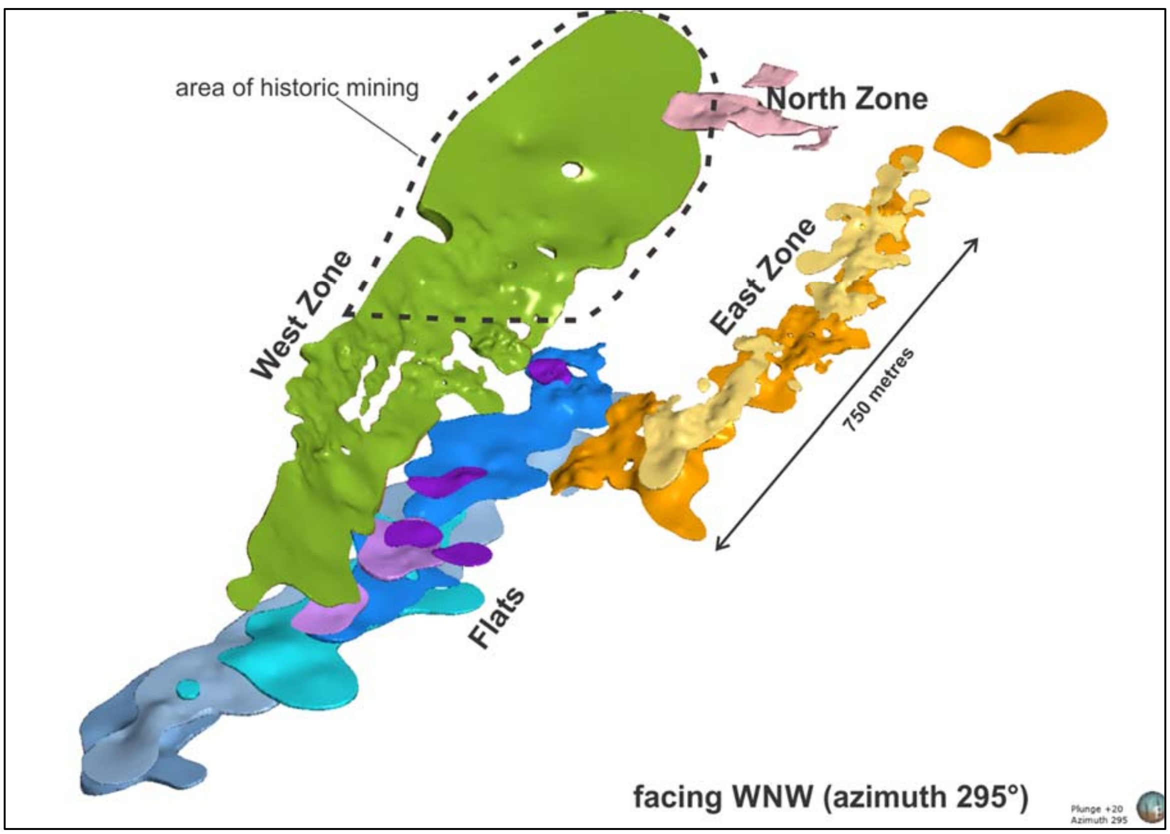


Figure 8-3: Three-dimensional oblique View of the Major Ore Bodies in the Olympias Mine looking west-northwest

Mine nomenclature classifies the mineralization into eight ore types. Ore types 1 to 3 are base metal-pyrite dominant, ore types 6 and 7 are arsenopyrite-rich and silica bearing, ore type 8 is manganese-rich and ore types 4 and 5 are sub-economic pyritic wall rock alteration.

Ore types 1, 2 and 3 are gradational and reflect end-members of galena-sphalerite dominant (Ore Types 1) to pyrite dominant (Ore Type 2) to transitional mixed galena-sphalerite-pyrite (Ore Type 3). Arsenopyrite is common in all three ore types but is not the dominant sulphide. The ore types typically occur as massive to banded sulphide zones with medium to coarse grained sphalerite-galena-pyrite-arsenopyrite and calcite gangue. Ore Types 1 to 3 are dominant in the Flats. Ore Type 7 is arsenopyrite-rich and has the highest gold content. The mineralization is typically siliceous with massive to banded sulphide dominated by blocky to acicular arsenopyrite with lesser pyrite, galena and sphalerite. Ore Type 7 is locally gradational to Ore Type 3, and banded zones commonly comprise intergrown Ore Types 1 to 3 and 7. Ore Type 7 is dominant in the East Zone. Ore Type 6 is a paragenetically younger quartz-rich sulphide assemblage that locally overprints the early replacement massive sulphide ore horizons. Ore Type 6 can vary from banded siliceous zones to extensive intervals of grey siliceous matrix breccia that contains angular altered wallrock fragments. These quartz-rich sulphide bodies consist of interlocking, euhedral and growth-zoned quartz accompanied by interstitial arsenopyrite and boulangerite with subordinate pyrite, galena and sphalerite. The breccia matrix consists of dark gray chalcedonic quartz containing disseminated, euhedral pyrite, fibrous boulangerite and bladed arsenopyrite. In places Ore Type 6 grades into Ore Type 7, and commonly these quartz-rich ore types are surrounded by lower grade quartz-rhodochrosite alteration of the marble (Ore Type 8). Ore Type 6 and 8 are most commonly developed in the northern and eastern portions of the Olympias deposit, and in and around the Kassandra fault.

8.2 DEPOSIT MODEL

Olympias is an example of a polymetallic carbonate replacement deposit, however it is somewhat unusual due to the high Au content of the deposit (cf. Siron et al. 2019). Key characteristics of this class of deposit include carbonate host rocks, massive sulphide mineralization, spatial and temporal relationship with magmatism and zoned metal distribution (Megaw, 1998). Geochronological and paragenetic relationships described by Siron et al. (2016, 2018) confirm that Olympias was coeval with late Oligocene (~ 25-22 Ma) magmatism in the region although the source intrusion has not been identified. Nonetheless, carbon and oxygen isotopic data support a magmatic fluid source (Siron et al. 2019), and the zonation towards Mn-rich alteration in the upper levels of the deposit (North Zone) are consistent with shallow, distal portions of a magmatically derived carbonate replacement system.

Similar carbonate replacement deposits of Oligocene age elsewhere in the West Tethyan belt include the massive sulphide deposits of Maden-Thermes district of southern Bulgaria and northeastern Greece, as well as the deposits in the Lavrion district of southern Greece and skarn and replacement sulphide deposits in the Trepca district of Kosovo (Siron et al., 2019 and references therein). However, for the most part these deposits are gold-poor. Global examples of Au-rich polymetallic carbonate replacement deposits include deposits in the Eureka (Nevada), Bingham (Utah) and Leadville (Colorado) districts of the USA.

SECTION • 9 EXPLORATION

Eldorado has not undertaken any significant exploration work at the Olympias.

There have been geological studies carried out on the deposit as follows:

- Surface geological mapping in the area surrounding the minesite.
- A study of the deposit geology and structural controls, and a separate petrographic study were carried out in 2013.
- In 2018 a review of the geological controls of the deposit was carried out. There was also a review of the logging database; this work simplified the logging codes to bring consistency to the process and recommended further studies on the alteration rock geochemistry.
- Some Terraspec (Hyperspectral) SWIR measurements have been carried out on core to measure alteration footprints to assist in future exploration targeting.

SECTION • 10 DRILLING

10.1 INTRODUCTION

Diamond drillholes are the sole source of subsurface geologic and grade data for the Olympias deposit. There is a large amount of historical drilling in the database. Data generated prior to the drilling by TVX (prior to 1996) was not utilized for Mineral Resource estimation purposes, as there is no means of validating that data.

The previous operator, TVX, drilled 764 drillholes for a total of 93,246 m.

A list of drillholes completed up to 30 June 2019 is shown in Table 10-1.

Table 10-1: Drilling by Year

Year	No. of Holes	Total Metres	Average Depth	Comments
Pre 2000	764	93,246	122	Pre-Eldorado drillholes -TVX only
2014	10	1,929	193	Eldorado drilling 2014-2019
2015	68	8,377	123	
2016	228	21,778	96	
2017	311	32,583	105	
2018	295	33,073	112	
2019	179	19,907	111	Up to 30 June
Total	**1,855**	**210,893**	**114**	

The core drilled by TVX was located on site in secure containers when Eldorado acquired the property. In 2012-13, Eldorado relogged over 70,000 metres, representing 75% of the TVX drill core, to gain an understanding of the deposit geology and to validate the geological model. The relogging effort facilitated the present ore type classification referred to in Section 8. Since 2014, delineation drilling has been concentrated on the East and West (Kassandra) Zones, in and around areas of mining. This drilling has been carried out predominantly on a 15 x 15 metre grid spacing; local areas have been drilled on a grid spacing of 10 x 10 metres where more detail was necessary. Areas covered by this delineation drilling provide the data used in updating the resource model, superseding the older TVX drill information. Use of TVX data remains in areas that are yet, undrilled by Eldorado.

Since 2014, drilling has been carried out by Greek and Canadian contractors drilling HQ or NQ-size core (63.5 mm or 47.6 mm nominal core diameter). The current contractors are Geotest-GAIA JV from Greece, who drill NQ core, and Cabo Drilling from Canada, who drill both HQ and NQ core. The average hole depth since 2014 has been around 100 m. Fans of drillholes are designed to maximize good intersection angles to the structure and mineralized zones. The mine geology team supervises the rigs and has weekly meetings with the contractors.

10.2 CORE HANDLING AND SAMPLING PROCEDURES

Drillers place the core into sturdy, locally-made, wooden core boxes. The driller keeps track of the drilling depth and places wooden marker blocks, later nailed in place, at the end of each run to indicate the depth from the collar. Core boxes are clearly labelled. The underground core is delivered to the core shed west of the offices on the mine site, and is logged in a secure area.

The core is logged in detail on tablet computers using the logging software Logchief™ from Maxgeo. Data are then uploaded into an acQuire database. Data collected includes lithology, alteration, mineralization (including ore types), RQD, core recovery, and other geotechnical factors for input into the Q rating system.

Core photos are routinely taken both on wet and dry core, using a camera stand to ensure consistent photographs.

10.3 SURVEYS

Mine surveyors set out the location of drill collars prior to the setup of the drill. A survey is taken at the actual collar once the drill is set up. Downhole surveys are also routinely taken using multishot instruments, either a Reflex Gyro™ or a Devico Deviflex. Calibration is done annually.

10.4 CORE RECOVERY

Core recoveries are recorded in the geotechnical logs for all drillholes. The overall average recovery is 88.6%, which is considered to be reasonable.

10.5 BULK DENSITY DETERMINATIONS

In 2013, Eldorado measured bulk density on over 900 samples of mineralization from historical drillcore using the Archimedes water immersion method (waxed samples) and incorporated the results into the Eldorado database. For a subset of 617 samples, for which there was a complete assay dataset that included Fe, a calculated density was derived as follows:

- All Zn from assay was assigned to sphalerite; the composition of Olympias sphalerite is assumed to be 86% Zn and 14% Fe based on an historical mineralogical study.
- All As from assay was assigned to arsenopyrite (along with a portion of the total Fe).
- All Pb from assay was assigned to galena. While boulangerite is known to be present in Olympias ores, it is not volumetrically significant enough to affect the densities to a great degree and is not included in the density calculation.
- All remaining Fe (after assignments of Fe to sphalerite and arsenopyrite) was assigned to pyrite.
- Mineral percentages for sphalerite, galena, arsenopyrite and pyrite were derived from the assay data and were added together to yield the total sulfide mineral content. Gangue mineral content was calculated by subtracting the total sulfides from 100%.

- The calculated bulk density was then derived by accumulating the proportion of each of the minerals multiplied by its mineral density (gangue was assigned a density of 2.69 based on measured data from unmineralized samples).

A comparison of the measured versus calculated bulk densities was carried out using a quantile-quantile plot. It was determined that most calculated densities were about 6% higher than measured and that this disparity was most likely due to voids that could not be accounted for in a calculated bulk density. Thus, each block in the resource model has a calculated bulk density (based on the interpolated metal grades for each block) that is then downwardly adjusted by 6% to reflect the typical volume of voids.

SECTION • 11 Sample Preparation, Analyses and Security

11.1 Introduction

A number of the samples used in the current resource estimates originated from the 1996 – 98 drill campaign carried out by TVX. The procedures for sampling, analysis and security are described in previous Technical Report on Olympias (2011). Eldorado has reviewed the report and data and agreed with that report's conclusions that those drill data are acceptable to be used for mineral resource estimation. The remainder of this section describes the sample preparation and quality assurance / quality control (QA/QC) results of the work carried out by Eldorado, in particular during 2018 and 2019.

11.2 Sample Preparation and Assaying

Drill core samples are routinely sent to the ALS facility in Romania. They are bagged and packed in large sealed wooden bins before being trucked to ALS. The sample rejects are returned to the mine site in the same bins. The samples are prepared for assaying at the ALS facility.

Upon delivery to the laboratory the samples were crushed to 90% passing minus 3 mm and prepared according to the following protocol.

- A 1 kg subsample was riffle split from the crushed minus 3 mm sample and pulverized to 90% passing 75 µm (200 mesh).
- A 200 g subsample was split off by taking multiple scoops from the pulverized 75 µm sample.
- The 200 g subsample was placed in a kraft envelope, sealed with a folded wire or glued top, and prepared for shipping. The rest of the pulverized sample was then stored in plastic bags.
- All equipment was flushed with barren material and blasted with compressed air between each sampling procedure. Regular screen tests were done on the crushed and pulverized material to ensure that sample preparation specifications were being met.

All samples were assayed for gold by 30 g fire assay with an AAS finish, with Au values above 10 ppm determined by a gravimetric finish. Multi-element determination was carried out by Inductively Coupled Plasma Mass Spectrometry (ICP-MS) analysis and / or Inductively Coupled Plasma Emission Spectroscopy (ICP-ES) analysis. (Table 11-1).

11.3 QA/QC Program

11.3.1 Overview

Assay results were provided to Eldorado in electronic format and as paper certificates. The QA/QC procedure included inserting either a CRM, blank and duplicate into the samples stream every 10th samples. Upon receipt of assay results, values for Certified Reference Materials (CRMs) and field blanks were tabulated and compared to the established CRM pass-fail criteria:

- Automatic batch failure if the CRM result is greater than the round-robin limit of three standard deviations.
- Automatic batch failure if two consecutive CRM results are greater than two standard deviations on the same side of the mean.
- Automatic batch failure for each element if the field blank or analytical blanks are over the values given in Table 11-2.

Table 11-1: Summary of Assay Methods

ALS	
Code	**Description**
Au-AA25	Ore grade Au 30 g FA AA finish
Au-GRA21	Ore grade Au 30 g FA AA finish
ME-ICPORE	Oxidizing digestion w/ ICP-AES finish

Table 11-2: Blank Failure Criteria for Au, Ag, Pb and Zn

Element	Field Blank results	Analytical Blank Results
Au (g/t)	0.2 >	0.01>
Ag (g/t)	20 >	1 >
Pb %	0.2 >	0.01 >
Zn %	0.2 >	0.01 >

If a batch failed, it was re-assayed until it passed. Override allowances were made for barren batches. Batch pass/failure data were tabulated on an ongoing basis, and charts of individual reference material values with respect to round-robin tolerance limits were maintained.

The number of samples, duplicates, CRMs and blanks are presented in Table 11-3 along with the compliance figures. The QA/QC compliance of above 5% in each case aligns with accepted norms. A total of 147 batches were sent to the lab during 2018 – 2019.

Table 11-3: Summary of QA/QC sampling in 2018 – 2019

	Original Samples	Blanks	Gold CRMs	Base metal CRMs	Total CRMs	Pulp dups	Coarse dups	Total dups
Number	8,847	653	299	332	631	557	512	1069
Compliance		7.40%	3.40%	3.80%	7.10%	6.30%	5.80%	12.10%

11.3.2 CRM Results

CRMs used represent all economic minerals at Olympias. For each economic mineral, there should be three corresponding standards:

- At around the expected cut-off grade of the deposit
- At the expected average grade of the deposit
- At a higher grade

In Eldorado's opinion, both the grade range of the CRMs used, and the percentage of CRM samples submitted are appropriate.

Table 11-4 summarizes the details of the CRMs that were inserted by Eldorado in 2018 and 2019.

Table 11-4: Summary of CRM expected values and standard deviations

Standard ID	Au		Ag		Pb		Zn		No. of Assays
	Expected Value (g/t)	SD	Expected Value (g/t)	SD	Expected Value (%)	SD	Expected Value (%)	SD	
G310-8	7.970	0.290							42
G903-9	11.260	0.447							39
G908-7	4.820	0.182							23
G909-6	0.560	0.014							30
G912-4	1.907	0.081							54
G915-4	9.160	0.350							26
G317-2	12.970	0.410							26
G915-6	0.674	0.042							28
G318-10	4.580	0.170							31
GBM310-14			59.6	2.50	8.9465	0.3355	17.9106	0.8084	84
GBM911-11			10.2	1.10	0.1722	0.0084	0.1422	0.0067	75
GBM913-11			32.5	2.30	3.5846	0.1541	7.8969	0.2849	74
OREAS 133a			99.9	2.42	4.9000	0.1620	10.8700	0.3540	41
OREAS 134a			201.0	7.00	12.7900	0.7660	17.2700	0.5530	30
Oreas 137			25.9	1.18	0.6730	0.0200	4.9200	0.0800	28

Figure 11-1 shows the results for a selection of the gold CRMs. They show good performance with variation about the CRM mean values within acceptable ranges. Note in each CRM figure below, that the red line indicates three times the standard deviation, the yellow line indicates two times the standard deviation, the black line represents the expected value and the blue dots are the results of the assayed samples which were sorted by assay date, and blue dashed line shows either 2 or 4 moving average of consecutive samples.

Figure 11-2 to Figure 11-4 shows a selection of plots showing results for the Ag, Pb, and Zn CRMs.

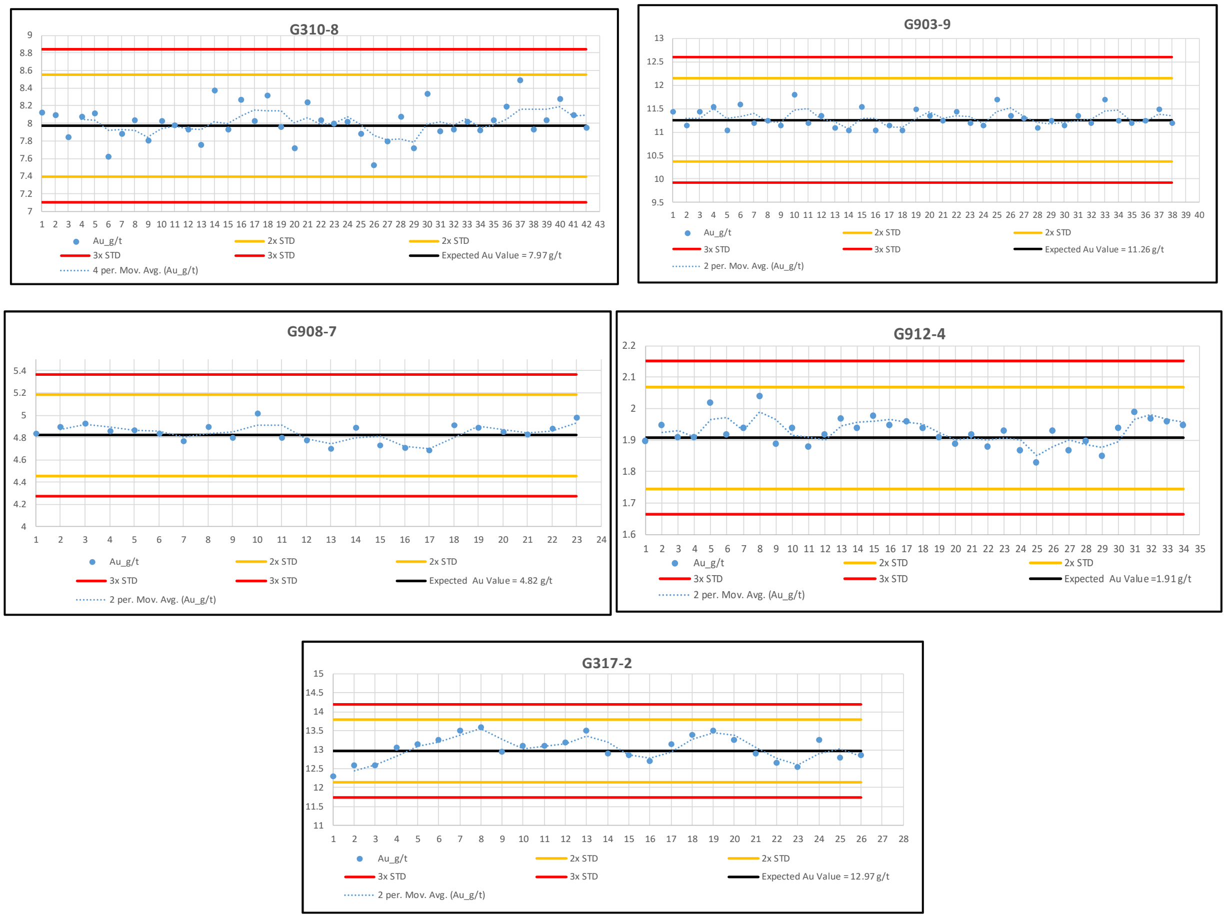


Figure 11-1: Gold CRM Charts, 2018-2019 Olympias Mine

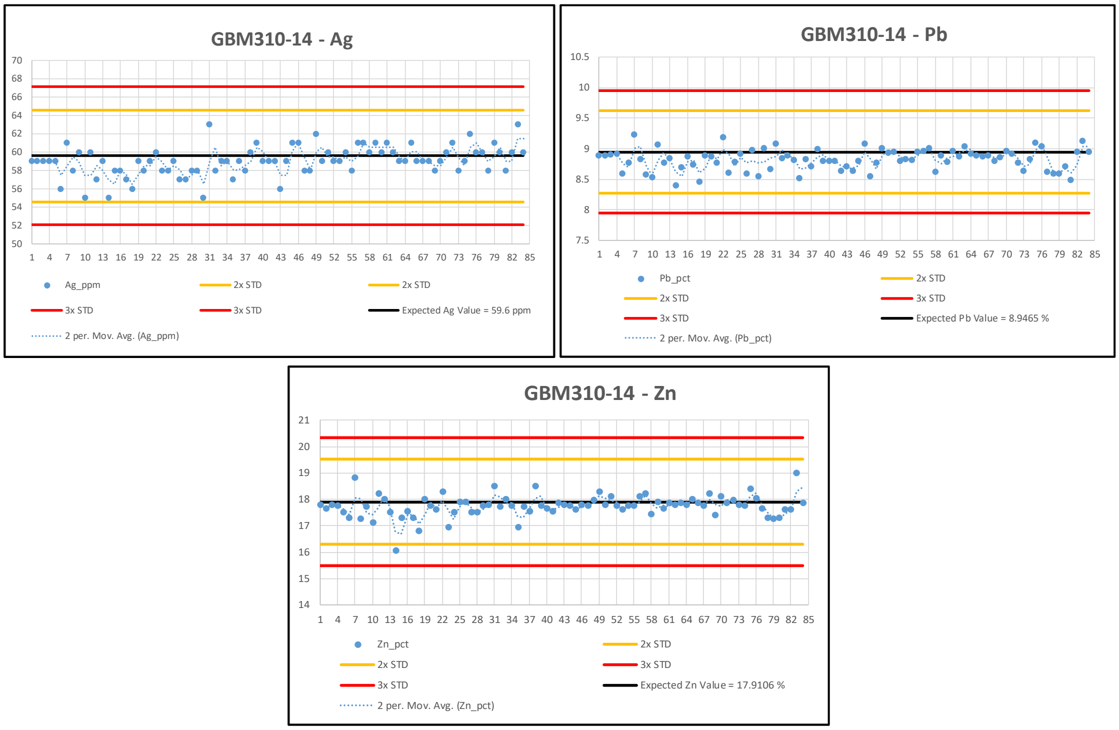


Figure 11-2: Ag, Pb, Zn GBM310-14 Charts, 2018-2019 Olympias Mine

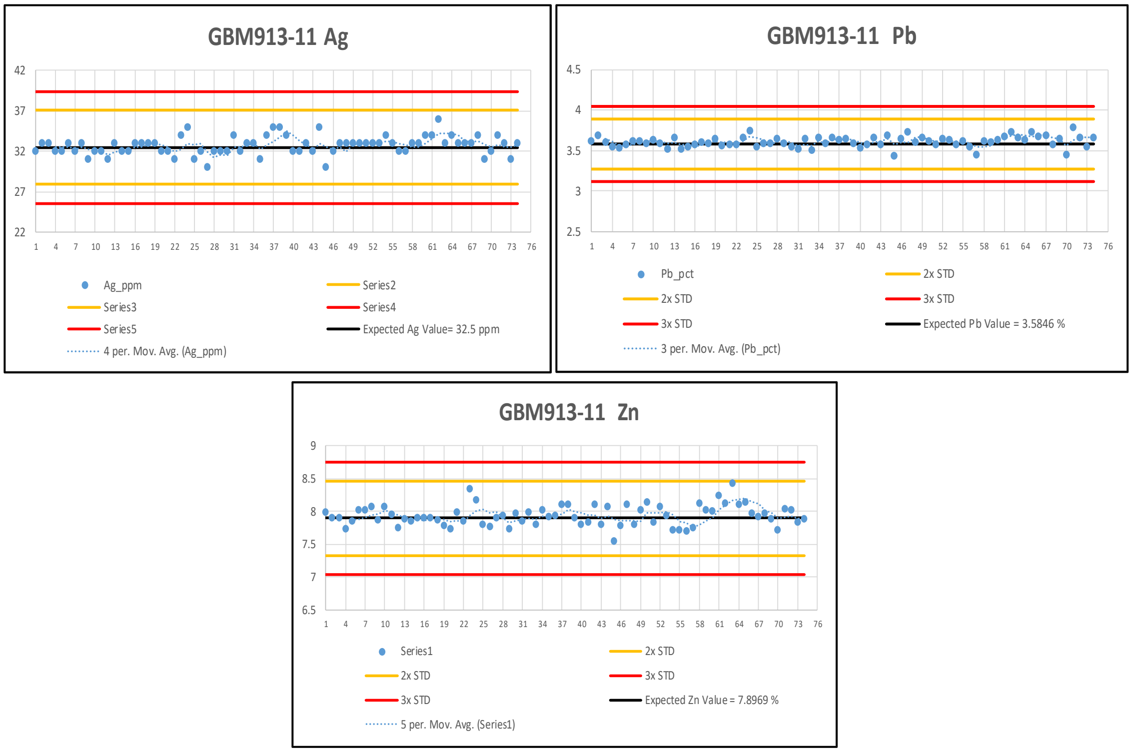


Figure 11-3: Ag, Pb, Zn GBM913-11 Charts, 2018-2019 Olympias Mine

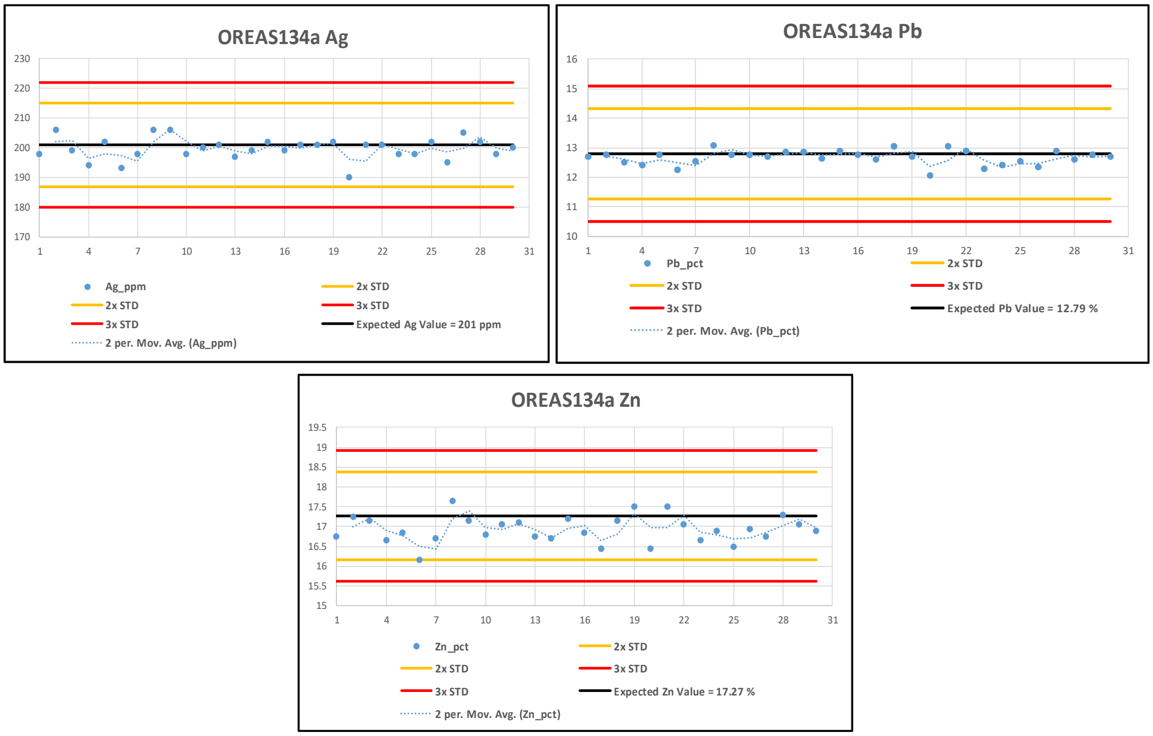


Figure 11-4: Ag, Pb, Zn OREAS134A Chart, 2018-2019 Olympias Mine

11.3.3 Blank Samples

Blank samples monitor for contamination during sample preparation and assay process. A total of 653 blank samples were submitted for Au, Ag, Pb, and Zn in 2018 - 2019. Figure 11-5 show the results of the blank samples for Au, Ag, Pb, and Zn, respectively No warnings or failures were identified in the blank samples.

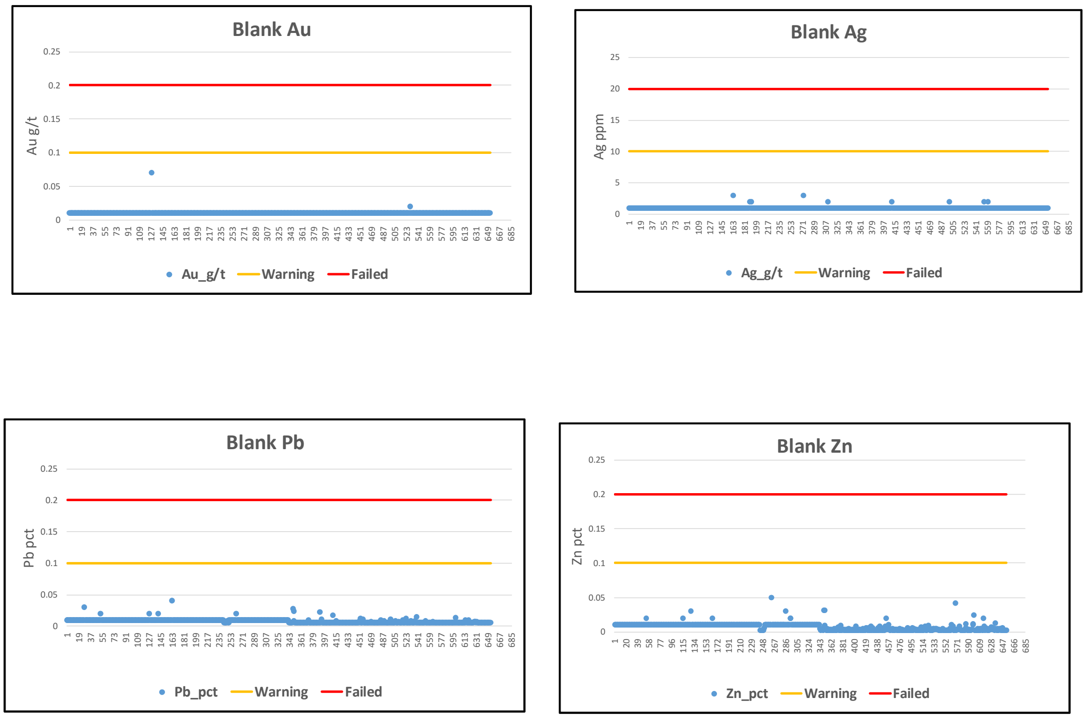


Figure 11-5: Au, Ag, Pb, Zn Blank Performance by Metal

11.3.4 Duplicate Samples

Eldorado implemented a program, which monitored data from regularly submitted coarse reject duplicates and pulp duplicates. Coarse, uncrushed duplicate samples monitor sampling variance, including that arising from crushing, analytical variance and geological variance. The pulp duplicates monitor the assaying procedure. In 2018 – 2019, 512 coarse duplicates and 557 pulp duplicates were submitted for analysis. The duplicate data for Olympias reproduce well and are shown in percentile rank or absolute difference charts for Au and Zn in Figure 11-6 to Figure 11-9. The coarse reject charts show that all data fall well within the 20% limits at the 90th percentile of the population (Eldorado's threshold for coarse reject duplicates). The pulp duplicate data shows similar compliance to its threshold of 10%.

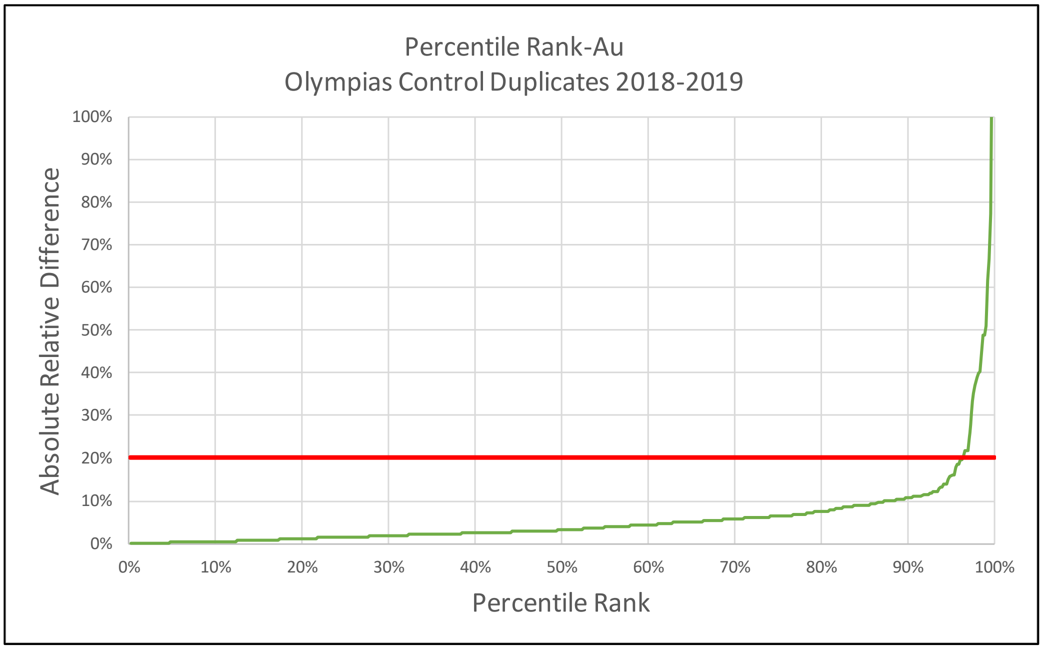


Figure 11-6: RPD plot for coarse duplicates Au

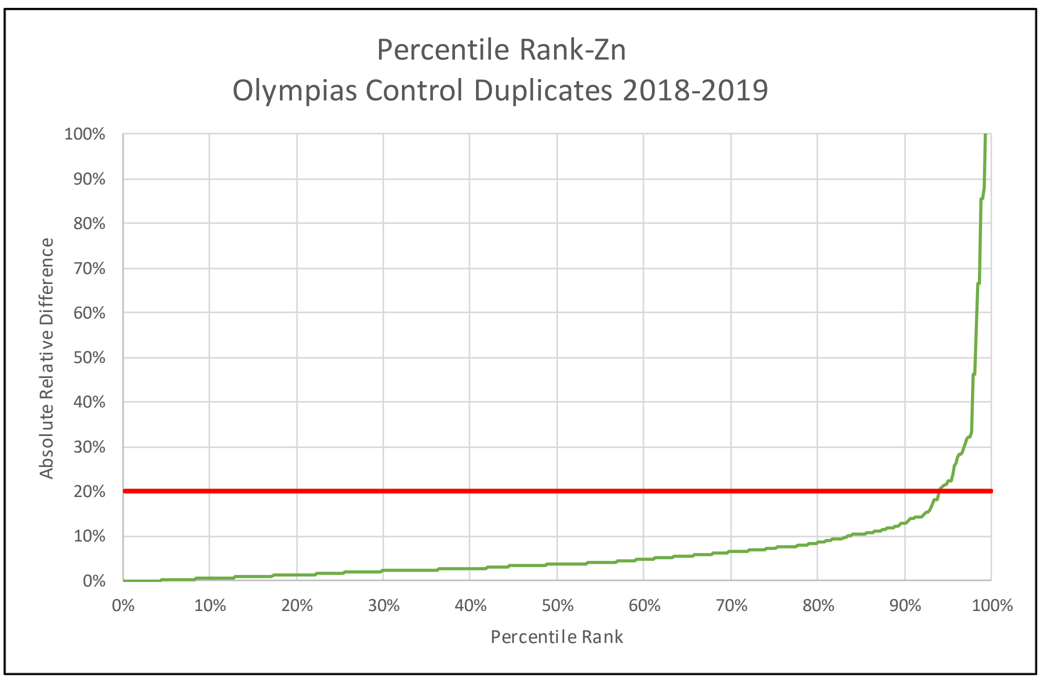


Figure 11-7: RPD plot for coarse duplicates Zn

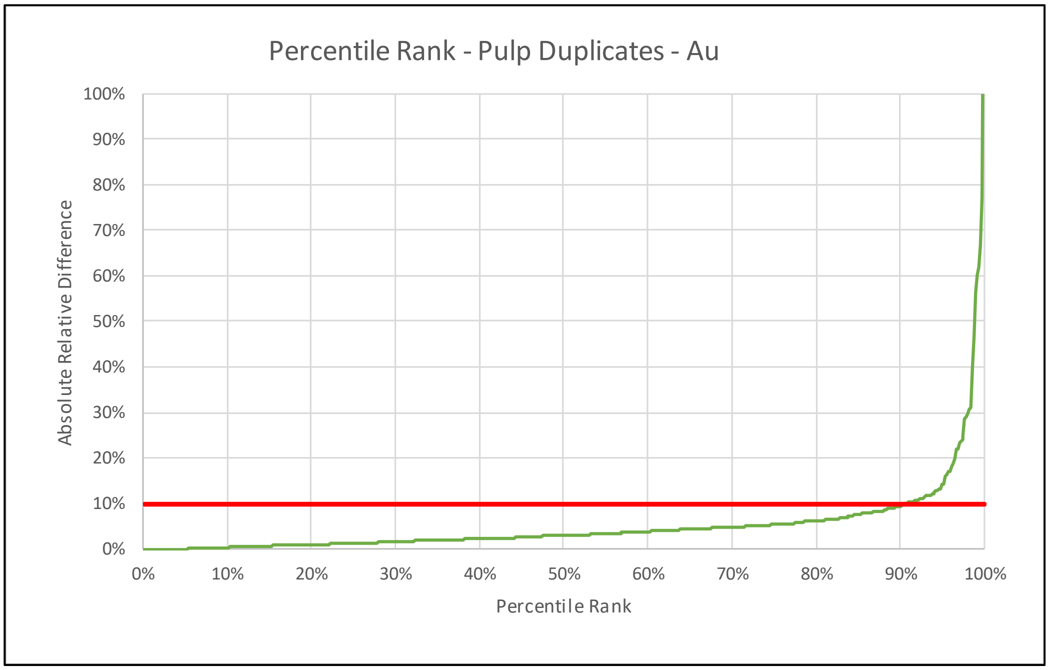


Figure 11-8: RPD plot for pulp duplicates Au

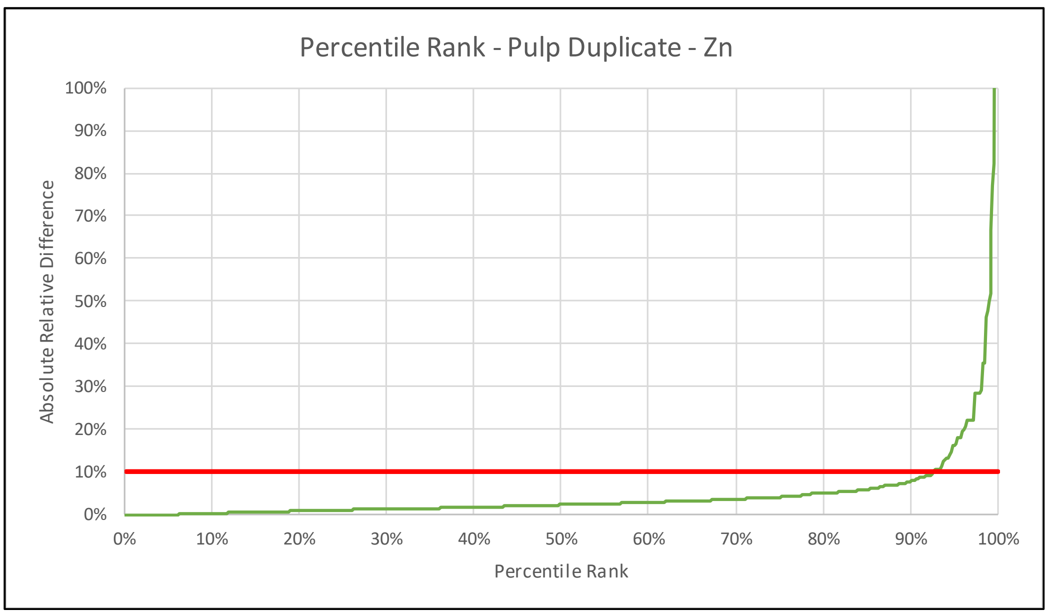


Figure 11-9: RPD plot for pulp duplicates Zn

11.4 CONCLUDING STATEMENT

In Eldorado's opinion, the sampling, sample preparation, security, and analytical procedures, as demonstrated by the QA/QC results, show that the Olympias mine's assay database, in particular for data since 2017, is sufficiently accurate and precise for resource estimation.

SECTION • 12 DATA VERIFICATION

12.1 INTRODUCTION

Data verification is regularly carried out by Eldorado, and comprises the following:

- Review of available data
- Review of QA/QC protocols and QA/QC performance
- Regular discussions and interviews with Olympias mine personnel regarding data collection and database compilation
- Regular site visits to the minesite since
- Cross checking of assays in the Eldorado sample database with original laboratory certificates
- Reconciliation of production to the block model estimates

Eldorado considers that these regular reviews adequately assess the Olympias database.

12.2 DATABASE CHECKS

Eldorado made checks of a selection of the original laboratory assay certificates against the database used for estimation. Eldorado has been operating at the site from 2014 but did some re assay of the historic drill core in 2013 as well. There is a total of 25,325 records in the database that were sent for analysis by Eldorado Gold as of end of 2019. 3,009 of these were standards and blanks and 1,716 duplicate records, these constitutes 11.9% and 6.80% of the total database respectively. As a result of these checks, data supporting the Olympias resource work are sufficiently free of error and adequate for resource estimation.

12.3 MODEL TO MILL RECONCILIATION

An important measure of performance at any producing mine is reconciliation of the block model to the final mill production figures, adjusted for stockpiles as necessary.

Reconciliation studies are carried out on site on a regular basis. Comparison is made between the depleted block model and the mill month-end figures. There is an adjustment for stockpiles that is applied to the mill figure, and dilution from multiple sources is taken into account when calculating the mine delivered tonnage.

For Olympias, the dilution is calculated from the difference between the mineralized wireframe and the as-built survey of the mined stope. The paste fill and waste contributions to the ore stream are then determined.

The first period, April 2017 to June 2018, tracked initial production. These results are shown in Table 12-1.The second, January to December 2019, represents the first year of regular quarterly reconciliation studies. Table 12-2 shows the summary of those results.

Table 12-1: Reconciliation for April 2017 to June 2018

	Tonnes	Au g/t	Ag g/t	Pb%	Zn%	Au oz	Ag oz	Pb tonnes	Zn tonnes
Mined	313,000	7.76	78	2.4	3.5	78,100	784,900	7,500	11,000
Mill	312,000	7.83	83	2.7	3.7	78,500	832,600	8,400	11,500
Mill / Mined	100%	101%	106%	113%	106%	101%	106%	112%	105%

Table 12-2: Reconciliation for January to December 2019

	Tonnes	Au g/t	Ag g/t	Pb%	Zn %	Au oz	Ag oz	Pb tonnes	Zn tonnes
Mined	285,000	7.78	89	2.8	4.1	71,300	813,600	7,900	11,700
Mill	288,000	7.30	92	2.8	3.7	67,700	854,600	8,200	11,100
Mill / Mined	101%	94%	104%	102%	94%	95%	105%	103%	95%

The reconciliation is detailed and thorough. It is currently providing a quarterly snapshot and demonstrating that the block model, and thus the mineral resources, are valid and robust. This validates the data underpinning the model and is, by association, a good verification of the work done.

12.4 CONCLUDING STATEMENT

The data collection by Eldorado is done in a diligent fashion. The data transfer to the database is by a mix of electronic and manual means, with many internal checks. The production reconciliation that is based on the data is very good. In Eldorado’s opinion, the data are adequate and verified for the purposes of the technical report.

SECTION • 13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 HISTORICAL METALLURGICAL TESTING

A number of extensive metallurgical reports were commissioned by TVX Hellas SA. The testwork confirmed that the ore responds well to flotation with high recoveries. Other metallurgical testwork and mineralogical investigations have been carried out on samples from the Olympias deposit. These are summarized in the following sections.

13.2 SGS METALLURGICAL TEST REPORT NO. 14607-001 – FINAL, 2015

In 2015, SGS Lakefield conducted a metallurgical test program on two samples originating from the Olympias Mine. Details and results of this test program were issued in a report titled "*An Investigation into Flotation Flowsheet Development of Two Samples from the Olympias Mine for HELLAS GOLD S.A. Project 14607-001 – Final Report, September 23, 2015*" (the 2015 Report).

The two samples for this test program were originally denoted as Fresh Ore and Old Ore from the Upper West Zone of the Olympias Mine. The objectives of the program were to define the ore characteristics and confirm the process flowsheet for restarting the mining operation. The scope of the testwork program included head characterization, grindability, flotation, solid-liquid separation, and tailings characterization. A key focus of the test program was to evaluate smelter penalty elements, arsenic and antimony, in the lead and zinc concentrates.

13.2.1 Sample Characterization

The head assays of Old Ore and Fresh Ore samples are summarized in Table 13-1.

The samples contained a significant amount of lead, zinc and sulphur. Of particular interest is the grade of arsenic, which was 10.5% for the Fresh Ore and 5.90% for the Old Ore. The gold content is related to the content of arsenic and, based on the ratio of Au/As, the Fresh Ore was classified from the Upper West Zone while the Old Ore was classified from the Upper East Zone.

Table 13-1: Old Ore and Fresh Ore Head Assays

Sample ID	Pb %	Zn %	Cu %	Fe %	Cd %	Sb %	Au g/t	Ag g/t	As %	[1]S= %	[2]S %
Old Ore	10.2	8.43	0.39	24.5	0.047	0.4	12.4	277	5.9	28.5	29.8
Fresh Ore	5.04	4.49	0.11	27.7	0.02	0.1	22.8	159	10.5	28.2	29.2
Sample ID	**Al g/t**	**Ba g/t**	**Be g/t**	**Bi g/t**	**Ca g/t**	**Co g/t**	**Cr g/t**	**K g/t**	**Mg g/t**	**Mn g/t**	**Mo g/t**
Old Ore	1780	24.5	0.06	<20	18600	<4	169	823	1110	1410	<5
Fresh Ore	1630	5.8	0.04	<20	4950	<4	201	766	360	689	<5
Sample ID	**Na g/t**	**Ni g/t**	**P g/t**	**Se g/t**	**Sn g/t**	**Sr g/t**	**Ti g/t**	**Tl g/t**	**U g/t**	**V g/t**	**Y g/t**
Old Ore	31	<20	<70	<30	108	10.9	46.2	<30	<50	<40	0.3
Fresh Ore	18	<20	<70	<30	163	3.94	51.9	<30	<50	<40	<0.2

Notes:
[1] Sulphide Sulphur
[2] Total Sulphur

13.2.1.1 Mineralogical Examination

Each of the Fresh Ore and Old Ore samples, after stage-pulverizing to 80% passing 120 µm, was screened into five size fractions (+106 µm, -106/+75 µm, -75/+45 µm, -45/+10 µm, and -10 µm) and submitted for mineralogical examination using Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEMSCAN).

Mineralogical modal analyses are summarized in Table 13-2 and Table 13-3 respectively. As evident from the head assays, each sample is dominated by sulphide minerals.

For the Fresh Ore sample, the sample is dominated by pyrite (37.3%), arsenopyrite (23.1%), and quartz (23.3%) with lesser amounts of galena (6.29%) and sphalerite (6.94%).

For the Old Ore sample, the sample contains pyrite (36.3%), arsenopyrite (12.5%), and quartz (16.9%), but contains higher quantities of galena (12.8%) and sphalerite (13.9%).

Calcite is present in fairly low quantities in the Fresh Ore (1.17%) and Old Ore (4.32%) samples, and all remaining minerals are present in contents less than 1%.

Table 13-2: Mineral Modals – Fresh Ore

Sample	Fresh Ore									
Fraction	Combined	+106 um		-106/+75 um		-75/+45 um		-45/+10 um		-10 um
Mass Size Distribution	**100**	27.5		18.1		21.2		21.8		11.4
Calc'd grain size	24	100		65		39		17		7
Mineral Mass (%)	Net	Net	Fraction	Net	Fraction	Net	Fraction	Net	Fraction	Net	Fraction
Pyrite/Marcasite	37.3	13.9	50.6	6.29	34.7	7.01	33.0	6.62	30.4	3.48	30.6
Arsenopyrite	23.1	5.47	19.9	4.97	27.4	5.61	26.4	4.75	21.8	2.32	20.4
Sphalerite	6.94	1.06	3.87	1.28	7.05	1.65	7.78	1.94	8.91	1.00	8.82
Galena	6.29	0.76	2.75	0.77	4.26	1.35	6.34	2.16	9.92	1.25	11.0
Chalcopyrite	0.32	0.06	0.22	0.04	0.21	0.05	0.24	0.07	0.34	0.09	0.82
Other-Cu-Sulphides	0.02	0.00	0.01	0.00	0.01	0.00	0.01	0.00	0.02	0.00	0.04
Boulangerite	0.11	0.02	0.07	0.02	0.09	0.02	0.11	0.02	0.09	0.03	0.30
Fe-Oxides	0.09	0.01	0.02	0.01	0.04	0.01	0.04	0.02	0.10	0.04	0.38
Quartz	23.3	5.89	21.4	4.44	24.5	5.12	24.1	5.39	24.8	2.50	22.0
Plagioclase/K-Feldspar	0.16	0.03	0.12	0.02	0.12	0.03	0.16	0.04	0.21	0.02	0.21
Micas/Clays	0.68	0.10	0.37	0.06	0.35	0.11	0.52	0.20	0.94	0.20	1.78
Other Silicates	0.06	0.01	0.05	0.01	0.06	0.01	0.05	0.02	0.08	0.01	0.09
Calcite	1.17	0.13	0.47	0.18	0.98	0.21	1.01	0.35	1.59	0.30	2.64
Dolomite	0.30	0.02	0.06	0.03	0.19	0.04	0.17	0.13	0.58	0.09	0.75

Table 13-3: Mineral Modals – Old Ore

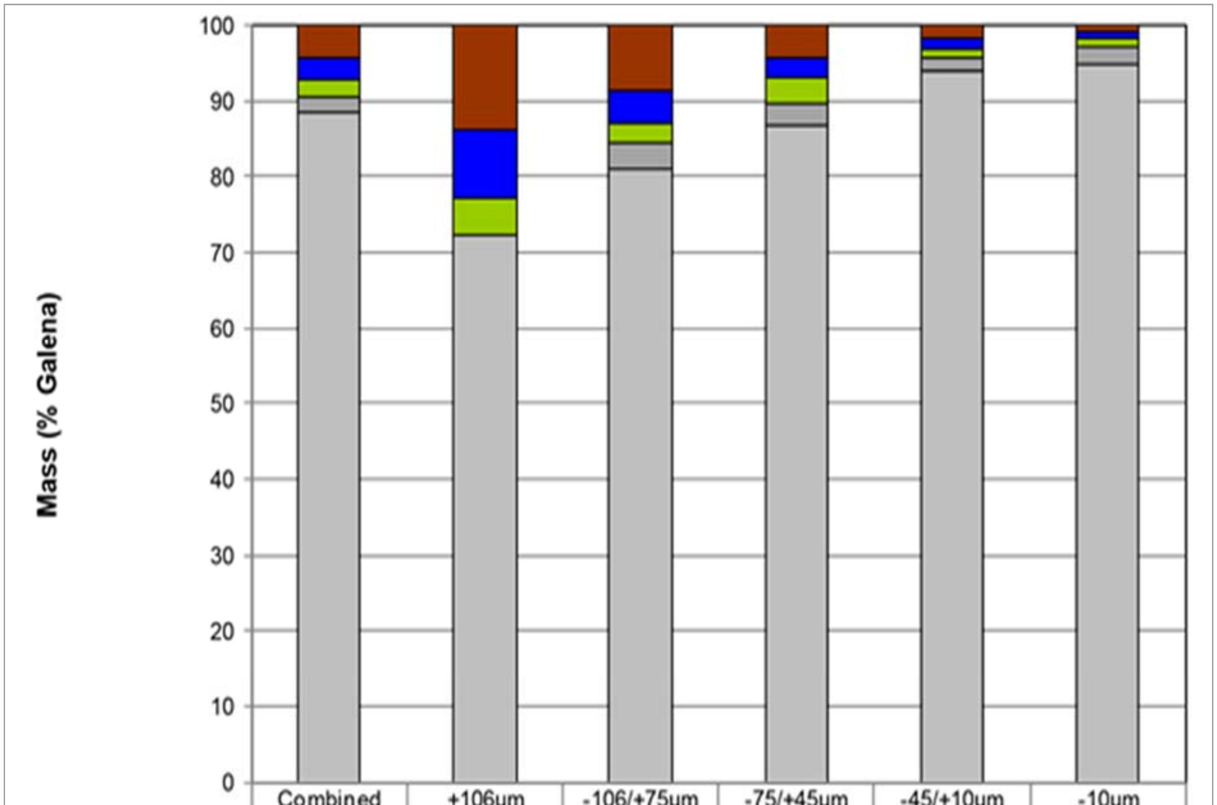
Sample	Old Ore									
Fraction	Combined	+106 um		-106/+75 um		-75/+45 um		-45/+10 um		-10 um
Mass Size Distribution	**100**	28.8		15.8		19.6		22.1		13.8
Calc'd grain size	21	98		57		36		17		6
Mineral Mass (%)	Net	Net	Fraction	Net	Fraction	Net	Fraction	Net	Fraction	Net	Fraction
Pyrite/Marcasite	36.3	14.7	51.2	5.93	37.6	6.50	33.2	6.23	28.1	2.87	20.9
Arsenopyrite	12.5	3.50	12.2	1.92	12.1	2.63	13.4	2.69	12.1	1.79	13.0
Sphalerite	13.9	2.54	8.83	2.43	15.4	3.29	16.8	3.69	16.6	1.97	14.3
Galena	12.8	1.53	5.32	1.75	11.1	2.58	13.2	4.06	18.3	2.86	20.8
Chalcopyrite	0.53	0.07	0.26	0.06	0.38	0.07	0.34	0.12	0.55	0.21	1.51
Tetrahedrite/tennantite	0.13	0.00	0.02	0.02	0.14	0.04	0.19	0.05	0.22	0.01	0.11
Other-Cu-Sulphides	0.12	0.00	0.01	0.01	0.07	0.02	0.11	0.04	0.20	0.04	0.31
Boulangerite	0.34	0.05	0.19	0.06	0.36	0.06	0.31	0.09	0.41	0.08	0.58
Fe-Oxides	0.19	0.07	0.23	0.01	0.06	0.01	0.07	0.02	0.09	0.08	0.55
Quartz	16.9	5.38	18.7	2.79	17.7	3.22	16.5	3.41	15.4	2.08	15.1
Plagioclase/K-Feldspar	0.23	0.04	0.15	0.03	0.19	0.04	0.19	0.04	0.19	0.08	0.55
Micas/Clays	0.72	0.10	0.33	0.07	0.44	0.11	0.56	0.21	0.95	0.24	1.71
Other Silicates	0.09	0.02	0.08	0.01	0.06	0.01	0.05	0.02	0.07	0.03	0.22
Calcite	4.32	0.59	2.06	0.61	3.87	0.86	4.38	1.25	5.66	1.01	7.31
Dolomite	0.81	0.11	0.37	0.07	0.47	0.11	0.56	0.19	0.87	0.33	2.42

13.2.1.2 *Mineral Liberation and Association*

The main minerals that are recoverable by flotation are galena, sphalerite, pyrite, and arsenopyrite, to produce lead, zinc, and gold sulphide concentrates. The liberations of these minerals are summarized in the following sections.

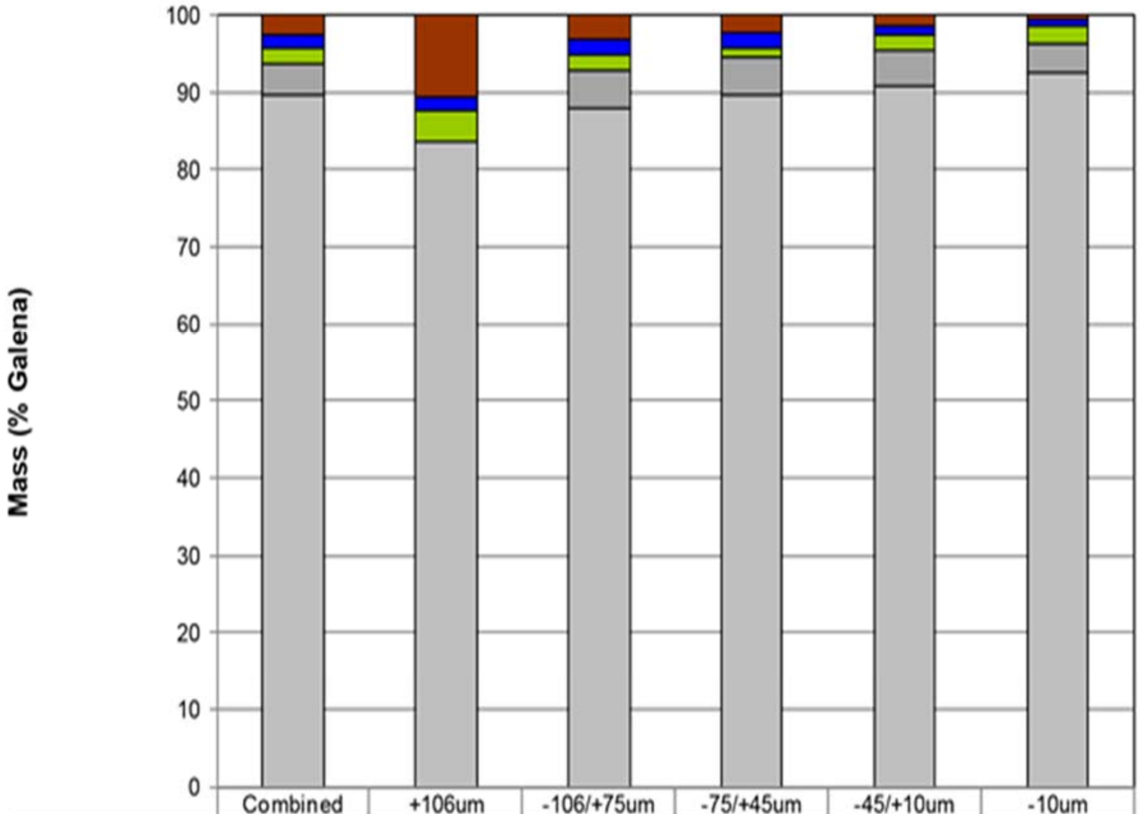
Galena Liberation

The liberation of galena in the Fresh and Old Ore samples at grind size of 80% passing 120 µm is summarized in Figure 13-1and Figure 13-2, respectively.



	Combined	+106um	-106/+75um	-75/+45um	-45/+10um	-10um
Locked Galena	4.31	13.7	8.53	4.19	1.65	0.76
Sub Midds Galena	2.87	9.04	4.52	2.71	1.42	0.80
Midds Galena	2.35	5.03	2.55	3.38	1.27	1.35
Lib Galena	2.07	0.00	3.47	2.91	1.73	2.15
Free Galena	88.4	72.2	80.9	86.8	93.9	94.9

Figure 13-1: Galena Liberation – Fresh Ore



	Combined	+106um	-106/+75um	-75/+45um	-45/+10um	-10um
Locked Galena	2.67	10.8	3.06	2.29	1.26	0.48
Sub Midds Galena	1.54	1.63	2.12	1.94	1.36	1.03
Midds Galena	2.10	4.13	1.89	1.33	1.87	2.16
Lib Galena	3.99	0.00	5.00	4.76	4.69	3.81
Free Galena	89.7	83.5	87.9	89.7	90.8	92.5

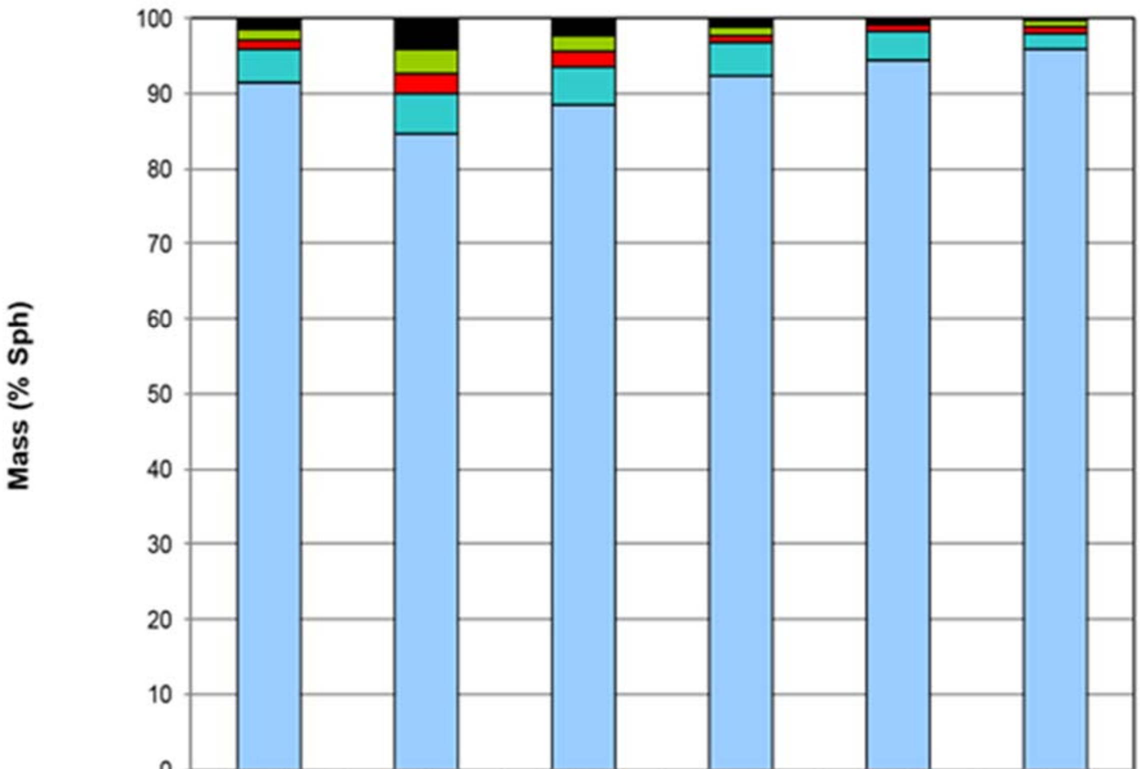
Figure 13-2: Galena Liberation – Old Ore

As the data indicates, both samples are well liberated, with the amount of free and liberated grains being 90.4% and 93.7% for the Fresh and Old Ore samples, respectively. In both cases, the proportion of locked and sub-middling particles reduces considerably in the finer fractions, and this suggests that regrinding would be required to attain high concentrate grades.

For the Fresh Ore sample, galena is mainly associated with pyrite (4.48%) and arsenopyrite (3.34%), with smaller amounts associated with sphalerite (0.90%) and complex particles (0.49%). Galena contained in the Old Ore sample is mainly associated with pyrite (3.00%), but significantly less so with arsenopyrite (0.60%), sphalerite (1.20%), and complex particles (0.62%).

Sphalerite Liberation

The liberation of sphalerite at grind size of 80% passing 120 µm is summarized in Figure 13-3 and Figure 13-4 for the Fresh and Old Ore samples, respectively.

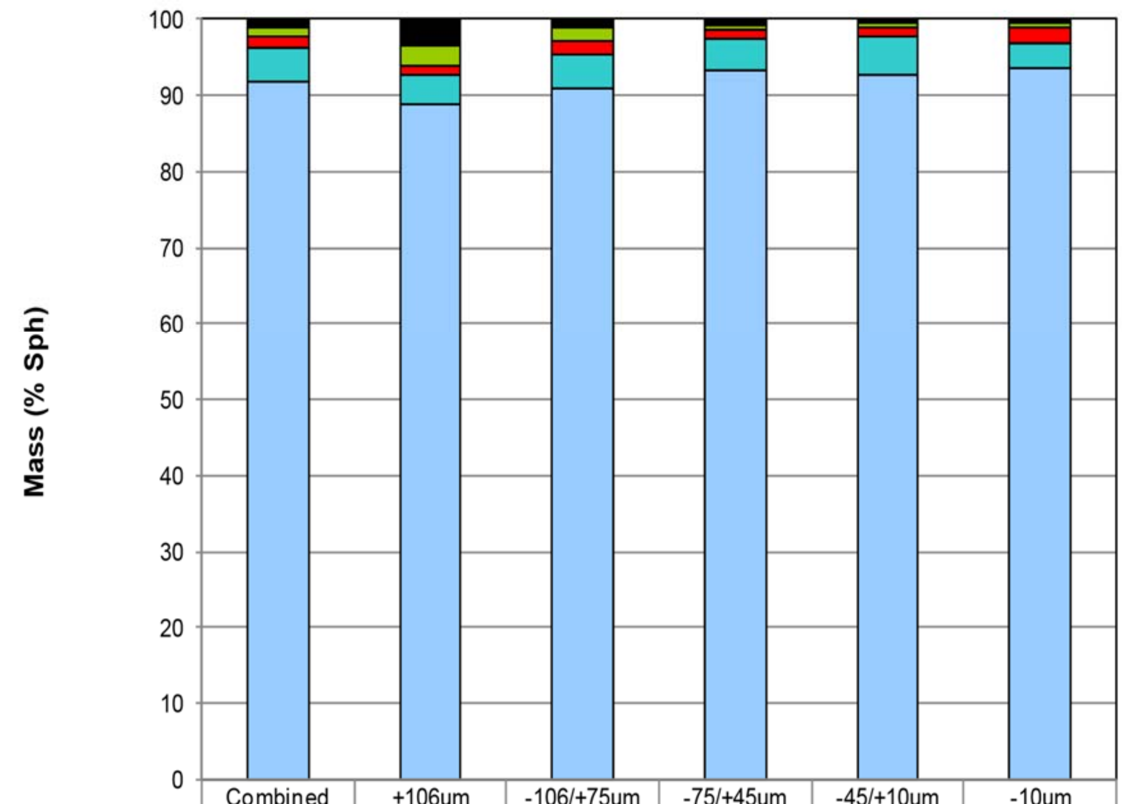


	Combined	+106um	-106/+75um	-75/+45um	-45/+10um	-10um
Locked Sph	1.49	4.03	2.26	1.23	0.41	0.35
Sub Midds Sph	1.38	3.36	2.28	1.06	0.36	0.67
Midds Sph	1.37	2.65	2.02	0.86	0.85	1.02
Lib Sph	4.30	5.45	5.09	4.63	3.96	2.17
Free Sph	91.5	84.5	88.3	92.2	94.4	95.8

Figure 13-3: Sphalerite Liberation – Fresh Ore

The charts demonstrate that the degree of sphalerite liberation is higher than galena, with the content of free and liberated grains being 95.8% in the Fresh Ore and 96.2% in the Old Ore sample, respectively. The amount of locked and middling particles in the coarser size fractions indicates that regrinding would be required to attain high concentrate grades.

For the Fresh Ore and Old Ore samples, the amount of sphalerite associated with pyrite is 1.51% and 1.60%, respectively, with the association with other minerals being less than 1%.



	Combined	+106um	-106/+75um	-75/+45um	-45/+10um	-10um
Locked Sph	1.10	3.28	1.03	0.69	0.39	0.37
Sub Midds Sph	1.16	2.68	1.69	0.52	0.56	0.77
Midds Sph	1.49	1.39	1.79	1.25	1.36	1.89
Lib Sph	4.31	3.84	4.65	4.32	4.86	3.45
Free Sph	91.9	88.8	90.8	93.2	92.8	93.5

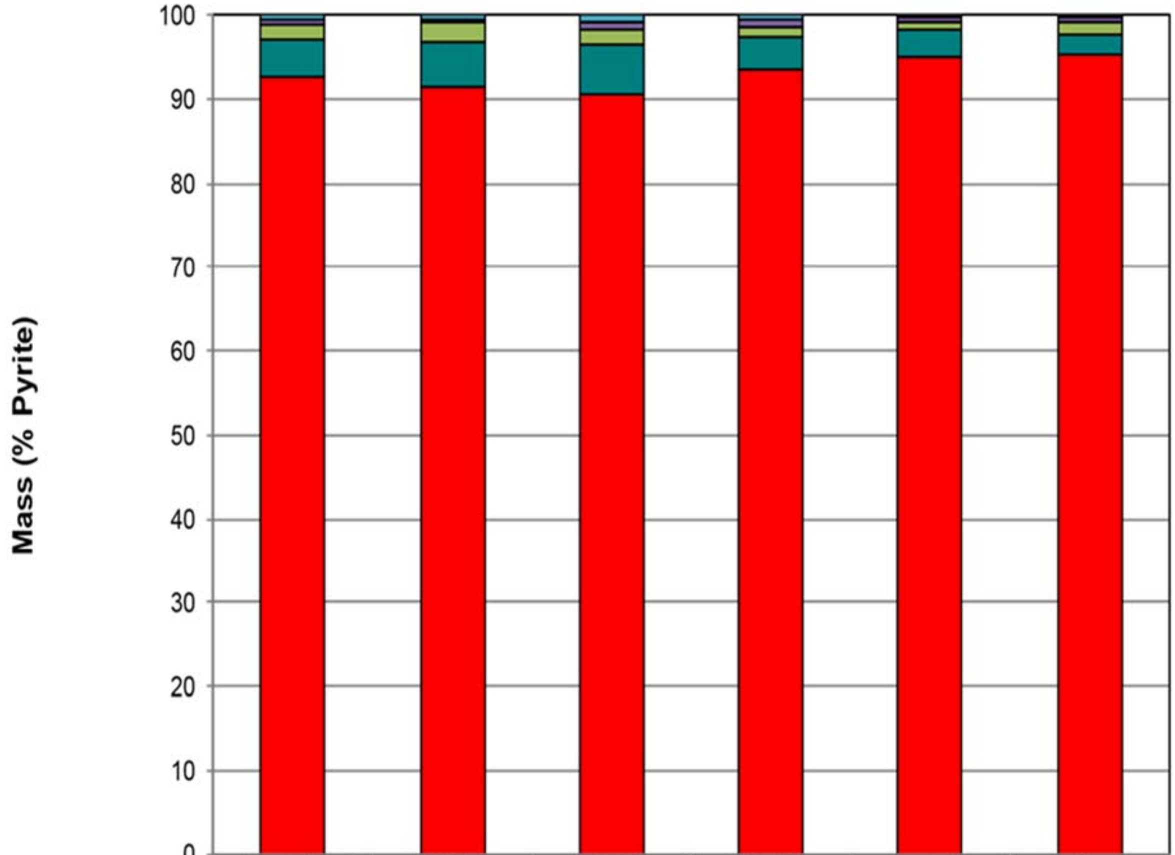
Figure 13-4: Sphalerite Liberation – Old Ore

Pyrite Liberation

The liberation of pyrite is summarized in Figure 13-5 and Figure 13-6 for the Fresh and Old Ore samples, respectively. The liberation of pyrite is very good, with the proportions being 97% and 96% for the Fresh and Old Ore samples, respectively. As the liberation is consistent across the size fractions, regrind would not be required.

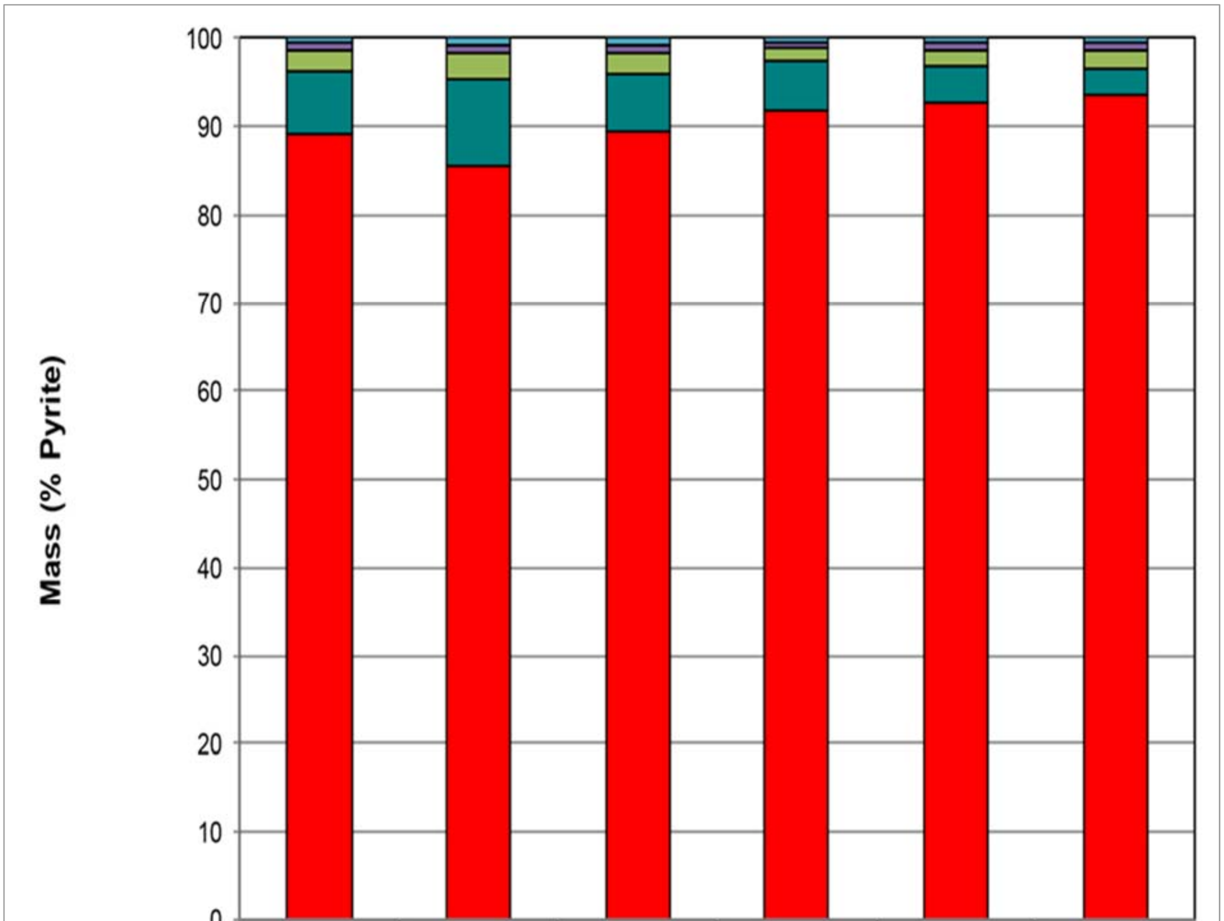
Arsenopyrite Liberation

The liberation of arsenopyrite is 96% and 92% in the Fresh and Old Ore samples, respectively. The coarsest size fraction (+106 μm) has the lowest degree of liberation, but at 91% and 86% for the Fresh and Old Ore samples, the liberation is quite good.



	Combined	+106um	-106/+75um	-75/+45um	-45/+10um	-10um
Locked Pyrite	0.57	0.56	0.83	0.69	0.38	0.29
Sub Midds Pyrite	0.50	0.34	0.82	0.67	0.36	0.47
Midds Pyrite	1.75	2.43	1.80	1.28	0.98	1.41
Lib Pyrite	4.52	5.28	6.04	3.82	3.30	2.46
Free Pyrite	92.7	91.4	90.5	93.5	95.0	95.4

Figure 13-5: Pyrite Liberation – Fresh Ore



	Combined	+106um	-106/+75um	-75/+45um	-45/+10um	-10um
Locked Pyrite	0.59	0.66	0.68	0.47	0.49	0.45
Sub Midds Pyrite	0.87	0.97	0.94	0.71	0.77	0.84
Midds Pyrite	2.29	2.87	2.30	1.46	1.78	2.23
Lib Pyrite	7.01	9.88	6.53	5.51	4.11	2.98
Free Pyrite	89.2	85.6	89.6	91.8	92.8	93.5

Figure 13-6: Pyrite Liberation – Old Ore

Electron Microprobe Analysis

Electron microprobe analysis was conducted to determine element deportment in various minerals and mineral identification. The mineral assay determined by microprobe analysis is summarized in Table 13-4. An important objective of this work was to identify minerals that contain arsenic and antimony that can report to the lead and zinc concentrates.

Table 13-4: Electron Microprobe Mineral Assay Summary

Mineral	Sample	%								
		S	As	Zn	Ag	Fe	Pb	Sb	Cu	Total
Pyrite	Fresh Ore	52.1	1.40	0.026	0.0043	46.2	0.0	0.0045	0.019	99.8
	Old Ore	52.3	1.22	0.047	0.0017	46.0	0.0	0.0020	0.010	99.5
	Global Average	52.2	1.31	0.036	0.0030	46.1	0.0	0.0032	0.014	99.7
Galena	Fresh Ore	13.4	0.010	0.046	0.18	0.072	85.9	0.27	0.0004	99.8
	Old Ore	13.3	0.019	0.057	0.13	0.067	85.4	0.27	0.0100	99.3
	Global Average	13.4	0.015	0.051	0.15	0.070	85.6	0.27	0.0052	99.6
Arsenopyrite	Fresh Ore	22.4	41.6	0.021	0.006	35.7	0.0	0.042	0.0009	99.7
	Old Ore	22.4	41.6	0.110	0.004	35.7	0.0	0.069	0.0026	99.8
	Global Average	22.4	41.6	0.064	0.005	35.7	0.0	0.055	0.0018	99.8
Sphalerite	Fresh Ore	33.5	0.0042	59.0	0.0098	6.97	0.0	0.0003	0.220	99.7
	Old Ore	33.6	0.0000	57.9	0.0060	8.43	0.0	0.0026	0.052	100.0
	Global Average	33.6	0.0021	58.5	0.0078	7.72	0.0	0.0015	0.130	99.9
Boulangerite / Bournonite	Fresh Ore	18.2	0.24	0.42	0.0130	0.91	56.0	24.1	0.013	99.9
	Old Ore	18.9	0.44	0.13	0.0000	0.15	50.9	24.6	4.27	99.3
	Global Average	18.6	0.35	0.26	0.0061	0.50	53.2	24.4	2.34	99.6
Tetrahedrite	Old Ore	25.1	1.01	3.66	1.16	3.76	0.0	27.4	37.2	99.3

The only mineral present that contains appreciable amounts of arsenic is arsenopyrite. Tetrahedrite and pyrite contain a relatively high amount of arsenic at approximately 1%, but due to the likely small amount of these minerals being present in the concentrate, they are not expected to be problematic. For antimony, galena contains 0.27%. Other minerals containing antimony at significant levels are boulangerite, bournonite, and tetrahedrite. These antimony minerals would be expected to report to the lead concentrate and could cause excessive antimony contamination.

13.2.2 Grindability testwork

13.2.2.1 Unconfined Compressive Strength Test

One rock from each sample was used for unconfined compressive strength (UCS) testing (Table 13-5). The compressive strength was 83.5 MPa and 35.6 MPa for the Fresh and Old Ore samples, respectively. Based on these data, the Fresh Ore sample was classified as 'soft', while the Old Ore sample was classified as 'very soft'.

Table 13-5: UCS Test Results

Sample	Length (cm)	Diameter (cm)	Compressive strength (MPa)	Specific Gravity (t/m^3)
Fresh Ore	6.81	3.74	83.5	3.90
Old Ore	9.20	3.76	35.6	4.63

13.2.2.2 Bond Rod Mill Grindability Tests

The test results for Bond rod mill grindability testing are summarized in Table 13-6.

Table 13-6: Bond Rod Mill Grindability Test Results

Sample name	Mesh of grind	Feed F_{80} (mm)	Product P_{80} (mm)	Work index (kWh/t)
Fresh Ore	14	11,294	832	6.8
Old Ore	14	9,320	872	6.7

The Fresh Ore sample had a rod mill work index of 6.8 kWh/t and was classified as 'very soft', while the Old Ore sample had a similar rod mill work index of 6.7 kWh/t and was also classified as 'very soft'.

13.2.2.3 Bond Ball Mill Grindability Test

The Bond ball mill grindability test was conducted at two closing mesh grind sizes, 65 mesh and 80 mesh (212 µm and 180 µm), to attain a product size distribution within the grind target between 120 µm to 150 µm. The test results are summarized in Table 13-7.

Table 13-7: Bond Ball Mill Grindability Test Results

Sample name	Mesh of grind	Feed F_{80} (µm)	Product P_{80} (µm)	Work index (kWh/t)
Fresh Ore	65	2,357	168	8.0
	80	2,357	145	8.5
Old Ore	65	2,182	171	8.9
	80	2,182	148	9.2

The Fresh Ore sample had similar ball mill work indexes of 8.0 kWh/t and 8.5 kWh/t at the mesh of grinds of 65 and 80, respectively, with classification as 'very soft' in both cases. The Old Ore sample also had similar ball mill work indexes of 8.9 kWh/t and 9.2 kWh/t at the mesh grinds of 65 and 80, respectively, and was also classified as 'very soft' in both cases.

13.2.2.4 Bond Abrasion Tests

The Fresh Ore and Old Ore samples were submitted for the Bond abrasion index test. The Fresh Ore sample was considered 'medium abrasive' with an abrasion index of 0.274 g, while the Old Ore sample was also considered 'medium abrasive' with an abrasion index of 0.244 g.

13.2.3 Flotation Testwork

Both the Fresh Ore and Old Ore samples were subjected to flotation testwork, with the purpose of producing separate lead, zinc, and gold concentrates.

Testwork included an initial flotation tests under conditions from past flotation testwork at primary grind of 80% passing 100 to 120 µm and isopropyl xanthate as a collector in all three circuits. A series of rougher and open-circuit cleaning tests; and a locked cycle test (LCT) were completed with the Fresh Ore sample. A total of 45 batch flotation tests were completed, using Fresh Ore sample (40 tests) and Old Ore sample (5 tests)

13.2.3.1 Fresh Ore Sample

Lead Rougher Flotation

A variety of variables were evaluated including the following: Collector type (Aerophine 3418A or SIPX) and dosage, $ZnSO_4$ dosage, lime dosage, primary grind size, and rougher flotation residence time.

The primary grind size was generally in the order of 80% passing 110 µm ~ 158 µm. The effect of various parameters on lead recovery and selectivity against other metals was relatively small. Lead recovery ranged from 91.8% to 94.1%, with an average of 92.9%.

Following conditions provided the best results:

- 12 minutes of rougher flotation time
- 750 g/t of lime and 50 g/t of NaCN added to the mill
- pH maintained at 9.5
- 35 g/t of Aerophine 3418A distributed to the first three rougher concentrates

Lead Cleaner Flotation

Twenty-three (23) tests were completed that involved lead cleaner flotation, with the purpose of reducing the amount of arsenic reporting to the final lead concentrate. Rejecting arsenic to the desired level of 0.5% was not achieved during this testwork program and the level of arsenic in the final lead concentrate was seldom less than 3%. The content of antimony in the lead concentrate was not an issue with the Fresh Ore sample and was generally less than 0.5%.

Many varying conditions and reagents were trialed in an attempt to reject arsenic. These included: regrind size, addition of lime, high sodium cyanide dosage, depressants such as guar gum, Aero 7260HFP, hydrogen peroxide, sodium citrate, sodium metabisulphite, extended regrind with multiple stages, and reverse flotation of arsenopyrite from lead concentrate.

Based on the results from all rougher and open-circuit cleaner tests, it was concluded that entrainment is the main factor for arsenic reporting to the lead concentrate. Flotation of arsenopyrite associated with galena was also partially responsible. It was estimated that, for arsenic grade in the lead concentrate to reach 0.5%, arsenic head grade in the feed should be lower than 5% and the final concentrate lead grade should be above 80%. When the concentrate lead grade is less than 80%, arsenic content in the concentrate will increase accordingly.

Lead Concentrate Characterization

The size-by-size analysis was completed on two lead concentrate samples. One concentrate was high grade from initial lead cleaning test, while the second sample was lower grade from the combined lead concentrate. Size-by-size analysis of the lead concentrate is provided in Table 13-8.

Table 13-8: Lead Concentrate Size-by-Size Analysis

size range	Wt. %	Assay %						Distribution %					
		Pb	Cu	Zn	Fe	As	S	Pb	Cu	Zn	Fe	As	S
+53 µm	10.9	36.6	0.38	4.88	15.3	7.80	20.9	6.9	14.1	14.9	17.7	16.2	12.8
38 ~ 53 µm	18.1	49.8	0.29	4.49	12.6	7.33	19.5	15.7	17.8	22.7	24.2	25.2	19.9
25 ~ 38 µm	15.2	55.7	0.25	4.21	10.9	6.52	18.4	14.7	12.9	17.9	17.6	18.9	15.8
14 ~ 25 µm	31.1	64.9	0.21	2.51	7.70	4.97	16.4	35.3	22.3	21.9	25.5	29.5	28.8
-14 µm	24.8	63.4	0.39	3.28	5.64	2.17	16.2	27.4	32.9	22.7	14.9	10.2	22.6
Head (Calc)	100	57.3	0.29	3.57	9.39	5.25	17.7	100	100	100	100	100	100
Head (Direct)	100	56.1	0.28	3.60	9.19	5.44	18.0	NA	NA	NA	NA	NA	NA

These data show that lead grade is lowest in the coarser size fractions, while arsenic grade is highest, indicating that liberation may be an issue. Mineralogical examination indicated that the above lead concentrate contained 65.4% galena, 14.1% arsenopyrite, 9.13% pyrite, 6.36% sphalerite, and 2.93% quartz. Arsenic is present as arsenopyrite, with a minor amount associated with pyrite.

The arsenopyrite is fairly well liberated with 81% of arsenopyrite grains being greater than 90% liberated. Rejection of the liberated arsenopyrite should be possible, while further regrinding might be necessary to liberate the arsenopyrite associated with galena.

Zinc Rougher Flotation

Zinc flotation conditions included activation using copper sulphate and flotation using a weaker xanthate collector (SIPX) to limit recovery of pyrite and arsenopyrite, high pH of 11.8. Similar to the lead rougher, a series of variables were investigated, including collector dosage, $CuSO_4$ dosage, slurry pH, and primary grind size.

Unlike the lead flotation, zinc flotation appeared to be more sensitive to flotation variables. Important variables were SIPX dosage, $CuSO_4$ dosage, and lime dosage (pH).

60 g/t SIPX was the best compromise between zinc recovery and selectivity. Lime dosage appeared to have limited impact on zinc recovery. However, the selectivity of zinc flotation over pyrite/arsenopyrite was improved considerably at the high lime dosage of 1,800 g/t. The higher

dosage of 600 g/t of $CuSO_4.5H_2O$ provided the greatest zinc recovery and selectivity compared to the lower $CuSO_4.5H_2O$ dosages of 300 and 500 g/t.

Coarser grind sizes of 141 and 158 µm resulted in zinc recovery ranging from 81.9% to 84.5%. The zinc recovery increased to 88.6% at the finer grind size.

Following conditions provided the best results for zinc rougher flotation: Lime dosage of 1,890 g/t (pH 11.8), $CuSO_4.5H_2O$ dosage of 600 g/t, and SIPX dosage of 60 g/t.

It was found that using soda ash as opposed to lime in the primary grind avoided the formation of an extremely hydrophobic phase following $CuSO_4$ activation. The use of soda ash in the primary grind in two tests demonstrated that the requirement for $CuSO_4$ could be significantly reduced.

Zinc Cleaner Flotation

The best zinc open-circuit cleaner flotation test achieved zinc recovery of 74.2% at concentrate grade of 55.9% zinc. The contents of arsenic and antimony in the zinc concentrate were low at 0.46% and 0.038%, respectively.

Pyrite and Arsenopyrite Flotation

Following the flotation of lead and zinc, the gold is recovered by flotation of remaining sulphide minerals into a bulk sulphide concentrate with concentrate grade of 20 – 25 g/t Au. The main gold bearing minerals are predominately pyrite and arsenopyrite. Flotation is carried out by reducing slurry pH to 6 with sulphuric acid, and with a large dosage of SIPX.

The highest recovery of gold was 90.6%. A single stage cleaning test resulted in very little upgrading for gold grade from 34.0 g/t to 34.8 g/t. The cleaner stage recovery was quite high at 99.7%.

The preferred conditions of pyrite and arsenopyrite flotation were pH 6.0, 250 g/t SIPX, and MIBC as frother.

Locked-Cycle Flotation Test

A locked-cycle test was completed with the Fresh Ore sample. 1 kg test charges were used. Target and actual grind sizes are provided in Table 13-9.

Table 13-9: LCT-1 Grind Size Summary

Sample	Target P_{80}, µm	Actual P_{80}, µm
Lead Concentrate	25	26
Zinc Concentrate	25	26
Gold Pyrite/Arsenopyrite Concentrate	109	102
Final Tail	103	106

The flowsheet is summarized in Figure 13-7 and the conditions are summarized in Table 13-10.

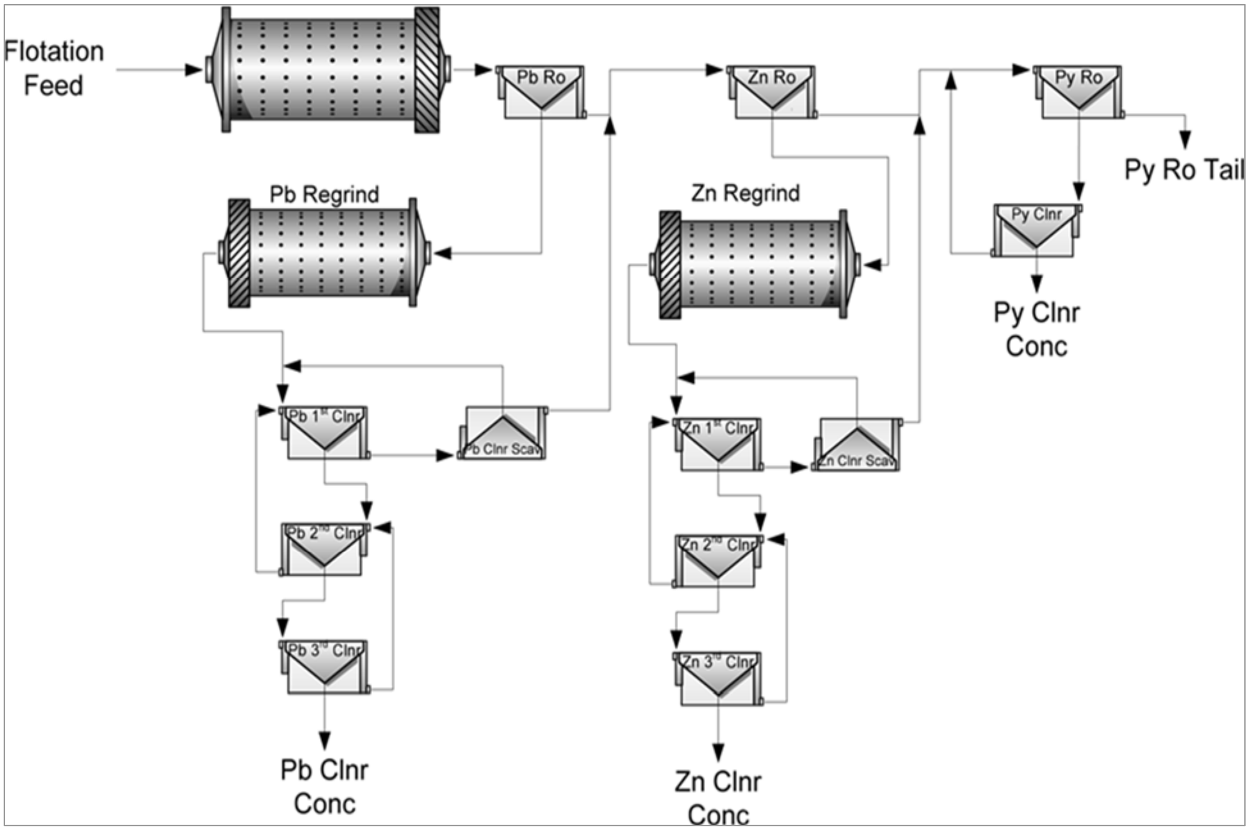


Figure 13-7: Locked-Cycle Test Flotation Flowsheet

Table 13-10: LCT-1 Flotation Conditions

Stage	Reagents added, g/t							Time, minute			pH
	Lime	NaC	3418	MIB	CuS	H_2S	SIPX	Grin	Con	Frot	
Lead Rougher	900	50	35	20				7.5	4	6.5	9.5
Lead Regrind	50	20						3			8.2
Lead Cleaner	100	30	5	35					4	7	9.5
Zinc Rougher	156			20	600		60		3	6.5	11.8
Zinc Regrind	250				50			3			10.9
Zinc Cleaner	980			25			5		4	7.5	11.8
Py / AsPy Rougher				30		2990	250		3	6.5	6.0
Py / AsPy Cleaner				10		150			2	3.5	6.0

The locked-cycle test demonstrated reasonably good stability overall.

The metallurgical performance for Fresh Ore sample from the locked-cycle test is summarized in Table 13-11.

Table 13-11: Locked-Cycle Test Metallurgical Projection – Fresh Ore

Product	Wt	Assays, %, g/t							% Distribution						
	%	Pb	Zn	Au	Ag	S	As	Sb	Pb	Zn	Au	Ag	S	As	Sb
Pb 3rd Cleaner Conc	5.7	69.5	1.63	7.76	1830	14.5	3.22	0.34	83.6	2.0	1.9	75.8	2.9	1.8	30.1
Zn 3rd Cleaner Conc	7.5	6.64	53.6	1.34	207	31.2	0.49	0.046	10.5	85.9	0.4	11.3	8.3	0.4	5.4
AsPy / Py Cleaner Conc	62.1	0.38	0.72	35.8	26.3	39.6	16.1	0.064	5.0	9.5	96.1	11.9	87.4	96.1	62.0
Final Tail	24.6	0.18	0.49	1.44	6.0	1.56	0.73	0.007	0.9	2.6	1.5	1.1	1.4	1.7	2.5
Head (back calculated)	100	4.75	4.70	23.1	138	28.2	10.4	0.064	100	100	100	100	100	100	100
Head (direct)	100	5.04	4.49	22.8	159	29.2	10.1	0.10	NA	NA	NA	NA	NA	NA	NA

High-grade lead and zinc concentrates were produced containing 69.5% Pb and 53.6% Zn, respectively. Concentrate recoveries were 83.6% for lead and 85.9% for zinc in lead and zinc concentrates, respectively.

The lead concentrate contained 3.22% arsenic and 0.34% antimony. In view of 10.4%, arsenic in the feed, 3.22% arsenic in the lead concentrate is quite good. For the plant operation, arsenic content in the mill feed is controlled normally to less than 5%. Because arsenic in the lead concentrate is proportional to arsenic in the mill feed, less than 2% arsenic in the lead concentrate is expected if arsenic content in the mill feed is less than 5%.

The zinc concentrate contained 10.5% Pb, which is high and suspected to be the result of elevated level of lead in the lead cleaner scavenger tail reporting to the zinc circuit. Re-routing this stream to the lead rougher circuit is an option to consider for improvement. Due to the lower recovery of zinc in the zinc circuit, a considerable amount of zinc reported to the gold concentrate. Improving zinc recovery by higher collector dosages during the cleaning stage or re-routing the zinc cleaner scavenger tail to the zinc rougher were noted as options to consider for improvement.

The pyrite / arsenopyrite concentrate grade was extremely high containing 39.6% S, 35.8 g/t Au, which represents 96.1% gold recovery.

13.2.3.2 Old Ore Sample

The flotation testing with the Old Ore sample was limited to five flotation tests. The Old Ore sample had considerably higher grade than the Fresh Ore sample, containing almost twice the amount of lead and zinc than the Fresh Ore sample. However, the Old Ore sample contained approximately half the amount of arsenic compared to the Fresh Ore sample. All flotation tests responded well with 30 ~ 40 g/t Aerophine 3418A and 75 ~ 80 g/t sodium cyanide addition, except for one test (F7) which resulted in excessive arsenic recovery at increased collector and reduced cyanide additions. There is similarity in the zinc and arsenic selectivity, but the mass pull to achieve the same recovery was significantly higher due to the substantially higher grade of lead in the feed.

For zinc rougher flotation, due to the higher feed grades, a higher mass pull would be required to attain the same level of recovery. pH 11.8 was used for activation and resulted in considerably greater selectivity for zinc over arsenic.

The performance of pyrite/arsenopyrite flotation was relatively poor, due to high recovery of gold-bearing sulphides in the zinc circuit. Nevertheless, the stage-recovery of gold approaches 100% for all tests

Due to the high head grade of lead and low head grade of arsenic compared to the Fresh Ore sample, the content of arsenic in the lead concentrates was quite low at 0.47 ~ 0.70%. However, the content of antimony in the lead concentrate was quite high at 2.32% ~ 2.64%. Antimony floated strongly together with lead. The overall lead recovery for the Old Ore sample was 74.4%, despite higher lead rougher recovery of 94.4%.

It appears that the addition of lime considerably reduced lead recovery. The low arsenic and high antimony grade in the lead concentrate was due to the lower arsenic head grade and higher antimony head grade in the feed.

13.2.3.3 Large Batch Flotation

Large batch flotation was completed on 300 kg of Fresh Ore sample to produce 90 kg of tailings for backfill testing by a third-party laboratory. The large batch flotation test resulted in lower concentrate grades and recoveries than anticipated. This was attributed to the fairly coarse lead regrind P_{80} of 44 µm compared to the 25 µm used during the smaller-scale laboratory testwork. Solid-liquid separation testwork was conducted on the products from the large batch flotation test, including final tail, gold/pyrite/arsenopyrite cleaner concentrate, zinc cleaner concentrate and lead cleaner concentrate.

13.2.3.4 Solid-Liquid Separation Testing

Flocculant scoping and static settling testwork indicated that all samples responded well to BASF Magnafloc 10 flocculant. Static settling tests showed underflow (U/F) densities of 66% w/w for the final tail, and 71% ~ 79% w/w for the concentrates. The results are presented in Table 13-12.

Table 13-12: Static Settling Results Summary

Sample ID	Dosage flocc't g/t	Feed[1] %w/w	U/F[2] %w/w	CSD U/F[3] %w/w	TUFUA[4] m^2/t/day	Supernatant[5] visual	TSS mg/L
Final tail	9	10	66	68	0.04	Clear	<10
Gold/pyrite/arsenopyrite concentrate	5	15	75	n.d	0.01	Clear	39
Zinc concentrate	9	15	71	74	0.03	Clear	19
Lead concentrate	8	15	79	80	0.02	Clear	18

Notes:
All values were calculated without a safety factor. Common conditions: Raked, ambient temperature. Flocculants: BASF Magnafloc 10
[1] Diluted Thickener Feed.
[2] Ultimate Underflow Density.
[3] Underflow Density as predicted by the Critical Solids Density (CSD) as detailed in the Rheology Test Section.
[4] Thickener Underflow Unit Area corrected with actual CSD.
[5] Supernatant Clarity after 60 minutes of elapsed settling time.

Dynamic settling tests were conducted on all samples except for the zinc cleaner concentate due to insufficient mass. Table 13-13 summarizes the optimized dynamic settling operating conditions and results, confirming the U/F densities obtained in the static settling tests.

Table 13-13: Optimized Dynamic Settling Results Summary

Sample ID	Dosage flocc't	Feed[1]	U/F[2]	U/F[3] extended	TUFUA4	Solids loading	Net Hyd. loading	Res. time solid vs UF	O/F	TSS
	g/t	% wt.	% wt.	% wt.	$m^2/t/d$	$t/m^2/h$	$m^3/m^2/d$	h	visual	mg/L
Final tail	10	10	64.5	72.4	0.10	0.416	3.70	0.84	Clear	62
Gold/pyrite/arsenopyrite concentrate	10	15	80.6	87.0	0.07	0.592	3.30	1.1	Clear	10
Lead concentrate	12	15	79.1	83.4	0.08	0.520	2.88	1.4	Clear	22

Notes:
All values were calculated without a safety factor. Flocculants: BASF Magnafloc 10 underflow raking, ambient temperature.
[1] Autodiluted Thickener Feed.
[2] Underflow (UF) Density.
[3] Underflow (UF) Density after 30 minutes of extended thickening (raked, no feed).
[4] Thickener Underflow Unit Area.

Concentric cylinder rotational viscometry (CCRV) rheology was conducted on the final tail, zinc concentrate, and lead concentrate underflow samples, while vane rheology measurements were conducted on the gold/pyrite/arsenopyrite concentrate underflow sample due to its fast settling nature. Table 13-14 summarizes the critical solids densities (CSD) and corresponding yield stress values.

Table 13-14: Underflow Rheology Results Summary

Sample I.D.	CSD[1], %w/w solid	Yield stress, Pa Un - sheared
Final tail	67.6	21
Zinc concentrate	74.0	14
Lead concentrate	80.0	10

Notes:
[1] CSD: Rheology-determined Critical Solids Density.

Vane rheology was conducted at pulp densities from 85.8% to 87.7% w/w solids using a Haake VT550 viscometer. The resulting vane-yield stress values ranged from 222 Pa to 989 Pa at 86.1% to 87.7% w/w solids (see Table 13-15). At 85.8% w/w solids, the sample settled and did not provide a yield point.

The critical solid density (CSD) is the solids density at which a small increase of the solid density causes a significant decrease of flowability of the slurry. The CSD value is also predictive of the maximum underflow solids density achievable in a commercial thickener and pumpability achievable in practice.

Table 13-15: Vane Rheological Data Summary – Gold/Pyrite/Arsenopyrite Concentrate

Test No.	Size, P_{80} (µm)	SG[1] (g/cm^3)	Temp (^{0}C)	Solids %	Shear stress [2]Peak (Pa)
1	103	5.10	20.0	87.7	989
2	103	5.10	20.0	86.9	527
3	103	5.10	20.0	86.1	222
4	103	5.10	20.0	85.8	No yield

Notes:
The gold/pyrite/arsenopyrite concentrate samples were flocculated with Magnafloc 10 at a dosage of 10 g/t.
Shear Stress value was produced by subjecting the slurry sample to a constant rotation at 0.022 s^{-1} for 4 minutes.
[1] SG - Dried solids specific gravity, measured with pycnometer
[2] Maximum (peak) initial shear stress achieved.

Results of vacuum filtration tests on the final tail underflow sample are summarized in Table 13-16. Using a Testori 6583 TC filter cloth, a cake moisture of 13.6% w/w was achieved from a 65% w/w pulp density with a dry time of 30 seconds. A qualitative confirmation test at lower filter feed solids (60% w/w) achieved a similar cake moisture of 15% w/w.

Table 13-16: Vacuum Filtration Results Summary of the Final Tail

Operating conditions					Filter throughput[1]		
Feed solids %w/w	Vacuum level, inch Hg	Form time (s)	Dry time (s)	Form / dry ratio	Cake thickness (mm)	Throughput, dry kg/m^{2}h	Cake[2] moisture %w/w
65	23.4	5	5	1.00	20.0	10329	21.5
		4	4	1.06	14.5	9111	19.5
		4	6	0.65	15.0	7016	17.4
		5	15	0.32	15.0	3853	16.3
		4	20	0.21	15.0	3140	14.7
		3	30	0.10	10.5	1594	13.6
		6	2	4.00	20.0	13397	22.3
60	24	6	30	0.20	15.0	2123	15.0

Notes:
[1] Filter throughput predictions versus test conditions using raw test data. Results are not for sizing of any specific type of filter. Refer to individual test results for additional sizing information.
[2] Common cake characteristics for all samples: Cohesive, Good cake release from cloth.

Conditions and results of pressure filtration tests on the concentrate underflows are summarized in Table 13-17.

A qualitative confirmation test at lower filter feed solids was conducted using the best conditions of the pressure filtration test. Cake moistures ranged from 3.8% to 5.1% w/w for the gold/pyrite/arsenopyrite concentrate, from 5.2% to 7.2% w/w for the zinc concentrate, and from 5.0% to 6.0% w/w for the lead concentrate.

13.2.3.5 Environmental Characterization

The final tailing sample from the large batch flotation testing was subjected to a basic environmental test program, including elemental analysis (qualitative X-ray diffraction, whole rock, and Aqua regia digest analyses), toxicity characteristic and synthetic precipitation leaching procedure (TCLP and SPLP) extractions, modified acid base accounting (ABA), net acid generation (NAG) testing, and NAG liquor analysis.

Elemental analyses indicate that the tailing sample was predominantly comprised of quartz with minor amounts of calcium, iron, and aluminum. Environmentally significant concentrations of arsenic, lead, and zinc were also observed.

Analysis of the TCLP leachate reported arsenic and lead at concentrations expected to be of environmental concern. It should be noted that since the TCLP is a very aggressive extraction procedure, the limits applicable to this test procedure are much higher than those used for the SPLP or NAG liquor analyses. With the exceptions of pH, arsenic, and lead, the SPLP extraction leachate reported all of the typically controlled parameters well within the limits characteristically applied to this test procedure.

Table 13-17: Pressure Filtration Results Summary for the Concentrates

Sample identification	Filter cloth	Operating conditions					Filter throughput[1]		
		Feed solids %w/w	Pressure level, PSI	Form time (s)	Dry time (s)	Form / dry ratio	Cake thickness (mm)	Throughput, dry kg/m^{2}h	Cake[2] moisture, % w/w
Gold/pyrite/ arsenopyrite concentrate	Micronics 861	85.0	5.5	1.7	92	0.02	31.5	3672	3.8
				1.2	59	0.02	24.5	4461	4.0
				1.1	45	0.02	20.4	4784	3.8
				1.1	31	0.03	14.5	5002	3.8
			6.9	1.7	62	0.03	30.4	5052	4.3
				1.3	50	0.03	25.5	5518	4.2
				1.5	49	0.03	20.0	4293	4.0
				1.0	36	0.03	15.0	4503	4.0
Gold/pyrite/ arsenopyrite concentrate		80.0		1.7	33	0.05	21.0	6046	5.1
Zinc concentrate	Micronics 889	74.0	5.5	3.5	85	0.04	32.5	2906	6.1
				2.4	55	0.04	25.0	3489	6.1
				2.1	37	0.06	20.0	4124	6.3
				1.4	27	0.05	14.5	4029	6.0
			6.9	2.3	56	0.04	30.5	4113	6.0
				1.8	55	0.03	25.5	3476	5.7
				1.8	56	0.03	20.4	2858	5.2
				1.3	33	0.04	14.0	3364	5.1
Zinc concentrate		64.0		1.9	39	0.05	20.4	3764	7.2
Lead concentrate	Micronics 861	78.0	5.5	3	72	0.04	32.0	4635	5.2
				2	55	0.04	24.5	4724	5.1
				2	42	0.04	20.5	4969	5.0
				1	30	0.04	14.0	5200	5.0
			6.9	2	45	0.05	29.5	6697	5.2
				2	38	0.04	24.3	6671	5.7
				1	26	0.05	19.0	7295	5.9
				1	24	0.04	14.0	6268	6.0
Lead concentrate		68.0		3	40	0.08	19.0	4660	5.7

Notes:
[1] Filter throughput predictions versus test conditions using raw test data. Throughputs are calculated based on cycle time of form and dry only. Results are not for sizing of any specific type of filter. Refer to individual test results for additional sizing information.
[2] Common cake characteristics for all samples: Cohesive, Good cake release from cloth.

Modified ABA testing results show the final tailing sample as uncertain about acid generation potential. Although NAG testing reported no net acidity generated, due to the elevated sulphide content, the single peroxide addition may not have completely oxidized the sulphide present. Sequential NAG tests using multiple peroxide additions were noted as being required to confirm this assumption. Analysis of the NAG liquor reported arsenic at an environmentally significant concentration.

13.2.3.6 Transportable Moisture Level Testing

A portion of the lead, zinc, and gold concentrates produced from the large batch flotation testwork was used for determining the transportable moisture level. Results are summarized in Table 13-18.

Table 13-18: Transportable Moisture Level Test Summary

Sample	P_{80}, µm	Flow moisture, %	Transportable moisture, %
Lead Concentrate	25	11.4	10.3
Zinc Concentrate	23	13.8	12.4
Gold/pyrite/arsenopyrite Concentrate	109	9.2	8.3

13.3 CONCLUSIONS AND RECOMMENDATIONS

13.3.1 Conclusions

- The main sulphide minerals for the Fresh Ore and Old Ore samples were galena, sphalerite, pyrite, and arsenopyrite. Other than quartz, all other minerals can be considered minor.
- The liberation of all sulphide minerals was excellent at a P80 of 120 µm.
- Flotation of galena, sphalerite and pyrite / arsenopyrite in a sequential flowsheet was found to be effective at producing lead, zinc and gold concentrates.
- Lead flotation was found to work well with Aerophine 3418A, with lime and sodium cyanide.
- Copper activation and zinc flotation were found to be optimum at a pH of 11.8 with 600 g/t $CuSO_4.5H_2O$ and 60 g/t SIPX.
- Flotation of pyrite and arsenopyrite was effective at pH 6.0 and 250 g/t SIPX.
- The rejection of arsenic in the Fresh Ore sample and antimony in the Old Ore sample was found to be problematic in the lead concentrate. It was extremely difficult to reduce the content of arsenic in the lead concentrate from the Fresh Ore to less than 3%. Antimony was not an issue with the Fresh Ore sample but was identified with the Old Ore sample as being present in the lead concentrate at a grade of approximately 3%.
- The locked cycle test indicated a high amount of lead (6.64%) reporting to the zinc concentrate and a high amount of zinc (0.72%) reporting to the gold concentrate.

- A large batch flotation campaign was completed to produce tailings for backfill testing by a third-party. Issues with zinc flotation were identified. The lead and zinc concentrates produced were relatively low grade and the recovery was low.
- All flotation products responded well to BASF Magnafloc 10 flocculant.
- Tests confirmed good thickening characteristics. Thickener overflow was visually clear with total suspended solids (TSS) of 62 mg/L, 10 mg/L, and 22 mg/L for the final tail, gold/pyrite/arsenopyrite concentrate and lead concentrate, respectively.
- Final tail, zinc concentrate and lead concentrate underflows displayed Bingham plastic rheological behaviour characterized by yield stress.
- The critical solid density was ~68% for the final tail underflow, ~74% for the zinc concentrate underflow, and ~80% for the lead concentrate underflow. The yield stress for the gold/pyrite/arsenopyrite concentrate ranged from 22 to 989 Pa between 86.1% and 87.7% solids w/w.
- Vacuum filtration of the final tail underflow showed high throughput and low residual cake moisture, which ranged from 13.6% to 22.3%.
- Pressure filtration on all concentrate underflows indicated high throughput and low residual moisture of 3.8%~4.3% for the gold/pyrite/arsenopyrite concentrate, 5.1%~6.3% for the zinc concentrate, and 5.0%~6.0% for the lead concentrate.

The environmental characterization of the final tailing sample indicated the following:

- The final tailing was comprised primarily of quartz with minor amounts of calcium, iron, and aluminum and environmentally significant concentrations of arsenic, lead, and zinc.
- The TCLP leachate reported arsenic and lead at concentrations expected to be of environmental concern.
- The SPLP extraction leachates reported a strongly alkaline pH value and concentrations of arsenic and lead that may be environmentally significant.
- ABA testing indicated uncertain acid generation potential for the final tailing.
- NAG testing of the final tailing reported no net acidity generated.
- Analysis of the NAG liquor reported arsenic at an environmentally significant concentration.

13.3.2 Recommendations from Metallurgical Testwork

- Column flotation which involves a high degree of froth washing was recommended to aid in the rejection of arsenic in the lead concentrate in cleaner flotation stages.
- Mineralogy of the lead concentrate indicated that a portion of the remaining arsenic is present as arsenopyrite locked with galena grains. The use of two regrinding stages in the lead

cleaning circuit was found to benefit arsenic rejection and this was recommended to be investigated further as well as optimization of depressant schemes.

- The high content of antimony in the lead concentrate from the Old Ore sample was due to the high content of antimony-bearing minerals such as boulangerite, bournonite, and tetrahedrite that appear to report to the lead concentrate. These minerals contain copper and lead, and thus were expected to behave similarly to chalcopyrite and galena. Further testwork for depression of antimony bearing minerals was recommended as the lead concentrate containing appreciable amounts of antimony would face significant penalty charges.
- Further study was recommended on replacing the lime addition with soda ash in the primary grind and a lower dosage of $CuSO_4$ to overcome the formation of an extremely dry and flat froth causing low zinc rougher recovery.
- In light of the high pH requirements for selective flotation in the zinc rougher circuit, it was recommended that flotation of pyrite / arsenopyrite at the naturally elevated pH be evaluated in order to reduce sulphuric acid consumption.
- The locked cycle test identified the issues with excessive lead and zinc reporting to the zinc and gold concentrates, respectively. Further evaluation of the flowsheet, specifically the cleaner scavenger conditions and retention times with locked cycle testing, was recommended
- Both Old Ore and Fresh Ore samples had challenges with rejection of arsenic and antimony, which are smelter penalty elements. Further testing with different samples was recommended to ascertain how widespread the issue of arsenic and antimony reporting to the lead concentrate is, and if any other issues arise with zinc and gold flotation circuits.
- It was noted that during dynamic thickening testing, there was an observed sensitivity of the flow response of the underflows to the solid’s residence time. This could lead to bed over-compaction or underflow plugging during extended residence times. Thickener design and operation was advised to be carefully controlled to mitigate any issues relating to this phenomenon.

SECTION • 14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

Mineral resource estimates for Olympias mine were made from a 3D block model utilizing MineSight 3D software. Project limits, in UTM coordinates, are 478105 to 479700 East, 4491165 to 4493480 North and -800 to +60m elevation. Cell size for the project was 5 m east x 5 m west X 5 m high.

Three areas (East, West, and Flats) have been included in the mineral resource estimate. Wireframes were prepared in LeapFrog software to model these three zones. Block grades for Au, Ag, As, Pb, Zn, and Fe were estimated using inverse distance to the power of 4 (ID^4). SG and Sulfur grades calculated by formula and are of the function of the Pb, Zn, As and Fe grade estimations.

14.2 MINERALIZED DOMAINS

The historical approach to creating mineralized domains at Olympias involved defining limits to what was thought as the dominant mineralization style: massive sulphide bodies. The relogging effort of the pre-existing core and the growing database of infill drilling and mapped stopes showed this assumption to be simplistic. The sulphide mineralization at Olympias, as described in Section 8, comprises many styles and these styles often change abruptly over the scale of mining. Instead, a grade based discriminant would allow for more consistent interpretations to be made. The challenge was to find the formulation of this discriminant because the Olympias deposit contains multiple revenue-contributing metals: Au, Ag, Pb and Zn. In the end, a simplistic value formulae based on the logic of a Net Smelter Return formulae, that used a combination of metal prices and metal recoveries to act as weighting factors against each metal, showed to be an excellent surrogate for a comprehensive equivalent grade. Inspection of these resource defining values (RDV) showed that for the parameters used, a value of $50 best defined what one would classify as likely economically mineralized zones. Seequent's Leapfrog Geo™ software was used to model the mineralized domain wireframes.

For the Olympias modeling, the deposit was divided into three zones: East, West, and Flats, shown in plan view in Figure 14-1. Within each of these zones, modeling domains were created using the $50 RDV. Assays and composite samples were tagged by these domain shapes ahead of data analysis and grade interpolation.

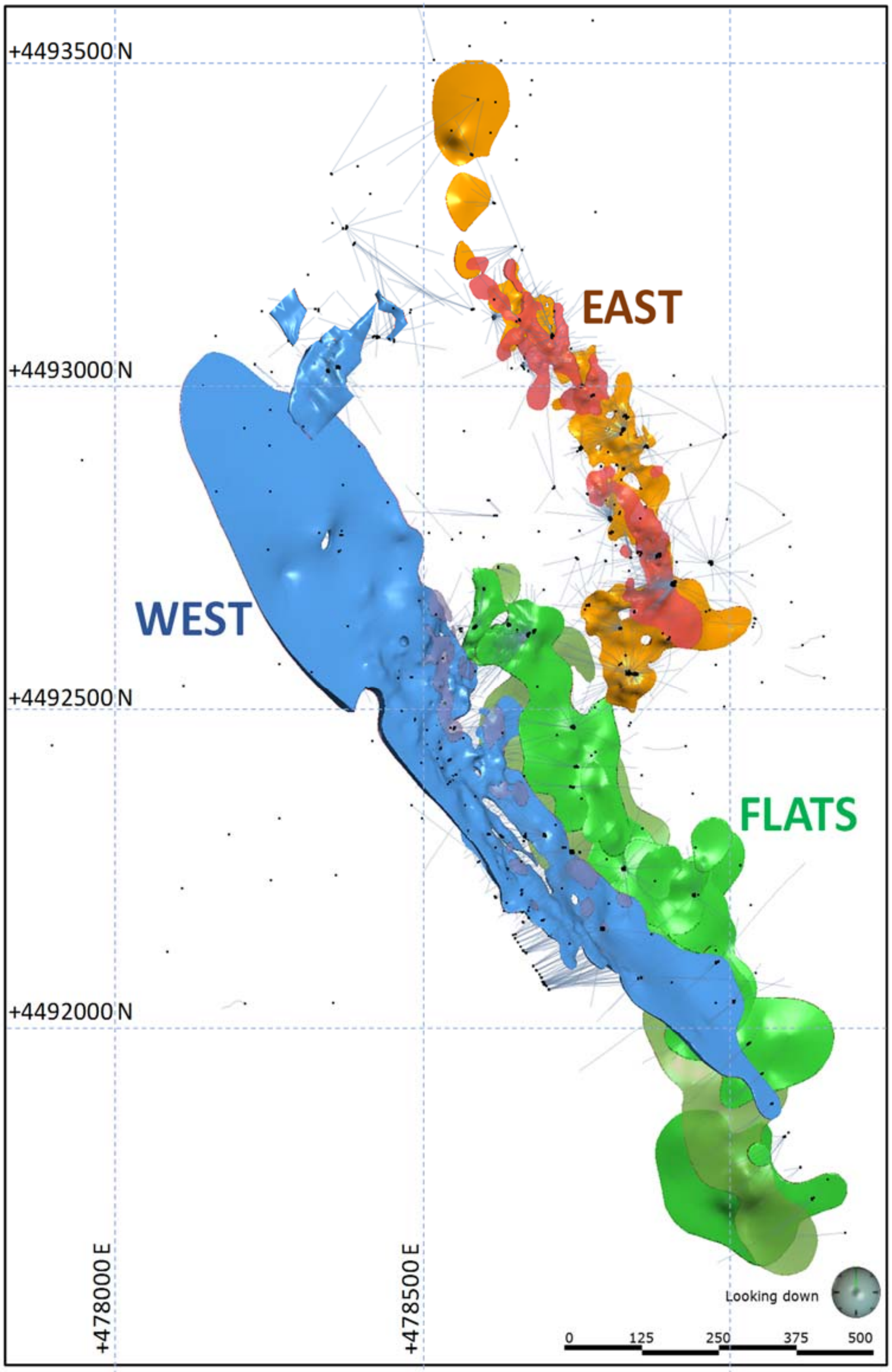


Figure 14-1: Ore Zones, Olympias Mine

14.3 DATA ANALYSIS

The dominant assay sample length inside the mineralization shapes is 1 m. This then became the compositing length. Assay samples were composited to 1 m lengths within each mineralization lenses. During compositing, samples less than 0.25 m in length were merged with the previous sample.

Assays were also composited to 5 m lengths within each mineralization lens. The 5 m composites are used in the nearest neighbour estimation for model validation purposes.

Statistical properties of uncapped 1m composite data for gold, silver, lead and zinc, by zone are shown in Table 14-1 to Table 14-4

Table 14-1: Uncapped Gold Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3511	14.29	0.97	13.81	4.15	10.44	20.56	137.1
FLAST	2889	7.46	1.03	7.66	2.16	5.24	10.54	111.7
WEST	5086	8.52	1.03	8.75	3.01	6.14	11.15	140.3

Table 14-2: Uncapped Silver Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3510	166	1.2	197	23	97	238	1460
FLAST	2889	87	1.5	134	8	31	112	1410
WEST	5078	139	1.3	175	21	76	193	1580

Table 14-3: Uncapped Lead Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3511	5.16	1.16	5.97	0.78	3.14	7.30	43.0
FLAST	2889	3.18	1.46	4.64	0.31	1.35	4.12	39.2
WEST	5083	4.47	1.23	5.48	0.71	2.56	6.38	56.4

Table 14-4: Uncapped Zinc Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3511	5.65	1.08	6.10	0.60	3.42	8.93	33.1
FLAST	2889	4.83	1.13	5.47	0.65	2.76	7.14	35.3
WEST	5083	5.37	1.08	5.79	0.88	3.45	7.96	41.2

14.3.1 Evaluation of Extreme Grades

Outlier sample grades can cause overestimation in the resource model if left untreated. Extreme grades were examined for gold, silver, lead and zinc by means of cumulative probability plots and histograms for each element. Generally, 1.0 to 1.5% of the samples showed susceptibility to extreme grades. The cap value and number of samples capped are shown in Table 14-5.

Table 14-5: Top-Capping Grades

	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	As (%)	Fe (%)
Cap Value	50	900	25	25	25	35
No. Samples	12,673	12,654	12,667	12,669	12,665	11,725
No. Capped	153	134	174	172	190	173
Percent capped	1.21%	1.06%	1.37%	1.36%	1.50%	1.48%

Statistical properties of capped 1m composite data for gold, silver lead and zinc, by zone, are shown in Table 14-6 to Table 14-9.

Table 14-6: Capped Gold Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3511	13.83	0.87	12.09	4.15	10.4	20.55	50
FLATS	2889	7.41	0.99	7.32	2.16	5.24	10.54	50
WEST	5086	8.42	0.95	8.02	3.01	6.14	11.15	50

Table 14-7: Capped Silver Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3510	163.1	1.1	186.9	22.6	96.5	237.5	900
FLATS	2889	86.5	1.5	130	8	31	112.2	900
WEST	5078	136.4	1.2	163.6	21	76.4	193.1	900

Table 14-8: Capped Lead Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3511	5.04	1.09	5.51	0.78	3.14	7.27	25
FLAST	2889	3.13	1.4	4.37	0.31	1.35	4.12	25
WEST	5083	4.36	1.14	4.97	0.71	2.56	6.36	25

Table 14-9: Capped Zinc Composite Statistics for each Zone

ZONE	No of Comps	Mean	CV	SD	q25	q50	q75	Maximum
EAST	3511	5.62	1.07	6	0.6	3.42	8.93	25
FLAST	2889	4.79	1.11	5.34	0.65	2.75	7.14	25
WEST	5083	5.32	1.05	5.59	0.88	3.45	7.95	25

14.4 BLOCK MODEL

The mineral resource was estimated using a block model with limits and size as shown in Table 14-1. MineSight software was used to model the deposit. The block size selected based on the minimum unit for drift and fill mining. Ore blocks were coded to have domain percentages to honor the wireframe boundaries; coded with zones and domain values to control hard boundary between mineralization units and to define various search parameters for the interpolation stage.

Table 14-10: Block Model Parameters

	Minimum (m)	Maximum (m)	Block size (m)	Number of blocks
East	478,105	479,700	5	319
North	4,491,165	4,493,480	5	463
Elevation	-800	60	5	172

14.5 GRADE ESTIMATION

Grade estimates for Au, Ag, As, Pb, Zn and Fe were interpolated using an inverse distance to the power of 4 (ID^4) method. Nearest–neighbour (NN) grades were also interpolated as a declustered distribution to validate the estimation method.

The search orientation and radii were based on drillhole spacing and the orientation and size of each mineralized zone.

A multi-pass approach was instituted for interpolation. The first pass required a grade estimate to include composites from 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.

For the East zone, three estimation passes were used to ensure each block had all grades estimated. During the first pass estimation, the search distances used was 60 x 60 x 10 m in X, Y and Z directions respectively. At this step of the estimation, model blocks received a minimum number of 4 composite samples, maximum of 3 composite from a single hole and total maximum number of composites adjusted as 12. The second pass, which aimed to capture more of the surrounding nearby samples for the interpolation to provide a good local estimation, had a smaller search ranges of 30 x 30 x 5 m in X, Y and Z direction. Maximum number of samples per hole varied between 2 to 5 for different elements, and maximum samples varied from 12 to 24 composite samples. The minimum number of samples required varied between 3 to 6 samples for the different elements. A final large search was employed to place grade for those remaining uninterpolated blocks using a 10-sample maximum for the interpolation and 4-sample maximum per hole.

The Flats zone contains seven separate mineralization lenses. For each lens a large search radius was used as a first pass to ensure each block had a grade estimated. For this pass, blocks received a minimum of 4 and a maximum of 18 composites.

Search distances for the second pass run consisted of 70X x 70Y x 20Z m and lay in the direction of mineralization. Blocks required to have minimum 5 and maximum 16 composite samples for this interpolation, with a maximum 4 samples allowed from a single hole.

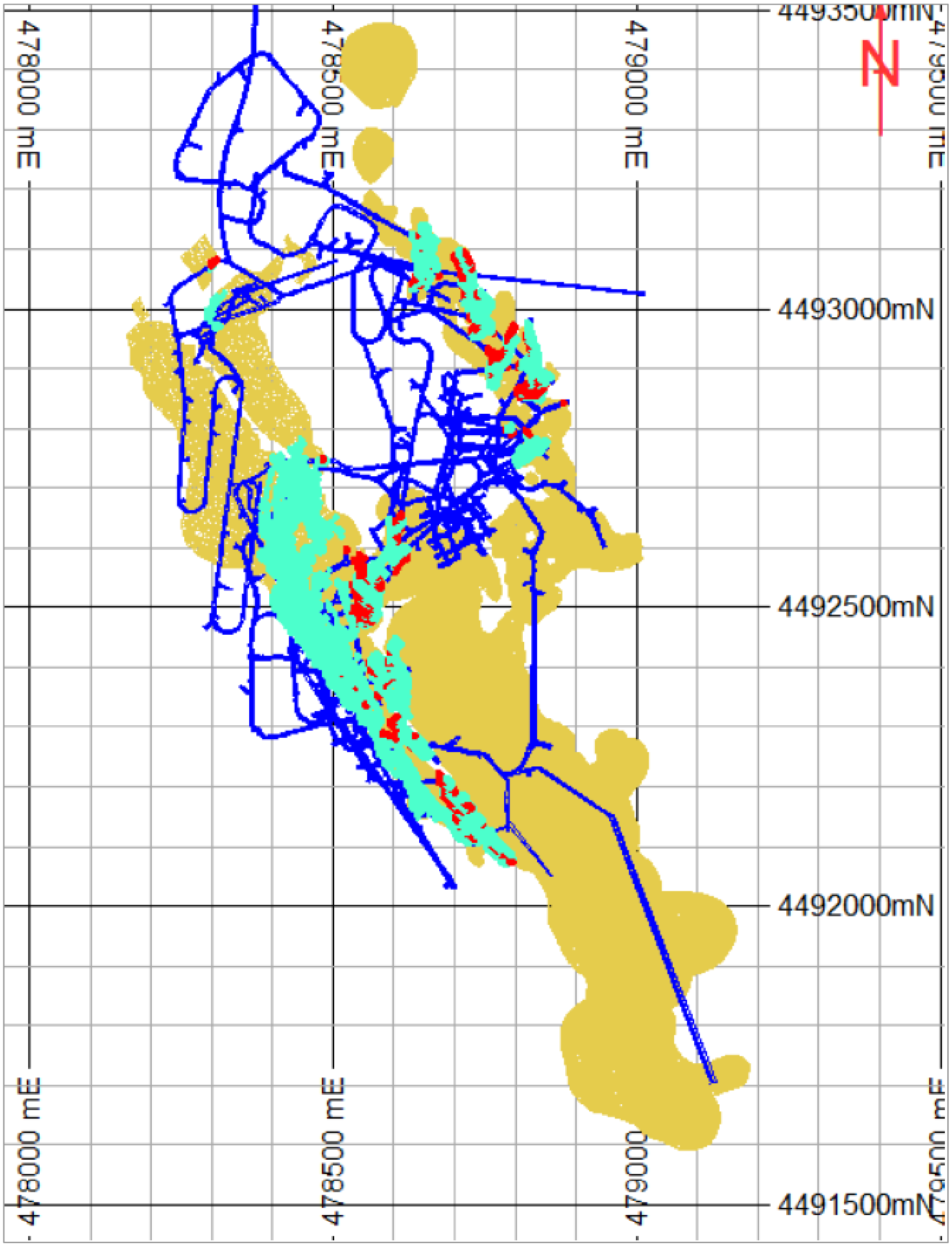
In the West zone, the short range estimation ellipse, comprised 70X x 70Y x 15Z m. Blocks required to have minimum 4 and maximum 18 composites for this interpolation, with a maximum 3 samples allowed from a single hole. The longer ranged pass had a minimum 4 and maximum 12 composites protocol.

In East and West zones, an outlier restriction was used to control the effects of high-grade composites for gold in local areas of less dense drilling. The restricted distances ranged 20 and 55 m with threshold gold grades of 30 g/t (West) and 45 g/t (East).

Bulk density in the block model is function of the estimated Pb, Zn, As and Fe grades. Values for each model block were calculated following the description of the method in Section 10.

14.5.1 Areas of Previous Mining

Areas of previous mining are located in the East and West Zones. Mined out areas do not form part of the stated mineral resource estimate. Figure 14-2 is a plan showing the location of the mineral resources and areas mined out.



Notes:

- Yellow colour equals area of mineral resources.
- Green colour represents areas of pre-Eldorado mining.
- Red colour shows areas mined by Eldorado.
- Mine development shown with blue lines.

Figure 14-2: Location of Areas of previous mining

14.6 BLOCK MODEL VALIDATION

14.6.1 Visual Inspection

Eldorado completed a detailed visual validation of the resource model. Models were 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 views of block model grades (Au) and drill hole composites for each zone are shown in Figure 14-5.

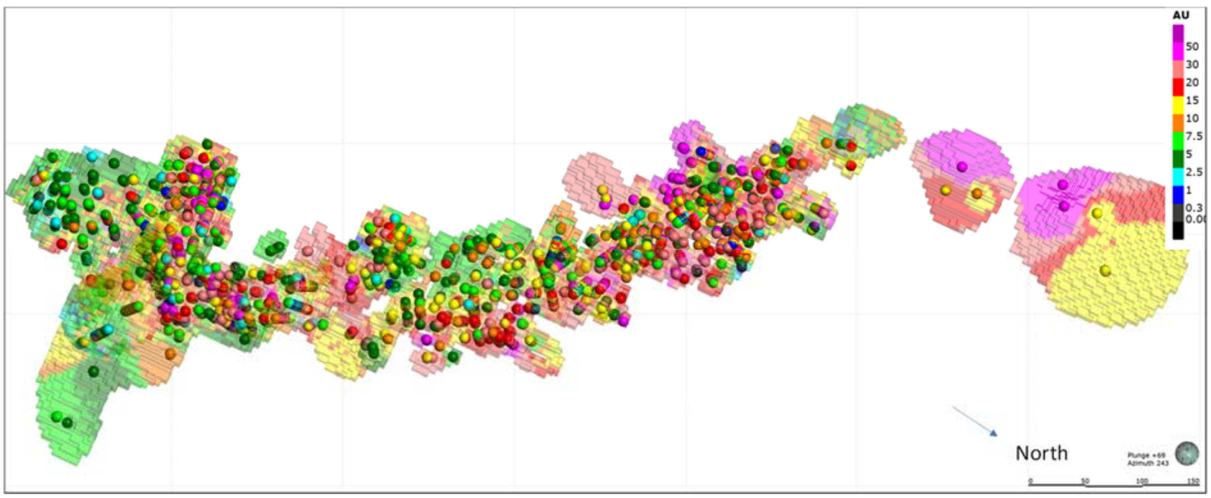


Figure 14-3: 3D Oblique View of EAST Zone comparing Block Au Grades to Composite Au Grades

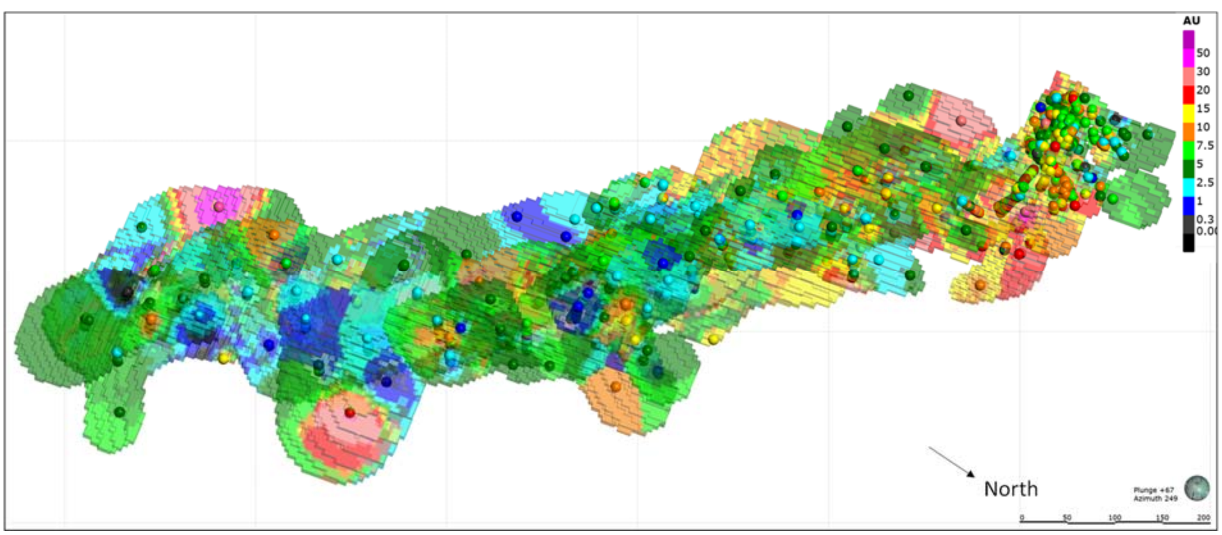


Figure 14-4: 3D Oblique View of FLATS Zone comparing Block Au Grades to Composite Au Grades

Figure 14-5: 3D Oblique View of WEST Zone comparing Block Au Grades to Composite Au Grades

14.6.2 Model Checks for Bias

Eldorado checked the block model estimates for global bias by comparing the average metal grades (with no cut-off) from the ID4 models with means from nearest-neighbour estimates. The nearest-neighbour estimator declusters the data and produces a theoretically unbiased estimate of the average value when no cut-off grade is imposed. It is a good basis for checking the performance of different estimation methods. Results, summarized in Table 14-11 show no global bias in the estimates.

Table 14-11: Global Model Mean Grade Gold Values (g/t) by Domain

ZONE	ID4	NN Estimate	Difference
	AU	**AUNN**	**%**
EAST	13.82	13.66	1.2%
FLATS	6.97	6.93	0.5%
WEST	8.61	8.29	3.7%
	AG	**AGNN**	**%**
EAST	169	172	-2.0%
FLATS	136	140	-2.8%
WEST	149	150	-0.6%
	ZN	**ZNNN**	**%**
EAST	6.4	6.2	2.1%
FLATS	6.5	6.8	-4.4%
WEST	5.3	5.3	-0.3%
	PB	**PBNN**	**%**
EAST	5.4	5.5	-1.1%
FLATS	4.7	4.9	-3.9%
WEST	4.7	4.7	0.0%
	AS	**ASNN**	**%**
EAST	6.8	6.7	1.3%
FLATS	3.5	3.5	-1.5%
WEST	4.0	3.8	6.2%
	FE	**FENN**	**%**
EAST	14.7	14.6	0.8%
FLATS	14.6	14.7	-0.4%
WEST	15.4	15.2	1.1%

14.6.3 Swath Plots

Eldorado also checked for local trends in the grade estimates. This was done by plotting the mean values from the nearest-neighbour estimate versus the ID4 modeled results for benches, eastings and northings in 20 m swaths. Examples for Au, Pb and Zn, along eastings of all three zones are shown in Figure 14-6 to Figure 14-8. Since 5 m composites were used for nearest neighbour (NN) estimation, distribution is already smoothed; therefore, model estimates should be closer or slightly smoother than the nearest-neighbour estimate. The observed trends behave as predicted and show no significant trends in the estimates in models.

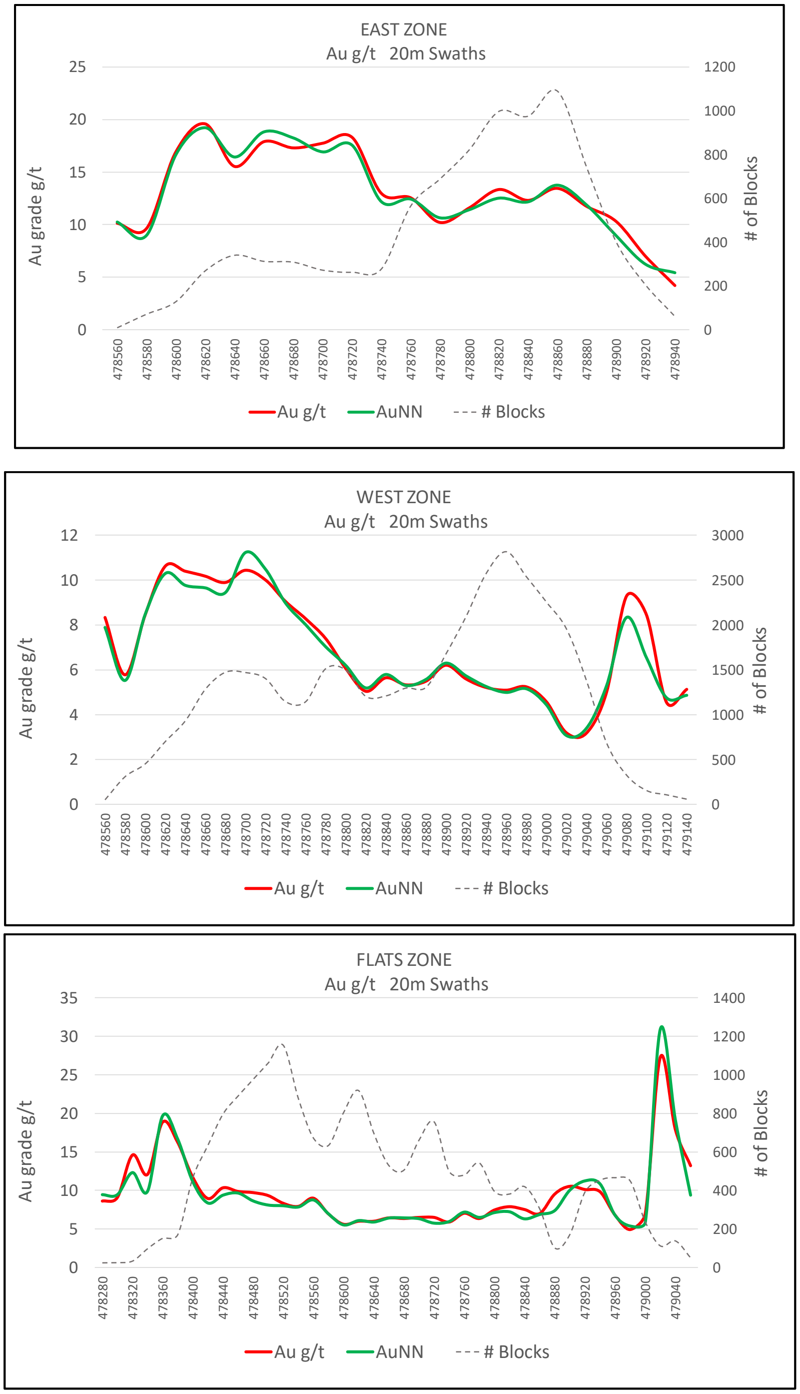


Figure 14-6: Au SWATH Plots

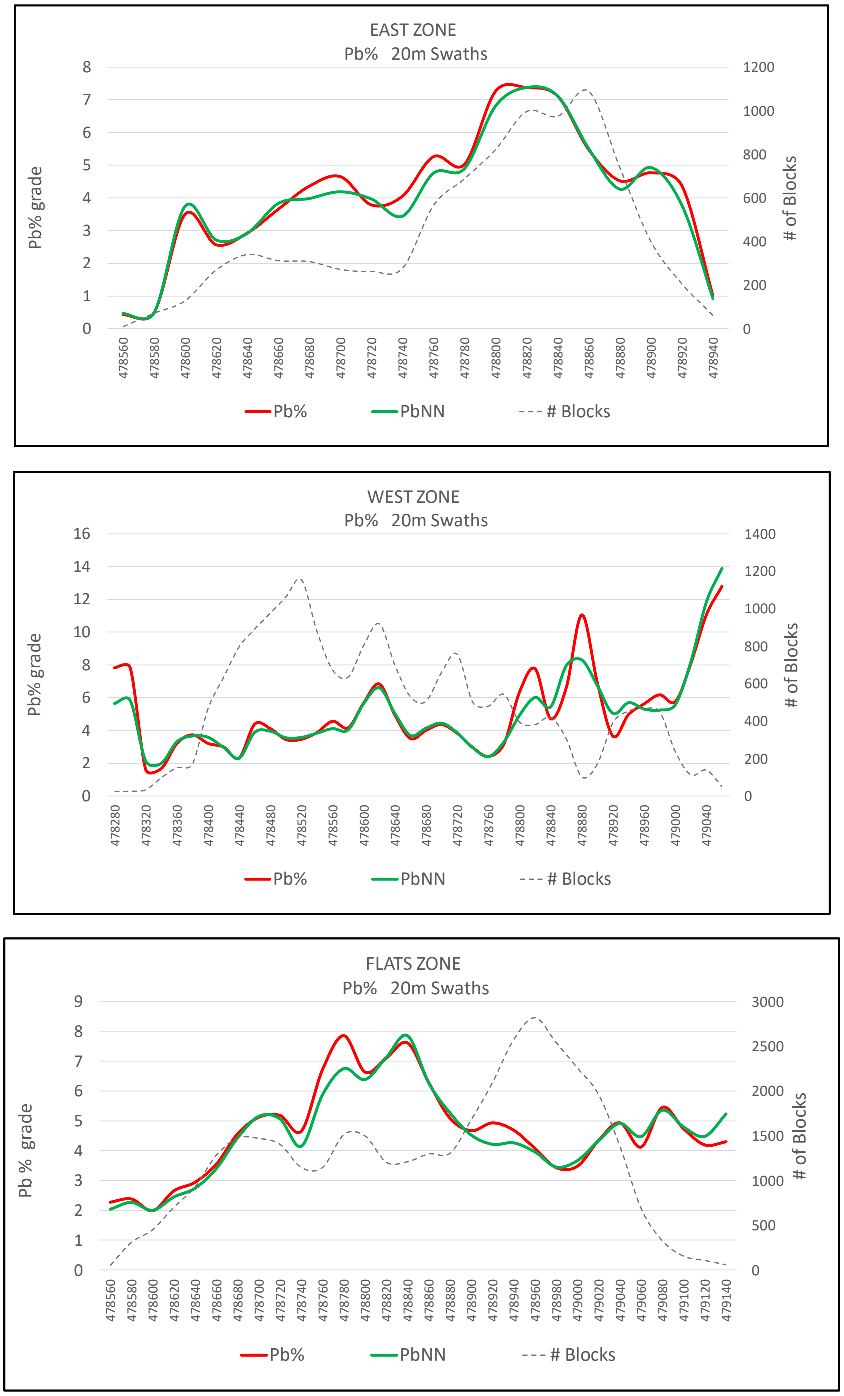


Figure 14-7: Pb SWATH Plots

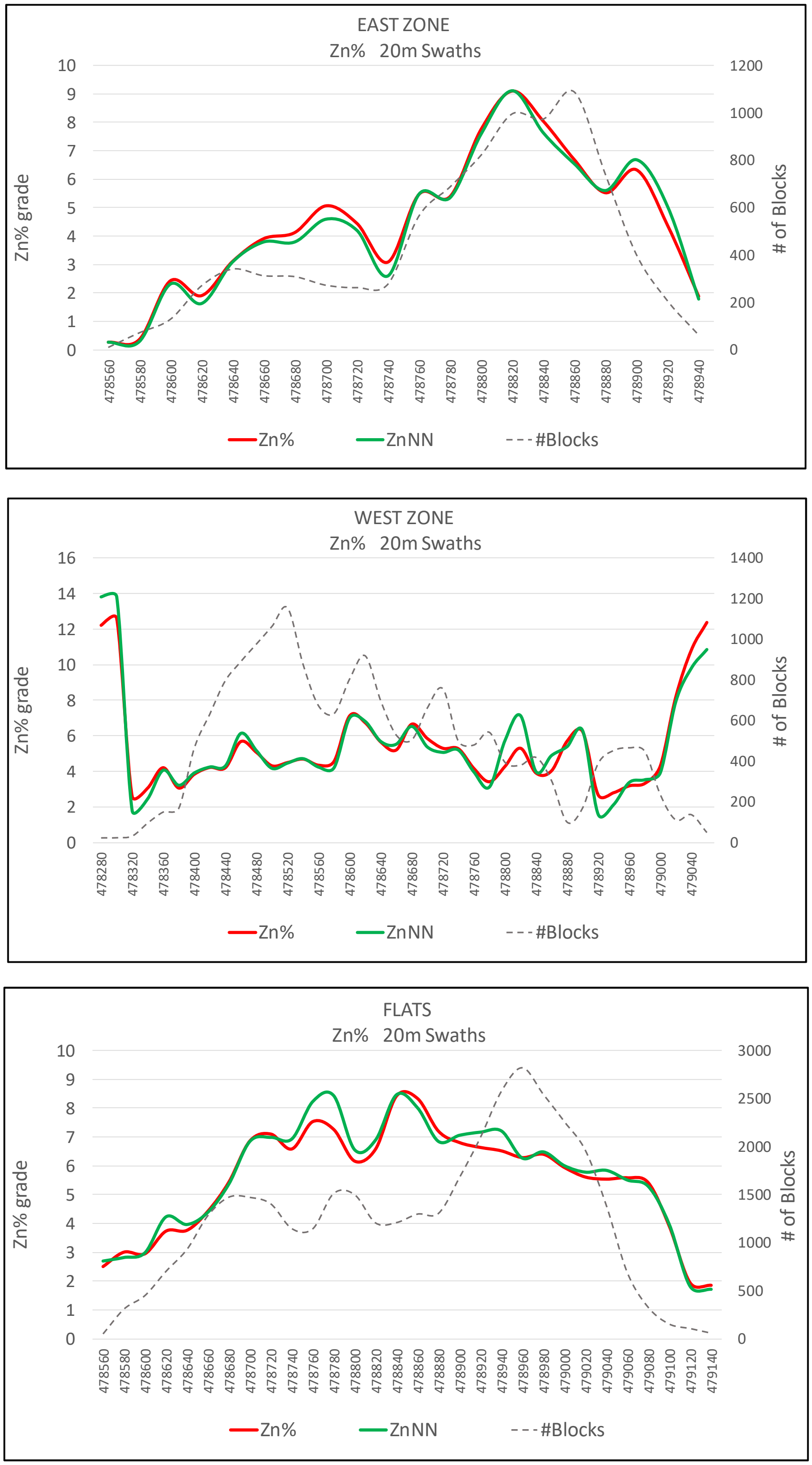


Figure 14-8: Zn SWATH Plots

14.7 MINERAL RESOURCE CLASSIFICATION

The mineral resource was classified as measured, indicated and inferred using logic consistent with the CIM Definition Standards referred to in NI 43-101. The mineral resource was classified as measured, indicated or inferred, based on:

- Location and number of drillholes
- Location of blocks close to areas that were mined

The measured mineral resources were generally located near areas of active mining and within approximately 15 m of at least three drillholes. The indicated mineral resources were generally located a maximum of 45 m from two drillholes (actual average distance is 30 to 35 m). Blocks that honored these conditions for measured and indicated resources were examined in longitudinal section. Polygons were digitally drawn around contiguous areas of appropriately tagged blocks for each class type. These shapes were subsequently used to formally classify the model blocks as measured or indicated mineral resources. All remaining interpolated blocks were classified as inferred mineral resources.

Figure 14-9 to Figure 14-11 show plan views of classified blocks relative to drill holes for each zone.

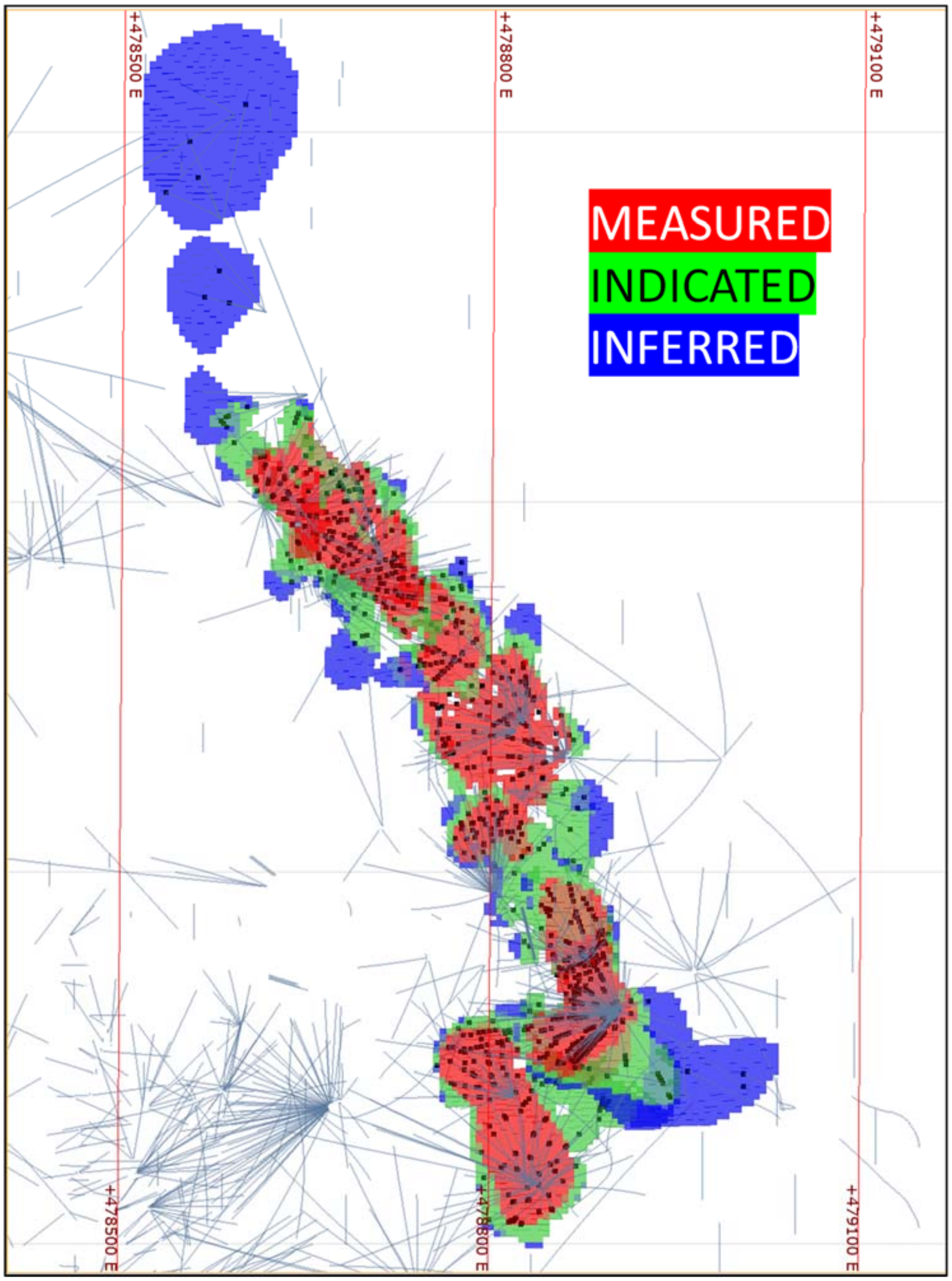


Figure 14-9: East Zone Mineral Resource Classification

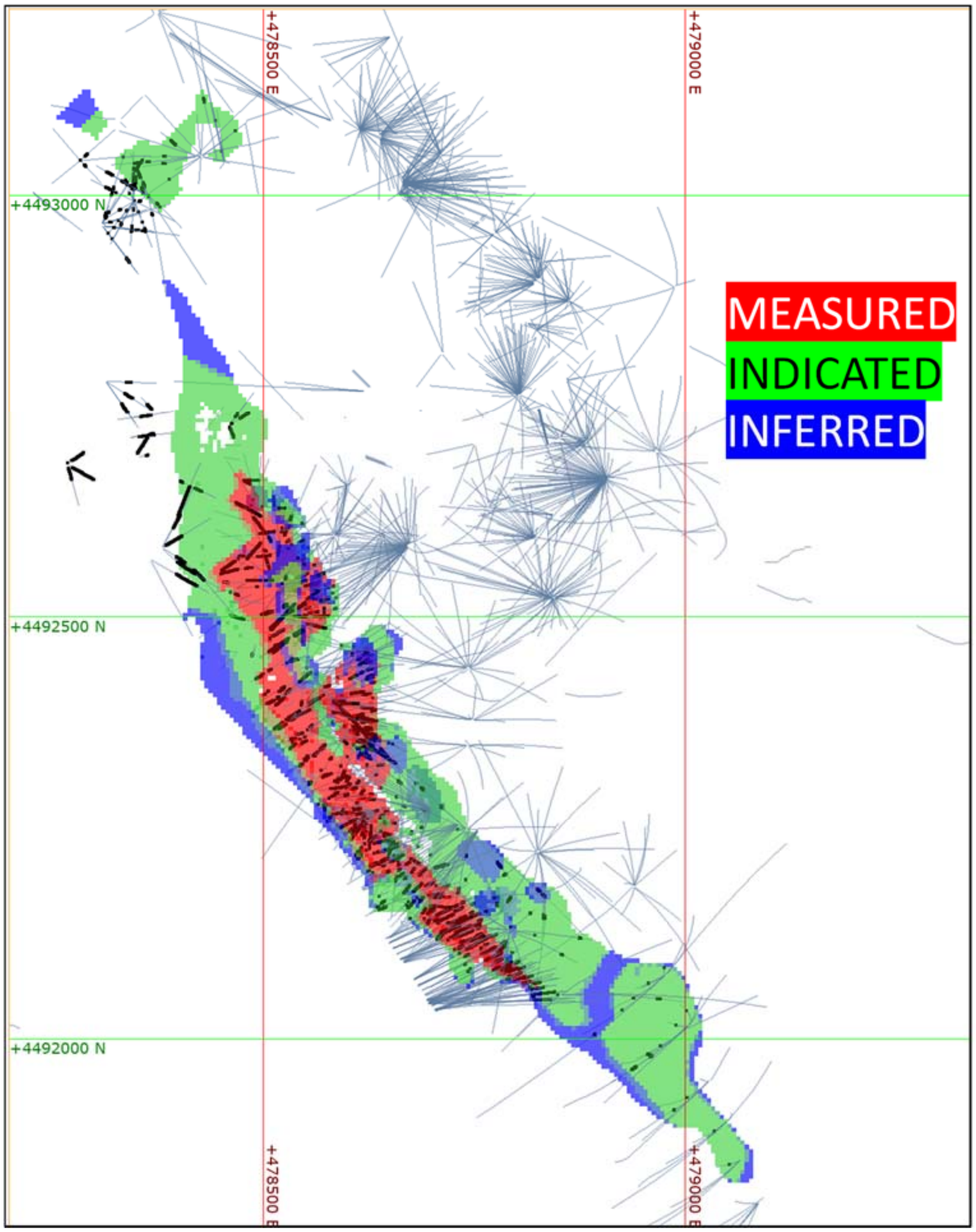
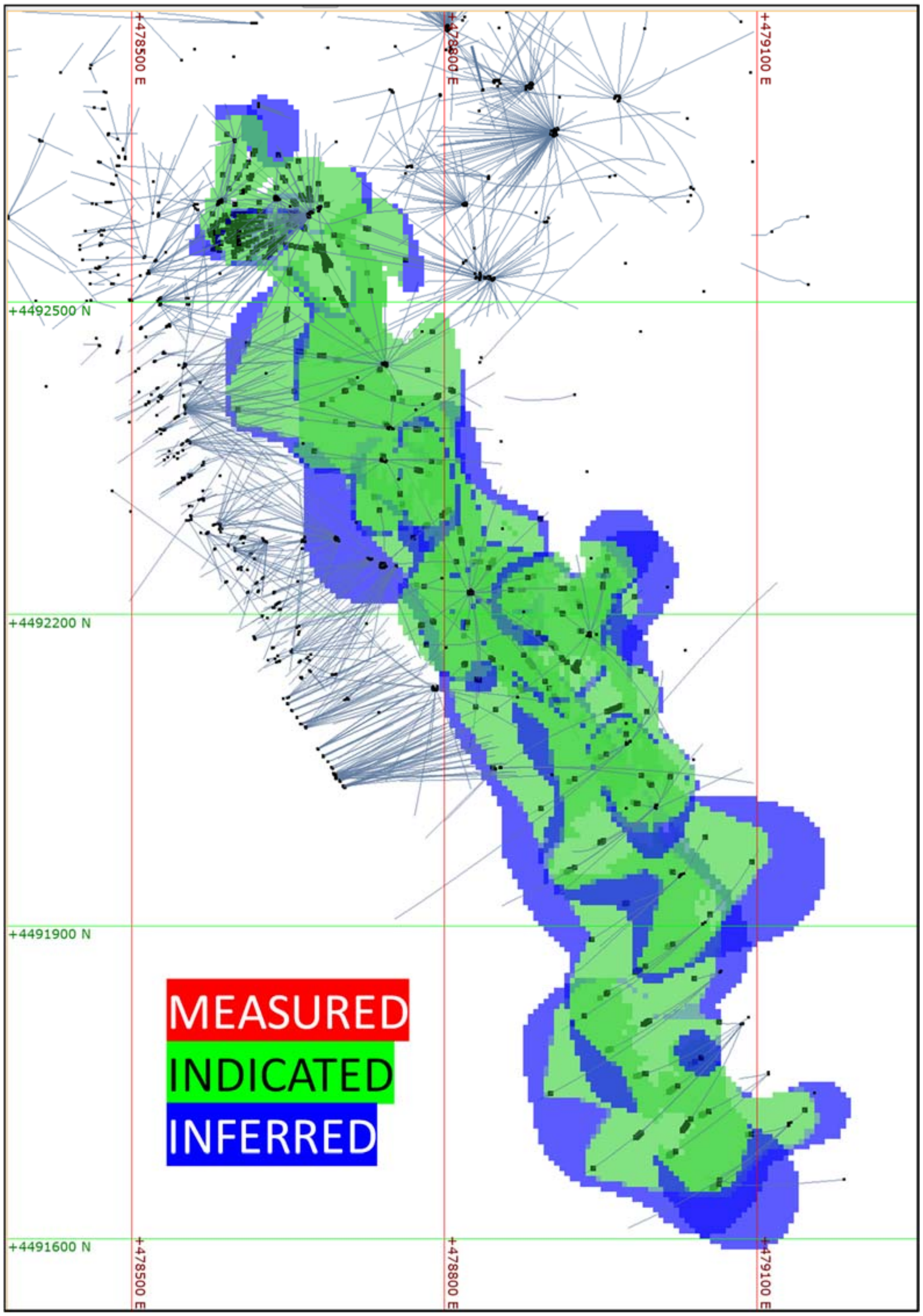


Figure 14-10: West Zone Mineral Resource Classification



Note: Measured Mineral Resources are not defined in the Flats Zone.

Figure 14-11: Flats Mineral Resource classification

14.8 MINERAL RESOURCE ESTIMATE

The Olympias mineral resources as of September 30, 2019 are shown in Table 14-12. All material contained within the mineralization domains, defined by a RDV of $50, are reported.

Table 14-12: Olympias Mineral Resources, as of September 30, 2019

Classification	Tonnes (Kt)	Au (g/t)	Au (Koz)	Ag (g/t)	Ag (Koz)	Pb (%)	Pb (Kt)	Zn (%)	Zn (Kt)
Measured	2,702	10.93	950	156	13,552	5.0	135	6.0	162
Indicated	11,779	7.52	2,848	134	50,746	4.6	542	6.2	730
Measured and Indicated	**14,481**	**8.16**	**3,798**	**138**	**64,298**	**4.7**	**677**	**6.2**	**892**
Inferred	3,720	7.98	954	137	16,385	3.9	145	4.0	149

SECTION • 15 MINERAL RESERVE ESTIMATES

The mineral reserve estimates conform to CIM Definition Standards referred to in NI 43-101. All design and scheduling have been completed using the mineral resource model and estimate described in Section 14. The mineral reserves are as of September 30, 2019.

Only measured and indicated resources have been used for mineral reserves estimation. The estimation assumes that the mining methods employed at the mine will be drift and fill (DAF) and transverse longhole open stoping (TLHOS).

The proven and probable underground mineral reserves for the Olympias deposit are summarized in Table 15-1.

Table 15-1: Olympias Mineral Reserves as of September 30, 2019

Class	Tonnes (kt)	Au (g/t)	Au (Moz)	Ag (g/t)	Ag (Moz)	Pb (%)	Pb (kt)	Zn (%)	Zn (kt)
Proven	2,601	9.19	0.77	133	11.1	4.3	112	5.1	133
Probable	10,324	6.47	2.15	115	38.2	4.0	413	5.3	547
Total	12,925	7.02	2.92	119	49.3	4.1	525	5.3	680

Notes:

- Mineral reserves are included in measured and indicated mineral resources.
- Figures in the tables may not compute due to rounding.
- The mineral reserves are based on a planning cut-off grade of $133/t for DAF and $116/t for TLHOS, and $50/t for development ore.
- Cut-off grades are based on a gold metal price of $1,250/oz, silver metal price of $16/oz, zinc metal price of $2,000/t, and lead metal price of $1,800/t.
- Metallurgical recoveries are based on feed grade and metallurgical algorithms.
- Exchange rate used is €1.20 = US$1.00.
- Average mining dilution and mining recovery factors of 14% and 95%, respectively, for LHOS, and 13% and 98%, respectively, for DAF are assumed.
- Mineral reserves are reported on a 100% ownership basis; Eldorado owns 95% of Hellas Gold.

15.1 CUT-OFF GRADE

The cut-off values supporting the estimation of underground mineral reserves were developed in 2018 and based on future projected operating costs at a steady-state production rate of 650,000 tonnes per annum. The operating cost assessment indicated that NSR values of $133/t for DAF mining and $116/t for TLHOS mining would adequately cover all site operating costs on a breakeven basis. The weight averaged operating cost can be estimated at $125/t considering the balance between TLHOS and DAF. These DAF and TLHOS cut-offs were used to create potentially mineable stope shapes from the NSR block model (NSR BM). The NSR BM was created by Eldorado and is based on metallurgical recovery experience and historical sales and includes recognition of transport and refining costs.

It is recognized that the Olympias mine is currently in a ramp up phase and has not yet achieved the proposed production rate and mining cost targets. Table 15-2 contains target costs over the peak production phase (2024 – 2038) when Olympias will be operating at full 650 ktpa capacity.

Table 15-2: Target Operating Costs

Category	Target
Mining Cost	53.30
Process Cost	50.40
G&A Cost	21.30
Total	125.00

The operational improvement plan includes the following elements:

- Increased equipment availability
- Upskilling and multiskilling the workforce
- Increase in manpower utilization
- Maximizing the use of paste backfill
- Infrastructural projects to increase underground mining efficiency:
    - Underground workshop
    - Slick line to deliver shotcrete
    - Fuel line to underground
    - Underground slimes filtration system
    - Introduction of remote operations and central control system

Eldorado has reviewed the Olympias plans for operational improvements and considers them adequately achievable to support the current mineral reserves estimate. The impact of operational improvements will be closely monitored over 2020 and later years, failure to achieve the target cost performance by 2023 would potentially reduce mineral reserves and shorten the mine life. This is a base case scenario and any subsequent metallurgical processing would enable Eldorado to run another reserve calculation.

Table 15-3 presents the estimated impact to reserves and mine life should it become necessary to raise the breakeven cut-off grade in the future.

Table 15-3: Estimated Impact of Raised Breakeven Cut-off Grade on Reserves and LOM

Class	Tonnes (kt)	Tonnes (kt)	Tonnes (kt)	Tonnes (kt)	Tonnes (kt)
Proven	2,601	2,592	2,543	2,479	2,418
Probable	10,324	9,797	9,467	8,940	8,630
Total	12,925	12,389	12,010	11,419	11,048
Cut-off grade (averaged $/t)	**125**	**135**	**145**	**155**	**165**
Years of Mine Life	21	20	19	18.5	18

Eldorado has also reviewed the material planned to be mined during the ramp-up period prior to the steady-state production value of 650 ktpa being achieved and, therefore, prior to the full benefits of the operational improvements being realized. The value of this material validates its inclusion as part of the estimated mineral reserves.

15.2 MINING SHAPES

DAF stope shapes were generated by Eldorado using the software mineable shape optimizer (DSO) to produce stope shapes based on the NSR cut-off of $133/t, minimum stope width of 5 m, a fixed stope height of 5 m, and a maximum FW / HW angle of 45°. The TLHOS stope shapes were created by Eldorado using a cut-off of $116/t to generate grade shells. The grade shells were used to manually design the stope shapes.

On the outliers of the ore-body, some marginally economic areas were not included in the mine plan and, therefore, is not included in the mineral reserves. Capital and operating development cost estimates were used to determine economic viability for these areas.

15.3 DILUTION AND RECOVERY FACTORS

In the evaluation of underground mineral reserves, modifying factors were applied to the tonnages and grades of all mining shapes to account for dilution and ore losses. In the DAF stopes, a mining dilution factor of 13% and a mining recovery of 98% were estimated. The sources of dilution for DAF will be floor mucking on fill (estimated 0.1 m), and side wall dilution of fill or waste (estimated 0.1 m on each wall) from overbreak and sloughing.

In the TLHOS stopes (predominantly blind-uppers) a mining dilution factor of 14% and a mining recovery of 95% were estimated. The sources of dilution for TLHOS were considered for primary and secondary stopes. In the primary stopes, there will be waste rock overbreak on the back of the blind uppers stopes and end wall fill dilution from the preceding TLHOS (when applicable). The floor will invariably be on country rock and associated dilution is anticipated to be minimal. For the secondary stopes, there will be waste rock overbreak in the back, and end wall and side wall fill dilution from the preceding secondary stope and adjacent primary stope, respectively.

Ore losses (mining recovery factors) are related to the practicalities of extracting ore under varying conditions, including difficult mining geometry, problematic rock conditions, losses of ore into backfill, and blasting issues. Mining recovery has been estimated to be 98% by weight for DAF and 95% for TLHOS.

Figure 15-1 and Table 15-4 present 2019 reserves dilution estimation.

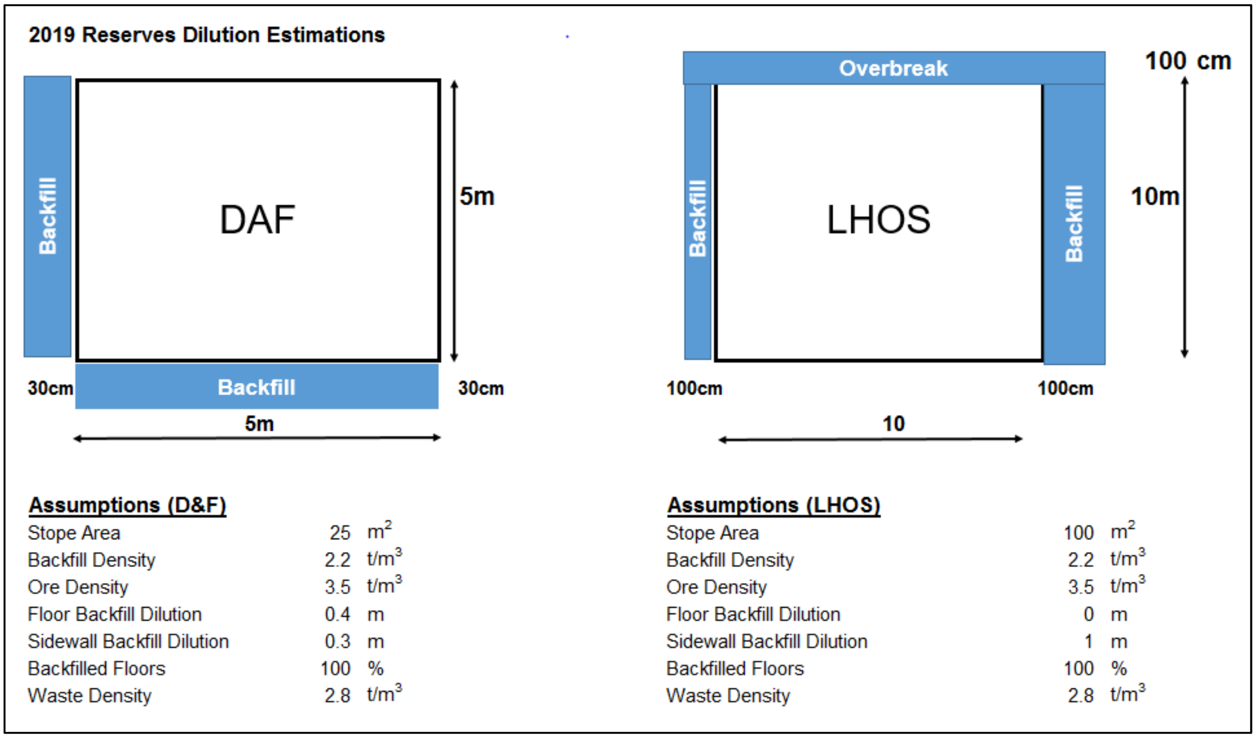


Figure 15-1: 2019 Reserves Dilution Estimation

Table 15-4: 2019 Reserves Dilution Estimation

	Sidewall Backfill				Backfill Dilution (%)			Overbreak	Total
Drift and fill									
Ore Zone	**Frequency**	**Width (m)**	**Height (m)**	**Backfill (m^2)**	**Sidewall**	**Floor**	**TOTAL**	**All**	**All**
Upper East	50%	0.3	5.0	0.75	1.9%	4.8%	6.7%	6.0%	12.7%
Lower East	75%	0.3	5.0	1.13	2.8%	4.8%	7.6%	6.0%	13.6%
Flats D&F	75%	0.3	5.0	1.13	2.8%	4.8%	7.6%	6.0%	13.6%
Middle West	75%	0.3	5.0	1.13	2.8%	4.8%	7.6%	6.0%	13.6%
South West	50%	0.3	5.0	0.75	1.9%	4.8%	6.7%	6.0%	12.7%
LHOS									
Flats LH	50%	2	10.0	10.00	6.3%	0.0%	6.3%	8.0%	14.3%

15.4 RESERVE RESOURCE RECONCILIATION

Table 15-5 shows mineral resource to mineral reserve tonnes and metal content conversion factors.

Total mineral reserve tonnes (proven plus probable) are approximately 89% of mineral resource (measured plus indicated) tonnes. The tonnage conversion from measured to proven is 96% and that for Indicated to probable is 88%. Total metal conversion percentages for gold, silver, lead, and zinc are 77%, 77%, 78%, and 76% respectively.

Table 15-5: Conversion of Mineral Resources to Mineral Reserves

Olympias		Mineral Resources			Mineral Reserves*			Conversion factors*		
		Measured	Indicated	M+I	Proven	Probable	P+P	Prov / Meas	Prob / Ind	P+P / M+I
Tonnes	kt	2,702	11,779	14,481	2,601	10,324	12,925	96%	88%	89%
Au	g/t	10.93	7.52	8.16	9.19	6.47	7.02	81%	75%	77%
Ag	g/t	156	134	138.0	133	115	119	82%	75%	77%
Pb	%	5.0	4.6	4.7	4.3	4.0	4.1	83%	76%	78%
Zn	%	6.0	6.2	6.2	5.1	5.3	5.3	82%	75%	76%

Notes:
*Tonnes and metal content.
Numbers may not compute exactly due to rounding.

SECTION • 16 MINING METHODS

16.1 INTRODUCTION

The Olympias mine is an underground mining operation extracting ore from three zones: East, West and Flats, and two sub-zones: Remnants, and North. These are shown in Figure 16-1 as an isometric view of the Olympias underground zones looking north-west.

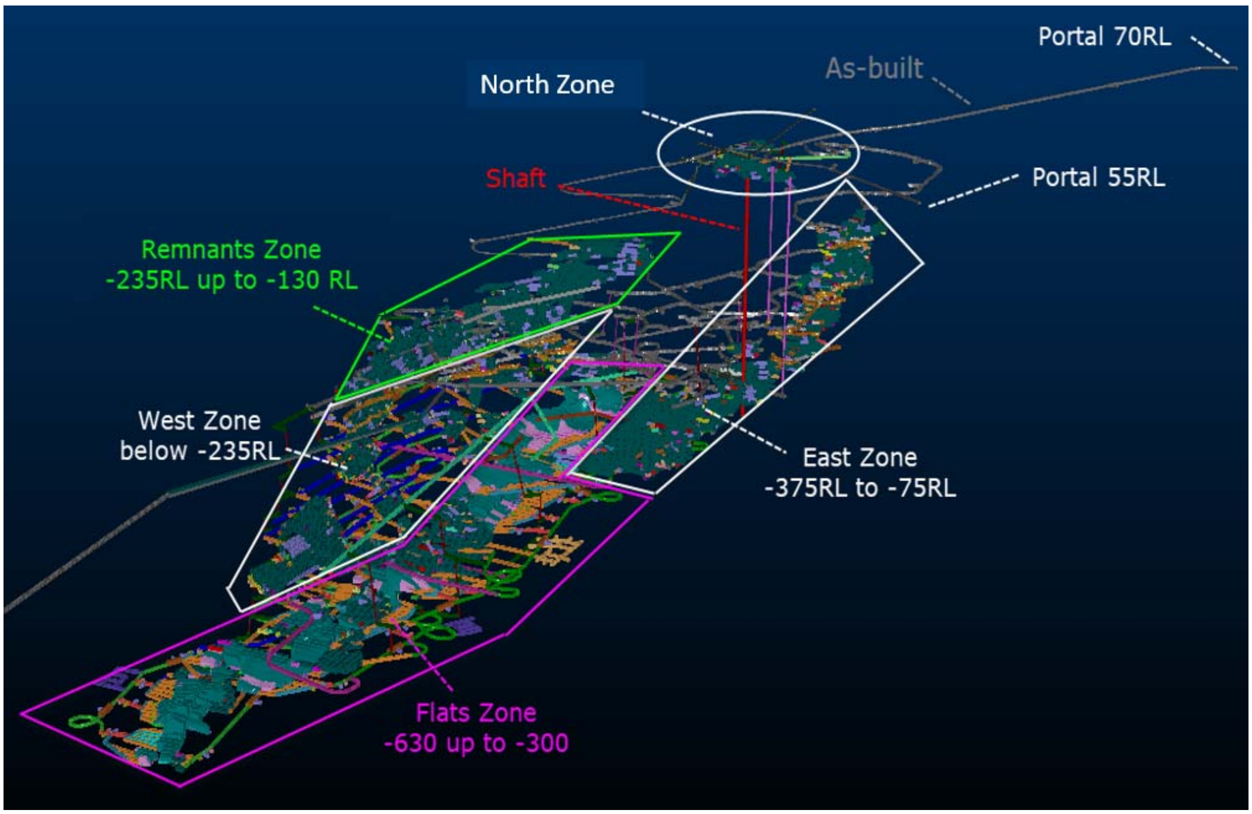


Figure 16-1: Isometric View of Olympias Mining Areas

The East zone is one of three current active mining areas at Olympias mine. It extends approximately 700 m horizontally and has a vertical height of 300 m (-375 masl to -75 masl). The zone dips from 20° to 25° and is mined exclusively using DAF as the mining method. Existing development in the East zone is established at 20 m to 25 m sublevels. The lower East zone (below -240 masl) has not been developed but is included in both short and long-range mine plans.

The West zone is the second active mining area at Olympias; it is also mined entirely as DAF. The zone dips between 45° to 65° and has a horizontal extent of approximately 900 m and a vertical rise of approximately 260 m (-500 masl to -240 masl). Existing development between -280 masl and -240 masl accesses the West zone DAF stopes. Active levels below -280 masl are also currently being mined.

The third active mining area is the Flats zone. This zone dips from 15^0 to 20^0, has a horizontal extent of approximately 1,000 m and elevation difference of approximately 355 m (-275 masl to -630 masl). The current mining area in the Flats is at -280 masl and mining is via DAF. Approximately half of the total mineral reserve tonnage in the Flats is planned to be mined using blind up-hole TLHOS. The Flats currently has limited existing development.

The majority (~90%) of the mineral reserve tonnes will be extracted from these three zones (East, West, and Flats). The remaining ore will be mined from the two other zones, North and Remnants, which are further described below.

The Remnant sub-zone (part of the overall West zone that was mined historically) has a dip of 30° to 50° and is situated on the West side of the Olympias orebody between elevations -235 masl and -130 masl. The zone includes the remaining areas of previously mined DAF stopes, as well as unmined areas that will be removed using the DAF mining method.

The smallest of the mining areas is the North sub-zone that has a dip between 15° and 25° and is situated from -65 masl to -15 masl. This area will be mined as DAF and has no existing development.

Main access to underground is through the existing portal at 70 masl to the north, as well as a portal at 55 masl to the east. An existing shaft (previously utilized for hoisting) extends from surface down to -315 masl. This shaft is not, nor is planned to be, used to transport ore / waste to surface. It is planned to be utilized for ventilation throughout the LOM. The existing underground development plus an increased emphasis on future development will allow the mine to increase the annual production rate from the current 360,000 tonnes up to 650,000 tonnes by 2024. As development progresses, additional stope mining levels will be accessed from lower parts of the mine, in the East, West and Flats, to achieve the production increase to 650,000 tpa.

An average mining rate of 1,800 tonnes per day of ore is planned to reach the steady-state of 650,000 tonnes of ore per year. Mine life is projected to be 21 years for the total mineral reserve of 12.9 million tonnes.

16.2 GEOTECHNICAL CONSIDERATIONS

16.2.1 General Description

The ground conditions in the West zone vary considerably. The majority of the poor ground is within the orebody or at the HW contacts due to the presence of the Kassandra fault and strong to intense clay alteration. The fault and weak rock mass strength may cause instability or HW overbreak / sloughing. The West Zone is deemed not to be suitable for longhole mining and the drift and fill method “DAF” is used throughout, with the length of the cut varying depending upon the conditions. The DAF method is also used in the East Zone and in the Remnants area where the dip and the orebody geometry dictate the preferred method rather than ground conditions.

The overall rock mass quality in the Flats zone appears good based on field observation and review of core photos and the analysis of three recently completed geo-technical drill holes in 2019. If the ground conditions observed and the cores examined and from photos, are representative of the overall geotechnical conditions in the Flats zone, longhole mining can be anticipated to be appropriate.

16.2.2 Longhole Stope Stability

Information obtained from the three 2019 geotechnical drill holes contributed towards a rock mass rating system to quantify the geotechnical properties in prospective longhole areas. Key data from these drill holes are shown in Table 16-1.

Table 16-1: Results from logging of geotechnical drill holes in Flats Zone

Domains	Density (KN/m^3)	UCS (MPa)	Poisson's Ratio	Young's Modulus (GPa)	Rock Quality Designation (RQD %)	Rockmass Rating (Q system)
GN (Gneiss)	27	132	0.31	30	86.2	22.79
MA-10A (Marble)	26,5	74	0.31	32	76.7	19.01
MA-IWZ (Marble)	26.8	79	0.27	38		
MZ (Massive Sulphides	37.5	62	0.28	27	45.3	12.28
PE (Pegmatite)	25.6	132	0.25	29	77.5	44.38

These data allowed for an empirical stope stability analysis based on the stability graph method. This method considers the geometry of the stope plane (the wall, the back or the end) as described by its hydraulic radius, and the geotechnical condition expressed as a modified rock mass rating designation N', which takes into account the orientation and persistence of joint sets, the orientation of the stope plane which is being evaluated, and other geotechnical parameters. The results obtained from the data are plotted against empirical data collected and normalized to fit the model. This creates a spectrum of stable to unstable spans and are a means of determining the predicted stability and the type of ground support if required.

The inputs into the stability graph method used to analyze the proposed longhole stopes in the Flats zone based on the average geotechnical parameters are shown in Table 16-2.

Table 16-2: Inputs to the Stability Graph Method

Stope Span	Span	Length	UCS (Mpa)	Stress (Mpa)	RQD	Jn	Jr	Ja	Q'	A	B	C	N'	Stope HR	Stable HR
Backs	10	20	80	36	87	4	1.5	1.0	32.5	0.12	0.3	2.0	2.4	3.3	3.3
Walls	25	20	60	36	60	4	1.5	2.0	11.3	0.1	1.0	8.0	9.0	5.6	5.3
Ends	25	10	60	36	60	4	1.5	2.0	11.3	0.1	1.0	8.0	9.0	3.6	5.3

The resultant stability graph is shown in Figure 16-2. The analysis shows that the proposed longhole stoping geometries will be stable with up to 20 m height and a span width of up to 10m. The results using the averages from the rock mass ratings suggest that support will not be necessary to maintain these stope geometries although both the sidewall and the backs of the stopes points lie close to the support necessary transition zone boundary.

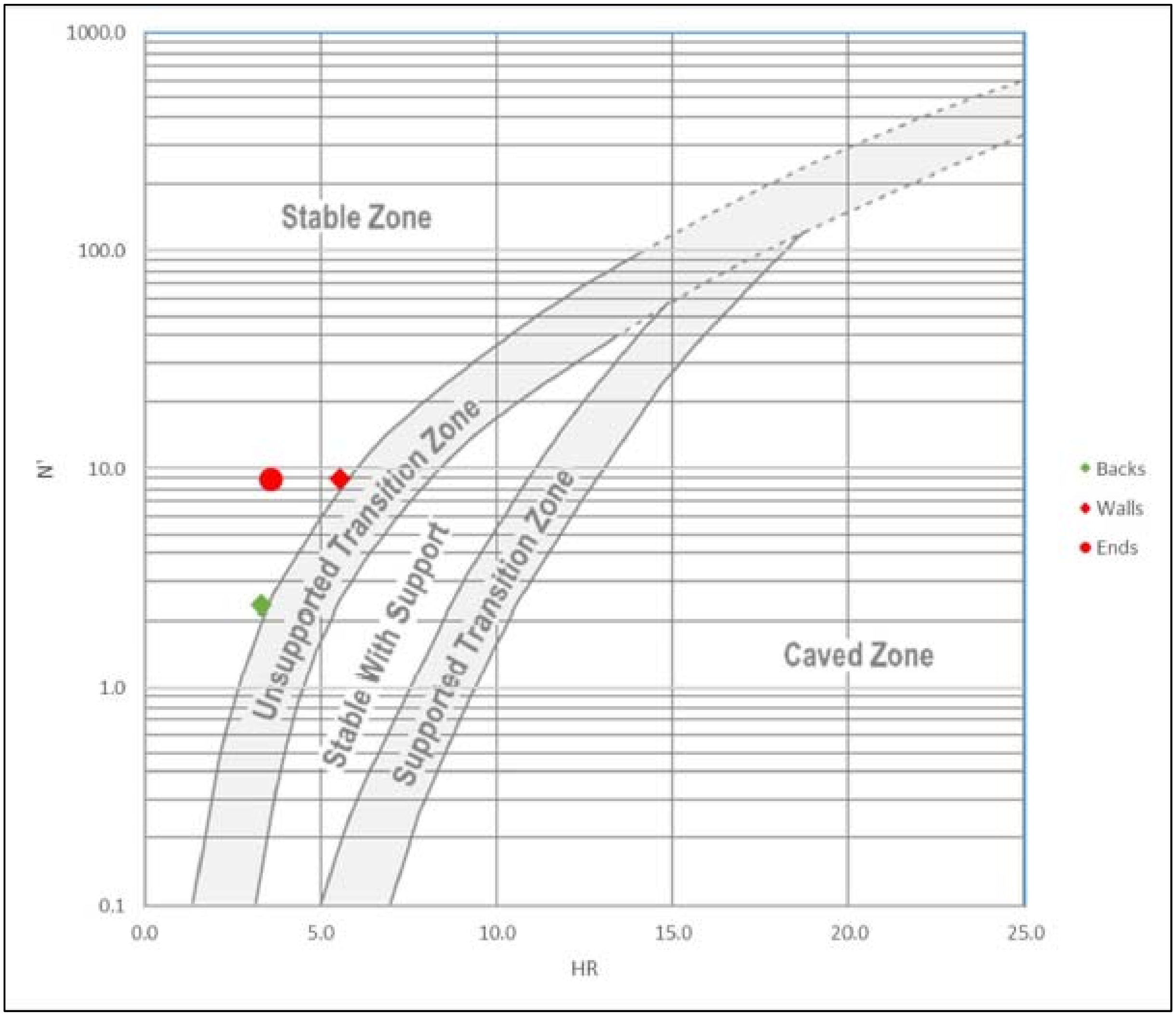


Figure 16-2: Empirical Stope Stability Analysis using 50th percentile Geotechnical Data

Since no longhole open stoping has been done at Olympias mine since the current Phase 2 has been operating (it was used by previous owners) and is currently scheduled to take place in Olympias in the Flats zone. Eldorado is looking to test longhole stoping to confirm the geotechnical behavior of likely areas for LHOS.

16.2.3 Ground Support

Field observations supports the current Olympias ground support system. The implementation of ground control at the mine appears adequate, with no unsecured ground being noted and support performance apparently good. Regular QA/QC (rock bolt pull testing and shotcrete compressive testing) is conducted to ensure quality installation of the ground support and the effectiveness of the support system.

Based on the rock conditions at Olympias mine, the ground support class of Fair to Good may be predominantly applied. The underground openings appear stable, being generally supported with 50 mm of shotcrete and 2.4 m long Swellex bolts on a 1 m by 1.5 m spacing. The dominant condition in ore is similar to slightly weaker than that of the hosting waste rock.

Pull testing on 1% of installed bolts is carried out and the overall bond performance of Swellex bolts in poor ground is satisfactory, which implies that the poorer quality of rock mass is of limited width and extent. In particular it appears that any clay alteration is not extensive and the frictional bolts have a good bond strength at the interface with competent rock. Therefore, the integrated ground support (shotcrete and frictional bolts) is seen to be effective in the observed weak ground and mixed ground conditions.

16.3 MINING METHODS

The mining methods planned for future operations at Olympias mine are DAF (the current method) and TLHOS. DAF is envisaged to be used in all mining areas (East, West, Flats, Remnants, and North), whereas TLHOS is projected for use in the Flats. The overall geometry of the Olympias orebody and the planned locations of the two types of mining methods are shown in Figure 16-3 as an isometric view of the Olympias underground zones looking north-west. The beige colouring indicates planned areas of DAF, the magenta TLHOS.

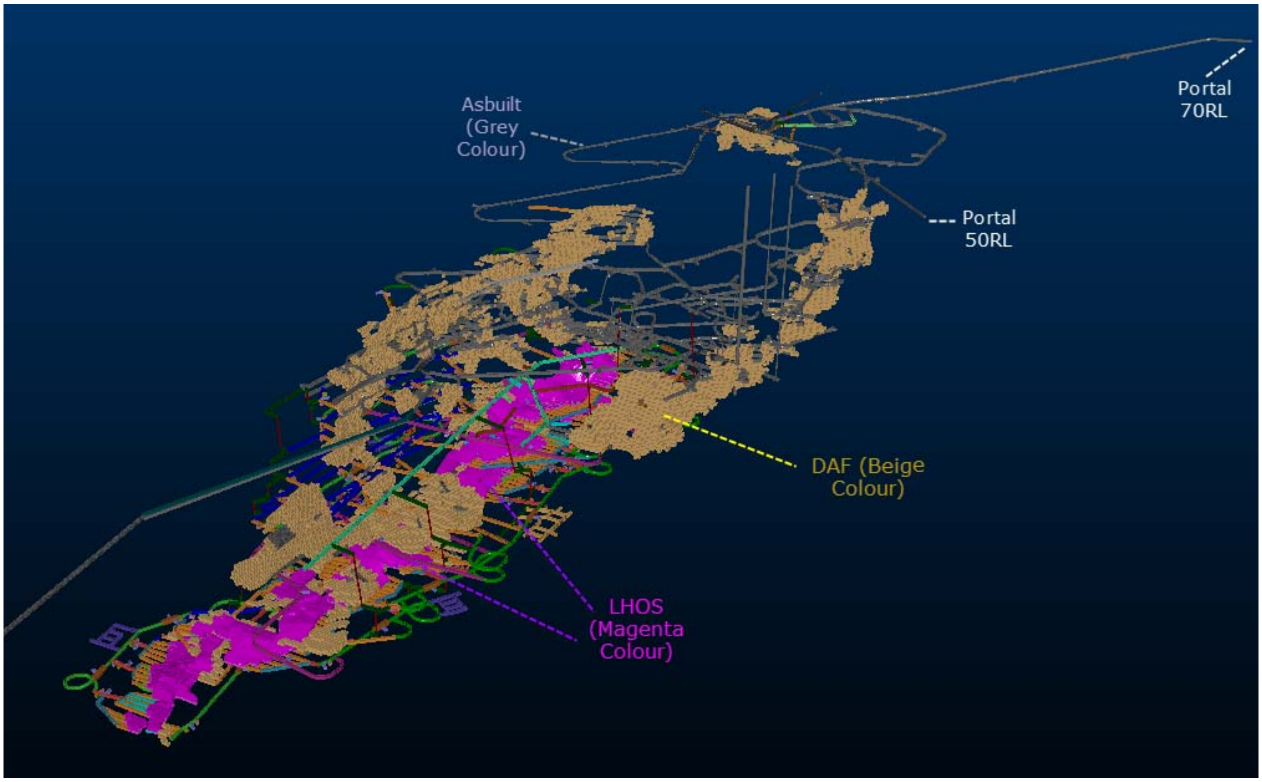


Figure 16-3: Isometric View of Olympias Underground – Mining Methods

16.3.1 Blind-Uppers Transverse Longhole Open Stoping (TLHOS)

Blind-uppers TLHOS will be utilized only in the Flats zone. Flats are gently dipping (15^{o} to 20^{o}) tabular lenses with thicknesses up to 30 m. TLHOS is to be used in over 50% of this zone.

An isometric view of typical primary and secondary TLHOS stopes is shown in Figure 16-4. The ore is accessed through a FW drive and by individual accesses to each stope from the drive. The accesses are driven to the ore contact, then the remaining development in ore is driven at 5.0 m W x 5.0 m to the FW contact. A blind-uppers (conventional or large diameter) raise is driven at the end of the TLHOS followed by longhole drilling of the production rings.

Slot blasting will occur once the raise and production rings are completed and after the backfill curing of the preceding stope. The ore will be mucked from the stope undercut drive with a mixture of manual mucking when the brow is choked with ore, and remote scoop operation when it is not choked-off. Confirmation of complete extraction of the stope by operations supervision will initiate technical services to complete the stope cavity monitor survey (CMS). After the CMS, an engineered fill fence will be constructed back from the brow of the stope and paste filling will proceed until the stope is tight-filled. Other stopes will be mined following the same sequence.

Based on the current geotechnical assessment the primary and secondary stope dimensions will be 10 m wide (constant), with the maximum height being 30 m (undercut floor elevation up to the back of the blind uphole stope). The designed maximum unsupported stope strike length varies based on the stope height and average rock quality for the stope back.

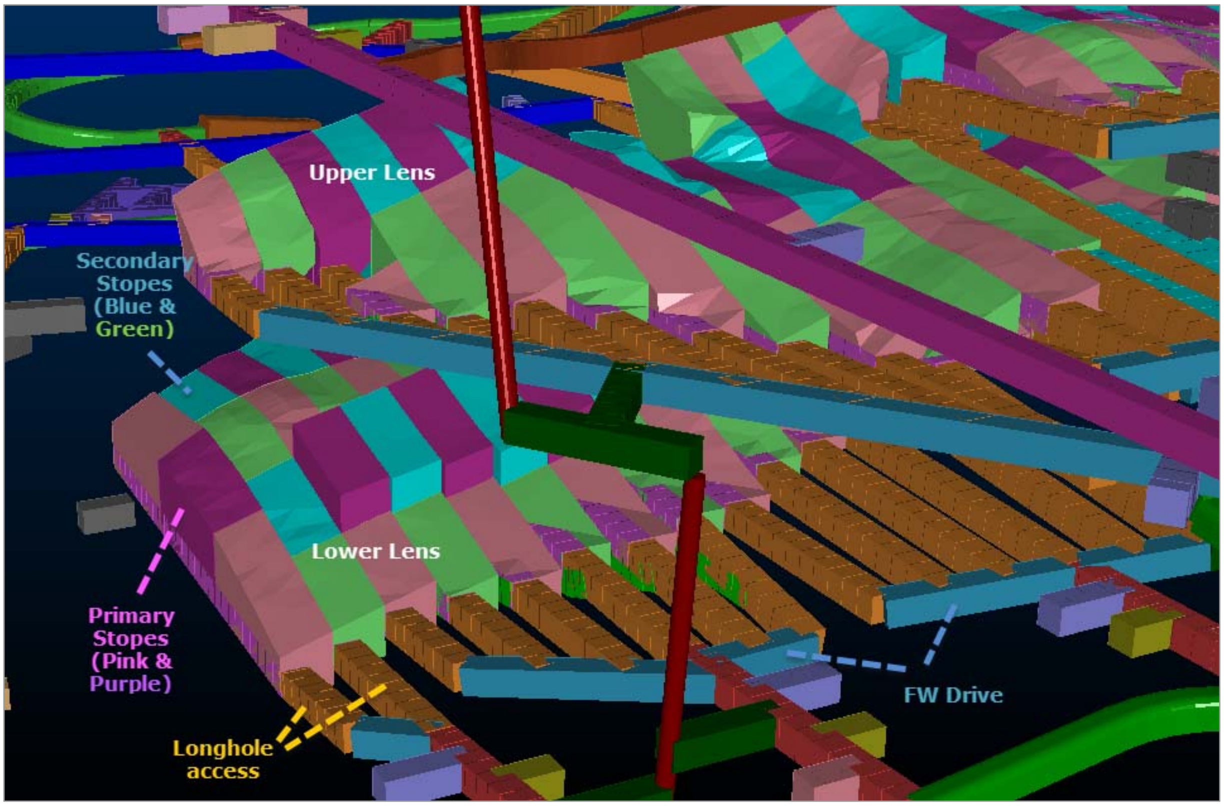


Figure 16-4: View of typical primary secondary Longhole Stopes in the Flats

16.3.2 Drift and Fill (DAF)

DAF at Olympias commences once the decline reaches the footwall drive or Level access elevation of the orebody, usually midway along its strike length (see representative sketch as Figure 16-5). DAF is an overhand mining method. The stope sequence begins with the lowest 5.0 m high lift. Then each subsequent lift requires the back of the level access to be slashed down (‘take-down-back or TDB) to reach the next lift. There are four lifts between levels for a total rise of 20 m from each access.

Future DAF mining is expected to be either in single or multiple passes depending on orebody geometry; this is further described below.

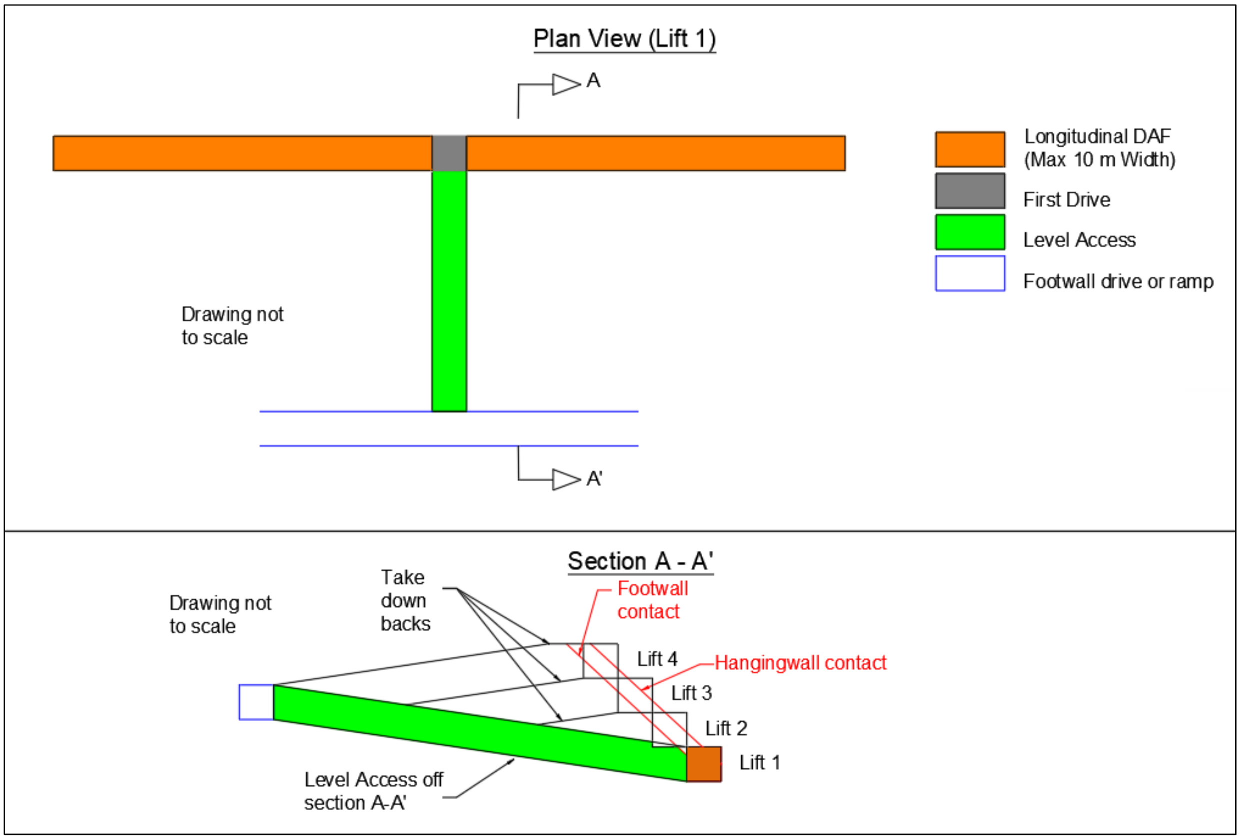


Figure 16-5: Longitudinal DAF

16.3.2.1 Longitudinal DAF

Generally, for FW to HW thickness of 10 m or less the stope will be developed as longitudinal DAF. The mining begins by driving the Level access to the FW contact of Lift 1 and then the drive is extended flat (zero gradient) to the HW contact of the ore. Next, the ore is mined longitudinally in a single pass along strike in both directions to the limits of the orebody. Any remaining ore on the HW side will be slashed out on retreat and then the drift will be backfilled with pastefill (PF) or lesser cemented aggregate fill (CAF). After the fill has cured, the Level access will be slashed down to provide access for the next lift, and the process will be repeated for subsequent lifts.

16.3.2.2 Multi-Pass DAF

Multi-pass DAF is generally for orebody widths greater than 10 m from FW to HW. The mining begins by driving the Level access to the FW contact of Lift 1 (Figure 16-6). Then the drift is extended to the HW contact of the orebody. The Primary drift will be mined (note: generally, all multi-pass DAF drives are planned to be 5.0 m H x 5.0 W) on the FW contact to the extremities of the orebody. The drift will then be filled mainly with PF. After the fill has cured, the Secondary drift will be developed parallel to the Primary drift, with fill on one side and ore on the other side of the drift. After filling and curing is complete for the Secondary, a Tertiary drift is driven beside the Secondary drift, followed by filling and curing. This process is repeated until the HW is reached, which is the Quaternary drift in Figure 16-6. Once the fill has cured for the Quaternary drift, the Level Access will be slashed down to reach the next lift and the process will be repeated for the remaining lifts of multi-pass DAF.

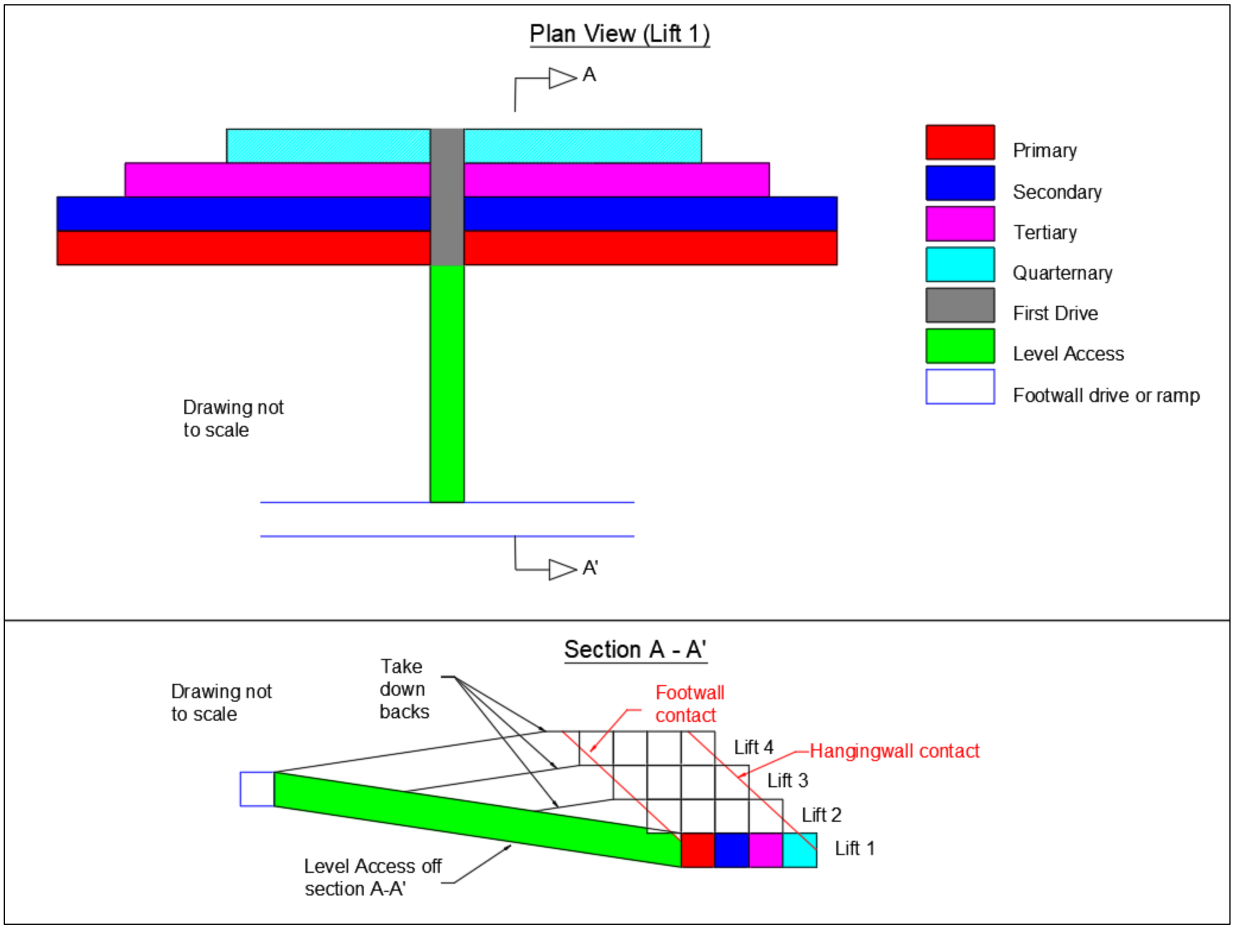


Figure 16-6: Multi-Pass DAF Stope

16.4 BACKFILL

Olympias is currently using CAF and PF for underground backfill operations. The PF system is intended to provide most future backfill for the mine.

The CAF is prepared in a batch plant adjacent to the portal. Aggregate and sand are combined with 8% binder to produce CAF with a 7-day strength of 1 MPa and 28 day-strength of 4.0 MPa. The CAF is backhauled in 40 t trucks and tipped at the entrance to the DAF production area. The CAF is then placed with a loader and, using a pusher arm attachment, tight filled to the back of the drift.

The PF system has been designed to meet future backfill requirements up to the 650 ktpa production rate and operating at up to 70% utilization. PF will be used in DAF and longhole stopes. Using dewatered flotation tailings and cement, the plant can produce 42 m^3/hr of PF. When backfill is not required, dewatered tailings are stockpiled and reclaimed with a front-end loader during the next fill run. Paste can be placed in continuous runs of up to 2,500 m^3.

PF is delivered by positive displacement pump via drill holes and pipelines to the stopes. For control purposes, pressure is monitored in the pipelines by instrumentation. PF is retained in stopes using shotcreted barricades. Tight filling is achieved by managing delivery via in-stope pipelines or drill holes, and location of barricades.

The current operational permit dictates a backfill directive for Olympias that recommends using a strength of 4 MPa for all backfill placed underground. Using best available design practice, Eldorado calculates that a maximum strength for 5 m x 5 m wide drift sill beams, at a Factor of Safety (FOS) of 3.0, should be 1.4 MPa and suggests that the 4.0 MPa requirement is overly conservative.

The backfill directive also states that vertical walls of cured backfill must not be exposed until the strength has reached 1.0 MPa. Using best available design practice, Eldorado calculates the maximum strength for 5 m high x 5 m wide drifts to be only 300 kPa at a FOS of 1.5.

Eldorado recommends that Olympias mine initiate a backfill geotechnical study to optimize the strength requirements of undercut sills and drift sidewalls using a combination of empirical and numerical modelling techniques. It further recommends that Olympias then prepare a technical brief for submission to the relevant authorities on an optimized backfill strength design, using current industry best practice, for stable openings of cured fill.

16.5 MINE DESIGN

16.5.1 Underground Access

A FW decline from the surface Portal at +70 m down to -300 masl connects with the new Olympias decline (the West decline) that will eventually be extended to the base of the orebody at -630 masl of the Flats zone. The FW decline and the West decline extension that is currently being developed, are used as a main haulage route.

A second main decline is currently being driven from -300 masl on the east side of the Flats zone and will be extended down to the -540 masl. This East decline will be used as a haulage ramp for the Flats and lower part of the East Zone.

The existing development above the Flats zone has twin declines and cross-over drifts for personnel and equipment to provide alternate haulage routes and secondary egress. Similamasly, the undeveloped East and West declines will also have multiple cross-over drifts in the Flats zone to provide alternate haulage routes and secondary egress from the bottom of the mine to surface.

16.5.2 Material Handling

Currently all ore and waste is hauled via 40 tonne trucks to surface. A material handling trade-off study was completed in 2018 and determined that, for the 650 ktpa expansion, this continues to be the most cost-effective solution for handling ore and waste.

16.5.3 Personnel

The mine currently operates 24 hours per day, 7 days per week on 3 shifts per day. One shift is lost per week (Sunday night) for a total of 20 worked shifts per week. Support staff at the mine works only a single shift.

The operation has a full complement of workers at the moment and extra underground workforce will be required (approximately 40 personnel) to operate at the 650,000 tpa proposed.

16.5.4 Underground Mining Fleet

The current mobile equipment list for underground mining is shown in Table 16-3. For the production increase to 650 ktpa, additional equipment will be required, including longhole drills, jumbo, bolters, trucks and loaders. These are also listed in Table 16-3.

Table 16-3: Olympias Mine Equipment List

Equipment	Current	650 ktpa
Jumbo – 2 boom	4	5
Bolter	2	4
Long hole drill – ITH	0	2
Haul trucks – 40 tonne	5	7
LHD – 14 tonne	4	7
LHD – utility	2	2
Transmixers	4	4
Shotcrete Sprayers	2	2

16.6 VENTILATION

16.6.1 Ventilation Strategy

The ventilation design is based on an exhausting system configuration with the main surface fans located at the single exhaust raise. At present, fresh air for Olympias is supplied from the +59 Portal, +70 Portal, Shaft and Fires area. Exhaust is via the main exhaust raise at -173 level via three 132 KW fans. This arrangement enables a total of 150 m^3/s to be available for total mine ventilation.

The planned LOM ventilation strategy, with a main exhaust raise an additional exhaust through +70 Portal, will increase the available airflow for the mine to approximately 360 m^3/s.

Return air is exhausted from the mine through internal raises adjacent to each ore block before feeding the -173 Level for exhaust into the primary exhaust raise and the ramp leading towards +70 Portal.

See Figure 16-7 for a representation of the ventilation design for Olympias mine to meet a production rate of 650 ktpa.

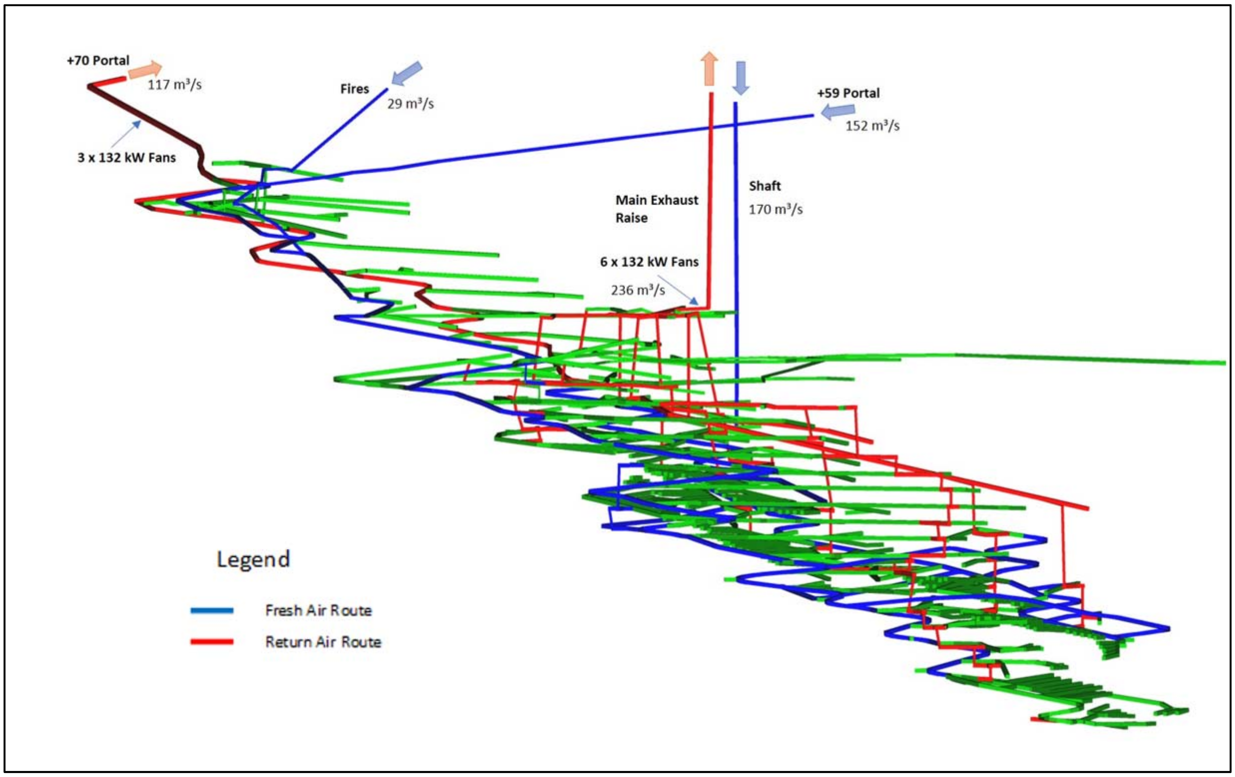


Figure 16-7: LOM Ventilation Strategy Planned

Ventilation for each production level is designed such that fresh air will be sourced from level accesses and FW drives. Delivery of fresh air to work areas will be via auxiliary fan and duct, with contaminated air from development and production level activities returning to an internal return air raise.

Two means of egress are provided for each production area of the mine. The primary means of egress is via the ramp system. Secondary emergency egress is provided in internal raises by means of installed ladderways, and also via crossover drifts.

16.6.2 Mine Airflow

In terms of the quantity of air to be supplied for the mine, Part 3, Article 75 of the Greek Mining Regulations states that, "The volume of clean air at all the work places must be at least 5.66 m^3 for every first minute and worker and 2.3 m^3 for every first minute and internal combustion engine power (0.051 $m^3/s/kW$), regardless the difference in operating time of machinery". The diesel engine ratings are based on technical specifications for major equipment.

In order to meet this requirement, it will be necessary for Olympias to move away from the use of high horsepower four-wheel drive vehicles and towards lower horsepower tractors for personnel transport. Furthermore, Olympias is recommended to consider the introduction of all-electric equipment, particulamasly jumbos, which are already tried and proven in other mines.

16.6.3 Emergency Preparedness

In development of the ventilation strategy for Olympias, consideration has been given to the potential for mine emergencies. As such, the following criteria have been established:

- In general, ramps will be in fresh air once developed.
- On almost all levels, escape can be either to a ramp or to the escape ladderway in the internal raises.
- In each ramp, escape may either be up the ramp or down the ramp to a safe area.
- Portable refuge chambers are installed for flexibility of location at the most appropriate points in the mine.
- Whilst the primary means of communication will be by radio, a stench system will be in place for introduction of ethyl mercaptan into the portal air stream in the event of fire.

16.7 DRILL AND BLAST

Olympias mine currently uses packaged explosives products for all blasting practices in the underground mine. There is no existing magazine on the property and explosives are delivered to site daily by the supplier. The following sections discuss the opportunity to implement best international practices for: explosives selection, drill and blast designs, and logistics and explosives management at Olympias mine.

16.7.1 Explosives Selection

This section examines the explosives to be utilized for stoping and lateral development at Olympias.

Emulsion is a commonly utilized bulk explosive product that can be used for lateral development and can also be pumped both up and down into stoping blastholes. Emulsion is water resistant, produces low levels of ammonium nitrate in leachate, can be handled in bulk deliveries to reduce costs, has reliable and consistent performance and, depending on the product, can be left in the ground for up to four weeks before firing. Bulk emulsion is proposed as the primary explosive product to be used for both lateral development and stoping (DAF and TLHOS) at Olympias.

ANFO is the most basic bulk explosive product. It is widely available and can be poured or blown into development or DAF blastholes, or used for blind uphole TLHOS. The equipment required to use ANFO is basic and easily maintained. However, the total ammonium nitrate in the leachate from blasting is higher for ANFO than bulk emulsion. If the total ammonium nitrate found in the leachate from explosives detonation is to be minimized, then ANFO should be replaced with emulsion. Further, ANFO is not water resistant, which makes it less versatile than emulsion when wet conditions are encountered. ANFO is also less desirable for blind upholes as it is limited to a blasthole size of 76 mm because of the explosive tending to fall out under gravity. Conversely, bulk emulsion can effectively be used in dry upholes to a maximum diameter of 102 mm. Emulsion is therefore the preferred bulk explosive.

Packaged explosive products are typically three to five times more expensive on a per unit basis compared to bulk explosive products such as emulsion or ANFO. Although packaged explosives can be used for small narrow blind uphole open stopes (such as in a typical narrow vein mine) they are not commonly used for a bulk mining method such as blind uppers TLHOS, primarily due to inefficiencies in loading large quantities of explosives. Therefore, packaged explosive products have also been eliminated in preference to using bulk emulsion.

It is noted that poor quality / highly fractured ground conditions exist in some DAF areas of the mine and that packaged explosives have been effectively used for blasting in these zones. The switching from primary explosives to bulk emulsion as the main blasting product does not preclude the possibility of maintaining a small inventory of packaged explosive products for use when fractured ground conditions are encountered.

16.7.2 Detonators

Non-electric detonators are the most basic detonation system and are suggested for lateral development and stoping. They are widely available, relatively simple to use, and are less expensive than electronic detonators.

Another viable blast initiation system is electric detonators (or electronic detonators), which have significant advantages over non-electric detonators; these include:

- Electric detonators are a single product (where the delay is programmable). This reduces the required site stock and increases stock turnover.
- The detonator delay is programmable. This allows increased blast size, precision blasting and better fragmentation.
- The delay scatter (prevalent in non-electric detonators) is eliminated.
- The complexity of the charge process for operators is reduced.
- Sensitivity of detonators to handling during the charge process is significantly reduced.
- Multiple security features make them virtually immune to interference, unauthorized use, or unplanned detonation.

Two commonly available electric detonator systems include Orica's IKON system and Dyno Nobel's SMARTSHOT system. All electric detonator systems are more complex than non-electric systems and require both operator and engineer training to be used safely and effectively. Training is provided by the system suppliers.

Electric detonators (or electronic detonators) are the highest quality detonating system available but are four to five times more expensive than non-electric systems; they have, therefore, been eliminated as the main detonating system.

16.7.3 TLHOS Drill and Blast Design

Figure 16-8 and Figure 16-9 show, respectively, a typical longitudinal section and cross section of blind uppers TLHOS of 17 m panel height (floor to stope back), fixed width of 10 m, and length of 26 m. TLHOS stopes account for approximately 31% of the planned stope tonnage for the entire mine. The remaining 69% of planned stope tonnage will be from ore development and DAF.

The overall powder factor for the TLHOS drilling stope design shown in Figure 16-8 is estimated at 0.40 kg/t. Depending on the stope shape and drill pattern, the design powder factor ranges from 0.35 to 0.50 kg/t for all types of longhole open stopes.

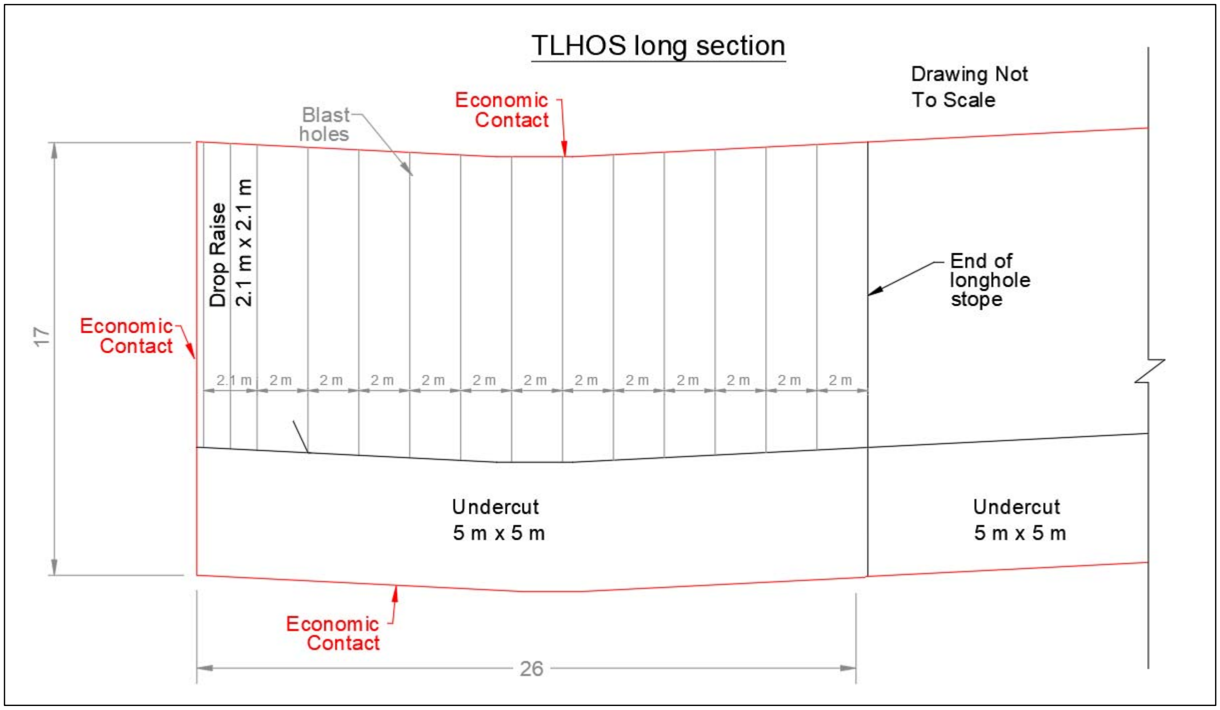


Figure 16-8: TLHOS Long Section

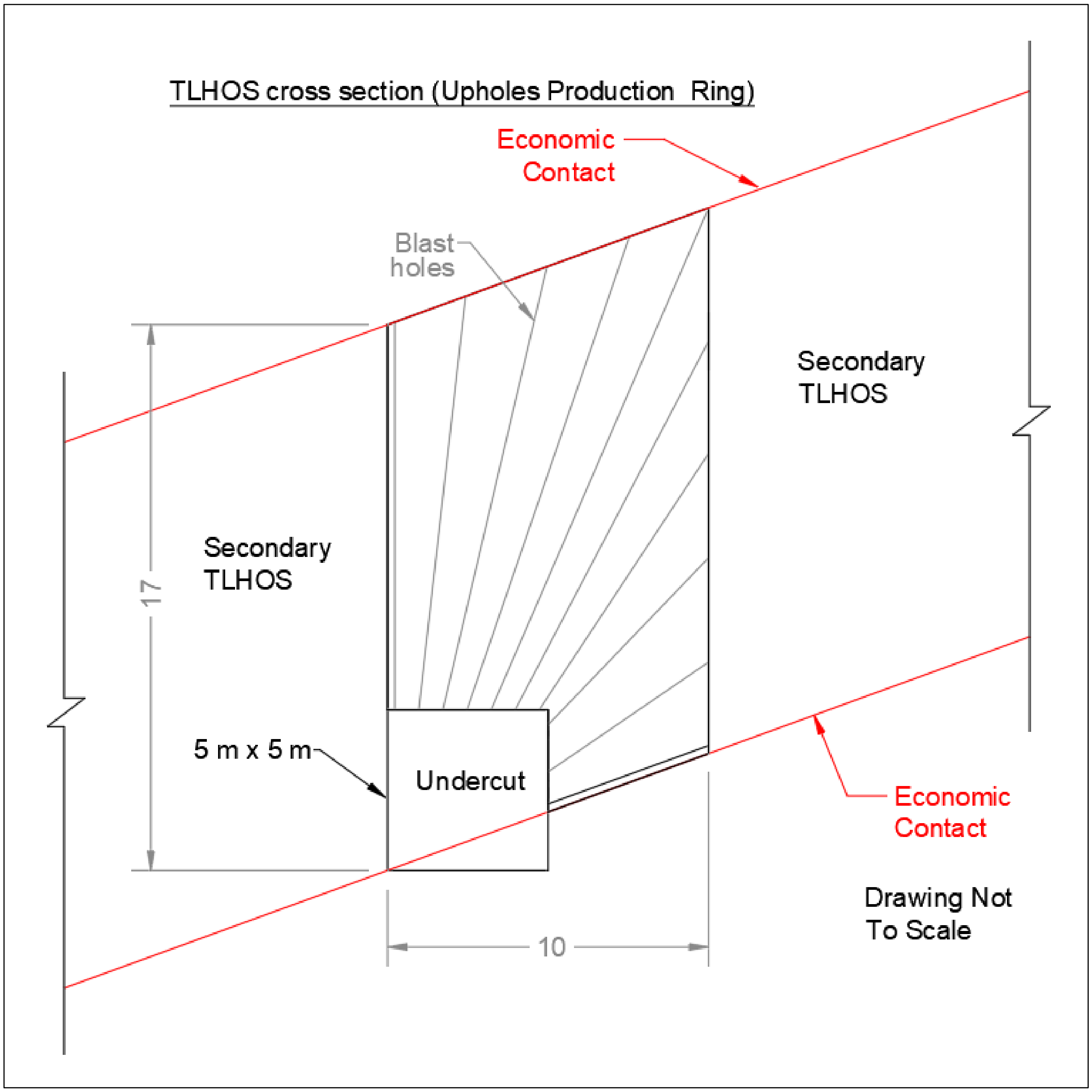


Figure 16-9: TLHOS Cross Section

16.7.4 Lateral Development Blast Design

Figure 16-10 shows the drill layout for a typical lateral development round measuring 5.0 m (H) x 5.0 m (W) x 3.9 m deep.

The overall powder factor for the lateral development design shown is 1.07 kg/t. The powder factor range for all lateral development is from 0.95 to 1.20 kg/t, including DAF stopes and the sill ore drives for the longhole open stopes.

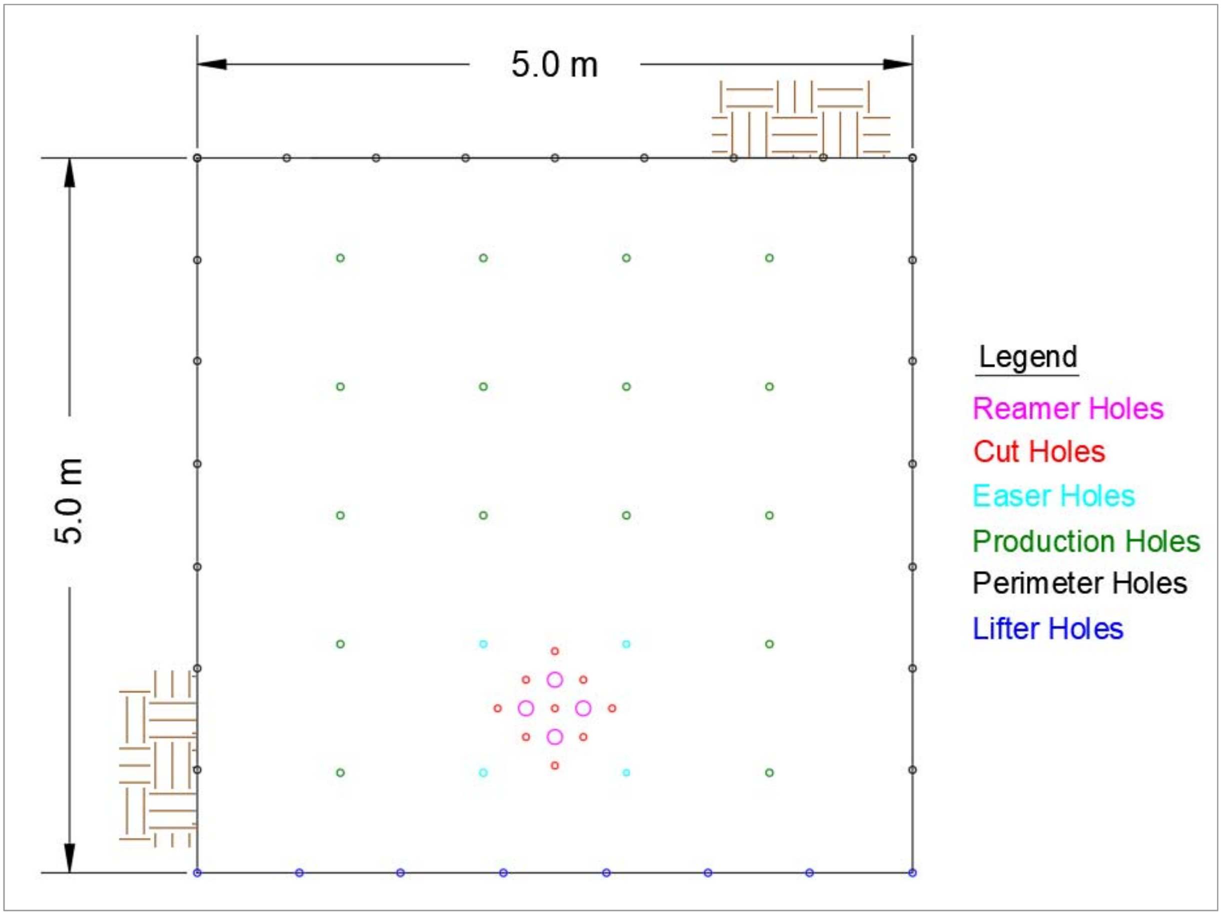


Figure 16-10: Typical Lateral Development Drill Pattern

16.7.5 Explosives Consumption

At steady-state production of 1,800 tpd (THLOS and DAF) and peak development advance of 12.5 m/day, the monthly mine explosives consumption will be approximately 53 tonnes of bulk emulsion, 12,500 detonators and 12,500 boosters. Table 16-4 shows typical powder factors for THLOS and lateral development (including DAF).

Table 16-4: Powder Factors for Longhole Stoping and Lateral Development/DAF

Description	Ore Tonnes (t)	Explosives (kg)	Powder factor (kg/t)
Longitudinal longhole stope	13,227	5,290	0.40
Lateral development (per round)	340	364	1.07

16.8 UNDERGROUND INFRASTRUCTURE

16.8.1 Service Water

Service water for underground mining is supplied from underground inflows captured in clean-water dams. It is distributed underground via DN100 HDPE lines with pressure regulators installed as required to maintain a pressure below 5.5 bar.

16.8.2 Stockpiles

Ore from the underground mine is transported by truck to the surface crushing plant via the haulage portal, where it is placed into one of four ore sorting bins, depending on its gold grade and point of origin (east versus west ore zones).

16.8.3 Underground Workshop

An area for a new underground workshop has been excavated at the -248 Level. Once equipped, this workshop will be used to maintain any equipment that is captive or rarely leaves the mine. The new shop will be equipped with three heavy equipment maintenance bays as well as a light vehicle service bay, areas for parts storage, office space, hose fitting, machine shop, wash bay, lube bay, tire repair station, first aid station, and restrooms.

Ventilation will be flow-through with a vent raise at the back corner of the shop. As mining progresses, a second workshop or satellite shop or service bay may be required.

16.8.4 Explosives / Cap Magazines

There are currently no magazines at the mine; explosives are delivered to site daily. Capital has been budgeted to develop an underground magazine in 2020.

16.8.5 Fuel Storage

There is currently no underground fuel storage at the mine. Funds have been budgeted to add an underground fuel bay once the new workshop facility is completed in 2020.

16.8.6 Compressed Air

Compressed air demand in the mine is typical for a mine this size. Blasthole loading equipment and shotcrete sprayers have on board compressors; all other mobile and fixed equipment is dependent on the compressed air reticulation system.

Currently an Atlas Copco GA 75 air compressor is set up at the +59 adit. This compressor has a capacity of 12.6 m^3/min at 10 bar. For the production increase to 650 ktpa, this will be replaced by an Atlas Copco GA 110 air compressor at the -254 level. The new system will have a capacity of 23.5 m^3/min max at 14 bar; however, mine pressure will be maintained below 8 bar. The compressed air is distributed throughout the mine via a network of DN63 HDPE pipes.

A smaller diesel compressor is also available for emergency situations or power outages.

16.8.7 Electrical Power

There are five underground 20 kV feeds for power underground, three main feeds and two feeds for the pump stations. Of the three main feeds, one runs down the East ramp and two run down the West ramp. All of these feeds are interconnected underground such that the loss of a single cable can be bypassed via a second route. The other two feeds are from the surface pump station to the -95 and -254 pump stations. There is a tie loop between the pump stations.

There is a total of ten main substations located at +46, -85, -173, and -284 in the East ramp and -100, -194, -248, -235, -254 and -260 in the West ramp. The West ramp interconnect is located between the -100 and -194 substations. The west to east interconnect is located between the -284 and -235 substations.

In addition to the main substations, portable 630 kVA substations with 20 kV to 660 V transformers are located where required for mining. All underground power for mobile equipment, fans and pumps is 660 V.

16.8.8 Mine Dewatering

Water inflow has been projected to increase to a maximum of 650 m^3/h. Greek regulations require that the pumping capacity must be 2x maximum anticipated flow + standby of rated maximum flow. This means a capacity of 1,300 m^3/h and standby pumping capability of 650 m^3/h.

The mine dewatering system is rated to meet this requirement and will be expanded to accommodate increased pumping at depth.

There are currently two pumping systems to dewater the mine. The first system is located in the upper East ramp starting at -254 masl and consists of a series of staged pump stations nominally spaced at 30 m vertical intervals. These stations consist of two submersible 37 kW pumps installed in parallel; they discharge into a 100 mm diameter pipe at the lower levels, increasing to 125 mm diameter pipe at higher elevations. The second system consists of two staged systems. These staged systems are located in the West ramp and in the lower East ramp. These systems currently pump water to the old shaft bottom station, where it is stage-pumped to an in-shaft pumping station and finally to surface.

A new pumping station has been excavated and pumps are being installed at the -286 Level. Once complete, it will replace the second system and all water from both the East and West ramps will report to the new pump station, where it will then be pumped via two DN300 steel lines encased in a drill hole (bypassing the existing in-shaft pump station) to the water treatment plant.

In the future, another pump station may be installed similar to the proposed main sump layout at a location near the bottom of the ramps. Any water from below this location would be pumped up the ramp via submersible pumps to the station.

16.9 MINING SCHEDULE

16.9.1 Stope Cycle Time and Mine Production Rate

Stope cycle times were estimated from an analysis of the various stoping activities. The resulting production rates, recognizing the projected impact of the improvement initiatives, are provided in Table 16-5.

Table 16-5: Olympias Mine, Estimated Stope Productivities

Drift and Fill	Unit	2019	2020	2021	2022
Number of hours per day	hrs	12	13.5	15	16.5
Tonnes / day / heading (DAF)	tpd	109	117	124	140
Time when two faces / heading are available	%	0	25	50	75
Tonnes / day / heading (DAF)	tpd	109	146	186	246
Transverse Llonghole (TLHOS)					
All-inclusive rate	tpd	185	191	197	202
Blast / muck / misc.	tpd	663	693	721	749
Primary / secondary sequence	tpd	317	336	353	369

The Olympias mine is projected to increase its annual production rate from the current 390,000 tonnes in 2019 up to 650,000 tonnes by 2024. To facilitate this production ramp-up, the mine has budgeted for increased mobile equipment, increased power and an increase in manpower.

The operational improvement plan includes the following elements:

- Increased equipment availability
- Upskilling and multiskilling the workforce
- Increase in manpower utilization
- Maximizing the use of paste backfill
- Infrastructural projects to increase underground mining efficiency:
    - Underground workshop
    - Slick line to deliver shotcrete
    - Fuel line to underground
    - Underground slimes filtration system
    - Introduction of remote operations and central control system

These operational improvements will increase the number of hours per day worked at a production face (face time). There are three eight-hour shifts per day and the total face time for all three shifts is indicated as 12 hours per day in 2019, or four hours of face time per shift. Through improved personnel transport, increase in available active headings and time management improvements, it is anticipated that the face time will increase to 5.5 hours per shift or 16.5 hours per day by the year

2022 (Table 16-5). This increase in face time will result in higher stope production rates for both DAF and TLHOS, as indicated in Table 16-5.

A further key initiative to assist the annual production ramp-up is the optimization of DAF mining. This will be accomplished by initiatives that include the introduction of multi-pass mining as shown in Table 16-4 and Table 16-5. The result will be an increase in the number of concurrently available faces in a DAF stope. The increase in the number of stopes with multiple faces will increase the production rate for individual DAF stopes, in addition to facilitating increased miner face time.

The plan for approximately one-third of total mine production to be provided by the more efficient bulk-mining THLOS system is also a key element of the planned production increase to 650 ktpa.

16.9.2 Development Schedule

The projected LOM development schedule is shown in Table 16-6.

Table 16-6: Projected Development Schedule over the LOM

Description		Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Waste development	km	4.20	0.36	0.38	0.42	0.16	0.21	0.16	0.26	0.09	0.28	0.17	0.18	0.09	0.11	0.12	0.12	0.12	0.09	0.08	0.21	0.17	0.17
Drift and fill access	km	6.14	1.00	1.05	0.78	0.31	0.37	0.80	0.52	0.42	0.34	0.02	0.03	-	0.04	0.05	0.03	0.03	0.03	-	-	-	-
West ramp	km	2.83	0.25	0.19	0.24	0.17	0.15	0.22	0.23	0.27	0.69	0.32	-	-	-	-	-	-	-	0.01	-	-	-
Longhole footwall drive	km	1.64	0.05	-	0.22	0.25	0.23	0.03	0.13	0.42	0.07	0.09	-	-	-	0.04	0.03	-	-	0.04	0.03	-	-
Sump	km	0.57	0.08	0.09	0.06	0.06	0.06	0.07	0.04	0.04	0.02	0.03	-	-	-	-	0.01	-	-	-	-	-	-
Connecting drive	km	2.77	0.23	0.27	0.11	0.46	0.21	0.42	0.36	0.48	0.06	-	-	-	0.07	0.05	0.01	-	-	-	-	-	-
Drift and fill footwall drive	km	4.25	0.63	0.83	0.43	0.68	0.46	0.42	0.27	0.02	0.22	-	-	-	-	-	0.06	-	-	-	-	-	-
Miscellaneous	km	0.50	-	-	0.50	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Longhole access	km	5.25	0.16	-	0.49	0.57	0.90	0.25	0.34	1.14	0.29	0.25	0.01	0.12	0.08	0.10	0.06	0.00	0.01	0.21	0.26	-	-
Ramp access	km	1.50	0.29	0.20	0.14	0.10	0.18	0.10	0.12	0.11	0.12	0.08	-	-	-	-	-	-	0.00	0.01	0.00	-	-
Remnant ramp	km	0.32	0.13	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
North ramp (not remnants)	km	0.20	0.08	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
East Ramp	km	2.43	0.23	0.24	0.29	0.25	0.16	0.42	0.55	0.17	-	-	-	-	-	-	-	-	-	-	-	-	-
Return air drive	km	2.81	0.30	0.31	0.30	0.08	0.25	0.36	0.35	0.36	0.25	-	-	-	-	-	-	-	-	-	-	-	-
Longhole ore drive	km	9.60	0.09	0.05	0.44	1.26	0.89	0.58	0.60	0.62	0.57	0.43	0.13	0.53	0.81	0.93	0.27	0.06	0.14	0.87	0.33	-	-
Passing bay	km	0.72	0.08	0.09	0.06	0.08	0.07	0.06	0.06	0.03	0.12	0.03	-	-	-	-	-	-	-	-	-	-	-
Floor bench drives	km	1.31	0.02	-	0.03	-	0.05	0.01	0.15	0.02	0.01	0.03	0.02	0.07	0.28	0.13	-	0.02	0.08	0.32	0.08	-	-
Recovery drives	km	0.46	-	0.03	-	0.06	-	-	-	0.03	-	0.01	0.12	-	-	0.02	-	-	-	0.02	-	0.11	0.05
Refuge / lunchroom	km	0.06	-	-	-	-	-	0.02	0.03	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-
Remuck	km	1.05	0.12	0.11	0.09	0.13	0.11	0.06	0.06	0.12	0.09	0.06	-	-	0.02	-	0.02	-	-	-	-	-	-
Small level sumps	km	0.07	0.01	0.01	0.00	-	-	-	0.02	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-
Electric Bay	km	0.57	0.07	0.08	0.04	0.05	0.05	0.07	0.06	0.06	0.02	0.03	-	0.00	-	-	0.01	-	-	-	-	-	-
Service drift (=Exploration drift)	km	1.07	0.20	0.67	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Garage / shops	km	0.53	-	-	-	-	-	0.39	0.14	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Paste fill drift	km	0.31	0.03	0.16	-	-	-	0.12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Powder magazine	km	0.04	-	0.00	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fuse magazine	km	0.01	-	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fuel bay	km	0.06	-	0.02	-	-	-	0.01	0.02	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-
Drill drift for flats	km	0.38	0.32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Main sump	km	0.76	-	-	-	-	0.23	0.03	0.23	0.02	0.13	0.12	-	-	-	-	-	-	-	-	-	-	-
Total	km	52.37	4.72	4.78	4.75	4.67	4.58	4.60	4.54	4.44	3.29	1.67	0.50	0.81	1.41	1.42	0.63	0.22	0.36	1.55	0.91	0.28	0.22

16.9.3 Underground Mining Production Plan

Table 16-7 shows the scheduled production tonnes and grade for the Olympias mine life. Recovery and dilution factors have been discussed in Section 15. The total projected LOM is 21 years.

Table 16-7: Olympias LOM Ore Production Schedule

Yrs	Total Tonnes (t)	East Tonnes (t)	West Tonnes (t)	Flats Tonnes (t)	Remnants Tonnes (t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
2020	415,000	77,146	233,606	19,069	85,179	7.22	100.28	3.3	4.0
2021	440,000	114,098	221,126	55,143	49,633	7.72	98.23	3.1	3.5
2022	500,000	100,000	305,888	44,112	50,000	7.65	126.95	4.0	3.9
2023	650,000	129,893	256,993	213,113	50,000	7.80	123.91	4.0	4.3
2024	650,000	129,815	167,110	303,076	50,000	7.31	140.78	4.7	4.9
2025	650,000	130,000	14,378	455,622	50,000	7.70	127.46	4.2	5.1
2026	650,000	129,611	10,223	430,291	79,874	7.35	149.86	4.9	5.3
2027	650,000	129,812	10,286	429,979	79,924	6.49	104.79	3.5	4.8
2028	650,000	129,992	11,608	428,433	79,967	6.77	112.67	3.5	5.1
2029	650,000	118,568	4,696	446,736	80,000	6.63	105.50	3.7	5.2
2030	650,000	100,144	-	469,917	79,939	7.19	125.41	4.3	6.1
2031	650,000	129,737	-	420,513	99,750	7.01	133.24	4.7	6.0
2032	650,000	80,991	-	469,373	99,636	7.29	112.50	3.8	5.4
2033	650,000	98,025	-	451,975	100,000	7.49	113.09	4.0	5.9
2034	650,000	-	-	550,000	100,000	7.26	131.22	4.5	5.5
2035	650,000	-	-	550,000	100,000	7.06	138.45	4.9	6.1
2036	650,000	-	-	500,000	150,000	6.99	126.66	4.6	5.6
2037	650,000	-	-	500,000	150,000	7.13	100.13	3.8	4.9
2038	650,000	-	-	468,563	181,437	7.89	103.14	3.8	5.2
2039	650,000	-	-	650,000	-	4.18	91.33	3.3	5.9
2040	550,000	230,937	-	319,063	-	5.11	103.60	3.6	5.5
Total	**12,925,000**	**1,844,094**	**1,284,779**	**8,207,886**	**1,715,338**	**7.02**	**119**	**4.1**	**5.3**

Note: Totals may not compute exactly due to rounding, 2019 not shown, but contained 2 months of reserve production

Figure 16-11 shows the LOM production profile of ore tonnes from each zone.

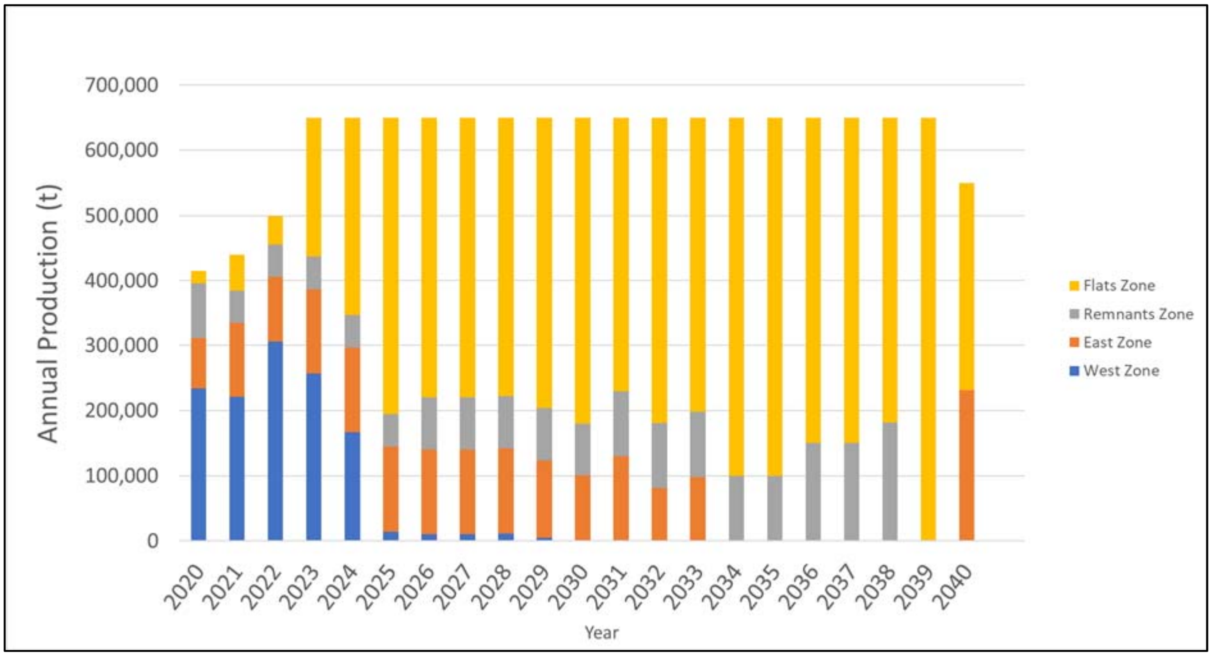


Figure 16-11: LOM Production Profile

SECTION • 17 RECOVERY METHODS

17.1 INTRODUCTION

A three-phase plan has been adopted for Olympias Project.

Phase I – A new bulk flotation plant was built and commissioned to process the old tailings from 2012 to 2016. A single gold/pyrite/arsenopyrite concentrate was produced.

Phase II – A new selective flotation plant was built and commissioned in May 2017 to process fresh ore from the underground mine at a throughput of around 400 ktpa. Three concentrates are produced, namely, lead/silver concentrate, zinc concentrate, and gold/pyrite/arsenopyrite concentrate. Lead/silver (lead) and gold/pyrite/arsenopyrite (gold) concentrates are sold through Thessaloniki port and zinc concentrate is sold through Stratoni port. A portion of flotation tailing is used for underground backfill. Any surplus of flotation tailings is filtered and then trucked to Kokkinolakkas TMF. The Kokkinolakkas TMF is 8.5 km south-east of Olympias mine.

Phase III – Expansion and upgrade will take place to the underground mine, Olympias process plant and Stratoni port to reach production rate of 650 ktpa of ore. AS indicated previously, a permit is required to increase throughput of the process plant up to 650 ktpa.

17.2 PROCESS PLANT DESIGN CRITERIA

A summary of major process design criteria for Phase II is provided in Table 17-1.

Table 17-1: Major Process Design Criteria for Phase II

Description	Unit	Parameter
1.0 Ore characteristics		
Ore Feed Grade - Gold, Silver	g/t	9.5, 124
Ore Feed Grade - Lead, Zinc, Arsenic, Sulphur	%	4.2, 5.7, 5.0, 17.5
Ore Specific Gravity		3.39
Crushing Work Index	kW.h/t	4.2 - 7.7
Rod Mill Work Index - Range, Design	kW.h/t	(7.6 - 12.1), 11.2
Ball Mill Work Index - Range, Design	kW.h/t	(11.4 - 14.3), 13.8
Abrasion Work Index - Range, Design		(0.244 - 0.323), 0.28
Ore Moisture Content	%	5
2.0 Production schedule		
Annual Throughput	tpa	400,000
Operating Days Per Year		350
Lead Concentrate Grade: Pb	%	63
Lead Concentrate Recoveries: Pb, Ag	%	85, 78
Zinc Concentrate Zn Grade & Recovery	%	52, 91
Gold-Arsenic-Pyrite Conc Gold Recovery	%	86
Gold-Arsenic-Pyrite Conc Sulfur Grade	%	38
3.0 Crushing & fine ore storage		
Crusher Operating Schedule; Shifts/d, h/shift		2, 8
Plant Operating Time	%	75

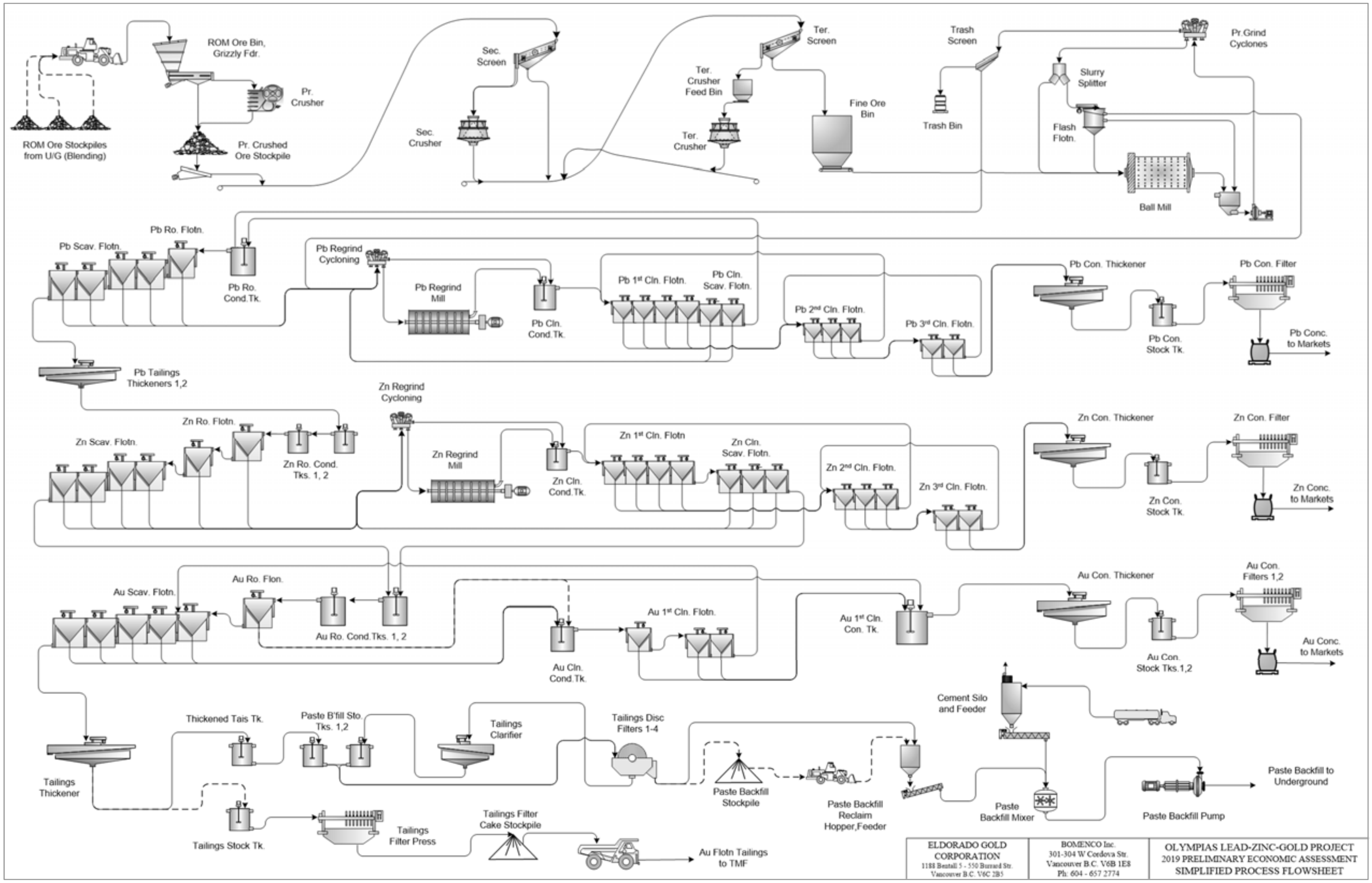
Description	Unit	Parameter
Required Daily Production	t	1,272
Primary Crusher Feeder Type		Vibrating Grizzly
Primary Crusher Type		Single Toggle Jaw
Crusher Product Size, P80	mm	90
Secondary Crusher Feed Screen Aperture – upper / lower deck	mm	50/20
Secondary Crusher Type		Standard Cone
Product Size (P_{80})	mm	28
Screen Aperture	mm	13
Tertiary Crusher Type		Short Head Cone
Product Size (P_{80})	mm	13
Fine Ore Bin	t	1,155
Fine Ore Reclaim System	Type	Belt Feeder
Number		5
Maximum Capacity	t/h	125
4.0 Grinding & classification		
Grinding Circuit Type		Single stage ball milling
Feed Size F_{80}	mm	10
Operating Schedule		
Throughput	t/h	53
Shifts/d, h/shift		3, 8
Plant Availability	%	90
Plant Operating Time	%	86
Primary Grinding Mill Type		Grate Discharge Ball Mill
Installed Motor Power	kW	900 (VS)
Classification Type		Hydrocyclone
Overflow Density	% solid	41
Overflow Product Size (P_{80})	µm	120
Circulating Load	%	250
Flash Flotation Cell Type		Dual Outlet SkimAir SK240
Proportion of Cyclone U/F to Flash	%	95
Residence Time	min	3.2
5.0 Lead flotation, regrind & tails thickening circuits		
Lead Flotation Feed Conditioner time	min	5.5
Lead Rougher Flotation Retention Time	min	4.6
Rougher Concentrate Mass Pull	%	5.7
Lead Scavenger Flotation Retention Time	min	20.4
Scavenger Concentrate Mass Pull	%	6.3
Lead Regrind Classification Type		Hydrocyclone
Overflow Density	% solid	2.5
Overflow Product Size (P_{80})	µm	14
Lead Regrind Mill Type/Size/Model		IsaMill, M500
Installed Motor Power	kW	200
Product Size P_{80}	µm	14
Lead 1st Cleaner Feed Conditioning time	min	12.4
Lead 1st Cleaner Flotation Retention Time	min	8.9
Concentrate Mass Pull	%	22
Lead 1st Cleaner Scav Flotation Retention Time	min	6

Description	Unit	Parameter
Concentrate Mass Pull	%	5
Lead 2nd Cleaner Flotation Retention Time	min	12.2
Concentrate Mass Pull	%	16
Lead 3rd Cleaner Flotation Retention Time	min	16.4
Concentrate Mass Pull	%	6
Lead Tailings Thickeners No, Dia	No., m	2, 8
Underflow Density	% solid	60
6.0 Zinc flotation & regrind circuits		
Zinc Flotation Feed Conditioner No 1	min	13.5
Zinc Flotation Feed Conditioner No 2	min	13.5
Zinc Rougher Flotation Retention Time	min	9.4
Concentrate Mass Pull	%	11
Zinc Scavenger Flotation Retention Time	min	21.7
Concentrate Mass Pull	%	9
Zinc Regrind Classification Type		Hydrocyclone
Underflow Density	% solid	53
Overflow Product Size (P_{80})	µm	15
Zinc Regrind Mill; Type, Size / Model		IsaMill, M500
Installed Motor Power	kW	200
Product Size P_{80}	µm	15
Zinc 1st Cleaner Feed Conditioning time	min	9.3
Zinc 1st Cleaner Flotation Retention Time	min	8.1
Concentrate Mass Pull	%	20
Zinc 1st Cleaner Scav Flotation Retention Time	min	7.6
Concentrate Mass Pull	%	7
Zinc 2nd Cleaner Flotation Retention Time	min	12.9
Concentrate Mass Pull	%	16
Zinc 3rd Cleaner Flotation Retention Time	min	15.4
Concentrate Mass Pull	%	10
7.0 Gold-pyrite flotation		
Au/Pyrite Flotation Feed Conditioner No 1	min	8.6
Au/Pyrite Flotation Feed Conditioner No 2	min	8.6
Au/Pyrite Rougher Flotation Retention Time	min	6.8
Concentrate Mass Pull	%	15
Au/Pyrite Scavenger Flotation Retention Time	min	23.4
Concentrate Mass Pull	%	49
Au/Pyrite Cleaner Feed Conditioning	min	7.6
Au/Pyrite 1st Cleaner Flotation Retention Time	min	17
Concentrate Mass Pull	%	15
8.0 Concentrate handling		
Lead Concentrate		
Lead Concentrate Thickener		8 m dia.
Underflow Density	% solid	65
Lead Concentrate Surge Tank Capacity	h	23
Lead Concentrate Filtration, Filter Type		Pressure - Plate & Frame
Filter Area, Total	m²	26
Residual Cake Moisture	%	9.9

Description	Unit	Parameter
Zinc Concentrate		
Zinc Concentrate Thickener		8 m dia.
Underflow Density	% solid	65
Zinc Concentrate Surge Tank Capacity	h	22
Zinc Concentrate Filtration, Filter Type		Pressure Plate & Frame
Filter Area, Total	m²	49
Residual Cake Moisture	%	11
Au/Pyrite Concentrate		
Au/Pyrite Concentrate Thickener		12 m dia.
Underflow Density	% solid	65
Au/Pyrite Concentrate Surge Tank	h	19.7
Au/Pyrite Concentrate Filtration, Filter Type		2 x Pressure Plate / Frame
Filter Area, Total	m²	620
Residual Cake Moisture	%	13
9.0 Tailings disposal		
Flotation Tailings Thickener Type		10 m dia.
Underflow Density	% solid	65
Tailings Filter Surge Tank Capacity	h	5.3
Tailings Filtration Filter Type		Plate and Frame
Filter Area, Total	m²	620
Residual Cake Moisture	%	13
Tailings Disposal Type (When not backfilling)		Dry Stacked Tailings

17.3 PROCESS FLOWSHEET

A simplified process flowsheet is shown in Figure 17-1.



Source: Eldorado

Figure 17-1: Simplified Process Flowsheet of Olympias Plant

17.4 PROCESS PLANT DESCRIPTION

The current process plant is currently able to treat 400,000 tonnes per annum at 53 t/h. A few key features are as follows.

- Three-stage crushing
- Single-stage ball milling in closed circuit with hydrocyclones
- Nearly all hydrocyclones underflow is fed to flash flotation.
- Lead flotation consists of rougher, scavenger, regrinding, three stages of cleaning
- Zinc flotation consists of rougher, scavenger, regrinding, three stages of cleaning
- Gold-Pyrite flotation utilizes rougher, scavenger, and single stage of cleaning
- Concentrate thickening, filtration, and packaging
- Tailings thickening and filtration
- Tailings paste backfill
- Reagent mixing, storage and distribution
- Water and air services

The following sections describe in detail the above processes.

17.4.1 Crushing, Ore Storage, and Reclaim

A ROM ore stockpile is situated adjacent to the primary crusher feed bin. The ROM stockpile is divided into a few areas according to ore type and grades to facilitate blending. Ore is reclaimed from the ROM stockpile by a front-end loader (FEL).

The crushing plant is operated 350 days per year, 16 hours per day at a crushing rate of 125 t/h.

The crushing product size is 100% passing 13 mm and 80% passing 10 mm.

A 600 mm x 800 mm inclined static grizzly is fitted to the primary crusher feed hopper for protection from oversize rocks. The oversize rocks are periodically broken to less than 600 mm with a mobile rock breaker. The primary crusher feed hopper is equipped with dust suppression spray.

The primary crusher is fed by a vibrating grizzly feeder that scalps the rocks greater than 75 mm ahead of crushing. The undersize material is directed onto the crusher discharge conveyor. Grizzly feeder oversize feeds a jaw crusher (Sandvik CJ411) at closed side setting (CSS) of 75 mm.

The jaw crusher product (100% passing 180 mm and 80% passing 90 mm) reports to the crusher discharge conveyor and is delivered, along with the fines from the vibrating grizzly feeder, to the primary crushed coarse ore stockpile feed conveyor. The coarse ore stockpile feed conveyor discharges onto the coarse ore stockpile with a live capacity of 79 tonnes. A cross-belt tramp metal magnet is positioned over the coarse ore stockpile feed conveyor. Tramp metal is discharged via a chute into a tramp metal bin located at ground level.

Primary crushed ore is withdrawn from the coarse ore stockpile by a vibrating feeder and transported to the secondary crusher feed screen conveyor. A metal detector is installed on the secondary crusher feed screen conveyor to ensure all tramp metal is removed.

The primary crushed ore then reports to a double-deck scalping screen with 50 mm aperture for the top deck and 20 mm aperture for the bottom deck. The oversize material from both decks is directed to the secondary crusher (Sandvik CH430 with CSS at 28 mm) whilst the screen undersize and the secondary crusher product are deposited onto the secondary crusher discharge conveyor, which in turn discharges onto the tertiary crusher feed screen conveyor.

Ore on the secondary crusher discharge conveyor transfers onto the tertiary crusher feed screen conveyor, and then to the tertiary product screen. A metal detector is installed on the tertiary feed screen conveyor to ensure all tramp metal is removed for protection of the tertiary crusher. When the presence of metal is detected by the metal detector, the crushing circuit automatically stops to allow the operator to pick the metal from the belt at a pre-determined location. The product screen is a single deck vibrating screen with an aperture of 13 mm. The screen oversize returns to the tertiary crusher.

The tertiary crusher feed hopper is fitted with a belt feeder. The tertiary crusher (Sandvik CH430 at CSS of 13 mm) is a short head cone crusher that operates in closed circuit with the product screen. The crusher discharges directly onto the tertiary crusher discharge conveyor, which in turn discharges onto the secondary crusher discharge conveyor (thus operating in closed circuit with the product screen).

The final crusher product after screening has a P_{80} of 10 mm and is deposited onto the tertiary screen undersize conveyor, which in turn discharges onto the fine ore bin feed conveyor. The fine ore bin feed conveyor transfers the final crusher product to the fine ore bin (FOB). The crushing rate is monitored by a weightometer located on the fine ore bin feed conveyor. The FOB has a live capacity of 1,155 tonnes to provide 21.8 hours of mill feed.

Mill feed is withdrawn from the FOB via five variable speed belt feeders. A weightometer indicates the instantaneous and totalized mill feed tonnage for control of belt feeder speed. Mill feed from the FOB is transported to the grinding circuit by the ball mill feed conveyor.

17.4.2 Grinding, Classification, and Flash Flotation

The grinding circuit consists of a single-stage overflow ball mill that operates in closed circuit with hydro-cyclones to produce a ground product slurry with P_{80} of 120 µm and with a flash flotation cell for recovery of galena from the cyclone underflow.

Mill feed enters the grinding circuit through the mill feed chute, where process water is added. The ball mill is 3.65 m diameter by 4.00 m long effective grinding length and powered by a 900 kW variable speed drive. Normal operation requires 700 kW power draw. The ball mill is fitted with a trommel screen. Trommel screen undersize slurry reports to the cyclone feed hopper.

Reagents are added to the cyclone feed hopper for depression of sphalerite, arsenopyrite and pyrite.

Cyclone overflow flows by gravity to the trash screen feed box. A single horizontal vibrating trash screen (0.8 mm x 12 mm slotted aperture) removes trash from the flotation feed. Oversize trash gravitates directly to a trash bin. Screen undersize gravitates to the lead rougher flotation conditioning tank.

Cyclone underflow gravitates to the underflow distribution box where the flow is split. Up to 100% of cyclone underflow stream can be directed to the flash flotation circuit to recover fast-floating galena. Dilution water and flotation reagents are added. Concentrate flows by gravity to the flash flotation concentrate transfer pump before being pumped to the inline stream analyzer (ISA) for analysis and then regrinding. The flash flotation coarse tailings flows by gravity to the ball mill for further grinding. The fine tailings from the top outlet flows to the cyclone feed hopper or to a surge tank and then to conditioning tank ahead of lead rougher flotation

17.4.3 Lead Flotation and Regrind

The lead flotation circuit consists of rougher, scavenger, and three stages of cleaner. A regrind circuit provides size reduction of the rougher, scavenger, flash flotation and cleaner scavenger concentrates to improve mineral liberation and clean particle surfaces.

Reagents are added to the lead flotation conditioning tank, lead rougher and first lead scavenger. Slurry flows by gravity from the conditioning tank to the first lead rougher flotation cell. Flotation feed (cyclone overflow) is automatically sampled for analysis by the ISA to provide a flotation feed assay and feed slurry density.

The lead rougher/scavenger flotation circuit consists of one 10 m³ rougher tank cell followed by four 10 m³ capacity scavenger flotation tank cells

The rougher concentrate flows by gravity to the lead rougher concentrate pump and the scavenger concentrate is directed to the lead scavenger concentrate pump. The lead scavenger flotation tail discharges via a dart valve to the lead scavenger tail hopper. The hopper is serviced by duty and standby variable speed pumps that pump lead scavenger flotation tail to the lead tails thickeners. An automatic sampler on the lead scavenger tail provides a sample to the ISA.

The flash flotation, lead rougher, and lead scavenger concentrates are pumped separately to the lead regrind cyclone feed hopper, where they are combined with the lead cleaner scavenger concentrate and reagents for sphalerite, arsenopyrite and pyrite depression. The combined concentrate streams are classified in the regrind cyclone cluster consisting of three 150 mm cyclones (two operating and one standby). The regrind cyclones are fed at a pulp density of approximately 18% solids. The cyclone overflow flows by gravity to the lead cleaner conditioning tank.

The cyclone underflow gravitates to the lead regrind mill feed hopper and is diluted to a density of 50% solids before being pumped to the lead regrind mill (IsaMill M500). Media addition is based on mill motor power draw. The regrind mill is configured in open circuit. Pulp discharge from the lead regrind mill joins the cyclone overflow in the lead cleaner conditioning tank. The regrind mill circuit can also be configured to enable a portion of the regrind mill discharge to be recycled to the feed hopper to assist in maintaining a constant volumetric flow rate. The lead cleaner conditioning tank has an effective volume of 12.6 m³. Reagents are added to the conditioning tank and various points in the lead cleaner circuit.

Four 4.25 m³ Outotec OK-3HG-U flotation cells operate as a single bank for the 1st cleaner. The lead 1st cleaner tail flows by gravity via a dart valve to the lead cleaner scavenger cells (two 4.25 m³).

Concentrate from the lead cleaner scavenger cells is pumped to the lead regrind mill cyclone feed hopper. The lead cleaner scavenger tail discharges via a dart valve to the lead cleaner scavenger tail hopper, and then pumped to the lead flotation tailing thickeners.

Concentrate from the lead 1st cleaner cells is pumped to the lead 2nd cleaner cells. Three 4.25 m^3 capacity Outotec OK-3HG-U flotation cells are arranged in a single bank. Concentrate from the lead 2nd cleaner cells is pumped to the lead 3rd cleaner cell feed box. Two 4.25 m^3 capacity Outotec OK-3HG-U flotation cells operate as a single bank. The lead 3rd cleaner tail is combined with the lead 1st cleaner concentrate to the lead 2nd cleaner circuit. The lead 3rd cleaner concentrate is final concentrate and reports to the lead concentrate thickener.

An ISA is installed in the plant to provide analysis of five key process streams in the plant. These are:

- Flash flotation concentrate
- Cyclone overflow
- Lead scavenger tail
- Lead cleaner scavenger tail
- Lead final concentrate

Reagent addition rates are controlled including frother (MIBC), collector (Aerophine 3418A) and depressants (sodium cyanide, zinc sulphate and lime)

17.4.4 Lead Flotation Tails Thickening

The lead scavenger tails and lead cleaner scavenger tails are pumped to a distribution box and then into two 8 m diameter tailings thickeners. The lead tails are thickened to 60% solids, then pumped to zinc flotation conditioning tank 1. Flocculant is added to the lead tailings thickener feed stream to assist in the thickening process. Lead tailings thickener overflow reports to the lead thickener overflow tank and is reused as process water in the grinding and lead flotation circuits.

17.4.5 Zinc Flotation and Regrind

The zinc flotation circuit consists of rougher, scavenger and three stages of cleaner. A regrind circuit provides particle size reduction of the rougher, scavenger and cleaner scavenger concentrates to improve mineral liberation and clean particle surfaces.

The thickened lead tailings are diluted to 35% solids by the addition of process water in the first conditioning tank.

The two agitated zinc flotation conditioning tanks have a total effective volume of 57 m^3. Reagents are added to the first and second conditioning tanks, first rougher and scavenger cell feed boxes.

The zinc rougher/scavenger flotation circuit consists of two 10 m^3 rougher tank cells followed by four 10 m^3 scavenger tank cells.

The rougher concentrate flows by gravity to the zinc rougher concentrate pump and the scavenger concentrate is directed to the zinc scavenger concentrate pump. The zinc scavenger flotation tail discharges via a dart valve to the zinc scavenger tail hopper and is pumped to the first gold-pyrite flotation feed conditioning tank. A sampler on the zinc scavenger tail stream provides a sample to the ISA.

The zinc rougher and scavenger concentrates are pumped separately to the zinc regrind cyclone feed hopper, where they are combined with the zinc cleaner scavenger concentrate and reagents for sphalerite activation and depression of arsenopyrite and pyrite. The combined concentrate streams are classified in a cyclone cluster consisting of four 150 mm cyclones (three operating and one standby). The regrind cyclones are fed at a pulp density of approximately 20% solids. The cyclone overflow flows by gravity to the zinc cleaner conditioning tank.

The cyclone underflow gravitates to the zinc regrind mill feed hopper and is diluted to a density of 50% solids before being reground. The zinc regrind mill is an IsaMill M500 fitted with a 200 kW motor. Media addition is based on mill motor power draw. The regrind mill is configured in open circuit. Pulp discharge from the regrind mill joins the overflow from the regrind cyclone overflow in the zinc cleaner conditioning tank. The regrind mill circuit can also be configured to enable a portion of the regrind mill discharge to be recycled to the feed hopper to assist in maintaining a constant volumetric flow rate.

The zinc cleaner conditioning tank has an effective volume of 10 m^3. Reagents are to the conditioning tank and various points in the zinc cleaner circuit.

Four 4.25 m^3 Outotec OK-3HG-U flotation cells operate as a single bank in the 1st cleaner

The 1st cleaner tail gravitates via a dart valve to the zinc cleaner scavenger cells (three 4.25 m^3 Outotec OK-3HG-U flotation cells). Concentrate from the cleaner scavenger cells is pumped back to the zinc regrind cyclone feed hopper. The zinc cleaner scavenger tail discharges via a dart valve to the zinc cleaner scavenger tail hopper and then pumped to the first gold-pyrite flotation conditioning tank. A sampler on the tail provides a sample to the ISA.

Concentrate from the 1st cleaner cells is pumped to the 2nd bank of cleaner cells (three 4.25 m^3 Outotec OK-3HG-U cells).

Concentrate from the 2nd cleaner cells is pumped to the 3rd cleaner cells (two 4.25 m^3 Outotec OK-3HG-U cells). The 3rd cleaner tail stream is combined with the 1st cleaner concentrate as the 2nd cleaner feed. The zinc 3rd cleaner concentrate as final concentrate reports to the zinc concentrate thickener feed hopper.

The ISA provides analysis of three key process streams in the plant.

- Zinc scavenger tail
- Zinc cleaner scavenger tail
- Zinc final concentrate

Reagent addition rates are controlled, including frother (MIBC), collector (sodium isopropyl xanthate), activator (copper sulphate), and depressant (lime).

17.4.6 Gold-Arsenopyrite-Pyrite Flotation

The gold-arsenopyrite-pyrite flotation circuit consists of rougher, scavenger, and a single stage of cleaner. The cleaner tail is recirculated to the scavenger feed. The scavenger tail reports as the final tail, and the cleaner concentrate joins the rougher concentrate as final concentrate.

The two agitated gold flotation conditioning tanks have a total effective volume of 56 m^3. Slurry gravitates from the first conditioning tank to the second conditioning tank. Reagents are added to the first and second conditioning tanks and to the first gold rougher and scavenger cell feed boxes.

The gold rougher/scavenger flotation circuit consists of one 20 m^3 rougher tank cell followed by five 20 m^3 capacity scavenger tank cells arranged to produce separate concentrates.

The first rougher and the first and second gold scavenger cells are each equipped with a feed box. The gold flotation tail discharges via a dart valve to the gold flotation tail hopper, and then pumped to the tailings thickener. A sampler on the discharge side of the gold flotation tail pump provides a sample to the ISA.

Concentrate from the gold rougher flotation cells is pumped directly to the final gold concentrate thickener or alternatively to the gold cleaner cells. The concentrate produced from the gold scavenger flotation cells is pumped to the gold cleaner conditioning tank.

The gold cleaner flotation conditioning tank has an effective volume of 12.6 m^3. Reagents are added to the conditioning tank and to the first gold cleaner cell feed box.

The 1st cleaner flotation circuit consists of a single 10 m^3 tank cell followed by a bank of two 10 m^3 tank flotation cells.

The 1st cleaner tail discharges via a dart valve to the gold 1st cleaner flotation tail hopper and then pumped to the scavenger bank, where it is combined with the rougher flotation tail.

The 1st cleaner concentrate gravitates to the final gold concentrate hopper, where it combines with the rougher concentrate, and then pumped to the gold concentrate thickener. A sampler on the discharge side of the gold concentrate transfer pump provides a sample to the ISA.

The ISA provides analysis of two key process streams in the plant.

- Gold scavenger tail
- Gold final concentrate

Reagent addition rates are controlled, including frother (MIBC), collector (sodium isopropyl xanthate) and activator (sulphuric acid)

17.4.7 Lead Concentrate Thickening and Filtration

Concentrate from the lead flotation circuit is pumped to the lead concentrate thickener feed hopper, and then to a de-aerator, from where it flows by gravity to the feedwell of an 8 m diameter concentrate thickener. Overflow from the de-aerator gravitates back to the lead concentrate thickener feed hopper with the level allowed to vary between a high and a low limit to maintain the de-aerator operating pressure set-point.

An auto-dilution system on the feedwell of the thickener increases the settling rate. Flocculant is added to the feedwell. The design thickener underflow density is 65% solids.

A bed level measuring device is installed to monitor the thickener bed depth. The addition rate of flocculant is controlled according to the bed depth. The thickener is equipped with two variable speed peristaltic underflow pumps arranged in a duty / standby configuration, which are controlled to maintain thickener bed pressure. A nuclear density gauge measures the density of the thickened slurry to the lead concentrate filter storage tank.

Thickener torque is automatically maintained in pre-set ranges via the thickener local control panel, which raises and lowers the thickener rakes according to torque readings. Torque readings and rake status (i.e. running / stopped / fault) are displayed on the control system. Lead concentrate thickener overflow gravitates to the lead thickener overflow tank. Combined lead concentrate thickener overflow and lead tailings thickener overflow is reused in the grinding and lead flotation circuits.

The lead concentrate filtration section consists of an agitated lead concentrate filter feed storage tank, dual filter feed pumps and a horizontal plate/frame pressure filter. The lead concentrate tank has a working volume of 49 m^3, equivalent to 23 hours of concentrate production

The thickened concentrate slurry is pumped from the lead concentrate tank to a pressure filter (26 m^2 filtration area) for dewatering. The pressure filter produces a filter cake containing 9 – 10% moisture.

The filter cake discharges onto the lead filter cake conveyor, which in turn transfers it to a bagging station. Bagged lead concentrate is weighed and loaded onto haulage trucks.

A filtrate air separator is used to remove air from the filtrate before it reports to the filtrate hopper. During normal operation, the filtrate is collected in the lead filter filtrate tank. Filtrate is then pumped to the concentrate thickener to remove fine solids prior to re-use in the plant. The filter has a local control panel that controls the operation of the filter and associated valves and pumps, including the feed pumps and the cloth wash water pump.

17.4.8 Zinc Concentrate Thickening and Filtration

Concentrate from the zinc flotation circuit is pumped to the zinc concentrate thickener feed hopper and then to a de-aerator, from where it gravitates to the feedwell of an 8 m diameter concentrate thickener. The de-aerator feed pump speed is controlled automatically to maintain a set pressure in the de-aerator. Overflow from the de-aerator gravitates back to the zinc concentrate thickener feed hopper, with the level allowed to vary between a high and a low limit while maintaining the de-aerator operating pressure set-point.

An auto-dilution system on the feedwell of the thickener increases the settling rate, and flocculant is added to the feedwell. The design thickener underflow density is 65% solids.

A bed level measuring device is installed to monitor the thickener bed depth. The addition rate of flocculant is controlled according to the bed depth. The thickener is equipped with two variable speed peristaltic underflow pumps arranged in a duty / standby configuration, which are controlled to maintain thickener bed pressure. A nuclear density gauge measures the density of the thickened slurry to the zinc concentrate filter storage tank.

Thickener torque is automatically maintained in pre-set ranges via the thickener local control panel, which raises and lowers the thickener rakes according to torque readings. Torque readings and rake status (i.e. running / stopped / fault) are displayed on the control system. Zinc concentrate thickener overflow gravitates to the water collection sump before being pumped directly to the process water tank.

The zinc concentrate filtration section consists of a zinc concentrate filter storage tank, dual filter feed pumps and a horizontal plate/frame pressure filter. The zinc concentrate tank has a working volume of 91 m^3, equivalent to 22 hours of concentrate production. The tank has an ultrasonic level device to monitor levels.

The thickened concentrate slurry is pumped from the zinc concentrate tank to a pressure filter (48 m^2 filtration area) for dewatering. The pressure filter produces a filter cake containing 12% moisture.

The filter cake discharges onto the zinc filter cake conveyor, which in turn transfers the filtered concentrate to a bagging station. Bagged zinc concentrate is weighed and loaded onto haulage trucks.

A filtrate air separator is used to remove air from the filtrate produced during the core blow cycle, and the filtrate is collected in the zinc filter filtrate tank before being pumped to the zinc concentrate thickener.

The filter has a local control panel that controls the operation of the filter and associated valves and pumps, including the feed pumps and the cloth wash water pump.

17.4.9 Gold Concentrate Thickening and Filtration

Concentrate from the gold-pyrite flotation circuit is pumped to the gold concentrate thickener stock hopper and then to a de-aerator, from where it gravitates to the feedwell of a 12 m diameter thickener. The de-aerator feed pump speed is controlled automatically to maintain a set pressure in the de-aerator. Overflow from the de-aerator gravitates back to the gold concentrate thickener feed hopper.

An auto-dilution system on the feedwell of the thickener increases the settling rate, and flocculant is added to the feedwell. The design thickener underflow density is 65% solids.

A bed level measuring device is installed to monitor the thickener bed depth. The addition rate of flocculant is controlled according to the bed depth. The thickener is equipped with two variable speed peristaltic underflow pumps arranged in a duty / standby configuration, which are controlled to maintain thickener bed pressure. A nuclear density gauge measures the density of the thickened slurry.

Thickener torque is automatically maintained in pre-set ranges via the thickener local control panel, which raises and lowers the thickener rakes according to torque readings. Torque readings and rake status (i.e. running / stopped / fault) are displayed on the control system. Gold concentrate thickener overflow flows by gravity to the gold thickener overflow tank before being pumped to the process water tank.

The gold concentrate filtration section consists of two storage tanks, two sets of dual filter feed pumps and two horizontal plate/frame pressure filters. The gold concentrate tanks have a working volume of 119 m^3, equivalent to 19.7 hours of concentrate production.

The thickened concentrate slurry is pumped to two pressure filters in parallel. Total filtration area is 1,240 m^2. The filter cake contains 10% moisture.

The filter cake discharges onto dedicated conveyors, which in turn transfer the filter cake from each filter to a bagging station. Bagged gold concentrate is weighed and loaded onto haulage trucks.

A filtrate air separator is used to remove air from the filtrate before it reports to the filtrate tank. During normal operation, the filtrate from both filters is collected in the gold filter filtrate tank before being pumped to the gold concentrate thickener.

17.4.10 Tailings Thickening and Filtration

Flotation tailings from the gold scavenger flotation are pumped to the 10 m diameter high rate thickener. Flocculant is added to increase the settling rate and underflow density to approximately 65% solids w/w. Tailings thickener overflow gravitates directly to the zinc thickener overflow tank for re-use in the grinding and flotation circuits.

A bed level measuring device is installed to monitor the thickener bed depth. The addition rate of flocculant is controlled according to the bed depth. The thickener is equipped with two variable speed peristaltic underflow pumps arranged in a duty / standby configuration, which are controlled to maintain thickener bed pressure.

Thickener torque is automatically maintained in pre-set ranges via the thickener local control panel, which raises and lowers the thickener rakes according to torque readings. Torque readings and rake status (i.e. running / stopped / fault) are displayed on the control system.

Tailings thickener underflow flows by gravity to the thickened tailings hopper and is then pumped to the tailings filtration or paste backfill plant.

The tailings filtration section consists of an agitated tailings holding tank, filter feed pumps and a horizontal plate/frame pressure filter. The tailings storage tank has a working volume of 137 m^3, equivalent to 5.3 hours of tailings production.

Thickened tailings are pumped from the tailings holding tank to a horizontal plate/frame pressure filter for dewatering. Total filtration area is 620 m^2. The pressure filter produces a filter cake containing 10% moisture. Filtrate gravitates directly to the tails filter filtrate tank before being pumped back to the tailings thickener. The filter cake discharges onto a stockpile, where it is reclaimed by a FEL. The filtered tails is either transported and stored off site or fed to the paste backfill plant.

17.4.11 Tailings Paste Backfill

There are four Eimco 51 m² (2.7 m diameter x 5 disc) filters, three operating and one standby. Three vacuum pumps are provided, two operating, one standby.

In paste backfill mode, thickened flotation tailings are pumped to two 137 m³ storage tanks which provide approximately 11.8 hours of storage at the design processing rate. Slurry is then pumped to three disc filters by dedicated centrifugal pumps. The level in each filter bath is controlled by a pinch valve.

Each vacuum pump is supplied with filtrate receivers. The vacuum pump sealing water system consists of a closed-circuit system with a chiller to remove heat build-up from the vacuum pump operation. Make-up sealing water is supplied from a filtered water system. The disc filters produces a filter cake containing 20% moisture.

Each filter discharges directly onto the paste backfill filter discharge conveyor, which in turn transfers the filtered tailings to the paste backfill cross conveyor. This conveyor is reversible and can discharge either onto a stockpile or the paste backfill mixer feed conveyor. There is a facility to reclaim stacked tails via the paste backfill feeder conveyor, which discharges onto the mixer feed conveyor.

Filtrate flows by gravity directly to the paste backfill clarifier, where it is mixed with flocculant solution to increase clarifier performance. The clarifier produces an overflow, which is pumped to the paste backfill mixer and paste plant clean water tank via a filter. Underflow from the clarifier is pumped back to the paste backfill storage tanks, with a bleed to the clarifier feedwell to provide seed material. Excess filtrate is sand filtered to provide clean water.

The paste backfill mixer feed conveyor transfers the filtered tailings to a paste plant. Paste components are directed to the feed chute of the paste mixer along with cement dosed at a proportion of the measured dry weight of feed. Cement is transported to site via bulk tankers and pneumatically conveyed via a truck-mounted pneumatic conveyor to the storage silo. A 65 tonne capacity storage silo is installed. The silo discharges via a screw feeder to the paste backfill mixer. The measured components are added to a dual-shaft continuous mixer. Mixed paste is pumped to the underground reticulation system.

17.5 REAGENTS MIXING, STORAGE, AND DISTRIBUTION

The following process reagents are necessary to operate the processing facilities:

- Hydrated lime
- Sodium cyanide
- Zinc Sulphate
- Frother: methyl isobutyl carbinol (MIBC)

- Collector: aerophine 3418A.
- Collector: sodium isopropyl xanthate (SIPX)
- Copper sulphate
- Sulphuric acid
- Flocculant
- Caustic soda
- Antiscalant

Packaged reagents are delivered to site and placed in the reagent compound. A forklift is used to transfer the drums or pallets to the preparation area.

17.5.1 Hydrated Lime

Lime slurry is used as the pH modifier in the flotation process. Hydrated lime is delivered to site as a powder in 20 tonne bulk tankers (containers). Hydrated lime is off-loaded from the tanker with a dedicated blower into a 25 tonne capacity silo. Hydrated lime is then automatically mixed with raw water on a batch basis to generate a 20% w/v slurry. The lime slurry is periodically transferred from the mixing tank to the holding tank. Both the mixing and holding tanks are agitated. Centrifugal slurry pumps distribute the slurry throughout the flotation circuits via a ring main. The pH of the circuit is measured in various places and lime dosage adjusted accordingly. Individual addition points are controlled by an automatic valve and a pH probe

Lime slurry is dosed to the following locations:

- Ball mill feed chute
- Lead regrind mill cyclone feed hopper
- Lead 2nd cleaner feed
- Lead 3rd cleaner feed
- Zinc conditioning tank 1
- Zinc regrind mill cyclone feed hopper
- Zinc cleaner scavenger feed, Zinc 1st cleaner feed and Zinc 2nd cleaner feed

17.5.2 Sodium Cyanide

Sodium cyanide is used as a sphalerite and arsenopyrite/pyrite depressant during lead flotation. Sodium cyanide is supplied as solid briquettes in 50 kg drums.

Sodium cyanide mixing is completed manually by the operator. Sodium cyanide is added to the mixing tank using a drum tipping device. Raw water is added into the sodium cyanide mixing tank to dissolve crystalline cyanide and the solution is then transferred to the 16 m³ holding tank. A small amount of caustic soda is also added into the mixing tank to provide protective alkalinity. The hood over the mixing tank is ducted to the ventilation scrubber, which removes, scrubs and vents any gases released during the mixing process. A sodium cyanide ring main system with dedicated dosing pumps enables the required additions to the following locations:

- Ball mill feed

- Lead cleaner conditioning tank
- Lead 2nd and 3rd cleaner feed

17.5.3 Zinc Sulphate

Zinc sulphate is used as a sphalerite depressant during lead flotation. Zinc sulphate is supplied in powder form in bags.

Zinc sulphate mixing is completed manually by the operator. The mixing facility comprises a dust enclosure with bag splitter and an agitated 10 m³ capacity mixing tank. The bags are split, and zinc sulphate is then mixed with raw water to generate a 20% w/v solution. After mixing the zinc sulphate solution is pumped to a header tank for distribution to the lead flotation circuit. Zinc sulphate addition is via dedicated dosing pumps running off a ring main.

The zinc sulphate is dosed to the following locations:

- Primary mill cyclone feed hopper
- Lead regrind mill feed pump box

17.5.4 Frother

The frother used in the flotation process is MIBC. The frother is supplied as a 100% concentrated solution in 1,000 litre bulk boxes or 200 litre drums. Frother is pumped undiluted to a head tank for distribution through the flotation circuits. The frother addition is via dedicated dosing pumps running off a ring main.

The frother is dosed to the following locations:

- Flash flotation feed
- Lead rougher feed
- Lead scavenger feed
- Lead 1st cleaner feed
- Lead cleaner scavenger feed
- Zinc rougher feed
- Zinc scavenger feed
- Zinc 1st cleaner feed
- Zinc 2nd cleaner feed.
- Zinc cleaner scavenger feed
- Gold rougher feed
- Gold scavenger feed
- Gold 1st cleaner feed

17.5.5 Collector (Aerophine 3418A)

Aerophine 3418A is the collector in the lead flotation circuit. It is supplied as a 100% concentrated solution in 1,000 litre bulk boxes or 200 litre drums. The collector is pumped to a head tank for distribution through the lead flotation circuit. The lead collector addition is via dedicated dosing pumps running off a ring main.

The lead collector is dosed to the following locations:

- Flash flotation feed
- Lead conditioning tank
- Lead scavenger feed
- Lead cleaner conditioning tank
- Lead cleaner scavenger feed

17.5.6 Collector (SIPX)

SIPX is the collector in the zinc and gold-pyrite flotation circuits. SIPX is supplied as solid pellets in 120 kg drums.

SIPX mixing is completed manually by the operator. SIPX is added to the mixing tank using a drum tipping device and then mixed with raw water to generate a 10% w/v solution. The hood over the mixing tank is ducted to the ventilation scrubber, which removes, scrubs and vents any gases released during the mixing process. After mixing, the SIPX solution is pumped from the mixing tank to a header tank for distribution to the zinc and gold-pyrite flotation circuits. The SIPX solution addition is via dedicated dosing pumps running off a ring main.

The SIPX is dosed to the following locations:

- Zinc conditioning tank 2 launder
- Zinc scavenger feed
- Zinc cleaner conditioning tank
- Zinc cleaner scavenger feed
- Gold flotation conditioning tank 2
- Gold scavenger feed

17.5.7 Copper Sulphate

Copper sulphate ($CuSO_4$) is the zinc and arsenopyrite/pyrite activator in the flotation process. Copper sulphate is supplied in powder form in 25 kg bags.

Copper sulphate mixing is completed manually by the operator. The bags are loaded manually to the mixing tank via a bag breaker and the powder is then mixed with raw water to generate a 15% w/v solution. The hood over the mixing tank is ducted to the ventilation scrubber, which removes, scrubs and vents any gases released during the mixing process. After mixing, the copper sulphate solution is pumped to a header tank for distribution to the zinc flotation circuit and gold flotation circuit. Copper sulphate addition is via dedicated dosing pumps running off a ring main.

- Zinc conditioning tank 2
- Zinc regrind cyclone feed hopper
- Gold flotation conditioning tank 2

17.5.8 Sulphuric Acid

Sulphuric acid is the pyrite/arsenopyrite activator in the gold flotation circuit. Sulphuric acid is delivered to site in liquid form at 95% w/v. Acid is stored in a 140 m³ capacity tank and dosing is performed using duty and standby variable speed pumps. The sulphuric acid is dosed to the gold flotation conditioning tank 1.

17.5.9 Flocculant

Flocculants are long chain molecules that aid solids settling by causing individual particles to stick together, thereby forming larger, heavier particles. Flocculant is supplied in powder form in 25 kg bags.

There is a dedicated flocculant mixing system and flocculant holding tank for the concentrate thickeners, tailings thickener and paste backfill clarifier (three systems in total). The flocculant mixing systems consist of a bag breaker and dry flocculant storage hopper, screw feeder and a 2.5 m³ mixing tank with agitator. Mixed solution gravitates to an agitated 5 m³ holding tank. Flocculant is mixed automatically with raw water on a batch basis to generate a 0.2% w/v solution. After a suitable hydration period, the flocculant is discharged into the flocculant holding tank, from where it is pumped by a positive displacement pump to the thickener.

17.5.10 Caustic Soda

Caustic soda is delivered to site in 25 kg bags of pellets. Caustic soda mixing is completed manually by the operator. The bags are loaded manually to the mixing tank via a bag breaker and the pellets are then mixed with raw water to generate a 20% w/v solution. The mixed solution gravitates to a 2.5 m³ holding tank. Caustic addition to the ventilation scrubber and the cyanide mixing tank is via dedicated lines running off a ring main.

17.5.11 Antiscalant

Antiscalant is added to the discharge of both process water pumps to inhibit the formation of gypsum ($CaSO_4.H_2O$) on the pipes in the flotation circuit. The antiscalant is supplied as a solution in 1,000 litre bulk boxes or 200 litre drums.

Antiscalant is distributed to the discharge of the process water pumps via a variable speed dosing pump.

17.6 SERVICES, WATER, AND AIR DISTRIBUTION

Services and utilities for the plant include process water, well, fresh water, low and high pressure compressed air, and instrument air.

The plant is serviced by a maintenance shop providing mechanical, electrical, and instrumentation facilities. An assay lab provides daily and shift quality monitoring information on solids and water samples for the safe and efficient operation of the plant.

17.6.1 Water Services

Due to the zero-discharge requirement, limited new water enters the circuit. The majority of the requirement for clean water (gland water, cooling water, etc.) is met by filtering recycled thickener and clarifier overflow. Fresh water from the underground aquifer by borehole is utilized for reagent mixing, potable water reticulation and other minor uses.

A seasonally variable quantity of water (up to 650 m^3/h) is pumped from the underground mine and treated in a water treatment facility. Total suspended solids (TSS) are reduced in settling ponds followed by two clarifiers. After settling, the water is further treated with lime for pH adjustment and precipitation of metals. Once it meets discharge criteria, the water is released into the nearby Mavrolakkas stream.

Make-up water for the plant is sourced from the water treatment plant as required.

17.6.1.1 Raw Water

Raw water sourced from the mine is directed to settling ponds (three operating and one emergency pond) to remove suspended solids. Raw water supply pumps draw from the final settling pond to feed the raw water tank.

Raw water is stored in a 9 m^3 water tank adjacent to the clean water and process water tanks. The raw water tank gravity feeds raw water to the following:

- Process water make-up (minimal)
- Reagent mixing
- Flocculant mixing

17.6.1.2 Clean Water

Clean water is used for:

- Filtration
- Regrind mill flushing and seal water
- Paste backfill vacuum pump cooling and seal water make-up
- Gland service requirements

Clean water is supplied by a sand-filtering paste plant clarifier, gold tailings, and zinc and gold concentrate thickener overflows.

17.6.1.3 Fire Water

The lower portion of the raw water tank provides a dedicated fire water reservoir for the fire water system. The fire water system includes an electric-driven “jockey” pump and a diesel-driven fire water pump to ensure a continuous supply of water to the fire system in the event of a power failure.

Fire hydrants and hose reels are placed throughout the process plant, fuel storage and plant offices at intervals that ensure coverage in areas where flammable materials are present.

17.6.1.4 Process Water

Process water is stored in a 181 m^3 tank. Duty and standby process water pumps provide process water to the following:

- Crushing area dust suppression
- Grinding area dilution water
- Trash screen sprays
- Flotation and regrind areas dilution and knock-down water
- De-aerator dilution water
- General purpose hose-down points

Process water is recycled from the zinc and gold/pyrite/arsenopyrite concentrate and tailings thickener overflows, concentrate and tailings filtrates and paste backfill clarifier overflow

17.6.1.5 Lead Flotation Process Water

Reclaimed water from the lead concentrate and tailings thickeners is stored in a 100 m^3 tank and reused in the grinding and lead flotation circuits. This minimizes the return of zinc flotation reagents to the lead flotation circuit.

17.6.1.6 Potable Water

Potable water is supplied in drinking bottles to the general workforce.

Well water is produced from a bore located in the regional aquifer and pumped to a water tank for storage. This water flows by gravity from the water tank to service the administration complex, plant offices, control room, site laboratory, washing facilities, and safety shower network. This water is not used as drinking water.

17.6.2 Air Services

A single screw compressor supplies plant and instrument air to the plant air receiver (via a refrigerated air dryer) and to the crushing area air receiver.

Two dedicated air compressors supply high pressure air to separate lead, zinc and gold concentrate filter air receivers for drying purposes.

Two additional air compressors provide air services to the tailings and paste backfill disk filters, as high pressure air, to assist with cake discharge; there are separate receivers at the filter building for each duty.

Low pressure air for flotation cells is provided by five blowers, four operating and one standby.

SECTION • 18 PROJECT INFRASTRUCTURE

18.1 SITE ACCESS

The Olympias mine is a producing gold, zinc and lead mine located approximately 2 km west of Olympiada, in the Halkidiki peninsula, Northern Greece. It is 8 km north of the Stratoni mine, also owned by Eldorado.

The Olympias mine lies about 100 km by road from Thessaloniki and is readily accessible by car and bus. Site roads are fully developed as paved roads, with access via Highway 16.

18.2 LOCAL SERVICES

The town of Olympiada can supply services; also, the towns of Stavros and Stratoni are both within 20 km of the mine site. The location of the mine relative to local services is shown in Figure 18-1.

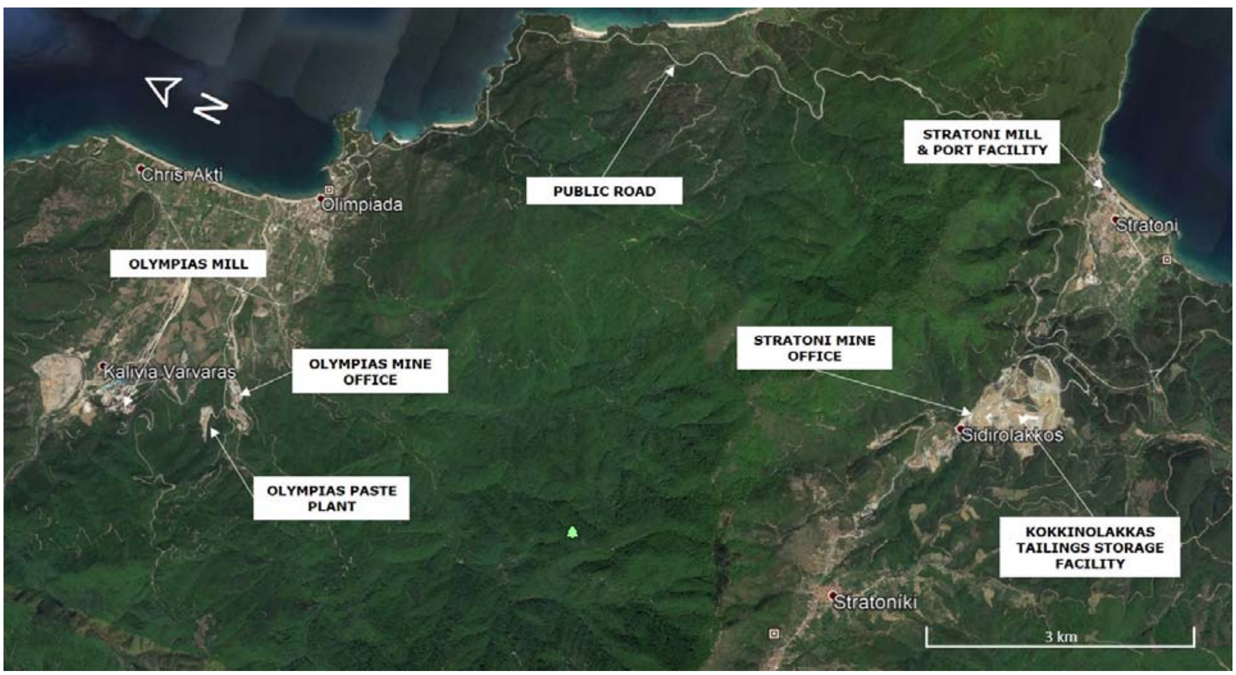


Figure 18-1: Location of Olympias Mine, Port and TMF

Mine infrastructure, including ancillary facilities and services, has been fully developed to support the existing mine production at 440 ktpa and requires minimal capital for expansion to 650 ktpa. Surface ancillary facilities are close to the decline and primary crusher. The ancillary facilities include the production services building, the surface workshop and warehouse, process plant, paste plant and surface fuel storage. The Olympias mine site layout is shown on Figure 18-2.

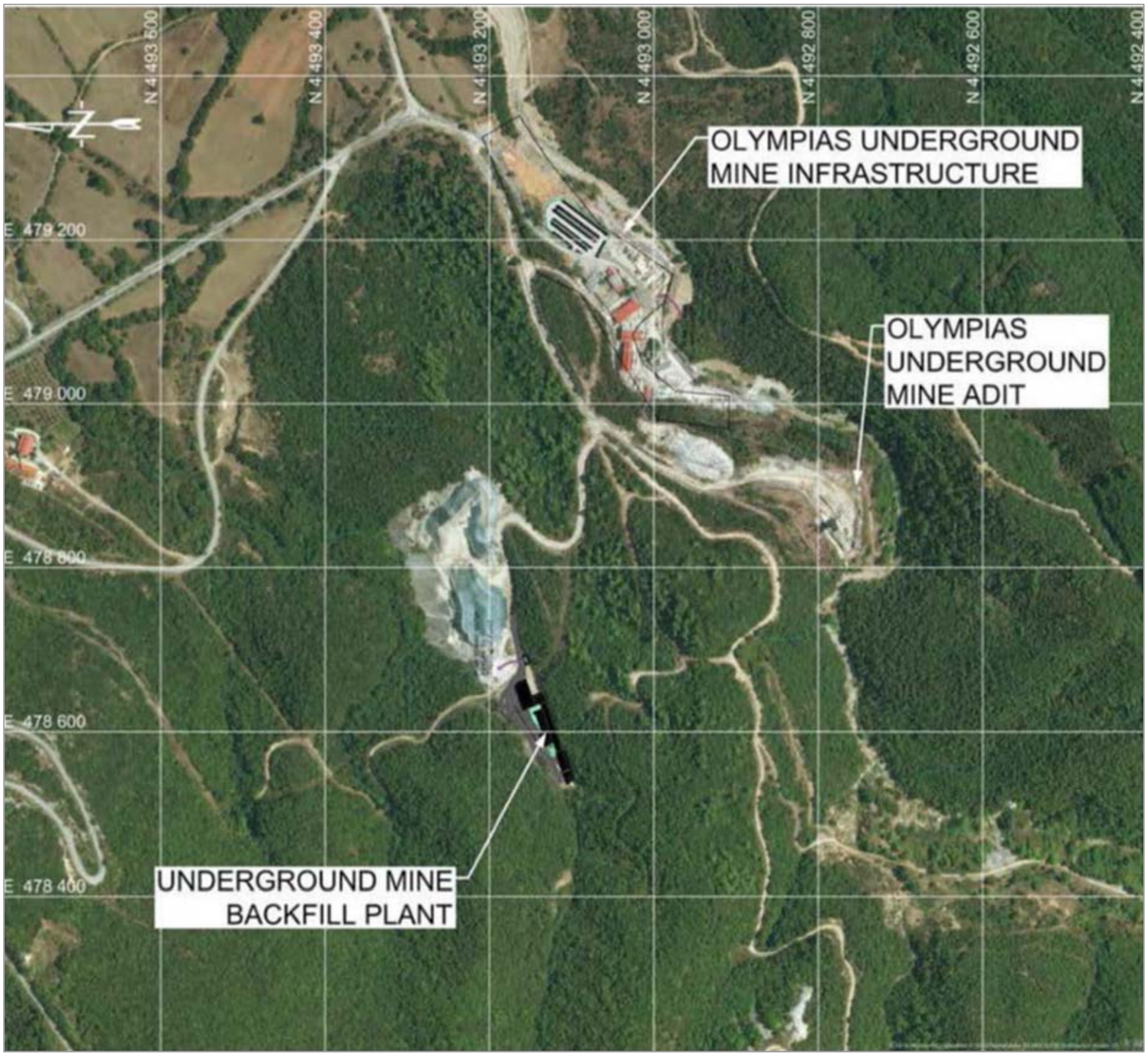


Figure 18-2: Olympias Site Layout

The Olympias concentrator is shown in Figure 18-3. Located near the concentrator are the ore storage bins, water treatment plant, and a nursery.

18.3 PORT FACILITIES

A port facility located at Stratoni, 26 km by paved road from the Olympias concentrator, is owned by Hellas Gold. Currently zinc concentrate is shipped via the port facility. Lead and arsenopyrite concentrate are bagged in the facility and shipped by truck to the port at Thessaloniki.

As part of the expansion to 650 ktpa, there is a two-phase plan to improve the port facility so that it can be fully utilized. The first phase is to rehabilitate and cover the conveyors and add dust collection. The second phase is to seismically upgrade the facility, increase the roof height, replace the floor slab, improve site water management, and increase the capacity of the existing material loading facility. These improvements will allow Hellas to ship all the concentrates to the port facility in International Carriage of Dangerous Goods by Road (ADR) certified trucks and eliminate the requirement to bag lead and arseno-pyrite concentrates. This will decrease transport costs. The changes have been budgeted and the two phases are scheduled to be completed in 2021 and 2023. They are part of the overall planned efficiency improvement initiative that will facilitate achievement of the NSR cut-off values used in the development of the mineral reserve.

Figure 18-3: Olympias Concentrator

18.4 COMMUNICATION SYSTEM

Primary means of communication is by radio, using a leaky feeder cable system that advances with ramp and level development. This system provides radio communications throughout the mine.

18.5 WATER SUPPLY

Water is supplied by both surface facilities' contact water run-off and from underground mine dewatering. A series of three settling ponds, with a fourth as a spare, is used to remove suspended solids. This water is pumped to a raw water tank next to the clean water and process water tanks.

Excess water from mine dewatering is treated in the surface water treatment plant. The water treatment plant consists of two clarification ponds followed by a neutralization unit (if required). Treated water that is not required is then discharged into the Mavrolakkas stream. The system total capacity of 400 m^3/hr is currently sufficient under normal circumstances. This system is being upgraded to handle the new anticipated maximum capacity of discharge from the underground of 650 m^3/hr as mining progresses to lower elevations. An allowance has also been made for a new secondary treatment plant to be constructed to ensure alignment of water quality with regulatory requirements.

Process water is reclaimed from the tailings thickening and filtration circuit and backfill clarifier overflow. A minimum amount of make-up water is supplied from the raw water tank. The lower portion of the process water tank provides a dedicated fire water reservoir for the fire water system. The fire water system includes an electrically-driven "jockey" pump and a diesel-driven fire water pump to ensure a continuous supply of water to the fire system in the event of a power failure.

Potable water is produced from a borehole located in the regional aquifer and pumped to the potable water tank for storage. Potable water gravitates from the potable water tank to service the administration complex, plant offices, control room, site laboratory, ablution facilities, and the safety shower network.

18.6 MINE DEWATERING SYSTEM

Details for the underground dewatering system are provided in Section 16 of this report.

18.7 MINE WASTE MANAGEMENT

Waste from the underground mine is brought to surface, where it is recycled underground as backfill or hauled to the waste disposal facility.

Thickened tailings are pumped to the paste plant. If thickened tailings production exceeds the maximum capacity of the paste plant, excess thickened tailings can be pumped to one of two tailings pressure filters located adjacent to the concentrator. Filtered tailings are then trucked to the paste plant for additional feed into the plant when required.

Excess thickened tails are pumped to a tailings filter to produce a cake with a moisture content of 13 wt% solids, which is then trucked to the TMF for dry stacking.

In addition to mine tailings from the Olympias mining operations, tailings from historic mining activities at Olympias are also being hauled to the Kokkinolakkas TMF. This facility is located 8.5 km South of Olympias (23 km by public paved road). It is designed to safely manage approximately 10.5

Mm^3 of mine waste at an average dry density of 1.6 t/m^3. An overview photograph of the facility under construction is provided in Figure 18-4.

Figure 18-4: Kokkinolakkas TMF

The TMF consists of upstream and downstream rockfill embankments, which act as tailings dams. The upstream embankment is designed to serve two purposes: as a water-retaining dam to prevent contaminated seepage leaving the facility, and to provide flood containment control. It is designed with a central clay core and associated filter and transition zones located upstream and downstream of the core. The downstream embankment is designed as a water retaining rockfill dam, with a steeper concrete face instead of geosynthetic bedding layers.

A liner system to minimize seepage for the facilities has been engineered that consists of a geosynthetic clay liner (GCL) overlain by a textured HDPE geomembrane and protected by a non-woven heavy-duty geotextile. A sandy gravel layer overlays the membrane to collect any seepage or drainage; it is also protected by a non-woven geotextile.

18.8 NEW FACILITIES

Construction of a new geology preparation laboratory, technical services building and expansion to the existing administration building is planned to finish in 2020. This is part of the support for the planned production increase.

18.9 SURFACE WORKSHOP

There is an existing surface maintenance workshop with two bays located near the East decline portal, with a nearby parts warehouse attached to it. Equipment that regularly exits the mine, such as the haulage fleet and light vehicles, can be serviced at the surface shop, whilst equipment such as production scoop trams (LHDs) jumbos, and production drills, will generally be serviced in the underground workshop, once completed.

18.9.1 Explosives / Cap Magazines

There is currently no surface or underground magazine. Explosives are brought daily to the mine from the supplier as required. There are plans to install a new set of magazines underground in 2020.

18.9.2 Fuel Storage

There is currently storage capacity for 60,000 litres of diesel fuel on site, with fuelling of equipment taking place on surface.

18.10 SHAFT

There is an existing headframe, Koepe hoist and shaft on site located near the mine shop area. Currently the hoist is operational and is used for inspection of the in-shaft pumping station only. The old underground crusher and loading pocket facility is in disrepair. A high-level trade-off study was completed and it was determined that rehabilitation of the shaft for ore haulage was not economically preferable to other material handling options.

18.11 ELECTRICAL POWER

Currently power is brought to site via a 10 mVA 20 kV overhead line from the local PPC grid. Power is distributed via overhead line to the various facilities throughout the site and to underground at 20 kV. Underground distribution is described further in Section 16.

To supply the required electrical power for the production increase, a new 150 kV 25 mVA Main Substation at +85 Level is being installed. As part of this capacity increase, new fiber optic cable will be installed between the various substations for rapid communications.

Currently there is 3,680 kVA of backup diesel power generation capacity available at site. An additional 2,500 kVA of back-up diesel generation in two generators will be added with the new substation, for a combined total of 6,180 kVA. Backup power is not centrally located, rather it is distributed at the various local substations, which should reduce the risk of power loss due to a failed overhead line.

SECTION • 19 MARKET STUDIES AND CONTRACTS

19.1 CONCENTRATE MARKETING

The Olympias plant produces three concentrates: an arsenopyrite/pyrite concentrate containing gold, a lead concentrate which also contains silver, and a zinc concentrate. Hellas Gold has negotiated multiple concentrate sales contracts with commodity traders for concentrates from Olympias. Agreements with several different customers are currently in use.

Maximizing revenue generation from Olympias requires optimization of the plant feed with respect to all three target metals. Optimized mine planning, carefully controlled blending of run-of-mine ore, and targeted operation of the concentrator are required to produce concentrate within the specifications of the current contracts and to maximize revenue at any particular time.

19.2 CONCENTRATE SALES

Eldorado sells gold, lead and zinc concentrates to various customers from its Olympias operation. The contracts in place are valid and represented correctly in the economic analysis.

19.3 GOLD CONCENTRATE

Contracts for sales of gold concentrate contain a range of settlement terms, generally based on the grade of gold contained within the concentrate. In summary the terms used in the financial model are as follows:

- Gold payability ranges from 59-71% and an average of 64% is used over the LOM
- No treatment or refining changes are assumed
- No smelter penalties are assumed

19.4 ZINC CONCENTRATE

Contracts for sales of zinc concentrate contain a range of settlement terms. In summary the terms used in the financial model and economic analysis are as follows:

- Treatment charges ranging from $80 - $275/dmt
- Zinc payability is fixed at 85%
- Gold and silver are not payable
- Smelter penalties applied for As and other metals, and average US$0.8 per year

19.5 LEAD CONCENTRATE

Contracts for sales of lead concentrate contain a range of settlement terms. In summary the terms used in the financial and economic analysis are as follows:

- Treatment charges ranging from of $150- $300/dmt

- Lead payability is fixed at 95%
- Silver is paid at 95%, gold is paid at an average of 60%
- Smelter penalties are applied for As, and other metals and average US$1 M per year
- Gold and silver refining costs are US$20/oz and US$1.00/oz respectively

19.6 OTHER CONTRACTS

19.6.1 Mining Contract

In 2019 Olympias initiated a mining development contract that is currently in place. The contractor will focus on capital development including the ramps, level access, ventilation drives, and sumps. The contractor supplies machinery, labour, consumables excluding explosives and shotcrete.

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

20.1 ENVIRONMENTAL IMPACT STUDY

The EIS for the Kassandra mines mineral deposits project includes an area of 26,400 ha, in north-eastern Halkidiki (Macedonia Region). The Complex includes the Skouries, Olympias and Stratoni sites.

The EIS considers the potential impact on the local and regional environment as it relates to:

- Open pit and underground workings
- Tailings impoundment
- Process plant
- Infrastructure necessary for the Project's operation

ENVECO S.A. (Environmental Protection, Management and Economy consultants) under Hellas Gold's management, has authored the full EIS. The EIS was submitted in August 2010 and approved in July 2011. The EIS covers all environmental issues for the Project.

20.2 LEGISLATIVE FRAMEWORK FOR THE PREPARATION OF THE EIS

For the preparation of the EIS, legislation, standards and directives required by the national and European Community legislation in force were used.

The following assessments have been made on the Olympias Project's environmental impact.

20.3 LANDSCAPE

Impact on the landscape of the property is of low significance, of local character and much will be non-permanent and partially reversible. All measures to minimize the anticipated impact will be taken.

The Olympias Project will benefit the Olympias area by allowing the rehabilitation of the Olympias valley at the end of the mine life.

20.4 GEOLOGICAL ENVIRONMENT

The DAF method is primarily used for the Olympias Project. The mined areas are backfilled with material comprised of inert tailings and cement. This prevents surface subsidence and ameliorates the acidity of underground water quality due to the alkalinity of the backfill.

20.5 GROUND – LAND USE

The impact will be localized to the areas of the disposal facilities but, with careful replanting at closure, will be minimized. No significant impact is anticipated on the rest of the Olympias Project areas. After the completion of operations, the Olympias Project areas will be rehabilitated according to appropriate and approved land uses.

20.6 WATER RESOURCES

No significant impact is expected either quantitatively or qualitatively on the water in the immediate mining and processing areas, or those areas within the immediate regional water system. The Olympias Project has been designed such that any leakage can be managed and controlled. The overall impacts have actually been assessed as moderately positive, due to the rehabilitation of the old tailings ponds and permanent restoration of the valley site when operations are complete.

20.7 ATMOSPHERIC ENVIRONMENT

Mining activities (mines, surface plants, disposal areas, etc.) are not assessed as causing any significant impact on the atmospheric environment, since all the necessary precautions have been taken for impact minimization. All potential impacts can be considered as moderate, temporary and reversible.

The Olympias Project's impact on the atmospheric quality of the area will be limited in the direct area of influence and will be mainly related to the local and restricted increase of dust and gas values. There will be no impact at all on the quality of the atmospheric environment of the nearby populated areas.

20.8 SOUND ENVIRONMENT

Aggravation of the sound environment is not seen as an issue in the direct and broader area of the operation. Accumulative impact on the sound environment is only anticipated in the area where the waste disposal facility is situated near to the process plant. The rest of the infrastructure elements are located sufficiently apart to avoid any significant accumulating impact on the sound environment of the Property.

20.9 ECOSYSTEMS – FLORA – FAUNA

All the areas used are outside of, but adjoin, Natura 2000 (an EU designated nature reserve); however, this places no restrictions on activities of Hellas Gold providing the terms of the EIS are adhered to.

Rehabilitation will prove positive for the return of certain species of mammals. The impacts are therefore estimated to be minor, permanent and partially reversible. In some occasions, impacts are positive.

20.10 SOCIAL AND FINANCIAL ENVIRONMENT

The positive consequences of the Olympias Project are very important for the Halkidiki Prefecture and for the National Economy. The general financial and social consequences deriving from the Olympias Project are the following:

- A significant contribution is made to the National Economy
- Significant infrastructure is constructed and equipped by local companies
- Service industries in the local economy expand
- New jobs are progressively created

20.11 CLOSURE PLAN

The fundamental criteria for Project closure and environmental rehabilitation include the following:

- The Project site must be handed back in a state that will not give rise to risks to the health and safety of people, the flora and fauna in the area and to environmental safety in general.
- All remaining structures, including interventions, in the natural terrain of the Project site, must not generate any risk to public health, safety, or the environment in terms of geotechnical stability.
- All remaining materials must not generate a risk to public health or the environment for future users of the area.
- Environmental rehabilitation must lead towards a self-sustaining ecosystem typical of the area. The purpose of the rehabilitation program must be to meet future land needs in the area, and rehabilitation must seek to re-create safe and stable biological conditions that encourage natural regeneration and the development of biodiversity.

Eldorado has undertaken to ensure that rehabilitation is in line with goals for safety, health and the environment. Surface facilities at the Olympias site will be decommissioned, and subsequently rehabilitated, once operations are completed.

Voids created from the extraction process will be hydraulically filled with a mixture of tailings and cement. When exploitation of the mineral reserves is complete, the only pending arrangements for full underground closure are the removal of mining and mechanical equipment from the workings, backfilling access works where appropriate, and the rehabilitation of the area around the entrances of the main access tunnels.

20.12 ENVIRONMENTAL COSTS AND GUARANTEES

Hellas Gold has provided a €50.0M ($57.5M) Letter of Guarantee to the MOE as security for the due and proper performance of rehabilitation works in relation to the mining and metallurgical facilities of the Kassandra Mines project and the removal, cleaning and rehabilitation of the old disturbed areas from the historic mining activity in the wider area of the project. Additionally, a Letter of Guarantee to the MOE, in the amount of €7.5M ($8.6M), has been provided as security for the due and proper performance of the Kokkinolakkas TMF (total €57.5M).

A ten-year monitoring period with estimated costs of US$1 Mpa per site (Stratoni, Olympias, and Skouries) is allowed for (US$30M) as a cost.

Five years after completion of the rehabilitation works and the change in land use of each individual facility in line with the master plan, the effectiveness of the guarantee will be examined by the External Auditing Committee, which will make a recommendation on any adjustment to the letter of guarantee that is needed.

SECTION • 21 CAPITAL AND OPERATING COSTS

All costs are presented in US Dollars (US$) based on the exchange rates shown in Table 21-1.

Table 21-1: Exchange Rates

Currency code	Currency name	Exchange rate
US$	United States Dollar	US$1.00 = US$1.00
C$	Canadian Dollar	C$1.00 = US$0.75
€	Euro	€1.00 = US$1.15

21.1 CAPITAL COSTS

The total capital cost estimate includes the expansion to 650 ktpa in the process plant, as well as capital development of the underground mine and sustaining capital, which is primarily for development, mine, processing infrastructure, and maintenance. Expansion capital specifically includes process plant upgrades, water management facilities, Phase 1 and Phase 2 of the Stratoni port facility upgrade as allocated amongst the Kassandra assets, related indirect costs, EPCM, owner's and contingency costs. The majority of the expansion capital is spent in the first three years (2020 – 2022) Sustaining capital costs are spread out over the LOM (to 2040). Capital cost estimates are summarized in Table 21-2. Sunk costs to end of 2019 are excluded from the capital cost estimate.

Table 21-2: Summary of Capital Costs

Area	Growth ($ x 1,000)	Sustaining ($ x 1,000)	Total ($ x 1,000)
Mining Dev't and Equipment	9,616	89,471	99,087
Mine Infrastructure	3,876	49,161	53,037
Process Plant	18,892	14,356	33,248
Tailings + Water	9,457	17,630	27,087
Surface Supports	5,739	2,056	7,795
Stratoni Port Facility	15,478	5,700	21,178
Others	1,566	2,829	4,395
Total Direct Cost	**64,624**	**181,203**	**245,827**
Indirect Cost	6,894	10,990	17,884
Spares and Fills	3,153	-	3,153
EPCM	6,306	3,060	9,366
Owners Costs	3,153	-	3,153
Contingency	15,765	-	15,765
Total Installed Costs	**99,894**	**195,253**	**295,147**

Direct costs were developed from a combination of firm or budget quotes, material take-offs, existing contracts, project specific references, and historical benchmarks.

Contingency was applied to each cost item in the estimate, based on the level of engineering definition and reliability of unit rates. The overall Project contingency is 19% for the growth capital and no contingency is applied to the sustaining capital.

Labour rates were derived from existing contracts for the Olympias Phase 2 Project. The all-in crew labour rates include all direct and indirect costs associated with contractors.

Quantities were generally based on detailed material take-offs and equipment lists, with some allowances for minor items.

Indirect costs were factored from direct costs and based on historical experience.

Owner's costs include labour and general and administrative costs for the owner's team during the period of active construction and were applied over the three years from 2020 to 2022.

21.2 OPERATING COSTS

Operating costs include allocations for the following.

- Underground mining
- Processing
- Concentrate transport
- Tailings filtration, materials handling, and stacking
- Water management
- General and administration

Operating costs are summarized in Table 21-3 and were calculated for each year of operation. It should be noted that operating costs calculated here are exclusive of refining and transport charges; these charges are excluded from the cut-off grade operating costs estimate as they are included as part of the calculation in developing the NSR model.

The underground mine operating costs were estimated based on actual operating costs to date and budget estimates that allow for the production ramp-up. Costs for THLOS which is currently not used, are estimated from factors to account for the difference in consumables and manpower related to the new methods.

The underground operating costs include all consumables (ground support, explosives, services, cement, aggregates, and fuel) and equipment required to meet the development and production schedule objectives. The operating unit costs for mobile equipment and fuel consumption rates were largely obtained from manufacturers and validated with the current operations costs. Labour requirements were developed to support the operation and maintenance of the fleet and for the general operation of the underground mine.

Table 21-3: Summary of Operating Costs

Category	LOM ($M)	LOM average ($/t ore)	Steady State: 2024 to 2038 ($/t ore)
Mining costs	908	69.6	48.9
Tailings and water management costs	56	4.4	4.4
Processing costs	652	50.4	50.4
G&A costs	310	23.9	21.3
Total operating cost	**1,925**	**148.3**	**125.0**

Process operating 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 consumption was calculated based on electrical load lists.

Operating costs for the water management facility are based on operation and maintenance requirements of pumps, pipelines, and mobile equipment. Labour requirements for operation of mobile equipment and supervision were calculated based on fleet sizing.

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

Underground mine operating costs were based on the production schedule, equipment operating costs from manufacturers, quotations on bulk items from primarily Greek suppliers and the staffing plan and consumables as per the 2020 budgetary quotes.

The process operating costs were based on consumption inputs from the design criteria, load lists, updated quotations of consumables and the staffing plan.

General and Administrative (G&A) costs were estimated based on actuals from the 2018-2019 years and include allocations of costs from a general Kassandra administrative centre and from the Athens office which overseas permitting and other related tasks.

Salaries and wages were provided as full-burden rates by Hellas Gold, and based on detailed personnel lists. Energy costs for diesel and power supply, consumable and maintenance costs have been obtained from supplier quotations, relevant in-house data, or were factored.

Current labour productivities are considered very low when compared to other international operations. Eldorado believes that the envisaged improvement initiatives are reasonable and achievable and has allowed for effective work hours per shift to gradually increase from 4 hrs to 5.5 hrs. This will be assisted by the increase in number of available faces for DAF mining though design and sequence changes. The introduction of more cost-effective THLOS mining for approximately one third of the steady-state tonnage is also a major factor in the future operating costs schedules. All of these initiatives should be in place as steady state by 2024. To illustrate the positive impact of these initiatives on LOM operating costs, a separate column was created in Table 21-3 showing estimated costs from 2024 to 2038.

SECTION • 22 ECONOMIC ANALYSIS

22.1 METHODS, ASSUMPTIONS AND BASIS

The economic analysis is based on the mineral reserves as defined in Section 15, the mining methods and production schedule as shown in Section 16, the recovery and processing methods as described in Section 17, and the capital and operating costs as outlined in Section 21.

The metal prices used in the economic model are $1,400/oz Au, $18/oz Ag, $2,100/t Pb and $2,400/t Zn. The economic model was also evaluated for sensitivity on metal prices, capital and operating costs at ±15%. All results reported in this section are in US$.

The economic model has been prepared on an annual life of mine basis. The effective date of the estimate is assumed to be December 31, 2019. The LOM for the current mineral reserves is projected to be 21 years.

22.2 CASH FLOWS

The annual cash flow forecast is built from the production schedule and capital and operating cost models. The model results are shown in Table 22-1 and Table 22-2. The build details of the production schedule, the operating and capital costs, the mill production and offsite costs, and the net cashflows over the LOM are shown.

The after-tax cash flow analysis shows that Olympias is a robust project once the additional capital is applied and the project reaches a steady state level of production. Beyond 2022 the project cashflows turn positive. The net present value (NPV) of the Project is estimated to be $1,067 M, using a discount rate of 5%, with positive after-tax net cash flows projected in all years other than 2019, the initial year of expansion capital expenditure.

Table 22-1: Key Economic Results

Olympias	Unit	Value
Total ore milled	kt	12,925
Gold grade	g/t	7.02
Silver grade	g/t	119
Lead grade	%	4.1
Zinc grade	%	5.3
Gold contained	koz	2,920
Silver contained	koz	49,300
Lead contained	kt	525
Zinc contained	kt	680
Payable gold	koz	1,637
Payable silver	koz	35,637
Payable lead	kt	408
Payable zinc	kt	494
Net revenue	$M	4,501
Total Capital costs	$M	295
Operating costs (total)	$M	1,925
Transport, treatment and refining costs	$M	408
Royalties	$M	67
Corporate tax	$M	423
Net post-tax cash flow	$M	1,866
Post-tax NPV @ 5% discount rate	$M	1,061
Post-tax NPV @ 8% discount rate	$M	783

Table 22-2: Olympias Mine Cashflow Analysis

Description		LOM	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Ore and Waste Production Schedule																							
East Zone Ore	t	6,890,063	389,241	376,469	392,632	357,649	251,146	350,584	299,611	295,480	325,598	352,132	413,562	313,222	215,021	221,602	298,568	307,410	300,431	316,216	247,376	320,912	309,895
West Zone Ore	t	4,135,387	-	24,294	34,820	130,419	257,457	189,242	235,044	227,937	219,091	216,740	142,285	219,693	270,108	269,370	269,370	269,369	267,830	180,041	293,527	266,161	152,587
Flats Ore	t	1,105,322	16,169	16,169	16,169	60,352	60,352	60,352	60,352	60,352	60,352	60,352	60,352	60,352	60,352	60,352	60,352	60,352	60,352	50,510	50,510	50,510	44,184
Remnants Ore	t	1,075,747	34,661	23,070	56,390	101,585	81,048	49,870	54,996	66,191	45,120	20,734	33,801	56,735	104,516	98,686	21,720	12,881	21,387	103,233	58,613	12,413	10,636
Lateral waste	t	2,682,377	285,316	301,379	270,703	229,363	243,957	270,231	256,898	249,397	190,815	96,966	7,075	14,302	17,230	21,569	24,487	4,900	6,442	31,192	17,181	9,167	6,058
Vertical waste	t	24,851	1,863	3,617	2,840	615	2,334	1,798	2,933	4,619	3,255	-	-	-	-	-	-	-	-	-	-	-	-
Total waste	t	2,707,228	287,179	304,996	273,543	229,978	246,291	272,029	259,831	254,016	194,071	96,966	7,075	14,302	17,230	21,569	24,487	4,900	6,442	31,192	17,181	9,167	6,058
Mill Feed Production Schedule																							
Ore tonnage	t	12,925,000	415,000	440,000	500,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	550,000
Gold	g/t	7.02	7.22	7.72	7.65	7.8	7.31	7.7	7.35	6.49	6.77	6.63	7.19	7.01	7.29	7.49	7.26	7.06	6.99	7.13	7.89	4.18	5.11
Silver	g/t	119	100.28	98.23	126.95	123.91	140.78	127.46	149.86	104.79	112.67	105.5	125.41	133.24	112.5	113.09	131.22	138.45	126.66	100.13	103.14	91.33	103.6
Lead	%	4.1	3.3	3.1	4	4	4.7	4.2	4.9	3.5	3.5	3.7	4.3	4.7	3.8	4	4.5	4.9	4.6	3.8	3.8	3.3	3.6
Zinc	%	5.3	4	3.5	3.9	4.3	4.9	5.1	5.3	4.8	5.1	5.2	6.1	6	5.4	5.9	5.5	6.1	5.6	4.9	5.2	5.9	5.5
Float Concentrates Produced																							
Lead concentrate	dmt	704,263	18,734	19,997	22,514	32,499	33,602	31,746	31,649	33,284	32,475	33,331	41,968	41,035	39,109	38,690	35,997	36,068	38,740	35,256	32,307	35,840	28,582
Zinc concentrate	dmt	1,257,253	29,855	28,367	37,119	55,484	61,620	54,622	55,581	53,999	53,922	57,263	67,156	74,624	72,167	73,121	66,902	71,726	69,344	68,452	69,985	66,806	51,171
As/Py concentrated	dmt	3,825,477	134,110	122,183	145,905	193,203	184,498	177,421	180,313	171,302	185,049	172,521	169,659	178,316	206,839	192,745	196,019	202,112	180,346	202,485	196,067	199,625	163,765
Flotation Tailings Produced																							
Floatation tailings tonnes	dmt	7,419,527	257,373	269,455	294,473	368,819	370,283	386,259	382,461	391,376	378,716	386,844	371,218	356,028	331,883	345,456	351,093	340,106	361,569	343,807	351,666	347,725	273,784
Float tailings as paste to ug	dmt	6,603,260	220,036	220,001	250,005	325,003	325,002	325,024	325,002	324,980	325,081	324,979	325,000	325,002	324,999	325,005	325,005	325,006	325,000	325,000	325,013	324,998	258,651
Float tailings to Kokinolakkas	dmt	816,267	37,337	49,454	44,468	43,817	45,282	61,235	57,459	66,396	53,635	61,865	46,218	31,026	6,884	20,450	26,088	15,100	36,569	18,807	26,653	22,727	15,133
Lead Concentrate																							
Recovery of Pb	%	83.52	70.00	70.00	70.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00
Recovery of Au	%	4.45	2.78	3.27	3.23	3.52	3.73	3.51	3.52	3.88	3.77	3.89	4.80	5.26	6.24	5.42	5.35	4.54	5.14	4.39	4.93	5.36	4.91
Recovery of Ag	%	77.57	73.00	73.00	76.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00	78.00
Grade of Pb	%	62.01	53.00	53.00	53.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00	63.00
Zinc Concentrate																							
Recovery of Zn	%	90.54	85.00	85.00	88.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00	91.00
Grade of Zn	%	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00	52.00
Gold Concentrate																							
Recovery of Au	%	86.00	84.5	85.0	85.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0	86.0
Payability of gold	%	64%	58.9	60.8	63.2	63.2	63.2	70.8	63.2	63.2	63.2	63.2	70.8	62.1	62.1	62.1	62.1	62.1	70.8	62.1	62.1	62.1	62.1
Payable Metals Production																							
Payable lead	t	408,183	10,944	10,406	15,506	20,417	23,687	21,489	24,988	17,512	17,529	18,922	21,526	24,001	19,400	20,033	22,839	24,883	23,154	19,175	19,384	16,657	15,730
Payable silver	oz	35,637,630	1,015,696	996,998	1,473,480	1,869,614	2,124,130	1,923,195	2,261,093	1,581,117	1,700,043	1,591,831	1,892,251	2,010,402	1,697,465	1,706,307	1,979,833	2,089,008	1,911,115	1,510,840	1,556,233	1,377,962	1,369,018
Payable zinc	t	494,297	12,668	11,115	14,150	20,519	23,241	23,989	25,268	23,032	24,354	24,934	29,122	28,699	25,497	28,231	26,299	29,078	26,493	23,221	24,739	28,141	21,507
Payable gold	oz	1,637,572	52,829	58,033	67,681	90,713	84,735	99,952	85,172	75,504	78,779	76,983	93,279	79,796	83,296	85,610	82,760	80,383	90,617	81,443	90,337	47,454	52,215
Revenue from Concentrate Sales																							
Gross lead revenue	$M	857	23	22	33	43	50	45	52	37	37	40	45	50	41	42	48	52	49	40	41	35	33
Gross zinc revenue	$M	1,186	30	27	34	49	56	58	61	55	58	60	70	69	61	68	63	70	64	56	59	68	52
Gross silver revenue	$M	641	18	18	27	34	38	35	41	28	31	29	34	36	31	31	36	38	34	27	28	25	25
Gross gold revenue	$M	2,293	74	81	95	127	119	140	119	106	110	108	131	112	117	120	116	113	127	114	126	66	73
Total gross revenue	$M	4,977	146	148	188	253	262	277	273	226	236	236	280	267	249	260	263	272	273	237	255	194	182
Treatment Transport and Refining and Royalties Costs																							

Description		LOM	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Total transport, treatment and refining cost	$M	408.4	28.6	23.1	13.6	18.3	19.8	19.0	20.7	17.0	17.8	17.9	19.8	20.7	19.1	19.6	20.3	21.9	20.0	18.4	18.6	18.4	15.7
Total royalties	$M	66.8	1.78	1.95	2.63	3.54	3.53	3.90	3.63	3.08	3.22	3.18	3.83	3.49	3.38	3.52	3.49	3.53	3.71	3.23	3.53	2.33	2.33
Net Revenue																							
Net revenue total	$M	4,501	115	123	172	231	239	254	249	206	215	215	256	243	227	237	239	247	250	216	232	173	164
Net rev per ore tonne	$/t	348	259	279	343	355	368	391	383	317	331	331	394	374	349	365	367	380	384	332	358	266	299
Operating Costs																							
Mining	$M	908.1	47.26	49.51	47.45	49.46	47.48	47.49	45.97	48.04	44.85	43.70	42.12	40.91	38.88	39.46	41.20	40.36	40.48	41.27	39.04	40.14	33.02
Processing	$M	651.7	21.84	21.84	25.29	32.74	32.74	32.74	32.74	32.74	32.75	32.74	32.74	32.74	32.74	32.74	32.74	32.74	32.74	32.74	32.74	32.74	26.15
Tailings and Water	$M	56.2	1.00	1.12	1.71	2.35	3.01	3.16	3.12	3.22	3.09	3.17	3.04	2.91	2.72	2.82	2.87	2.74	2.96	2.77	2.88	2.84	2.74
G+A	$M	309.8	18.11	16.11	14.25	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	14.60	13.13
Total operating costs	$M	1,925.8	88.22	88.57	88.70	99.15	97.83	97.99	96.43	98.59	95.29	94.21	92.50	91.16	88.94	89.62	91.42	90.45	90.78	91.38	89.26	90.32	75.03
Capital Costs																							
Development / expansion capital	$M	99.8	12.3	52.6	29.3	5.1	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining, mining capital	$M	195.2	35.0	29.0	13.7	14.1	8.4	21.6	10.2	9.2	10.0	4.0	2.7	6.0	2.8	2.6	6.3	2.4	2.3	5.9	2.8	2.6	2.6
Total capital	$M	295.1	47.4	81.6	43.0	19.2	8.7	21.6	10.2	9.2	10.0	4.0	2.7	6.0	2.8	2.6	6.3	2.4	2.3	5.9	2.8	2.6	2.6
Taxation																							
Tax rate			27%	26%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%
Corporate taxes paid	$M	423.3	0	0	0	0	16.6	19.8	18.5	7.6	10.3	10.4	34.2	33.2	30.8	33.6	33.5	36.2	37.0	28.7	33.5	19.0	20.6
Cash Flow																							
Undiscounted cashflow after tax	$M	1,851	(-35.8)	(-68.4)	31.8	113.7	121.9	117.1	125.4	91.5	100.1	107.8	127.9	113.1	105.0	111.9	106.9	118.0	119.8	89.1	107.3	61.2	77.6
Cash cost	$/oz	(-173)	889	812	176	(-52)	(-266)	(-164)	(-388)	(-24)	(-120)	(-167)	(-354)	(-502)	(-253)	(-325)	(-380)	(-544)	(-354)	(-125)	(-185)	(-342)	(-310)
All-in sustaining cost	$/oz	(-54)	1,552	1,312	379	103	(-167)	52	(-268)	99	6	(-115)	(-325)	(-427)	(-220)	(-294)	(-304)	(-514)	(-329)	(-52)	(-154)	(-287)	(-260)

Olympias Project Financial Results

After tax	Discount rate	Net present value
NPV0%	0%	$1,866
NPV5%	5%	$1,061
NPV8%	8%	$783

22.3 ROYALTIES AND OTHER FEES

Olympias is subject to a mineral production royalty, which is a sliding scale NSR type of royalty payable to the Greek government.

The relevant royalties for the relevant economic metals are shown in Table 22-3 to Table 22-6. The royalties are calculated on the payable metals that are produced by the site, with deductions allowable for the downstream smelter costs including refining and treatment. As such, the royalty can be considered as a net smelter return royalty.

For the Project case economics reported in this section, the gold royalty is assumed to be 2.0%, silver royalty is 1.0%, lead and zinc royalty are both 1.0%. The royalty legislation is prepared in Euros and the exchange rate used to convert to US$ is 1.2 US$: 1 Euro.

Table 22-3: Gold Royalty

Gold price ($/oz)		Royalty
From	To	
691	1035	1.0%
1035	1265	1.5%
1265	1495	2.0%
1495	1725	3.0%
1725	1898	4.0%
1898	2070	5.0%
2070	Above	6.0%

Table 22-4: Silver Royalty

Silver price ($/oz)		NSR royalty
From	To	
12	20	1.0%
20	29	1.5%
29	37	2.0%
37	46	3.0%
46	Above	4.0%

Table 22-5: Lead Royalty

Lead price ($/t)		NSR royalty
From	To	
1610	2070	0.5%
2070	2530	1.0%
2530	2990	1.5%
2990	3450	2.0%
3450	Above	2.5%

Table 22-6: Zinc Royalty

Zinc price ($/t)		NSR royalty
From	To	
1610	2070	0.5%
2070	2530	1.0%
2530	2990	1.5%
2990	3450	2.0%
3450	Above	2.5%

22.4 CLOSURE AND SALVAGE VALUE

Closure costs are captured by the economic model in the form of an asset retirement obligation (ARO) of €57.5M in favour of the Greek State that is offset by progressive rehabilitation and anticipated salvage value to be realized at the end of the mine life.

A ten-year monitoring period with estimated costs of $1 Mpa per site (Stratoni, Olympias, and Skouries) is allowed for ($30M) as a cost to the Kassandra mines as a whole.

22.5 TAXATION

Corporate income tax rates in Greece are 28% of net earnings for 2019. Favourable changes in the Greek Corporate Tax rate indicate a steady reduction in the rate by 1% each year until stable rates of 25% are applied from 2022 onwards. Corporate Tax has been applied in the financial model as follows:

- 2019, 28%
- 2020, 27%
- 2021, 26%
- 2022 and onward, 25%

Income from operations can be offset by operating costs and by depreciation of capitalized items according to a schedule of depreciation. For Olympias an average depreciation rate for all assets of 7.2% is used.

22.6 FINANCING COSTS

Cost of financing such as interest on loans are not included in the economic model. Olympias is assumed to be funded by Hellas Gold, and any costs or charges relating to Eldorado's funding of the Hellas Gold subsidiary are beyond the scope of the analysis.

22.7 THIRD PARTY INTERESTS

Hellas Gold is the 100% owner of the Kassandra mines, which includes Olympias. Eldorado owns a 95% interest in Hellas Gold, with the remaining 5% held by Aktor Enterprises Ltd., a Greek construction firm.

All data provided in this report is shown at the 100% ownership level.

22.8 SENSITIVITY ANALYSIS

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices, and of operating and capital cost assumptions on the projected financial returns. The results of the sensitivity analysis are provided in Table 22-7 and Table 22-8.

The test of economic extraction for Olympias is demonstrated by means of this sensitivity analysis. At the mineral reserve metals prices stated in Section 15, Olympias shows strong positive economics.

The sensitivity analysis also shows that the Olympias economics are robust when evaluated using lower metal price assumptions, or higher operating and capital costs.

Expansion to 650,000 tpa is required to obtain the economic performance outlined here. While there is limited capital required the process plant and no change in the overall footprint, the 650,000 tpa case still requires a permit from the Greek Government. If the project cannot expand, then the economics and the reserve that it is based on, would need further review.

Table 22-7: Metal Price Sensitivity Analysis

Sensitivity Ranges						
Parameters	Units	-15%	-7.5%	Project case	+7.5%	+15%
Gold price	$/oz	1,200	1,300	1,400	1,500	1,600
Silver price	$/oz	15.0	17.0	18.0	19.0	21.0
Lead price	$/t	1,800	1,900	2,100	2,300	2,400
Zinc Price	$/t	2,000	2,200	2,400	2,600	2,800
Results (after tax)						
NPV 0%	$M	1,327	1,604	1,866	2,164	2,415
NPV 5%	$M	724	897	1,062	1,241	1,395
NPV 8%	$M	519	655	783	925	1,046
Taxation	$M	248	336	423	512	591
Royalties	$M	42	52	66	76	120

Table 22-8: Capital and Operating Cost Sensitivity Analysis

Sensitivity Ranges				
Parameter	**Units**	**-15%**	**Project case**	**15%**
LOM Capex	$M	**256**	**295**	**347**
NPV 0% (after tax)	$M	1,927	1,866	1,836
NPV 5% (after tax)	$M	1,105	1,062	1,030
NPV 8% (after tax)	$M	823	783	755
LOM operating costs	$/t ore	**126**	**148**	**170**
NPV 0% (after tax)	$M	2,100	1,866	1,663
NPV 5% (after tax)	$M	1,205	1,062	929
NPV 8% (after tax)	$M	899	783	678

SECTION • 23 ADJACENT PROPERTIES

The Property is located within the Kassandra mines complex, which is comprised of a group of Hellas Gold mining and exploration concessions, covering 317 km^2. The other properties within the complex include Stratoni, currently in production, and the Skouries copper-gold porphyry deposit that is under development and on hold, pending permits.

SECTION • 24 OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation required to make the technical report understandable and not misleading.

SECTION • 25 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

Olympias is a mine with a long history that has only been recently put back into production by Eldorado. Commercial production was achieved in December 2017. Olympias is a gold-rich polymetallic carbonate replacement deposit hosted in an interlayered sequence of feldspar-biotite gneiss and marble. The orebody has a complicated geometry, with mining operations planned in five areas: East, West, Flats, Remnant and Tops. There are sufficient Mineral Reserves for a mine life of 20 years at a steady-state production rate of 650 ktpa. The mine has embarked on a series of initiatives to increase both productivity and production rates.

25.2 GEOLOGY AND MINERAL RESOURCES

The geological understanding of the deposit is good and, with the aid of an extensive core relogging program, the ore types have been well defined.

The data collection, sampling, sample preparation, security, and analytical procedures adopted by Eldorado for its exploration programs meet accepted industry standards, and the QA/QC results confirm that the assay results may be relied upon for Mineral Resource estimation purposes.

An important measure of performance at any producing mine is reconciliation of the block model to the final mill production figures. The reconciliation is detailed and thorough. It is currently providing a quarterly snapshot and demonstrating that the block model, and thus the mineral resources, are valid and robust. This validates the data underpinning the model and is, by association, a good verification of the work done.

The modelling and estimation are carried out in a diligent fashion and the reconciliation mentioned above has provided corroboration of the reasonableness of the mineral resource estimates. The mineral resource was classified as measured, indicated and inferred using logic consistent with the CIM Definition Standards (2014) referred to in NI 43-101.

25.3 MINING AND MINERAL RESERVES

Approximately 12.92 Mt of ore is planned to be mined at Olympias using a combination of DAF and TLHOS mining methods. DAF stope shapes were generated by Eldorado using MSO software to produce stope shapes based on the NSR COG of $133/t, minimum stope width of 4 m, a fixed stope height of 5 m, and a maximum FW / HW angle of 45°. The TLHOS stope shapes were created by Eldorado using a COG of $116/t to generate grade shells.

In the evaluation of underground Mineral Reserves, modifying factors were applied to the tonnages and grades of all in situ mining shapes to account for dilution and ore losses that are common to all mining operations.

Approximately 52% of the planned production is from DAF from all mining areas, whereas 30% is TLHOS and is planned to be mined only in the Flats zone. The remainder is from development ore and remnant extraction. The final selection of the most effective and economical mining method in any area of the mine should also consider the geometric configuration of the orebody, dilution, recovery, productivity and operating cost.

The achievement of increased productivity and the steady-state production rate of 650 ktpa is dependent on the successful implementation of a series of optimization initiatives, including:

- Introduction of the more cost effective TLHOS mining in the Flats area where ground conditions permit.
- A ramp-up period of three years to steady-state, with increasing realization of economies of scale.
- Improved worker and equipment productivity and reduced downtime.
- Modifying the mining method for larger DAF stopes such that there are multiple faces available in each stope.

The new PF system has been designed to meet future backfill requirements up to 650 ktpa ore production rate and operating at up to 70% utilization. PF will be used in DAF and TLHOS stopes. Backfill optimization study work is recommended. Successful implementation of best practice backfill design would reduce the regulated cement requirements and considerably reduce operating costs.

25.4 METALLURGY AND PROCESSING

EGC embarked on a three-phase execution of the Olympias Project:

- Phase I involved the recommissioning of the flotation circuit and processing of the existing tailings.
- Phase II (currently in operation) was commissioned in December 2017 and included upgrading of the process facilities to process 430 ktpa of ore from the underground mine. Lead / silver (lead) and arsenopyrite / pyrite gold (gold) concentrates are shipped through the Port of Thessaloniki and zinc concentrates are shipped through the Stratoni port. Tailings are used for underground backfill or filtered and trucked from the Olympias processing facility to the Kokkinolakkas TMF.
- Phase III involves upgrading of the existing Olympias process plant and port facilities at Stratoni to a rate of 650 ktpa.

The testwork completed to date demonstrated that the ore response to flotation was well understood, and the processing route is deemed appropriate. Actual results for the recoveries of all metals exceeded budget targets for first half of 2019. This would indicate that the processing plant is performing well above initial projections and this is anticipated to continue.

Other conclusions include:

- The main minerals of significance of the Olympias ores are galena, sphalerite, pyrite, and arsenopyrite. Other than quartz, all other minerals can be considered minor.
- The liberation of all sulphide minerals is excellent at a primary grind size P80 of 120 µm.
- Flotation of galena, sphalerite and pyrite / arsenopyrite in a sequential flowsheet is proven effective at producing lead, zinc and gold concentrates.

25.5 INFRASTRUCTURE

Mine infrastructure including ancillary facilities and services have been fully developed to support the existing mine production at 440 ktpa. Surface ancillary facilities are close to the decline and primary crusher.

As part of the expansion to 650 ktpa, there is a two-phase plan to improve the port facility such that it can be fully utilized. The first phase is to rehabilitate the conveyors, cover the conveyors and add dust collection. The second phase is to seismically upgrade the facility and increase the capacity of the existing material loading facility. These improvements will allow Hellas Gold to ship all concentrates from the port facility and eliminate the requirement to bag lead and arseno-pyrite concentrates, thereby decreasing transport costs. These phased changes have been budgeted and are scheduled to be completed in 2021 and 2023, respectively.

Currently power is brought to site via a 10 mVA 20 kV overhead line from the local PPC grid and is distributed via overhead to the various site facilities and to underground. For the production increase to 650 ktpa, a new 150 kV 25 mVA Main Substation at +85 Level is being installed, along with a new fibre optic cable for rapid communications.

Excess water from mine dewatering is treated in the surface water treatment plant (total capacity 400 m^3/hr). This system is being upgraded to handle the new anticipated maximum capacity of discharge from the underground of 650 m^3/hr.

25.6 FINANCIAL PERFORMANCE

Sufficient provision ($295M) has been made for the development of the Olympias project with respect to mine development, processing and infrastructure. Total LOM operating costs are estimated at $1.917B, for an average of $148/t ore milled. This reflects the assumed proportional use of the selected mining methods. The actual proportional use will determine actual mining costs at the time of extraction. The ability to increase productivity from the current labour force will be a key factor in realizing operating cost projections.

The after-tax cash flow analysis shows that Olympias is a robust project financially. On a post-tax basis, assuming the metal prices and payabilities used in the economic model (1400 $/oz Au, 18 $/oz Ag, 2100 $/t Pb, 2400 $/t Zn) the net present value (NPV) of the Project is estimated to be $1,061M at a discount rate of 5%, with positive cash flows in all years after 2022

25.7 CLOSURE

In order to ensure that rehabilitation of the Olympias Mine is in line with goals for safety, health and environment, surface facilities at Olympias will be decommissioned and the site rehabilitated once operations are completed. Voids created from the extraction process will be hydraulically filled. When mining ceases, equipment will be removed from the workings and the area around the portals rehabilitated. Hellas Gold has provided the Mining & Industrial Minerals Directorate of the Ministry of the Environment, with a letter of guarantee for €50M in favour of the Greek State as an assurance that the funds necessary for rehabilitation projects will be available. Additionally, a Letter of Guarantee to the MOE, in the amount of €7.5M, has been provided as security for the due and proper performance of the Kokkinolakkas TMF.

25.8 CONCLUDING STATEMENT

Olympias mine economics are projected to be very robust. Continued efforts to bring the mine into full production as soon as possible are recommended.

SECTION • 26 RECOMMENDATIONS

26.1 MINING AND MINERAL RESERVES

- It is recommended that additional drilling be carried out in the Inferred Mineral Resource and further along strike to the south and down plunge where the deposit is open in order to potentially extend mine life.
- Eldorado should continue to evaluate opportunities to apply TLHOS methods in good ground conditions.
- Olympias should initiate a backfill geotechnical study to optimize the strength requirements of backfill.
- Olympias should prepare a technical brief for submission to the mining authorities on an optimized backfill strength design using current industry best practice for stable openings of cured fill.
- Current labour productivities are considered low compared to other international operations and industry best practice. A detailed and monitored action plan should be in place to support the improvement initiative programs that include increasing productivity from the labour force over the next few years to support increased production rates.

26.2 METALLURGY AND PROCESSING

- Vibratory screening as a method of classification and the replacing the current cycloning in the primary grinding circuit should be considered for improved grinding, reduced slimes, and reduced operating costs through reduced recirculating loads.
- Olympias ores should be tested, for their amenability to dense media separation (DMS). This could be an important factor, particularly for Phase III plant expansion consideration. DMS, as utilized on many lead-zinc ores, would pre-concentrate ROM ores by separating low grade waste after crushing and ahead of grinding. Potential benefits of DMS include increased flotation feed grades, reduced cut-off grades (thereby increasing resource tonnes), reduced capital costs for mill expansion, reduced operating costs, and the possibility of producing NAG waste ahead of grinding.
- Use of ore sorting as a pre-concentration stage to remove waste material from the ROM stream should be investigated. Commercially available systems are available that separate particulate material into two streams based on particle bulk density. A crushed and sized fraction of the ROM stream is generally treated, with the lighter waste fraction being discarded as waste. Significant upgrading of the existing plant feed, and consequent significant savings in energy required for comminution, are possible if waste is rejected prior to grinding.
- Rejection of arsenic and antimony in the lead concentrate is challenging. Entrainment is believed to be a factor and thus column flotation, which involves a high degree of froth washing, is recommended to aid in the rejection of arsenic in the lead concentrate.
- Gold, sulphur, and arsenic grades of the gold-pyrite concentrate should be optimized in order to maximize smelter returns.

- Studies should be continued into options for extracting more value from the gold concentrate currently being sold directly to smelters and traders worldwide. Technical solutions do exist for extraction of much of the contained gold but these need to be able to demonstrate acceptable rates of return and to comply with environmental and safety requirements. If a viable extraction method is developed, then construction of the new metallurgical facility would follow.

26.3 INFRASTRUCTURE AND OTHER

- Eldorado should maintain a clear focus and progress on the expansion to 650 ktpa, inclusive of the two-phase plan to improve the port facility, which will decrease transport costs. These phased changes have been budgeted and are scheduled to be completed in 2021 and 2023 respectively.

SECTION • 27 REFERENCES

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SECTION • 28 DATE AND SIGNATURE PAGE

Date and Signature Page

The effective date of this report entitled "Technical Report, Olympias Mine, Greece" is December 31, 2019. It has been prepared for Eldorado Gold Corporation by David Sutherland, P. Eng., Colm Keogh, P.Eng, Paul Skayman, FAusIMM, Ertan Uludag, P.Geo. and Sean McKinley P.Geo. , each of whom are qualified persons as defined by NI 43-101.

Signed the 28th day of February 2020.

"Signed and Sealed"
David Sutherland

David Sutherland, P. Eng.

"Signed"
Colm Keogh

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I, David Sutherland, am a Professional Engineer, employed as Project Manager, of Eldorado Gold Corporation located at 1188 Bentall 5, 550 Burrard St., Vancouver in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Olympias Mine, Greece*, with an effective date of December 31, 2019.

I am a member of the Engineers & Geoscientists of British Columbia. I graduated from the Lakehead University with a Bachelor of Science (Physics) in 2003 and a Bachelor of Engineering (Mechanical) in 2005.

I have practiced my profession continuously since 2005. Since receiving my profession designation, I have worked exclusively on the design of mineral processing plants, assisted on numerous National Instrument 43-101 studies and have directed engineering and procurement on three mineral processing projects through construction. For 30 years I have been working in heavy industry including operations, maintenance and construction. During this time, I have lead the design and construction of major greenfield and brownfield construction projects in Canada, Turkey, and Greece.

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

I have visited the Olympias Mine on numerous occasions with my most recent visit occurring on October 2, 2019 (1 day).

I was responsible for coordinating the preparation of the technical report. I am responsible for the preparation or supervising the preparation of items 1, 2, 3, 4, 5, 6, 18, 20, 24, 25, 26, and 27 in the technical report.

I have had continual 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, Olympias Mine, Greece*, with an effective date of December 31, 2019, 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 28th day of February 2020.

"Signed and Sealed"
David Sutherland

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I, Colm Keogh, am a Professional Engineer, employed as Operations Manager, Olympias Mine of Eldorado Gold Corporation and residing at 1107 Miller Road, Bowen Island in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Olympias Mine, Greece*, with an effective date of December 31, 2019.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia) and also a member of l'Ordre des ingénieurs du Québec. I graduated with a Bachelor of Applied Science degree (Mining) from the University of British Columbia in Vancouver, British Columbia in 1988.

I have practiced my profession continuously from 1988 until 2001 and again from 2007 until present. I have been involved in mining engineering at underground base metal and precious mining operations in Canada, Ireland, Greece and Turkey. I have further provided mining engineering consultant services to underground base metal and precious metal properties in the United States, Peru, Brazil, Mexico, Mali, Canada and Ireland.

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

I have been working at the Olympias Mine since July 2019.

I was responsible for reviewing matters related to the estimation of mineral reserves, mining methods, project infrastructure, capital and operating costs, and economic analysis. I am responsible for sections 15, 16, 21 and 22.

I have had continual prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation as defined in 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, Olympias Mine, Greece*, with an effective date of December 31, 2019, 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 for which I am responsible contain all scientific and technical information that is required for disclosure.

Dated at Vancouver, British Columbia, this 28th day of February 2020.

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I, Paul Skayman, am a Professional Extractive Metallurgist, employed as Special Advisor to the COO, of Eldorado Gold Corporation and reside at 3749 West 39th Avenue in Dunbar, Vancouver, in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Olympias Mine, Greece,* with an effective date of December 31, 2019.

I am a fellow of the Australian Institute of Mining and Metallurgy. I graduated from the Murdoch University with a Bachelor of Science (Extractive Metallurgy) degree in 1987.

I have practiced my profession continuously since 1987 and have been involved in operation and management of gold and base metal extraction operations in Australia, Ghana, Tanzania, Guinea, China Turkey and Greece. This work has also included Feasibility Studies, Project Acquisition, Development / Construction and closure of said projects.

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

I have visited the Olympias Mine on numerous occasions with my most recent visit occurring on November 11, 2019 (1 day).

I was responsible for the preparation of the sections in this report that dealt with metallurgy and process operations and related costs and payability of the technical report. I am responsible for the preparation or supervising the preparation of items 13, 17 and 19 in the technical report.

I have had continual 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, Olympias Mine, Greece*, with an effective date of December 31, 2019, 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 28th day of February 2020.

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I, Ertan Uludag, am a Professional Geoscientist, employed as Resource Geologist, of Eldorado Gold Corporation and reside at 6779 Kitchener Street in the City of Burnaby, in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Olympias Mine, Greece*, with an effective date of December 31, 2019.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated from Middle East Technical University in Ankara Turkey with Bachelor of Science in Geological Engineering in July 1994.

I have practiced my profession continuously since 1996. I have been involved in ore control, mine geology and resource modelling work on gold, copper, zinc, lead and silver underground and open pit properties in Turkey, China, Greece, Canada and Romania, and rock mechanics work in South Africa

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

I have visited the Olympias Mine on numerous occasions with my most recent visit occurring on August 21 to 27, 2019.

I was responsible for the mineral resources and the preparation of related sections on sample preparation and analyses, data verification and mineral resource estimation for this technical report. I am responsible for the preparation or supervising the preparation of items 11, 12 and 14 in the technical report.

I have had continual 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, Olympias Mine, Greece*, with an effective date of December 31, 2019, 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 28th day of February 2020.

"Signed and Sealed"

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I, Sean McKinley, am a Professional Geoscientist, employed as Senior Geologist- Resource Development, of Eldorado Gold Corporation and reside at 2231 Bellevue Ave, in the City of Coquitlam in the Province of British Columbia.

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I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated from Queen's University in Kingston, Ontario with a Bachelor of Science (Honours) degree in geology in 1992 and subsequently obtained a Master of Science degree in geology from the University of British Columbia.

I have practiced my profession continuously since 1996 and have been involved in: mineral exploration (both greenfields and brownfields), mine geology (underground and open pit settings) and geological modelling on gold, copper, lead, zinc and silver projects in Canada, Ireland, Sweden, China, Mexico, Romania, Greece and Turkey.

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

I have visited the Olympias Mine on numerous occasions with my most recent visit occurring on July 2 to 5, 2019.

I was responsible for the preparation of the sections in this report concerned with geological information, exploration and drilling for this technical report. I am responsible for the preparation or supervising the preparation of items 7, 8, 9, 10 and 23 in the technical report.

I have had continual 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, Olympias Mine, Greece*, with an effective date of December 31, 2019, 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 28th day of February 2020.

"Signed and Sealed"

Sean McKinley

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Sean McKinley, P.Geo.