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April 3, 2019

It was brought to our attention that there was a typographical error in Table 22.11 on page 323-324 of the Gediktepe NI 43-101 Technical Report filed on April 3, 2019. The attached Gediktepe NI 43-101 Technical Report corrects this error.

OreWin
Independent Mining Consultants

Alacer Gold Corp.

Gediktepe

2019 Prefeasibility Study

March 2019

Job No. 18018

OreWin

IMPORTANT NOTICE

This notice is an integral component of the Gediktepe 2019 Prefeasibility Study (PFS19) and should be read in its entirety and must accompany every copy made of this report (the Technical Report). The Technical Report has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The Technical Report has been prepared for Alacer Gold Corp. (Alacer) by OreWin Pty Ltd (OreWin). The Technical Report is based on information and data supplied to OreWin by Alacer and other parties and where necessary OreWin has assumed that the supplied data and information are accurate and complete.

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Title Page

Project Name:	Gediktepe
Title:	Gediktepe 2019 Prefeasibility Study
Location:	Balıkesir Province, Turkey

Effective Date of Technical Report:	26 March 2019
Effective Date of Mineral Resources:	5 March 2019
Effective Date of Drilling Database:	21 March 2018
Effective Date of Mineral Reserves:	5 March 2019

Qualified Persons:

- Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining, was responsible for the overall preparation, mining and, the Mineral Reserve estimates in Sections: 1.1, 1.2, 1.6, 1.7, 1.10, 1.11; 2; 3; 4; 5; 15; 16; 19; 20; 21.1, 21.2.13, 21.3.1, 21.3.3; 22; 23; 24; 25; 26, 26.2, 27.
- Sharron Sylvester, BSc (Geology), MAIG, RPGeo (10125), employed by OreWin Pty Ltd as Technical Director – Geology, was responsible for Mineral Resources in Sections: 1.3 to 1.5, 1.11; 2; 3; 6 to 12; 14; 25; 26.1; 27.
- Peter Allen, BEng (Metallurgy), MAusIMM (CP 103637), employed by GR Engineering Services as Manager – Technical Services, was responsible for process plant and infrastructure in Sections: 1.8, 1.9,1.11; 2; 3; 13; 17; 18; 21.2.1 to 21.2.12, 21.2.14, 21.3.2; 25; 26.3, 26.4; 27.

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Signature Page

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/s Bernard Peters
Date of Signing: 26 March 2019
Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining

/s Sharron Sylvester
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Sharron Sylvester, BSc (Geology), MAIG, RPGeo (10125), employed by OreWin Pty Ltd as Technical Director – Geology

/s Peter Allen
Date of Signing: 26 March 2019
Peter Allen, BEng (Metallurgy), MAusIMM (CP 103637), employed by GR Engineering Services as Manager – Technical Services

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1 SUMMARY

1.1 Introduction

The Gediktepe project is located in the Balıkesir province of Western Turkey, some 67 km (air distance) south-east of Balıkesir province centre and 38 km east–south-east of the Bigadiç township (Figure 1.1). The Gediktepe project is a massive sulphide deposit hosted in metamorphic schist units. The upper portions of the Gediktepe deposit have been weathered, leached, and oxidised by naturally-occurring acidic surface water and ground water. The oxide zone is nearly devoid of base metals, but gold and silver remain relatively intact. The sulphide zone is polymetallic with potentially economic values of zinc, copper, gold, and silver. The major economic minerals are sphalerite and chalcopyrite. Pyrite is ubiquitous.

Figure 1.1 Location Map

Figure from Polimetal, 2018.

Polimetal Mining Industry and Trade Inc., otherwise known as Polimetal Madencilik San. ve Tic. A.Ş. (Polimetal), was formed in 2011 as a joint venture company between Lidya Madencilik San. ve Tic. A.Ş. (Lidya) (50%) and Alacer Gold Corp. (Alacer) (50%). Gediktepe mining licenses are held by Polimetal. In 2017, Polimetal assembled a study team made up of Polimetal personnel and consultants to carry out further feasibility assessment of the project. There has been one previous Technical Report describing the Gediktepe project: the Gediktepe 2016 Prefeasibility Study (PFS16).

This Technical Report, titled the Gediktepe 2019 Prefeasibility Study (PFS19), documents the outcomes of technical investigations by Polimetal as at the end of 2018.

In PFS19, mining is planned to use a conventional open pit mining method using excavators and trucks. Two main types of ore will be mined and processed: oxide ore to recover gold and silver, and sulphide ore to recover copper, zinc, gold, and silver. The proposed oxide treatment rate is 1.1 Mtpa in a carbon-in-pulp (CIP) plant. The sulphide treatment rate is 2.4 Mtpa, processing the polymetallic sulphide ore in a concentrator to produce separate copper and zinc concentrates.

1.2 Property Description and Ownership

The Gediktepe Operating License (OL) RN 85535 is held by Polimetal.

<u>Operating License – RN 85535</u>

The General Directorate of Mining and Petroleum Affairs (GDMPA) approved the merging of OL 20054077 and Exploration License (EL) 201400291 into one OL (RN 85535) on 29 July 2016. RN 85535 is valid until 23 June 2036. Figure 1.2 shows RN 85535.

On 21 February 2018, GDMPA also approved Polimetal's application for a production permit for clay and aggregate for three locations within RN 85535.

Figure 1.2 Current Gediktepe Operating License

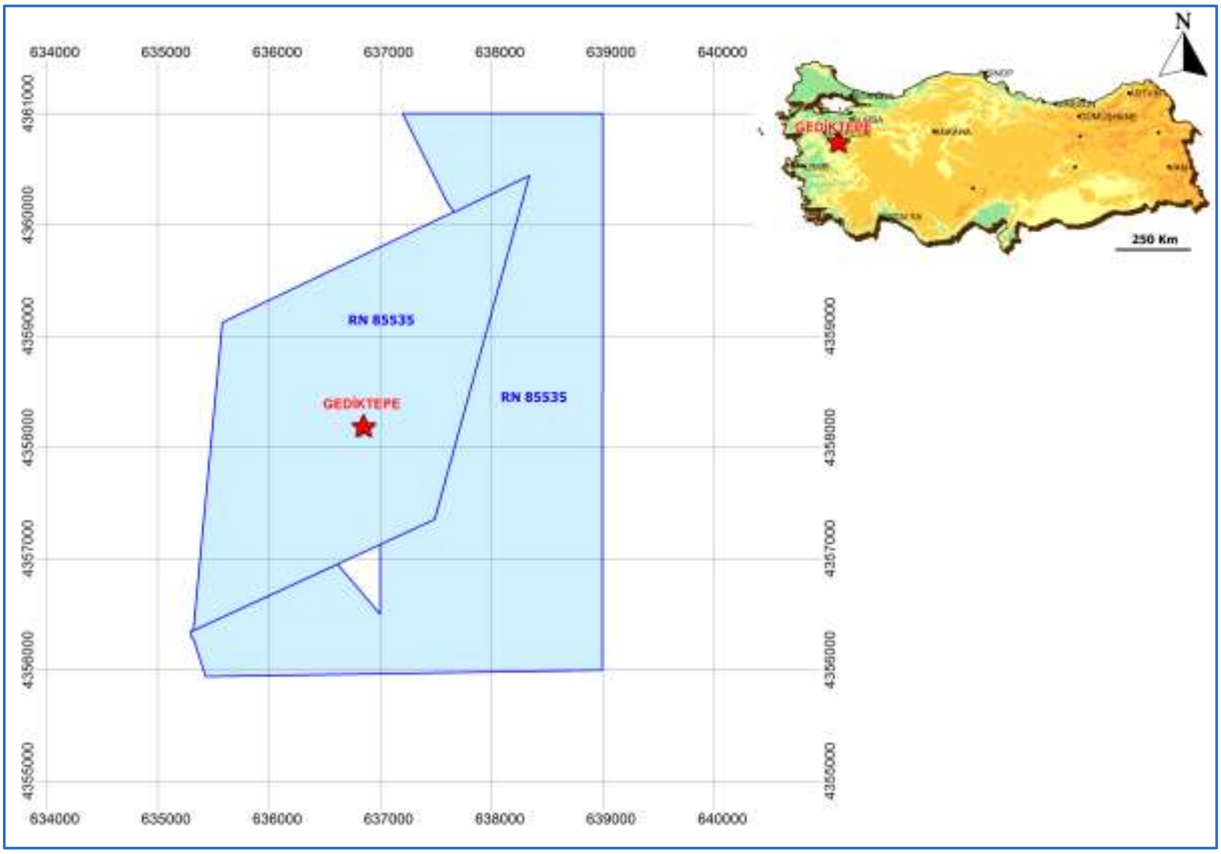

Figure from Polimetal, 2018.

1.3 Geology and Mineralisation

1.3.1 Geology

The Gediktepe project is located within the Afyon tectonic zone, which is one of the main tectonic domains in Turkey. The Afyon zone is a belt consisting of generally low-grade weathered metamorphic rocks. It is located between Menderes Massive to the west and the city of Denizli to the south.

The Gediktepe regional geology comprises Upper Paleozoic metamorphics and Lower to Middle Miocene intrusives and volcanics.

The metamorphics are generally composed of gneiss, schist, mica schist, chlorite schist, phyllite, amphibolite, marble, and quartzite, with varying degrees of metamorphism. These metamorphics are stratigraphically overlain by Triassic carbonates and fragmental units, Jurassic limestone, and upper Cretaceous ophiolitic mélange.

Magmatic rock intrusions developed later between the Oligocene and Lower Miocene, due to extensional features in western Anatolia. Those intrusions cut the Paleozoic metamorphic and Upper Cretaceous ophiolitic rocks, establishing in the region what is now called the Alaçam Mountains granites, which outcrop in an arc-shaped geometry over an area of nearly 30 km^2.

The Alaçam Mountain granites consist of granite porphyries and aplitic dykes, creating hornfelsic belts where they intruded Paleozoic metamorphic rocks. Skarn formations are abundant at the contacts of recrystallised limestone blocks of Upper Cretaceous ophiolitic mélange.

Lower Miocene volcanic rocks are positioned stratigraphically above Paleozoic to Upper Paleozoic metamorphics and Upper Cretaceous ophiolitic mélanges. Lower Miocene volcanic rocks comprise andesitic and dacitic intrusions, domes, lava flows, dykes, and volcanogenic sedimentary rocks.

Volcanic rocks, surrounding the Lower–Middle Miocene Alaçam Mountains, outcrop over an area of hundreds of square kilometres from the towns of Bigadiç to Simav, and from Dursunbey to Düver Hill. The volcanic suite includes ignimbrite of felsic (dacite and rhyolite) composition. Ignimbrites have the widest distribution among felsic volcanic rocks, with thicknesses of up to 350–400 m around the Alaçam Mountains. In some areas, these units are overlain by Pliocene terrestrial sediments and Quaternary alluvial deposits, sourced from the local metamorphics, ophiolitic mélange, granitoids, and felsic volcanic rocks.

Upper Paleozoic metamorphics are the most common units at Gediktepe, with the stratigraphic sequence, from top to bottom, being:

- Dacite and Pyroclastic
- Calcschist
- Feldspar–Quartz Schist
- Chlorite–Sericite Schist
- Quartz Schist

The second-most common rocks at the project are the Lower to Middle Miocene volcanics, observable around Karadikmen Hill, south-west of Gediktepe. These comprise altered dacites–rhyodacites, characterised by lava flows and pyroclastics.

The youngest units at the project are mineralised gossan and ferricrete, along with talus, colluvium, and alluvium, being weathering products of the host rock.

1.3.2 Mineralisation

The mineralisation at Gediktepe is associated with greenschist facies schist units. The mineralisation is thought to be developed syn-genetically in sedimentary units elongated along a north-east / south-west trending structure zone and metamorphosed to schist. Greenschist minerals are generally actinolite, chlorite, albite, and epidote.

Massive sulphide-type mineralisation occurs as lens shaped units trending north-east / south-west and dipping at approximately 20° to 40° to the north-west. Minerals include pyrite, sphalerite, tetrahedrite, tenantite, chalcopyrite, galena, and magnetite. The units are cut by later north-west / south-east trending post-mineralisation structures within the oxide zone, in which the sulphide mineralisation has been completely leached out, leaving gold and silver relatively intact.

Potentially-economic gold–silver–copper–zinc mineralisation is present to varying degrees in the sulphide zone. The mineralisation at Gediktepe is divided by Polimetal into five main types, as summarised in Table 1.1.

Table 1.1 Mineralisation Types

Horizon	Mineralisation Type
Oxide	Gossan
Sulphide	Massive Pyrite
	Massive Pyrite–Magnetite
	Enriched
	Disseminated Sulphide

The characteristics of the Gediktepe mineralisation have been interpreted as a convex massive sulphide type deposit, which implies a syngenetic style of sulphide mineralisation. Subsequent weathering and oxidation have been responsible for the development of oxide and gossan horizons.

1.4 Status of Exploration, Development, and Operations

Exploration drilling (Phase 1) commenced on April 2013. Throughout the phases drilling by both diamond core (DD) and reverse circulation (RC) drilling was completed by local contractor companies. Diamond core holes were predominantly started using PQ core size, and rarely with HQ holes. Most deeper holes, however, needed to switch to HQ at depth. RC drilling was restricted to Phases 2 and 3 and was used on the margins of the deposit to define extensions or set limits, and for infill in some parts of the deposit.

Most holes have been drilled vertically to intersect the low-angle zones of mineralisation. Eight of the initial 11 Phase 1 holes were angled holes, with the remainder of the holes vertical or sub-vertical. The average deviation of the surveyed holes is less than 1° per 100 m.

At the end of each phase of drilling, drillhole collars were surveyed by a local surveying firm. RC drillholes were not downhole surveyed.

1.4.1 PFS19 Drillhole Dataset

The geological modelling work proceeded with a subset of the total drillhole database, resulting in a dataset comprising 629 RC and diamond drillholes totalling 70,127 m of drilling (PFS Drillhole Dataset). Table 1.2 summarises the PFS19 Drillhole Dataset.

The cut-off date for the PFS19 Drillhole Dataset was 21 March 2018.

Of the 438 diamond drillholes in the PFS19 Drillhole Dataset, 388 have downhole survey data.

Table 1.2 Summary of PFS19 Drillhole Dataset to 21 March 2018

Drilling Phase	Period	Diamond Drilling		Reverse Circulation Drilling	
		No. of Holes	Metres Drilled	No. of Holes	Metres Drilled
1	2013	11	1,529	–	–
2	2013/2014	144	17,158	84	6,920
3	2014/2015	153	26,544	107	6,309
4	2017	94	5,252	–	–
5	2017/2018	36	6,414	–	–
Total		**438**	**56,898**	**191**	**13,229**

A cell model was constructed, with coding applied to represent volumes of geological units, and mineralisation and weathering domains. The sample data set was coded in a corresponding fashion and evaluated statistically and geostatistically. The major grades of economic interest to the project, Au, Ag, Cu, and Zn, were estimated into the mineralisation domains and background material portions of the cell model. The minor grades As, C, Pb, S, Fe, and Hg, along with bulk densities, were similarly estimated into both mineralisation and background domains.

The modelled estimates were assessed for levels of geological confidence, and accordingly classified into Measured, Indicated, and Inferred categories, referencing CIM guidelines (CIM, 2014). The Mineral Resource tonnages and grades have been reported using Net Smelter Return (NSR) cut-offs and constrained within an optimised pit.

1.5 Mineral Resources

The Measured, Indicated, and Inferred Mineral Resources for the project at specified NSR cut-offs are presented in Table 1.3, (Measured plus Indicated combined at the end).

The more-detailed breakdown of Mineral Resources by mineralogy-type is included in Table 14.23 (Measured, Indicated, and Inferred) and Table 14.24, (Measured plus Indicated combined).

Table 1.3 Summary of Gediktepe PFS19 Mineral Resources

MEASURED	Tonnes (kt)	Grade					Metal			
		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Total Oxide	–	–	–	–	–	–	–	–	–	–
Total Sulphide	3,999	0.67	25.1	1.01	1.83	0.34	86	3,221	40	73
Total Measured	**3,999**	**0.67**	**25.1**	**1.01**	**1.83**	**0.34**	**86**	**3,221**	**40**	**73**

INDICATED	Tonnes (kt)	Grade					Metal			
		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Total Oxide	2,674	2.71	66.3	0.10	0.10	0.47	233	5,703	3	3
Total Sulphide	23,544	0.74	27.6	0.85	1.69	0.33	560	20,865	200	399
Total Indicated	**26,217**	**0.94**	**31.5**	**0.78**	**1.53**	**0.34**	**792**	**26,568**	**203**	**402**

INFERRED	Tonnes (kt)	Grade					Metal			
		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Total Oxide	23	0.95	21.8	0.23	0.14	0.12	1	16	0	0
Total Sulphide	2,958	0.53	20.2	0.76	1.16	0.27	51	1,926	22	34
Total Inferred	**2,981**	**0.54**	**20.3**	**0.76**	**1.16**	**0.27**	**51**	**1,941**	**23**	**34**

MEASURED + INDICATED	*Tonnes (kt)*	*Grade*					*Metal*			
		Au (g/t)	*Ag (g/t)*	*Cu (%)*	*Zn (%)*	*Pb (%)*	*Au (koz)*	*Ag (koz)*	*Cu (kt)*	*Zn (kt)*
Total Oxide	*2,674*	*2.71*	*66.3*	*0.10*	*0.10*	*0.47*	*233*	*5,703*	*3*	*3*
Total Sulphide	*27,542*	*0.73*	*27.2*	*0.87*	*1.71*	*0.33*	*645*	*24,086*	*241*	*472*
Total M + I	***30,216***	***0.90***	***30.7***	***0.81***	***1.57***	***0.34***	***878***	***29,790***	***243***	***475***

Notes:
1 CIM definitions were followed for Mineral Resources.
2 Effective Date of Mineral Resource is 5 March 2019.
3 Mineral Resources are estimated at NSR cut-offs of $20.72/t for oxide and $17.79/t for sulphide.
4 Mineral Resources have been constrained using an optimised pit shell, to reflect reasonable prospects of economic extraction.
5 Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6 Mineral Resources are inclusive of Mineral Reserves, except for mining losses and grade dilution, which are determined through re-blocking of the resource model after declaration of the Mineral Resource.
7 Mineral Resources are quoted on a 100% project basis.
8 Totals may not match due to rounding.

1.6 Mineral Reserves

The Gediktepe Mineral Reserves, reported according to the CIM guidelines, are summarised in Table 1.4. Due to its polymetallic nature, the oxide and sulphide portions of the Mineral Reserves are quoted at different NSR cut-offs based on metal prices, metal recoveries, plus on and off-site processing costs.

Table 1.4 Gediktepe PFS19 Mineral Reserves

Classification	Tonnage (kt)	Grade				Contained Metal			
		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Oxide									
Proven	–	–	–	–	–	–	–	–	–
Probable	2,755	2.34	56.7	–	–	207	5,020	–	–
Proven & Probable	2,755	2.34	56.7	–	–	207	5,020	–	–
Sulphide									
Proven	3,620	0.68	26.7	1.03	1.93	79	3,105	37	70
Probable	14,960	0.89	33.1	0.89	1.99	429	15,903	133	298
Proven & Probable	18,580	0.85	31.8	0.92	1.98	509	19,008	170	368

Notes:
1 CIM definitions were followed for Mineral Reserves.
2 Effective Date of Mineral Reserve is 5 March 2019.
3 Mineral Reserves were reported using a Net Smelter Return (NSR) based on metal prices of $1,300/oz Au, $18.5/oz Ag, $3.30/lb Cu, and $1.28/lb Zn, smelter terms for treatment and refining charges and transport including ocean freight for sulphide ore concentrates.
4 Cut-offs applied were: oxide ore $20.67/t and sulphide ore $17.74/t. Additionally, enriched mineralisation with a Cu/Zn grade ratio < 0.75 is considered to be waste.
5 Metal prices used for economic analysis to demonstrate the Mineral Reserve are Au $1,315/oz, Ag $18.0/oz, Cu $3.20/lb and Zn $1.10/lb.
6 Reported Mineral Reserves incorporate and include mining losses and grade dilution that are not reported in the Mineral Resource.
7 Only Measured Mineral Resources (and dilution) were used to report Proven Mineral Reserves and only Indicated Mineral Resources (and dilution) were used to report Probable Mineral Reserves.
8 Mineral Reserves are a subset of, not additive to, the Mineral Resources and are quoted on a 100% project basis.
9 Totals may not match due to rounding.

1.7 Mining Methods

Open pit mining is planned to be carried out on 2.5 m flitches using small excavators (3–4 m^3 capacity) and trucks. Drilling and blasting will be required. All mining services will be performed by a suitably qualified and experienced Turkish mining contractor. It is currently anticipated that the same mining contractor will provide initial construction services, particularly construction of the tailings storage facility (TSF).

Grade control to determine material types and ore boundaries will be performed based on blasthole sampling and assaying, and under the control of the mine geologists. Feed to the process plants is expected to be a combination of both direct tipping and reclaim from run-of-mine (ROM) stockpiles to ensure optimal feed to the process plant, particularly for sulphide ore.

The open pit design (Figure 1.3) was based on pit optimisation analysis using the relevant cost, revenue, and physical parameters. The ultimate pit design was further sub-divided into a series of intermediate pit stages designed to defer waste mining and facilitate blending and project cash flow.

Figure 1.3 Ultimate Pit Design

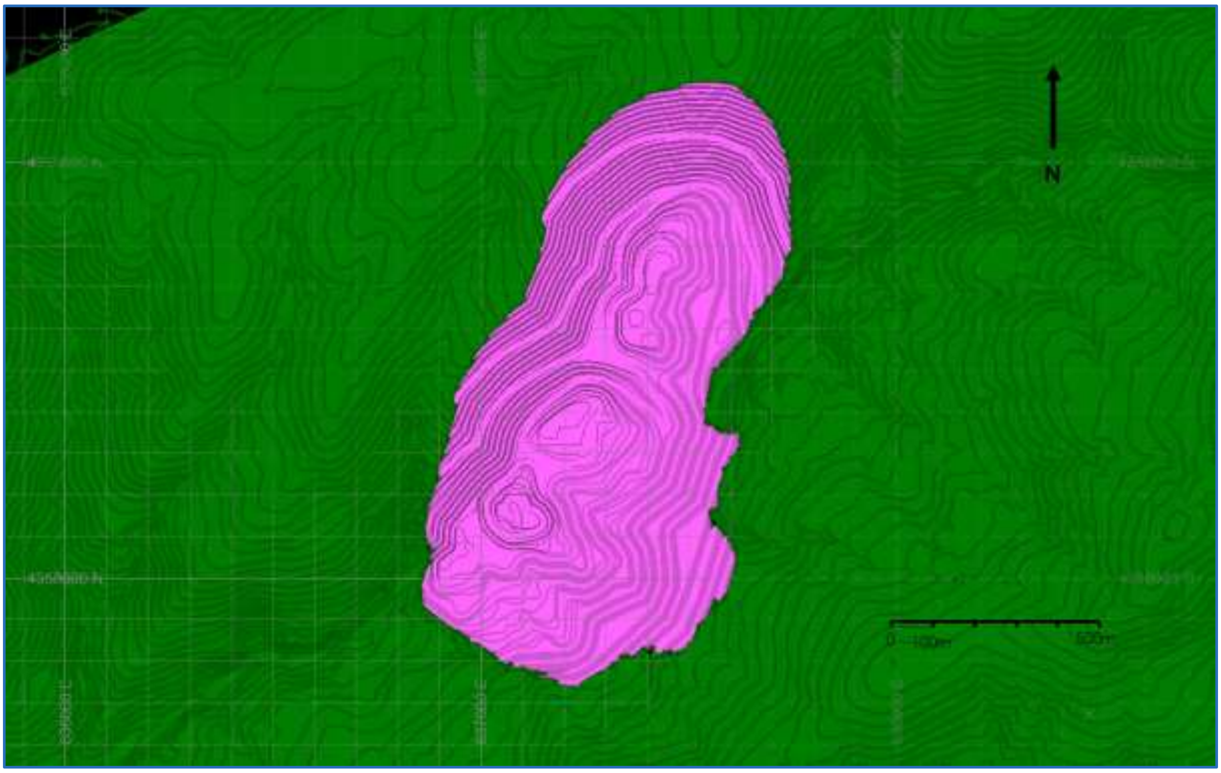

Figure by OreWin, 2019.

Mine and process scheduling was carried out on a monthly basis for the first five years (including a one-year pre-strip) and quarterly for the remainder of the mine life. It was guided by a linear programming tool to facilitate the required ore blending outcomes.

In addition to ore mining targets, waste mining in the pre-strip and initial years targeted minimum quantities of suitable waste to construct the clean water pond and the TSF to manage mine area run-off and ensure tailings storage availability at the commencement of oxide ore processing.

Figure 1.4 shows total mining by annual period.

Figure 1.4 Total Tonnage Mined by Period

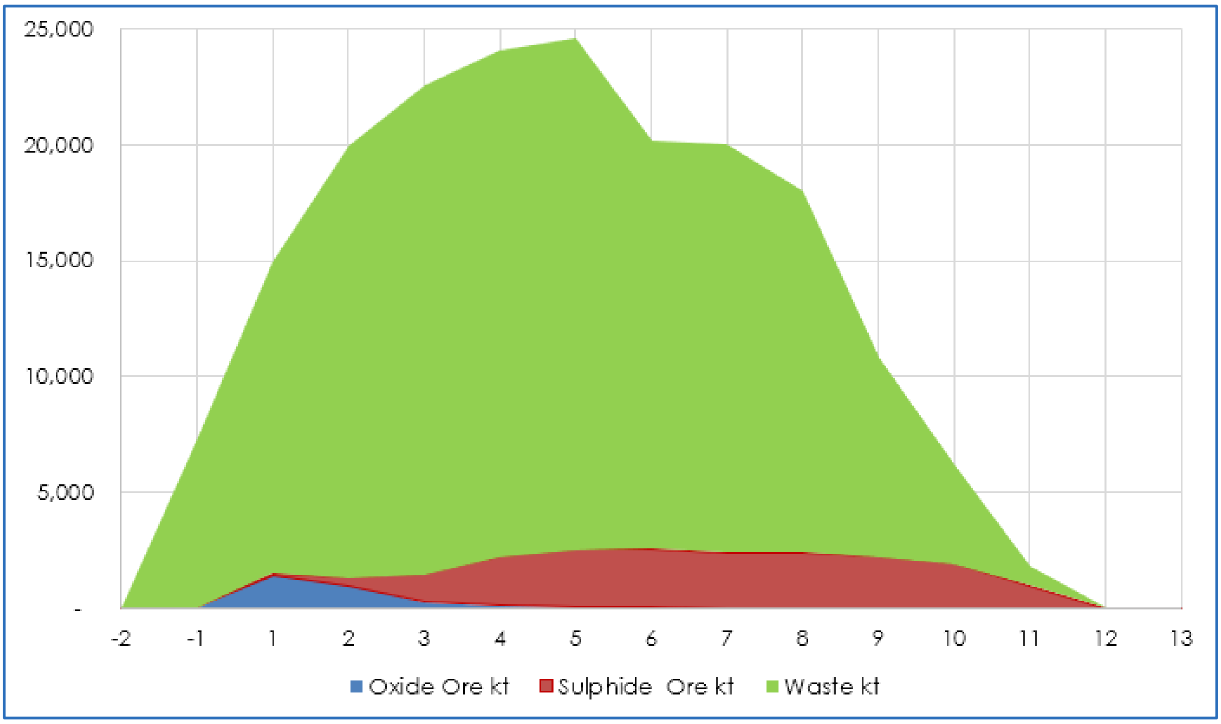

Figure by OreWin, 2019.

Oxide and sulphide processing schedules honouring the mining and processing constraints are shown in Figure 1.5 and Figure 1.6.

Figure 1.5 Oxide Processing

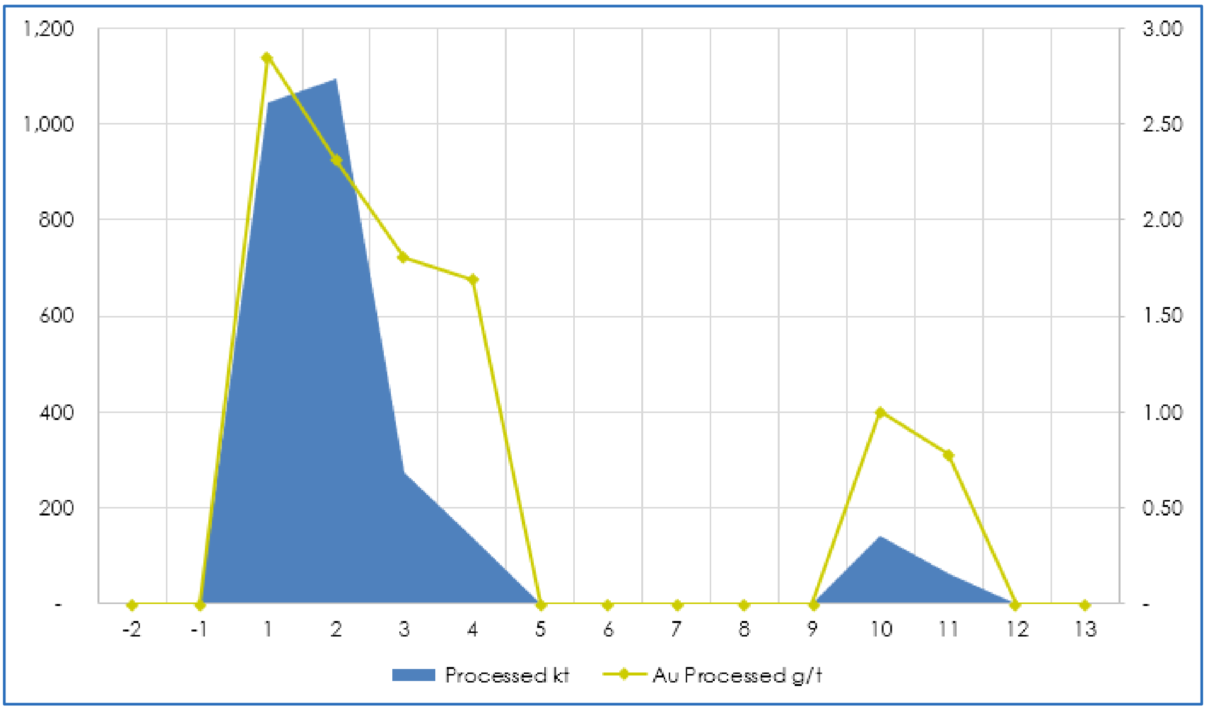

Figure by OreWin, 2019.

Figure 1.6 Sulphide Processing

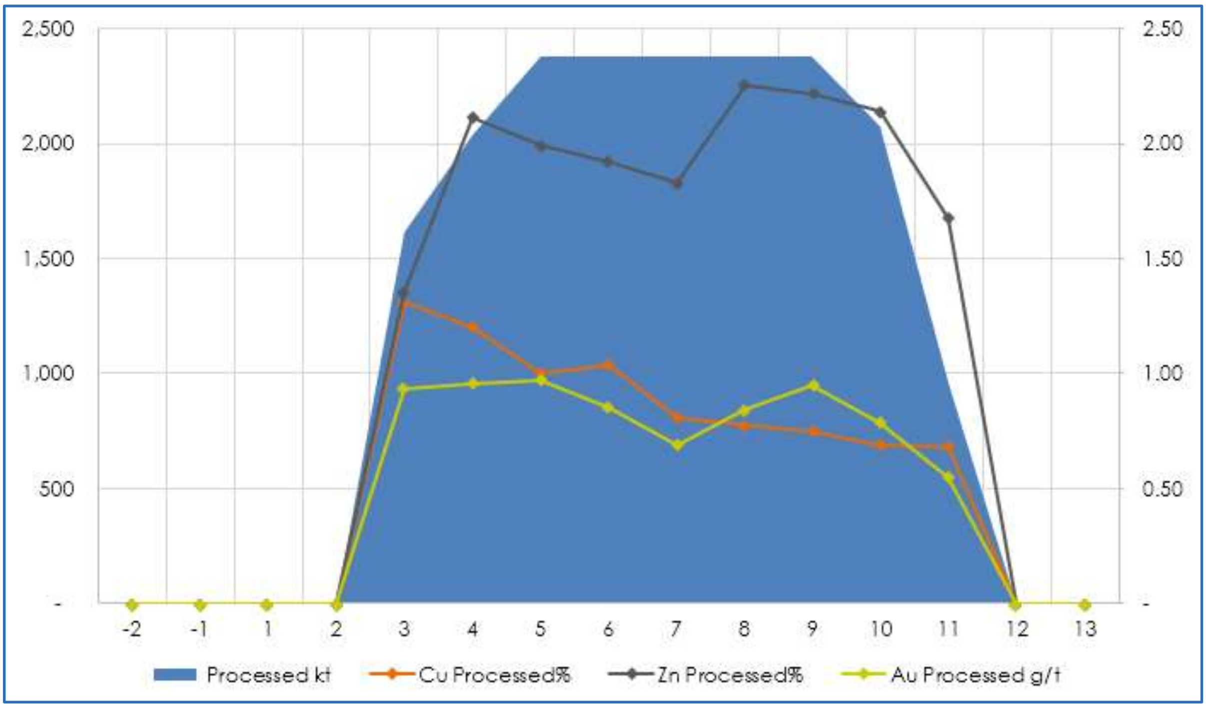

Figure by OreWin, 2019.

1.8 Recovery Methods

The oxide processing facility has been designed to treat 1.1 Mt per annum of oxide ore for approximately two years and will be followed by processing 2.4 Mt per annum of sulphide ore over a total mine life of approximately 11 years. The project will therefore be installed and commissioned in two stages:

- Stage 1 oxide ore – comprising a two-year period for processing gold and silver ore that will be treated in a single stage semi-autogenous grinding (SAG) mill circuit, followed by sodium cyanide leaching, carbon-in-pulp (CIP), and elution and electrowinning techniques to recover the gold and silver; and,
- Stage 2 sulphide ore – the oxide processing plant will be expanded to process copper and zinc-bearing ore by flotation. A 5.5 MW secondary grinding ball mill will be added to the grinding circuit. Sequential flotation will be employed to produce separate copper and zinc concentrates for export.

The major unit operations of the oxide and sulphide process flowsheets have been tested at bench scale, along with specialist vendor testwork as required.

The proposed oxide ore flowsheet is presented in Figure 1.7 and the flowsheet for sulphide ore in Figure 1.8.

Figure 1.7 Flowsheet for Oxide Ore Processing

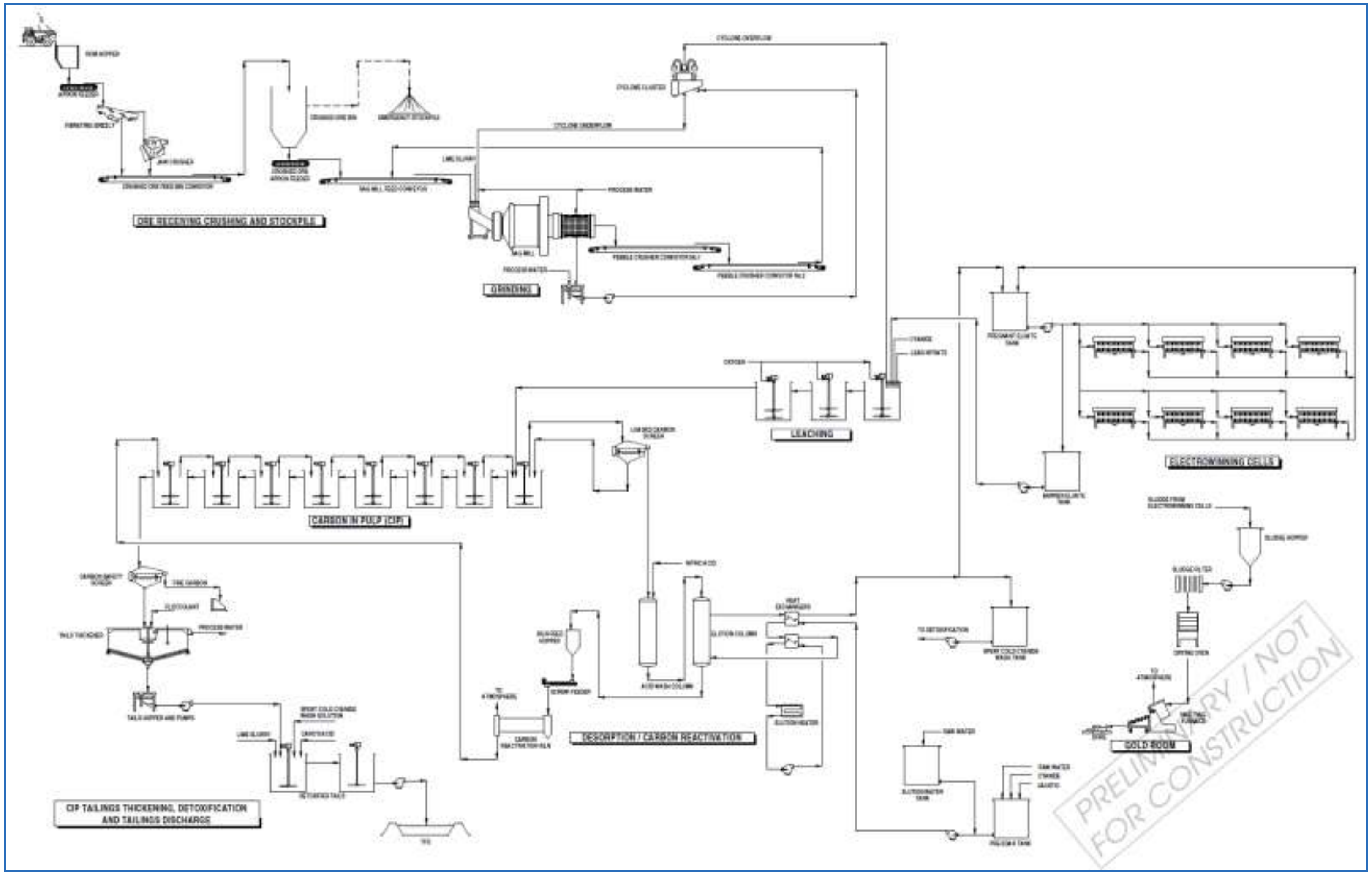

Figure from GRES, 2019.

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Figure 1.8 Flowsheet for Sulphide Ore Processing

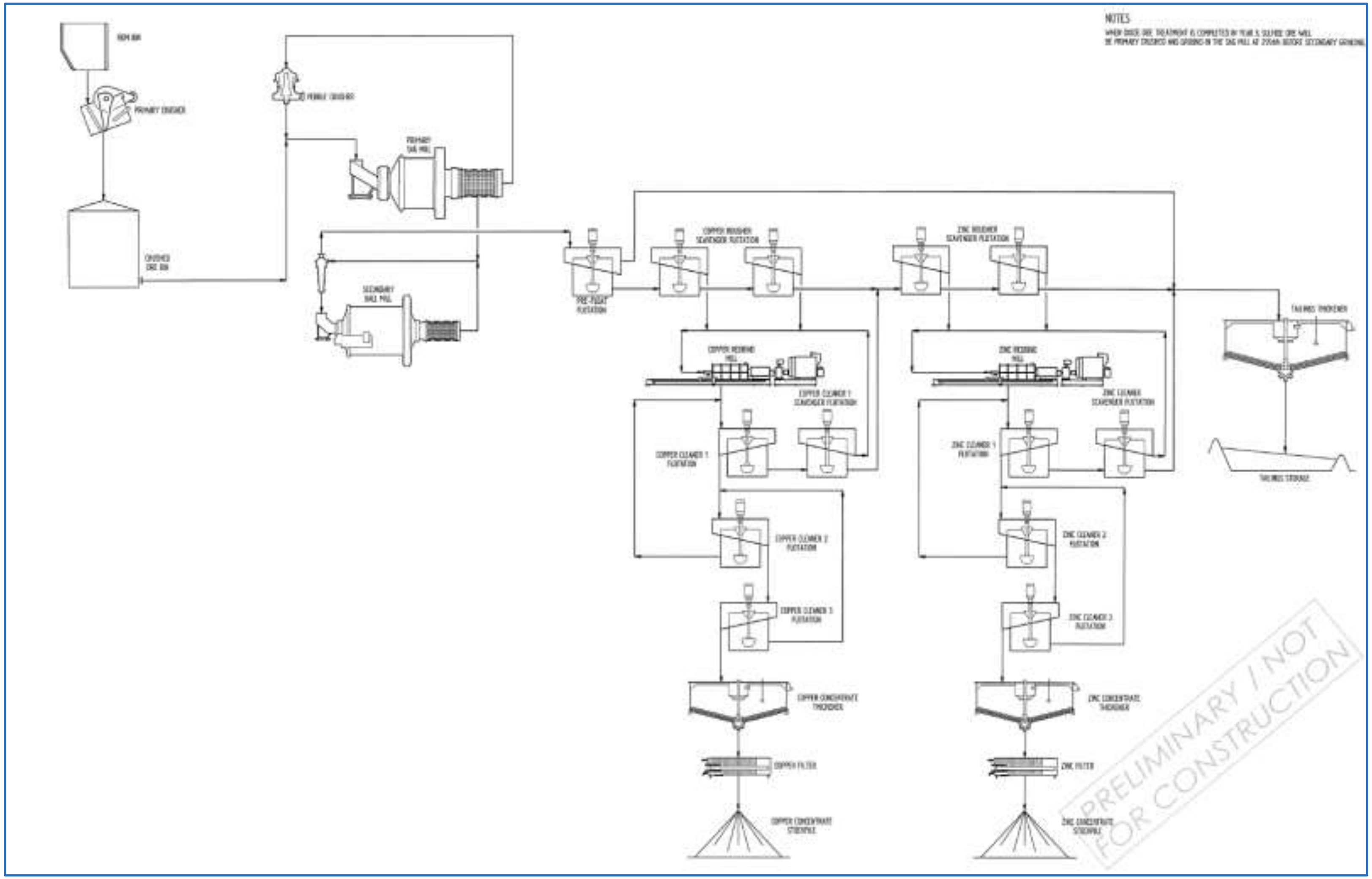


Figure from GRES, 2019.

1.9 Site Infrastructure

Buildings and facilities planned to be constructed adjacent to the process plants include workshop and warehouse, changeroom, security gate house, mine administration building, kitchen and dry mess, laboratory, fuel storage, control room, and other dedicated structures.

In addition, the following infrastructure will be located in the general mine area to support the project:

- Waste dumps (PAG and NPAG) and ROM stockpile area
- Topsoil stockpile areas
- Tailings Storage Facility for oxide and sulphide tailings disposal
- Clean water pond
- Water diversion structures
- 154 kVa power transmission line
- Operations personnel camp and facilities
- Mining contractor area

Proposed off-site infrastructure includes covered concentrate storage and blending bays at the selected export facility.

1.10 Financial Results

In PFS19, mining is planned to use a conventional open pit mining method using excavators and trucks. Two main types of ore will be mined and processed: oxide ore to recover gold and silver, and sulphide ore to recover copper, zinc, gold, and silver. The proposed oxide treatment rate is 1.1 Mtpa in a carbon-in-pulp (CIP) plant. The sulphide treatment rate is 2.4 Mtpa, processing the polymetallic sulphide ore in a concentrator to produce separate copper and zinc concentrates. The life of the project is approximately 11 years. A summary of the results is shown in Table 1.5.

The base case economic analysis returns an after-tax Net Present Value (NPV), at an 8% discount rate, of US$186M. It has an after-tax Internal Rate of Return (IRR) of 27% and a payback period of 4.1 years. The analysis calculates annual cash flows over the life of the mine and incorporates Turkish taxes, permit and license fees, and government royalties on metal sales.

The analysis is based on 2018 fourth quarter US Dollars and a Turkish Lira-to-US Dollar exchange rate of 6.0.

Financial results are summarised in Table 1.6. Table 1.7 summarises life-of-mine production, processing and concentrate quantities. Life-of-mine metal production is summarised in Table 1.8. Figure 1.9 shows the undiscounted after-tax cash flow modelled for the project.

Figure 1.9 Undiscounted After-Tax Cash Flow

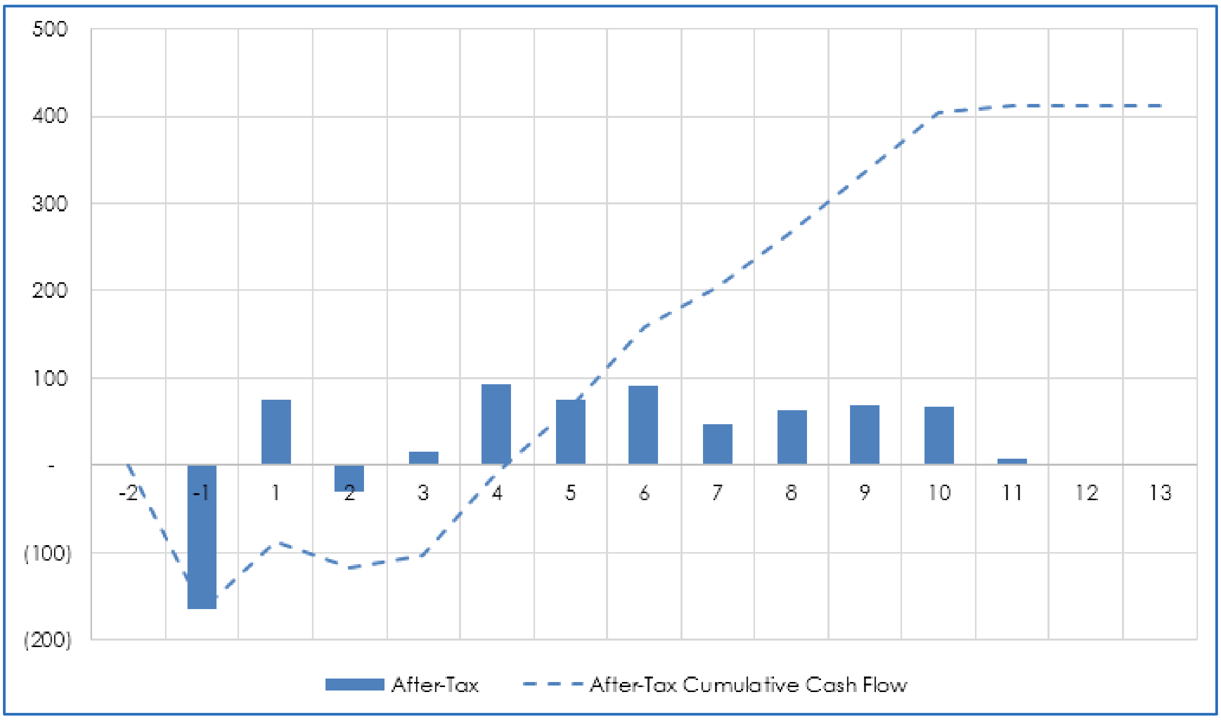


Figure by OreWin, 2019.

Table 1.5 PFS19 Results Summary

Metric	Unit	Value
Ore Mined	kt	21,335
Waste Mined	kt	169,206
Total Movement	kt	190,541
Stripping Ratio	waste:ore	7.9
Oxide Ore	kt	2,755
Oxide Grade – Au	g/t	2.34
Oxide Grade – Ag	g/t	56.7
Sulphide Ore	kt	18,580
Sulphide Grade – Cu	%	0.92
Sulphide Grade – Zn	%	1.98
Sulphide Grade – Au	g/t	0.85
Sulphide Grade – Ag	g/t	31.8
Copper Concentrate	kt	387
Zinc Concentrate	kt	503
Total Gold	koz	345
Total Silver	koz	8,148
Copper in Concentrate	kt	115
Zinc in Concentrate	kt	284
Before-Tax Undiscounted Cash Flow	US$M	420.4
Before-Tax NPV at 8% Discount Rate	US$M	191.0
Before-Tax IRR	%	27%
After-Tax Undiscounted Cash Flow	US$M	412.0
After-Tax NPV at 8% Discount Rate	US$M	186.1
After-Tax IRR	%	27%
Project Payback	years	4.1
Initial Capital (incl. contingency)	US$M	164.1
Operating Cost		
Mine	$/t ore	14.54
Oxide Process	$/t ore	20.85
Sulphide Process	$/t ore	19.88
Administration	$/t ore	5.07
Overall Operating Cost	$/t ore	39.62

OreWin

Table 1.6 Financial Results

	NPV	
	Before-Tax	After-Tax
	US$M	US$M
Undiscounted	420.4	412.0
5%	258.4	252.5
8%	**191.0**	**186.1**
10%	154.8	150.5
15%	86.8	83.5
IRR	27%	27%
Peak Funding	–164.1	
Payback (Years)	4.09	4.12

Table 1.7 Life-of-Mine Production and Processing Quantities

Life-of-Mine Production	Unit	Quantity
Oxide Ore	kt	2,755
Oxide Grade – Au	g/t	2.34
Oxide Grade – Ag	g/t	56.7
Sulphide Ore	kt	18,580
Sulphide Grade – Cu	%	0.92
Sulphide Grade – Zn	%	1.98
Sulphide Grade – Au	g/t	0.85
Sulphide Grade – Ag	g/t	31.8
Weathered Waste	kt	26,449
Fresh Waste	kt	142,757
Total Material	kt	190,541
Copper Concentrate	kt	387
Zinc Concentrate	kt	503

Table 1.8 Life-of-Mine Metal Production

Life-of-Mine Production	Unit	Quantity
Copper in Concentrate	**kt**	**115**
Zinc in Concentrate	**kt**	**284**
Gold		
Oxide	koz	187
Copper Concentrate	koz	128
Zinc Concentrate	koz	31
Total Gold	**koz**	**345**
Silver		
Oxide	koz	3,547
Copper Concentrate	koz	2,329
Zinc Concentrate	koz	2,272
Total Silver	**koz**	**8,148**

A summary of total project initial and deferred capital costs is shown in Table 1.9.

Table 1.9 Summary of Project Capital Costs

Capital Costs	Initial	Expansion	Sustaining	Total
	US$M			
Plant	44.4	53.2	2.9	100.5
Infrastructure	53.8	––	21.8	75.6
Closure	––	––	22.7	22.7
EPCM	9.4	9.0	––	18.4
Owner's EPCM Management Team	9.4	4.5	––	13.9
Pre-Production Mining	25.9	-	––	25.9
Contingency	21.2	3.8	9.5	34.5
Capital Costs	164.1	70.6	56.9	291.6

Table 1.10 shows the breakdown of estimated life-of-mine project operating costs.

Table 1.10 Project Operating Costs

	Total (US$M)	Breakdown Unit	Unit Cost (US$)
Mine			
Owner's Staff	40.2	$/t total moved	0.21
Mining Cost	270.0	$/t total moved	1.42
Mine	310.2	$/t total moved	1.63
Process			
Oxide Direct Cost	57.4	$/t ore Oxide	20.85
Sulphide Mill Direct Cost	369.3	$/t ore Sulphide	19.88
Process	426.8	$/t ore	20.08
Administration			
Sitewide G&A	43.8	$/t ore	2.06
Site Camp Costs	41.4	$/t ore	1.94
Land Usage / Forestry Fees	22.4	$/t ore	1.05
License and Compliance Fees	0.6	$/t ore	0.03
Administration	108.3	$/t ore	5.07
Overall Operating Cost	**845.2**	**$/t ore**	**39.62**

1.11 Conclusions and Recommendations

Feasibility Study

PFS19 is at a prefeasibility level of accuracy. It has identified a positive business case and it is recommended that the assessment of the Gediktepe project be continued to a feasibility study level in order to increase the confidence of the estimates.

There are a number of areas that need to be further examined and studied and arrangements that need to be put in place to advance the development of the Gediktepe project. The key areas for further work are as follows.

Mineral Resources

The resource classification categories assigned to the Gediktepe estimates (Measured, Indicated, and Inferred) have, at a global scale, identified different levels of confidence (uncertainty) across the deposit, and this is considered sufficient for prefeasibility assessment. However, these categories do not necessarily reflect variations in confidence at a more-local resolution, which may impact on the shorter term effectiveness, and hence profitability, of eventual mining.

It is recommended that additional work be undertaken in an effort to reduce this uncertainty. This may involve:

- Additional, focussed drilling.
- A short-range variability study to attempt to better understand the grade distributions.
- Selected resampling and assaying.
- Review of local geological and mineralogical interpretations.
- Refinement of resource modelling and grade estimation procedures.

The uncertainty in the mineralogical interpretations may necessitate that sampling for grade control be close-spaced and of a high degree of accuracy. A detailed plan in regard to grade control measures is required. To arrive at the most appropriate grade control strategy, studies into the accuracy and practicality of the various available measures should be undertaken, including, but not limited to, blasthole sampling, RC drillhole sampling, trenching, grab sampling, and portable XRF sampling, as well as methods for obtaining accurate and meaningful mapping data from already-mined benches. The feedback of this information into the grade control model in a timely and accurate way will be very important to ensure that knowledge in regard to the tenor and type of mineralisation that is due to be imminently exposed is available in a usable form when required.

Mining

The following mining work is recommended to be carried out for the feasibility study:

- Update and revise the open pit and waste dump designs based on updated process parameters from additional testwork recommendations.
- Prepare detailed designs and schedules for the waste dumps, including the PAG dump. Detailed specifications for the PAG dump should be prepared for the dump design, management, and closure.
- Investigate the possibility of encapsulating the PAG within cells in the main waste dump.
- Obtain updated mining contractor budget pricing based on the final feasibility study mine plan and schedules.

Process and Metallurgical Testwork

The following metallurgical testwork is recommended to be carried out for the feasibility study:

<u>Oxide samples</u>

- Variability testing of samples with a range of precious metal head grade, cyanide-soluble (CN^{sol}) copper content, silver-to-gold ratios, spatial and depth locations, and mine schedule composites.
- Investigation of acid washing and elution conditions for removal of copper and zinc, and recovery of gold and silver from loaded carbon.
- Effect of low temperature (climate) on leach extractions and adsorption efficiency.
- Optimisation of leach conditions (cyanide concentration, pulp density, and dissolved oxygen levels).

Sulphide samples

- Variability testing of samples from each ore type with a range of head grade, copper-to-zinc ratios, lead content, spatial and depth locations, and mine schedule composites.
- Investigate the influence of copper-to-zinc ratio on the behaviour of the enriched ore and blends of enriched ore with other sulphide ore types.
- Assess the impact of increased production of complex concentrate by treatment of higher proportions of enriched material and develop a strategy for concentrate blending.
- Process water treatment parameters for removal of residual reagent using activated carbon.

Infrastructure

The following infrastructure work is recommended to be carried out for the feasibility study:

- Optimise surface infrastructure layout.
- Prepare detailed closure planning and costing.
- Complete an assessment of road usage and travel arrangements for workforce access to site using a drive-in / drive-out strategy compared to provision of an on-site camp.
- Prepare a detailed project implementation schedule to cover all the activities from pre-production of the oxide plant through to the post-commissioning period of the sulphide plant.

2 INTRODUCTION

This Technical Report has been prepared for Alacer Gold Corp. (Alacer) by OreWin Pty Ltd (OreWin) in the first quarter of 2019. The document is in accordance with the guidelines provided in NI 43-101 and conforms to Form 43-101 F1 for Technical Reports.

The Gediktepe project investigations are being managed by Polimetal Madencilik San. ve Tic. A.S. (Polimetal), a 50/50 joint venture between Alacer and Lidya Madencilik San. ve Tic. A.Ş. (Lidya).

In 2017, after the completion of the 2016 prefeasibility study, titled 'Technical Report, Prefeasibility Study Gediktepe Project, Balikesir Province, Turkey', with an effective date of 1 June 2016 (PFS16), Polimetal appointed a team of consultants to carry out further feasibility assessment of the Gediktepe project. Polimetal is continuing the feasibility study work on the Gediktepe project.

This PFS19 report documents the outcomes of technical investigations by Polimetal as at the end of 2018.

The consultants appointed by Polimetal for PFS19 are detailed in Table 2.1. These consultants, directed by Polimetal, are the primary sources of the technical information compiled in the Technical Report.

Table 2.1 Consultants Appointed by Polimetal

Consultant	Abbrev.	Study Work Completed
AMC Consultants Pty Ltd	AMC	Geology, Mineral Resource, Mineral Reserve, and mine planning
GR Engineering Services	GRES	Process and infrastructure, and tailings and clean water pond peer review with CMW Geosciences
Hacettepe Mineral Technologies	HMT	Metallurgical testwork
Golder Associates	Golder	Mine and waste dump geotechnical
ENSU Engineering and Consulting Co. Ltd	ENSU	Tailings storage facility and clean water pond design
SRK Consulting	SRK	Mine water management, environmental and social impact assessment (ESIA), EIA update, and waste rock management

Information regarding the Qualified Persons who contributed to the Technical Report is as follows and summarised in Table 2.2:

- Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining, was responsible for the overall preparation of PFS19 and, the Mineral Reserve estimates. Bernard Peters visited the site on 15 January 2019. The site visit included briefings from Polimetal engineering, mining, and geology and exploration personnel. The visit included inspection of drill core, and site inspection of the mining and plant sites. Meetings with Polimetal and Alacer personnel were held at their respective offices in Ankara, Turkey during the week of the site visit. Bernard Peters has had a review role on the Gediktepe study work since 2018 and participated in reviews and meetings with Polimetal, Alacer, and their consultants in Perth 17–19 January 2018.

Bernard Peters was responsible for the overall report preparation, plus the mining and Mineral Reserve estimates in Sections: 1.1, 1.2, 1.6, 1.7, 1.10, 1.11; 2; 3; 4; 5; 15; 16; 19; 20; 21.1, 21.2.13, 21.3.1, 21.3.3; 22; 23; 24, 25; 26, 26.2; and 27.

- Sharron Sylvester, BSc (Geology), MAIG, RPGeo (10125), employed by OreWin Pty Ltd as Technical Director – Geology, was responsible for the preparation of the Mineral Resources. Sharron visited the site on 15 January 2019. The site visit included briefings from Polimetal engineering, mining, and geology and exploration personnel. The visit included inspection of drill core, and site inspection of the mining and plant sites. Meetings with Polimetal and Alacer personnel were held at their respective offices in Ankara, Turkey during the week of the site visit. Sharron Sylvester has had a review role on the Gediktepe study work since 2017 and participated in reviews and meetings with Polimetal, Alacer, and their consultants in Tucson 22–30 October 2017 and Perth 16–17 January 2018.

  Sharron Sylvester was responsible for Mineral Resources in Sections: 1.3 to 1.5, 1.11; 2; 3; 6 to 12; 14; 25; 26.1; and 27.

- Peter Allen, BEng (Metallurgy), MAusIMM (CP 103637), employed by GR Engineering Services as Manager – Technical Services, was responsible for process plant and infrastructure and visited the site on 11 September 2017. The site visit included briefings from Polimetal project, mining, and geology and exploration personnel. The visit included inspection of drill core and site inspection of the potential and proposed plant and infrastructure sites. A visit was also made to a potential equipment vendor on 16 September 2017. Peter Allen participated in technical meetings with Polimetal and Alacer personnel in Ankara, Turkey, including with other consultants, from 12–15 September 2017, and in Ankara 4–6 July 2018, and Perth 17–19 January 2018. Peter Allen also participated in fortnightly project meetings through 2018 and 2019 and attended testwork conducted at ALS laboratories, Perth, Western Australia.

  Peter Allen was responsible for process plant and infrastructure in Sections: 1.8, 1.9, 1.11; 2; 3; 13; 17; 18; 21.2.1 to 21.2.12, 21.2.14, 21.3.2; 25; 26.3, 26.4; and 27.

Table 2.2 Qualified Persons and Sections Responsibilities

Name	Company	Qualifications	Site Visit Date	Sections
Bernard Peters	OreWin Pty Ltd	B.Eng (Mining)	15 Jan. 2019	1.1, 1.2, 1.6, 1.7, 1.10, 1.11; 2; 3; 4; 5; 15; 16; 19; 20; 21.1, 21.2.13, 21.3.1, 21.3.3; 22; 23; 24; 25; 26, 26.2; 27
Sharron Sylvester	OreWin Pty Ltd	B.Sc (Geology)	15 Jan. 2019	1.3 to 1.5, 1.11; 2; 3; 6 to 12; 14; 25; 26.1; 27
Peter Allen	GR Engineering Services	B.Eng (Metallurgy)	11 Sept. 2017	1.8, 1.9,1.11; 2; 3; 13; 17; 18; 21.2.1 to 21.2.12, 21.2.14, 21.3.2; 25; 26.3, 26.4; 27.

3 RELIANCE ON OTHER EXPERTS

The QPs, as authors of PFS19, have relied on, and believe there is a reasonable basis for this reliance, upon the following Other Expert reports as noted below. Individual QP responsibilities for the sections are listed on the Title Page.

3.1 Mineral Tenure

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the project area, underlying property agreements, or permits. The QPs have fully relied upon, and disclaim responsibility for, information derived from Polimetal for this information through the following documents:

- Report on Gediktepe titled Property Description and Location.

This information was used in Sections 1 and 4 of PFS19.

3.2 Surface Rights

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Alacer for information relating to payment of land and surface rights taxes and other payments through the following document:

- Email from Alacer to OreWin dated 22 February 2019.

3.3 Market Studies and Contracts

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Polimetal staff and experts retained by Polimetal for information relating to the status of the market studies and contracts for the project, as follows:

- Gediktepe Project Port Study Report, Polimetal, 24 August 2017.

This information was used in Section 19 of PFS19.

3.4 Environmental and Work Programme Permitting

The QPs have obtained information regarding the environmental and work programme permitting status of the project through opinions and data supplied by experts retained by Polimetal. The QPs have fully relied upon, and disclaim responsibility for, information derived from such experts through the following documents:

- Gediktepe Final Environmental Impact Assessment Report, March 2016.

This information was used in Section 20 of PFS19.

3.5 Taxation and Royalties

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Alacer staff and experts retained by Polimetal for information relating to the status of the current royalties and taxation regime for the project, as follows:

- Memorandum by Alacer Re: Gediktepe PFS – tax and royalty assumptions February 2019.
- Email from Alacer Re: Gediktepe 2019 PFS – Financial Model 16 February 2019.

This information was used in Section 22 of PFS19.

4 PROPERTY DESCRIPTION AND LOCATION

The Gediktepe project is located in the Balıkesir Province of Western Turkey, east–south-east of the Bigadiç township.

The project coordinates are:

Latitude and Longitude:	39°21'38.7"N	28°34'43.0"E
UTM European Zone 35 coordinates:	4,358,000N	636,000E

4.1 Project Ownership

The Operating Licenses (OL) for Gediktepe project are held by Polimetal.

Operating License – RN 85535

RN 85535 is the main OL for the Gediktepe project. The General Directorate of Mining and Petroleum Affairs (GDMPA) approved the merging of OL 20054077 and Exploration License (EL) 201400291 into one OL (RN 85535) on 29 July 2016. RN 85535 is valid until 23 June 2036. Figure 4.1 shows RN 85535.

Operating License – 20054077

On 1 July 2005, the Gediktepe EL was acquired from GDMPA by tender on behalf of Yeni Anadolu Mineral Madencilik San. Tic. Ltd. Sti. (YAMAS). The license area covered 657.87 ha. That EL was changed to an OL on 23 June 2011 and was valid for ten years. The OL was transferred to Polimetal from YAMAS on 26 July 2011.

An Environmental Impact Assessment (EIA) permit application was submitted, and the EIA Permit was granted on 14 March 2012. A Forest Permit was granted on 11 October 2013 and a Workplace Opening and Working Permit (GSM) was obtained on 24 October 2013.

After obtaining all the necessary permits, the operation permit was acquired on 30 December 2014.

Exploration License – 201400291

On 17 September 2014, the EL, which is on the east side of 20054077, was acquired by Polimetal from GDMPA by auction tender. The license area covered 829.12 ha.

On 21 February 2018, GDMPA also approved Polimetal's application for a production permit for clay and aggregate for three locations within RN 85535.

Operating License – 200700250

OL 200700250, which covers an area of 480.88 ha, was transferred from EL to the operational stage on 13 May 2014 by the previous owner, Hakki Musa Nogay. Polimetal purchased the OL from Hakki Musa Nogay during June of 2014. Transfer of the license to Polimetal was completed on 18 November 2015. No work has been completed on this license area, and it does not form part of PFS19.

Figure 4.1 Current Gediktepe Operating License

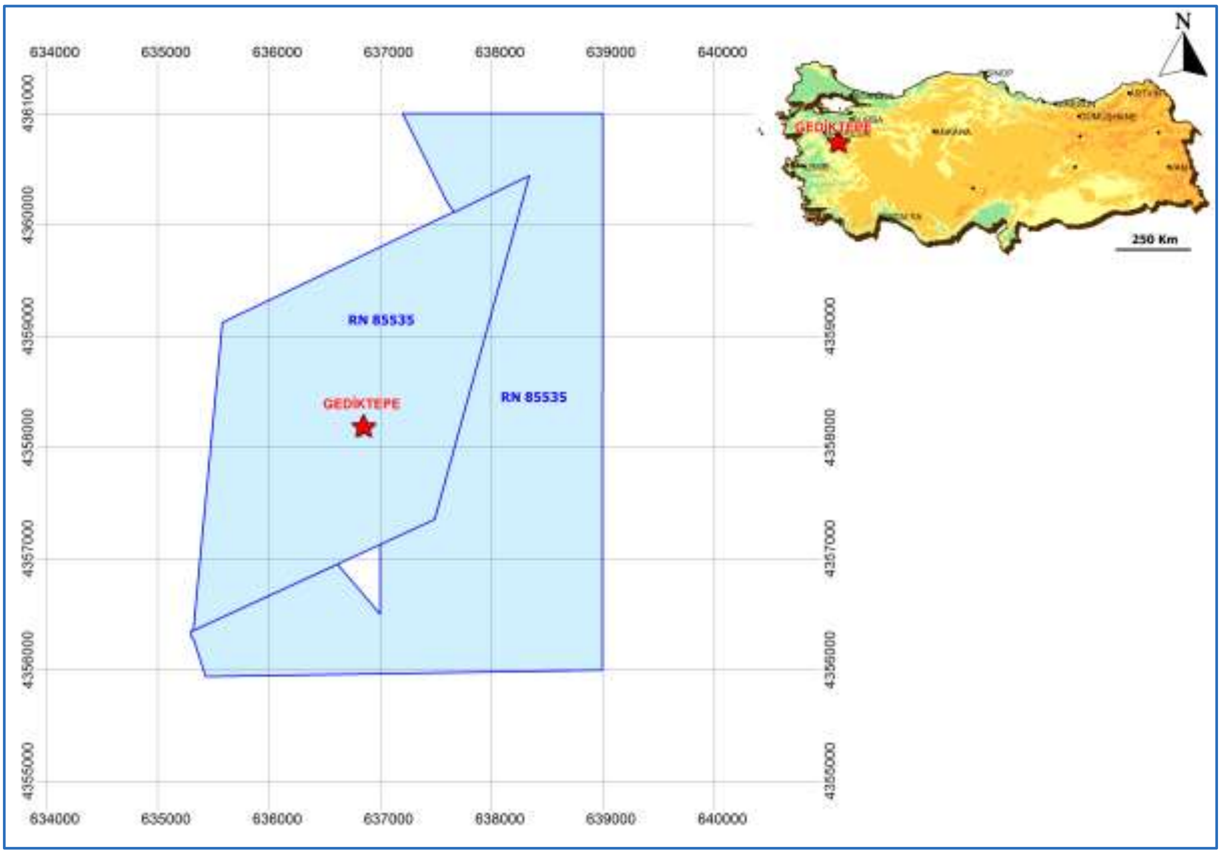


Figure from Polimetal, 2018.

4.2 Royalties or Encumbrances

Gediktepe mining licenses do not have any associated royalty to a third party other than the government royalty payment.

A forestry permit is required for any forest land that will be used in the project. To obtain the forestry permit, an application must be prepared by the forest engineer and should be submitted to the Regional Management of Forestry Department. Permit applications will be assessed and approved by the Operation Chief of Forestry Dept., Regional Management of Forestry Dept., General Management of Forestry Dept., and Prime Ministry, respectively.

The cost of obtaining a forestry permit depends on the location of the project, type of project (operating a mine, infrastructure or power line, etc.), type of forest and the quantum of trees.

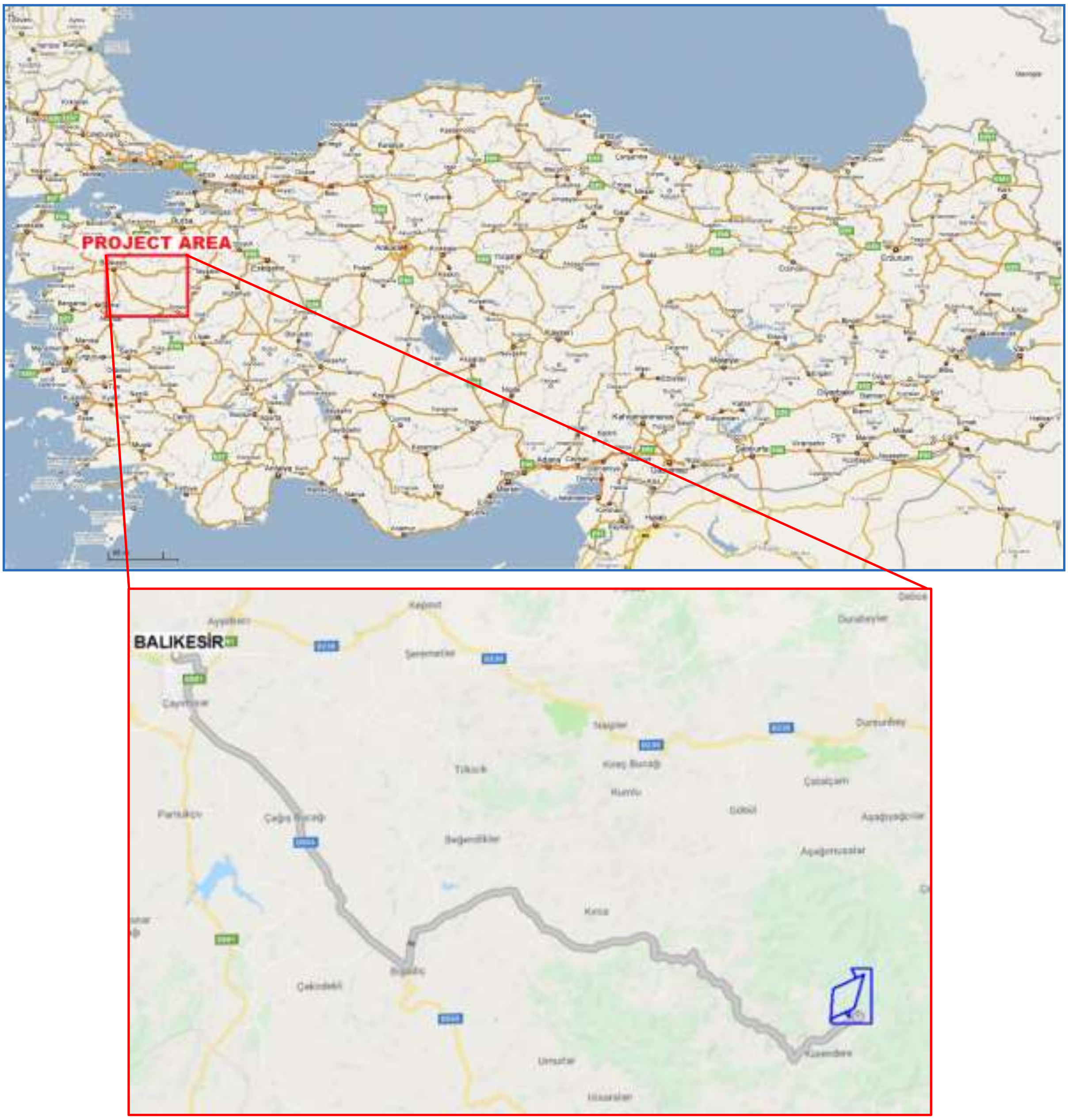
After obtaining approval, an agreement will be signed, and the forestry land permit fee will be paid every year until the end of the permit period, a one-time re-forestation fee and a deposit must also be paid. After reclamation of the used area, the deposit will be reimbursed.

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

5.1 Location and Access

The Gediktepe project is located in the Balıkesir Province of Western Turkey, some 67 km (air distance) south-east of Balıkesir Province centre and 38 km east–south-east of the Bigadiç township (Figure 5.1 and Figure 5.3).

Figure 5.1 Location Map



The Gediktepe project is accessed along 102 km of paved road from the city of Balıkesir (population 1,189,075) on paved highway D555 through the town of Bigadiç (population 48,470). The road from Bigadiç to the project site was recently widened and paved to be suitable for light and heavy vehicles (Figure 5.2). The road currently serves lumber trucks, concrete trucks, buses, and light vehicles. A 3.1 km by-pass road was constructed in 2017 to divert around the local Hacıömerderesi neighbourhood (Figure 5.3).

Figure 5.2 Paved Road

Figure from Polimetal, 2019.

The nearest airport, Balıkesir Koca Seyit Airport serving Balıkesir and Edremit, is approximately 185 km by road from site. There are also airline services to the nearest major city of Izmir, which is approximately 242 km by road via Bigadiç.

The project site is centrally located to access to several ports by road (distances are approximate from site via Bigadiç):

- Bandırma port is 194 km to the north
- Dikili port is 207 km to the west
- Aliağa port is 224 km to the west
- Izmir port is 242 km to the south-west

The nearest railway stations are in Dursunbey to the north and Balıkesir to the north-west.

The project land position consists of a single operating license number 85535, with a total area of 1,486.99 ha, of which 76% is forest area (see Figure 5.3).

The region covering the project area is classified as "1st Degree Earthquake Zone" according to the Seismic Zone Map of Turkey.

Figure 5.3 Gediktepe Project Area and By-pass Road

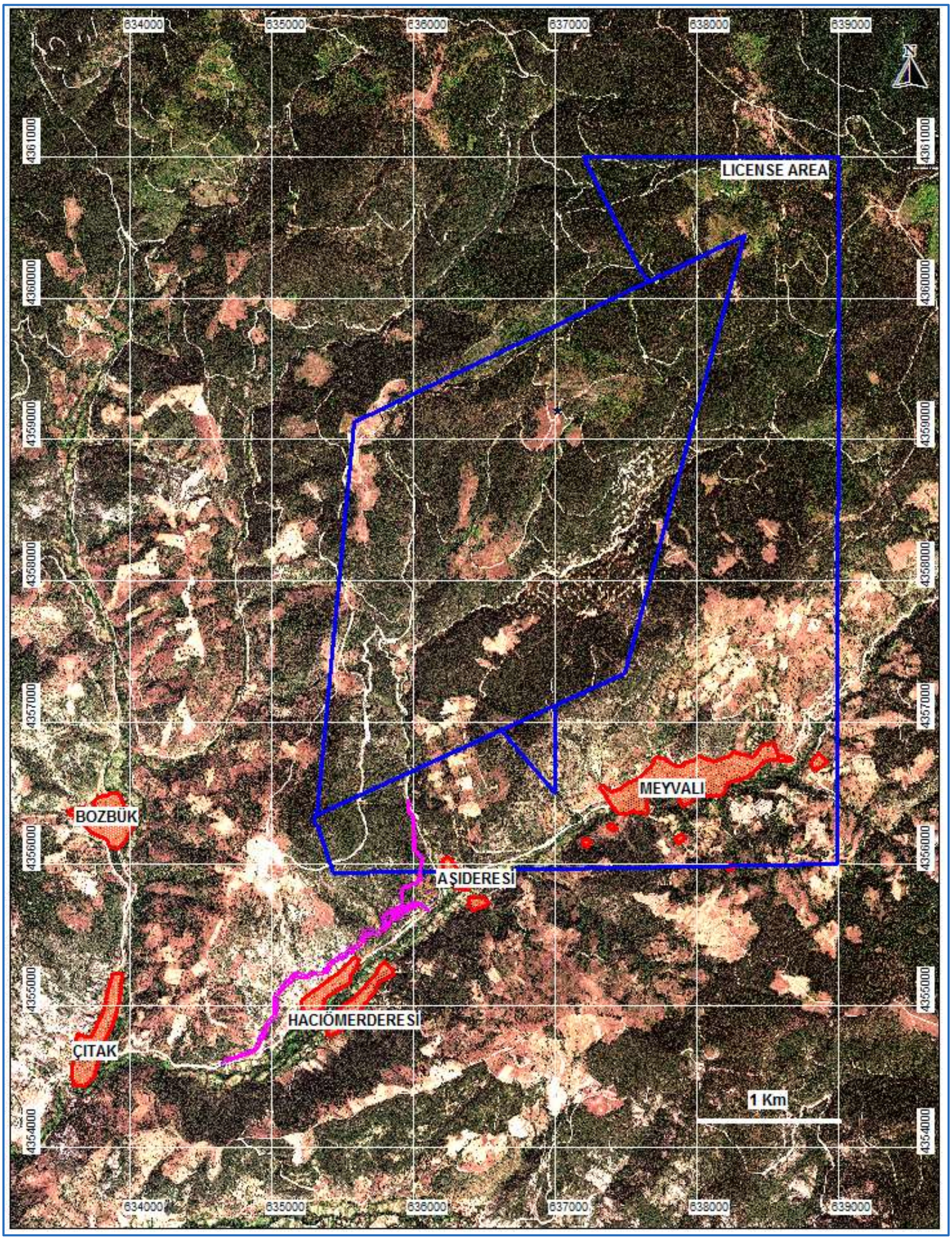


Figure from Polimetal, 2019. Pink colour shows the by-pass road, constructed in 2017.

5.2 Climate

Three climates are dominant in Balıkesir Province. The Mediterranean climate is seen in the Aegean coasts, the Marmara climate in the north, and the Continental climate in the inner regions. The temperature difference between summer and winter is small on the coastline. In the interior of the province, this difference is bigger. In the mountainous eastern region, winters are harsh, and summers are cool.

The local climate is hot and arid during the summer and warm during the fall. There is snow from December through February but not significant accumulation. Spring is often the rainy period. According to data from Dursunbey Meteorological Station for the years 1965–2014, the annual average temperature is 12.2°C. The highest measured temperature was recorded as 40.3°C in 2007 and the lowest temperature was recorded as –16°C in 1985.

The wind generally blows from the north or north-east.

Average evaporation from the Dursunbey Meteorological Station data is 943 mm per year with the highest average monthly evaporation of 190 mm experienced during July.

A meteorological station has been installed at site at the end of 2014 as part of the environmental base line data collection.

5.3 Local Resources

The closest settlements to the Gediktepe project site are:

- Hacıömerderesi neighbourhood,
- Aşıderesi neighbourhood, affiliated to Hacıömerderesi neighbourhood, and
- Meyvalı neighbourhood.

The main economic income sources in the area are forestry, agriculture, and animal husbandry.

The local area is serviced by a family doctor, who visits the neighbourhoods once per month, accompanied by a nurse and a midwife, when necessary. The closest hospital is the Bigadiç State Hospital, and there is a university hospital in Balıkesir Province.

A field camp was constructed at Kürendere, approximately 7 km (air distance) south-west of the project area and is currently partially in use. The field camp includes accommodation, kitchen, and social facilities (Figure 5.4).

There is an open pit borax mine in Bidagiç, operated by the State Enterprise, and an open pit gold mine in Sındırgı. Regionally, gold, silver, lead, copper, zinc, molybdenum, and chromite mines have operated for many years. The regional authorities and residents are familiar with co-existing with mining operations.

Figure 5.4 Aerial Photograph of Gediktepe Field Camp

Figure from Polimetal, 2019.

5.4 Infrastructure

A water supply will need to be established for the project as there is currently no developed system in the area capable of supporting a project of this size. For that reason, a clean water pond with 682,497 m^3 active reservoir capacity was engineered after signing the Protocol with DSI (State Water Works) and a village water supply pipeline was designed and approved by the Balıkesir Water Sewage Authority (BASKI). As per Environmental Impact Assessment (EIA) commitments, a water supply pipeline will be constructed before any site works will start (see Figure 5.5).

A 39.6 km-long 34.5 kV power transmission line (PTL) was constructed between Dursunbey substation and Kürendere to provide power to the project.

5.5 Physiography

The terrain at Gediktepe is mountainous with steep erosional valleys. Elevations in the project area range from 974–1,482 m above sea level (masl). Coniferous trees cover most of the project site, with occasional open meadows in areas of less-steep terrain.

Figure 5.5 shows the topography of the area.

Figure 5.5 Gediktepe Topography and Water Supply Route

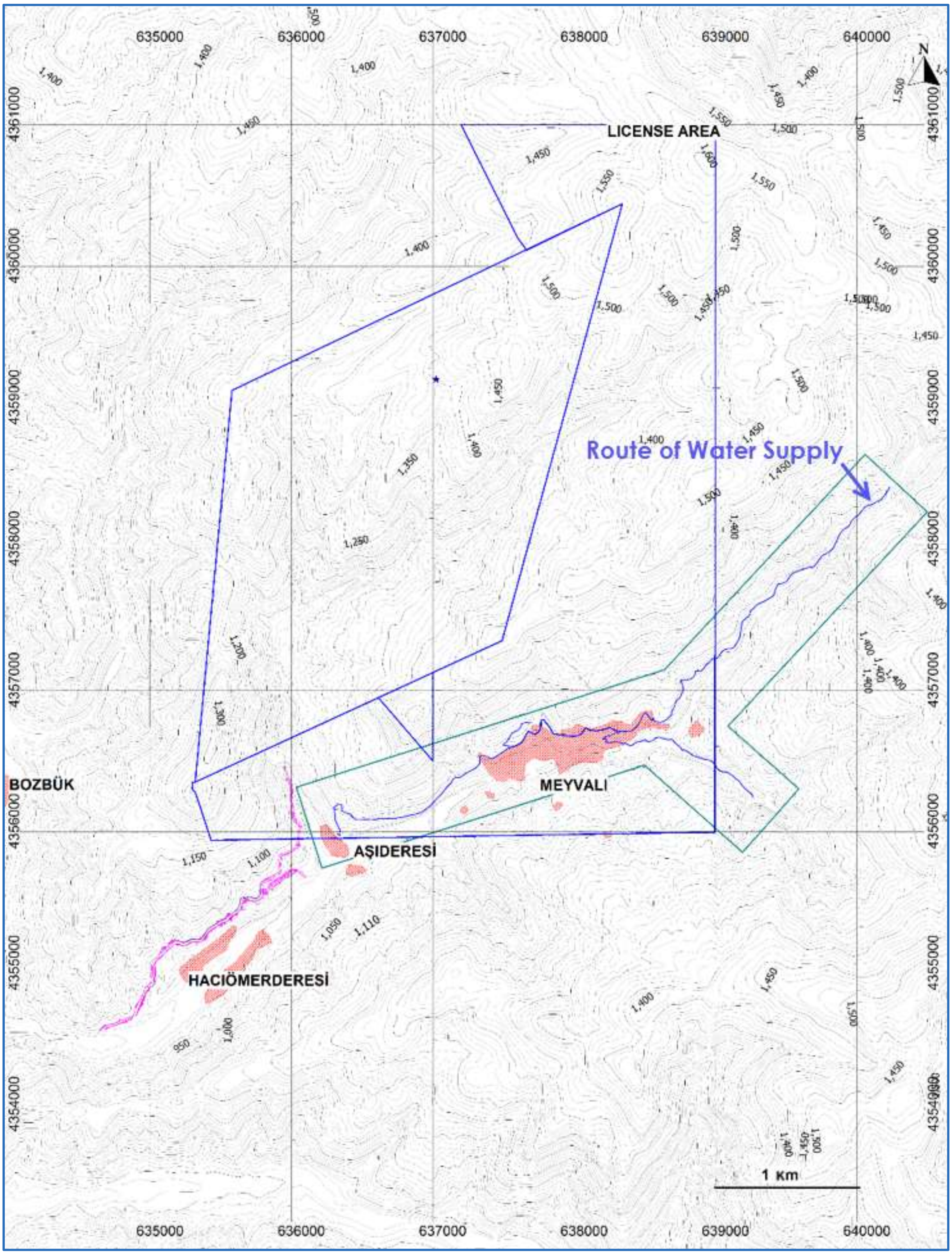


Figure from Polimetal, 2018.
Contoured at 10 m intervals.

6 HISTORY

Alacer obtained the first exploration license for the Gediktepe project in 2005. That license, number 20054077, constitutes the central area of the project.

Alacer completed geochemical stream sampling prior to 23 June 2011, at which time the license was transferred to Polimetal, the current joint venture operator.

Permit applications have been submitted at various times for site activities necessary to support technical investigations leading to the ultimate approval of a mining and mineral processing project.

Permit activity related to this license has included:

- An EIA Permit was obtained on 22 August 2012 for Phase 1 drilling that included 21 drill locations. The forestry permit for 11 drill locations was obtained on 17 March 2013. An EIA permit to undertake drilling at 234 locations was obtained on 14 March 2012 and 18 June 2013 and a forestry permit was obtained on 11 October 2013.
- For Phase 2 drilling, an EIA permit to undertake drilling at 139 locations was obtained on 18 December 2013 and 4 February 2014 and a forestry permit was obtained on 2 September 2014.
- For Phase 3 drilling, an EIA Permit to undertake drilling at 264 locations was obtained on 2 April 2014 and a forestry permit was obtained on 2 September 2014.
- In mid-2014, Polimetal commissioned a Preliminary Economic Assessment (PEA) of the project to determine economic potential. It identified a combined oxide and sulphide Indicated Mineral Resource of approximately 10Mt. Oxide processing was by heap leaching, while the subsequent sulphide processing was through a concentrator. The PEA did not identify a Mineral Reserve.
- Based on the positive PEA findings, Polimetal continued site investigations, including additional drilling aimed at increasing the size of the Mineral Resource.
- A Phase 4 Drilling EIA permit was obtained on 27 June 2014 for 344 drill locations, 175 of which received subsequent forestry approval.
- For the meteorological station, an EIA permit was obtained on 3 February 2014 and a forestry permit was obtained on 2 September 2014.
- Based on the PEA, a revised project operation was submitted to the General Directorate of Mining and Petroleum Affairs (GDMPA) on 25 September 2014 to enlarge the operation permit area and to change the annual production and processing capacity to as much as 2,375 kt of run of mine ore.
- An EIA Permit was obtained to undertake 242 drill and trench locations on 27 June 2014. The forestry permits for 17 drill and trench locations were received on 13 November 2015. Forestry permit approval of another 61 drill and trench locations planned for Stage 2 geotechnical investigations followed.
- An EIA application for oxide and sulphide mining and processing was submitted on 9 July 2015 and a public participation meeting was held on 11 August 2015. The EIA report was submitted to the Ministry of Environment and Urbanisation on 15 December 2015 and received a positive certificate on 1 July 2016.

During 2015, Polimetal commissioned a prefeasibility study (PFS16) on the project. This study used all drilling performed up to August 2015. PFS16, which was published in June 2016, identified a significant increase in combined oxide and sulphide Measured plus Indicated Mineral Resource to 36 Mt and, based on favourable technical and economic factors, identified a combined oxide and sulphide Mineral Reserve of 25 Mt and a potential mining and processing operation with a 12-year mine life.

A forestry permit for 157 drilling locations within the EIA boundary was received on 15 May 2017.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Gediktepe project is located within the Afyon tectonic zone, which is one of the main tectonic zones in Turkey. The Afyon Zone is a belt consisting of generally low-grade weathered metamorphic rocks. It is located between Menderes Massive to the west and the city of Denizli to the south (Figure 7.1).

Figure 7.1 Tectonic Map of Turkey

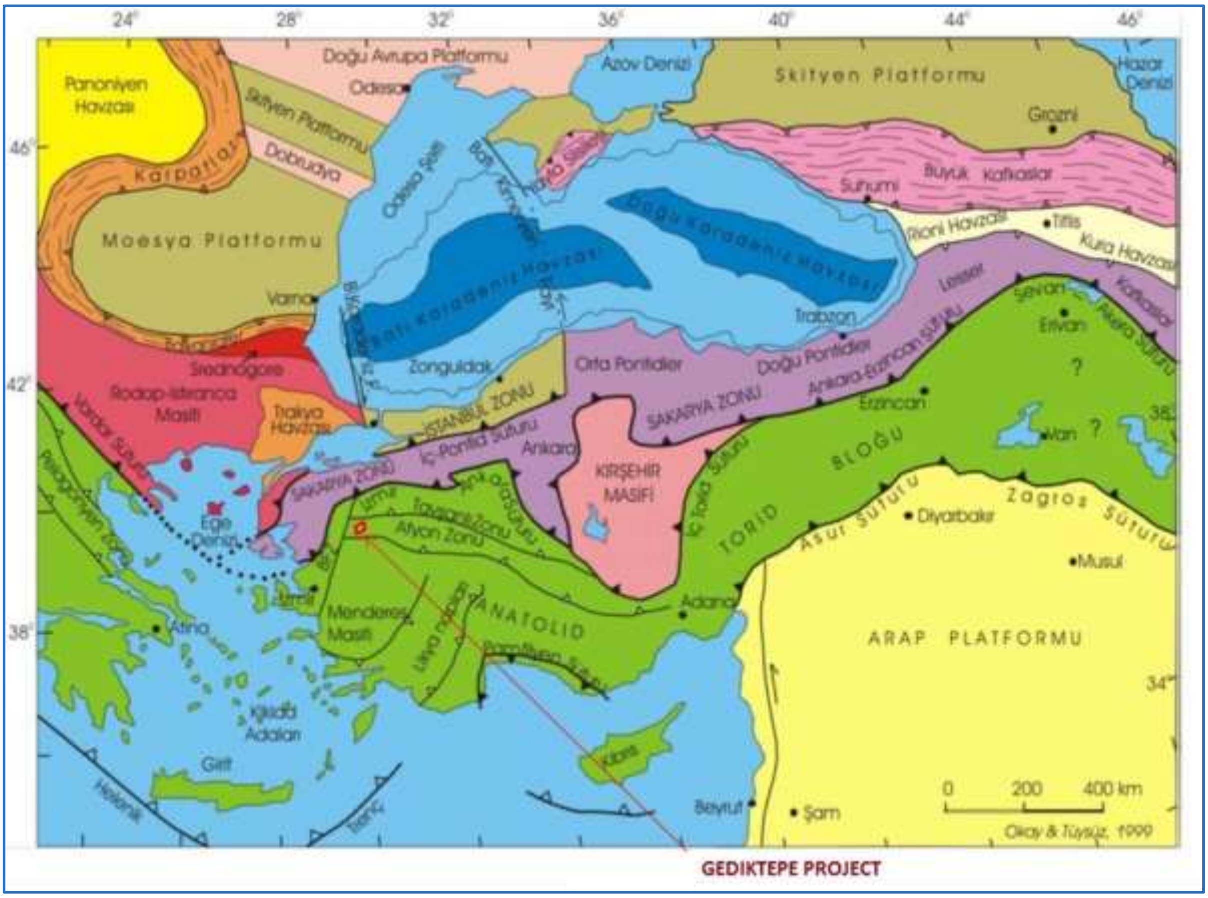


Figure from Polimetal, 2018, (after Okay and Tüysüz, 1999).

The Gediktepe regional geology comprises Upper Paleozoic metamorphics and Lower–Middle Miocene intrusives and volcanics.

The metamorphics are generally composed of gneiss, schist, mica schist, chlorite schist, phyllite, amphibolite, marble, and quartzite, with varying degrees of metamorphism. These metamorphics are stratigraphically overlain by Triassic carbonates and fragmental units, Jurassic limestone, and upper Cretaceous ophiolitic mélange. The upper Cretaceous ophiolitic mélange consists of flysch facies units, including olistostromal blocks and ophiolite sections. Grey coloured, recrystallised limestone olistolites and primary rock surrounds maroon–grey coloured sheared sandstone and shale.

Magmatic rock intrusions developed later between the Oligocene and Lower Miocene, due to extensional features in western Anatolia. Those intrusions cut the Paleozoic metamorphic and Upper Cretaceous ophiolitic rocks, establishing in the region what is now called the Alaçam Mountains granites, which outcrop in an arc-shaped geometry over an area of nearly 30 km^2 (Figure 7.2).

Figure 7.2 Gediktepe Regional Geology

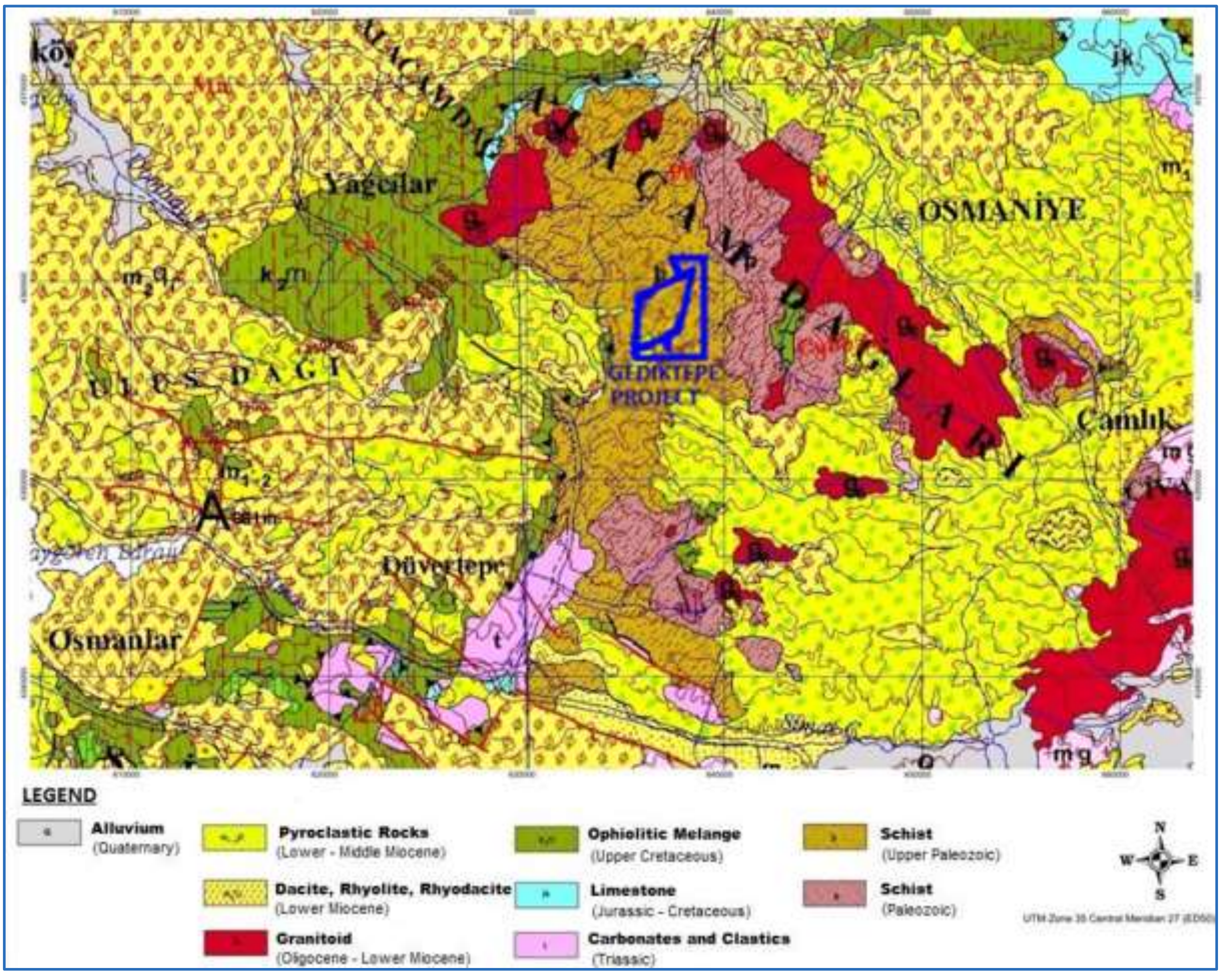


Figure from Polimetal, 2018, (after Okay and Tüysüz, 1999)

The Alaçam Mountain granites consist of granite porphyries and aplitic dykes, creating hornfelsic belts where they intruded Paleozoic metamorphic rocks. Skarn formations are abundant at the contacts of recrystallised limestone blocks of Upper Cretaceous ophiolitic mélange.

Lower Miocene volcanic rocks are positioned stratigraphically above Paleozoic to Upper Paleozoic metamorphics and Upper Cretaceous ophiolitic mélanges. Lower Miocene volcanic rocks comprise andesitic and dacitic intrusions, domes, lava flows, dykes, and volcanogenic sedimentary rocks.

Volcanic rocks, surrounding the Lower–Middle Miocene Alaçam Mountains, outcrop over an area of hundreds of square kilometres from the towns of Bigadiç to Simav, and from

Dursunbey to Düver Hill. The volcanic suite includes ignimbrite of felsic (dacite and rhyolite) composition. Ignimbrites have the widest distribution among felsic volcanic rocks, with thicknesses of up to 350 m to 400 m around the Alaçam Mountains. In some areas, these units are overlain by Pliocene terrestrial sediments and Quaternary alluvial deposits, sourced from the local metamorphics, ophiolitic mélange, granitoids, and felsic volcanic rocks.

Figure 7.3 represents a stratigraphic column of the Gediktepe project area. Mineralisation at Gediktepe is hosted in the Paleozoic units shown at the base of the column.

Figure 7.3 Regional Stratigraphic Column

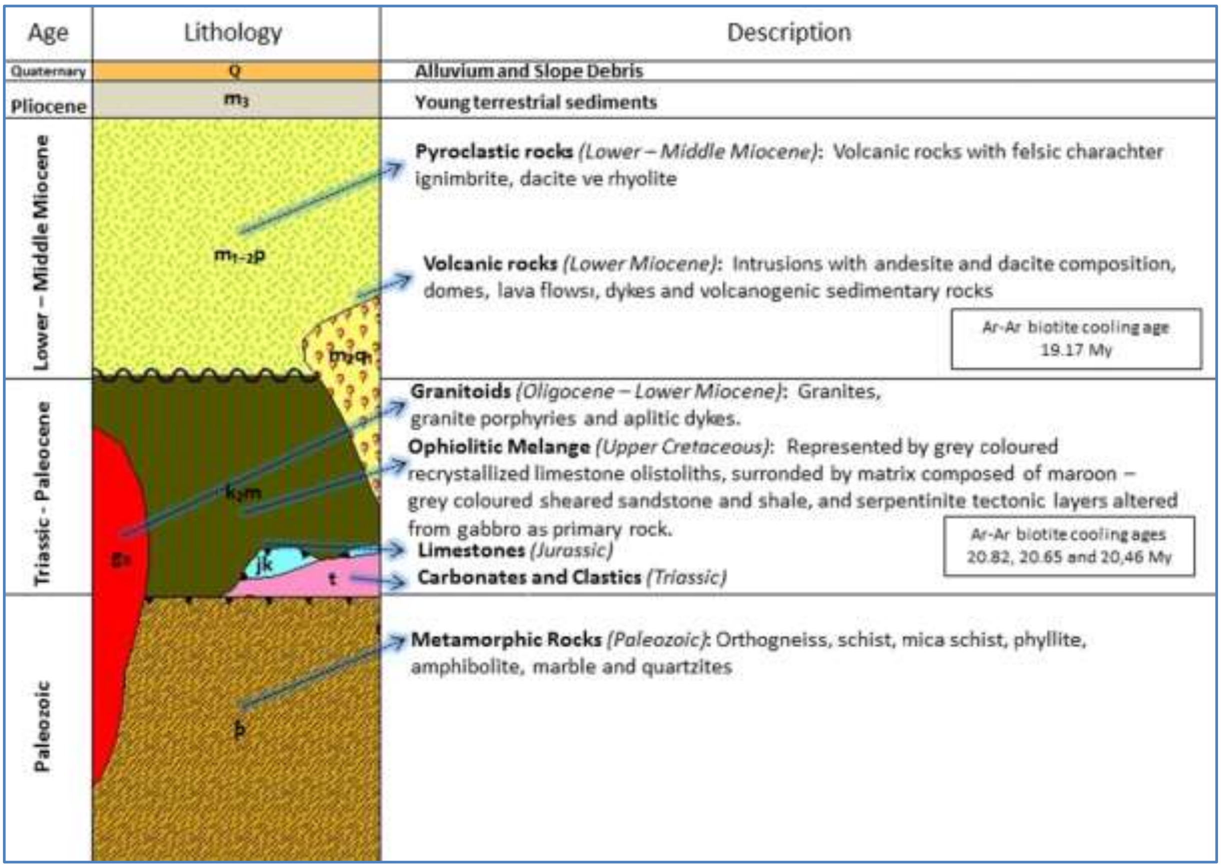


Figure from Polimetal, 2018, (after Okay and Tüysüz, 1999).

7.2 Deposit Geology

1:1,000 scale geological and structural mapping was conducted in the project area, followed up by 1:5,000 scale general mapping to outline the possible structures and alteration features (Figure 7.4).

Figure 7.4 1:1,000 Scale Project Geological Map

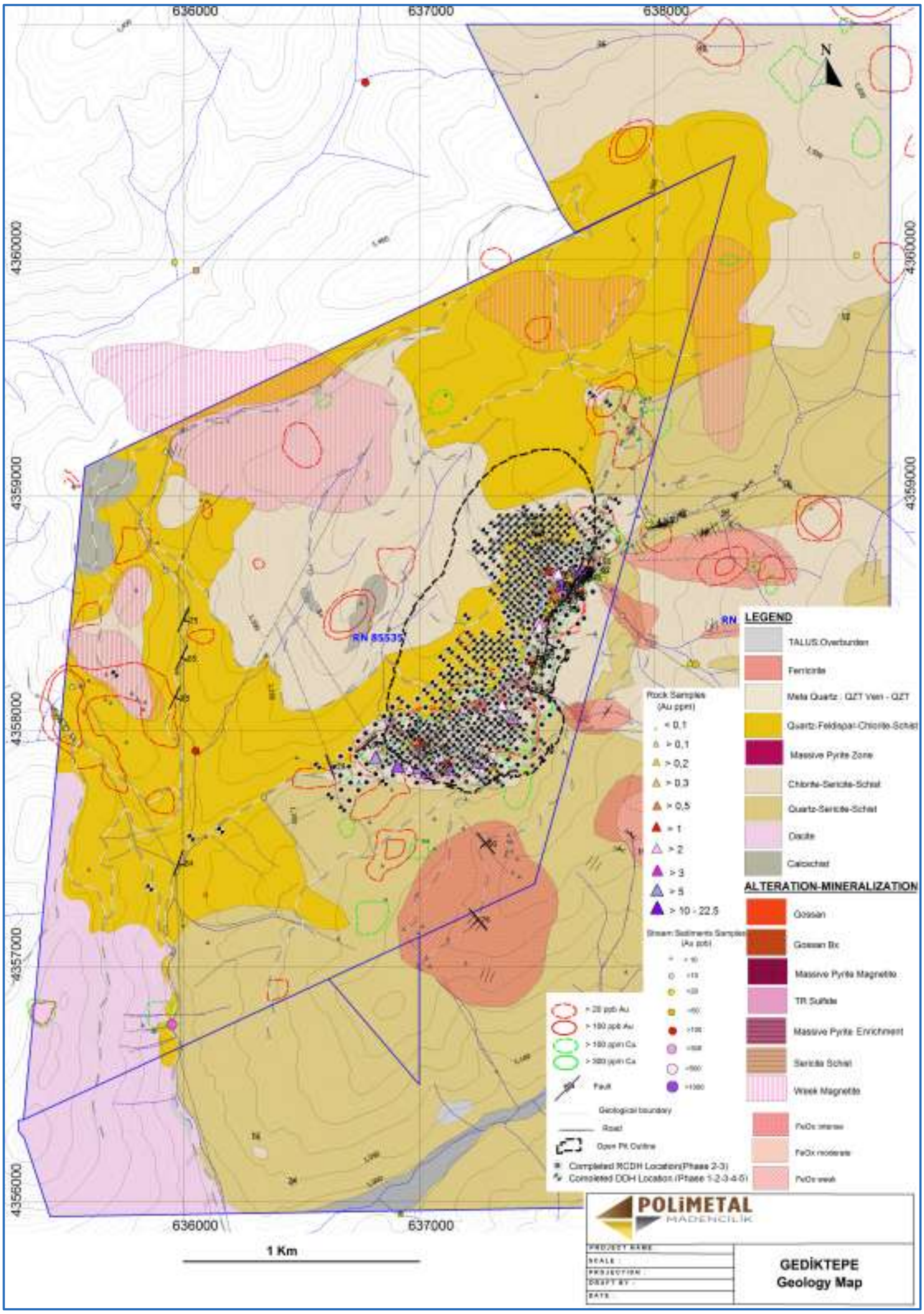


Figure from Polimetal, 2019.

Upper Paleozoic metamorphics are the most common units at Gediktepe, with the stratigraphic sequence, from top to bottom, being:

- Dacite and Pyroclastic
- Calcschist
- Feldspar–quartz schist
- Chlorite–sericite schist
- Quartz schist

The second-most common rocks at the project are the Lower–Middle Miocene volcanics, observable around Karadikmen Hill, south-west of Gediktepe. These comprise altered dacites–rhyodacites, characterised by lava flows and pyroclastics.

The youngest units at the project are mineralised gossan and ferricrete, along with talus, colluvium, and alluvium, being weathering products of the host rock.

7.2.1 Dacite and Pyroclastics (Lower–Middle Miocene)

The Dacites and Pyroclastics, of the Lower–Middle Miocene volcanics, are the second largest geological unit at the Gediktepe project.

The volcanics, located at southwest of Karadikmen Hill and Gaşakdoğrusu Hill, contain altered dacite to rhyodacite lava and pyroclastics (Figure 7.5a and Figure 7.5b). The units are grey to reddish colour, with a vuggy texture close to the surface, and traces showing flow directions. The vugs are filled with irregular-shaped quartz, and contain much higher amounts of feldspar, quartz, and biotite phenocrysts with depth (Figure 7.5c and Figure 7.5d).

Macroscopic features of the Dacite include porphyritic texture, large euhedral phenocrysts of clayey feldspar (orthoclase and plagioclase), quartz and biotite (very little chloritisation), cemented by feldspar, quartz, biotite, microlite, and crystallite, and volcanic glass. The matrix is intensely clay-altered and iron-oxidised. A hand specimen is shown in Figure 7.6.

Figure 7.5 Dacite Features *

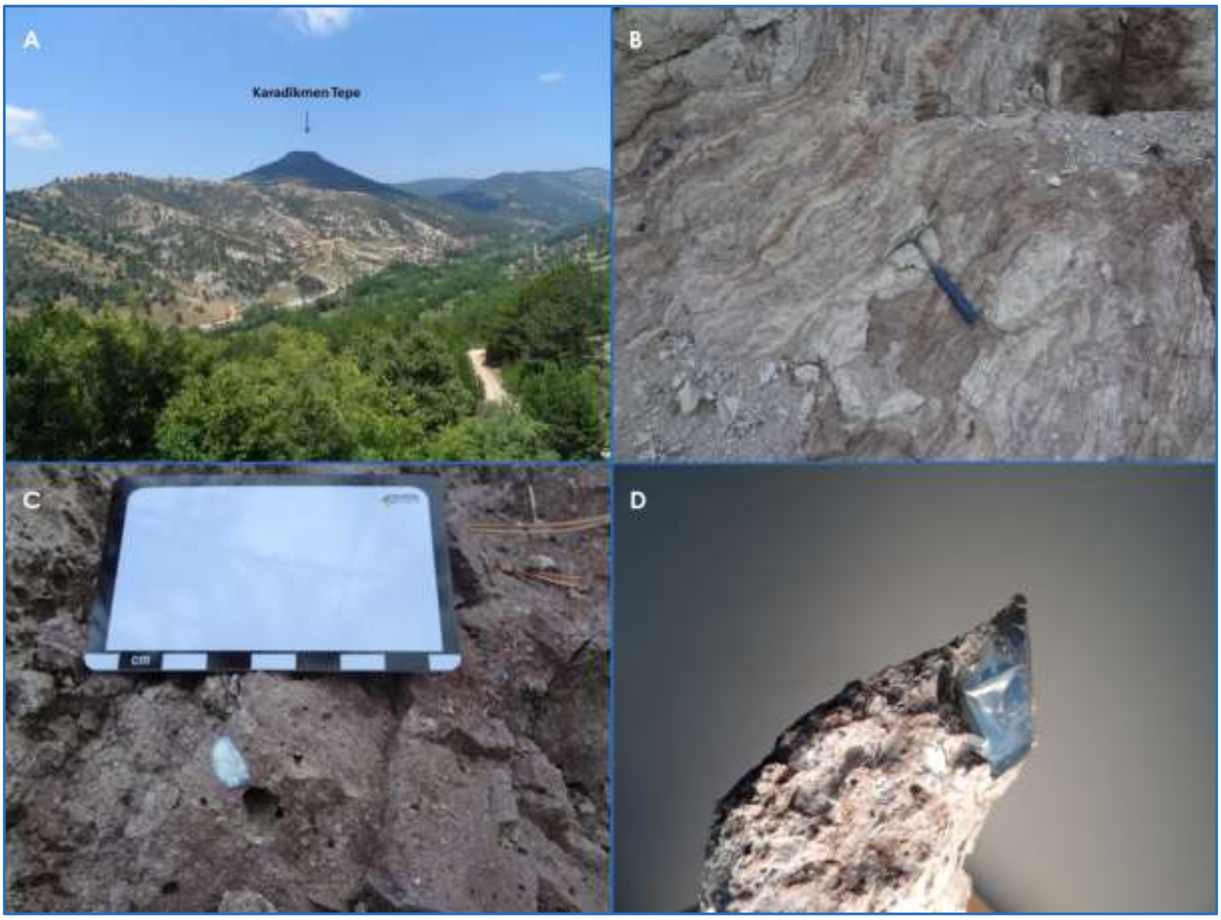


Figure from Polimetal, 2018, (* (A): Dacite dome Karadikmen Hill, (B): Flow structure in dacite, (C) and (D): Opal filling in voids)

Figure 7.6 Dacite Hand Specimen *

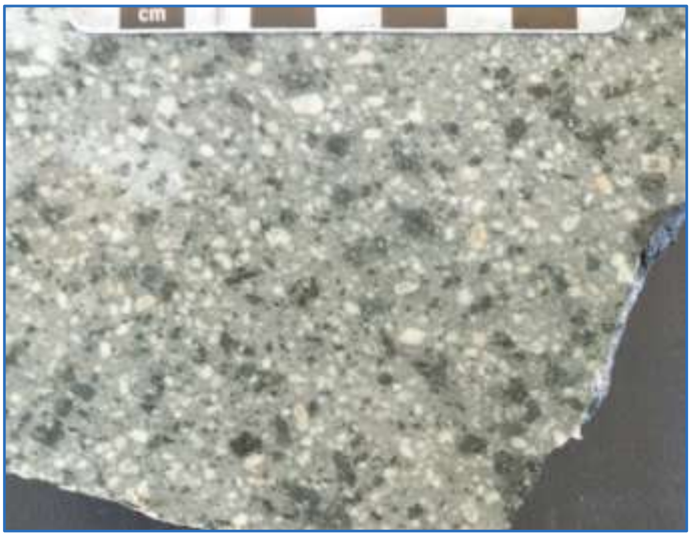


Figure from Polimetal, 2018.
(* Grey vitreous matrix incl. quartz, biotite, and feldspar phenocryst)

7.2.2 Calcschist (Upper Paleozoic)

The Calcschist is observed in outcrop at Küçük Yellice Hill and Fındıkalanı Ridge (Figure 7.7). It is beige to light grey in colour, has low hardness and schistosity, and is reactive to hydrochloric acid.

Figure 7.7 Calcschist in Outcrop *

Figure from Polimetal, 2018, (* Fındıkalanı Ridge).

7.2.3 Quartz–Feldspar Schist (Upper Paleozoic)

The quartz–feldspar schist is beige to light green in colour and is observable over a wide area at Gediktepe (Figure 7.8). It forms the primary unmineralised capping over the deposit, and generally contains virtually no sulphides. Macroscopically, it consists of 2–4 mm feldspar and quartz porphyroblasts and can be differentiated from other metamorphic rocks by its relatively weak schistosity. Chlorite and sericite minerals coating feldspar, and quartz porphyroblasts are other rock component minerals (Figure 7.9).

Thin section examination of the quartz–feldspar schist shows high quantities of feldspar minerals (orthoclase, plagioclase) and lesser quartz porphyroblasts. Porphyroblast fragments are composed of interlocked crystals and can reach up to 4–5 mm in size.

Figure 7.8 Quartz–Feldspar Schist in Outcrop *

Figure from Polimetal, 2018, (* around Büyük Yellice Hill and the Kaynarsu stream).

Figure 7.9 Quartz–Feldspar Schist Core Photograph

Figure from Polimetal, 2018.

7.2.4 Chlorite–Sericite Schist (Upper Paleozoic)

Chlorite–sericite schist is the main mineralisation host rock at Gediktepe, marked by gold and silver mineralisation in the oxide zone, and copper–zinc–lead with associated gold and silver in the sulphide zone.

The unit is observed in outcrop at Fındıkalanı Ridge, Çamdamı Ridge, Karaismailöldüğü, and north-west of Göğne Hill in the license area (Figure 7.10).

The colour of the chlorite–sericite schist varies between green and dark green due to mafic mineral banding. It has macroscopically strong schistosity (Figure 7.11). The orientation of the unit is generally 010–030° (north–north-east) with a dip of 20° to 40° to the north–north-west.

The rock composition, from lower to higher abundance, is: quartz, calcite, chlorite, and muscovite–sericite, with euhedral disseminated pyrite minerals observable in some cases. When disseminated pyrite in the chlorite–sericite schists exceed 10% to 45% by volume, the unit is logged by Polimetal as Transition Zone (Tr–Sulp, or disseminated sulphide). In Transition Zone material the disseminated pyrite minerals are aligned parallel to schistosity and appear as pyrite bands (Figure 7.11 and Figure 7.12).

Petrographic analysis indicates that the chlorite–sericite schist has been intensely altered (chlorite, epidote), silicified, carbonatised, and mineralised. Fractures and spaces between individual crystals of cataclastic structured epidote are filled with quartz, calcite, and chlorite. The largest euhedral epidote crystal size is up to 1 mm.

Figure 7.10 Chlorite–Sericite Schist in Outcrop

Figure from Polimetal, 2018.

Figure 7.11 Chlorite–Sericite Schist Core Photograph

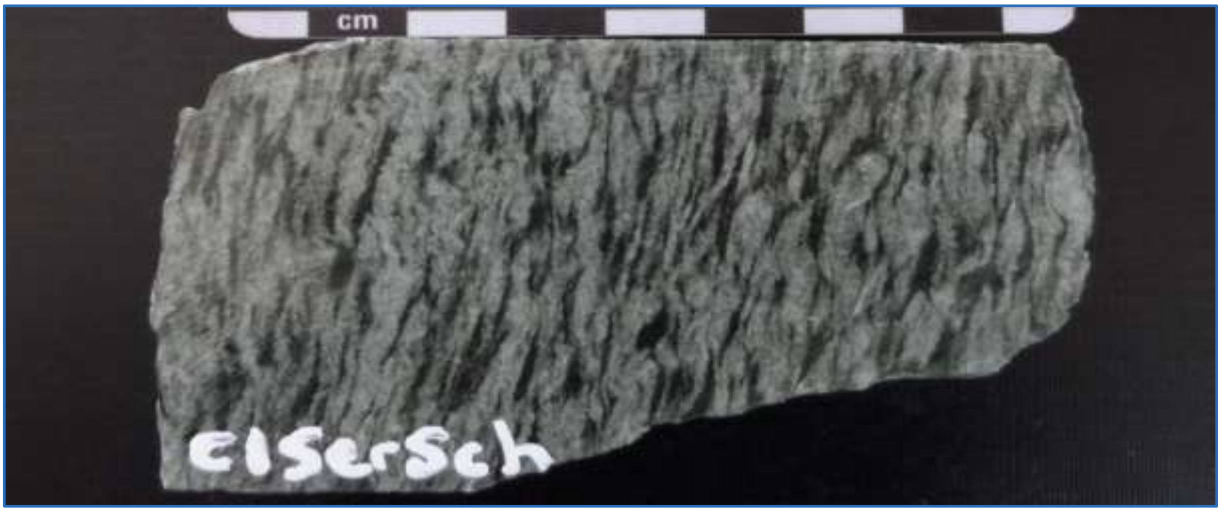


Figure from Polimetal, 2018.

Figure 7.12 Chlorite–Sericite Schist Altered to Tr–Sulp Core Photograph

Figure from Polimetal, 2018.

7.2.5 Quartz Schist (Upper Paleozoic)

Quartz schist is the lower-most stratigraphic unit at Gediktepe. It can be observed in outcrop in the southern part of the project area, from Üçoluk Hill to the Aşıdere stream, and in the north-east from Alçakgedik Hill to the Aşıdere stream in the south-east (Figure 7.13)

Macroscopically, the quartz schist is a beige–grey / beige–light green coloured unit containing large quartz porphyroblasts. Also observable are feldspar, chlorite, muscovite, and sericite (Figure 7.14).

Figure 7.13 Quartz Schist Containing Quartz Porphyroblasts in Outcrop *

Figure from Polimetal, 2018, (* south-east of Alçakgedik Hill).

Figure 7.14 Quartz Schist Core Photograph

Figure from Polimetal, 2018.

7.3 Mineralisation

The mineralisation at Gediktepe is associated with greenschist facies schist units, with the main mineralisation host rock unit being chlorite–sericite schist of the Upper Paleozoic. The mineralisation is thought to be developed syn-genetically in sedimentary units elongated along a north-east / south-west trending structure zone and metamorphosed to schist. Greenschist minerals are generally actinolite, chlorite, albite, and epidote.

Massive sulphide-type mineralisation occurs as lens shaped units trending north-east / south-west and dipping at approximately 20° to 40° to the north-west. Minerals include pyrite, sphalerite, tetrahedrite, tenantite, chalcopyrite, galena, and magnetite. The units are cut by later north-west / south-east trending post-mineralisation structures within the oxide zone, in which the sulphide mineralisation has been completely leached out, leaving gold and silver relatively intact.

Potentially-economic gold–silver–copper–zinc mineralisation is present to varying degrees in the sulphide zone.

The mineralisation at Gediktepe has been is divided by Polimetal into five main types, as summarised in Table 7.1

Table 7.1 Mineralisation Type Names

Horizon	Name
Oxide	Gossan
Sulphide	Massive Pyrite
	Massive Pyrite–Magnetite
	Enriched
	Disseminated Sulphide

Recent review of interpretations revealed that, in the northern part of the deposit and in the vicinity of the enriched mineralisation, areas within the sulphide horizon show high gold and silver and low base-metal (< 0.1% copper and zinc) concentrations.

7.3.1 Gossan (Oxide Mineralisation)

The upper portions of the Gediktepe deposit have been weathered, leached, and oxidised by naturally-occurring acidic surface water and ground water (Figure 7.15). The natural acidity is due to the presence of sulphides, particularly pyrite, within the oxide zone, and the sulphide mineralisation has been completely leached out, leaving gold and silver relatively intact. Relic 'lenses' of high-gold mineralisation remain in the oxide zone. There is some evidence that gold mineralisation has been transported downwards, chemically or mechanically, as there is often an increase in gold grade just above the oxide–sulphide contact (Figure 7.16).

Figure 7.15 Gossan in Outcrop

Figure from Polimetal, 2018.

The base of oxidation is generally abrupt, with rapid changes in metal grade across the oxide–sulphide contact. Copper and zinc grades are typically less than 0.10% within the oxide zone but increase to values typically around 1.40% Zn and 0.80% Cu immediately below the oxide horizon. Gold and silver follow the reverse trend, with gold in the range of 3.0 g/t in the oxide zone and often less than 0.7 g/t at the top of the sulphide zone (Figure 7.17).

Figure 7.16 Typical Profile of Mineralisation from Oxide to Sulphide

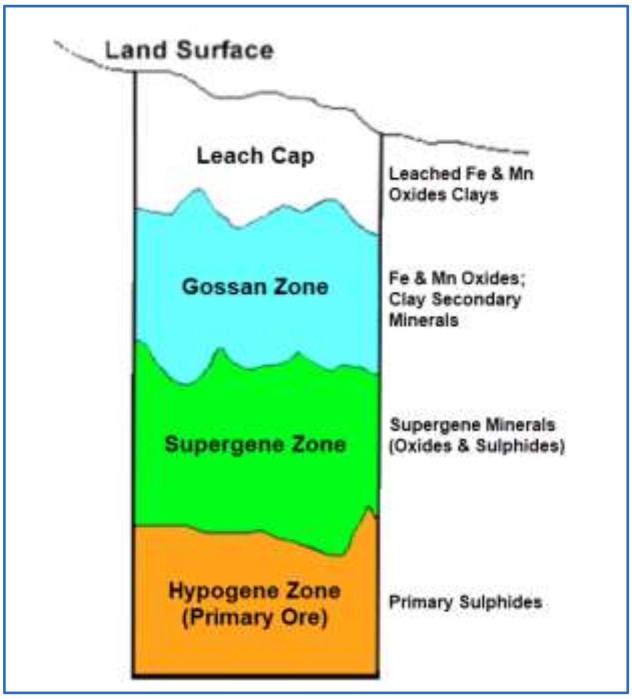


Figure from Polimetal, 2018, (after Okay and Tüysüz, 1999).

Figure 7.17 Mineralisation Profile at Gediktepe

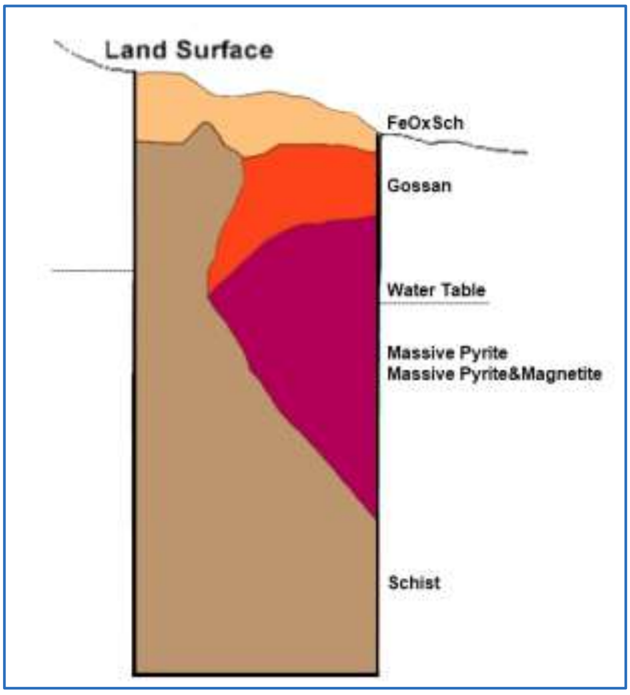


Figure from Polimetal, 2018, (after Okay and Tüysüz, 1999).

The Gediktepe oxide-type mineralisation is characterised by yellow-to-red leached zones of intense iron oxide gossan material. Near surface, is a leached cap, locally containing elevated gold values.

Figure 7.18 shows drill core through a typical vertical gossan profile at Gediktepe. The base-of-oxide (top-of-sulphide) is generally clearly discernible in drill core and is particularly clear in downhole assay trends.

Macroscopic investigation shows that the most common mineral is limonite with colloform textures, and consists of mostly goethite and, rarely, lepidochrosite.

Figure 7.18 High-Grade Gossan Mineralisation Drill Core

Figure from Polimetal, 2018.

7.3.2 Massive Pyrite (MPy)

The massive pyrite zone consists of fine to medium-grained pyrite, with massive-to-banded, vuggy textures, and locally sandy textures near structural features. The sphalerite–chalcopyrite–galena and weak covellite are observed as vug fracture fill and replacement mineralisation within a pyrite matrix. Locally, magnetite fragments are observed. The massive pyrite zone hosts high gold and copper mineralisation (Figure 7.19), (Çiftehan, 2015).

Figure 7.19 Massive Pyrite Drill Core

Figure from Polimetal, 2018.

7.3.3 Massive Pyrite–Magnetite (MPyMag)

Massive pyrite–magnetite has been distinguished from massive pyrite based on the presence of magnetite. Massive pyrite–magnetite shows the same textures as the massive pyrite. Quartz–magnetite fragments can be seen conformable with the schistosity, or primary bedding structures, within the massive pyrite–magnetite. The massive pyrite–magnetite characteristically shows lower gold–silver–copper–zinc–lead grades than the massive pyrite (Figure 7.20).

Figure 7.20 Magnetite Aligned with Schistosity in Core

Figure from Polimetal, 2018.

7.3.4 Enriched (Enrch)

The enriched zone consists of mainly chalcocite–covellite within fine to medium-grained pyritic mass. Occurring near or along structural features, the enriched zone is generally intensely fractured (Figure 7.21). Relative to other sulphide mineralisation zones, the enriched zone contains higher grade gold–silver–copper–zinc mineralisation.

Figure 7.21 Enriched Mineralisation in Drill Core (Blue Colour)

Figure from Polimetal, 2018.

7.3.5 Disseminated Sulphide Mineralisation (Tr–Sulp)

A lower grade sulphide mineralisation (gold–silver–copper–zinc–lead) is present within the rich disseminated (pyrite > 10%) chlorite–sericite schist (Figure 7.22). The total sulphide content in this zone exceeds 8.5%. Bands of 1–50 cm thickness appear parallel to bedding in this host rock below and above the sulphide mineralisation.

Figure 7.22 Chlorite–Sericite Schist with Pyrite Veining in Drill Core

Figure from Polimetal, 2018.

7.4 Structure

Structural features are not well mapped at surface due to the extensive ground cover and the degree of weathering of surface outcrops.

Offsets in mineralisation and related lithologies observed from drill cores indicate that the mineralisation is displaced by a series of steeply dipping north-west / south-east striking faults.

The tabular mineralised zones, particularly within the sulphide horizon, dip gently to the west. In the north-eastern portion of the deposit, mineralised zones may be shallower dipping. In several locations the overall trend is abruptly terminated, and the tabular mineralised zones are displaced downwards to the north-east, indicating post-mineralisation activity.

Progressing south-west to north-east across the deposit, this displacement geometry has been identified three to four times, and these features have been recognised as abrupt breaks or offsets during interpretation of mineralised bodies.

8 DEPOSIT TYPES

8.1 Deposit type

The characteristics of the Gediktepe mineralisation have been interpreted as a convex massive sulphide type deposit, illustrated in Figure 8.1, which implies a syngenetic style of sulphide mineralisation. Subsequent weathering and oxidation have been responsible for the development of oxide and gossan horizons.

Figure 8.1 Vertical Section of an Idealized Convex MS Deposit

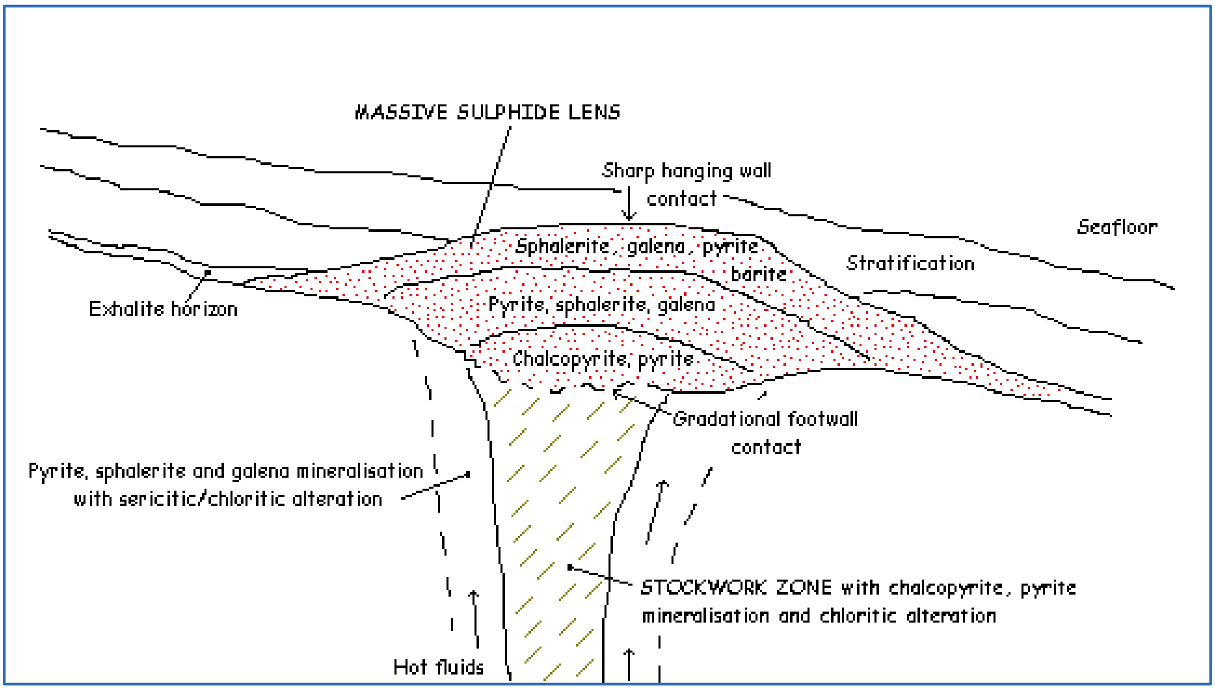


Figure from Polimetal, 2018, (after Okay and Tüysüz, 1999).

9 EXPLORATION

9.1 Introduction

The initial Gediktepe (Dursunbey) exploration license was acquired in auction by Anatolia Minerals (which became Alacer Gold Corp. following a merger with Avoca Resources). Anatolia Minerals conducted initial geological, geochemical, and geophysical activities on the license. Subsequent to the establishment of the Polimetal Madencilik San. ve Tic. A.Ş joint venture, the license was transferred to Polimetal.

A strong gold and copper geochemical anomaly was the catalyst for the first phase of drilling at the property by Polimetal in May 2013, leading to the discovery of the Gediktepe polymetallic deposit.

In addition to geochemical and geophysical surveys, 1:1,000 scale geological and structural mapping was conducted in the project area and followed up by 1:5,000 scale general mapping to outline possible structural and alteration features.

9.2 Geochemistry

Several surface geochemical sampling programmes were completed at Gediktepe from 2005 through 2014, with early work conducted by Anatolia Minerals prior to the establishment of Polimetal.

During 2014, a total of 1,048 soil samples, on a 100 m and 200 m grid pattern were obtained over the RN 85535 license area, representing 6.57 km^2. The soil sampling results were correlated with previous Anatolia Minerals soil sampling results to indicate a strong gold, copper, lead, and zinc anomaly, now known to directly overly the Gediktepe mineralisation. During the surface sampling, 151 rock chip samples were also collected from available outcrops.

The results of the surface geochemical sampling supported the presence of the gold–silver–copper–zinc–lead mineralisation along an elongated north-east / south-west structural zone. Further gold anomalies (> 20 ppb Au) north-west and north-east of the known mineralised zone remain untested and require further detailed work to define possible additional mineralisation.

The number and types of geochemical samples collected by the respective companies are summarised in Table 9.1 and Figure 9.1 is a compilation map of geochemical sampling on the property.

Table 9.1 Geochemistry Number of Samples

Company	Rock	Soil	Silt
Anatolia Minerals (Alacer Gold)	240	289	20
Polimetal	151	760	24
Total	**391**	**1,049**	**44**

Figure 9.1 Geochemical Sampling Compilation Map

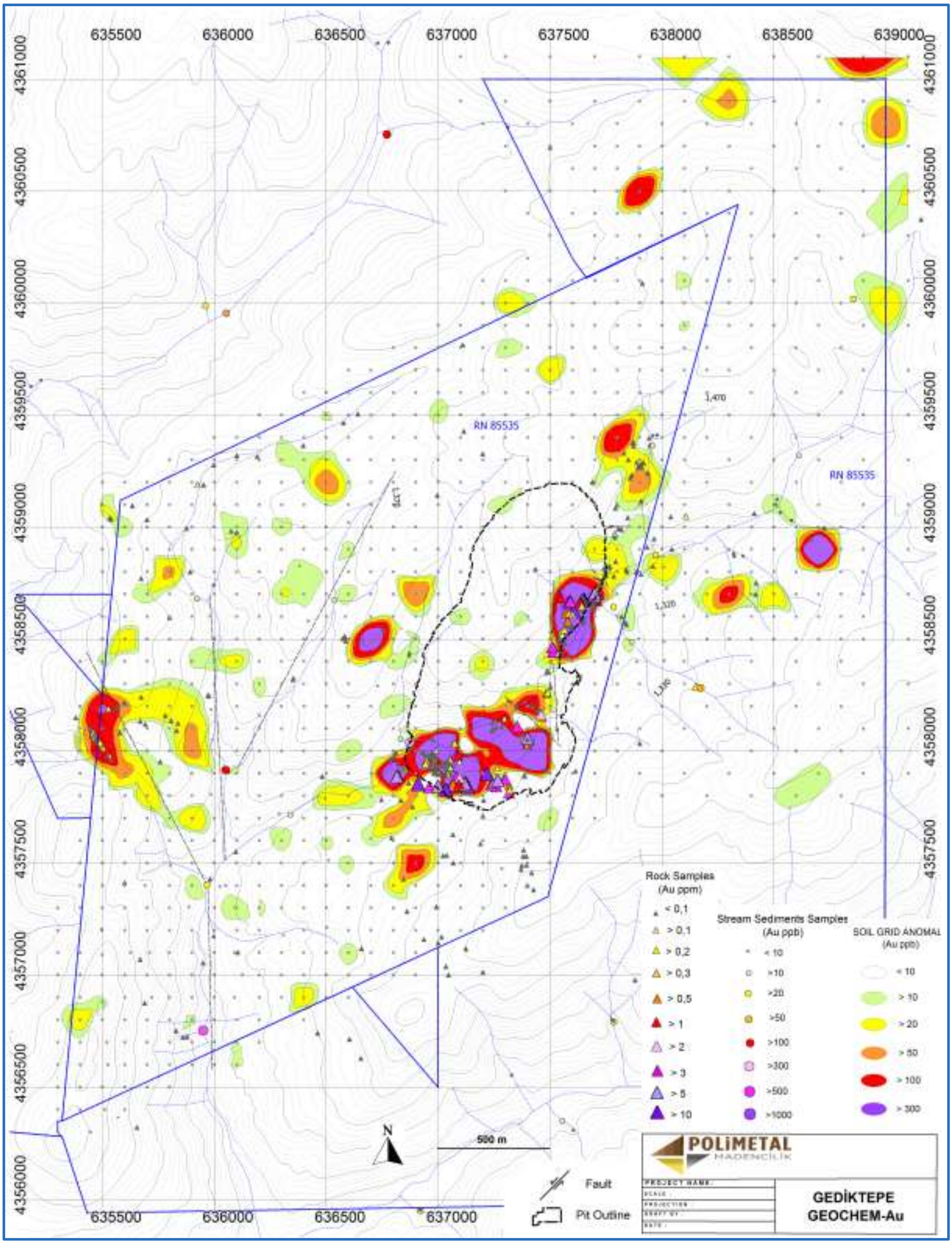


Figure from Polimetal, 2019.

9.3 Geophysics

Two types of ground geophysical surveys were completed at Gediktepe: Magnetic, and Induced Polarisation (IP).

9.3.1 Magnetic Survey

A magnetic survey was completed at Gediktepe during August of 2013. A total of 112.2 km of survey were conducted over 32 lines, at 100 m line spacing. The lines were oriented north–south and cover the entire area of the initial Gediktepe license 20054077 (Figure 9.2).

The magnetic anomalies identified in the survey indicate that medium and high-magnetic values correspond to the high-magnetite or massive sulphide mineralisation. The high-magnetic anomaly observed over the strong geochemical anomaly indicates that the high-magnetic anomalies may be a good indicator of other hidden sulphide zones containing magnetite. This observation provides support for further detailed evaluation of the strong magnetic and low-gold anomaly, observed approximately 500 m to the north-west of the Gediktepe deposit, and south of known mineralisation external to the license (Figure 9.3).

9.3.2 Induced Polarisation (IP) Survey

The IP survey, which was completed in-parallel with the magnetic survey, consisted of 22 IP section lines oriented north-west to south-east, for a total of 41.6 km, at 50 m, 100 m, and 200 m spacings. Most of the initial Gediktepe license 20054077 was covered by the IP surveying (Figure 9.2).

Higher chargeability results were obtained where disseminated pyrite mineralisation occurs within the chlorite–sericite schist (Figure 9.4), (Ibek, 2014).

The IP and magnetic geophysical surveys indicate that the low-resistivity and high-magnetic zones may correspond with richly mineralised zones, as supported by the drilling; therefore, detailed geological, geochemical, and structural work is recommended to explain the source of high-magnetic and low-resistivity anomalies in the area (Figure 9.4).

Figure 9.2 Magnetic (Left) and Induced Polarisation (Right) Survey Location Maps

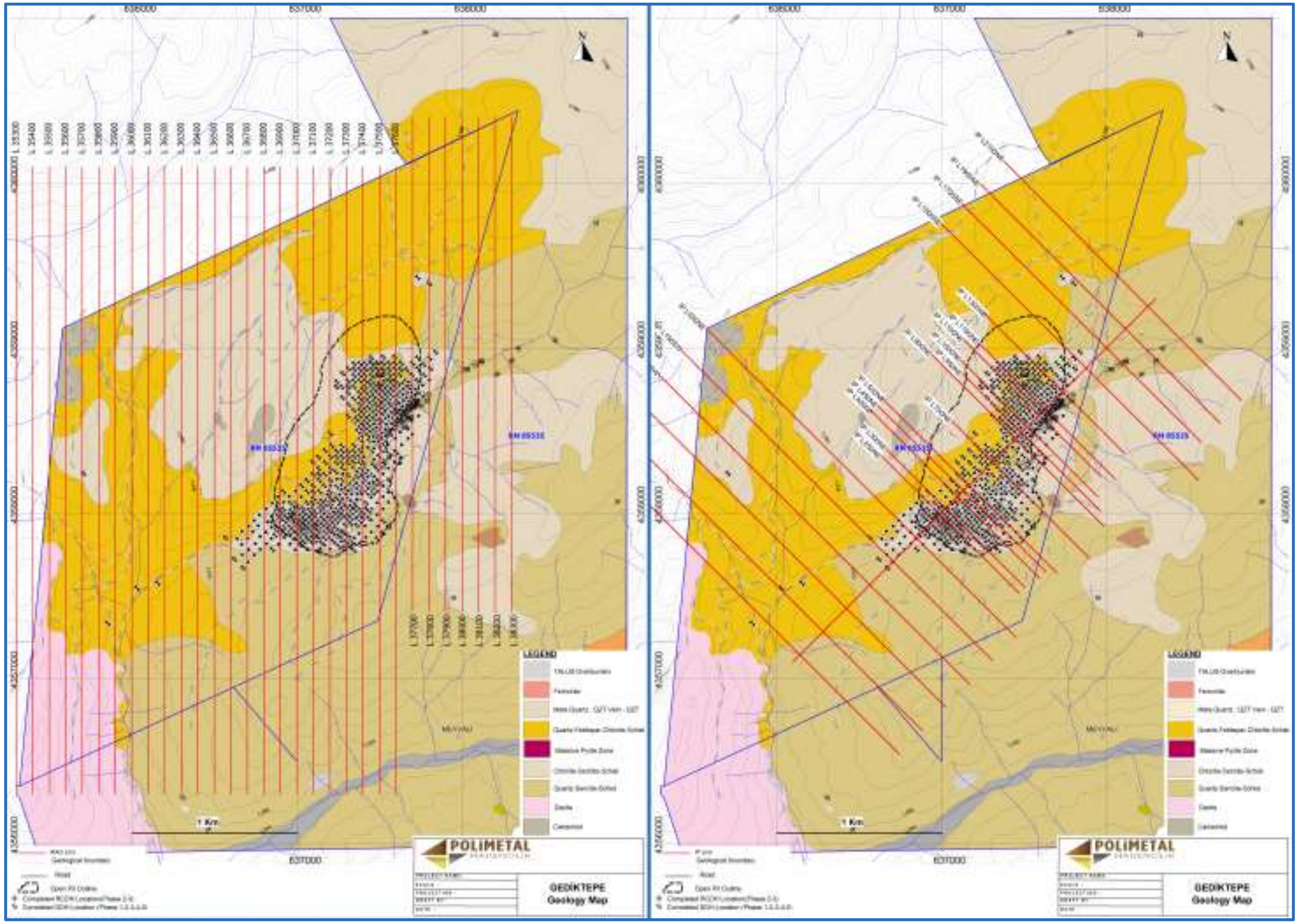


Figure from Polimetal, 2019.

Figure 9.3 Geophysical Potential: Mineralisation and Magnetic Anomaly Map

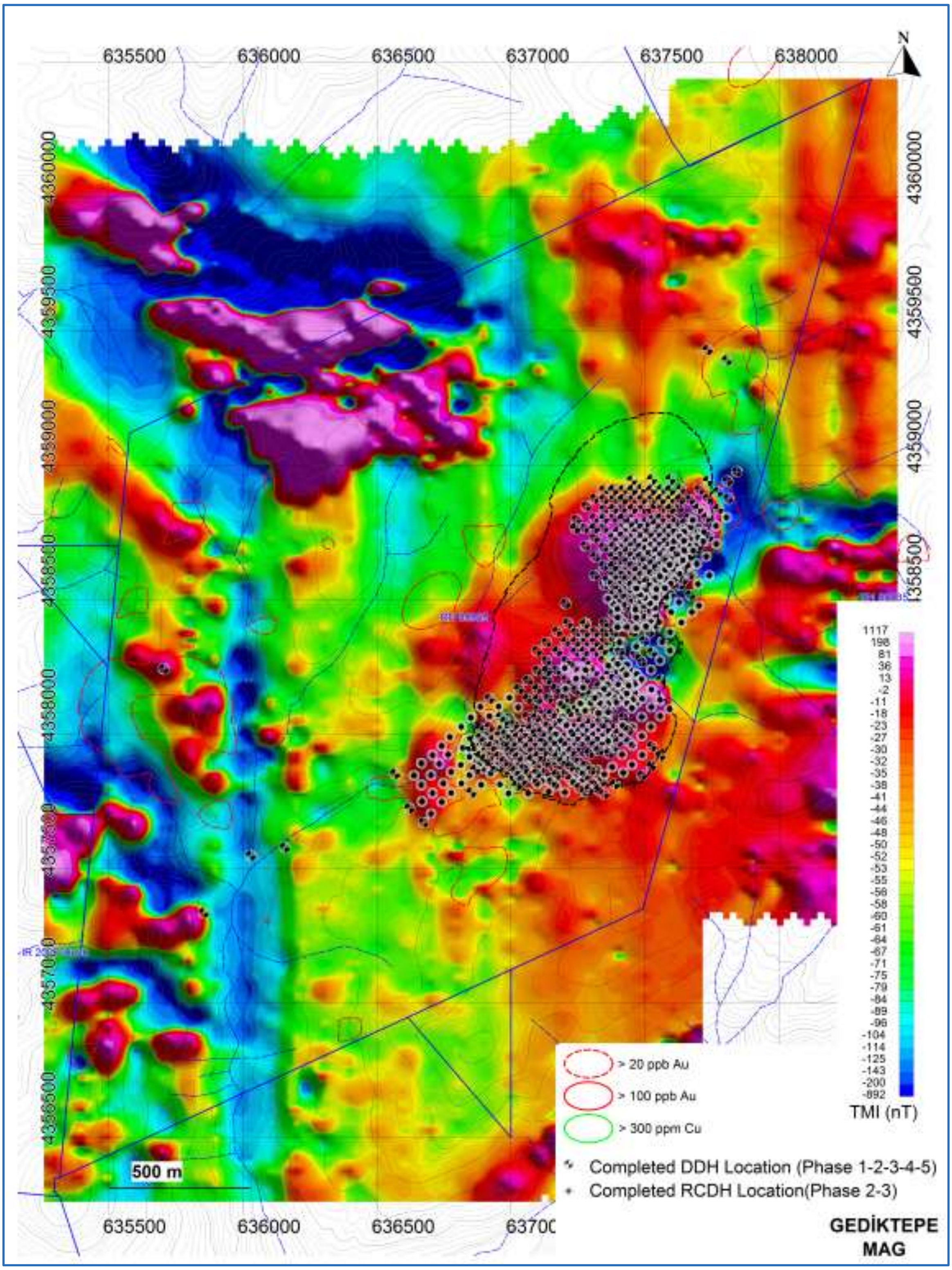


Figure from Polimetal, 2019.

Figure 9.4 Geophysical Potential: Relationship between Surveys and Mineralisation

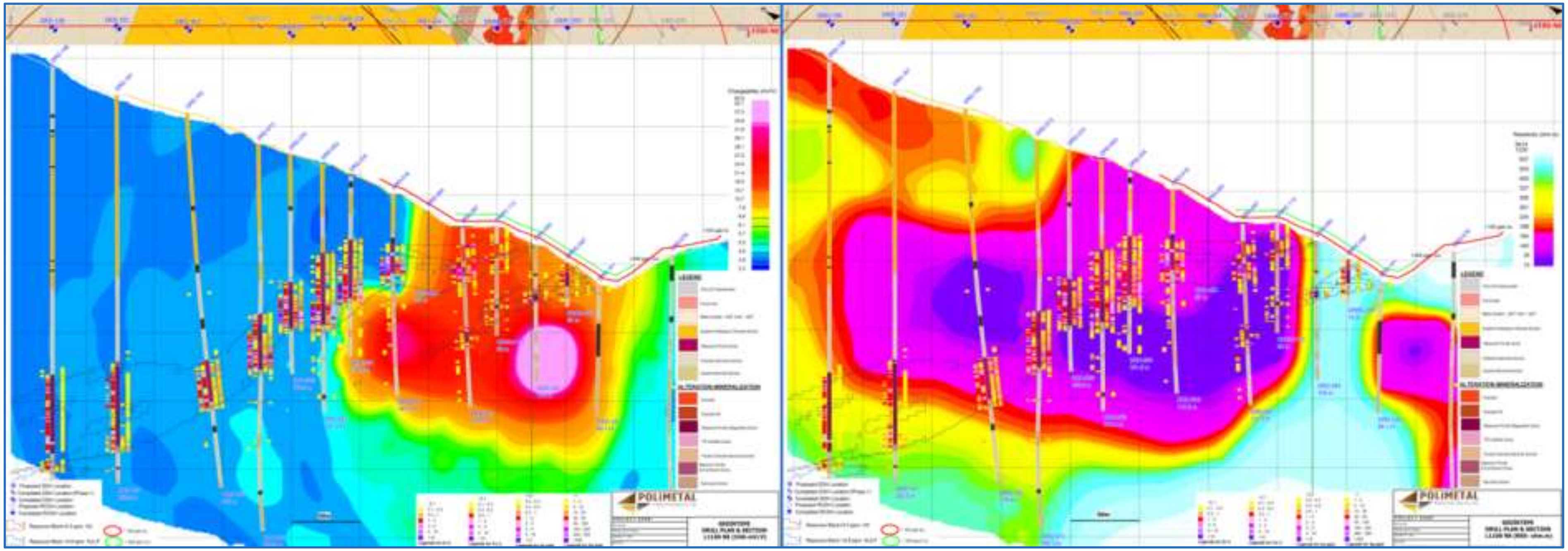


Figure from Polimetal, 2019.

9.4 Mineralogical Studies

Thin section and polished section analysis was completed on 19 drill cores and four hand specimens by Çağatay Madencilik – Mermercilik San. ve Tic. Ltd. Şti. in Ankara.

In addition, five potential ore sample composites were analysed as a mineralogical study by RDI as part of the metallurgical testwork. The objective was to determine the bulk mineralogy of the five selected composite mineralised samples, with an emphasis on gold and silver mineralogy.

Each sample was prepared as a standard polished thin section for study by reflected / transmitted light microscopy.

The following sections summarise the RDI (2015) report of identified mineral assemblages.

9.4.1 Silicate Mineralogy

Concentration of silicate mineralogy varies from sample-to-sample and is primarily composed of quartz and micaceous phases. Quartz occurs as angular fragments and mosaic aggregates with grain sizes up to approximately 150 µm. The majority of quartz is liberated; however, a small population carries inclusions of sulphides and other silicates (Figure 9.5). The primary mica phases muscovite, sericite, and chlorite vary in concentration. Both micas are very fine-grained, and generally occur as liberated plates with a grain size that varies greatly from 2 µm up to approximately 40 µm. Low amounts of pyrophyllite and talc are also present as small plates. With the exception of quartz, the micaceous phases do not appear to be associated with sulphides. A few angular shards of water-clear k–feldspar are present in some samples.

Figure 9.5 Fragment of Blue Covellite *

Figure from Polimetal, 2018. (* Covellite surrounded by iron oxide and silicates: Reflected light, 500x).

9.4.2 Carbonate Mineralogy

Calcite is present in all samples and varies in abundance from trace to a few percent. The carbonate is very fine-grained, with a grain size of up to 40 µm.

9.4.3 Sulphate Mineralogy

Fragments of barite are present in all samples, with concentrations that vary from trace to several percent. Individual fragments measure from 5 µm up to approximately 100 µm. Oxidised samples carry low levels of jarosite. Individual grains are very fine, up to 2–3 µm.

9.4.4 Oxide Mineralogy

Oxide mineralogy is represented in all samples, with concentrations that vary from trace to several percent. Iron oxide in the form of goethite is dominant, and occurs as fine-grained, granular material and large masses. The primary oxide found in the samples is magnetite, with a grain size up to 150 µm (Figure 9.6). Magnetite is liberated for the most part, but some grains are attached to pyrite. Larger grains frequently carry inclusions of sulphides and some show mild replacement by hematite. Other oxides in all samples include trace amounts of rutile and rare ilmenite (Figure 9.7).

Figure 9.6 Magnetite Grain with Inclusions of Pyrite and Chalcopyrite (Yellow)

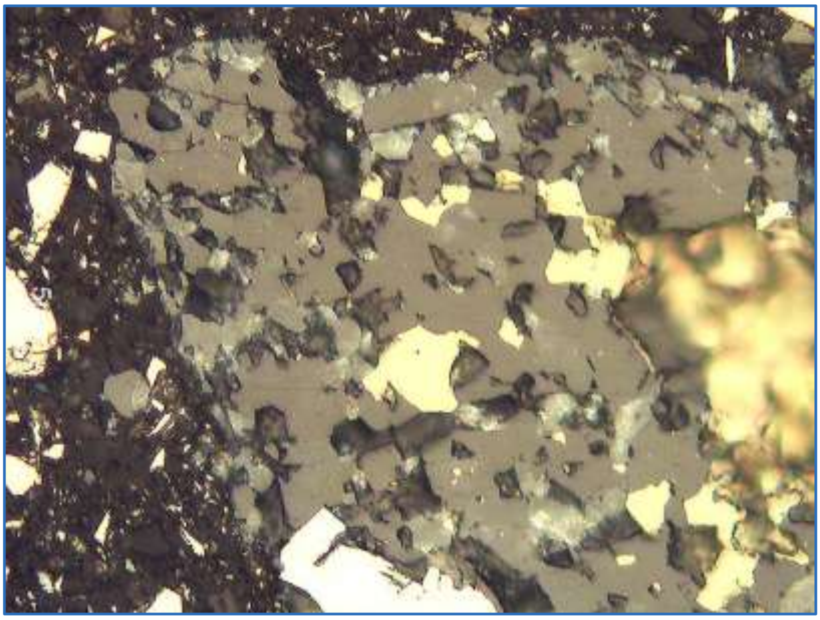


Figure from Polimetal, 2018.

Figure 9.7 Relict Pyrite in a Goethite Matrix *

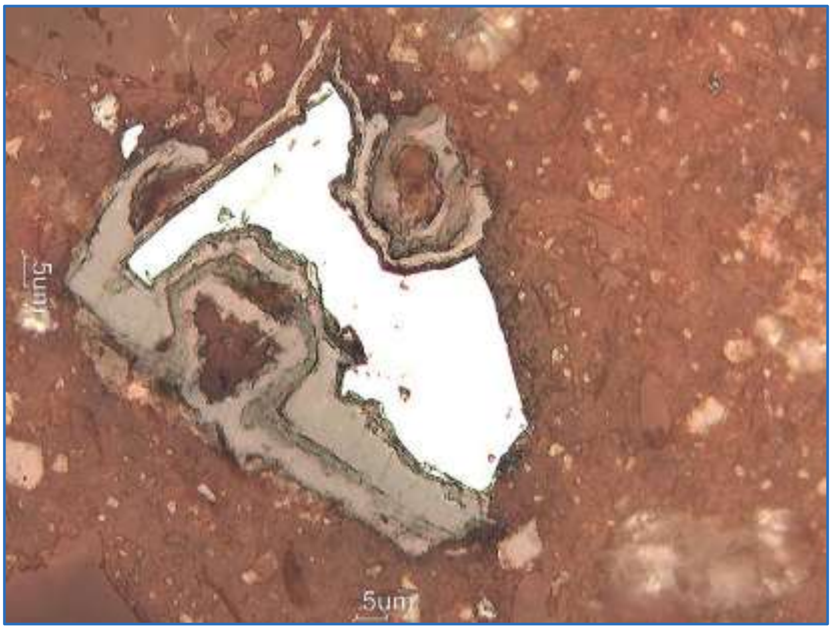


Figure from Polimetal, 2018, (* Reflected light, 500x).

9.4.5 Sulphide Mineralogy

Pyrite occurs as cubes and small fragments that range in size from 1 µm up to 150 µm. Large grains commonly carry minute inclusions of magnetite and other sulphides. Chalcopyrite appears as liberated fragments, but more-commonly as aggregates with pyrite and sphalerite. Grain size is generally very fine, with measurements in the 2 µm to 50 µm size range. Chalcopyrite commonly shows mild to strong alteration to covellite and chalcocite (Figure 9.8).

Figure 9.8 Aggregates of Pyrite, Sphalerite (Sp), and Magnetite (Mg)

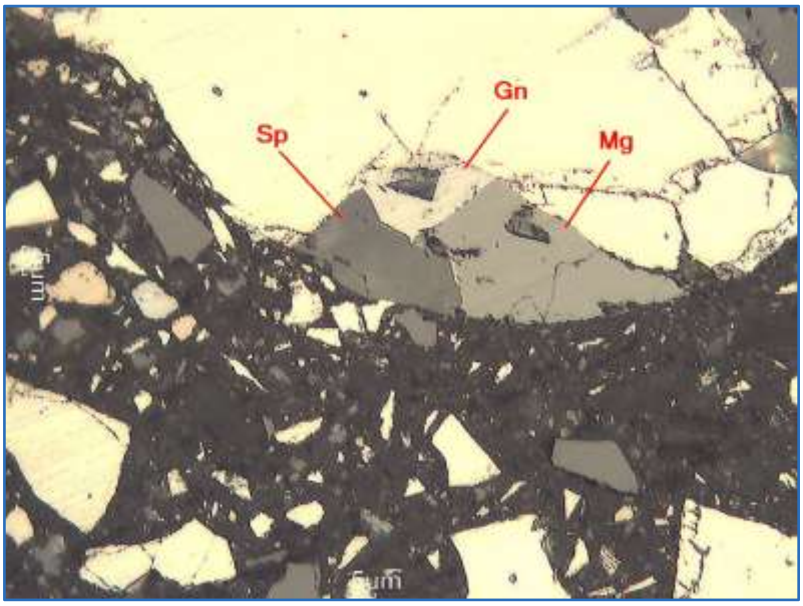


Figure from Polimetal, 2018.

A few grains of sphalerite also show minor covellite replacement. Trace galena is present in all samples, with a grain size up to 25 µm. A few liberated fragments are present; however, the majority of galena is seen as small inclusions in sphalerite and more commonly in pyrite (Figure 9.9).

Figure 9.9 Pyrite Attached to Grey Sphalerite Rimmed with Blue Covellite and Chalcocite and Galena

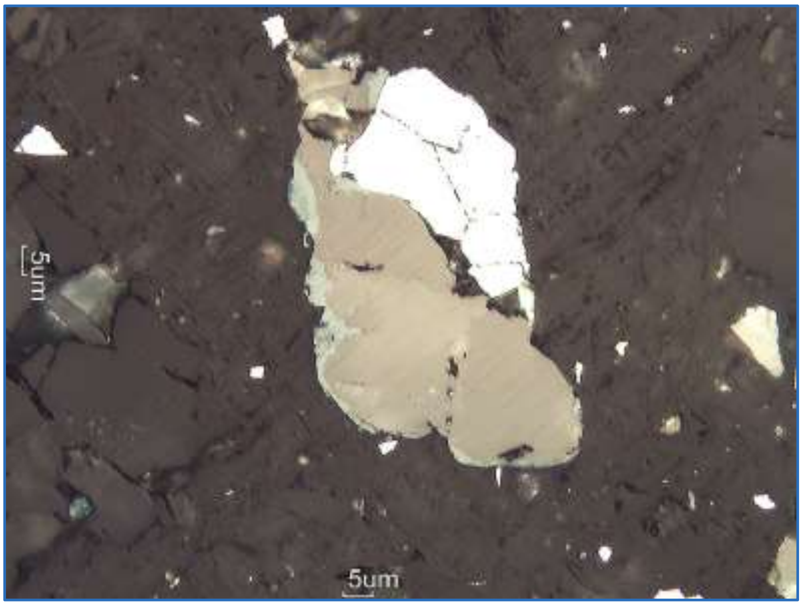


Figure from Polimetal, 2018.

9.5 Gold / Silver Mineralogy

An extensive search of all samples failed to identify discrete silver mineralogy, either as a sulphide or native metal. Fine-grained silver mineralogy may be associated as an impurity with galena, pyrite, iron oxide, or covellite. A few small (2–3 µm) gold grains were seen in granular iron oxide and appear to be liberated (Figure 9.10).

Figure 9.10 Gold Grain in Iron Oxide, (Reflected Light – 500x)

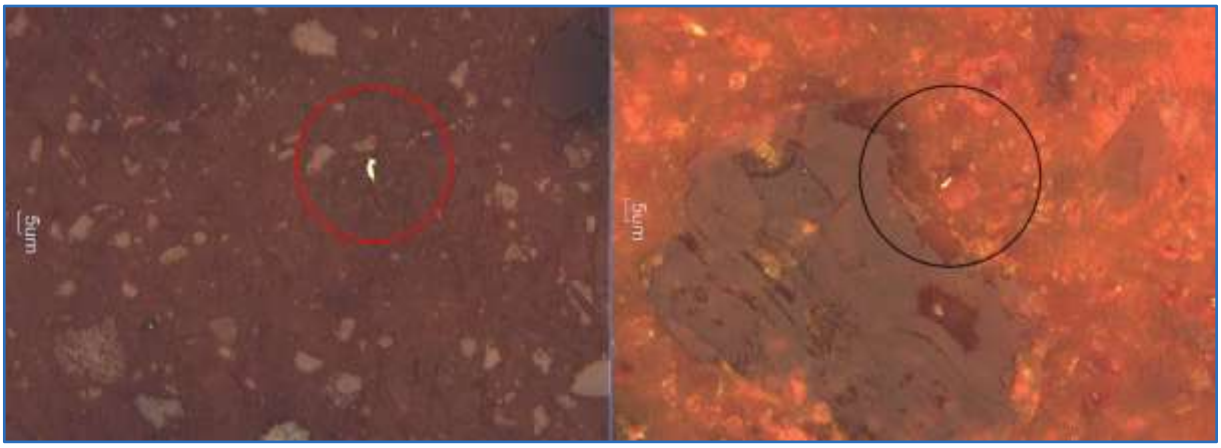


Figure from Polimetal, 2018.

9.6 Topographical Surveys

The project coordinate system references UTM European Zone 35.

A detailed topographic map, with 1 m contour intervals, incorporating all existing roads, was surveyed across the deposit area (Figure 9.11) for a total of 3,500 measured survey points.

Figure 9.11 Topographic and Road Survey Map

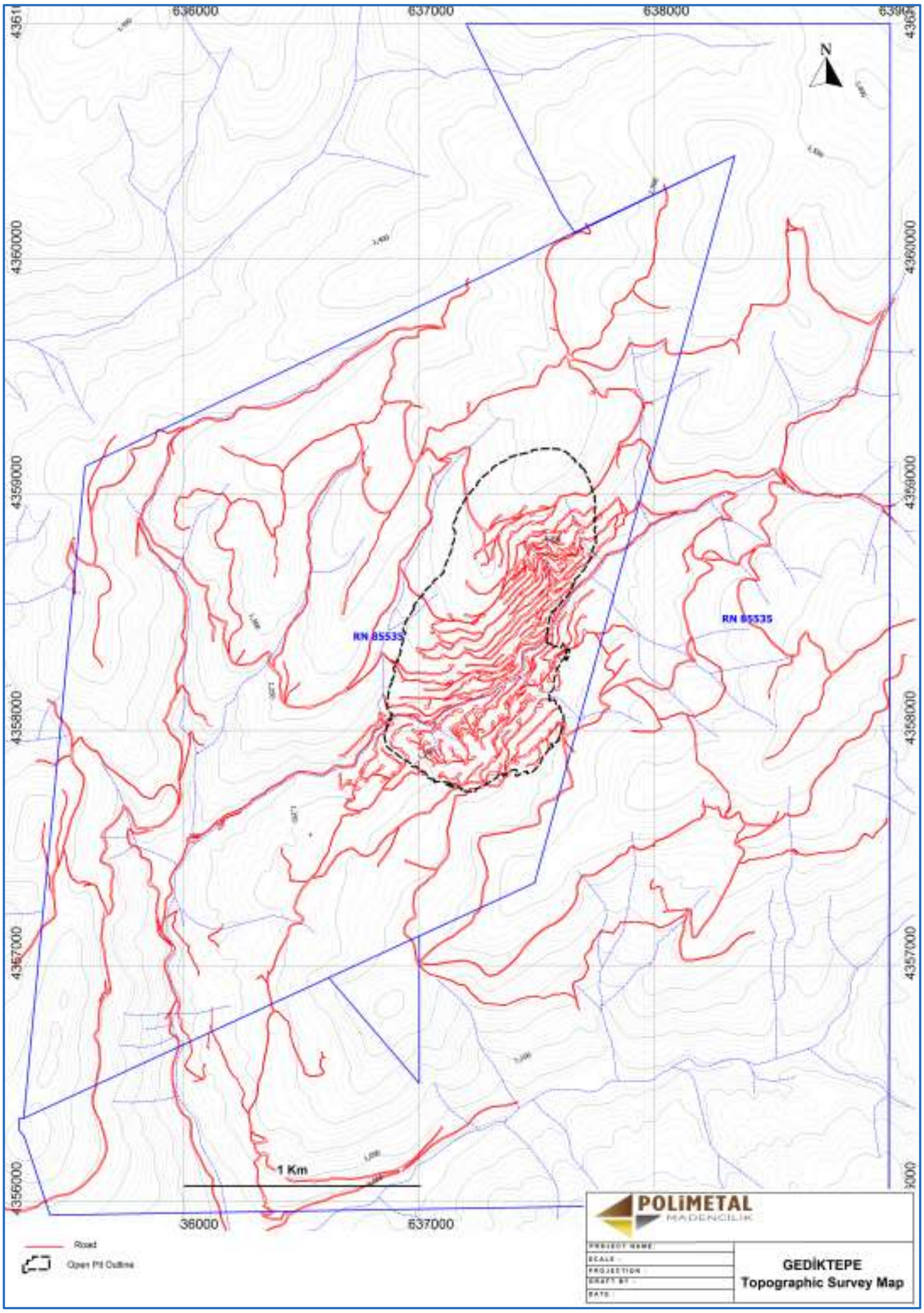


Figure from Polimetal, 2019.

10 DRILLING

The cut-off date for the drillhole data was 21 March 2018.

The majority of drilling at Gediktepe up to the cut-off date has focussed on outlining and then defining the main mineralisation over a strike length of 1.6 km and down-dip extents, projected to surface, of up to 600 m. The work has been conducted through five distinct phases (campaigns), with drilling layouts dominantly arranged along a set of 45° azimuth grid lines, with line spacing down to 25 m intervals, referencing the UTM European Zone 35 coordinate system. Magnetic declination for the area is +4.78°.

Additionally, there are a number of holes that have been drilled with other objectives, including geotechnical investigations, groundwater level determination, location selection for tailings storage and heap leach ponds, seismic data, etc.

Table 10.1 shows the total database listed by types of drilling and the number of holes drilled at Gediktepe up to the PFS19 cut-off date.

Table 10.1 Summary of All Drilling Completed at Gediktepe to 21 March 2018

Drillhole Purpose	BHID Prefix	No. of Holes
Resource Definition – Diamond Drilling	DRD	434
Resource Definition – Reverse Circulation	DRRC	191
Geotechnical	GEO, J, OPJT, S	39
Fresh Water Reservoir	BSK, DSK, EK, KSK, SK	16
Tailings Storage Facility	ABSK	12
Heap Leach Ponds	BH	12
Seismic	SIS	2
Water Hole	W	14
Waste Dump	WRD	10
	Total	**730**

10.1 Drilling Programmes

Exploration drilling (Phase 1) commenced on April 2013. Throughout the phases drilling by both diamond core (DD) and reverse circulation (RC) drilling was completed by local contractor companies (Asyatek, Spektra, IDC, Ortadoğu). Diamond core holes were predominantly started using PQ core size, and rarely with HQ holes. Most deeper holes, however, needed to switch to HQ at depth. RC drilling was restricted to Phases 2 and 3, and was used on the margins of the deposit to define extensions or set limits, and for infill in some parts of the deposit, (Polimetal, 2018).

The majority of holes have been drilled vertically, to intersect the low-angle zones of mineralisation. Eight of the initial 11 Phase 1 holes were angle holes, with the remainder of the holes drilled vertical or sub-vertical. The average deviation of the surveyed holes is less than 1° per 100 m.

At the end of each phase of drilling, hole collars were surveyed by a local surveying firm. Downhole surveys were performed on a majority of the diamond drillholes, generally at 40 m intervals, with a Devico reflex device. RC drillholes were not surveyed downhole.

10.2 PFS19 Drillhole Dataset

The geological modelling work proceeded with a subset of the drillholes listed in Table 10.1. Table 10.2 summarises the numbers of each type of hole per phase of drilling for the drillhole data used in the 2018 geological modelling for PFS19 (PFS19 Drillhole Dataset). Table 10.3 summarises the meterage of those holes. Figure 10.1 shows the locations of these holes.

Table 10.2 Count of Drillholes in the PFS19 Drillhole Dataset by Phase and Type

Drilling Phase	Count by Hole Type					Total PFS19 Drillhole Dataset
	Diamond Drillholes				RC Drillholes	
	DRD	GEO	J	OPJT	DRRC	
1	11					11
2	144				84	228
3	153				107	260
4	93	1				94
5	32†		2	2		36
Total Count	**433**	**1**	**2**	**2**	**191**	**629**
Proportion	**69%**	**0% ***	**0% ***	**0% ***	**30%**	**100%**

* Sum of individual proportions does not add to 100% due to rounding
† Excludes DRD-401

Table 10.3 Metreage of Drillholes in the PFS19 Drillhole Dataset by Phase and Type

Drilling Phase	Metres by Hole Type					Total PFS19 Drillhole Dataset
	Diamond Drillholes				RC Drillholes	
	DRD	GEO	J	OPJT	DRRC	
1	1,529					1,529
2	17,158				6,920	24,078
3	26,544				6,309	32,853
4	5,189	63				5,252
5	5,319†		615	480		6,414
Total Metres	**55,739**	**63**	**615**	**480**	**13,229**	**70,127**
Proportion	**79%**	**0%**	**1%**	**1%**	**19%**	**100%**

† Excludes DRD-401

Of the total 438 diamond drillholes in the PFS19 Drillhole Dataset, 388 have downhole survey data.

Figure 10.1 Drillhole Location Plan – PFS19 Drillhole Dataset

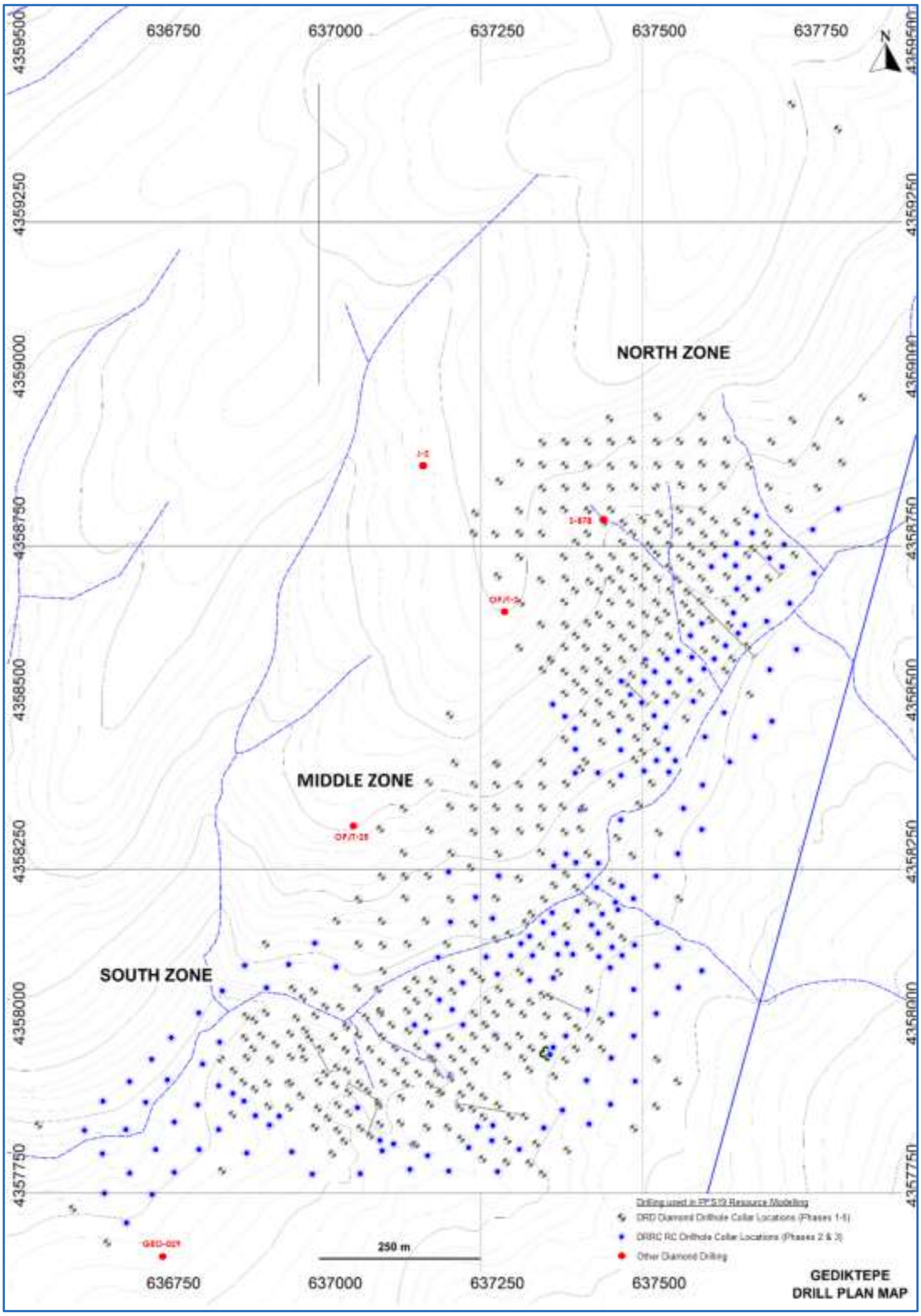


Figure from Polimetal, 2018.

10.3 Drilling Methods

10.3.1 Diamond Core

Diamond core samples are boxed at the drill rig and transported by company vehicle to the core logging facilities nearby. Core is washed prior to being logged for geotechnical and geological parameters, including lithology, alteration, mineralisation, and structures.

10.3.2 Reverse Circulation

RC samples are collected using a rotary splitter at the drill rig. Chip samples are logged for features including lithology, alteration, mineralisation, and, where possible, structures. Approximately 55% of the RC samples were taken at 2 m intervals. The remainder of the samples are shorter, with the shortest and most common length being 1 m. Weights of RC samples are recorded and are typically approximately 3 kg.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 On Site Sample Preparation

All samples of drill core and RC chips were subjected to quality control procedures that prescribed handling, sampling, analysis, and storage of the drill core.

11.1.1 Diamond Core

Sampling for assay is nominally at 1–2 m intervals, selected on a geological basis, but may be reduced to as little as 0.40 m at boundaries within the mineralised zones.

Drill core samples were cut by a diamond blade rock saw, with half of the sawn core placed in individual bags in preparation for despatch to the laboratory for assaying, and the remaining half returned to the original core box for historical reference. The retained core is stored in a core shed at the field camp.

Polimetal inserts standards, field duplicates, and blanks into the sample shipments. Duplicates are additional splits of the core (i.e. quarter-core).

11.1.2 Reverse Circulation (RC)

The RC sample splits for assaying are approximately 3 kg. The remnant (approximately 3 kg) of sample residues after splitting at the rig is retained in storage at the field camp.

Similar to core sampling, standards, blanks, and duplicates are submitted with RC samples. RC duplicates are second splits taken at the drill rig.

11.2 Sample Quality Control and Assurance (QA/QC)

11.2.1 Standard Samples

Certified reference materials (CRMs) were used to test the accuracy of the assays and to monitor the consistency of the laboratory results. Standards are inserted on a nominal 1-in-20 basis.

Four CRMs were used for the project; two of the CRMs are for gold, providing confirmation at 0.63 g/t Au and 3.84 g/t Au respectively. The third and fourth CRMs are base metal standards. The CRMs were selected randomly from the available suite and inserted into the sample sequence every 20 samples.

The names of the CRMs and their corresponding values are summarised in Table 11.1.

Table 11.1 Certified Reference Materials

Name	Source	Element	Unit	Value
G907-4	Geostats Pty Ltd	Au	g/t	3.840
G910-8	Geostats Pty Ltd	Au	g/t	0.630
GBM398-1	Rocklabs	Cu	%	1.482
		Zn	%	2.030
		Pb	%	2.667
		Ag	g/t	5.100
GBM914-10	Geostats Pty Ltd	Au	g/t	0.137
		Cu	%	1.864
		Zn	%	9.697
		Pb	%	4.671
		Ag	g/t	9.400

A total of 1,931 CRMs were analysed during the 2013 through 2018 drilling programmes, which comprised a total of 37,856 drill samples.

11.2.2 Blanks

Blanks are generally used to check the cleanliness of the laboratory. Blanks are inserted on a nominal 1-in-20 basis, and typically inserted as the first and last sample of a drillhole to assure no carryover of values from hole-to-hole. In total 1,737 blanks were inserted into the sample batches, which calculates out to an average insertion rate of 1-in-25 samples.

Five blank samples, AuBlank_S50, AuBlank62, AuBlank65, AuBlank66, and BlankST154, were used. The blank samples, purchased from Rocklabs, consist of a mixture of finely pulverised feldspar and basalt. Prior analysis of the blanks had confirmed low-Au. The sample sachets were stored in an environment free from potential Au contamination.

11.2.3 Duplicate

Pulp samples were re-submitted to ALS Chemex, Izmir, to ascertain the repeatability and precision of assays. During the period from 2013 through 2017, duplicate samples were inserted on a nominal 1-in-40 basis, and after the 2017 drilling programme the rate of insertion of duplicate samples was increased to a nominal 1-in-20.

11.3 Laboratory Sample Preparation and Analysis

Following standard procedures, drill samples were assigned unique sample tag numbers and weighed. Samples from each drillhole were prepared as a single batch, along with the associated blanks, duplicates, and CRM samples.

Transportation from Gediktepe to the respective laboratories was the responsibility of Polimetal. The despatched samples were accompanied by a completed sample shipment form (GSS form), which includes the project code, collar coordinates, sample type, analytical methods, QA/QC procedures, and sender details. GSS forms are completed by field staff and approved by the database team prior to shipment. Once samples are delivered to the laboratory, laboratory staff register the samples into their system and confirm with Polimetal that the transfer of the sample has taken place.

During Phase 1 drilling, all assays were submitted to SGS laboratory in Ankara. From Phase 2 (2013), all samples were submitted to the ALS Chemex laboratory in Izmir. Both the SGS laboratory in Ankara and the ALS laboratory in Izmir are ISO–9001:2008 certified. The same set of CRMs were submitted throughout the phases.

Gold was assayed using the Fire Assay Fusion technique with a nominal 30 g sample weight (ALS Code Au–AA25) with additional 33 element analysis by ICP–AES with Aqua Regia Digestion (ALS code ME–ICP61a).

11.3.1 SGS Procedures

The SGS procedures applied to the Phase 1 core during 2013 were as follows:

- The samples were logged in and weighed on arrival.
- The samples were dried and crushed by SGS protocol CRU24.
- Pulps were prepared. The laboratory certificates from SGS did not list the pulp protocol, but the nominal pulp criteria for the Atomic Absorption Spectroscopy (AAS) and Inductively Coupled Plasma Mass Spectrometry (ICP) analysis at SGS is 75 µm.
- Gold was assayed by protocol FAA303, a fire assay with AAS finish on a 30 g aliquot.
- Copper and silver were assayed by protocol AAS42S, which is an AAS finish.
- All other metals were assayed by protocol ICP40B, which is a four-acid digestion and multi-element ICP procedure.

11.3.2 ALS Procedures

The ALS sample preparation and assay procedures were applied to the Phase 2 through Phase 5 drilling for both core and RC samples.

- The samples were logged in and weighed on arrival.
- The core samples were dried and crushed by ALS protocol CRU–31 with 70% passing less than 2 mm. RC samples were dried before splitting, without crushing.
- Samples were split with a riffle splitter before pulping.
- Pulps were prepared with ALS protocol PUL–32, where 1 kg is reduced to 85% passing 75 µm.
- Gold was assayed by protocol Au–AA25, a fire assay with AAS finish on a 30 g aliquot.
- All other metals were assayed by protocol ME–ICP61a, which is a four-acid digestion to report 33 elements by ICP methods. After a three-month period of storage at the ALS laboratory, pulps were transferred to Polimetal's field camp storage facility.

The ALS laboratory also inserted internal standards into every assay batch and the results are reported to Polimetal.

11.3.3 Third Party Check Assays

Additional to routine QA/QC procedures and analysis, a set of 726 pulp check samples from each phase of drilling were sent to AcmeLabs, SGS, and Argetest to confirm the original assay results provided by the ALS laboratory.

11.4 Review of QA/QC

On completion of each drilling phase, Polimetal undertakes an in-house analysis of the QA/QC laboratory results. As soon as the results of the analysis are received, they are checked according to QA/QC protocols. Any failed results are re-analysed. Final accepted results are transferred to the database entry process.

Polimetal commissioned an independent consultant, AMC Consultants Pty Ltd (AMC), to undertake data compilation and verification as part of a geological and resource model update in 2018. An analysis of the Gediktepe sample QA/QC results provided by Polimetal was undertaken. The analysis was undertaken for Au, Ag, Cu, Pb, and Zn. A selection of charts from the analysis are shown Figure 11.1 to Figure 11.3.

Figure 11.1 QA/QC Chart Example: Phase 2 CRM: Au

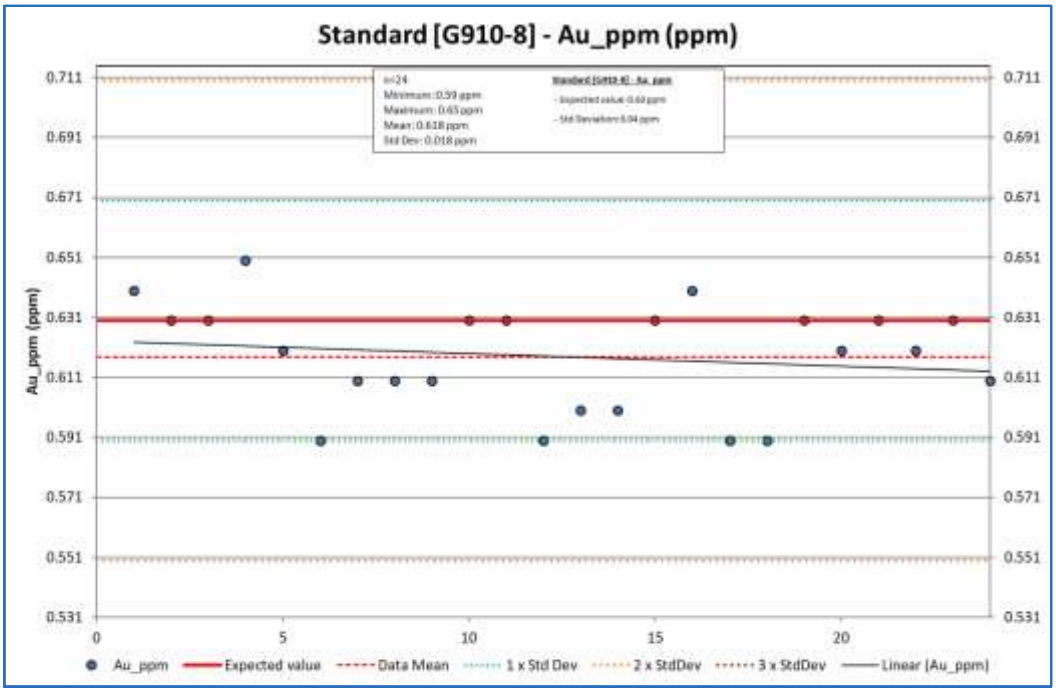


Figure from Polimetal, 2018.

Figure 11.2 QA/QC Chart Example – Phase 2 Duplicates Cu

Statistic	Original	Duplicate
Sample Count	742	742
Minimum	0.5000	0.5000
Maximum	9840.0000	9830.0000
Mean	654.9272	649.0748
Median	41.0000	45.5000
Standard Deviation	1509.5809	1478.9562
Correlation Coefficient	0.990	
Pairs within +/- 20% variance	63%	
Pairs with Pair Mean >=10	502	68% of total data
Pairs ≤ 20% HARD	458	91%
Pairs ≤ 20% RPD	402	80%
Average RPD	-0.019	Skewness 0.123

Project	218028_Gediktepe
Data Series	???
Data Type	Phase II Duplicates
Analytical Method	??
Detection Limit	1 ppm
Comments	**Insert Comments**

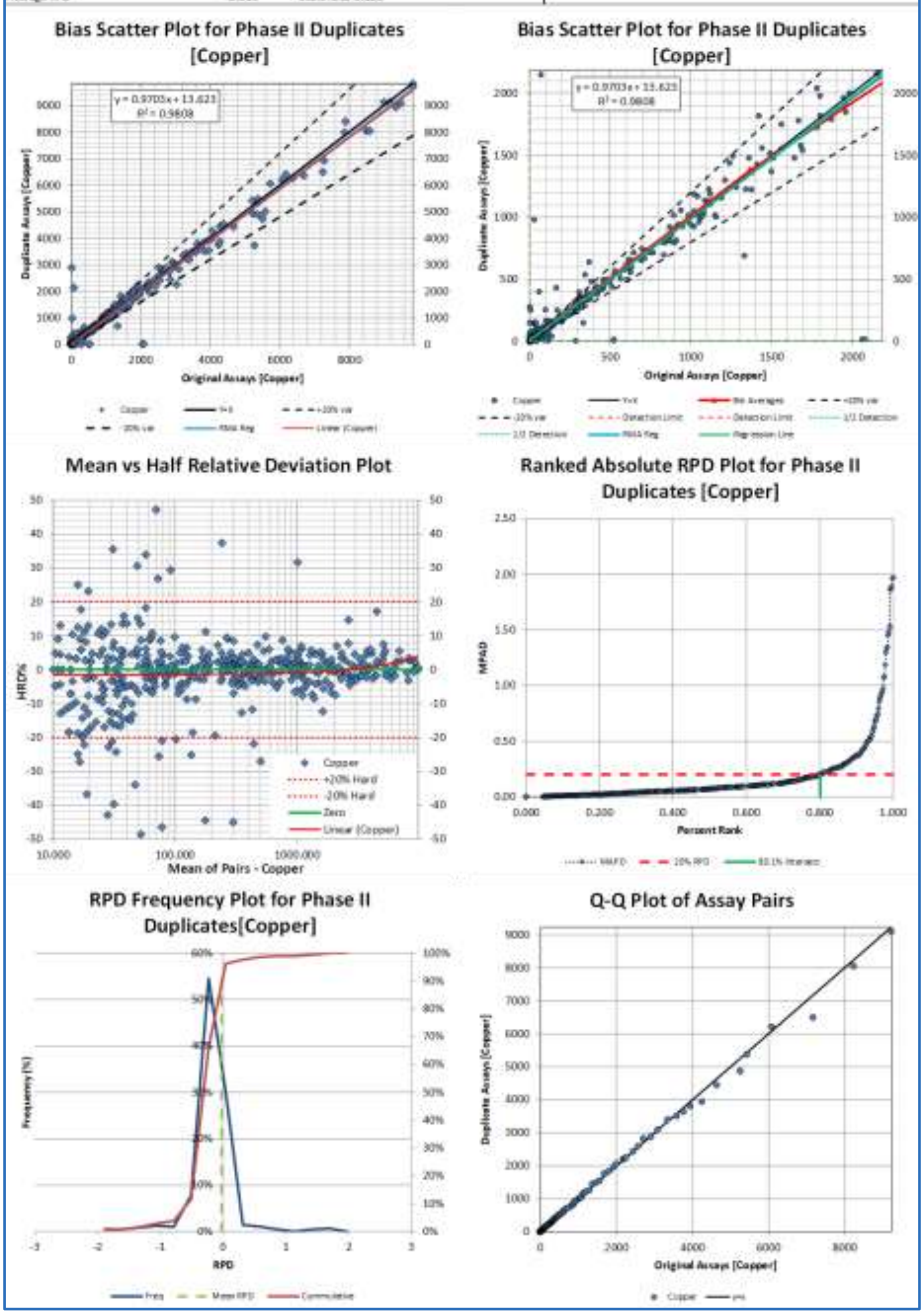


Figure from Polimetal, 2018.

Figure 11.3 QA/QC Chart Example: Phase 5 Blanks: Au

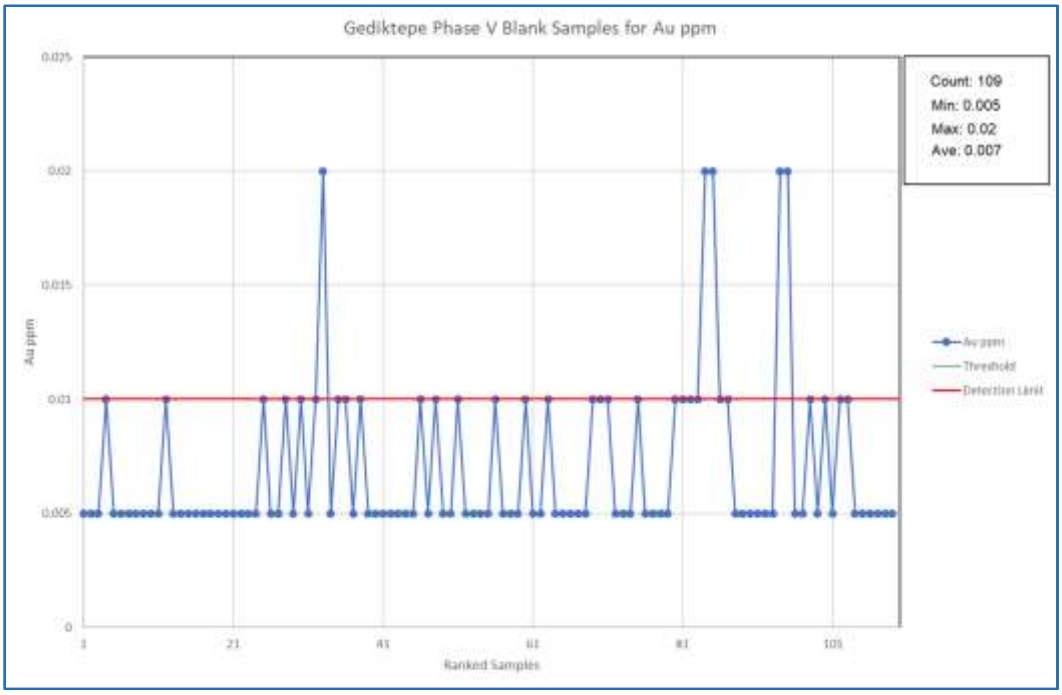


Figure from Polimetal, 2018.

No material issues were identified during the analysis of QA/QC data. However, the following observations were made:

- The latest QA/QC report (Polimetal, 2018) does not disclose results for Ag.
- Blanks are reported for Au only.
- Early-stage assays, especially from Phase 1, have higher variance but remain within limits. This possibly reflects subsequent laboratory refinement of analytical processes and internal quality control.
- Phase 5 exhibits higher variances, but again within limits.
- Some clear mislabelling of standards is evident, suggesting that data management is a minor issue.
- Inconsistent data definitions also impacted analysis:
  - Non-standard CRM naming.
  - Inconsistent methods for reporting of below detection samples, e.g. 0, < 0.01, or –0.01.
- The lack of adequate sample identification of the QA/QC samples limited the ultimate usefulness of the QA/QC programme. For example, it was not possible to distinguish between the various types of duplicate samples (field duplicates, pulp duplicates, laboratory duplicates), illustrating a processing issue. It was also not possible to track the blanks through the preparation process as a consequence of the lack of this data definition.

Recommendations for future QA/QC work include:

- Keep record of assay dates, to allow for time-based ranking.
- Make use of the ALS 'Webtrieve' process to obtain real-time laboratory results, including time stamp information, and incorporate this data into DataShed database software.
- Store analytical process per analyte rather than in concatenated 'assay requested' field.

11.5 Database Assembly

Upon receipt of analytical batches, blanks, standards, and duplicates were examined for evidence of laboratory contamination, analytical error, assay reproducibility, and drill-bit contamination.

Assay certificate information was forwarded electronically to Polimetal, where employees in Ankara maintain a master assay database using DataShed.

The list of assay fields exported in the data provided is Au, Ag, Al, As, Ba, Be, Bi, C, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn, and Hg.

11.6 Bulk Density

Density measurements are routinely undertaken by Polimetal geology staff on whole-core samples at the logging facility.

Core samples of approximately 10 cm lengths were selected every 5 m within mineralised zones, and every 10 m outside of mineralisation. Samples were dried in an oven at 105°C for 24 hours, before being coated in wax. Samples were then weighed in air, and then again while immersed in water. The difference in the two weights is the weight in the water displaced by the volume of the core sample.

After measurements had been completed, core samples were labelled and returned to relevant positions within the core boxes.

Calculations of specific gravity (SG) are conducted according to the following formula:

$$\boldsymbol{SG = \frac{Mdry}{Mwax - Mwater - \left(\frac{Mwax - Mwater}{0.86}\right)}}$$

The SG values for each primary logged unit at Gediktepe are given in Table 11.2.

Table 11.2 Bulk Density Values for Gediktepe Lithologies

Lithology	No. of Samples	SG
Ovb	33	2.56
Qzt	44	2.86
Dac	2	2.53
QFClSch	767	2.68
Gos	491	2.56
Clay-like Gos	29	2.50
ClSerSch	1,755	2.71
Tr–Sulp	907	3.27
MPy	827	4.33
MPyMag	676	4.39
Enrch	121	4.20
QSch	608	2.68

12 DATA VERIFICATION

12.1 Verifications

Polimetal commissioned an independent consultant, AMC Consultants Pty Ltd (AMC), to undertake data compilation and verification as part of a geological and resource model update in 2018. AMC principal geologist, Chris Arnold visited the Gediktepe project on two occasions, and during the first visit spent two weeks working in the site offices. In addition to inspecting the project site and reviewing drill core from a suite of representative diamond drillholes, the visit also facilitated regular interactions with site professionals. No field or sampling operations were being undertaken at the time of the site visit, and no inspection of laboratory facilities was undertaken.

A full set of drill core photographs, collated into PDF documents, was supplied. During the geological modelling and interpretation and statistical analysis phases of work, these photographic records enabled the cross-checking observations relating to assays and logged geology. This process represents a spot-check confirmation of relationships between geology and assays, and in this way provided additional assurance concerning the validity of data.

A number of data verification activities were conducted, including the independent analyses of QA/QC data outlined in Section 11.4. In addition, a set of routine tests of database validity was completed as part of the data preparation phase for the resource estimation work; these include both specific and general tests. No matters of concern were identified.

12.2 Diamond Core vs. Reverse Circulation and Twin Comparisons

12.2.1 2018 Diamond Core vs. Reverse Circulation and Twin Comparisons

The 2018 analysis was informed by two sets of twinned drillhole information:

- Alacer shared a graphical comparison of the twin hole pairs evaluated in 2016.
- Polimetal provided a table of comparisons between twinned DD and RC data sets (Table 12.1), which included an additional two pairs of data not available in 2016, each of which penetrated both the gossan and massive pyrite mineralisation.

OreWin

Table 12.1 Diamond Core vs. Reverse Circulation Twin Comparison Statistics

BHIDs	Dist. (m)	Min. Zone	Intersections			Grades				
			From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
DRRC-001 DRD-053	3.41	MPy	6.0 3.7	17.0 16.9	11.0 13.2	1.10 0.92	39 45	0.91 1.13	0.62 0.69	3.25 3.47
DRRC-002 DRD-051	1.50	MPy	12.0 15.5	41.0 42.1	29.0 26.6	0.48 0.41	29 27	0.60 0.58	0.18 0.16	1.93 2.09
DRRC-062 DRD-142	6.18	MPy	46.0 48.2	73.0 63	27.0 14.8	1.21 1.46	39 52	0.92 0.89	0.55 0.77	3.15 4.52
DRRC-116 DRD-370	5.60	Gos	32.0 34.0	48.0 49.5	16.0 15.5	1.76 1.59	26 17	0.04 0.04	0.22 0.14	0.07 0.04
		MPy	48.0 49.5	52.0 53.0	4.0 3.5	2.53 1.06	88 32	2.27 3.17	0.17 0.08	1.75 3.70
DRRC-183 DRD-324	4.20	Gos	0 0	10.0 10.7	10.0 10.7	3.88 0.99	208 47	0.11 0.07	2.26 0.68	0.09 0.16
		MPy	10.0 10.7	13.0 13.0	3.0 2.3	3.07 2.20	88 476	2.46 2.43	0.16 0.13	0.10 0.14

Each of the pairs of twin holes were reviewed graphically and it was concluded that, overall, the statistics and graphical comparisons indicate that any differences are within acceptable bounds, particularly with respect to known variabilities in gold distributions. In one RC hole, (DRRC-062), evidence of downhole contamination relative to the twin DRD-142 was identified. This feature was not replicated in the other RC holes, and it was therefore considered to not be reflective of a consistent matter of concern. It was concluded that there was no basis for questioning the RC data referenced against the DD data.

12.3 Conclusions

The verification and data validation undertaken by independent professionals have not highlighted any issues of material concern. Consequently, the Gediktepe drilling data was concluded to be suitable as input into the evaluation of mineral resources.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The metallurgical testwork has been completed using parallel programmes for samples from each of the oxide and sulphide zones of the Gediktepe deposit. Material from the oxide zone has been tested using cyanidation for the recovery of gold and silver. The sulphide material has been assessed using sequential flotation to recover separate, marketable copper and zinc concentrates.

Testwork was undertaken from 2014 through 2015 by Resource Development Inc. (RDI; Colorado, USA), SGS (England), and Hacettepe Mineral Technologies (HMT; Ankara, Turkey) for PFS16. Further testwork was performed from 2016 through 2018 at Wardell Armstrong International (WAI; Truro, England), HMT, and ALS (Perth, Australia).

As a result of the testwork outcomes and trade-off studies, the treatment of oxide material has been changed from the crush–agglomerate–heap leach–zinc precipitation flowsheet proposed in the scoping study and PFS16 to a crush–grind–leach–CIP–elution flowsheet in PFS19.

The 2016 through 2018 sulphide testwork identified variable performance due to surface oxidation (aging effects), mineralogical and head grade variations, material type blends, and pulp chemistry conditions. An understanding of the complexity of the project geology and mineralogy, and the methods to control the metallurgical performance, continue to be investigated. Variations due to spatial location, depth in the deposit or sequence of mineralised layers, and mine schedule have yet to be completed. Associated precious metal (Au and Ag) deportment also has not been confirmed.

To assess the metallurgical performance of the sulphide flotation flowsheet, the results of locked cycle tests (LCT) have been used with additional batch roughing and cleaning tests. The data from the LCTs has been balanced using two methods: the standard method, as described in the SME handbook, and the concentrate production balance method, in which the tailing is calculated by difference between the feed and concentrates. LCT balances have been completed by the testing laboratories and independently by GR Engineering Services (GRES; Perth, Australia). The laboratory calculations and GRES SME method results have been used for prediction of the concentrate grades and recoveries.

Interpretation of the test results has been cognisant of the effect of concentrate grade used for an individual circuit on the performance of subsequent stages. For example, the zinc reporting into the pre-float and copper concentrates impacts the zinc available for recovery into zinc concentrate. Therefore, nominating a specific grade for copper in the copper concentrate that was different to that achieved in a LCT will reduce the confidence in the grades and recoveries of the zinc concentrates.

The range of data available for correlation of the effect of head grade, ore type, and spatial location will be expanded and hence improve the confidence level of the predicted performance when results from the continuing testwork has been completed. This variability testwork will be completed at ALS during 2019.

13.2 Mineralisation

From a mineralogical perspective, the Gediktepe deposit is characterised as a massive sulphide skarn, in which processes such as weathering, leaching by the acidic (pH 5.5) groundwater, and oxidation of the sulphides in the upper regions have depleted sulphur and base metals, leaving an oxide zone.

The oxide zone has been further characterised into two lithological types: a gossan, and a disseminated oxide, (the disseminated oxide has also been labelled as 'low gossan'). In addition, a light yellow(ish) layer, described as clay and of the order of 1 m wide, is present in outcrop at the southern end of the proposed pit. The minerals in this layer are finer and softer than the surrounding layers.

Four main categories have been used to describe the sulphide mineralisation:

- Massive pyrite (MPy);
- Magnetite-rich massive pyrite (MPyMag);
- Disseminated pyrite or transitional sulphide (Tr–Sulp); and
- Enriched massive pyrite (Enrch).

These various mineralogies occur in layers, lenses, or pods hosted in a chlorite–sericite schist. They tend to sequence (vertically down): gossan, disseminated oxide, magnetite-rich massive pyrite, massive pyrite, then enriched massive pyrite.

The massive pyrite and massive pyrite–magnetite may alternate and interfinger with layers of oxide zone lithologies.

The enriched massive pyrite has elevated levels of copper, lead, and zinc and occurs near the contact of the mineralisation with the host; it is generally spotty in distribution, located at the base or up the sides of mineralised zones at contacts with the schist, and more typical in the southern part of the deposit. While it is enriched in chalcopyrite, covellite and chalcocite are also present in significant amounts. Sphalerite also tends to be high.

Disseminated pyrite mineralisation, or veins of massive sulphide in the host rock, have sometimes been referred to as transitional sulphide, only because the massive sulphide mineralisation abundance is diminishing to waste grades as a result of phenomena other than post-emplacement alteration of the minerals.

13.2.1 Sample selection and location

The location of the samples used in the testwork are shown in Figure 13.1. The specific details of sample compositing from drill core intervals is reported in each of the testwork reports.

Figure 13.1 Metallurgical Drillhole Locations

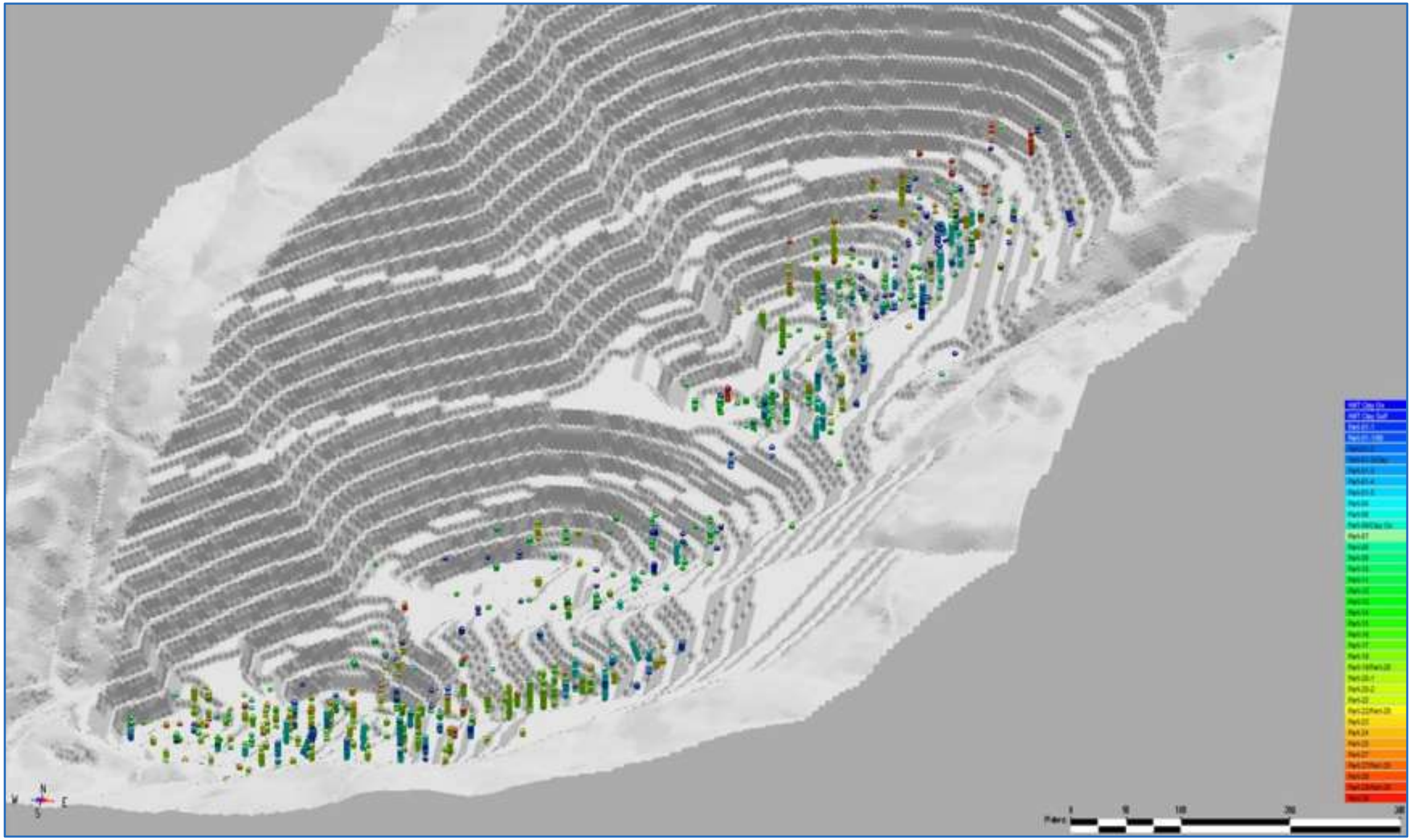


Figure from GRES, 2018.

13.2.2 Oxide Mineralogy

Drilling identified the presence of clay bands in the oxide zones. These clays comprise the minerals illite, smectite, montmorillonite, and kaolinite (HMT, March 2016). All these minerals, except kaolinite, are classified as 'swelling' clays whereby they expand when water is added. Clays impact on handling, heap leach percolation / voidage, and viscosity of slurries in processing plants. The three-stage crushing plant selected in PFS16 for the more-competent ore expected at that time, would experience issues such as: build-up of the clay material in chutes, bins, and crusher chambers; sticking of wet clays to conveyor belts; blinding of screen decks; and packing in the cone crushers, which causes tramping and damage.

The clay minerals would affect a heap leach operation in the following ways:

- Reduce percolation through clumps of agglomerated clay and other rock / particles and through filling voids between rocks;
- Increase the retention of cyanide solution within the heap, which will result in lower gold and silver extraction.

Column leach tests of composites comprising various proportions of clay were tested by SGS in 2015. Those composites with clay levels exceeding approximately 15% required cement additions averaging 20 kg/t in order to achieve stable agglomerates, which significantly slowed down the leaching kinetics and exhibited high slump. WAI performed additional test on samples in 2017 and likewise indicated some issues with slump and percolation.

The master composite used in the 2018 tank leach testwork was submitted for quantitative mineralogical analysis by Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEMSCAN) and x-ray diffraction (XRD) for mineral speciation. Trace Mineral Search–Energy Dispersive Spectroscopy (TMS–EDS) was used to identify gold. The sample was separated into a gravity concentrate and a gravity tail using a Knelson concentrator and hand-panning of the Knelson concentrate. The gravity tail was screened into five size fractions.

The gravity concentrate (unmounted) particles were examined optically using a stereo-microscope and identified the presence of coarse free gold grains. Seven free gold grains ranging in size from 100 µm to 250 µm were detected. The particles in the gravity concentrate had a D_{80} size of 148 µm and D_{50} of 88 µm.

In addition, two native gold grains were detected during the QEMSCAN trace mineral search. One of these, shown in Figure 13.2, was 7 x 3 µm in size and enclosed in a composite particle comprising quartz–goethite–(Pb,K)–Fe–sulphate. The other gold grain, which was 2 x 2 µm in size, occurred in barite ($BaSO_4$). The gold grains were analysed to be 91% Au / 9% Ag, and 97% Au / 3% Ag respectively. A 12 µm gold (electrum) grain observed by SGS (February 2016) analysed 75% Au / 25% Ag. The other gold grain found by SGS was 1 µm in size and locked within goethite. DCM Science (mineralogy for RDI in 2014) reported some 2 µm to 3 µm grains of gold in iron oxide. Note that QEMSCAN cannot detect gold grains smaller than 1 µm (colloidal gold) and gold that is in solid solution (possibly present in pyrite and goethite). SGS determined, using Dynamic Secondary Ion Mass Spectrometry (D-SIMS), that pyrite contained 1 g/t Au and arsenopyrite 6 g/t Au.

Figure 13.2 Fine Gold Detected in a Composite Particle of Goethite, Quartz, and Plumbojarosite by QEMSCAN Trace Mineral Search

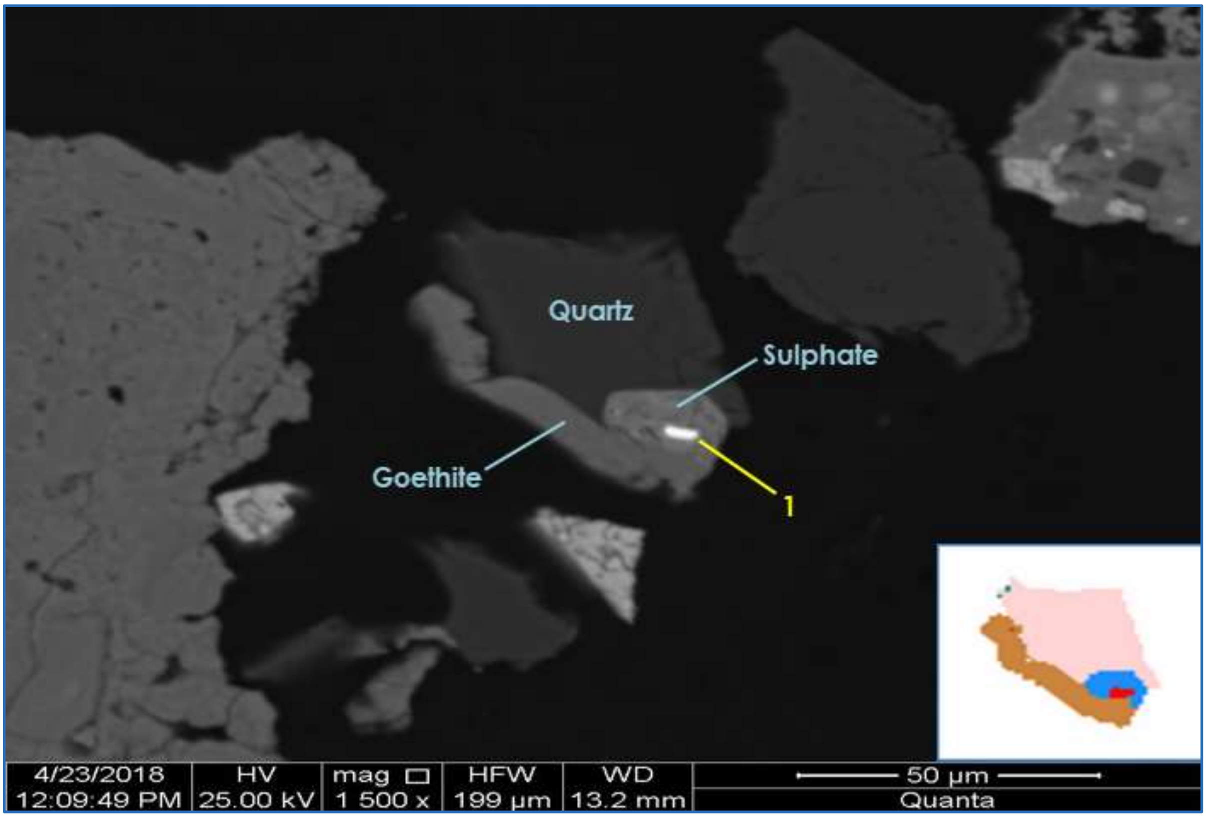


Figure from GRES, 2018. Porosity also evident in the particles.

The mass of the gravity concentrate (prepared by ALS for mineralogy) of 8% compares well with the gravity recovery testing done by RDI, which indicated 10% gravity recoverable gold. However, no coarse gold has been documented in previous testing. Identification of +100 µm gold can help explain the variability in calculated versus assay head observed in all the testwork programmes.

The ALS mineralogy indicates potential for inclusion of a gravity recovery circuit in the plant.

Silver minerals, (silver-halide (Ag–(Cl,Br,I)), and acanthite (Ag_2S)) were detected in the gravity concentrate by ALS. Two silver-halide grains approximately 80 µm in size were associated with goethite and pyrite respectively. The remaining silver minerals were fine-grained (ranging from several µm to 30 µm in size) and associated with goethite. SGS (2016) identified minor amounts of silver as cosalite, pearceite, acanthite, and marrite, which were associated with the iron oxides (one instance of acanthite coating pyrite). The silver in halides and acanthite is soluble in cyanide solution.

The bulk mineralogy results are summarised in Table 13.1. Goethite, quartz, and other silicates dominate. Optical observations by ALS indicated that pores and voids are present in goethite, (see Figure 13.3), which also contains up to 1% by weight copper – this represents

94.2% of the copper in the ALS samples as being hosted in goethite although the level of copper is below or close to the QEMSCAN detection limit and therefore this deportment for copper in goethite is indicative only. Nevertheless, it supports the extraction of only 50% of the cyanide-soluble copper (CN^{sol} Cu) in the cyanidation testwork.

The remaining copper is distributed between chalcopyrite, chalcocite, covellite, enargite–tennantite, Cu–(Fe) oxides, and Cu–Sn–Pb–Fe–(Zn) metal (in one agglomerate only). The copper minerals are fine-grained (P_{80} of 39 µm), poorly liberated, and mainly associated with pyrite and goethite. DCM also reported chalcocite rims on chalcopyrite (Figure 13.4). There appears to be more zonal or colloform banding in the grains than previous analyses.

Pyrite is the dominant sulphide in the oxide samples. In the ALS sample, 88.4% of the pyrite was classified as liberated and it had a D_{80} of 94 µm.

Other major minerals in the ALS master composite were micas (9.3%), barite (3.5%), (Pb,K)–Fe–sulphates (4.4%, also porous), rutile / ilmenite (2.1%) and hematite / magnetite (2.0%).

The iron oxide minerals (goethite, hematite, and magnetite) with a P_{80} of 72 µm were 69% liberated (58% liberated in the +106 µm fraction and 78% liberated in the –38 µm fraction). The copper minerals were finer with a P_{80} of 39 µm and lower liberation of 42%, however the coarse copper minerals (+106 µm) were 79% liberated, with 36% liberation of the –38 µm representing the copper included in pyrite and goethite.

Pitting of magnetite particles was observed, and a review of the DCM Science images indicated pitting was also present in the sample investigated in 2014.

Table 13.1 Summary of Mineral Composition for Oxide Samples

<table>
<tr><th rowspan="5">Oxide Ore Mineral Group</th><th colspan="8">Mass% in sample</th></tr>
<tr><th>ALS</th><th colspan="2">WAI (XRD)</th><th>DCM Science</th><th>SGS</th><th>DCM Science</th><th>SGS</th><th>SGS</th></tr>
<tr><th>MIN3199</th><th colspan="2" rowspan="2">XRD</th><th>Petrology</th><th>14963-001</th><th>Petrology</th><th>14963-001</th><th>14963-002</th></tr>
<tr><th>Apr'18</th><th>Apr'14</th><th>Apr'15</th><th>Apr'14</th><th>Apr'15</th><th>Dec'15</th></tr>
<tr><th>90:10 Master Comp</th><th>Gossan</th><th>Cly-like Gossan</th><th>Gossan</th><th>Gossan</th><th>Dissem. Oxide</th><th>Low Gossan</th><th>Column 6 Residue</th></tr>
<tr><td>Gold</td><td>0.02</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>Pyrite</td><td>0.43</td><td></td><td></td><td>Trace</td><td>3.66</td><td>1</td><td>0.7</td><td>2.24</td></tr>
<tr><td>Chalcocite / Covellite</td><td>0.00</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>Chalcopyrite</td><td>0.00</td><td></td><td></td><td></td><td>0.01</td><td></td><td></td><td></td></tr>
<tr><td>Other Cu</td><td>0.00</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>Arsenopyrite</td><td></td><td></td><td></td><td></td><td>0.04</td><td></td><td>0.03</td><td>0.09</td></tr>
<tr><td>Other Sulphides</td><td>0.02</td><td></td><td></td><td></td><td>0.02</td><td></td><td>0.01</td><td></td></tr>
<tr><td>Barite</td><td>3.50</td><td></td><td></td><td>14</td><td>6</td><td>3</td><td>2.34</td><td>3.5</td></tr>
<tr><td>(Pb,K)–Fe–Sulphates</td><td>4.43</td><td>4.7</td><td>3.1</td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>Hematite / Magnetite</td><td>1.99</td><td>6.4</td><td>3.5</td><td></td><td rowspan="2">55.2</td><td></td><td rowspan="2">19.7</td><td rowspan="2">31</td></tr>
<tr><td>Goethite</td><td>47.3</td><td>30.4</td><td>15.7</td><td>25</td><td>15</td></tr>
<tr><td>Goethite–Quartz Intergrowths</td><td>3.00</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>Quartz</td><td>25.1</td><td>30.2</td><td>30.9</td><td>43</td><td>22</td><td>39</td><td>38.2</td><td>36.8</td></tr>
<tr><td>Micas and Illite</td><td>9.28</td><td>15.7</td><td>23.6</td><td></td><td></td><td></td><td></td><td>13.9</td></tr>
<tr><td>Albite</td><td>0.75</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>Chlorite</td><td>0.89</td><td>1.5</td><td>1.4</td><td>Trace</td><td>1.8</td><td>11</td><td>9.4</td><td>1.35</td></tr>
<tr><td>Talc and similar</td><td>0.42</td><td></td><td></td><td></td><td>0.11</td><td></td><td>0.1</td><td>0.77</td></tr>
<tr><td>Kaolinite and Clays</td><td>0.32</td><td>Trace</td><td>1.2</td><td></td><td>0.6</td><td></td><td>1.73</td><td>1.85</td></tr>
<tr><td>Clinochlore</td><td></td><td></td><td></td><td></td><td>0.4</td><td></td><td>4.4</td><td></td></tr>
<tr><td>Muscovite</td><td></td><td></td><td></td><td>12</td><td>9.3</td><td>25</td><td>18.8</td><td></td></tr>
<tr><td>Feldspars</td><td></td><td></td><td></td><td></td><td></td><td>1</td><td>6.8</td><td>5.14</td></tr>
<tr><td>Other Silicates</td><td>0.09</td><td></td><td></td><td></td><td></td><td></td><td></td><td>0.34</td></tr>
<tr><td>Rutile / Ilmenite</td><td>2.06</td><td></td><td></td><td></td><td>0.57</td><td></td><td>0.69</td><td>0.57</td></tr>
<tr><td>Other Minerals</td><td>0.17</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>Steel</td><td>0.23</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td></tr>
<tr><td>TOTAL</td><td>100</td><td></td><td></td><td>94</td><td>99.7</td><td>95</td><td>103</td><td>97.6</td></tr>
</table>

Figure 13.3 Goethite Showing Porosity

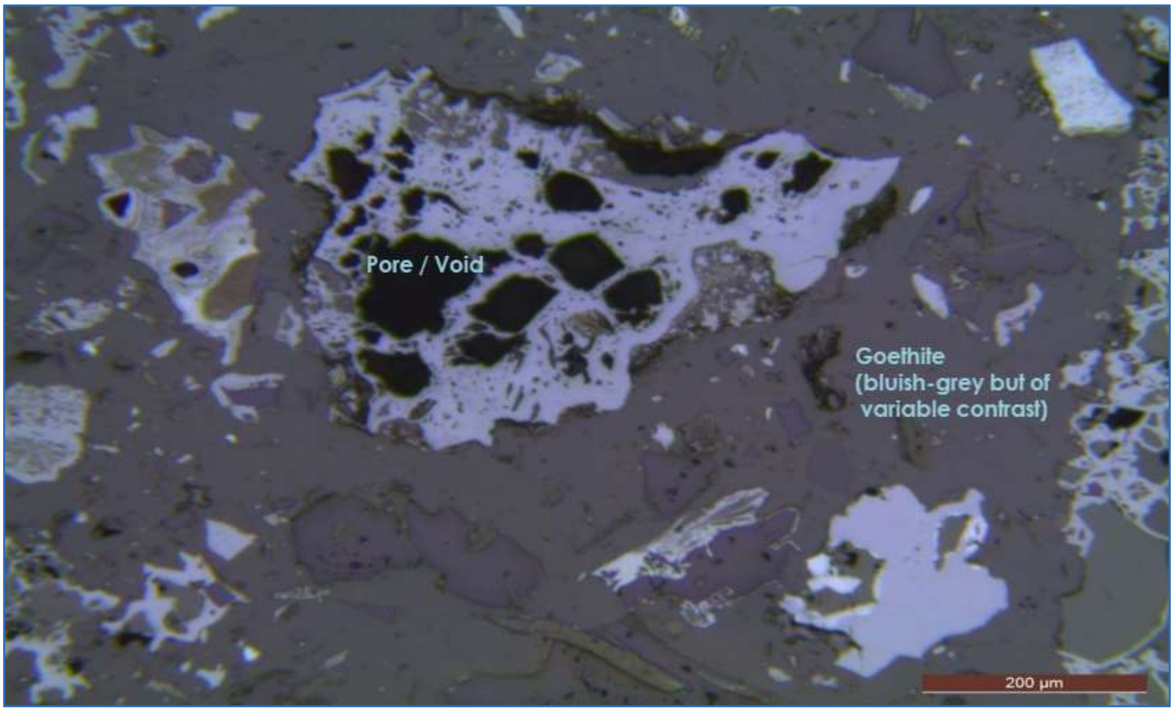


Figure from GRES, 2018.

Figure 13.4 Chalcocite Rimming of a Pyrite–Sphalerite Composite Particle

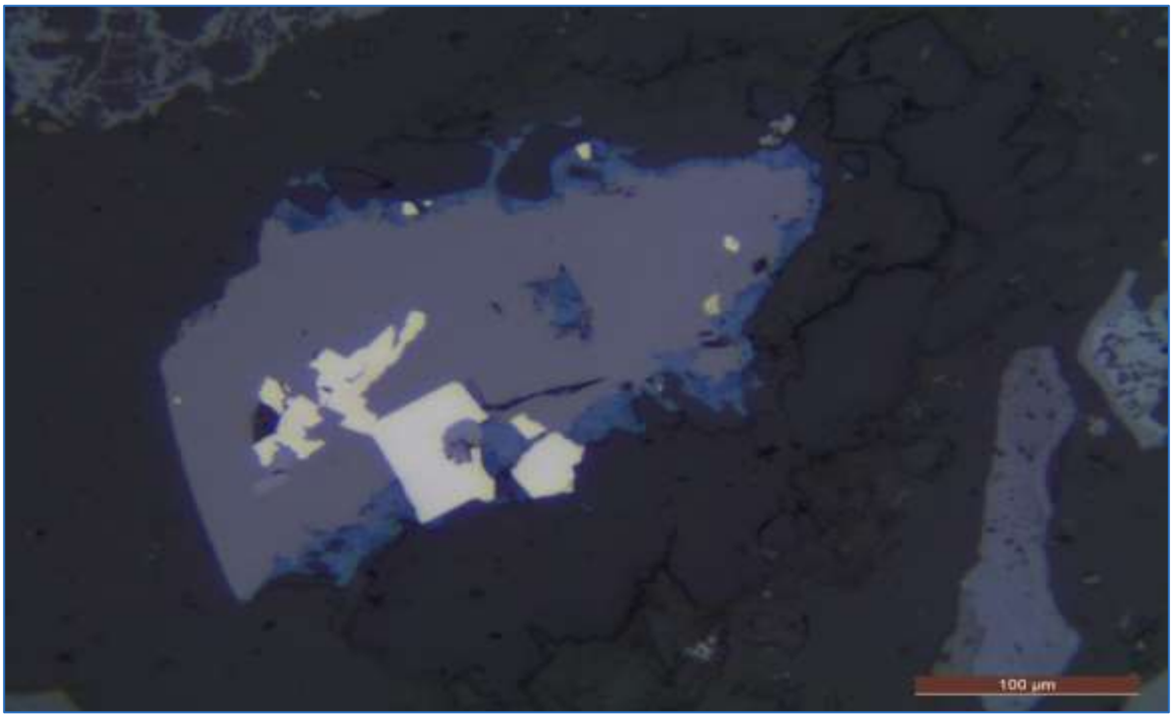


Figure from GRES, 2018.

13.2.3 Sulphide Mineralogy

The results of mineralogical analyses of sulphide samples are summarised in Table 13.2.

The mineralogy of the sulphide zones has the following impact on metallurgical performance:

- Mineral liberation – grind size: mineral grain size D_{50} is 30 µm indicating the need for fine primary grind and regrind target sizes;
- High pyrite content;
- Variable chalcopyrite, sphalerite, and galena contents and ratios;
- Presence of secondary minerals (notably secondary copper minerals); and
- Presence of naturally floating silicates (non-sulphide gangue).

Arsenic, in the form of enargite and arsenopyrite, could report as a penalty element in the copper concentrate.

The high ratio of pyrite to galena indicates the likely activation of sphalerite by lead ions that have been released by oxidation in the plant pulps (mainly due to galvanic reaction between these minerals). Consequently, analysis of solutions from EDTA extractions were completed during the testwork to measure the level of surface oxidation.

Pre-activation of sphalerite in situ by secondary copper minerals may also have occurred. Weathering and oxidation of sulphide bearing drill core stored on surface was also observed.

Any clay present in the sulphide feed will increase the viscosity of slurries and manifest as reduced efficiency in size reduction, classification, flotation, and dewatering. This will necessitate operation at lower pulp densities, thereby increasing the volumetric capacity of plant equipment and associated capital cost, and with higher mixing power intensities, thereby increasing operating costs. The 'swelling' clays will have a tendency to absorb and retain not only water but also reagents.

The presence of secondary copper minerals will result in metal ions in solution that will adversely affect flotation recovery. The ability to achieve clean separations of copper and zinc minerals to counteract these effects will necessitate the use of a broad range of reagents, resulting in high reagent consumption and costs.

Mineral liberation data shows the average grain sizes of all minerals are less than 50 µm, indicating fine grinding will be required to achieve high-grade products and improve mineral separation efficiency.

Pyrite is the dominant mineral: It can contain inclusions of magnetite, chalcopyrite, galena, and sphalerite in the coarser grains. SGS described pyrite as having a vuggy texture with deposition of other sulphide minerals in the cracks, fractures, and openings of the vugs, (SGS,2015).

Table 13.2 Summary of Mineral Composition for Sulphide Samples

Sulphide Ore Mineral Group	Mass% in sample								
	WAI	SGS	WAI	SGS	SGS	SGS	SGS	SGS	WAI
	DFS	PFS	DFS	PFS	PFS	PFS	PFS	PFS	DFS
	Petrolab	SGS	Petrolab	SGS	SGS	SGS	SGS	SGS	Petrolab
	AM2701b	15082-001	AM2660b	15082-001	15082-001	15082-001	15082-001	15082-001	AM2842b
	Jan-18	Jun-15	Nov-17	Jun-15	Jun-15	Jun-15	Jun-15	Jun-15	Jun-18
	Master Composite	Master Composite	Tr–Sulp	Tr–Sulp	Tr–Sulp Average	MPy	MPyMag	Enrch	Master Composite Blend 8
Gold									
Pyrite	80.4	56.2	54.5	56.2	24.1	78.9	75.8	86.2	64.3
Chalcocite / Covellite	< 0.1	0.2	0.2	0.13	0.2	0.3	0.1	3.0	
Chalcopyrite	1.8	1.9	0.6	1.87	1.1	1.9	3.0	2.5	1.9
Enargite	< 0.1		0.3	0.06					
Tetrahedrite									
Other Cu (Bornite)	< 0.1		< 0.1	0.05					
Sphalerite	2.5	3.2	1.2	3.24	2.3	3.8	3.2	4.1	2.7
Galena	0.1	0.4	0.3	0.38	0.5	0.3	0.4	0.1	0.3
Arsenopyrite				0.10					
Other Sulphides		0.2		0.06	0.2	0.1	0	0.8	
Barite		0.5	4.8		0.8				

Sulphide Ore Mineral Group	Mass% in sample								
	WAI	**SGS**	**WAI**	**SGS**	**SGS**	**SGS**	**SGS**	**SGS**	**WAI**
	DFS	**PFS**	**DFS**	**PFS**	**PFS**	**PFS**	**PFS**	**PFS**	**DFS**
	Petrolab	**SGS**	**Petrolab**	**SGS**	**SGS**	**SGS**	**SGS**	**SGS**	**Petrolab**
	AM2701b	**15082-001**	**AM2660b**	**15082-001**	**15082-001**	**15082-001**	**15082-001**	**15082-001**	**AM2842b**
	Jan-18	**Jun-15**	**Nov-17**	**Jun-15**	**Jun-15**	**Jun-15**	**Jun-15**	**Jun-15**	**Jun-18**
	Master Composite	**Master Composite**	**Tr–Sulp**	**Tr–Sulp**	**Tr–Sulp Average**	**MPy**	**MPyMag**	**Enrch**	**Master Composite Blend 8**
Iron Oxides	4.5	5.2	17.9	4.65	4.5	4.9	6.6	0.9	4.0
Quartz / Feldspars	2.8	14.2	15.0		30.2	2.6	1.3	1.0	8.7
Micas / Clays	0.8	5.2	1.3		10.9	0.7	0.2	0.1	3.3
Chlorite	3.8	8.2	2.6		21.6	1.8	3.5	0.3	5.2
Talc and Similar		0.8			0.4	0.4	1.9	0.1	
Biotite		0.7			1.6	0.2	0	0	
Other Silicates			1.0						0.2
Carbonates	2.1	2.6			1.3	3.3	3.5	0	3.9
Rutile / Ilmenite									
Other Minerals	0.9	0.5	0.3		0.5	0.2	0.2	0.1	0.5
Steel									
TOTAL	**99.7**	**100.0**	**100.0**	**66.74**	**100.2**	**99.4**	**99.7**	**99.2**	**95.0**

13.3 Oxide Testwork

Introduction

Testwork programmes have been completed and reported in the following:

- RDI, Metallurgical Testing of Oxide Samples from Gediktepe Prospect, Turkey. Colorado, USA; Revised Report. 13 January 2015.
- SGS, Report on Oxide Metallurgical Test Programme – Update. Project No 10866–573, Cornwall, UK. 1 February 2016.
- SGS, An Investigation into The Mineralogical Characteristics of Eight Feed Samples from Turkey. Project 14963-001 Final Report; SGS Lakefield, Canada. 4 February 2016.
- SGS, An Investigation into Gold Deportment & QEMSCAN Study on One Metallurgical Sample from The Polimetal Madencilik Copper–Zinc–Lead Deposit, Turkey. Project 14963-002 Final Report; SGS Lakefield, Canada. 18 February 2016.
- HMT, Evaluation of Clay Sections in Oxide Zone of Gediktepe Ore Deposit. March 2016.
- WAI, Gediktepe Oxide Testwork Report. ZT64-0609 R001, Report MM, Version V0.2 Draft; Cornwall, UK. 1 August 2018.
- ALS, Metallurgical Testwork conducted upon Oxide Ore Samples from the Gediktepe Gold/Silver Project. Report No A18762; Perth, Australia. September 2018.

A trade-off study comparing the heap leach flowsheet with a hybrid agitation leach–thicken–CIP–zinc precipitation flowsheet was completed, (GRES, 2017).

A further trade off study was completed by GRES (2018), recommending an all CIP circuit in place of the more complex hybrid leach–thicken–CIP–zinc precipitation flowsheet and a conventional leach-thicken–filter–zinc precipitation flowsheet.

Samples

The two mineralisation types, gossan and disseminated gossan (including low or clay-like gossans) constituted the main composite samples tested in each programme (Table 13.3). SGS, WAI, and ALS completed tests on composites that represented either spatial location or head grade variation.

Table 13.3 Oxide Master Composite Assays

Description	ALS 2018		WAI		SGS		RDI
	90% Gossan 10% ClyGoss	50:50 Gossan: ClyGoss	Gossan	ClyGoss	Gossan	Low Gossan	Master Composite
ASSAYS							
Au (g/t)	2.59	2.87	2.76	2.68	3.57	0.87	2.72
Ag (g/t)	68	75	79.3	79	44.2	29	74.3
Cu (%)	0.079	0.068	0.12	0.10	0.085	0.047	0.1208
CN sol Cu (%)	8	4.8	3.2	10	ND	ND	ND
Pb (%)	0.49	0.29	0.90	0.57	0.38	0.16	0.422
Zn (%)	0.072	0.072	0.13	0.10	0.098	0.071	0.088
As (%)	0.229	0.145	0.24	0.18	0.217	0.054	0.072
S (total) (%)	1.8	1.58	1.47	1.49	2.52	0.56	
S (sulphide) (%)			0.97	1.44			
C (organic) (%)	0.18	0.18	0.08	0.09	0.04	0.05	
Hg (ppm)	3.8	4.9			4	2	
Fe (%)	30.2	23			21.4	5.4	17.86

Variability samples for the SGS programme were selected by SGS Geostat (Canada). The WAI variability samples were blends representing ratios of gossan and clay-like gossan material. Two ALS variability samples represented grade variations and another one a 50:50 blend of gossan and clay-like gossan. Additional testing of twelve samples representing mine schedule and grade variation has been initiated but not completed.

Details of the sample compositions and drill core sources are given in the respective testwork reports.

13.3.1 Heap Leach Testwork

13.3.1.1 General

Bottle roll (coarse and fine particle size) and column cyanidation testwork was undertaken by RDI, SGS, and WAI.

Agglomeration of the material was required to overcome plugging.

13.3.1.2 Agglomeration and Percolation Tests

The conditions required to provide suitable drainage characteristics and strength of the heap or column of material were established by WAI in a 75 mm diameter column. The accepted parameters associated with suitable heap performance are a drainage or percolation rate of at least 10,000 L/m²h and a slump of less than 2% prior to tapping and 10% after tapping. Preliminary tests established suitable agglomeration using 7 kg/t cement and 1.7 kg/t lime as binding agents. The clay-like gossan sample failed the test criteria at these conditions and required 20 kg/t cement to make competent agglomerates.

SGS required high cement additions of 15–20 kg/t and 3 kg/t lime to form stable agglomerates in the percolation integrity tests completed in 2016 on samples of gossan and disseminated gossan. No fines breakthrough was recorded, and drained solutions were clear. The high cement additions were used in the column tests.

One column test was completed on a blend of north, middle, and south ores (N/M/S Blend) at a lower cement addition of 7 kg/t. Measured slump after leaching was high (up to 18.3%) and geotechnical testing by Golders of the Column 6 residue, simulating loading in a 36 m-high commercial heap, indicated an overall slump of up to 33% could occur from the combined effects of wetting, agglomeration breakdown, and heap loading.

Results of tests are summarised in Table 13.4 and Table 13.5.

Table 13.4 Summary of Agglomeration and Percolation Tests

Description	Size (mm)	Cement (kg/t)	Lime (kg/t)	Average Drainage (L/m²h)	Slump (%)	Tapped Slump (%)	Final Slump (%)
SGS – 2016							
Low Gossan	–6.3	20	3				1.6
	–6.3	30	3				1.0
	–19	15	3				1.8
Gossan	–6.3	20	3				1.8
	–19	15	3				2.6
N/M/S Blend	–19	7	1.8				17
WAI – 2018							
Clay-like Gossan 1	–19	7	1.7	770	7.0	11.0	12.7
	–19	12	1.7	3,511	7.0	7.0	11.4
	–19	15	1.7	5,137	0.5	6.2	8.8
	–19	20	1.7	16,123	0.5	2.3	2.3
	–19	20	0	20,955	0.4	1.7	4.3

Description	Size (mm)	Cement (kg/t)	Lime (kg/t)	Average Drainage (L/m²h)	Slump (%)	Tapped Slump (%)	Final Slump (%)
Trans / Oxide	–19	7	1.7	17,724	1.6	3.2	4.8
	–19	5	1.7	6,656	3.0	11.0	12.9
VAR1 5% Clay-like 95%Gossan	–19	7	1.7	13,206	1.6	7.7	
	–19	5	1.7	24,510	3.2	11.0	
VAR2 10% Clay-like 90%Gossan	–19	7	1.7	18,749	1.4	5.0	
	–19	5	1.7	11,331	4.6	11.7	
VAR3 15% Clay-like 85%Gossan	–19	7	1.7	26,190	0.7	6.7	
	–19	5	1.7	1,794	9.8	16.4	
VAR4 20% Clay-like 80%Gossan	–19	7	1.7	24,761	1.7	9.5	
	–19	5	1.7	9,128	9.9	16.7	
VAR5 50% Clay-like 50%Gossan	–19	7	1.7	17,737	2.5	10	
	–19	5	1.7	4,291	9.9	17	
	–19	10	1.7	35,170	0.3	8.5	

13.3.1.3 Column Leach Tests

Columns of 150 mm diameter by 2 m high were used at WAI (2018) to test 40 kg samples of –19 mm over a 70-day period with cyanide solution maintained at a pH of 10.5 and 0.5 mg/L NaCN, and applied at 10–12 L/m²h.

Parameters and results of the column leach tests are summarised in Table 13.5.

SGS (2016) also used 150 mm diameter columns, a leach solution application rate of 12 L/m²h, and sodium cyanide concentration maintained at 1 g/L. The column discharge solution pH was excessively high (> 12) for four of the column tests due to the high cement additions required for stable agglomerates, and resulted in significantly slower leach kinetics. Column Test 5 on a blend of 14.5% gossan, 6% low gossan, 20.3% north, 27.6% middle, and 31.5% south material failed to reach a suitable pH > 9 and consequently cyanide addition was not made and the test was cancelled. Column Test 6 on a N/M/S Blend using a lower cement addition of 7 kg/t had a pH of 10.3, which improved leach kinetics but resulted in a high slump of 17%.

Average extractions from all tests were 82% Au, 45% Ag, 8% Cu and 3% Zn. It should be noted that it is standard industry practice to discount column tests, particularly the small diameter column used in the test programme, by 3–5% to reflect scale-up to a commercial heap.

Table 13.5 Summary of Column Leach Test

Lab	Sample ID	Test Column (mm)	Size (mm)	CN (kg/t)	Lime (kg/t)	Time (h)	Cement (kg/t)	Au Assay (g/t)	Au Extractⁿ (%)	Ag Assay (g/t)	Ag Extractⁿ (%)	Cu Assay (ppm)	Cu Extractⁿ (%)	Slump (%)
RDI	35G65Diss	100	16.5	2.02	8.54	1,080	2.25	3.04	87.5	79	50.6			
RDI	35G65Diss	100	9	1.92	8.58	1,080	2.25	3.3	87.7	79.5	48.4			
RDI	DissemOx	100	19	1.55	8.72	936	2.25	0.38	92.3	18.4	66.8			
SGS	Gossan	150	–6.3	0.66	> 3	2,568	30	3.6	80.5	48.8	41.9	900	5.3	2.5
SGS	Gossan	150	–19	0.39	> 3	2,568	20	3.7	79.2	42.2	38.9	900	6.1	2.6
SGS	Low Gossan	150	–6.3	0.54	> 3	2,568	30	0.9	77.3	30.3	48.0	500	1.4	2.4
SGS	Low Gossan	150	–19	0.55	> 3	2,568	20	1.0	85.9	37.2	49.4	400	2.1	2.0
SGS	26Nth 35Mid 40Sth	150	–19	1.85	> 1.8	2,280	7	2.8	90.7	66.2	63.6	800	25.5	18.3
WAI	Trans/Ox	150	–19	2.68	1.98	1,680	7	1.65	56.5	55.5	17.1	5,347	21.6	0.5
WAI	ClyGossan	150	–19	0.54	0.16	1,680	20	1.8	91.6	50.9	17.1	617	1.3	1.7
WAI	95G5ClyG	150	–19	1.13	1.82	1,680	7	3.08	77.9	77.5	46.9	1,158	6	1.3
WAI	90G10ClyG	150	–19	1.02	1.83	1,680	7	3.12	79.1	73.3	49.1	1,142	5	1.7
WAI	85G15ClyG	150	–19	1.09	1.87	1,680	7	3.08	81.8	80.8	49.8	1,150	5.6	2.4
WAI	80G20ClyG	150	–19	1.09	1.85	1,680	7	3.06	82.2	79.6	48.3	1,152	5.4	2.5
WAI	50G50ClyG	150	–19	0.84	1.81	1,680	10	2.97	82.4	77.5	43.5	1,077	6.5	6.6
Average				**1.19**	**3.72**	**1,827**	**11.9**	**2.5**	**82.2**	**60**	**45.3**	**1,262**	**7.7**	**3.7**
Median				**1.09**	**1.86**	**1,680**	**7**	**3.04**	**82.2**	**66**	**48.3**	**989**	**5.5**	**2.4**
Standard Deviation				**0.67**	**3.42**	**565**	**9.6**	**1.06**	**8.7**	**21**	**13.5**	**1,314**	**7.7**	**4.8**

13.3.1.4 Coarse Bottle Roll Leach Tests

Bottle roll tests on coarse particle size distributions (D_{80} > 1 mm) were used to indicate the maximum extractions achievable from a heap leach operation and provide relative performance of samples in place of column leach tests. The RDI, SGS, and WAI programmes included coarse bottle roll tests and results are summarised in Table 13.6.

The results of the bottle roll tests are similar to those achieved in the column tests, reflecting the porous nature of the material observed in the ALS fine feed testwork.

Extraction of copper into the leach solution for the variability samples tested by SGS were high and indicate that methods for mitigating the interference in recovering gold and silver will need to be implemented.

Table 13.6 Summary of Coarse Bottle Roll Leach Tests

Lab	Sample ID	Size (mm)	Solids (%)	CN (kg/t)	Lime (kg/t)	Time (h)	Au Assay (g/t)	Au Extractn (%)	Ag Assay (g/t)	Ag Extractn (%)	Cu Assay (ppm)	Cu Extractn (%)
RDI T17	35G65Diss	–19	40	0.78	6.08	96	2.71	79.0	59.6	49.3		
RDI T16	35G65Diss	–12.5	40	0.42	8.17	96	2.11	88.5	66.4	41.6		
RDI T18	35G65Diss	–12.5	40	0.91	6.96	96	3.38	79.3	77.9	48.2		
RDI T19	35G65Diss	–6.35	40	1.03	7.20	96	3.03	87.3	92.2	42.3		
RDI T6	35G65Diss	–3.35	40	1.56	4.97	72	2.43	83.3	62.5	53.6		
SGS	Gossan	–25	45	1.77	1.43	336	3.52	77.4	41.6	34.2	840	6.1
SGS	Gossan	–19	45	1.74	1.38	336	3.7	79.9	42.3	40.2	710	6.8
SGS	Gossan	–16	45	1.76	1.45	336	3.88	79.5	40.2	37.9	790	5.5
SGS	Gossan	–12.5	45	1.05	2.45	336	3.98	80.5	44.6	35.7	750	5.3
SGS	Gossan	–6.3	45	1.16	2.67	336	3.85	86.9	45.7	37.6	750	7.2
SGS	Gossan	–3.35	45	1.05	2.81	336	3.92	85.6	42.5	42.4	770	7.4
SGS	LowGossan	–25	45	1.65	0.91	336	1.01	84.2	34.4	36.9	430	6.8
SGS	LowGossan	–19	45	1.45	0.97	336	0.92	85.5	28.0	40.9	470	6.0
SGS	LowGossan	–16	45	0.81	1.65	336	1.16	85.7	35.0	41.2	470	7.4
SGS	LowGossan	–12.5	45	0.87	1.57	336	1.00	84.5	32.4	43.7	470	6.7
SGS	LowGossan	–6.3	45	0.83	1.78	336	1.05	84.8	29.7	51.0	460	5.2
SGS	LowGossan	–3.35	45	0.85	1.76	336	1.07	81.7	32.2	53.3	480	5.2
SGS	High Au	–19	45	1.22	3.2	1,008	8.65	89.1	184.0	47.9	989	27.6
SGS	High BM	–19	45	2.04	1.68	1,008	1.47	83.6	131.0	30.0	2,545	29.9
SGS	Low BM	–19	45	0.65	1.9	1,008	1.09	90.9	15.0	33.7	278	6.5
SGS	Middle	–19	45	2.53	2.23	1,008	2.09	94.5	70.0	74.1	873	27.7
SGS	North	–19	45	2.24	2.03	1,008	2.78	88.4	89.0	59.5	496	18.7

Lab	Sample ID	Size (mm)	Solids (%)	CN (kg/t)	Lime (kg/t)	Time (h)	Au Assay (g/t)	Au Extract^n (%)	Ag Assay (g/t)	Ag Extract^n (%)	Cu Assay (ppm)	Cu Extract^n (%)
SGS	South	–19	45	2.52	2.6	1,008	4.71	87.2	215.0	69.6	884	36.1
WAI	95G5ClyG	–19	40	1.28	2.68	72	2.78	74	74.2	41.3	1,200	5.1
WAI	90G10ClyG	–19	40	1.28	2.5	72	2.24	76.7	63.2	37	1,100	4.6
WAI	85G15ClyG	–19	40	1.13	2.66	72	2.9	75.3	72.9	41.2	1,200	5.4
WAI	80G20ClyG	–19	40	1.01	2.79	72	2.48	78.3	65.5	37.4	1,200	5.7
WAI	50G50ClyG	–19	40	0.97	2.72	72	2.83	79.2	75.4	37.4	1,000	4.6
Average				**1.31**	**2.9**	**389**	**2.74**	**83.2**	**67**	**44.3**	**833**	**10.8**
Median				**1.15**	**2.5**	**336**	**2.75**	**83.9**	**61**	**41.3**	**770**	**6.5**
Standard Deviation				**0.6**	**1.9**	**348**	**1.60**	**5**	**45**	**10.3**	**464**	**9.7**

13.3.1.5 Zinc Cementation Tests

The 5 kg bulk leach test completed by RDI produced a pregnant solution assaying 1.78 ppm Au, 43.9 ppm Ag, and 339 ppm Cu indicating that processes and/or conditions would be needed to avoid high-copper content doré, which attracts higher refining charges (or can lead to rejection by a mint).

Zinc precipitation tests by SGS and ALS extracted 98.6% of the gold and 98.7% of the silver from the pregnant solution for tests using a 20:1 stoichiometric addition of zinc dust. The testwork at ALS showed that increasing the zinc addition ratio resulted in increased extraction of copper with gold and silver into the precipitate, therefore the amount of copper co-precipitated could be controlled to some extent by controlling the addition of zinc. A stoichiometric zinc to gold ratio of 7.5:1 plus silver content was determined to be the minimum addition to ensure extraction of gold and silver by zinc precipitation was above 97%. Results are summarised in Table 13.7.

Due to limited solution volume, a fixed lead nitrate ($Pb(NO_3)_2$) addition, cyanide concentration and pH were used in the ALS tests. The pregnant solutions also contained zinc (up to 15 ppm Zn in the ALS solution and 17.2 ppm Zn in the SGS solution), iron (3.5 ppm Fe; ALS) and mercury (0.2 ppm Hg; SGS). The pregnant solution generated from the high-grade sample in the ALS tests had a lower copper content than the master composite tested.

Table 13.7 Summary of Zinc Cementation Test Results

Lab	Feed Solutions			pH	Zinc Addition	Total CN	Lead Nitrate Addition	Cement-ation Time	Extraction		
	Au (g/t)	Ag (g/t)	Cu (ppm)		Zn/ (Au+Ag)	(ppm)	% of Zn wt.	(min.)	Au (%)	Ag (%)	Cu (%)
RDI	1.78	43.9	339								
SGS	1.48	19.1	130	11	20:1	374	25	60	98.6	98.7	3.1
ALS	2.16	54.6	21.8	10.5	5:1	1,000	14	60	42.1	54.9	15.6
					8:1	1,000	9	60	97.5	98.9	46.8
					10:1	1,000	7	60	99.8	99.8	66.6
					15:1	1,000	4.7	60	98.8	99.5	66.9
					20:1	1,000	3.5	60	100	99.8	56.8
					7.5:1	1,000	9.5	60	99.9	99.9	62.0
ALS	2.72	34.4	5.54	10.5	2.5:1	1,000	44	60	62.1	84.9	6.1
					5:1	1,000	22	60	93.4	98.0	3.2
					10:1	1,000	11	60	97.4	98.5	13.7
					15:1	1,000	7.3	60	97.1	98.0	43.7
					20:1	1,000	5.5	60	98.2	98.8	32.1

Removal of Copper and Zinc from Barren Liquor

Subsequent to the ALS zinc precipitation tests, the barren liquor was subjected to sulphide precipitation testing to remove copper and zinc from the solution that would be recycled in the plant process water (to avoid build-up and interference of these metal ions in the process). Both sodium hydrosulphide (NaHS) and sodium sulphide (Na_2S) were tested by ALS.

The results in Table 13.8 show that NaHS removed 97% of the copper and zinc from the barren solution at 300% stoichiometric levels compared to 58% of the copper and 20% of the zinc when using Na_2S.

Table 13.8 Summary of Sulphide Precipitation Tests at ALS

Feed Solutions				pH	NaHS Addition	Na_2S Addition	Eh Start	Eh End	Precipit[n] Time	Extraction			
Cu (ppm)	Zn (ppm)	Au (g/t)	Ag (g/t)		S_2/(Cu+Zn)	S_2/(Cu+Zn)	(mV)	(mV)	(min.)	Cu (%)	Zn (%)	Fe (%)	Pb (%)
9.2	72.6	0	0.05	2.70	100		276	–40	30	73.9	16.5	0	0
				2.74	125		283	–58	30	73.9	20.7	0	0
				2.63	150		264	–25	30	47.8	16.3	0	0
				2.70	200		277	–67	30	98.9	58.1	0	0
				3.16	200		361	–114	60	99.3	87.8	40	60
				4.91	300			–105	60	97.2	99.6	0	80
				2.65		100			30	58.7	9.1	0	0
				2.89		125	289	–35	30	54.3	20.4	0	0
				2.81		150			30	58.7	23.7	0	0

Copper Levels in Pregnant Solution

The copper levels in the leach liquor produced in the RDI (339 ppm Cu) and SGS (130 ppm Cu) tests, were significantly higher than the levels in the ALS tests (21.8 ppm Cu). This is a function of the cyanide-soluble copper (CN^{sol} Cu) in the feed samples used in the tests. To establish the range of cyanide-soluble assays within the oxide material, a total of 447 drillhole intercepts were assayed for CN^{sol} Cu. The distribution of values, as shown in Table 13.9, indicates that 88.5% of the samples contained < 10% of the copper present in the sample as soluble in cyanide and is close to the values observed in the ALS tests.

Table 13.9 Summary of Cyanide-Soluble Assays (Oxide Material)

CN^{sol} Cu Recovery Ranges (%)	No. Samples	CN^{sol} Cu Average Recovery (%)	Cu Grade (%)	Ratio (%)
< 10	396	2.45	0.12	88.59
10–20	23	14.11	0.26	5.15
20–40	19	26.55	0.21	4.25
40–60	5	51.85	0.33	1.12
≥ 60	4	65.14	0.38	0.89

13.3.2 Agitated (Tank) Leach Testwork (on finely ground samples)

Cyanidation testwork of finely ground material (< 212 μm) was conducted at RDI, SGS, and reported in PFS16, and more recently at WAI and ALS. The testwork included, comminution, agitated leaching, recovery of gold and silver from leach liquors, removal of base metals, and cyanide destruction from effluents.

13.3.2.1 Comminution

Breakage parameters used for design of the comminution circuit were measured in the RDI and ALS test programmes and results are summarised in Table 13.10. All samples indicated the oxide material is very soft – the disseminated gossan had the hardest SMC breakage parameters. Further comminution testing on variability samples are yet to be completed.

Agitated Leach tests

The RDI, SGS and WAI cyanidation tests on samples ground to < 212 μm were performed as Bottle Roll tests (BRT). The ALS test programme was performed in an agitated tank because it is considered that the agitated tank reflects plant operation more so than bottle rolls, especially for reagent consumptions. The ALS tests achieved slightly higher extractions than the other test programmes – oxygen was added to maintain 15–20 ppm of dissolved oxygen in the leach pulp and although a test relying on air addition (8 ppm dissolved oxygen) showed no decrease in extractions, oxygen addition has been recommended.

Test results are summarised in Table 13.11.

A reddish precipitate was noted in the filtrate solution of assay samples during the ALS testwork. This colloidal material was identified using XRD by ALS to comprise 25%–33% goethite, 20%–24% quartz, 8%–22% mica, 5%–21% calcite, 4%–9% jarosite, and 6% barite. Observation of activated carbon under the microscope showed this colloidal matter enters the pores of the carbon.

Table 13.10 Summary of Oxide Sample Breakage Parameters

Composite	Avg. Density	CWi	Rod Mill Work Index @1,180 µm			Ball Mill Work Index					SMC Parameters							
		kWh/t	F_{80} (µm)	P_{80} (µm)	kWh/t	F_{80} (µm)	P_{80} (µm)	g/rev	kWh/t	Closing Screen	A	b	A*b	ta	DWi (kWh /m³)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)
Gossan	2.85	7.4				2,190	110	2.354	10.43	150	77.4	1.8	139	1.27	2.05	7.1	4.2	2.2
Gossan	2.78		7,864	900	11.5	2,577	82	1.774	11.60	106	70.1	1.71	120	1.12	2.32	8.1	4.8	2.5
Dissem/Low Gossan	2.73					1,924	113	2.703	9.69	150	67.9	1.52	103	2.73	2.64	9.1	5.6	2.9

Table 13.11 Summary of Fine Grind Agitation Leach Tests

Lab	Sample ID	Test	Grind	% Solids	CN (kg/t)	Lime (kg/t)	Lead Nitrate (g/t)	Time (h)	Au Assay (g/t)	Au Extractn (%)	Ag Assay (g/t)	Ag Extractn (%)	Cu Assay (ppm)	Cu Extractn (%)
RDI T1	35G65Diss	BRT	208	40	1.37	5.60	0	48	2.43	86.3	65.8	58.7		
RDI T2	35G65Diss	BRT	150	40	1.38	5.60	0	48	2.41	87.6	69	58.5		
RDI T7	35G65Diss	BRT	150	40	1.20	2.63	100	48	2.52	88.1	65.5	65.2		
RDI T3	35G65Diss	BRT	104	40	1.80	5.50	0	48	2.52	88.5	66.2	61.2		
RDI T4	35G65Diss	BRT	74	40	1.38	5.60	0	48	2.67	89.3	76.5	64.9		
RDI T10	35G65Diss	BRT	150	45	1.62	5.58	0	48	2.43	88.0	62.4	63.1		
RDI T11	35G65Diss	BRT	150	50	1.47	5.43	0	48	2.21	86.0	63.0	62.3		
RDI T12	35G65Diss	BRT	150	40	1.26	5.59	0	48	2.38	81.4	65.0	52.4		
RDI T13	35G65Diss	BRT	150	40	1.46	5.19	0	48	2.52	85.2	65.0	63.3		

Lab	Sample ID	Test	Grind	% Solids	CN (kg/t)	Lime (kg/t)	Lead Nitrate (g/t)	Time (h)	Au Assay (g/t)	Au Extract[n] (%)	Ag Assay (g/t)	Ag Extract[n] (%)	Cu Assay (ppm)	Cu Extract[n] (%)
RDI T12	35G65Diss	BRT	150	40	1.13	5.53	0	48	2.87	52.9	65.0	9.5		
RDI T12	35G65Diss	BRT	150	40	1.26	5.72	0	48	2.58	73.7	64.0	36.0		
RDI T22	35G65Diss	BRT	74	40	1.62	8.14	0	48	3.14	86.4	80.2	50.2		
RDI T23	35G65Diss	BRT	74	40	1.02	4.24	100	48	3.20	86.2	75.5	52.5		
RDI Bulk 5kg	35G65Diss	BRT		45						73.4		62.1		33.4
SGS T21	Gossan	BRT	106	40	1.88	3.50	0	48	3.63	88.8	40.2	51.1	800	7.0
SGS T22	Low Gossan	BRT	106	40	1.57	2.69	0	48	1.08	86.6	29.5	72.0	500	7.2
SGS T23	High Gold	BRT	106	40	2.23	3.24	0	48	11.21	90.7	166	54.2	1,100	23.9
SGS T24	High Base Metals	BRT	106	40	5.07	1.81	0	48	1.53	89.9	111	70.4	3,300	38.7
SGS T25	Low Base Metals	BRT	106	40	1.70	1.30	0	48	1.39	95.0	13.9	47.7	300	7.1
SGS T26	Middle	BRT	106	40	2.16	3.54	0	48	2.01	96.0	32.3	75.3	1,000	19.3
SGS T27	North	BRT	106	40	1.87	2.05	0	48	2.22	92.6	58	78.4	600	20.0
SGS T28	South	BRT	106	40	2.15	2.08	0	48	3.56	88.1	114	72.8	1,184	53.6
WAI	90G10ClyG	BRT	210	40	1.09	4.92	100	48	3.12	87.5	77.9	62.2	1,200	8.0
WAI	90G10ClyG	BRT	150	40	1.24	5.27	100	48	3.13	87.7	76.9	62.7	1,200	8.8
WAI	90G10ClyG	BRT	106	40	1.54	5.43	100	48	3.16	88.4	76.3	63.3	1,200	9.1
WAI	90G10ClyG	BRT	74	40	2.10	5.48	100	48	3.18	89.1	76.9	63.6	1,200	11.8
ALS	90G10ClyG	AgiTank	106	40	1.78	6.54	0	48	2.99	92.0	81.4	80.3	821	3.8
ALS	90G10ClyG	AgiTank	125	40	2.52	6.35	0	48	3.04	93.3	76.2	79.0	884	3.7
ALS	90G10ClyG	AgiTank	150	40	2.01	6.58	0	48	2.79	92.6	77.3	76.7	813	3.4
ALS	90G10ClyG	AgiTank	106	40	3.02	6.40	0	48	2.81	92.5	75.0	78.7	812	3.5

Lab	Sample ID	Test	Grind	% Solids	CN (kg/t)	Lime (kg/t)	Lead Nitrate (g/t)	Time (h)	Au Assay (g/t)	Au Extract^n (%)	Ag Assay (g/t)	Ag Extract^n (%)	Cu Assay (ppm)	Cu Extract^n (%)
ALS	90G10ClyG	AgiTank	106	40	2.70	5.76	100	48	2.72	92.6	76.6	79.1	828	3.6
ALS	90G10ClyG	AgiTank	106	40	2.93	5.82	250	48	2.98	93.3	78.3	79.6	829	3.7
ALS	90G10ClyG	AgiTank	125	40	2.48	5.57	100	48	2.75	93.4	75.3	78.7	864	3.6
ALS	90G10ClyG	AgiTank	125	40	3.77	5.50	100	48	3.08	92.5	79.9	75.0	952	3.9
ALS	90G10ClyG	AgiTank	125	45	2.75	5.51	100	48	2.73	91.3	81.3	72.9	886	3.7
ALS	90G10ClyG	AgiTank	125	50	2.38	6.01	100	48	2.79	91.9	76.2	79.8	790	3.8
ALS	90G10ClyG	AgiTank	125	55	3.14	5.95	100	48	2.64	92.2	76.6	73.9	833	4.0
ALS – CIP	90G10ClyG	AgiTank	125	45	2.44	5.67	100	24 + 48	3.27	93.3	58.0	69.0	794	3.7
ALS 30 kg Bulk	90G10ClyG	AgiTank	125	45	0.88	6.40	100	24	2.86	92.5	87.7	76.1	871	3.1
ALS 20 kg Bulk	90G10ClyG	AgiTank	125	45	2.42	5.41	100	48	2.94	93.0	84.0	78.6	830	3.9
ALS	50G50ClyG	AgiTank	125	45	2.10	3.88	100	48	3.08	92.5	84.8	76.4	714	4.2
ALS	Low Grade	AgiTank	125	45	3.06	7.18	100	48	1.38	95.7	48.6	71.2	688	1.2
ALS	High Grade	AgiTank	125	45	1.49	3.68	100	48	3.59	92.5	55.0	76.4	1520	0.4
Overall Median			**125**	**40**	**1.8**	**5.5**	**0**	**48**	**2.79**	**89.9**	**75**	**69.0**	**832**	**3.9**
ALS Only Median			**125**	**43**	**2.46**	**5.8**	**100**	**48**	**2.90**	**92.6**	**77**	**77.6**	**829**	**3.7**

13.3.2.2 Grind Size Effects

The effect of grind size on leach extraction was investigated in each programme, except the 2016 SGS work. Good correlations, with the expected decrease in extraction as grind size increases, are shown for gold in the data from RDI and WAI, and for silver in the data from WAI and ALS, (see Table 13.11, Figure 13.5, and Figure 13.6). The ALS data for gold indicates no effect of grind size on gold extraction, which may be due to the high porosity measured in the ALS samples or that other conditions have more impact on cyanidation than grind size.

The trends show that gold extraction is not particularly sensitive to grind size with a 25 µm increase in grind size resulting in a gold extraction loss of 0.5% and a silver loss of between 0.3% and 2%.

Based on the low sensitivity to grind size, a target grind P_{80} of 125 µm was chosen to complete the remainder of the oxide leach testing at ALS.

Figure 13.5 Grind Size Effect on Gold Extraction

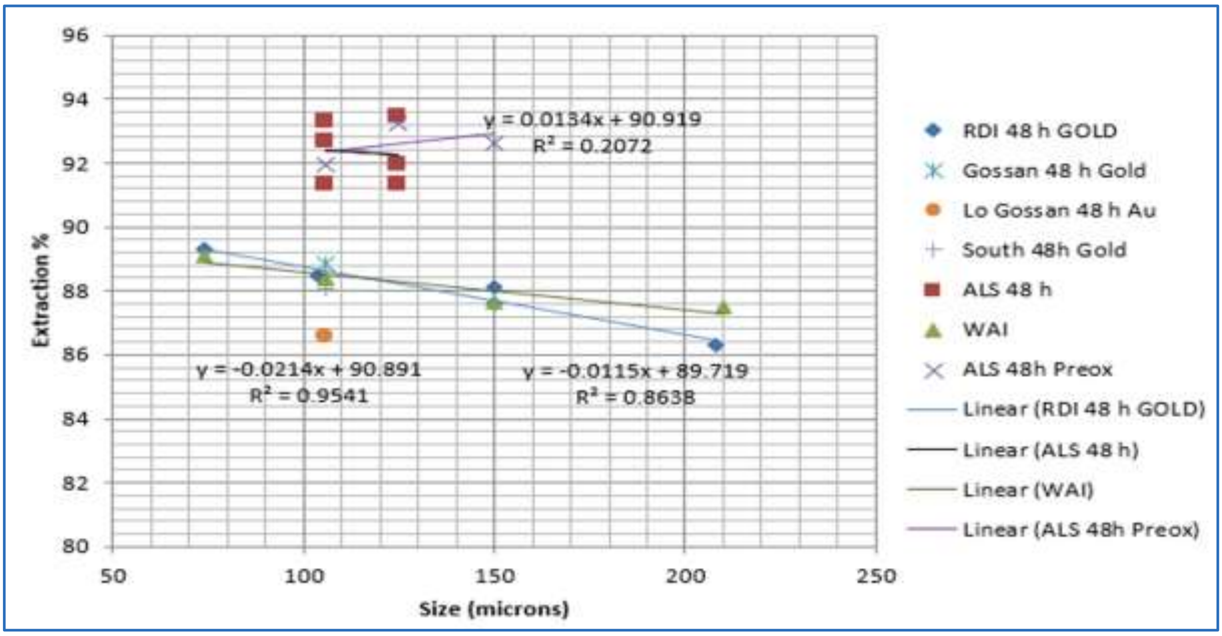


Figure from GRES, 2019.

Figure 13.6 Grind Size Effect on Silver Extraction

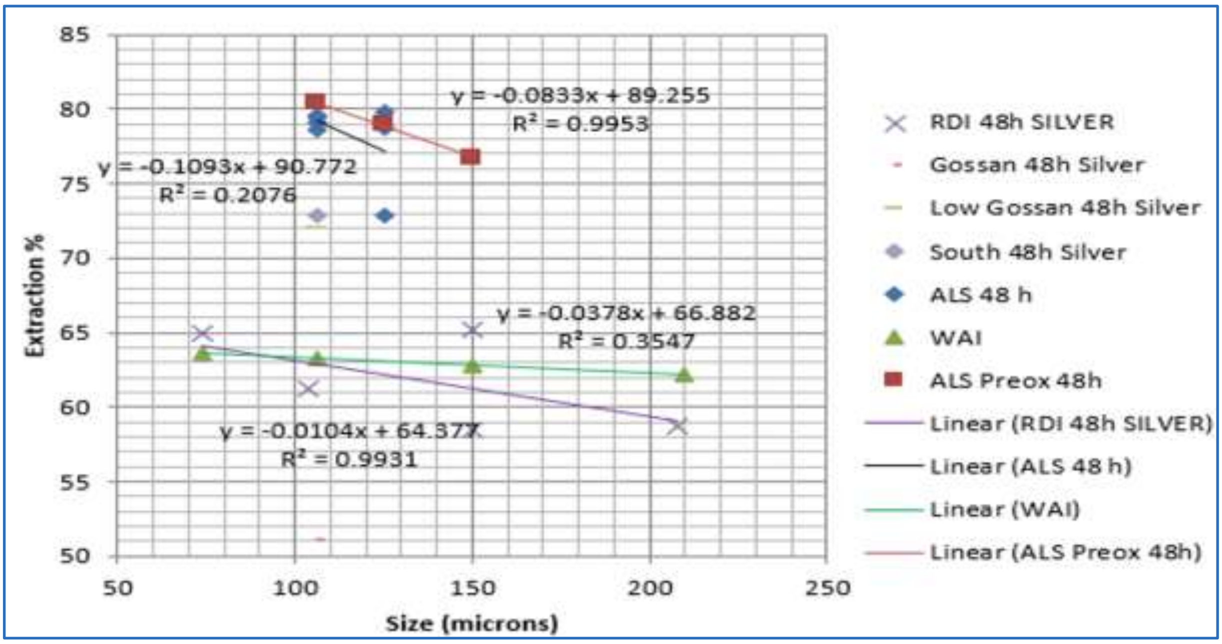


Figure from GRES, 2019.

13.3.2.3 Leach Time – CIP / CIL Tests

RDI concluded that the majority of the precious metals had been extracted into solution by cyanide within 48 hours – leach times up to 96 hours were used.

The SGS tests were carried out with 48 hours residence time, while the WAI tests ran for 72 hours.

ALS tests ranged from 24–72 hours and the majority of the tests indicated that the leaching of gold is completed by 24 hours (Figure 13.7) . Leaching of silver is somewhat slower (Figure 13.8).

Figure 13.7 Leach Kinetics – Gold

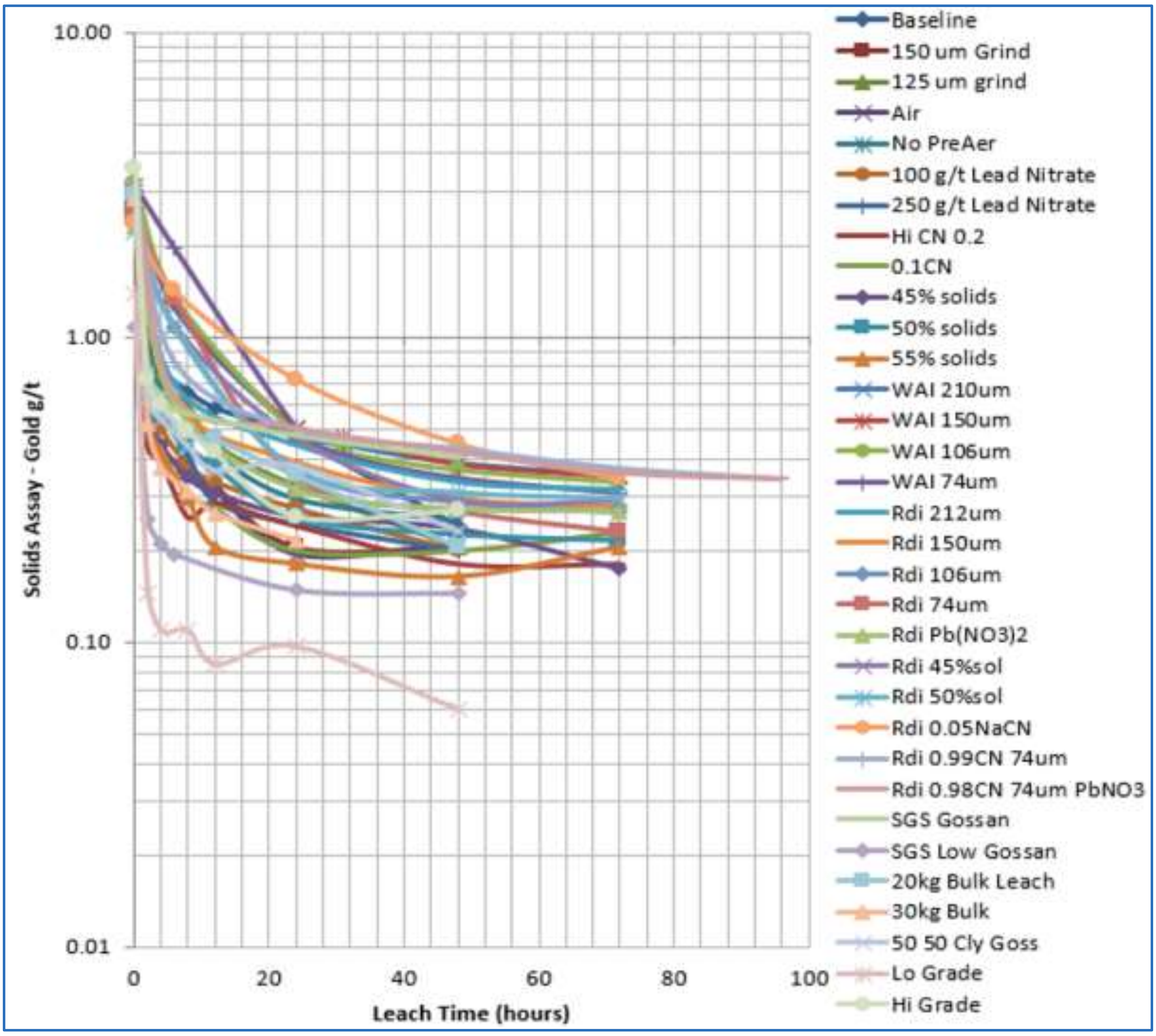


Figure from GRES, 2019.

Figure 13.8 Leach Kinetics – Silver

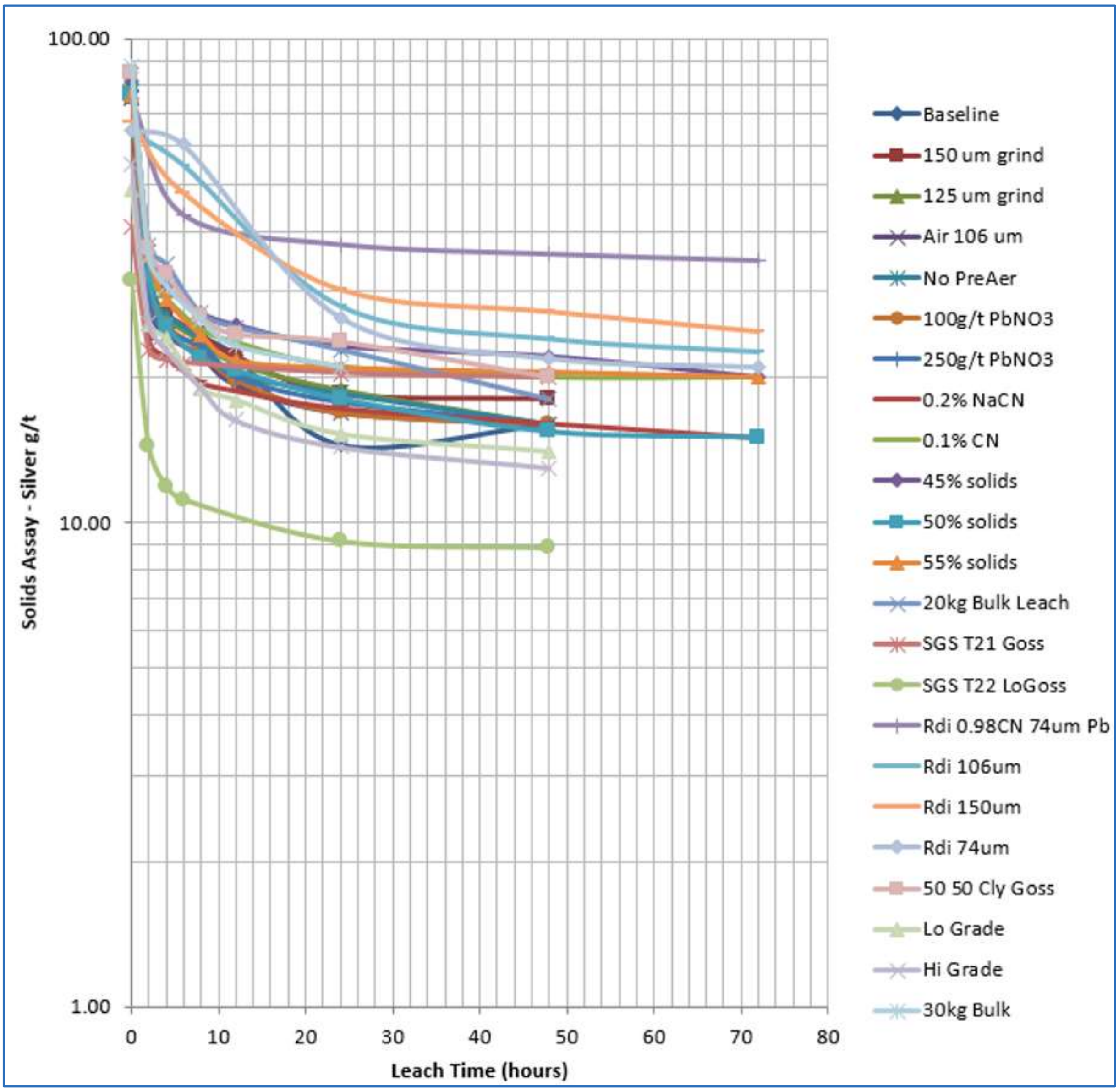


Figure from GRES, 2019.

Pulp Density

Tests by RDI and ALS both indicated little variation in extraction of gold or silver as pulp density varied. These results are summarised in Table 13.12.

Table 13.12 Summary of 48-Hour Cyanide Extractions vs. Pulp Density

Lab	Pulp Density % solids	Gold Extraction (%)	Silver Extraction (%)	Cyanide Consumption (kg/t)	Lime Consumption (kg/t)
RDI	35	87.1	64.1	1.58	4.7
	40	87.6	58.5	1.38	5.6
	45	88.0	61.5	1.56	5.6
	50	86.0	60.4	1.43	5.4
ALS	40	92.5	75.0	3.77	5.5
	45	91.3	72.9	2.75	5.5
	50	91.9	79.8	2.38	6.0
	55	93.8	73.3	2.60	6.0

The testwork in all programmes indicated no preg-robbing.

13.3.2.4 Testing of Hybrid (Leach–Zinc Precipitation–CIP) Circuit

To simulate a hybrid leach–zinc precipitation–CIP circuit, ALS completed bulk tests with a 12-hour leach stage, thickening / decanting to 65% solids to provide 'pregnant' solution for zinc precipitation testing, and 48 hours CIP stage of the solids re-pulped to 45% solids density. A range of carbon concentrations indicated that higher loadings of copper and zinc resulted from higher carbon concentrations. The gold loaded within two hours, while silver adsorption kinetics were slower, requiring 24 hours to achieve 90% adsorption. High iron, copper, zinc, and mercury loadings are also achieved, as shown in Table 13.13.

Table 13.13 Summary of Loaded Carbon Assays – ALS CIP Tests

Carbon Concentration (g/L)	Au (g/t)	Ag (g/t)	Cu (ppm)	Fe (ppm)	Zn (ppm)	Hg (ppm)
Feed Solution	1.12	27.2	12.8	1,000	7.8	0.79
3	273	6,098	128	7,910	176	112
6	117	2,766	162	3,190	211	77
9	81	1,898	180	3,615	285	61
12	60	1,436	210	2,950	309	44
15	48	1,030	204	2,405	266	36

Acid Wash Tests

Acid wash tests were completed using 3% hydrochloric acid or 3% nitric acid. Although nitric acid removed less calcium, magnesium, and iron, it removed more copper and zinc but less silver than the hydrochloric acid wash.

13.3.2.5 Reagent Conditions

Conclusions from testwork completed by RDI and ALS include:

- Concentration of sodium cyanide affects the extraction rates:
  - Lower NaCN concentration resulted in lower gold and silver extractions.
  - When NaCN concentration was allowed to decay (not maintained at initial levels), gold and silver extractions were significantly lower but with little change in cyanide consumption.
- Addition of lead nitrate $Pb(NO_3)_2$ increased silver extraction by up to 6% and reduced cyanide and lime consumption by 10% in the RDI tests but had no effect on silver extraction or cyanide consumption in the ALS tests although gold kinetics and extraction improved.

Graphs of leach kinetics for gold and silver versus NaCN and lead nitrate levels are shown in Figure 13.9, Figure 13.10, and Figure 13.11.

Figure 13.9 Sodium Cyanide Effect on Gold Extraction and Kinetics

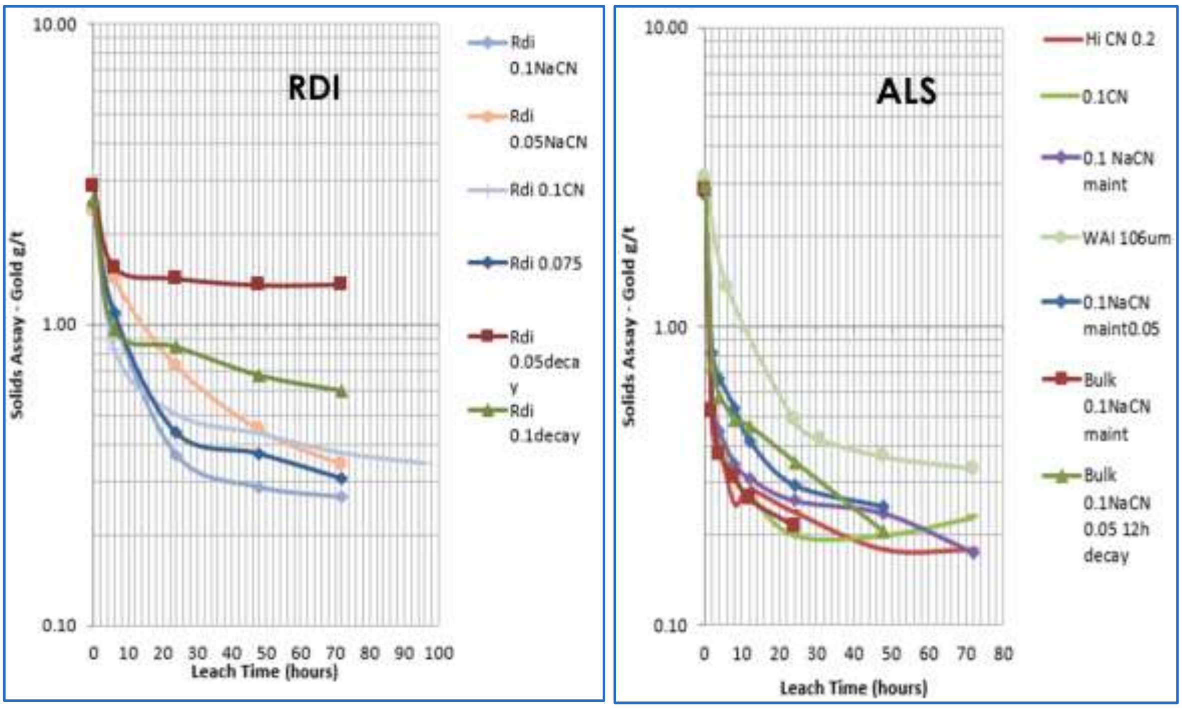


Figure from GRES, 2019.

Figure 13.10 Sodium Cyanide Effect on Silver Extraction and Kinetics

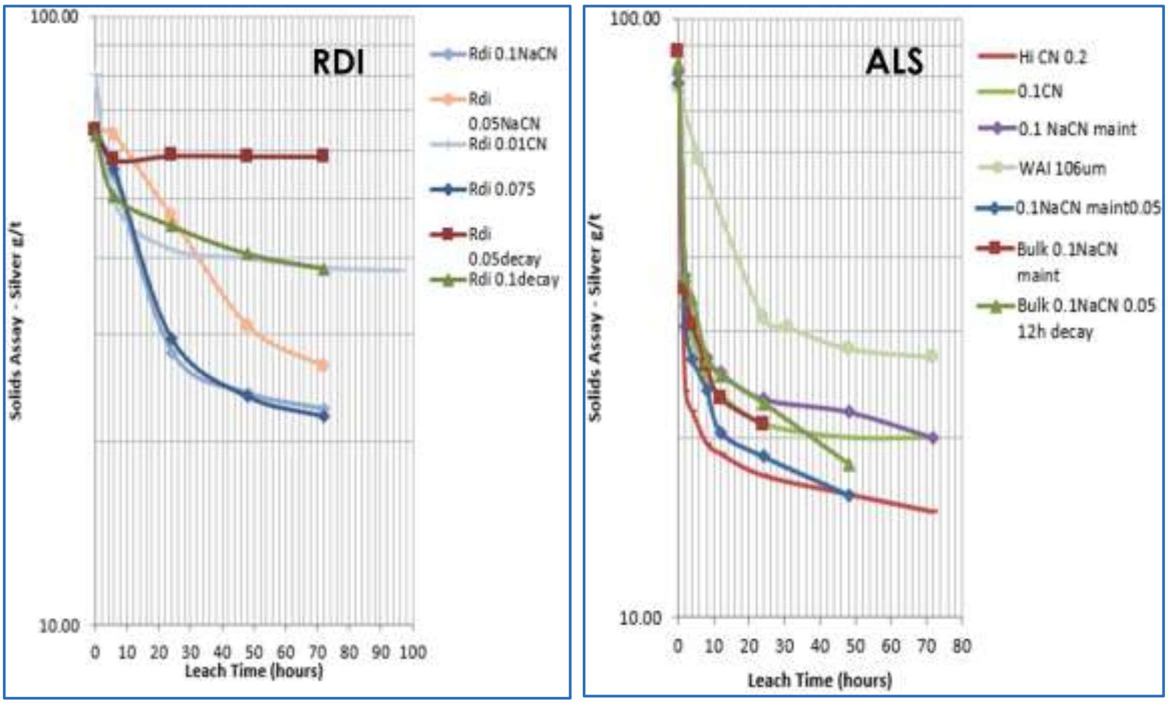


Figure from GRES, 2019.

Figure 13.11 Lead Nitrate Effect on Extraction and Kinetics

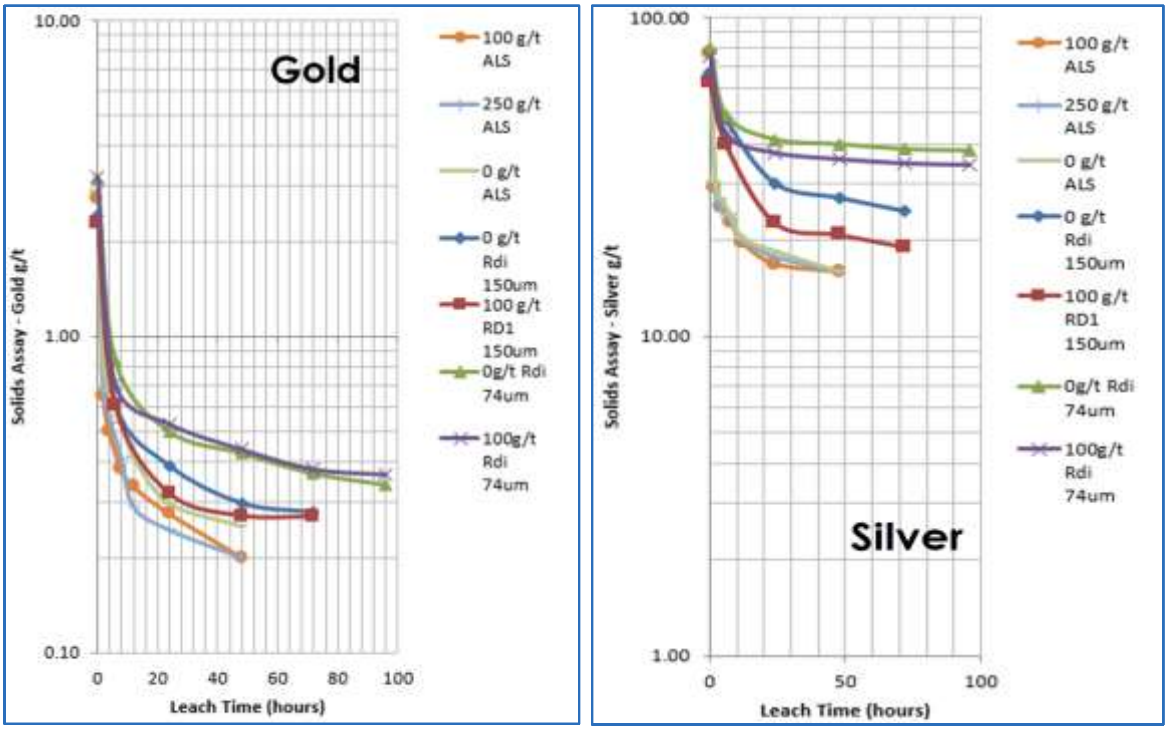


Figure from GRES, 2019.

Lime Consumption

The variability tests completed showed lime consumption depended on the 'natural' pH of the sample. Figure 13.12 shows an increased lime requirement as the natural pH of the samples decreased. The different test programmes show a difference in natural pH, which may reflect the effect of the respective local 'tap' water rather than the sample compositions and could explain some of the variations in reagent consumptions. The lowest natural pH levels were measured for the ALS samples (Australian water) with the highest pH measured for the SGS samples (England). The RDI (USA) and WAI (England) samples had similar average natural pH around 6.

Figure 13.12 Effect of Sample 'Natural' pH on Lime Addition

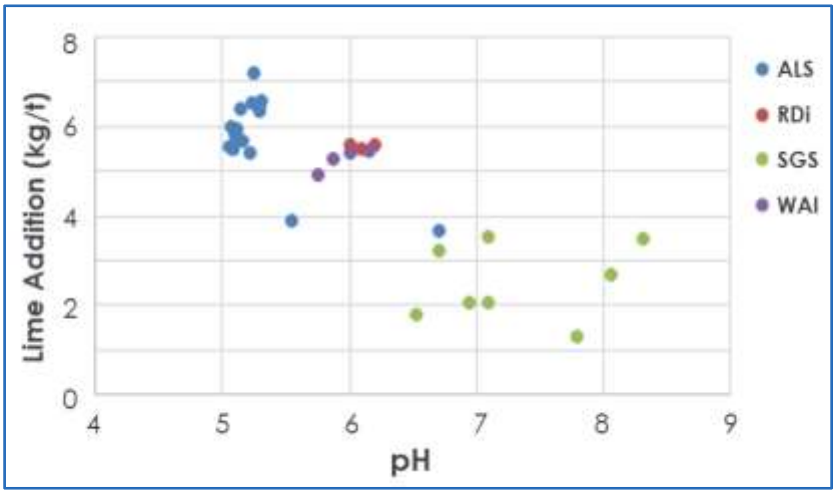


Figure from GRES, 2019.

Oxygen Uptake Tests

Oxygen uptake testing by ALS gave low oxygen consumption for the low-grade and 50:50 gossan / clay-like gossan samples (< 0.006 mg/L/min.). The master composite had an uptake of 0.016 mg/L/min., which equates to an oxygen consumption of 0.12 m³/t ore feed and is typical of oxide ores.

Reagent consumptions applied in process design for the cyanidation process reflect the average from the ALS test programme:

- 2.5 kg/t sodium cyanide
- 5.59 kg/t lime
- 0.1 kg/t lead nitrate
- 0.12 m³/t oxygen (15–20 ppm dissolved oxygen)

13.3.2.6 Elution Tests

No carbon desorption / elution testwork has been completed. The variability test programme includes cold cyanide wash, nitric acid wash, and elution testing.

13.3.2.7 Rheology

The viscosity of leach slurries (solids P_{80} of 125 µm) at different densities between 40% solids and 60% solids was determined from shear stress versus shear strain tests. All conditions returned viscosities less than 100 cPs at a shear rate of 4.2 s^{-1} indicating centrifugal pumps will be suitable. No viscosities were greater than 200 cPs at shear rates of 98 s^{-1} indicating leach tank agitation will be achieved at standard power input however settlement of coarse particles and even distribution of carbon in the CIP tanks may occur at low pulp densities. The ALS results confirm the earlier rheology testing completed by RDI.

13.3.3 Tailings Disposal – Oxide

13.3.3.1 Thickening

The settling characteristics of the leach tailing were investigated by RDI and ALS (Outotec conducted the testing). Results are summarised in Table 13.14.

The settling results combined with the high yield measurements indicate that a small change in underflow density can result in a paste-like slurry. Compression thickener testing by Outotec (ALS, 2018) showed that slurry is paste-like at a density of 72% solids.

Settling rates were relatively low due to the presence of clay minerals and a high fines component. Overflow clarity was good in the dynamic testwork.

Table 13.14 Oxide Tails Thickening Testwork

Description	Units	RDI (2014)		RDI CCD (2014)			ALS (2018)	
Test		Static Cylinder	Static Cylinder	Static Cylinder	Static Cylinder	Static Cylinder	99 mm Dynamic	99 mm Dynamic
Feed Size – P_{80}	µm	150	150	106	106	106	129	129
Feed Size – %Passing 10 µm	%			Stage 1	Stage 2	Stage 3	19.4	19.4
Feed Density	% solids	25	25	22	22	22	15	15
Solids Specific Gravity		2.82	2.8	2.81	2.81	2.81	3.39	3.39
Feed pH		8.8	8.8	10	9.8	9.3	10.5	10.5
Flocculant		HCD47	Coag + HCD47	HCD47	M5250	M5250		
Flocculant Addition	g/t	0	20	20	10	10	15	15
Underflow Density	% solids	50	50	55	55	55	67.6	64.2
Underflow Yield Stress	Pa						142	86
Specific Settling Flux Rate	m^2/tpd	0.496	2.38	0.037	0.036	0.048	0.166	0.028
	t/m^2h	0.08	0.02	1.13	1.16	0.87	0.25	1.5
Underflow Density	% solids	45	45	60	59	57	70.3	72.7
Underflow Yield Stress	Pa						225	320
Specific Settling Flux Rate	m^2/tpd	0.238	0.007	0.044	2.446	2.367	0.083	Comprn
	t/m^2h	0.18	5.95	0.95	0.017	0.018	0.5	
Turbidity / Clarity	NTU/mg/L			22	56	144	< 100	< 100

13.3.3.2 Filtration Tests

Vacuum filtration tests on leach residue completed in the RDI programme gave a minimum residual moisture content of 23.6% indicating high wash ratios would be required to achieve low precious metals losses.

Outotec conducted filtration tests on the ALS samples that included cake washing. Both vacuum and pressure filtration tests were done. Moistures less than 15% were achieved with a cake wash ratio of 0.8 m³ water per tonne solids achieving 90% wash efficiency for pressure filtering and 70% for vacuum filtration (see Table 13.15). The filtrate was observed to be brown with colloidal iron silicates present.

Table 13.15 Summary Filtration Tests

Lab	Method	Feed (%Solids)	Flocc'nt Addition (g/t)	Cake Wash (m³/t)	Wash Efficiency (%)	Cycle Time (min.)	Cake Thickness (mm)	Cake Moisture (%)	Filtration Rate (kg/m²h)
RDI	Vacuum	51	0			2.2	19	26.9	718
		48	20			1.9	17	27.2	841
		47	20			4.1	17	24.2	381
		45	20			2.1	11	23.6	487
ALS (Outotec)	Vacuum	66	0	0.77	70		10	15.2	236
	Pressure	66	0	0.79	90	12.5	37	13.8	165

13.3.3.3 Cyanide Destruction

Sodium metabisulphite added to slurry with copper in solution, aeration, and pH control can be used to convert free and weak acid dissociable (WAD) cyanide to carbon dioxide and inert chemicals. Tests by RDI (2014) reduced the residual cyanide species from 900 ppm to 36 ppm total cyanide, as shown in Table 13.16.

Caro's acid (peroxymonosulphuric acid, H_2SO_5) treatment was tested in the ALS (2018) programme due to its ability to reduce thiocyanate (SCN) levels in solution, which can interfere with flotation (during the period that both oxide and sulphide feeds will be treated). Note that potassium monopersulphate, $KHSO_5$ was used in the testwork as a substitute to Caro's acid due to the unstable nature of Caro's acid (sulphuric acid and hydrogen peroxide are mixed at the point and time of addition) – $KHSO_5$ contains 45% active SO_5. Cyanide speciation analyses were performed by the ChemCentre WA for ALS. The Caro's acid method reduced total cyanide from 200 ppm to 6 ppm.

Table 13.16 Summary of Cyanide Destruction Tests

Description	Method	Reagent Addition Molar Ratio WAD CN	Free CN	WAD CN	Total CN	SCN	OCN	Cu	Zn
			(ppm)						
RDI – 2014									
Before			900	790	910	87		336	
After	SMBS	3.0	26.9	26.8	36	46			
ALS – 2018									
Before				190	200	22	36	24.9	10
After	Caro's	1.75		2.2	3	3		0.02	0.01
After		1.5		3.1				0.04	0.01
After		1.25		7.8				0.10	0.01
After		1.0		8.0	6	< 1		0.16	0.01

Comparison of Heap Leach and Fine Grind Agitation Leach Test Results after Scale-up.

Testwork under agitation leach conditions produced 7% higher gold and 17% higher silver extractions than column leach tests. Table 13.17 summarises the median and average results. Cyanide addition was higher than the consumptions reported for the column tests. The coarse particle size testwork extractions were variable as shown in Figure 13.13 and Figure 13.14. The bottle roll tests completed by RDI, SGS, and WAI gave similar results while the agitated tank leach tests completed by ALS gave higher extractions (and higher cyanide and lime usage).

Column leach test recoveries, particularly in small diameter columns, are typically discounted by 3% to 5% to reflect scale-up to a commercial heap, while no discount is applied to agitation leach tests, therefore a recovery differential in favour of agitation leaching of at least 7% for gold and 16% for silver has been used in trade-off studies.

Table 13.17 Comparison of Agitation and Column Leach Test Results

	Description	Units	Column Leach Tests	Coarse Bottle Roll Tests	Fine BRT and Agitated Tank
Median	Gold Extraction	%	82.2	83.9	89.1
	Silver Extraction	%	48.3	41.3	64.9
	Copper Extraction	%	5.5	6.5	7.1
	NaCN Addition	kg/t	1.1	1.2	1.8
	Lime Addition	kg/t	1.9	2.5	5.4
	Cement Addition	kg/t	7.0	0	0
Average	Gold Extraction	%	82.2	83.2	89.9
	Silver Extraction	%	45.3	44.3	65.9
	Copper Extraction	%	7.7	10.8	11.5

The sensitivity of gold and silver recovery to particle size for both column leach and agitation leaching are shown in Figure 13.13 and Figure 13.14

Figure 13.13 Effect of Particle Size on Gold Extraction

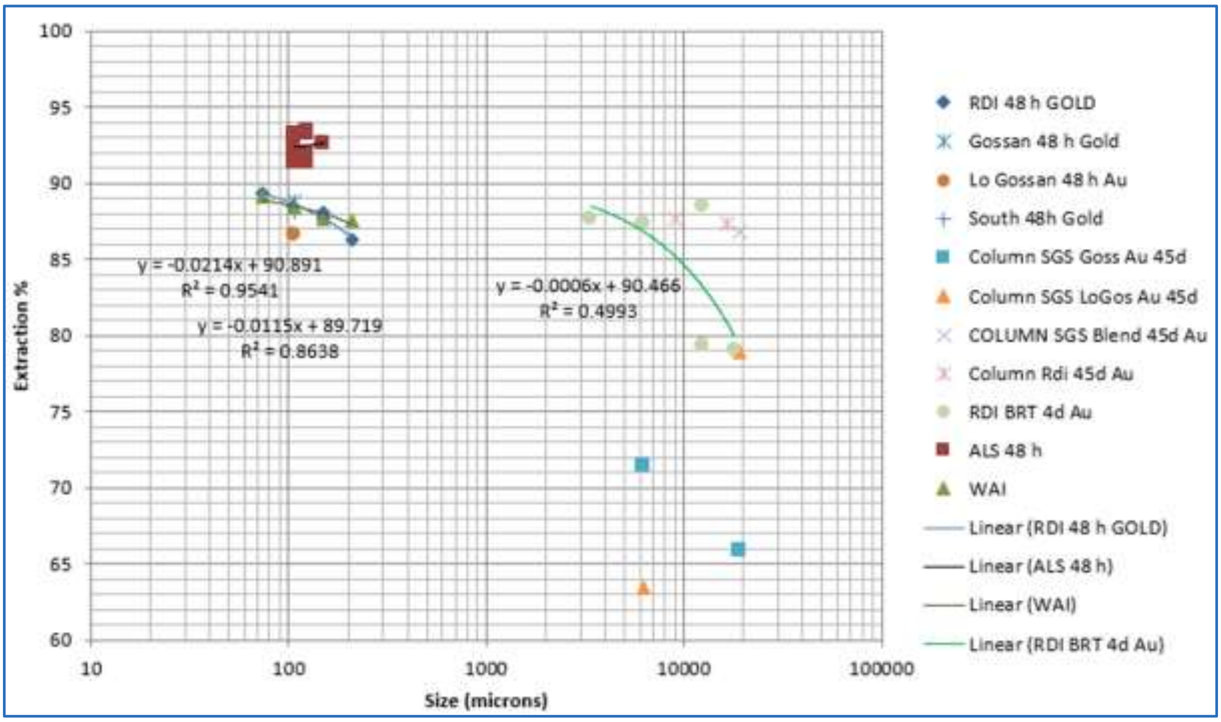


Figure from GRES, 2019.

Figure 13.14 Effect of Particle Size on Silver Extraction

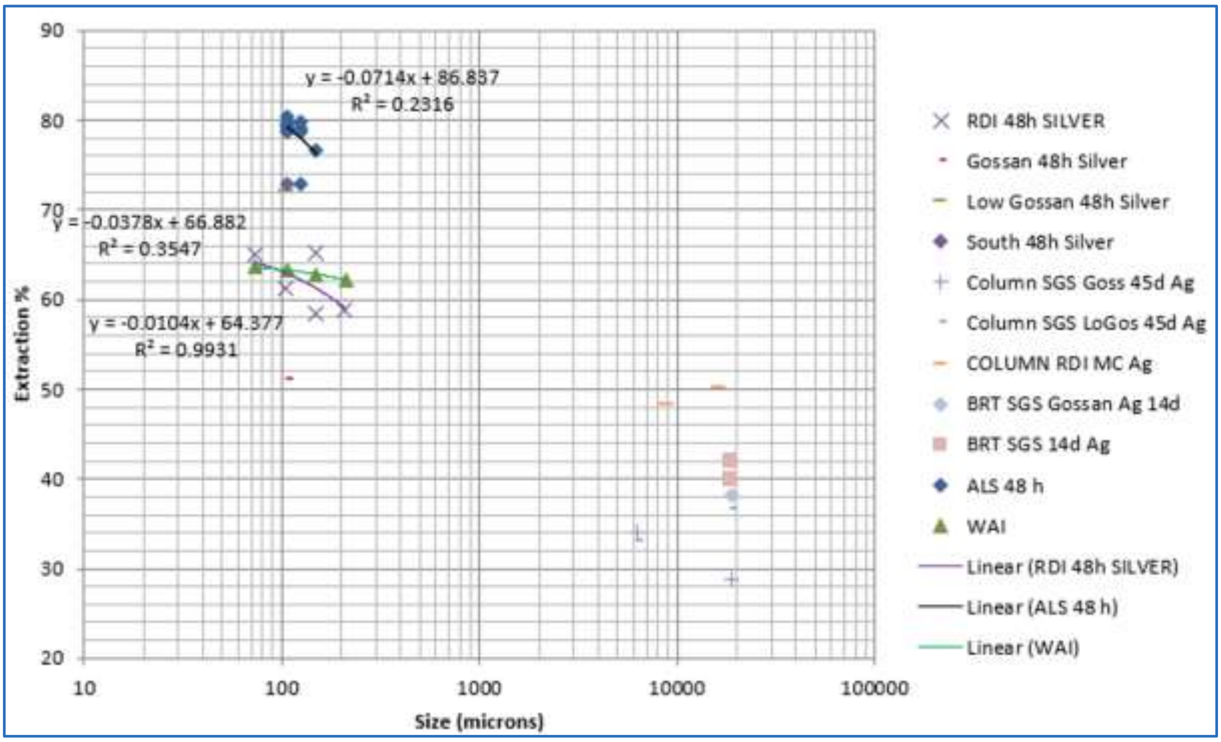


Figure from GRES, 2019.

13.3.4 Flowsheet Selection

Testwork has been completed to support three different flowsheets for processing the oxide resource. PFS16 was based on a heap leach of agglomerated ore with precious metal recovery from solution via Merrill Crowe zinc precipitation, due to the relatively high silver content in the oxide ore.

Subsequent geotechnical surveys found that the soils and sub-surface in the only area available for the heap leach pads and ponds were of poor quality and would require significant excavation and ground preparation costs. Studies indicated that grinding to around 100–125 µm followed by agitation (tank) leaching would provide an increase of 10% recovery for gold and 16% for silver, compared to heap leaching after taking into account typical scale up factors for column leach tests to a commercial scale heap leach.

Engineering studies indicated that a 'hybrid' (Merrill Crowe plus CIP) process, similar to that installed at the Mt Muro plant in Indonesia (now inactive), would result in a much smaller footprint than a heap leach with reduced site preparation cost.

The GRES trade-off study, (GRES, 2017) concluded that the additional revenue from a hybrid tank leach operation would exceed the higher capital differential of $2M and operating cost differential of $5.70/t ore. Metallurgical testwork at ALS therefore focussed on a tank leach flowsheet and engineering.

Subsequent studies and further testwork indicated that a tank leach plant followed by solution recovery by CIP alone, showed advantages over the more complex hybrid flowsheet, and this CIP flowsheet was adopted for PFS19.

The advantages of the selected CIP flowsheet compared to heap leaching are:

- Site layout – unstable soils and high site preparation cost for heap leaching; smaller footprint for agitation leach process.
- Concerns with high slump during heap leaching of high-clay ores and ability to achieve projected recoveries.
- Better control and reaction to peaks of copper ions in solution with agitation leach as final CIP plant will include a cold cyanide wash for copper removal.
- More-effective use of equipment in an integrated oxide / sulphide project. For example, the single mill in the oxide circuit will be suitable for the SAG mill duty for the sulphides treatment, offsetting additional capital for the sulphide plant and total project.

Disadvantages of the selected CIP flowsheet compared to heap leaching include:

- Higher operating costs for the agitation leach (offset by higher recovery and return from revenue).
- Higher initial project capital expenditure.

13.3.5 Recovery Projections for Model

The ALS testwork extractions shown in Table 13.18 (refer to Table 13.11 for detailed data) have been assumed for processing the oxides through the agitation leach–CIP plant. The extraction values used are the median values of the ALS test results. The ALS tests used an agitated tank and maintained oxygen levels during the leach stage while the bottle roll tests did not measure nor include oxygen or air addition. The agitated tank testwork is considered representative of plant operation.

An adsorption and elution recovery of 97.4% for gold and 91% for silver has been applied to the leach extraction values to allow for solution losses in the adsorption and elution (acid wash, cold cyanide wash) stages. These recoveries are more-typical of plant performance than the 99% recovery from solution of gold and silver achieved in the ALS testwork.

Table 13.18 Gold and Silver Recoveries for Financial Model

	Extraction (%)	Adsorption and Elution (%)	Total (%)
Gold	92.6	97.4	90.16
Silver	77.6	91.0	69.65

The extraction and adsorption recovery values will be reviewed following completion of additional variability leaching, adsorption and elution tests. No correlation of extraction relative to head grade was evident in the existing data as shown in Figure 13.15 and

Figure 13.16 – consequently, fixed recovery values have been applied to the oxide resource.

Figure 13.15 Head Grade vs. Extraction – Gold

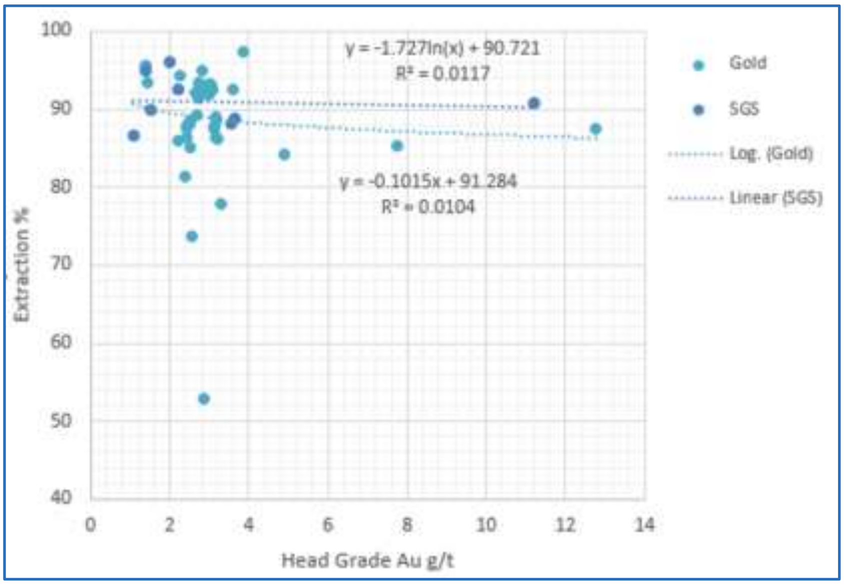


Figure from GRES, 2019.

Figure 13.16 Head Grade vs. Extraction – Silver

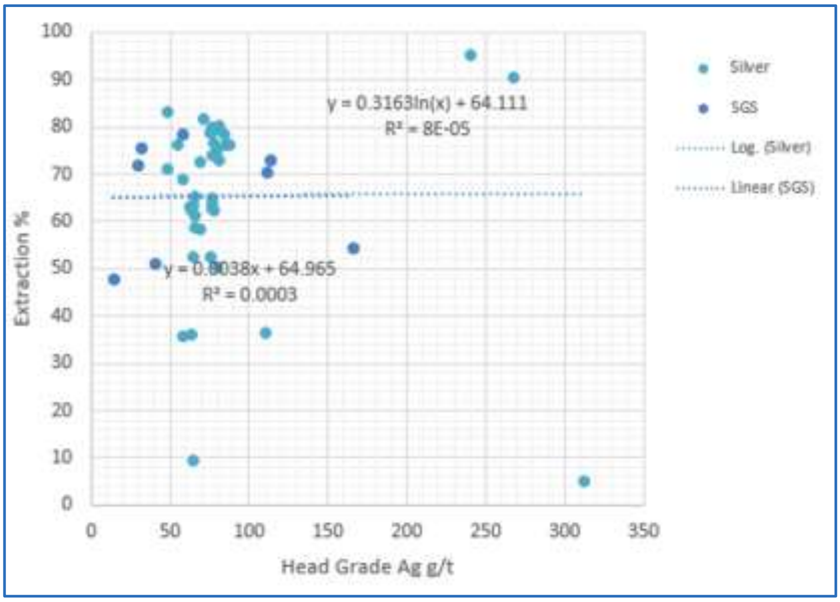


Figure from GRES, 2019.

13.4 Sulphide Metallurgical Testwork

13.4.1 Introduction

The results of the sulphide testwork programmes completed have been detailed in the following reports:

- RDI, Metallurgical Testing of Sulfide Samples from Gediktepe Prospect, Turkey. 2 June 2015.
- SGS, Metallurgical Analysis of the Gediktepe Sulfide Ore Deposit – Interim Report, 10866-577/100866-609. February 2016.
- HMT, Flotation of Gediktepe Cu-Zn Sulfide Ore. January 2015.
- HMT, Optimisation of Gediktepe Cu-Zn Sulfide Flotation Conditions. August 2015.
- HMT, Metallurgical Study on Gediktepe Cu-Zn Sulfide Deposit, DFS Phase – Part 1, Rev 2. 7 March 2018.
- HMT, Metallurgical study on Gediktepe Cu-Zn Sulfide Deposit, DFS -Part 2 final. October 2018.
- WAI, Gediktepe Sulfide Ore Metallurgical Testwork Draft Report ZT64-0609 R001 V0.1. August 2018.
- HMT, Flotation of Massive Pyrite and disseminated Sulfide Ore Sample: Influence of Lead Content – Interim Report. October 2018.
- WAI, Gediktepe Sulfide Ore Metallurgical Testwork Final Report. January 2019.

13.4.2 Samples

Testwork used master composites that reflected the proportion of mineralisation types determined by the resource model at the time of each phase of the project and therefore the distribution of mineralisation types changed as understanding of the deposit developed. The test conditions established for the master composite were then applied to variability samples in each phase of work.

RDI (2014) prepared composites from drill core reject samples that represented the three main sulphide ore types identified at that time – massive pyrite, massive pyrite–magnetite and disseminated sulphide. A master sulphide composite was then prepared from these in the proportions outlined in Table 13.19, and flotation testwork was conducted on the master composite.

A master composite comprising the same blend of the identified ore types as used by RDI was used by HMT to develop a sequential copper and zinc flotation flowsheet, (January 2015).

The subsequent optimisation testwork by HMT (August 2015) and SGS (2015 to 2016) used a different master composite that represented an updated model of the mine and included 1% of enriched material. An analysis of the mine geology and elemental distributions by SGS Canada (report ‘Sample Selection Report for Gediktepe Deposit, Turkey’, undated) identified nine variability samples – disseminated, enriched, massive pyrite, massive pyrite–

magnetite, high-zinc, low-zinc, high-gold, high-lead, and high-gold+silver – for variability testing.

Table 13.19 Summary of Master Composites used in the Testwork

Description	RDI (2014)	HMT (Jan'15)	HMT (Aug'15)	SGS (2016)	HMT (Mar'16)	WAI (Aug'18)	HMT Met Drill (2018)
Ore Type – Master Composite							
MPy	30%	30%	30%	34%	34%	48%	48%
MPyMag	30%	30%	30%	26%	26%	36%	36%
Dissem.	38%	38%	38%	39%	39%	15%	15%
Enrch	2%	2%	2%	1%	1%	1%	1%
Assay							
Cu (%)	0.85		0.82	0.81	0.75	0.74	0.70
Pb (%)	0.18		0.27	0.29	0.23	0.3	0.16
Zn (%)	1.36		1.56	1.76	1.90	2.01	1.64
Fe (%)	28.3		25.5	18.5	28.9	37.8	35.6
S (%)	26.5			30.3		40.1	
As (%)	0.055						0.046
Au (g/t)	0.57			0.62	0.69		
Ag (g/t)	24			28.8	23.8		23

Notes:
1 RDI 2014 and HMT 2015 master composite lithologies were re-defined and updated 28 November 2017. Previously the sample was classified as 33% MPy, 10% MPyMag, 57% Dissem, with 0% Enriched.
2 Minor Split from WAI sample Aug 2018 was also tested at HMT.

Details of the sample drillhole composition are reported in the respective testwork reports.

13.4.3 Comminution Testwork

A summary of the results of comminution parameter testwork is given in Table 13.20.

Due to the 38 µm target grind size selected for flotation, the closing screen size used for determining the Bond ball mill work index was reduced to 75 µm in the 2018 tests. Overall the sulphide material can be described as moderately soft. The hardest component was the material classified as disseminated ore and had a work index of 11.22 kWh/t.

The 80th percentile values were used for power consumption determinations in the plant design.

Table 13.20 Summary of Comminution Test Results

Composite	SG	Ai	CWi (kWh/t)	Rod Mill Work Index @1,180 µm			Ball Mill Work Index					SMC Parameters								
				F_{80}	P_{80}	kWh/t	F_{80}	P_{80}	g/rev	kWh/t	Closing Screen (µm)	A	b	A*b	ta	SCSE (kWh/t)	DWi (kWh/m³)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)
Massive Pyrite							2,293	87	3.66	6.7	106									
							2,233	79	3.3	6.9	106									
							2,332	84	3.9	6.2	106									
							2,296	86	2.97	7.9	106									
							2,202	83	3.42	6.9	106									
	4.35		3.2				2,060	117	4.586	6.3	150	80.4	2.04	164	0.98		2.65	5.8	3.5	1.8
	4.41	0.1852		10,636	795	7.59	2,111.7	60.6	1.77	10.66	75	70.4	1.22	86	0.5	6.86	5.14	9.6	6.8	3.5
Massive Pyrite–Magnetite							2,110	88	3.06	7.9	106									
							2,499	88	3.24	7.4	106									
							2,162	87	3.5	7	106									
							2,250	83	3	7.7	106									
							2,388	83	3.45	6.8	106									
							2,486	84	2.18	9.9	106									
	4.69						2,409	101	4.584	5.66	150	78.6	1.6	126	0.69		3.73	7	4.6	2.4
	4.38	0.2207		10,112	803	7.82	1,625	62	1.75	11.22	75	68.1	1.06	72	0.43	7.4	6.08	11.1	8.1	4.2

Composite	SG	Ai	CWi (kWh/t)	Rod Mill Work Index @1,180 µm			Ball Mill Work Index					SMC Parameters								
				F_{80}	P_{80}	kWh/t	F_{80}	P_{80}	g/rev	kWh/t	Closing Screen (µm)	A	b	A*b	ta	SCSE (kWh/t)	DWi (kWh/m³)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)
Disseminated Sulphide							2,205	79	2.59	8.5	106									
							2,317	86	2.52	9	106									
							2,512	82	2.43	9	106									
							2,403	83	2.08	10.3	106									
							2,382	83	1.96	10.8	106									
							2,326	84	2.1	10.3	106									
							2,391	83	1.9	11.1	106									
							2,333	84	2	10.8	106									
	3.31		18.8				2,061	116	2.629	9.81	150	65	1.59	103	0.81		3.2	8.8	5.6	2.9
	3.46	0.2237		9,884	858	10.28	1,708.5	66	1.58	11.88	75	61.3	1.06	65	0.49	8.6	5.3	12.6	8.9	4.6
Enriched Sulphide							2,141	88	2.4	9.7	106									
							2,234	89	2.64	8.9	106									
							2,035	80	2.94	7.8	106									
	4.36	0.1274		10,221	747	5.41	1,622.7	62.4	2.06	9.88	75	68.8	1.92	132	0.78	5.86	3.3	6.8	4.4	2.3

Composite	SG	Ai	CWi (kWh/t)	Rod Mill Work Index @1,180 µm			Ball Mill Work Index					SMC Parameters								
				F_{80}	P_{80}	kWh/t	F_{80}	P_{80}	g/rev	kWh/t	Closing Screen (µm)	A	b	A*b	ta	SCSE (kWh/t)	DWi (kWh/m³)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)
Overall	4.14	0.189		10,213	801	7.78	2,211	84	2.77	8.7	106	70	1.50	107	0.67	7.2	4.2	8.8	6.0	3.1
80th Percentile	4.40	0.222		10,387	825	8.80	2,389	88	2.04	10.4	106	77	1.09	75	0.80	7.9	5.3	10.8	7.8	4.1
Dissem. Sulphide Average	3.39	0.22		9,884	858	10.28	2,264	85	2.18	10.15	107	63	1.33	84	0.65	8.60	4.3	10.7	7.3	3.8
80th Percentile	3.43	0.22		9,884	858	10.28	2,393	84	1.95	10.86	106	64	1.17	73	0.75	8.60	4.9	11.8	8.2	4.3
Massive Pyrite	4.38	0.19		10,636	795	7.59	2,218	85	3.37	7.37	108	75	1.63	125	0.74	6.86	3.9	7.7	5.2	2.7
80th Percentile	4.40	0.19		10,636	795	7.59	2,295	87	3.04	7.70	106	78	1.38	102	0.88	6.86	4.6	8.8	6.1	3.2
Massive Pyrite–Magnetite	4.54	0.22		10,112	803	7.82	2,241	85	3.10	7.95	108	73	1.33	99	0.56	7.40	4.9	9.1	6.4	3.3
80th Percentile	4.63	0.22		10,112	803	7.82	2,455	88	2.51	9.10	106	77	1.17	83	0.64	7.40	5.6	10.3	7.4	3.8
Enriched Sulphide	4.36	0.13		10,221	747	5.41	2,008	80	2.51	9.07	98	69	1.92	132	0.78	5.86	3.3	6.8	4.4	2.3
80th Percentile	4.36	0.13		10,221	747	5.41	2,178	88	2.26	9.77	106	69	1.92	132	0.78	5.86	3.3	6.8	4.4	2.3
Ratioed Master Composite (39,34,26,1)	4.08	0.21		10,202	821	8.68	2,374	86	2.47	9.32	106	72	1.25	86	0.77	7.67	5.0	10.4	7.3	3.8
Ore Reserve Ratio Jan'18 (19xDissem,76xMpy,5xEnrch)	4.30	0.20		10,273	808	8.08	2,369	87	2.59	8.94	106	75	1.29	90	0.76	7.35	5.0	9.9	6.9	3.6

13.4.4 Flotation Testwork

The objective of the testwork was to develop a flowsheet that could produce separate marketable copper and zinc concentrates, providing the highest net smelter return. Typically, this would be a copper concentrate containing > 20% Cu, < 6% Zn, < 2% Pb, and a zinc concentrate containing > 50% Zn, and < 2% Cu.

The marketing team indicated that it may be possible to sell some of the copper as a 'complex' concentrate with less-restrictive zinc and lead smelter rejection limits. This is discussed further in Section 19.

The main challenge for the Gediktepe sulphide ore is in the copper circuit. A fine primary grind (P_{80} of 38 µm) and a fine regrind of the copper rougher concentrate (P_{80} of 15 µm) is required to achieve acceptable, although still incomplete, liberation of the fine-grained mineral assemblage. Selectivity between copper and zinc minerals is also affected by pre-activation of zinc minerals, due to the presence of secondary copper minerals in situ and/or due to galvanic effects between galena (PbS) and pyrite.

Production of saleable zinc concentrates, grading in excess of 50% Zn at recoveries of around 80%, has been consistently achieved in the testwork.

13.4.4.1 Historical Flotation Testwork – 2014 to 2016

RDI performed both differential (sequential) and bulk flotation tests in 2014 on the sulphide master composite. It was concluded that the differential flowsheet using zinc sulphate, sodium cyanide, and sodium bisulphite ($NaHSO_3$) was not appropriate. The bulk flowsheet used a pH of 12 to depress the pyrite. However, subsequent separation of copper from the zinc was not successful as the copper product assayed 10% Cu and 9% Zn. Depression of the copper by sodium hydrosulphide (NaHS) and flotation of the zinc did produce a 45% zinc grade concentrate at 75% zinc recovery, but neither of these products were considered to be marketable.

HMT, using experience from the Cayeli and other European complex fine-grained Cu–Zn operations, developed a sequential copper and zinc flotation flowsheet with a depressant reagent regime of sodium sulphide, zinc sulphate, and metabisulphite to effect selectivity between the copper minerals and the zinc and iron sulphide minerals, (January 2015) using a composite comprising the ore type blend from the 2014 testwork. Due to the pre-activation of sphalerite ((Zn,Fe)S) in situ, it was found that high additions of depressants were required to affect the separation of the copper sulphides from the sphalerite to produce a copper concentrate assaying less than 6% Zn.

Optimisation (HMT, August 2015) was completed on a different master composite that represented an updated model of the resource and included 1% of enriched material. The nine variability samples nominated from a geostatistical analysis by SGS – disseminated, enriched, massive pyrite, massive pyrite–magnetite, high-zinc, low-zinc, high-gold, high-lead, and high-gold+silver – were tested by both HMT and SGS. Although most tests comprised rougher stages only, eight locked cycle tests (LCTs) were completed by HMT – LCT8 replicated the selected flowsheet and conditions (see Table 13.21 and Table 13.22).

Table 13.21 Summary of LCT Results With and Without Recycle Water

Test	Lab.	Copper Concentrate						Zinc Concentrate		
		Cu Grade (%)	Cu Recov. (%)	Pb Grade (%)	Pb Recov. (%)	Zn Assay (%)	Zn Recov. (%)	Zn Grade (%)	Zn Recov. (%)	Fe Grade (%)
LCT1	SGS	31.0	67.1	1.3	10	2.7	2.7	53.7	83.5	6.2
W-LCT1	SGS	24.5	45.2	0.8	5.1	2.0	1.6	55.4	87.1	8.2
W-LCT2	SGS	27.3	62.9	1.3	9.4	2.5	2.4	54.3	82.3	5.9
LCT8	HMT	25.0	58.3	0.9	6.9	1.5	1.1	54.2	84.9	ND

The HMT optimisation testing provided guidelines and support for the SGS programme. SGS completed a series of tests with different combinations of grinding mill and media to achieve the reducing pulp potentials for selective flotation. The results showed the importance of using some mild steel media; iron tends to consume oxygen in the pulp and in alkaline conditions precipitates iron-hydroxy compounds. A primary grind P_{80} size of 45 µm was used after comparing results with a 30 µm grind test. Following rougher and open circuit cleaner tests, three locked cycle tests were completed by SGS.

The use of recycled water in locked cycle test W-LCT1 resulted in increased loss of copper into the pre-float concentrate, which reports to final tailing, and a higher recovery of non-sulphide gangue into the copper concentrate (lower copper concentrate grade) – mass reporting into the pre-float concentrate doubled, copper recovery to the pre-float concentrate increased from 1% to 20% and zinc loss from 0.5% to 1.5%, (see Table 13.21 and conditions in Table 13.22).

Table 13.22 LCT Test Conditions for Water Recycle Tests SGS, 2015

Test	No. of Cycles	Comments
LCT1	8	Pre-float, four cleaner stages and pyrite stage included; no recycle of decanted stream water or filtrate; build-up of Pb, Ag, and Au in recirculating streams
W-LCT1	8	Pre-float, four cleaner stages; no pyrite stage; water recycle from Cycle 3; Pb and Ag building up.
W-LCT2	8	No pre-float – aeration and CMC depressant used in Cu stage, three cleaner stages; no pyrite stage; SIPX instead of A7279; water recycled; Cu, Pb, and Ag, build-up.
LCT8	6	Pre-float, four cleaner stages and pyrite stage included. Fresh water.

Consequently, the pre-float stage was removed from the third locked cycle test, W-LCT2, which used recycled water and, carboxymethyl cellulose (CMC) was added.to depress the silicate minerals. Loss of copper and zinc was avoided and although the copper performance improved, the concentrate still had a high-silica content (15%) – LCT1 copper concentrate had a non-sulphide gangue content of 8% and W-LCT1 a non-sulphide gangue content of 19%. Other changes that may have impacted the performance in

W-LCT2 were the use of sodium silicate (a dispersant of fine silicate particles) in the regrind stage and changing the zinc collector from A7279 (a blend of isobutyl thionocarbamate and sodium diisobutyl dithiophosphate) to sodium isopropyl xanthate.

Analysis of the tailing water by SGS showed high levels of sulphate (1,130 mg/L), sodium (255 mg/L), chloride (209 mg/L), calcium (117 mg/L), and sulphite (30 mg/L) compared to the tap water. There were increases in the copper, zinc, and iron content, although not considered significant. Potassium and nitrate levels also increased and, as for sulphate, sulphite, and calcium, could be indicators for residual reagent.

The presence of organic reagents in the recycle water was also postulated as a contributor to the poor performance, therefore, in the subsequent test programme, HMT (March 2018) assessed the effect of synthetic, recycled process water and concluded that the addition of activated carbon to the recycle water restored performance to that when using fresh tap water. This was confirmed in the locked cycle tests completed by WAI.

13.4.4.2 Current Flotation Testwork – 2017 to 2018

As done previously, HMT undertook development and confirmatory testwork to support the programme undertaken at WAI (October 2017 to July 2018) where testing used a master composite that reflected a changed ore type blend resulting from the updated mining model. WAI also undertook a pilot plant operation, where the main objective was to generate samples of copper and zinc rougher concentrates for regrind power tests (signature plots), final concentrates for thickening, filtration and concentrate specifications, and tailings for thickening, paste disposal and geotechnical testing. The composite samples tested are outlined in Table 13.23.

As with SGS, tests were required to establish the primary grind and regrind conditions (mill and media types) necessary to generate the pulp conditions favourable for flotation. In conjunction with the HMT work, the optimum primary grind size was reduced to a P_{80} of 38 µm from 45 µm and the regrind sizes were established at a P_{80} of 15 µm for the copper cleaning and 20 µm for zinc cleaning stages.

Due to the complex metallurgy, difficulty of achieving mineral separations and the use of a sequential flowsheet, the use of rougher only plus open circuit cleaning batch test procedures does not reflect the metallurgical potential of the samples – the batch test results report lower recoveries than achieved in the locked cycle tests.

As shown in Table 13.24, the significant increase in recovery of the target metal is accompanied by a lesser increase in recovery of penalty elements.

To avoid the need to conduct complex and expensive locked cycle testing on every test, HMT developed a JKSimFlot model to use the data from open circuit cleaner tests to simulate the performance in a closed circuit plant operation.

Table 13.23 Summary of Master Composites used in the Current Programme (2017–2018)

Ore Type	WAI (2018)	HMT (Jan'18)	WAI (2018)	WAI (2018)	WAI (2018)	HMT (Mar'16)
	Master Composite			Metallurgical Drill Master Composite	N-Master Composite Blend 8	Master Composite
MPy	48%	48%	48%	50%	50%	34
MPyMag	36%	36%	36%	30%	30%	26
Dissem.	15%	15%	15%	20%	20%	39
Enrch	1%	1%	1%	0%	0%	1
Assay						
Cu (%)	0.74	0.807	0.701	0.65	0.72	0.75
Pb (%)	0.30	0.31	0.163	0.18	0.38	0.23
Zn (%)	2.01	1.75	1.64	1.8	1.83	1.90
Fe (%)	37.8	34.2	35.6	36.75	33.34	28.9
S (%)	40.1	44.3		43.3	33.88	
As (%)	0.053			0.047	0.066	
Au (g/t)	0.64					0.69
Ag (g/t)	21.9	25.5	23			23.8

Table 13.24 Comparison of Locked Cycle and Batch Test Results in Pre-Float and Copper Circuit

Source	Sample	Stream	Mass (%)	Grade (%)						Distribution (%)					
				Cu	Pb	Zn	Fe	S	As	Cu	Pb	Zn	Fe	S	As
FCT-37	Blend 7	Talc Conc.	1.7	0.73	0.47	2.04	15.5	18.6	0.04	1.7	2.5	1.7	0.8	0.8	1.7
WAI-LCT2			1.8	0.61	0.39	1.56	12.4	14.3	0.04	2.7	4.8	3.0	1.2	1.3	6.3
FCT-40			1.6	0.79	0.50	1.97	15.9	17.8		1.6	2.4	1.5	0.7	0.7	
WAI-LCT3			1.7	0.84	0.44	1.94	17.6	18.8	0.04	3.6	4.6	3.6	1.6	1.7	2.0
FCT3-B8b			1.2	0.93	0.49	1.78	15.3	17.3	0.04	1.5	1.5	1.1	0.6	0.6	0.7
WAI-LCT4	Blend 8		1.3	0.90	0.49	1.75	14.1	15.6	0.06	3.0	3.6	2.7	1.1	1.2	2.3
FCT-37	Blend 7	Cu Conc.	0.7	31.4	3.45	1.56	26.1	33.8	0.02	28.9	7.4	0.5	0.5	0.6	0.3
WAI-LCT2			1.1	28.0	2.17	6.67	22.6	33.1	0.87	44.5	8.5	4.0	0.7	0.9	12.7
FCT-40			0.3	35.0	0.87	0.73	29.9	36.1		12.1	0.7	0.1	0.2	0.2	
WAI-LCT3	Blend 7 (PP)		0.8	34.4	1.59	1.63	29.3	34.3	0.01	37.3	3.8	0.8	0.6	0.7	0.2
FCT3-B8b	Blend 8		0.6	33.4	0.52	1.48	29.8	35.6	0.02	29.2	0.9	0.5	0.6	0.6	0.2
WAI-LCT4			1.3	29.3	6.77	3.98	25.1	33.0	0.61	49.3	24.6	3.0	1.0	1.2	11.7

13.4.4.3 Selection of Primary Grind Size

WAI performed a series of tests on the master composite in 2018 evaluating the effect of primary grind on rougher flotation. The results in Table 13.25 show that Cu–Zn–Fe selectivity improved with finer grinding down to P_{80} of 25 µm, but due to the high power requirement needed to achieve this fineness of grind in a commercial plant it was decided to select a P_{80} of 38 µm for the standard grind in future tests and address the liberation requirements in the regrind circuits. It should also be noted that the zinc recovery to copper concentrate was relatively high in these tests indicating possible surface oxidation and pre-activation of the zinc minerals.

Table 13.25 Effect of Primary Grind on Rougher Flotation – Master Composite

Test	Grind P_{80}	Mass Pull (%)	Grade %			Distribution %		
			Cu	Zn	Fe	Cu	Zn	Fe
FT2	45	16.6	3.60	5.15	40.6	74.6	42.7	16.8
FT3	38	17.6	3.24	3.71	42.2	74.4	33.4	17.7
FT4	25	6.5	9.85	5.42	30.7	76.0	18.5	4.8

13.4.4.4 Pre-Float Circuit

In the flotation optimisation tests (HMT, 2015), upgrading of the copper concentrate was affected by the presence of talc and other silicate minerals. Addition of a modified guar (8860-GL) and a polymeric depressant (7261-A) did not improve the separation and a pre-float stage using frother (MIBC) to remove the naturally floating silicates was employed.

SGS (2016) investigated the impact of the pre-float stage by comparing W-LCT1 (pre-float) and W-LCT2 (no pre-float) however, due to the use of untreated recycle water, 22% of the copper reported to the pre-float concentrate in W-LCT1. The copper concentrate results of the three SGS LCTs are shown in Table 13.26. Although no effective conclusions can be made, SGS recommended trialling additional CMC.

Table 13.26 Comparison SGS (2015) Pre-Float Tests

Description	LCT1	W-LCT1	W-LCT2
Circuit	Pre-Float	Pre-Float	No Pre-Float
	Tap Water	Water Recycle	Water Recycle
Copper Circuit Collectors	NaAero + A8761	NaAero + A8761	NaAero + A8761
Talc Depressant			CMC (Depramin 347)
Zinc Circuit Collector	A7279	A7279	SIPX
Copper Concentrate			
Cu Grade (%)	30.9	24.4	27.3
Chalcopyrite (%)	66.2	54.8	59.2
Covellite (%)	12.3	10.0	10.8
Pyrite (%)	9.2	12.1	11.2
Galena (%)	0.9	0.9	0.6
Sphalerite (%)	2.6	2.9	3.5
Non-sulphide Gangue (%)	7.6	19.4	14.7
Copper Recovery			
Pre-Float Concentrate	2.3	22	0
Copper Concentrate	75	47	55
Total	**77.3**	**69**	**55**

HMT (March 2018) completed open cleaner flotation tests using four types of CMC available in Turkey:

- Rheolon 30N – 98% active content, 400 cps viscosity at 4%
- FiltraPAC LV 6014 – technical grade, high-DS, low-viscosity, 150 cps at 2%
- Blend 21
- QS2

The results showed that the use of these CMCs did not selectively depress non-sulphide gangue and therefore did not improve the grade of the copper concentrate. There was an increase in copper recovery, which was related to the increase in mass pull.

In tests on massive sulphide-only samples, with over 80% pyrite and low amounts of hydrophobic gangue, pre-aeration was used instead of the pre-flotation stage.

A pre-float of five minutes collectorless flotation using only frother was sufficient to remove less than 2% of the mass and loss of less than 4% each of the copper and zinc in the pre-float concentrate and enable generation of copper concentrate grades above 28% Cu in three stages of cleaning, (note: SGS used four stages of cleaning).

HMT tests (2015) showed that addition of sodium silicate in the primary and regrind stages depressed silicates.

13.4.4.5 Depressant Selection

The earlier testwork indicated that sphalerite particles are pre-activated and could not be depressed at high pH, which was the main reason for abandoning the more conventional bulk flotation route in favour of the sequential flotation developed by HMT. Depressants sodium sulphide (Na_2S), zinc sulphate ($ZnSO_4$) and metabisulphite (MBS), have been applied in grinding and copper flotation to achieve selective Cu/Zn flotation at natural pH. In zinc flotation, standard conditions of high-pH to depress pyrite, activation using copper sulphate ($CuSO_4$) and a xanthate type collector have been used.

HMT (March 2018) tested lower Na_2S dosage, use of sodium hydrosulphide (NaHS) and the mode of depressant addition to assess the impact on performance and opportunities to lower the reagent consumption (cost). A mixture of dextrin and sodium monophosphate ($NaHPO_4$) was also tested as an alternative depressant scheme. One flotation test at 40°C pulp temperature was conducted to investigate effects of pulp temperature on flotation performance.

The findings from these tests were:

- The highest grade per unit recovery was obtained with the standard depressant scheme.
- Use of NaHS in place of Na_2S did not improve the performance (confirmed in tests reported by HMT in 2015).
- Lower Na_2S dosage (250 g/t) increased copper recovery but negatively affected Cu/Pb selectivity.
- Lower MBS (1 kg/t) and $ZnSO_4$ (0.5 kg/t) did not improve copper recovery and negatively affected Cu/Pb and Cu/pyrite selectivity.
- Selective copper flotation could not be achieved with the dextrin and $NaHPO_4$ mixture.
- Post-grinding depressant addition increased copper recovery but with lower selectivity.
- Staged addition of depressants (grinding, copper Rougher 2 and copper Rougher 3 stages) achieved higher copper recovery with better Cu/Pb selectivity.
- Flotation selectivity was reduced at high pulp temperature.

It was concluded that the standard flotation conditions developed in the 2015 test programme were the optimum conditions and that dosages of depressants and collectors be adjusted according to the grade of the flotation feed.

13.4.4.6 Collector Selection

Aerophine 3418A (3418A) was tested in place of sodium aerofloat (NaAF), and also Aero 8761 and Aero 404 in mixture with NaAF.

Performance of the collectors was evaluated based on copper grade-recovery curves, selectivity curves, and also collector dosage vs. copper recovery. The mixture of NaAF+Aero 8761 gave the highest grade for a unit copper recovery, but the copper recovery remained at approximately 65% due to low mass pull.

3418A is a dithiophsphinate type collector and considered selective against pyrite and sphalerite in flotation of copper and lead minerals. The results showed that 3418A was stronger than the other types of collectors, and that it gave the highest copper recovery but at lower grade. The high copper recovery was due to high mass pull (13.01%). Selectivity against zinc and lead was consequently poor.

Aero 404 is a mercaptan type collector and considered beneficial for flotation of sulphide minerals having slightly oxidised and tarnished surfaces. Aero 404 was tested in combination with NaAF, to increase Cu recovery with acceptable selectivity over sphalerite and galena. The results showed that Aero404 did not significantly affect the performance.

Stage addition of collector in the rougher flotation stage is very common in many sulphide flotation plants. This type of operation could improve the selectivity with similar recoveries and more-controlled mass pull. Hence, stage addition of NaAF was tested. Higher copper recovery per unit collector addition was obtained. The final copper recovery was slightly higher but with longer flotation time (10 min.) compared to the test with NaAF+Aero 8761 (6 min.). The selectivity between Cu/Zn and Cu/Pb was similar to that obtained in the other collector tests. It was concluded that three stages was considered suitable for upgrading the copper rougher concentrate. Stage addition of collector improved both recovery and selectivity.

The mixture of NaAF+Aero 8761 was the most suitable collector mixture for copper flotation section.

13.4.4.7 Optimisation of Regrind Size

Liberation analysis of copper cleaner scavenger tail containing 5.6% chalcopyrite (WAI, 2018) indicated 45% of the copper was (> 80%) liberated while 19% was locked. The cleaner scavenger tail contained 22.5% sphalerite, 55% of which was liberated.

WAI completed a series of tests evaluating the copper cleaner metallurgy as a function of regrind size using the master composite blend sample. Based on the results shown in Table 13.27, 15 µm was selected as the standard condition for the copper circuit regrind.

Table 13.27 Copper Cleaner Flotation as a Function of Regrind Size

Test	R/G P_{80} (μm)	Mass Pull (%)	Grade (%)			Distribution (%)		
			Cu	Zn	Fe	Cu	Zn	Fe
FCT 8	10	1.0	25.0	6.4	17.6	31	3.3	0.5
FCT 9	15	1.3	25.0	4.97	17.6	39.1	3.0	0.6
FCT 10	20	0.9	23.8	6.76	25.7	28.1	3.2	0.7
FCT 11	25	1.4	20.75	11.0	24.7	35.2	7.5	0.9
FCT 18	10	1.4	26.9	6.4	22.1	47.1	4.4	0.8
FCT 19	15	1.2	27.2	7.0	23.0	39.4	4.0	0.7
FCT 30	15	1.1	29.7	4.71	22.5	39.6	2.8	0.9
FCT 32	15	0.9	28.9	5.21	26.9	37.7	4.0	0.5
FCT 33	10	0.8	29.8	6.31	23.1	28.9	2.8	0.5

A similar series of tests were completed for the zinc cleaner metallurgy Table 13.28 shows a significant upgrade in zinc grade in the zinc cleaner concentrate between the 30 μm and 20 μm zinc regrind size, but a smaller increase when reducing from 20 μm to 15 μm.
A 20 μm P_{80} was selected for the standard zinc regrind size.

Table 13.28 Zinc Cleaner Flotation as a Function of Regrind Size

Test	R/G P_{80} (μm)	Mass Pull (%)	Grade (%)			Distribution (%)		
			Cu	Zn	Fe	Cu	Zn	Fe
FCT 26	30	1.7	0.54	48.3	10.0	1.1	43.7	0.5
FCT 27	20	1.5	1.91	55.7	4.94	3.6	46.0	1.3
FCT 28	10	1.2	1.93	57.3	3.54	3.6	47.3	1.3

13.4.4.8 Aging Effects

Surface oxidation of sulphide minerals causes lower selectivity and recoveries in flotation. Each ore type has a different tolerance to surface oxidation depending on mineralogy and ore genesis. The flotation tests showed that the sulphide ore was sensitive to surface oxidation and testwork was done by aging samples for up to four weeks to assess any impact of mine stockpile time on flotation performance. Two samples were tested: a blended master composite:enriched (90:10) sample, and an enriched-only sample from a metallurgical drilling campaign to provide 'fresh' material. Due to the presence of secondary minerals in the enriched material, these samples were considered the most likely to suffer any detrimental aging effects.

The tests were performed using the standard flotation conditions after exposure to air and being sprinkled with water daily to simulate the effects of light rain. Weekly aged samples were tested to provide a time line.

For the 90:10 composite, copper recovery was not negatively affected in the rougher stage, however, the final copper grade decreased after two weeks of aging, but did not fall below 22% Cu. Nevertheless, the grade and recovery of zinc into the copper rougher concentrate increased with increasing aging time. However, the majority of the zinc was rejected in the cleaner flotation stage, irrespective of aging time.

Unlike the 90:10 composite ore sample, the copper recoveries for the enriched-only material decreased with increasing aging time. Similar behaviour occurred with the zinc and lead recoveries, which increased after one week of aging. Copper grade in the rougher concentrate decreased and zinc assay increased. Lead variation was low due to the low lead feed content of the enriched-only sample.

Zinc flotation was affected by the performance of the copper flotation stage. Therefore, stage recovery (i.e. recovery based on zinc rougher feed) was used to evaluate the effects of aging on zinc performance and is shown in Figure 13.17, with both recovery and grade decreasing as aging time lengthened.

Figure 13.17 Effects of Aging on Zinc Flotation Performance (90:10 Master Composite: Enriched Blend)

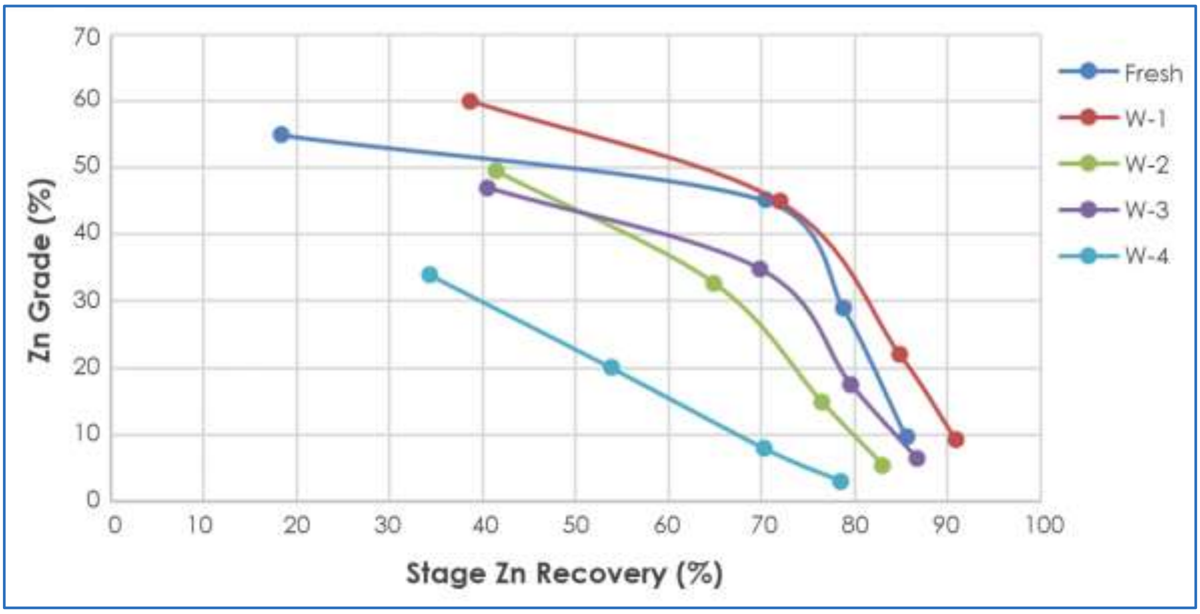


Figure from GRES, 2019.

Treatment of mine production within one to two weeks is therefore recommended.

EDTA extraction tests were used as a standard characterisation method to assess the state of surface oxidation of samples. EDTA extractable copper also provides an indication of the presence of secondary copper minerals – these tests showed that enriched samples contain higher amounts of secondary copper minerals. The relative amount of EDTA extracted metal showed that the enriched sample used in GD Mix tests (referenced as Part 17 from June 2017) was more oxidised than the enriched sample from metallurgical drill sample (referenced as Part 21d, September 2017). The sulphide minerals in master composite samples (GD3 master composite; referenced as Part 18, July 2017) also had higher surface oxidation than the samples from the metallurgical drilling programme (Part 21 October 2017).

13.4.4.9 Effect of Water Quality

The sulphide ore contains naturally floatable silicates (NSG), mainly talc, which could reduce the copper grade of the copper concentrate. In the standard flotation flowsheet, NSG is removed in a pre-flotation stage prior to copper rougher flotation. However, circulated water (zinc rougher flotation tail) contains residual organics, mainly xanthate ions, which cause flotation of sulphide minerals in the pre-flotation stage and loss of copper metal with pre-flotation concentrate.

Loss of copper and zinc into the pre-float concentrate was directly related to the increase in xanthate concentration in the process water. Tests were done to determine the effect of removal of the residual organic ions from the process water by activated carbon; results were similar to tests using fresh tap water.

Water treatment with activated carbon did not affect inorganic ions; only residual organic ions were removed.

The recycle water in LCTs was therefore treated using 8 g/L of activated carbon and agitated for 15 minutes to remove the organics. LCT1 on the master composite sample completed by WAI (2018) however reported a loss of 20% of the copper and 6% of the zinc in the pre-float concentrate (copper concentrate graded 25.5% copper at a recovery of 42.7%). Investigation revealed that the coarse 5 mm activated carbon used for treating the recycle water did not remove the residual frother and collector. Finer carbon and powder-activated carbon were then used to successfully treat the process water.

Locked Cycle Flotation Tests

Results for batch testing of pre-float and copper roughers are useful for establishing optimum conditions for primary grind and reagent additions, as no recycle is involved. Typically, after primary grinding to 38 µm (P_{80}) the rougher conditions for the massive pyrite and master composites were adjusted to maintain a mass pull of around 6%, resulting in a recovery of about 70%–75% Cu, < 20% Zn to the copper rougher concentrate. Increasing mass pulls above this level resulted in over loading of the copper cleaner circuit and poor cleaner performance.

Batch cleaning tests provided a reasonable indication of final copper concentrate grades, but at significantly lower recoveries than were achieved in the locked cycle testing. Simulations using JKSimflot software were used to predict the outcome of recycling cleaner tails from the batch cleaner test to reflect the conditions in a locked cycle test (and in a commercial plant).

The sequential flowsheet used in the standard locked cycle test is shown in Figure 13.18. Six to eight cycles were required to reach equilibrium.

Figure 13.18 Locked Cycle Test Flowsheet

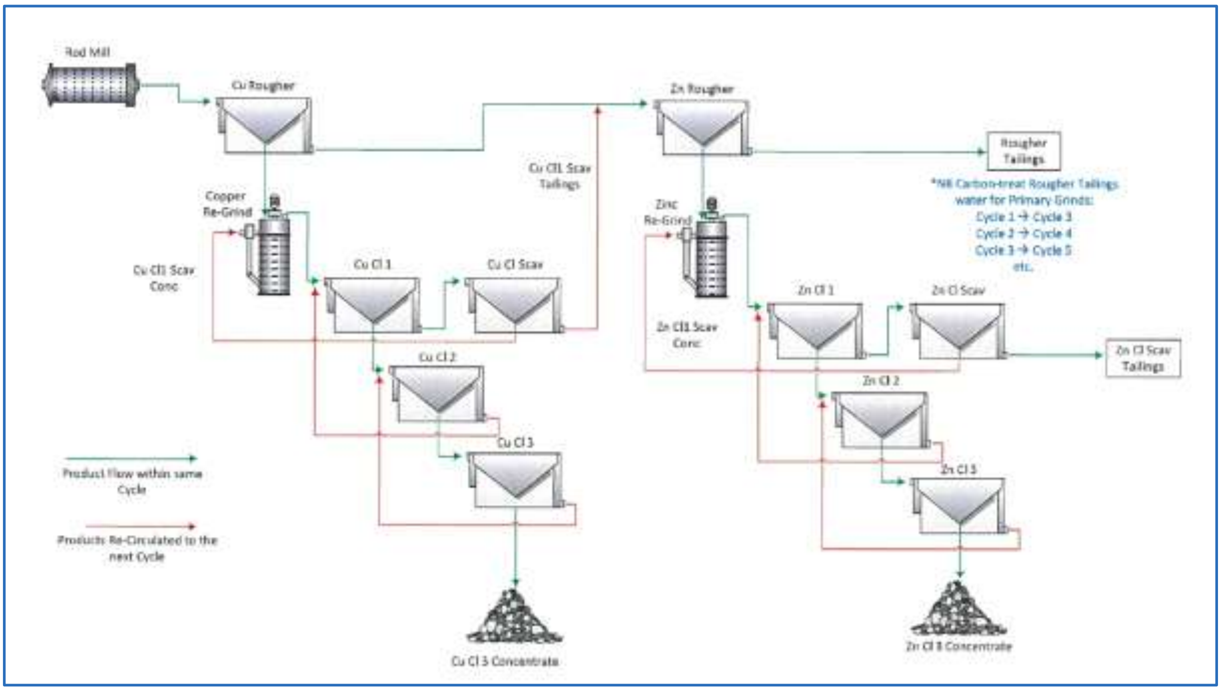


Figure from GRES, 2019.

A summary of locked cycle test results is shown in Table 13.29 and Table 13.30. The tables exclude tests not completed using non-standard conditions, e.g. when untreated recycle water was used. The WAI test showed high levels of lead reporting to the copper concentrate and generally produced lower copper recoveries than in the HMT tests. Mineralogical analysis of the WAI sample (Petrolab Report AM2842c) indicated that all the galena was present locked in other minerals or composited with other mineral particles – no liberated galena was observed. For master composite Blend 3, 20% of the galena was present as composites with chalcopyrite, while for master composite Blend 8, over 30% of the galena was composited with chalcopyrite. The majority of the galena was associated with pyrite.

Due to the complex galena mineralogy, WAI reduced collector additions attempting to minimise the lead recovery however copper recovery was reduced. A series of locked cycle tests was completed by HMT (results are summarised in Table 13.30) on disseminated and massive pyrite composites to investigate the possible source of the problematic ore. The results indicate that further variability testing within ore types is required to determine operating strategies to deal with the mineralogical fluctuations experienced in the different testwork samples.

Table 13.29 Summary of Locked Cycle Tests on Master Composites

Date	Lab.	Composite	Test No.	Feed Assay (%)			Copper Concentrate						Zinc Concentrate					
							Grade (%)			Distribution (%)			Grade (%)			Distribution (%)		
				Cu	Pb	Zn	Cu	Pb	Zn	Cu	Pb	Zn	Cu	Pb	Zn	Cu	Pb	Zn
Aug'15	HMT	MCS	LCT8	0.63	0.24	1.38	23.80	1.16	1.87	66.06		1.96			53.38			84.75
Aug'15	SGS	MCS	LCT1	0.80	0.28	1.75	30.95	1.30	2.49	75.51	8.96	2.77	0.80	0.28	1.75	8.79	24.62	80.14
Dec'17	HMT	MCS Met Drill	100%	0.67	0.15	1.51	28.74	2.21	4.57	68.30	23.20	4.80	1.98	0.35	52.87	6.87	5.34	81.53
Dec'17	HMT	MCS Met Drill	90:10	0.87	0.18	1.87	29.88	2.63	11.49	68.14	29.57	12.20	2.64	0.48	50.18	8.47	7.59	75.01
May'18	WAI	MCS Blend 7	LCT3	0.71	0.29	1.89	28.00	2.17	6.67	44.50	8.46	3.99	4.64	2.65	46.07	22.18	31.04	82.83
Jun'18	WAI	MCS Blend 8	LCT4	0.69	0.39	1.70	29.72	6.51	3.68	47.49	18.72	2.41	3.69	3.56	45.26	16.10	27.92	80.68

MCS = master composite. 90:10 = 90% master composite : 10% enriched ore composite.

Table 13.30 Summary of Locked Cycle Tests on Massive Pyrite and Disseminated Composites

Date	Lab.	Composite	Test No.	Feed Assay (%)			Copper Concentrate						Zinc Concentrate					
							Grade (%)			Distribution (%)			Grade (%)			Distribution (%)		
				Cu	Pb	Zn	Cu	Pb	Zn	Cu	Pb	Zn	Cu	Pb	Zn	Cu	Pb	Zn
Aug'15	HMT	Disseminated		0.67	0.15	1.50	31.10	6.90	5.10	73.80	48.50	6.60	3.70	1.90	50.70	9.90	14.80	74.20
Aug'18	HMT	High lead Diss.		0.25	0.40	0.96	24.02	14.24	6.09	67.70	24.90	4.50	2.29	5.20	50.23	12.60	17.70	71.80
Aug'18	HMT	Med. lead Diss.		0.42	0.22	0.91	21.70	6.31	9.08	72.20	40.20	14.00	2.96	2.26	48.51	9.20	13.50	70.00
Aug'18	HMT	Low lead Diss.		0.48	0.10	0.81	28.09	1.61	9.36	58.00	16.80	11.50	6.48	1.53	36.89	21.30	25.40	72.10
Aug'18	HMT	High lead MPY		0.98	0.69	4.67	31.27	1.28	5.16	71.20	4.12	2.46	1.75	2.72	54.28	12.20	26.80	79.10
Aug'18	HMT	Med. lead MPY		0.67	0.25	2.58	30.87	1.95	2.68	65.55	11.11	1.48	1.60	1.08	60.00	0.46	14.40	77.60
Aug'18	HMT	Low lead MPY		0.87	0.18	1.12	30.58	1.76	6.40	72.10	20.30	11.70	3.96	0.59	44.51	7.80	5.70	68.00

13.4.4.10 Pilot Plant Results

WAI conducted a 40 kg/h pilot plant operation treating a total of 1.8 tonnes of master composite Blend 8 material to generate rougher concentrates for regrind signature plot tests, final concentrates for thickening, filtration and transport tests, and final tailing (zinc rougher tail and zinc cleaner scavenger tail) for thickening tests. The products generated for these tests are summarised in Table 13.31.

Table 13.31 Summary of Products Generated in WAI Pilot Plant

Run	Stream	Mass (%)	Grade %			Distribution %		
			Cu	Pb	Zn	Cu	Pb	Zn
400 kg Run	Pre-Float Conc.	6.7	0.67	0.41	1.81			
	Cu Rougher Conc.	28.9	3.26	0.95	5.06			
	Zn Rougher Conc.	24.6	1.01	0.64	5.35			45
1.453 t Run	Pre-Float Conc.	3.9	0.95	0.45	1.91			
	Cu Conc.	0.74	26.0	3.6	5.4			
	Zn Rougher Conc.	15	1.3	0.76	8.7			85
	Zn Conc.				40			23

The zinc recoveries reported in Table 13.31 are indicative only as it was not possible to calculate a mass balance due to the configuration of the plant, programme of sampling, and optimisation required during the short operation. What the indicated zinc recoveries show is that the introduction of the copper cleaner stage rejects much of the zinc that mis-reports into the copper rougher concentrate, thereby making it available for recovery in the zinc circuit confirming this observation during the bench scale batch flotation tests.

Figure 13.19 shows photographs of the pilot plant equipment.

Figure 13.19 Pilot Plant Equipment

Figure from GRES, 2019.

13.4.5 Variability Tests

Introduction

Variability samples tested by SGS (2016) were selected by SGS Geostat. The results showed that flotation behaviour of disseminated sulphide and enriched samples was different from massive pyrite and massive pyrite–magnetite samples.

Metallurgical Domain – Massive Pyrite

The massive pyrite and massive pyrite–magnetite rock types have generally similar metallurgical characteristics and have been classified as a single metallurgical domain (MPy).

The main sulphide mineral is pyrite. Copper is mainly present as chalcopyrite with lesser amounts of secondary copper minerals and some enargite (copper arsenic sulphide). Zinc is present as sphalerite.

MPy is the predominant mineralisation type, making up around 60% of the sulphide resource. This has been reflected in the composition of master composites prepared for metallurgical testing.

The metallurgy of the MPy material is more consistent and therefore more predictable than the other mineralisation types:

- Lead levels in copper concentrates are generally within smelter rejection limits. Recovery of lead to copper concentrate averages around 10%.
- Zinc levels in copper concentrates are generally within smelter limits although may increase due to weathering effects.
- Arsenic recovery to copper concentrate is generally low, despite the presence of enargite in some samples.

A series of locked cycle tests was conducted by HMT in 2018 to evaluate the metallurgy of MPy samples containing varying amounts of penalty elements in the feed. Results are shown in Table 13.32 Analyses of the feed can be found in Table 13.30.

Table 13.32 Distribution of Penalty Elements in Concentrates from Massive Pyrite Samples Locked Cycle Tests

Test	Mass Pull (%)	Grade %				Distribution %			
		Cu	Pb	Zn	As	Cu	Pb	Zn	As
Copper Concentrate									
High Pb LCT1	2.20	31.23	1.32	5.20	0.11	68.69	4.19	2.45	3.16
High Pb LC2 T2	1.84	30.20	1.03	4.61	0.11	64.74	2.34	1.79	1.88
Med. Pb	1.48	30.49	1.98	2.78	0.07	66.22	11.20	1.69	2.27
Low Pb	1.47	27.79	1.24	1.93	0.05	63.23	12.97	1.64	1.53
Zinc Concentrate									
High Pb LCT1	7.32	1.69	2.66	50.63	0.15	12.35	28.05	79.19	13.92
High Pb LC2 T2	6.42	0.79	1.37	59.61	0.05	5.89	10.84	80.77	2.99
Med. Pb	3.27	1.66	1.15	57.85	0.05	7.95	14.35	77.35	3.56
Low Pb	2.36	2.16	0.56	58.06	0.04	7.90	9.34	79.27	1.98

13.4.5.1 Metallurgical Domain – Disseminated

Disseminated ores make up about 30% of the orebody and contain lower amounts of pyrite and more non-sulphide gangue than the MPy ores.

Metallurgy is more variable than for the massive pyrite, and penalty elements in copper and zinc concentrates can exceed smelter rejection levels in some samples.

Lead levels in copper concentrates can be above smelter rejection levels. Recovery of lead to copper concentrate averages around 25%. Tests on traditional lead depressants, including dichromate, were not successful (WAI Report, January 2019) in rejecting or depressing the lead.

Zinc levels in copper concentrate were generally elevated and a function of the Cu/Zn ratios in the feed and the susceptibility of samples to weathering effects.

Arsenic levels in copper concentrate were generally higher than for MPY samples.

A series of locked cycle tests was conducted by HMT in 2018 to evaluate the metallurgy of disseminated samples containing varying amounts of penalty elements in the feed – results are shown in Table 13.33. Analyses of the feed can be found in Table 13.30.

Table 13.33 Distribution of Penalty Elements in Concentrates from Disseminated Samples Locked Cycle Tests

Test	Mass Pull (%)	Grade %				Distribution %			
		Cu	Pb	Zn	As	Cu	Pb	Zn	As
Copper Concentrate									
High Pb	0.68	23.49	14.28	6.11	0.30	65.64	25.59	4.45	3.06
Medium Pb	1.48	21.73	6.32	9.06	0.68	72.70	40.40	14.40	18.14
Low Pb	0.99	28.04	1.61	9.35	0.43	57.37	16.82	11.32	6.97
Zinc Concentrate									
High Pb	1.36	2.29	5.21	50.32	0.34	12.77	18.63	73.13	7.04
Medium Pb	1.35	2.96	2.26	48.45	0.54	9.10	13.20	70.40	13.22
Low Pb	1.60	6.48	1.53	36.81	1.15	21.42	25.78	72.03	29.99

13.4.5.2 Metallurgical Domain – Enriched

Enriched ore comprises approximately 5% of the tonnes in the sulphide reserve (model 2018), but 17% of the contained copper and 6.5% of the contained zinc, due to its high grade. The zinc minerals have been pre-activated in situ and do not respond as effectively as the massive pyrite material, to the depressant regime used in the standard flowsheet resulting in poor selectivity between copper and zinc in the copper roughers as shown in Figure 13.20.

Figure 13.20 Enriched Ore – Lack of Selectivity to Standard Flowsheet

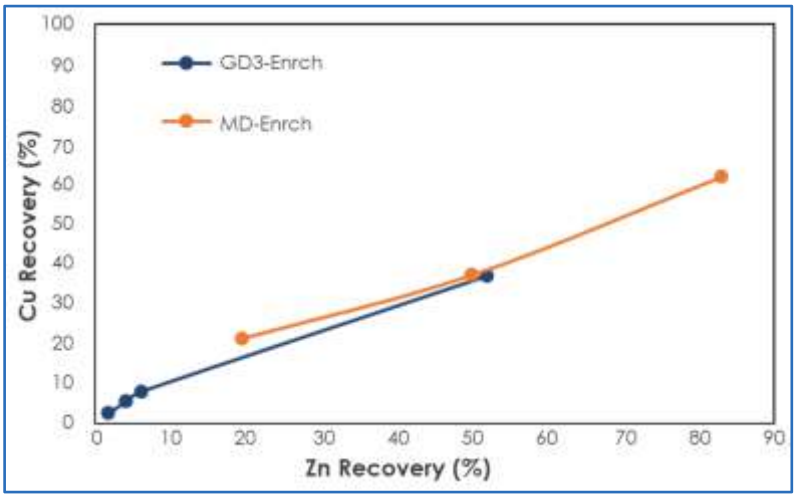


Figure from GRES, 2019.

An alternative approach to realising value from the enriched material was to evaluate the effect of blending various amounts of enriched with a typical master composite containing massive pyrite and disseminated material types. The results in Table 13.34 showed that up to 10% (and possibly 20%) of enriched material could be blended and still produce close to saleable grade 'complex' copper concentrate. The blend also showed that the enriched adds value due to its high grade. It should be noted that the enriched sample tested had a Cu/Zn ratio of 0.59, which is significantly lower than the average Cu/Zn ratio of 1.21 in the current mine production schedule. The higher Cu/Zn ratio would proportionately reduce the grade of zinc in the copper concentrate, assuming the same %Zn recovery.

Table 13.34 Effect of 90:10 Enriched Ore Blend on Concentrate Quality

Sample	Feed			Copper Concentrate					Zinc Concentrate				
	Wt (%)	Grade (%)		Wt (%)	Grade (%)		Recovery (%)		Wt (%)	Grade (%)		Recovery (%)	
		Cu	Zn		Cu	Zn	Cu	Zn		Cu	Zn	Cu	Zn
Enriched Assay Feed	10	2.57	4.35										
MCS LCT (HMT Dec'17)	90	0.67	1.50	1.44	28.74	4.60	68.6	4.9	2.30	1.98	52.90	6.8	81.1
Enriched (calc'd by difference)	10	2.67	5.50	0.55	32.94	29.60	67.7	29.5	0.50	6.36	62.00	11.9	56.4
90:10 Blend LCT	100	0.87	1.90	1.99	29.90	11.50	68.1	12.2	2.80	2.60	50.18	8.4	73.9

The test results showed that for 100% enriched material under the standard flowsheet conditions (see Figure 13.20), about 70% of the zinc and 50% of the copper was recovered into the copper rougher concentrate. However, in the 90:10 Blend, the results of back-calculation (Table 13.34) show only 29.5% of the zinc and 67.7% of the copper in the enriched material was recovered into the final copper concentrate, i.e. much-improved selectivity. A possible reason for this apparent synergistic effect is that the high amount of reagents added to depress the zinc in the 100% enriched sample also depressed the copper minerals in the sample, whereas the amount of depressant per unit of copper in the 90:10 blend was much lower.

Based on the poor results of treating 100% enriched material in the standard flowsheet, HMT evaluated alternative flowsheet conditions. Some success was achieved by partitioning the enriched material into a high Cu/Zn ratio feed and a high Zn/Cu ratio feed. The high Cu/Zn material would produce only a copper concentrate, with the zinc reporting to the tails, and the high Zn/Cu feed would produce only a zinc concentrate. However, this approach is complex, and would preferably be performed in a separate circuit, requiring additional capital. This approach was not evaluated further in PFS19.

The main conclusion is that processing of enriched ore presents challenges due to the pre-activation of zinc in situ resulting in a relatively high proportion of zinc reporting to the copper concentrate. This may be further affected by the weathering of mined ore in stockpiles prior to feeding to the mill. The mining schedule in PFS19 is based on a blending constraint that limits the enriched ore feed to the mill at < 10%, but will require enriched stockpiles of up to 40 kt in some months of year-4.

In future studies the following options should be evaluated:

- Improved scheduling to maintain more of the unbroken enriched ore in the mine until needed in the mill.
- Relaxing the constraint of < 10% enriched in the mill feed to approximately < 15%.
- Relaxing the constraint on the zinc content in copper concentrate by producing and selling more as 'complex' concentrate (rejection limit 10% Zn) at the expense of less standard concentrate (rejection limit 7% Zn). The indicative marketing terms allow for up to 50% of the copper concentrate to be sold as a 'complex', while the current financial model shows < 10%.
- Blending of concentrates at the port to meet marketing requirements for each batch shipped.
- Further testwork to establish the impact of the Cu/Zn ratio of the feed.
- Testwork on samples representing the spatial location, depth, and ratios of feed grades (Cu, Pb, and Zn) for each of the ore types and blends is planned for 2019, following drilling of some fresh core.

13.4.6 Concentrate Quality

Detailed analysis of copper and zinc concentrates produced from master composites is shown in Table 13.35. These are from the SGS 2016 report for LCT1 and from HMT tests completed in 2018 and reported in the 2018 phase 2 report.

Table 13.35 Detailed Analysis of Copper and Zinc Concentrates from Master Composites

Element	Unit	SGS LCT1 Cu Cleaner 4 Conc. Cycle 8	HMT 2018 MCS LCT1 Cu Conc.	SGS LCT1 Zn Cleaner 4 Conc. Cycle 8	HMT 2018 MCS LCT1 Zn Conc.
Au	g/t	4.53	6.92	3.77	1.37
Ag	g/t	112	323	147	91.3
Al_2O_3	%	0.86	0.06	< 0.01	< 0.04
As	ppm	370	2,820	1,320	1,900
Bi	ppm	132	742	259	198
Cd	ppm	50	194.5	973	1,750
Cl	ppm	100	60	2,100	160
Co	ppm	< 10	5	< 10	3
Cr	ppm	100	110	< 100	70
Cu	%	30.39	30.47	2.84	1.98
F	ppm	100	160	100	< 20
Fe	%	23.78	23	6.13	8.01
Hg	ppm	1	4.51	17	23.1
MgO	%	1.74	0.56	< 0.01	0.1
Mn	ppm	< 100	70	< 100	150
Mo	ppm	< 10	10.3	< 10	3.8
Ni	ppm	10	22	10	11
Pb	%	1.23	2.29	2.77	0.354
S	%	31.75	33.2	34.35	35.9
Sb	ppm	105	985	204	388
Se	ppm	59	400	45	80
SiO_2	%	4.05	1.5	2.04	0.2
Te	ppm	< 1	4.1	< 1	0.6
Zn	%	2.78	5.2	51.52	51.64
CaO	%	0.02		0.09	
CO_2	%		0.3		0.4

13.4.7 Regrind Power Tests – Rougher Concentrates

Signature plot tests were completed by Grinding Solutions Ltd (GSL; Truro, England) on the SGS rougher concentrates generated from bulk batch flotation tests (August 2016) and on the rougher concentrates produced in the WAI pilot plant (July 2018). The results of the tests are summarised in Table 13.36.

The finer feed size shown for the WAI sample reflects the finer primary grind size – P_{80} of 38 µm (WAI) compared to 45 µm (SGS). The 2016 tests were batch tests conducted in a stirred bead mill (see Figure 13.21(A)) intended to replicate a Metso Stirred Media Detritor (SMD) and the 2018 tests used a Netzsch LM4 horizontal mill, a replica of Glencore's IsaMill (see Figure 13.21(B)).

Note that the 'continuous' test procedure used by GSL is the same as applied by Glencore, however GSL is not an accredited Glencore test laboratory. The signature plots generated are shown in Figure 13.22.

Figure 13.21 GSL Test Equipment – (A) SMD Mill and (B) Netzsch LM4

Figure from GRES, 2019.

The media wear rate was estimated from the 2018 continuous tests to be 15–17 g/kWh. The pulp conditions were measured with both the copper and zinc discharge being devoid of dissolved oxygen (0 ppm DO_2) and the copper regrind product had a slightly reducing potential (–85 mV) at a neutral pH of 7.5, while the zinc discharge was slightly oxidising (+34 mV) with an alkaline pH of 9.3.

The results of the LM4 continuous tests have been used for design to size the power requirements for the regrind mills due to GRES's experience that the IsaMill signature plots are representative of plant requirements while the alternative procedures tend to underestimate the power required for a specific duty. In support of this, the Netzsch LM4 signature plots realised higher specific energies than the SMD batch tests.

Table 13.36 Summary of Regrind Power Tests on Copper and Zinc Rougher Concentrates

Parameter	Units	Copper Aug'16	Copper Jul'18	Zinc Aug'16	Zinc Jul'18
Test Procedure		Batch	Continuous	Batch	Continuous
Test Equipment		SMD	Netzsch LM4	SMD	Netzsch LM4
Test Mill Volume	L	1.5	4	1.5	4
Media Type		Kings CM270	Magotteau x M	Kings CM270	Magotteau x M
Media Size	mm	3	3	3	3
Media Density		2.65	3.6	2.65	3.6
Feed Size					
- F_{95}	µm	63.7	55	69	43
- F_{80}	µm	38.3	27.7	45.9	26.5
Feed Density	% solids	45	49.1	45	47.4
Concentrate Density		3.0 (assumed)	4.03	3.0 (assumed)	3.90
Product Size					
- P_{90}	µm	21.8	22	28.7	28
- P_{80}	µm	14.7	15	22.5	20
Signature Plot Equation P_{80} Size	y=kWh/t	y=17287x –2.542	y = 285652x –3.369	y = 4552.1x –2.001	y = 61953x –2.671
Specific Energy (Target P_{80})	kWh/t	17.7	31.2	11.5	20.6
Media Consumption / Wear	g/kWh		17.4		15.3

Figure 13.22 Signature Plots – SMD and IsaMill

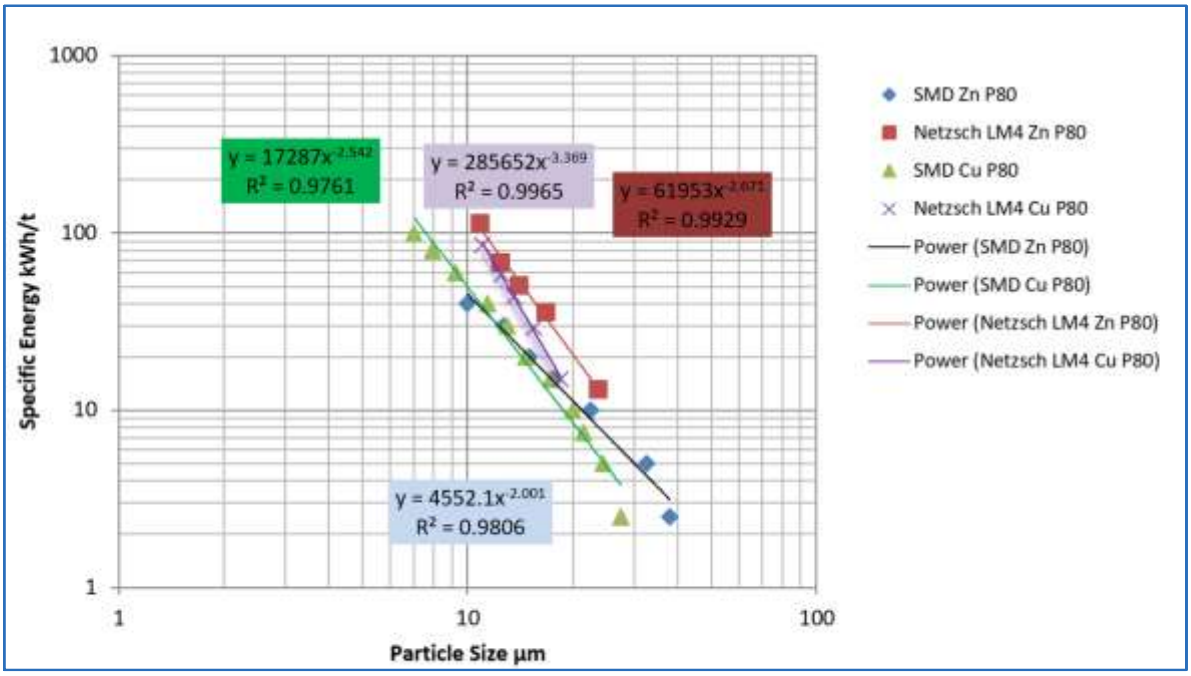


Figure from GRES, 2019.

13.4.8 Tailings Disposal

Static cylinder settling tests were completed on flotation tailing by RDI (August 2014), SGS (2016), and WAI (2018) (Table 13.37). The thickening tests of flotation tailing from the WAI pilot plant were done by Paterson and Cooke ('Gediktepe Testwork', report WTT-51-0185, 8 August 2018) and included high rate and compression (paste) procedures.

Flocculant screening tests concluded that Magnafloc 5250 gave faster settling rates and clearer supernatant than the other flocculants tested. The optimum feed density was established to be 10% solids and tests indicated there was a risk of over-flocculation.

Table 13.37 Tailings Thickening Tests

Description	Units	RDI (2014)	RDI (2014)	SGS (2016)	SGS (2016)	WAI (2018)		
Test		**Static Cylinder**	**Static Cylinder**	**Static Cylinder**	**Static Cylinder**	**100 mm Dynamic**		
Feed Size – P_{80}	µm	45	45			39		
Feed Size – %Passing 10 µm	%					28		
Feed Density	% solids	21	22	12.4	10.8	10		
Solids Specific Gravity		3.36	3.33	3.30	3.30	4.24		
Feed pH		11.1	11.0	11.0	11.0	7.47		
Flocculant			Anionic	Nasfloc 2225 Anionic	Nasfloc 2225 Anionic	M5250		
Flocculant Addition	g/t	0	20	15	25	14.7	15	15
Underflow Density	% solids	50	50	47.3	42.3	71	68.1	71.8
Underflow Yield Stress	Pa					63	46	84
Specific Settling Flux Rate	m^2/tpd	0.032	0.012	0.278	0.308	0.068	0.054	0.087
	t/m^2h	1.3	3.47	0.15	0.135	0.612	0.765	0.48
Underflow Density	% solids	55	55	50.3	45.2			
Specific Settling Flux Rate	m^2/tpd	0.038	0.035	0.366	0.405			
	t/m^2h	1.35	1.19	0.114	0.102			
Turbidity/Clarity	NTU or mg/L	31.4 NTU	24.6 NTU			560	830	380

Paterson & Cooke noted the fast settling rate of the solids to develop a high density underflow (with implications for the torque loading on the rake mechanism) and the shear-thinning nature of the underflow when subjected to pumping. High compression / paste consolidation testing produced a 79% solids underflow after three hours and 82% solids after 24 hours. High wall height thickeners (2.7 m minimum for high-rate thickener selection – preferably 3 m) and a relatively steep base cone angle of > 7° were recommended.

13.4.9 Concentrate Handling

SGS (October 2016) completed settling and filtration tests on rougher concentrate samples (17% Cu, 11.7% Zn, 23.4% Fe, and 35.7% S for copper concentrate, and 15.3% Zn, 30.3% Fe, and 44.6% S for the zinc concentrate), which had been reground to 97% passing 20 µm, resulting in the testwork samples having a P_{80} of 7.6 µm for the copper rougher concentrate and P_{80} of 17.8 µm for the zinc rougher concentrate – the copper being significantly finer than the current design P_{80} of 15 µm. Also, due to limited copper rougher concentrate sample, only static settling tests could be completed. The results of the testwork were reported in 'Settling, Filtration, Grindability and Flotation Tests on Samples from Gediktepe Ore' Project No 10866-647, 25 October 2016.

FLS was engaged by WAI to undertake thickening and filtration tests on the concentrates generated from the pilot plant – 'Gediktepe Sedimentation and Filtration Testwork', P2395 report ZT64-0609, 20 August 2018.

Thickening test results are summarised in Table 13.38. The FLS results indicate the concentrates are fast settling and low thickener areas are required with specific flux values of over 1.0 t/m²h. These flux rates are significantly higher than any plant operating data available to GRES. Consequently, a flux rate of 0.25 t/m²h, the industry standard, which allows for aerated froths, has been applied in design. Underflow densities of 60% solids for copper concentrate and 65% solids for zinc concentrate have been selected for design.

FLS completed vacuum, pressure and Pneuma-press filtration tests on the concentrates (Table 13.39). The feed density for all tests was 60% solids. The minimum cake moisture achieved from vacuum filtration was 21% for both concentrates, significantly higher than the transportable moisture limits (TMLs) of 13.2% moisture. The Pneuma-press filter did achieve moistures below the TML for one test on each concentrate however experience at Porgera (pyrite concentrate) and TasMines (magnetite concentrate) indicate inconsistent performance for fine feeds as will be required at Gediktepe.

Table 13.38 Summary of Thickening Tests on Copper and Zinc Concentrates

Description	**Units**	**SGS (2016)**	**SGS (2016)**	**FLS (2018)**	**FLS (2018)**
Test		**Cu Rougher Conc. Static**	**Zn Rougher Conc. Dynamic**	**Cu Conc. 100 mm Raked**	**Zn Conc. 100 mm Raked**
Feed Density	% solids			6	6
Solids Specific Gravity					
Flocculant		BASF M333 Anionic	BASF M333 Anionic	M10	M10
Flocculant Addition	g/t	35	20	30	30
Underflow Density	% solids	53	70.7	62.5	63.3
Yield Stress	Pa			16	25
Flocculant Addition	g/t			40	40
Underflow Density	% solids			63.8	65
Specific Settling Flux Rate	m²/tpd	0.2	0.09	0.037	0.04
	t/m²h	0.21	0.463	1.13	1.04
Overflow Clarity	mg/L		0.463	5	25

Table 13.39 Summary of Pressure Filtration Tests on Copper and Zinc Concentrates

Description	Units	Copper Concentrate			Zinc Concentrate		
Feed Size – P_{80}	µm	15.3			24.4		
Feed Size – % Passing 10 µm	%	71			57		
Cake Thickness	mm	25.5	33.6	51.4	26.2	33.6	51.3
Cake Density (dry)	kg/m³	2,095	2,133	2,137	2,060	2,136	2,100
Moisture	% solids	10.0	10.8	12.1	11.3	11.8	15.2
Specific Filtration Rate	kg/m²h	137	170	233	138	193	267
Target Moisture	% solids	12	12	12	12	12	
Filtration Rate	kg/m²h	201	241	230	175	213	

FLS did not provide details of filtration rate calculations or cycle times (only filtration feed and air blow times were reported). The cake density values have been used to size the plate and frame filters selected in the process design with vendor advice on cycle times. Chamber depths of 30 mm to 40 mm have been recommended based on the testwork that used 25 mm, 32 mm, and 50 mm deep chambers.

TMLs for the concentrates generated in the WAI pilot plant were measured by Bureau Veritas, Estonia (Certificates EEESTJ 18001885, 17 August 2018) to be 13.32% for the copper and 13.2% for the zinc. Bureau Veritas also completed self-heating tests (UN Test N.4) that concluded both concentrates were negative (< 60°C temperature rise) and not classified as MHB (Materials Hazardous only in Bulk, Division 4.2) having a variance of +21.1°C for copper and +0.4°C for zinc. Despite these results, the copper concentrate, in particular when secondary copper minerals are present, may become self-heating due to the fine particle size distribution, changes in moisture and oxidation of the sulphides over time.

13.4.10 Flowsheet Selection – Sulphide

The 6,500 tpd flowsheet for processing the sulphide ores has not changed significantly since the development of the sequential (differential) flotation process, developed by HMT in 2015. The three-stage crush–two-stage ball mill grinding circuits have been replaced with a single crushing stage and a SAG–pebble crush–ball mill grinding circuit to produce a flotation feed P_{80} size of 38 µm. The SAG mill will be the same unit as used for the single stage milling of the oxide ore.

Reagents will be added to the mill and the ground product will be pumped to the pre-float section for removal of naturally floating non-sulphide gangue, mostly talc. Pre-float concentrate will be directed to final tailings and the pre-float tails will be discharged to the copper roughers, operating at a natural pH 6.5.

Rougher concentrate will be reground to 15 µm and sent to a three-stage cleaner circuit. First cleaner tails will be sent to cleaner scavengers, the concentrate from which will be returned to the regrind mill and the tailings directed to the zinc circuit. Depressants will be added in the copper cleaner circuit to depress zinc sulphides and pyrite, producing a final copper concentrate of about 30% Cu.

Copper rougher tailings will be combined with copper cleaner scavenger tails and treated in the zinc roughers, where pH will be increased to pH 11 with the addition of lime. Zinc rougher concentrate will be reground to 20 µm (P_{80}) and then upgraded to above 50% Zn in a three-stage cleaner circuit.

The combined zinc rougher tail and zinc cleaner scavenger tail will be pumped to the tailings storage facility after thickening. Copper and zinc final concentrates will be thickened then pressure-filtered to less than 12% moisture and discharged into the storage shed prior to trucking to the port facilities.

Overflows from the thickeners will be combined with the reclaim water from the tailings impoundment and processed through the water treatment plant for removal of organic reagents by activated carbon. The treated water will then be returned to the process.

13.4.11 Metallurgical Projections for Financial Model – Sulphide

The metallurgical model is based on estimates of concentrate grades and recoveries from the three ore types; massive pyrite, disseminated, and enriched. The individual components from the mine production schedule are then summed to produce the expected quantity and quality of copper and zinc concentrate by period or quarter. Blending of concentrate will be necessary to maintain products within the smelter specifications.

Three types of concentrates will be produced.

- Standard copper concentrate: containing > 20% Cu, < 7% Zn , < 2.5% Pb
- Complex copper concentrate: containing > 20% Cu, < 10% Zn, < 6% Pb
- Zinc concentrate: > 50% Zn, < 5% Cu, < 5% Pb

Treatment and refining charges (TC/RC) along with penalties are discussed in detail in Section 19.

The average concentrate grades and recoveries for the sulphide resource for each feed type are shown in Table 13.40. The estimates are based on the following analysis:

- Head grade effects (e.g. copper recovery is related to copper in feed)
- Fixed concentrate grades for the primary metals in each concentrate (copper in copper concentrate, zinc in zinc concentrate) identified with 't' in Table 13.40.
- Fixed recoveries for other metals in the copper concentrate, identified with '*' in Table 13.40.
- Mass balances and stage recoveries to calculate grades of metals in zinc concentrate.
- Enriched ore recoveries and grades assume a maximum blend of 10% enriched material in the feed. Enriched with a Cu/Zn ratio < 0.75 is considered to be waste.

Table 13.40 Average Concentrate Grades and Recoveries from the Sulphide Ore

Copper Concentrate						
Grades	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Au (g/t)**	**Ag (g/t)**	**As (%)**
Massive Pyrite	30.00 *	2.41	1.26	11.94	178.05	0.07
Disseminated	25.80 *	6.45	6.66	10.79	358.99	0.38
Enriched	31.90 *	8.69	1.24	3.82	84.53	0.45
Recovery	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Au (%)**	**Ag (%)**	**As (%)**
Massive Pyrite	68.46	2.24	5.85	26.21	10.49	2.47
Disseminated	69.21	7.04	33.51	26.71	22.07	11.49
Enriched	67.70 †	29.50 †	45.50 †	10.00 †	10.00 †	50.00 †
Zinc Concentrate						
Grades	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Au (g/t)**	**Ag (g/t)**	**As (%)**
Massive Pyrite	1.82	58.14 *	2.64	1.62	124.07	0.05 *
Disseminated	3.40	49.72 *	2.53	2.70	213.14	0.48 *
Enriched	4.01	53.13 *	1.33	3.12	147.13	0.13
Recovery	**Cu (%)**	**Zn (%)**	**Pb (%)**	**Au (%)**	**Ag (%)**	**As (%)**
Massive Pyrite	6.03	79.01	18.30	5.28	10.88	2.51
Disseminated	12.74	75.92	17.85	9.39	18.41	19.98
Enriched	4.51	56.40 †	13.78 †	9.98 †	9.98 †	5.96 †

† Fixed recoveries
* Fixed concentrate grades

Processing of enriched ore presents some challenges due to the pre-activation of zinc in situ resulting in a relatively high proportion of zinc reporting to the copper concentrate. This may be further affected by the weathering of mined ore in stockpiles prior to feeding to the mill. The first pass schedule included in this report is based on a blending constraint that limits the enriched ore feed to the mill at < 10%, but will require enriched stockpiles of up to 40 kt in some months of year-4. It should be noted that this is a limited effect as in the remaining years the enriched stockpile levels will generally be less than 5 kt.

In the feasibility study the following options will be evaluated:

- Improved scheduling to maintain more of the unbroken enriched ore in the mine until needed in the mill.
- Relaxing the constraint of < 10% enriched in the mill feed to approximately < 15%.
- Relaxing the constraint on the zinc content in copper concentrate by producing and selling more as ‘complex’ concentrate (rejection limit 10% Zn) at the expense of less standard grade (rejection limit 7% Zn). The current financial model shows < 10% as ‘complex’ versus a maximum of 50% allowed under the indicative marketing terms.
- Blending of off-specification concentrate at the port warehouse.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The 2018 update of the Mineral Resources for the Gediktepe project was completed by AMC, based on available diamond core and reverse circulation drilling data, geological, mineralisation, structural, and weathering interpretations prepared by Polimetal, and supplementary mineralisation-constraining interpretations prepared by AMC.

A cell model extending beyond the mineralisation limits and covering the area shown in Figure 14.1, was constructed and truncated by topography. Domain codes were embedded in the model cells to represent volumes of geological units, and mineralisation and weathering zones. The sample dataset was coded in a corresponding fashion and statistical and geostatistical evaluations were undertaken to inform estimation of the major grades of economic interest (Au, Ag, Cu, and Zn) and minor grades (As, C, Pb, S, Fe, and Hg), along with bulk densities, into the mineralisation domains and background material in the cell model.

The modelled estimates were assessed for levels of geological confidence and classified into Measured, Indicated, and Inferred categories, referencing CIM guidelines (CIM, 2014). The Mineral Resource tonnages and grades were reported using net smelter return (NSR) cut-offs and constrained within an optimised pit.

Figure 14.1 Gediktepe Plan showing Cell Model Extents and Drillhole Collars

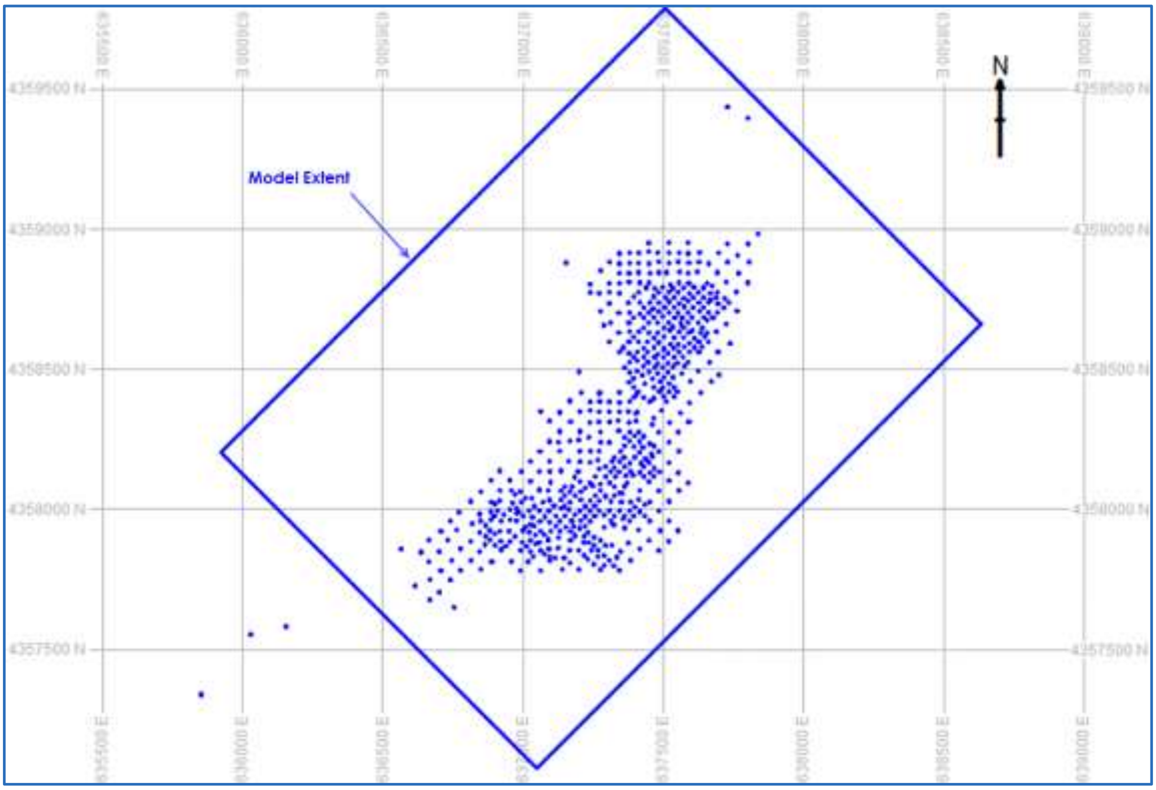


Figure from Polimetal, 2018.

14.2 Drilling and Sampling Data

The final drilling database files were released by Polimetal on 21 March 2018. Table 14.1 summarises these database files. A series of basic and standard checks of database validity were undertaken and no matters of concern were identified. Based on the validation comparisons reported in Section 12, the diamond core (DD) and reverse circulation (RC) drilling data were deemed to be valid inputs for interpretation and resource estimation.

Table 14.1 Gediktepe Drillhole Database Files as at 21 March 2018

Database File	No. of Records	Description
Gediktepe_Collar_20180321.xls	730	Drillhole collars
Gediktepe_Survey_201800321.xlsx	2,160	Drillhole downhole surveys
Gediktepe_Lithology_20180321.xlsx	43,926	Geological logs
Gediktepe_All_Assay_MasterData_20180321.xls	38,003	Sample assays
Gediktepe_Specific_Gravity_20180321.xls	6,262	Bulk density measurements

A quantity of drillholes that were drilled for reasons other than resource definition (i.e. hydrological, geotechnical, seismic, metallurgical, etc) were considered not relevant for geological modelling and resource estimation. These drillholes were excluded from the PFS19 Drillhole Dataset.

The total number of drillholes included in the PFS19 Drillhole Dataset is 629, totalling 70,127 m. A summary of the drillhole series (identified by Hole Type) that were retained in the PFS19 Drillhole Dataset is shown in Table 10.2 and Table 10.3.

14.3 Geological Interpretations

14.3.1 Lithologies

Polimetal supplied a set of wireframe solids representing interpretations of the three main schist lithologies plus overburden based on drillhole logging. The wireframes had been assigned codes corresponding to lithology (LTHZONE), as shown in Table 14.2.

Table 14.2 Lithology codes

LTHZONE	Description
OVBN	Overburden
SHQF	Quartz–Feldspar Schist
SHCS	Chlorite–Sericite Schist
SHQZ	Quartz Schist

Figure 14.2 shows an oblique view of the lithology wireframes.

Figure 14.2 Oblique View of Lithology Interpretations

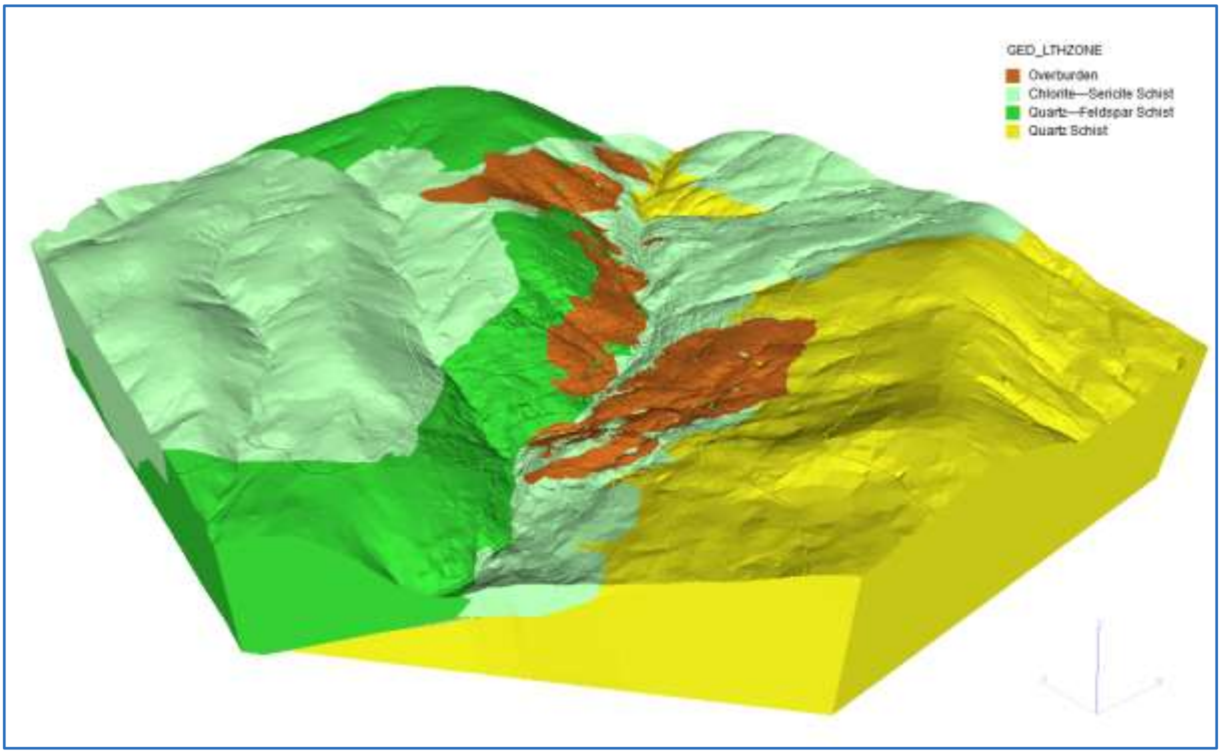


Figure from Polimetal, 2018.
Elevated view towards north–north-east.

14.3.2 Weathering (Oxidation)

Based on drillhole logging of the degree of oxidation, Polimetal prepared a wireframe surface representing the interpreted base-of-oxidation (top-of-sulphide) horizon. The oxidation surface was extrapolated laterally beyond the area of drillhole coverage to follow topography. It was also offset vertically downwards to reflect the tendency for the base-of-oxidation to follow the water table, which tends to track a profile below the topography.

Subsequently, the oxidation surface was also extended to the model boundaries by similarly following the topography, with a downward offset of approximately 10 m.

Figure 14.3 shows an oblique view of the interpreted base-of-oxidation surface.

Figure 14.3 Oblique View of Base-of-Oxidation (Top-of-Sulphide) Surface

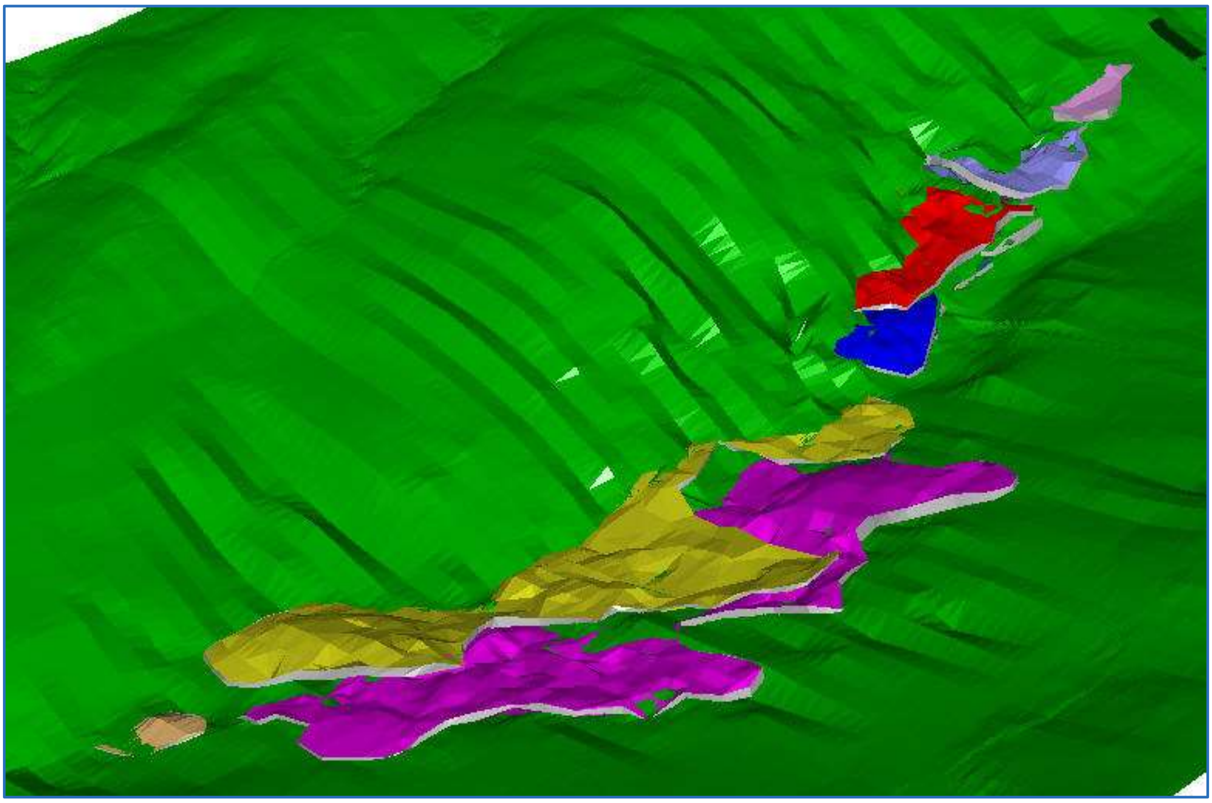


Figure from Polimetal, 2018.
Elevated view looking approximately towards north.
Oxidation surface shown in green. Low-grade mineralisation shell solid wireframes are shown for reference.

14.3.3 Mineralisation

14.3.3.1 Massive Pyrite and Gossan Solids

Polimetal prepared four sets of mineralisation interpretations, in the form of solid wireframes, based on drillhole logging and assay results.

The gossan and clay-like gossan wireframes are confined to the oxide zone, while the massive pyrite and enriched wireframes fall within the sulphide zone. Previous studies distinguished between massive pyrite and massive pyrite-magnetite. However, evaluations by Hacettepe Mineral Technologies (HMT), GRES, Polimetal, and others indicated that there was no benefit, from either a resource estimation or mineral processing perspective, in partitioning the massive pyrite, therefore the massive pyrite and massive pyrite–magnetite have been combined into one domain for the current study.

Figure 14.4 shows an oblique view of the mineralisation interpretations.

Figure 14.4 Oblique View of Mineralisation Solid Interpretations

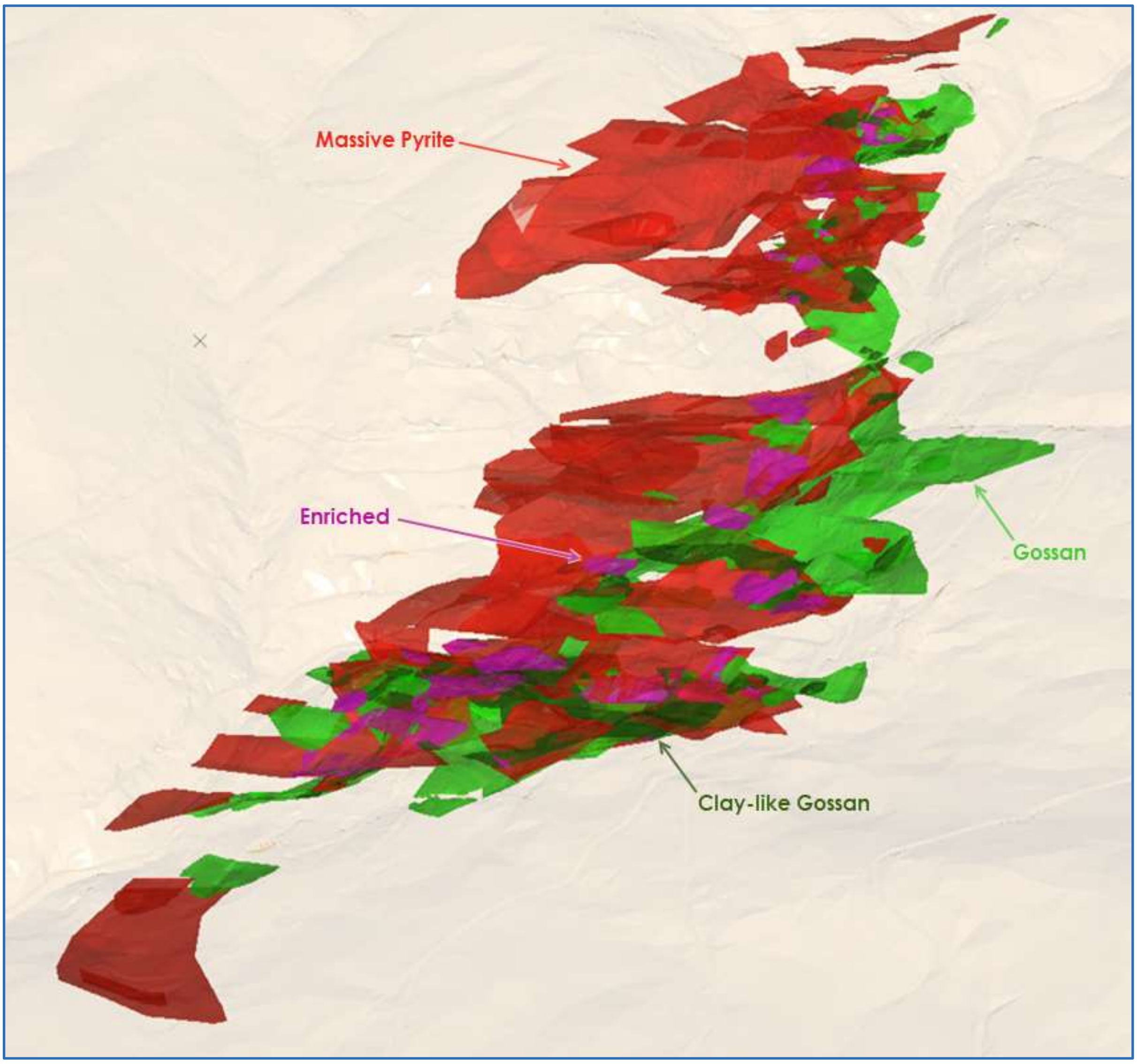


Figure from Polimetal, 2018.
Elevated view looking approximately towards north–north-east.

14.3.3.2 Low-Grade Mineralisation Shells

A set of solid wireframes were developed to capture and constrain lower grade mineralisation occurring outside of the mineralisation wireframes. The low-grade mineralisation shells were developed from interpreted strings developed by Polimetal. In many cases the boundaries and extents of the low-grade shell are relatively well defined, however, there are many instances where establishing continuity proved to be challenging. The existing mineralisation interpretations were used to inform and guide the low-grade shell interpretation. A significant number of mineralised intersections can still be observed outside of the low-grade mineralisation shells; these are usually isolated low grades, or downhole intervals in which some component grades are elevated (e.g. Zn) while others are not.

The low-grade mineralisation shell often imitates the boundaries of the massive pyrite or gossan zones, or extends laterally (up-dip, down-dip, or along strike) away from the interpreted mineralisation zones. Often these trends are not well defined by grade but can be observed in drillhole logs of disseminated sulphide and higher sulphur grades. The low-grade mineralisation shells occur as variable thicknesses around the massive pyrite or gossan interpretations.

There is evidence that the gossan and massive pyrite zones once formed continuous bodies of mineralisation across what is now interpreted as the base-of-oxidation. The low-grade mineralisation shell interpretations were therefore constructed without consideration of the base-of-oxidation boundary; any subsequent need for partitioning between oxide and sulphide can be achieved using the base-of-oxidation surface and associated domain coding.

There are 11 low-grade mineralisation shell solids, mostly encapsulating the mineralisation, but at times deviating locally to honour interpreted continuity. Such deviations are not considered to have a material impact, given that the primary objective of these shells was to constrain the grade estimation processes outside of the main mineralised zones.

Figure 14.5 shows an oblique view of the low-grade mineralisation shell interpretations.

Figure 14.5 Oblique View of Low-Grade Mineralisation Shell Interpretations

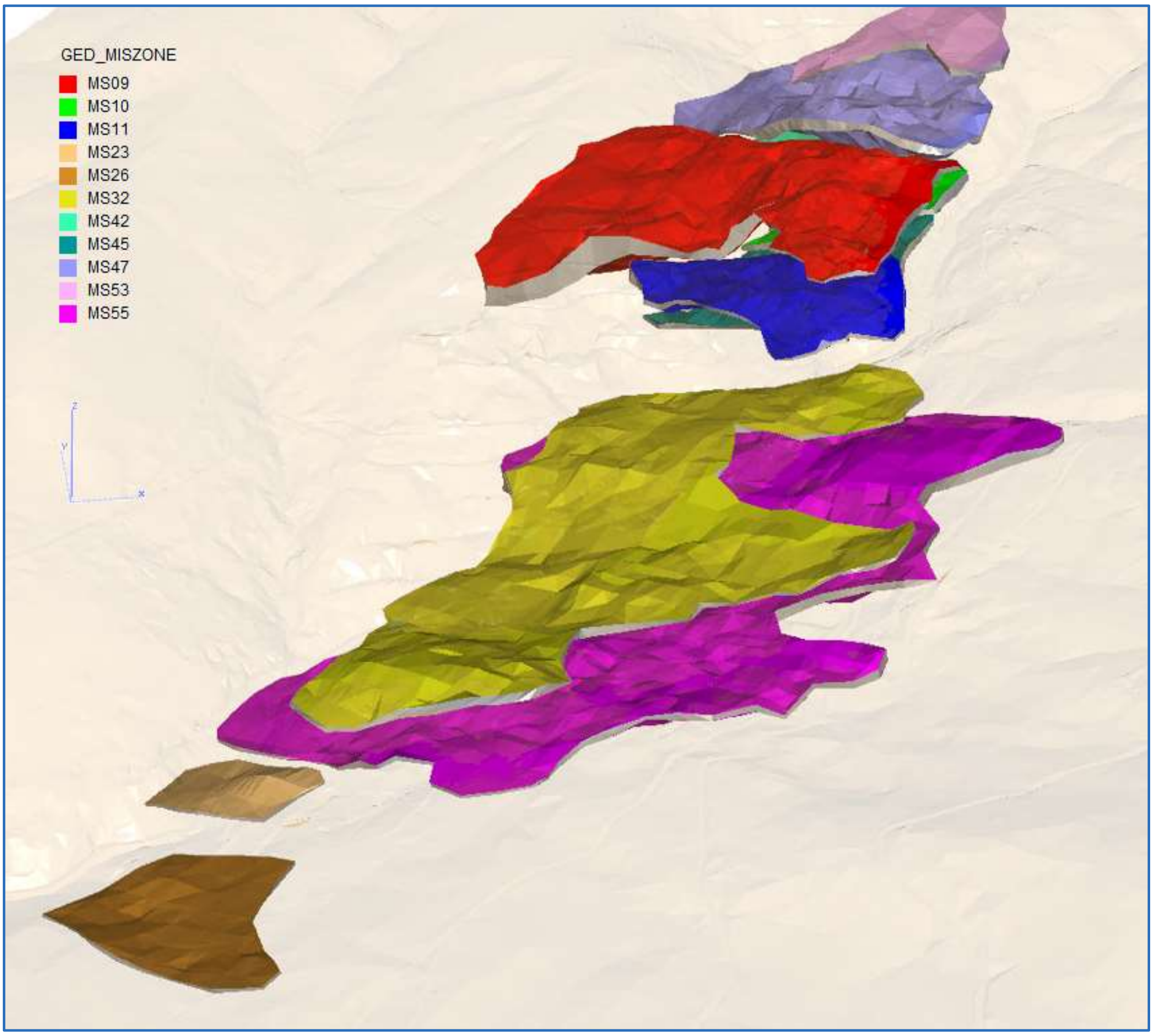


Figure from Polimetal, 2018.
Elevated view towards north–north-east.

14.3.3.3 Mineralisation Sub-Zones

During the review of the spatial distributions of grades across the Gediktepe deposit, a range of patterns were observed, often specific to individual metals or showing consistency between two or more elements. Two particularly marked distribution patterns showed potential for compromising the grade estimation outcomes unless a strategy could be developed for managing them. It was concluded that sub-zone interpretations were required to enable the segregation and individual management of these features.

The first mineralisation sub-zone relates to copper within the massive pyrite and was observed in plan view. Figure 14.6 shows a boundary string enclosing massive pyrite sample intervals with lower copper grades relative to the non-enclosed higher copper grades to the east (at the top of the image) and to the south (to the right in the image) of the interpreted boundary. The distinctive grade characteristics indicated a need for separate domaining of the higher and lower copper grade areas for further individual evaluation.

Figure 14.6 Copper Grade Distribution Zonation in Massive Pyrite

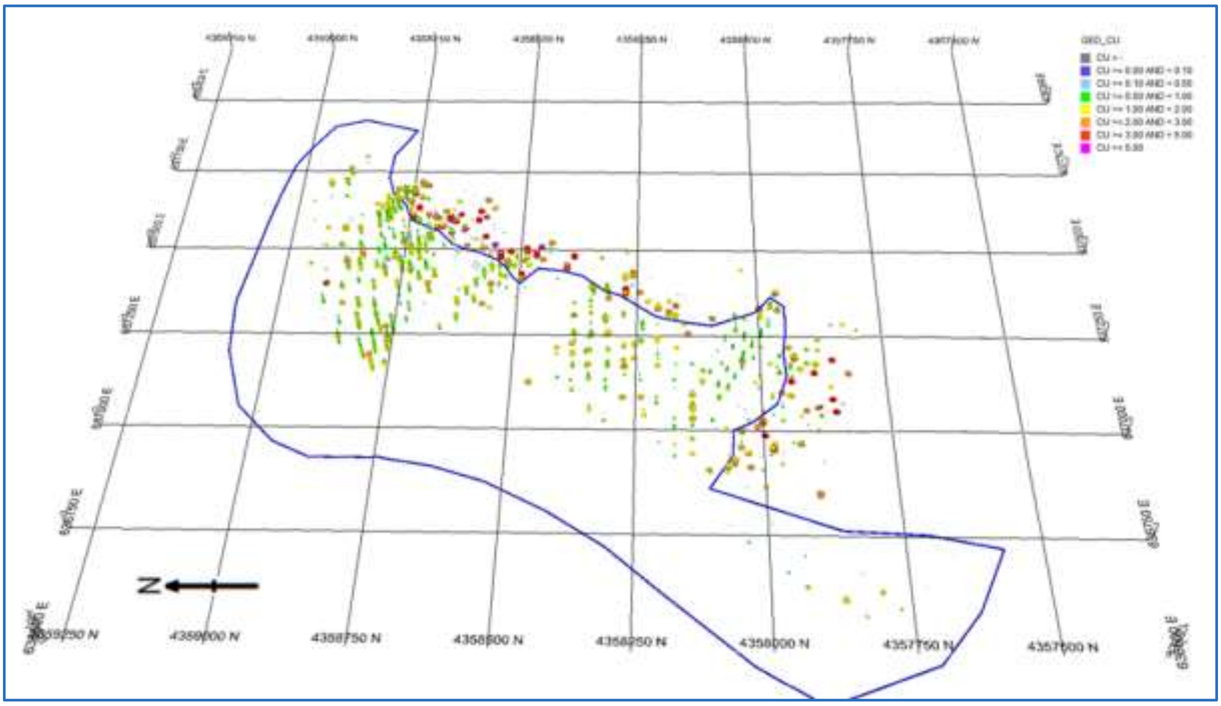


Figure from Polimetal, 2018.
Oblique view from above looking approximately east.

The second sub-zone was observed in cross-section, initially in relation to zinc grades. Figure 14.7 shows a cross-section with drillhole traces annotated with copper and zinc grades. From this cross-section it can be observed that, while the copper grade is generally strongly elevated across the full intersection of the massive pyrite, zinc remains depleted from the hangingwall through to some point within the intersection, after which the zinc grades increase sharply through to the footwall of the massive pyrite interval. This trend is consistent over a number of adjacent drillholes, both on the illustrated section and, to a lesser degree, on neighbouring sections.

To ensure that this clearly-defined trend in the zinc mineralisation could be represented, a wireframe surface linking the intersection points where zinc grades change sharply was created for use as a sub-zoning boundary.

Further inspection of the trend in other grades show that gold, silver, and mercury closely mirror this zinc trend, and minor component assays are also conformable (Mn, and Cd and Co (inversely)). These give further weight to recognising the zone as geochemically distinctive.

Figure 14.7 Depletion of Zinc in Massive Pyrite: Cross-Section

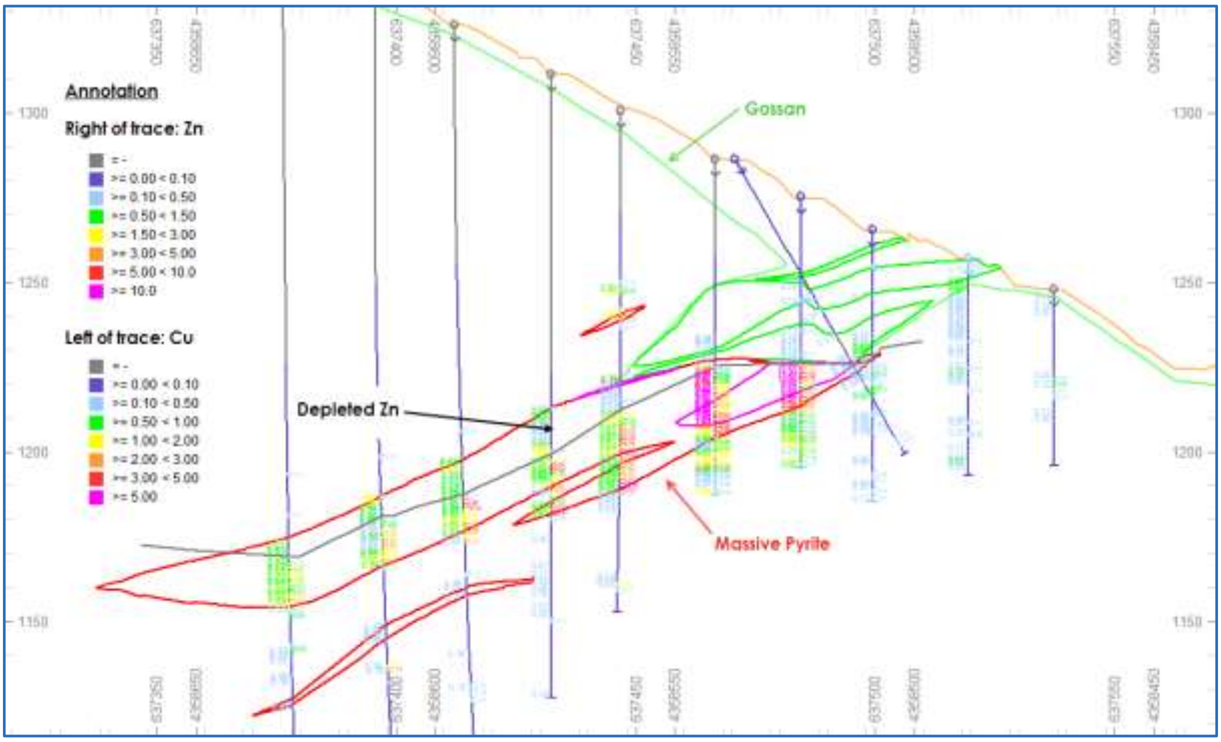


Figure from Polimetal, 2018.

14.3.4 Faults

The presence of faulting at the Gediktepe deposit is evident from a three-dimensional view of filtered grade values. In some cases, the dislocations are clearly defined, while in others the faults are more subtle.

Polimetal has identified and modelled the different interpreted faults as wireframe surfaces. The faults were used to define the limits to and offsets of the interpretations of mineralisation.

14.4 Sample Coding

Prior to domain-coding of the sample data, the drillholes listed in Table 14.3 were excluded from the drillhole dataset. These holes, which were drilled for metallurgical sampling purposes, had been retained for the geological interpretation phase of the work but were eliminated from further processing due to the absence of assay data.

Table 14.3 Drillhole Exclusions Prior to Sample Coding

BHID
DRD-413
DRD-414
DRD-415
DRD-417
DRD-431

The remaining drillhole intervals were coded with the relevant lithology, weathering (oxidation), and mineralisation interpretations. Table 14.4 shows the code fields for each of the domains and the methods used to apply the codes.

Table 14.4 Domain Coding Fields and Method of Application

Feature	Domain Field	Method
Lithology	LTHZONE	Within solids
Weathering	WEAZONE	Above / below base-of-oxidation surface
Mineralisation solids	MINZONE	Within solids
Mineralisation shells	MISZONE	Table of intersections

The coding used for the various lithological units is consistent with that shown in Table 14.2 (LTHZONE), while the coding for weathering (WEAZONE) is shown in Table 14.5. The coding for the mineralisation solids and low-grade mineralisation shells (MINZONE) is shown in Table 14.6.

Table 14.5 Weathering (Oxidation) Codes

WEAZONE	Description
WEAT	Weathered (oxide)
FRSH	Fresh (sulphide)

Table 14.6 Mineralisation Codes

MINZONE	Description
GOSS	Gossan
GSCL	Clay-like Gossan
MSPY	Massive Pyrite
MSEN	Enriched
MISZ	Low-grade Mineralisation Shell
BKGR	Background

An additional field, MISZONE, provides capability for filtering, analysing, and potentially estimating data according to individual low-grade mineralisation shells. The MISZONE codes are all of the form MSx, where x is a colour code.

14.5 Treatment of Unsampled Intervals

Unsampled intervals within the dataset can be attributed to a number of causes, including loss of sample during drilling, or intervals not selected for assaying due to being considered to be unmineralised. These unsampled intervals can range from isolated gaps typically at the standard 1 m or 2 m sampling lengths, or a sequence of missing samples downhole.

Intervals with absent grades were replaced with default values. This strategy was implemented prior to compositing of the raw domain-coded dataset.

The default values specific to each mineralised zone are shown in Table 14.7.

Table 14.7 Default Values for Unsampled Intervals

Component	Units	Domain	Value
Au	g/t	All	0.005
Ag	g/t	All	0.05
Zn	%	All	0.01
Cu	%	All	0.01
As	ppm	All	50
Hg	ppm	All	0.001
Pb	%	All	0.01
Fe	%	All	20
C	%	All	0.01
S	%	Weathered	0.001
		Fresh	10

14.5.1 Cancelled Drillholes

Nine drillholes in the PFS Drillhole Dataset are recorded in the collars database as having been cancelled for various reasons (listed in Table 14.8). These drillholes do not have any assay data associated with them, however they do have logging data, and as such were continued through the same process as the remainder of the drillhole dataset for use in the geological interpretation stage of work. These nine holes in their entirety have default grades assigned.

Table 14.8 Cancelled Drillholes – Assays Set to Default Grades

BHID
DRD-021
DRD-232
DRRC-027
DRRC-043
DRRC-056
DRRC-152
DRRC-176
DRRC-177
DRRC-179

14.6 Statistical and Geostatistical Analyses

14.6.1 Compositing

The sampling practice at Gediktepe was based on default sampling intervals of 1 m within the mineralised zones and 2 m outside of these zones, with exceptions permitted for shorter sample lengths to better honour the boundaries of geological features. The resultant frequency distribution of raw sample lengths is shown Figure 14.8 through Figure 14.10.

Figure 14.8 Distributions of Raw Sample Lengths – All Samples

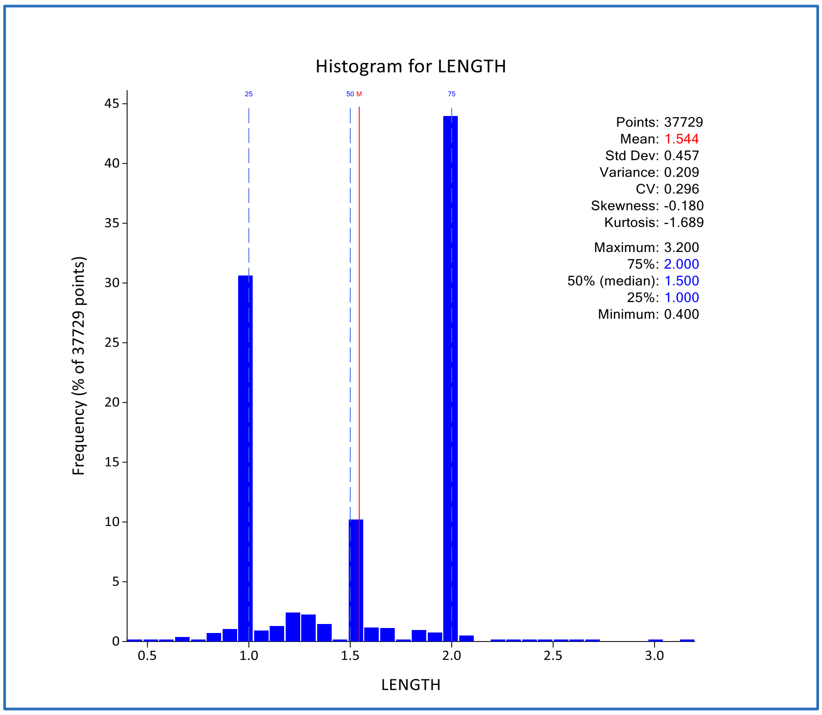


Figure from Polimetal, 2018.

Figure 14.9 Distributions of Raw Sample Lengths – Gossan and Clay-like Gossan

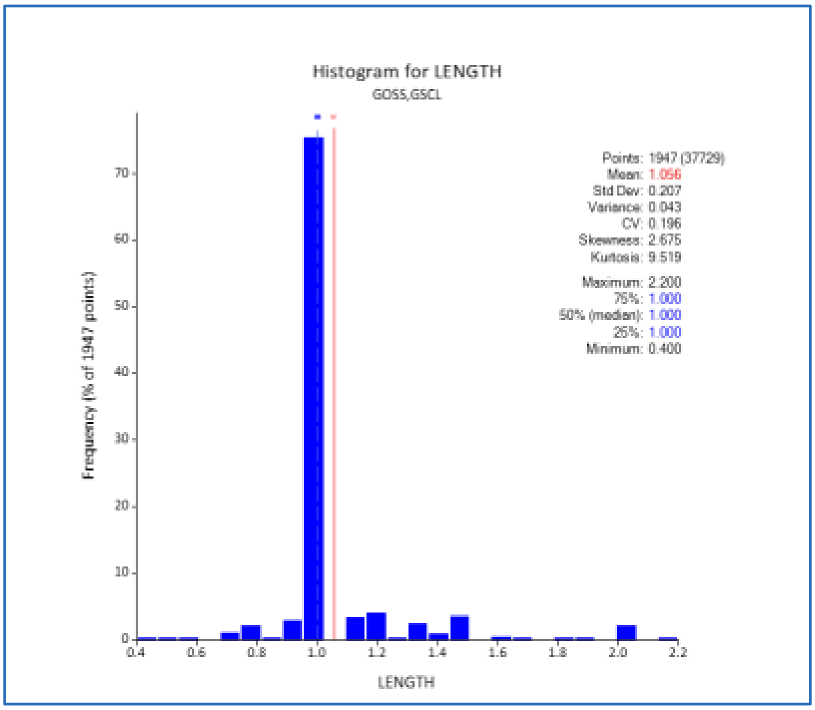


Figure from Polimetal, 2018.

Figure 14.10 Distributions of Raw Sample Lengths – Massive Pyrite

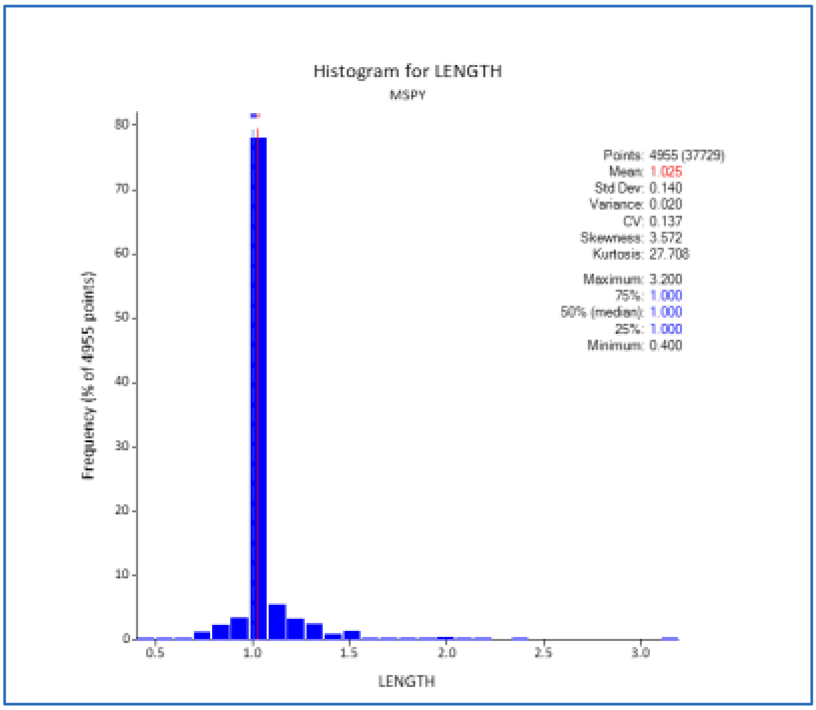


Figure from Polimetal, 2018.

The composting process used for the PFS19 geological modelling is one that leaves to the user's discretion the option of either: (1) an 'exact length' approach, or (2) an 'approximate-but-equal length' approach.

In the 'exact length' compositing approach, the majority of the samples within a like-domain segment down the drillhole are composited to be exactly the user-defined composite length, and any smaller remnant samples that result at boundaries are either retained at that shorter length or discarded, depending on a parameter used to define the minimum allowable composite length, (defaults to 0.5 x user-defined interval). This approach generally results in a composite database that is largely exactly the user-defined composite length, with some shorter composites and/or some missing composites, depending on whether the remnant samples met or failed to meet the minimum allowable length setting.

In the 'approximate-but-equal length' approach, the ultimate lengths of the individual composites within a domain are permitted to self-adjust to produce the smallest number of equal-length composites possible that approximately conform to the user-defined composite length (maximum is 1.5 x the user-defined interval length) and incorporate the entire length of a like-domain segment down the drillhole. This approach can result in more of the original samples being retained and incorporated into a composite, but some or even most of the individual composites in the dataset may only approximate the user-defined interval length, rather than exactly conform to that length, depending on whether each downhole segment in a domain is exactly divisible by the user-defined composite length or not.

In the case of the work done for the PFS19, the approximate-but-equal length strategy was adopted. Hence many of the composite lengths fluctuate around the user-defined 1 m or 2 m interval length (depending on domain). These will be referred to collectively as notional 1 m or 2 m composites from herein.

In the low-grade mineralisation shells, the sample length distributions show the samples are mostly clustered around 1 m and 1.5 m, while the background material is dominated by 2 m sample lengths.

Statistical analyses on mineralised zones were conducted using 1 m composites, while the low-grade mineralisation shell and background samples were analysed using 2 m composites.

14.6.2 Composite Grade Distribution Statistics

Table 14.9 shows the summary of the composites grade univariate statistics and population characteristics for each of the major grade fields, subset by mineralisation and weathering zones. The corresponding sample distributions were also plotted graphically as histograms and log probability charts.

The composite statistics in Table 14.9 are generally consistent with expectations for the different zones and sub-zones. Copper and zinc are strongly elevated in the massive pyrite and enriched zones but have reduced values in the gossan. A similar, but less marked change is evident in the grades in the low-grade mineralisation shell across the oxide–sulphide boundary. Gold and silver are elevated in both the oxide and sulphide zones but are slightly higher in the former.

Coefficients of variation (CoV) of precious and base metals in the massive pyrite and enriched zones are low-to-moderate, while visual observations of high-grade variabilities for gold and silver in the low-grade mineralisation shell below the base-of-oxidation boundary are confirmed by the high CoVs for these elements.

Statistics were also computed for the minor grade fields.

Table 14.9 Summary of Composites Grade Statistics by Domain

MINZONE	WEAZONE	Sub-Zone	Unit	Component	Composites		Minimum	Maximum	Mean	Std. Dev.	Coefficient of Variation	Variance
					Length*	Count						
All	All	All	g/t	Au	1	12,795	0.01	150	0.72	2.62	3.65	6.8
			g/t	Ag	1	12,795	0.00	2,203	25	70	2.82	4,858
			%	Cu	1	12,795	0.00	15.8	0.58	0.95	1.62	0.90
			%	Zn	1	12,795	0.00	30.9	1.00	1.86	1.86	3.46
GOSS	All	n/a	g/t	Au	1	1,350	0.01	43	2.04	3.80	1.87	14.5
			g/t	Ag	1	1,350	0.20	2,203	59	153	2.58	23,412
			%	Cu	1	1,350	0.00	1.0	0.10	0.10	1.03	0.01
			%	Zn	1	1,350	0.00	0.9	0.08	0.08	1.03	0.01
GSCL	All	n/a	g/t	Au	1	346	0.01	62	2.36	5.31	2.25	28.2
			g/t	Ag	1	346	0.50	948	70	126	1.79	15,791
			%	Cu	1	346	0.00	1.4	0.06	0.11	1.68	0.01
			%	Zn	1	346	0.00	0.6	0.06	0.07	1.17	0.01
MSPY	All	n/a	g/t	Au	1	4,858	0.01	10	0.77	0.91	1.19	0.8
			g/t	Ag	1	4,858	0.50	1,500	30	49	1.67	2,432
	All	All	%	Cu	1	4,858	0.00	15.8	0.91	0.90	0.99	0.81
	All	1	%	Cu	1	998	0.00	15.8	1.50	1.68	1.12	2.82
	All	2	%	Cu	1	3,748	0.00	3.0	0.76	0.40	0.53	0.16
	All	n/a	%	Zn	1	4,858	0.00	30.9	1.89	2.43	1.28	5.91

MINZONE	WEAZONE	Sub-Zone	Unit	Component	Composites		Minimum	Maximum	Mean	Std. Dev.	Coefficient of Variation	Variance
					Length*	Count						
MSEN	All	n/a	g/t	Au	1	454	0.01	45	1.22	2.25	1.85	5.1
			g/t	Ag	1	454	0.50	244	45	35	0.78	1,227
			%	Cu	1	454	0.00	13.4	3.26	2.17	0.67	4.72
			%	Zn	1	454	0.01	14.7	2.57	2.33	0.90	5.41
MISZ	WEAT	n/a	g/t	Au	2	252	0.01	9	0.28	0.86	3.13	0.7
			g/t	Ag	2	252	0.50	652	16	51	3.22	2,560
			%	Cu	2	252	0.00	1.7	0.13	0.19	1.47	0.04
			%	Zn	2	252	0.00	1.3	0.12	0.17	1.44	0.03
	FRSH	n/a	g/t	Au	2	2,733	0.01	120	0.24	2.66	11.23	7.1
			g/t	Ag	2	2,733	0.00	1,080	8	34	4.28	1,155
			%	Cu	2	2,733	0.00	3.4	0.25	0.29	1.17	0.09
			%	Zn	2	2,733	0.00	11.3	0.42	0.83	1.95	0.68

* Notional length

14.6.3 Variography

Variographic analysis was focussed on the major grade fields and only in those mineralised zones that demonstrate suitable continuity. The selected zones were (1) combined gossan and clay-like gossan, and (2) massive pyrite. The enriched zones were considered to be too discrete and discontinuous, and the low-grade mineralisation shell grades are not considered to represent sufficiently-defined populations to be meaningful for variography.

Experimental variograms were generated on un-transformed 1 m composites.

Directions of preferred continuity were tested within the primary planes of orientation for each zone, and structures were obtained for each of the strike (045°/00°), down-dip (315°/20°), and across-plane orientations (using downhole variograms as a proxy).

During variogram modelling, the position of the nugget variance was fixed using the downhole variogram, and anisotropic variogram parameters were derived using two or three-structure spherical models. Table 14.10 summarises the modelled variogram parameters for the major elements in the gossan and massive pyrite zones.

The downhole variograms typically displayed low nugget variances, around 10% to 20% of the total, particularly for base metals in the massive pyrite. This observation is consistent with the generally low variability of copper and zinc observed visually in profiles down mineralised intersections. Similarly, the downhole grade trends noted in some of the thicker massive pyrite intersections are reflected in the observation that some downhole variograms do not settle on to a horizontal sill.

A further feature is that many variograms in the plane of the materialisation (along-strike and down-dip) are not well formed, suggesting that the drill spacings is at or near the ranges in these directions.

The modelled sills for the three directions are commonly quite different. This zonal anisotropy is to be expected from observations of internal grade zonation within the plane of the mineralisation, particularly in wider portions of massive pyrite.

In some cases, very long ranges were invoked for the final structures to ensure that, where zonal anisotropy is evident, variogram models for all directions reach a common sill. These ranges are well beyond the search neighbourhood during estimation and therefore have no influence on the interpolation.

Table 14.10 Variogram Parameters

MINZONE	Grade Field	Sub-Zone	Dip Dir.	Dip	Nugget	Structure 1				Structure 2				Structure 3			
						Var.	Strike	Dip	Cross-Strike	Var.	Strike	Dip	Cross-Strike	Var.	Strike	Dip	Cross-Strike
GOSS + GSCL	Au		315	20	0.15	0.38	25	63	3	0.42	70	87	50	0.31	90	5,000	1,000
	Ag		315	20	0.15	0.23	20	36	5	0.45	35	79	7	0.22	60	89	200
	Cu		315	20	0.1	0.50	50	60	5	0.37	95	80	1,000	–	–	–	–
	Zn		315	20	0.1	0.65	25	37	7	0.33	75	1,000	100	–	–	–	–
	S		315	20	0.3	0.10	10	10	6	0.50	40	20	8	–	–	–	–
	As		315	20	0.1	0.45	10	10	4	0.30	25	25	8	0.38	500	40	16
	Hg		315	20	0.35	0.20	10	10	4	0.38	40	25	7	0.57	65	60	500
	Pb		315	20	0.05	0.50	10	10	3	0.30	25	25	5	0.35	40	40	50
	Fe		315	20	0.08	0.17	10	10	4	0.50	45	25	11	0.28	90	250	25
MSPY	Au		315	20	0.18	0.26	76	10	5	0.48	170	200	150	0.15	180	2,000	1,000
	Ag		315	20	0.04	0.22	11	60	5	0.50	70	105	45	0.09	200	140	60
	Cu	1	315	10	0.02	0.18	15	6	8	0.45	50	25	15	1.00	120	350	60
	Cu	2	315	20	0.1	0.36	25	22	3	0.33	120	125	13	0.41	230	1,000	1,000
	Zn		315	20	0.15	0.25	15	20	5	0.40	40	60	20	0.27	300	500	1,000
	S		315	20	0.1	0.20	8	8	3	0.14	25	25	9	0.60	100	250	250
	As		315	20	0.2	0.20	20	20	4	0.44	70	100	13	0.55	1,000	1,000	25
	Hg		315	20	0.1	0.18	10	45	3	0.20	30	120	15	0.48	160	500	60
	Pb		315	20	0.05	0.35	8	8	4	0.08	20	25	5	0.45	45	50	27
	Fe		315	20	0.1	0.20	10	20	4	0.14	20	95	8	0.40	30	250	150

14.6.4 Grade Capping

A detailed review of grade characteristics for the major grade fields was undertaken for each of the mineralisation and weathering zones as a basis for determining whether grade capping (high or low) was necessary and, if so, determine suitable values to use.

Several steps were followed to assess whether there was a requirement for capping of high grades to reduce any undue influence these grades might impose during grade estimation.

The grade capping reviews were conducted using 1 m or 2 m composites depending on the mineralised zone.

Initially, the composites statistics (Table 14.9) were referenced to understand the relationship between population mean grades and variances, and the magnitudes of the CoV. Histograms of grade distributions and log probability charts were also reviewed, paying particular attention to the relative frequency of higher grades (e.g. upper 5% of the population).

The major grade components on the composites were then visually analysed within each mineralised zone, highlighting composites with moderately high to anomalously high grades. The spatial locations of these high-grade samples were assessed relative to their surroundings and careful consideration was given to their possible impact during grade estimation. A list of high-grade caps was developed (Table 14.11).

In the case of the minor grade components, time did not permit the visualisation steps, and the values shown in Table 14.12 were determined from statistical tables and charts.

Samples that exceeded the high-grade cap were reset to equal the relevant cap value.

Table 14.11 High-Grade Caps by Domain: Major Elements

Component Grade	Comp. Length (m)	Domain			Upper Cap
		MINZONE	WEAZONE	Sub-Zone	
Au (g/t)	1	GOSS	All	–	25.0
	1	MSPY	All	All	6.0
	1	MSEN	All	–	6.0
	2	MISZ	WEATH	–	2.5
	2	MISZ	FRSH	–	5.0
Ag (g/t)	1	GOSS	All	–	350.0
	1	MSPY	All	All	150.0
	1	MSEN	All	–	150.0
	2	MISZ	WEATH	–	100.0
	2	MISZ	FRSH	–	100.0
Cu (%)	1	GOSS	All	–	0.7
	1	MSPY	All	1	12.0
	1	MSPY	All	2	2.0
	1	MSEN	All	–	10.0
	2	MISZ	WEATH	–	0.6
	2	MISZ	FRSH	–	2.0
Zn (%)	1	GOSS	All	–	0.5
	1	MSPY	All	All	12.0
	1	MSEN	All	–	10.0
	2	MISZ	WEATH	–	0.8
	2	MISZ	FRSH	–	5.0

Table 14.12 High-Grade Caps by Domain: Minor Elements

Component Grade	Comp. Length (m)	Domain		Upper Cap
		MINZONE	WEAZONE	
Pb (%)	1	GOSS	All	6.0
	1	MSPY	All	4.0
	1	MSEN	All	2.5
	2	MISZ	WEATH	–
	2	MISZ	FRSH	3.0
As (ppm)	1	GOSS	All	8,000
	1	MSPY	All	6,000
	1	MSEN	All	2,200
	2	MISZ	WEATH	2,500
	2	MISZ	FRSH	5,000
S (%)	1	GOSS	All	10.0
	1	MSPY	All	5.0
	1	MSEN	All	25.0
	2	MISZ	WEATH	20.0
	2	MISZ	FRSH	–
Fe (%)	1	GOSS	All	–
	1	MSPY	All	15.0
	1	MSEN	All	15.0
	2	MISZ	WEATH	25.0
	2	MISZ	FRSH	–
Hg (ppm)	1	GOSS	All	30.0
	1	MSPY	All	12.0
	1	MSEN	All	7.0
	2	MISZ	WEATH	10.0
	2	MISZ	FRSH	9.0

14.6.5 Bulk Density Capping

Evaluation of bulk density data was undertaken on the sample data points after coding with the LTHZONE, MINZONE, and WEAZONE domain codes. A total of 6,202 coded density samples were available for this assessment.

As per the grade capping strategy described in Section 14.6.4, bulk density samples were assessed to determine whether any outliers exist in the raw dataset and, if capping (high or low) was necessary, to determine suitable values to use.

Statistics were computed for densities in each of the mineralisation zones (and weathered and fresh for low-grade mineralisation shell), and for background material in each of the lithology domains (Table 14.13).

Significantly different statistics are observed for the different domains, consistent with expectations for variably-mineralised and variably-weathered material types.

Table 14.13 shows the low and high-density caps.

Table 14.13 Bulk Density Statistics by Domain

Domain	Samples	Minimum	Maximum	Mean	Cap	
					Low	High
GOSS	463	1.62	3.84	2.57	2.00	–
GSCL	37	1.85	3.54	2.45	2.00	3.00
MSEN	127	2.30	4.92	4.16	3.40	–
MSPY	1,414	2.21	5.91	4.34	3.20	5.00
MISZ – WEAT	49	1.84	4.07	2.58	2.10	3.30
MISZ – FRSH	724	2.00	4.84	3.39	2.50	–
BKGR – OVBN	33	2.16	2.69	2.56	2.30	–
BKGR – SHQF	759	1.49	4.29	2.68	2.40	2.80
BKGR – SHCS	1,974	1.83	5.25	2.81	2.30	3.80
BKGR – SHQZ	606	2.37	4.29	2.68	2.40	2.80
BKGR	16	2.49	3.29	2.67	–	–

Table 14.14 shows the revised statistics following the truncations.

Samples that exceeded the high-density cap or did not reach the low-density cap were removed from the dataset.

Table 14.14 Bulk Density Statistics after Removal of Outliers

Domain	Samples	Minimum	Maximum	Mean
GOSS	439	2.00	3.84	2.60
GSCL	34	2.00	2.88	2.45
MSEN	121	3.44	4.92	4.23
MSPY	1,389	3.25	4.92	4.37
MISZ – WEAT	45	2.17	3.23	2.54
MISZ – FRSH	709	2.52	4.84	3.41
BKGR – OVBN	32	2.41	2.69	2.57
BKGR – SHQF	741	2.43	2.79	2.67
BKGR – SHCS	1,905	2.30	3.80	2.78
BKGR – SHQZ	570	2.45	2.78	2.66
BKGR	13	2.49	3.29	2.67

14.7 Volume Model

The volume model was constructed using a base configuration of 20 m (easting) x 20 m (northing) x 10 m (RL) parent cells, as shown in Table 14.15.

The initial model geometry was selected on the basis of the overall dimensions of the geology, and also with a view to subsequent refinement for compatibility with estimation objectives (refer to Section 14.8).

Table 14.15 Volume Model Prototype

Coordinate	Origin	Parent Cell	
		Dimension	No. of Cells
Easting	636,000	20	100
Northing	4,357,000	20	120
RL	1,000	10	55

Domain coding in the model cells followed a similar logic and sequence of steps to the coding of drillhole samples (Section 14.4).

The lithology, mineralisation, and weathering domain coding was achieved by filling above, below, or within the respective wireframes, and then assigning the corresponding LTHZONE, MINZONE, and WEAZONE codes. An 'air' code was created in cells that were located above the topographic surface.

Splitting of parent cells at domain boundaries was permitted to better honour the interpretations. The smallest sub-cell size permitted was 5 m (E) x 5 m (N) x 2 m (RL).

Cells that were located above the topographic surface were eliminated from the volume model.

Figure 14.11 shows a stylised oblique view of the coded volume model.

Figure 14.11 Stylised Oblique Sectional through the Volume Model

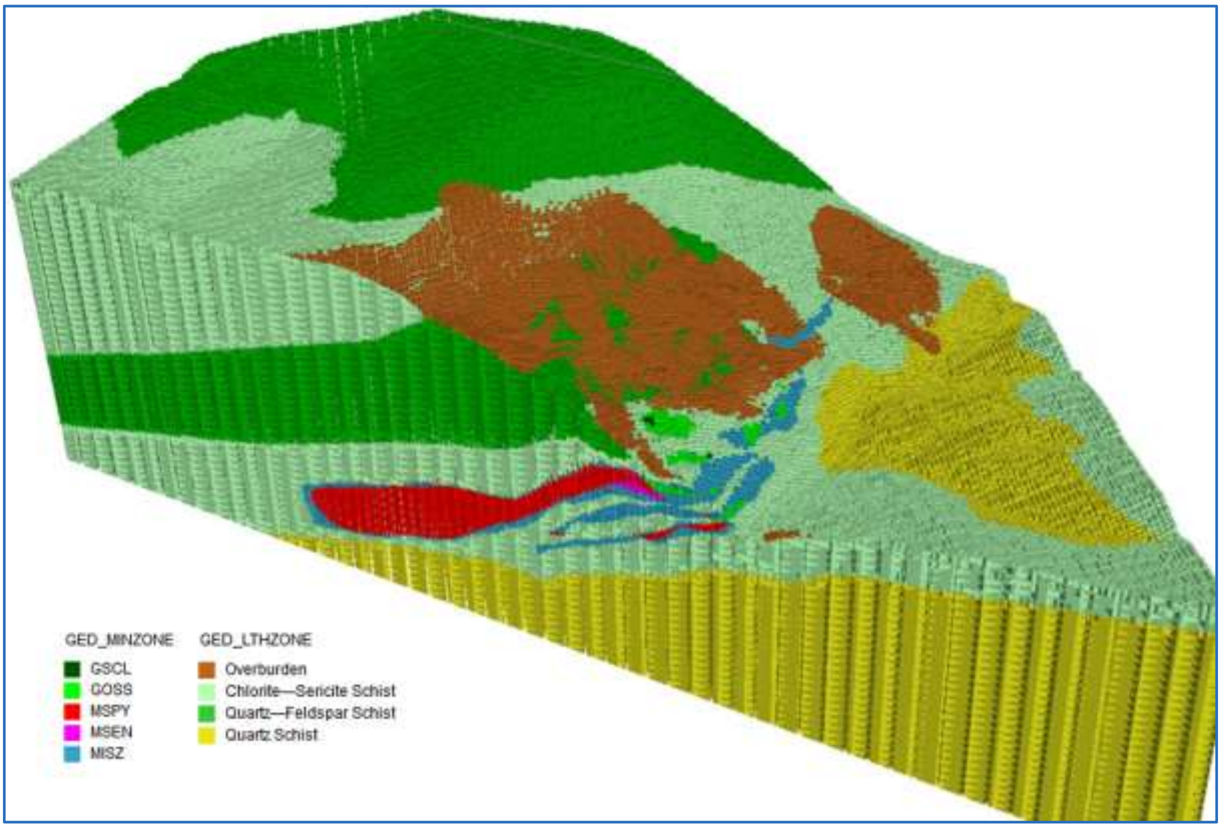


Figure from Polimetal, 2018.
View from above looking approximately north.

14.8 Grade and Density Estimation

14.8.1 Grade Estimates

The scope for the Gediktepe resource estimate update specified the following grade fields for estimation: Au, Ag, Zn, Cu, As, Hg, Pb, Fe, and S. This was later extended to include carbon (C), in particular for estimates in the background domain.

Grades were estimated using either ordinary kriging (OK) or inverse distance weighting to the power of two (ID2). Depending on the domain being estimated, composites of either 1 m or 2 m (notional) length were used (composites lengths discussed in Section 14.6.1).

Grade capping was applied after completion of compositing (grade capping discussed in Section 14.6.4).

The estimation of grades, by domain, using either OK or ID2 interpolation methods are summarised in Table 14.16.

Table 14.16 Estimation Methods

MINZONE	Sub-Zone	Component	Estimation Method
GOSS + GSCL	n/a	All – except C	OK
	n/a	C	ID2
MSPY	Zn Sub-Zone = 1	Au, Ag, Zn, & Hg	ID2
	Zn Sub-Zone = 2		OK
	All Cu Sub-Zones	Cu	OK
	n/a	As, Fe, Pb, & S	OK
	n/a	C	ID2
MSEN	n/a	All	ID2
MISW	n/a	All	ID2
MISF	n/a	All	ID2

For optimal processing, the domained volume model was constructed on the basis of a parent cell dimension of 20 m (E) x 20 m (N) x 10 m (RL) (see Section 14.7). However, this cell size was considered too coarse to be suitable for grade estimation. Therefore, in advance of grade estimation, the parent cell size of the volume model was temporarily reduced to 10 m (E) x 10 m (N) x 2.5 m (RL), using the model prototype shown in Table 14.17.

The parent cell estimates were mapped to individual like-domained sub-cells within the parent cell.

Table 14.17 Estimation Model Prototype

Coordinate	Origin	Parent Cell	
		Dimension	No. of Cells
Easting	636,000	10	200
Northing	4,357,000	10	240
RL	1,000	2.5	220

Grade estimation was conducted into parent cells under hard-bounded domain control, referencing the 'ESTDOM' field. The ESTDOM field was derived using MINZONE and WEAZONE field codes, as shown in Table 14.18. The massive pyrite, enriched, and gossan domains had already been constrained during interpretation to their relevant weathering zones, and therefore required no further subdivision. Table 14.19 shows the additional partitioning applied with respect to the identified copper and zinc grade distribution sub-zones.

Table 14.18 Estimation Domains

ESTDOM	Equivalent to:	
	MINZONE	WEAZONE
GOSS	GOSS	n/a
	GSCL	
MSPY	MSPY	
MSEN	MSEN	
MISW	MISZ	WEAT
MISF	MISZ	FRSH

Table 14.19 Estimation Sub-Zones

Zone Field	Code	Description
SUBZONCU	1	Higher grade Cu massive pyrite (around enriched)
	2	Other massive pyrite
SUBZONZN	1	Zinc-depleted massive pyrite
	2	Other massive pyrite

In view of the relatively regular distribution of drilled intersections across the deposit, and the similarity of the geometries for each of the interpreted mineralised zones, a limited set of search ellipsoid configurations was applied. These search dimensions were chosen with consideration of (a) capturing sufficient samples for estimation within the search neighbourhood, (b) the observed continuities of grades, and (c) evidence of zonal anisotropies in variograms.

Constraints including minimum and maximum numbers of like-domained composites, number of composites from a single drillhole, and octant search criteria were required to be met before a cell estimate was accepted. If these criteria could not be met in the first search pass, the search dimensions were expanded for a second and, if required, third search pass, each with new criteria applied (refer to Table 14.20).

At a whole-of-deposit scale, the Gediktepe mineralisation shows a relatively consistent strike, dip direction, and dip. The overall 045° strike (and corresponding 315° dip direction) and 15° to 20° dip, evident in the southern and central areas, is observed to swing west to strike approximately 025° (dip direction 285°) with a slightly steeper dip. Locally the dip orientations can be considerably more varied, particularly in long-section, often as a consequence of faulting. A default search orientation of 315°/20° for dip direction and dip was applied. To account for a limited number of variations from this default, the model cells within designated volumes were coded with local dip directions and dips, including a 285°/23° orientation for most of the northern area. This embedding of dip directions and dips into the model allowed the Datamine Studio Dynamic Anisotropy function to be applied to exploit these local orientations.

Cell discretisation during grade estimation was applied using a 4 x 4 x 2 (XYZ) matrix.

Following grade estimation, any cells coded as being within a mineralised domain that failed to receive an estimate were assigned default values using the same values as those used for unsampled sample intervals (see Table 14.7).

Figure 14.12 and Figure 14.13 are example cross-sections of model grade estimates, showing copper in the northern area and gold in the southern area, respectively.

Figure 14.12 Example Cell Model Estimates: Cross-Section: Copper: Northern Area

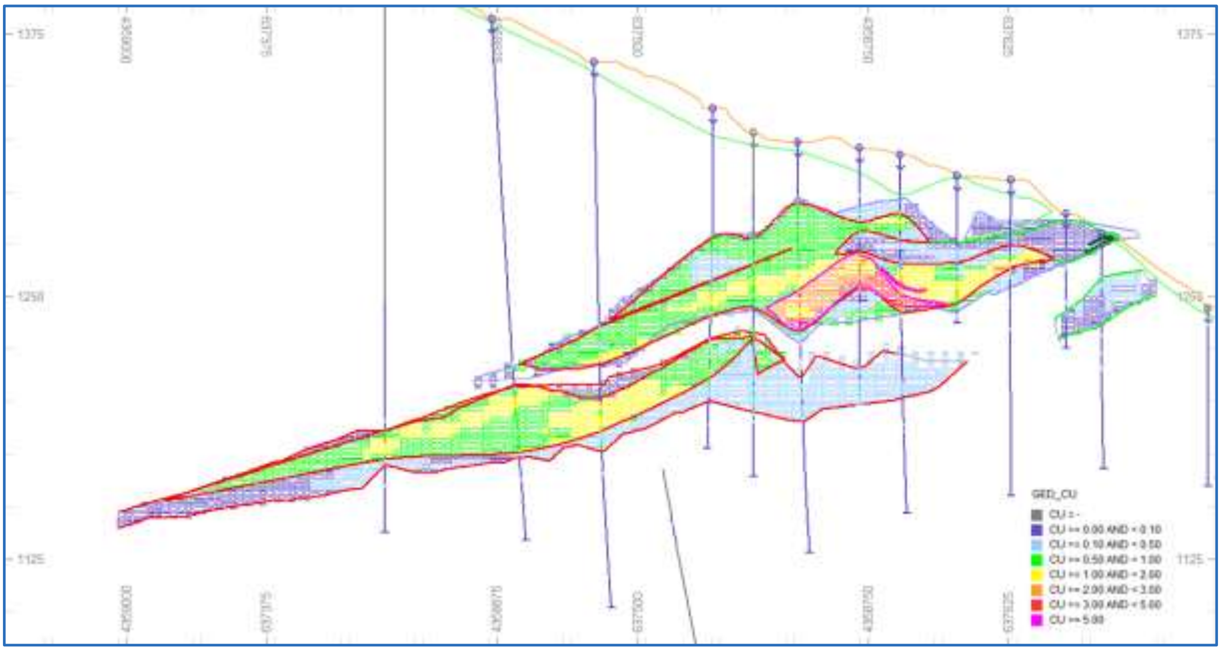


Figure from Polimetal, 2018.

Figure 14.13 Example Cell Model Estimates: Cross-Section: Gold: Southern Area

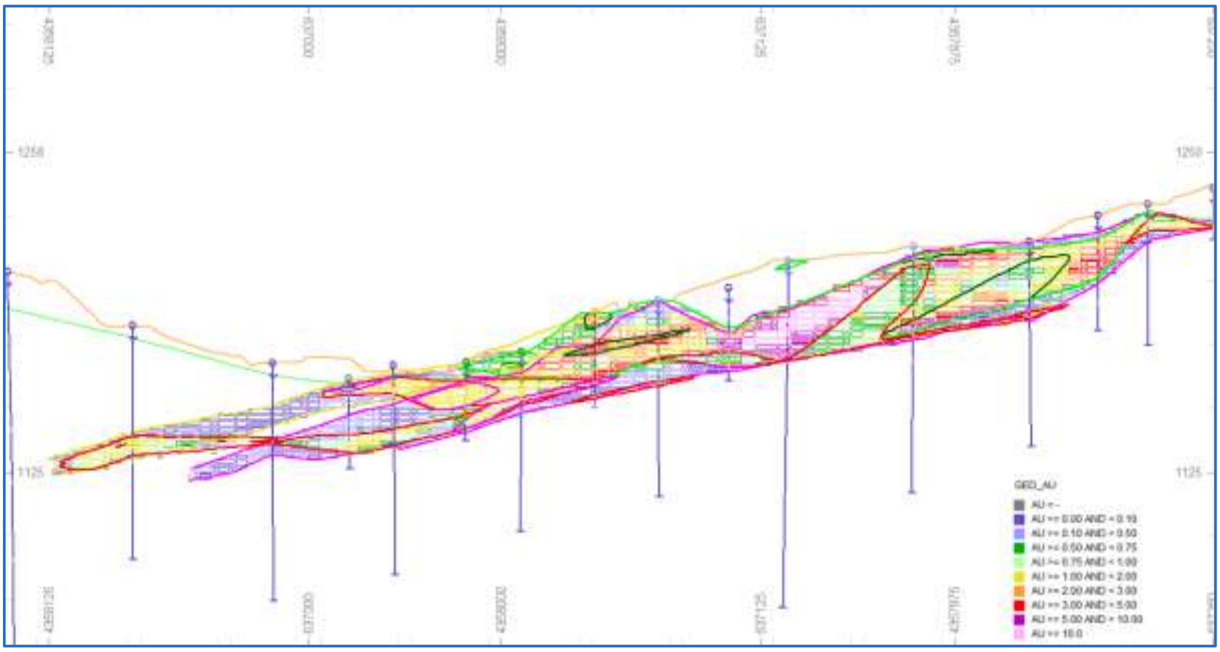


Figure from Polimetal, 2018.

Table 14.20 Search Parameters

ESTDOM	First Search Pass											Subsequent Search Passes					
	Search Distance (m)			Dip Dir.	Dip	Octant Search			Search 1		Maximum Comps. from any Drillhole	Search 2			Search 3		
	Strike	Dip	Across-Strike			Minimum No. of Octants	Minimum Comps. per Octant	Maximum Comps. per Octant	Minimum No. of Comps.	Maximum No. of Comps.		Expans^{n} Factor	Minimum No. of Comps.	Maximum No. of Comps.	Expans^{n} Factor	Minimum No. of Comps.	Maximum No. of Comps.
GOSS	40	50	5	315	20	2	2	4	5	24	5	1.5	3	24	3	2	20
MSPY	40	50	5	315	20	2	2	4	5	24	5	1.5	3	24	3	2	20
MSEN	40	50	5	315	20	–	–	–	4	24	5	1.5	3	24	3	2	20
MISW	20	25	5	315	20	–	–	–	2	15	5	1.5	2	24	–	–	–
MISF	20	25	5	315	20	–	–	–	2	15	5	1.5	2	24	–	–	–
BKGR	50	50	10	315	20	–	–	–	2	15	–	1.5	2	24	3	1	20

14.8.1.1 Density Estimates

The number, frequency, and broad spatial distribution of density values were considered to be a sufficient basis for estimating density values into model cells. Statistical analysis had demonstrated that the values within the different mineralisation and weathering populations show distinct density characteristics. Consequently, in preparation for estimation, density values were coded with the same ESTDOM field applied to control the estimation of grades. Outlier density values were eliminated from the dataset (see Section 14.6.5).

Densities were estimated using ID2 methods under zonal control of the ESTDOM field codes. Search orientations were aligned with the same orientations used for grade estimations, but with larger search ellipse dimensions to account for the lower quantum of density data. Any cells that did not receive a density estimate, which typically occurred as a result of insufficient data in the search neighbourhood, were assigned default values, derived from statistical analysis according to mineralisation and weathering domain.

14.8.2 Background Model Estimates

Background material, outside of the defined mineralisation domains, is not considered for inclusion in the Mineral Resource estimates. However, Polimetal requested that available data be used to generate grade and density estimates in background model cells.

As background material is located outside of the mineralisation domains, grades and densities were partitioned for analysis according to lithologies (LTHZONE) and weathering (WEAZONE). Grades and densities were estimated using ID2 methods.

14.9 Validation

The resulting estimated geological model incorporates the fields listed in Table 14.21.

Table 14.21 Post-Estimation Model Fields

Field	Description
ESTDOM	Domains used to constrain estimation
SUBZONZN	Zonation to partition small volume of depleted Zn, (plus depleted Au, Ag, and Hg)
SUBZONCU	Zonation of Cu to partition high-grade area from the remainder
DENSITY	Bulk density estimates
DADIPDIR	Estimation search dip direction to control 'Dynamic Anisotropy' (cell specific)
DADIP	Estimation search dip to control 'Dynamic Anisotropy' (cell specific)
DAPLNG	Estimation search plunge direction – not used
AU	Estimated grade: Gold
AG	Estimated grade: Silver
CU	Estimated grade: Copper
ZN	Estimated grade: Zinc
S	Estimated grade: Sulphur
AS	Estimated grade: Arsenic
HG	Estimated grade: Mercury
PB	Estimated grade: Lead
FE	Estimated grade: Iron
NUMSAM	Number of samples used to estimate into each parent cell (captured for gold estimates in GOSS and MISW, and for copper estimates in MSPY, MSEN, and MISF)
PASS	Search ellipse pass for grade estimation into each parent cell (captured for gold estimates in GOSS and MISW, and for copper estimates in MSPY, MSEN, and MISF)
DENDEF	Cells that received a default density (domain-specific average) as a result of failing to receive an estimate

Global and zonal statistics were generated to confirm that estimated model grades values fall within acceptable limits.

The grade and density estimates in the cell model were thoroughly scrutinised using graphical visualisation utilities. Model and drillhole data were overlain and viewed in various sectional and plan views, and in three-dimensions, with colour legends highlighting grade or zonal attributes.

These processes were undertaken repeatedly and continuously throughout the study, during which adjustments and refinements to the model were tested against the predicted consequences of any changes.

Validation processes revealed that the proportion of material estimated in the first search pass varied according to material type; specifically, with massive pyrite and gossan volumes more-commonly receiving first pass estimates than the material within the low-grade mineralisation shells.

The model also progressed through the various iterations from an exclusively ID2 estimate to one where OK was used for suitable domains. This progression of estimation methods provided insight into the effects on local estimates of different techniques.

The model development and grade estimation procedures were subject to a Peer Review process. Similarly, and prior to acceptance by Polimetal, a draft model was made available for review to Polimetal and Alacer.

Alacer generated and reviewed sectional plots and composite trend (swath) plots.

14.10 Resource Classification

14.10.1 Classification Method

Procedures for classifying the reported resources were undertaken within the context of the Canadian Securities Administrators National Instrument 43-101 (NI 43-101).

Gediktepe estimated resources have been classified with consideration of the following general criteria:

- Confidence in the geological interpretation.
- Knowledge of grade continuities gained from observations and geostatistical analyses.
- Number, spacing, and orientation of drillhole intercepts through mineralised domains.
- Quality and reliability of the raw drillhole data (sampling, assaying, surveying).
- The likelihood of material meeting economic mining constraints over a range of reasonable future scenarios, and expectations of relatively high selectivity of mining.

During the interpretation and evaluation phases of the resource modelling, a considerable body of knowledge was established in relation to the characteristics of the mineralisation and the quantum and configuration of sampling data from drilling. The geometric and grade continuities were observed to vary considerably, both in and across the general planes of the mineralised units, and the various grade attributes exhibit different variabilities and spatial trends. The notional 25 m x 25 m drill spacing is also seen to vary, with locally increased concentrations of drilling in some central areas, while reduced towards the margins.

Knowledge of better continuities and drilling intensity was used to identify the most-likely areas for higher resource classification potential.

Geological considerations affecting confidence:

- Mineralisation boundaries.
  - Sharpness within individual drill intersections.
  - Lateral continuities between adjacent intersections (are boundaries easily correlated?).

- Continuities (or variabilities) of grades.
  - Within individual intersection profiles.
  - Lateral continuities (or variabilities) between adjacent intersections (are intersection profiles consistent?).
- Structural effects – faulting, folding.

Other indicators of confidence:

- Observations from statistical and variographic work – low / high CoV, quality of variogram structures, ranges, nuggets, etc.
- Data quality and how it varies across the deposit.
- Output from the estimation process – e.g. number of samples, search ellipse pass.

All the above needed to be considered with respect to the individual characteristics of each estimated grade and each domain.

Classification of the Gediktepe PFS19 model was undertaken as follows:

1. Identify areas of different drilling intensity, and consider, as a default, what level of classification these might represent.
2. Digitise Inferred/Indicated boundaries (plan view) around default identifiable areas of higher drilling intensity. Separate boundaries were generated for each of the gossan, massive pyrite and low-grade mineralisation shell sets of intersections.
3. Apply these boundaries to code the model, using a vertical projection cookie-cutter method.
4. View the coded model in plan view, and cross-section and long-section views, and adjust boundaries, with consideration of the geological and other criteria described above.
5. Re-run model coding (Item 3), and review and adjust as necessary (Item 4).
6. Identify potential areas for Measured material (gossan and massive pyrite only), using identified zones of good continuity and suitable drillhole spacing.
7. Digitise initial Indicated/Measured boundaries.
8. Cycle through the same refinement process as for Inferred / Indicated (Items 3 through 5).

Because of the three-dimensional nature of the mineralised domains, the two-dimensional cookie-cutter method inevitably resulted in some localised volumes being inappropriately coded (as Measured). Several small solid wireframes were created to recode these volumes appropriately to Indicated.

The model cells were, by default, assigned an Inferred classification, and those cells falling within the digitised strings were re-coded as either Indicated or Measured. Table 14.22 shows the 'RESCAT' classification model field codes.

Table 14.22 Resource Classification Model Codes

Category	RESCAT
Measured	1
Indicated	2
Inferred	3
unclassified	*4*

A view of the distributions of the different resource categories is shown in Figure 14.14.

Figure 14.14 Oblique View of the Classified Cell Model

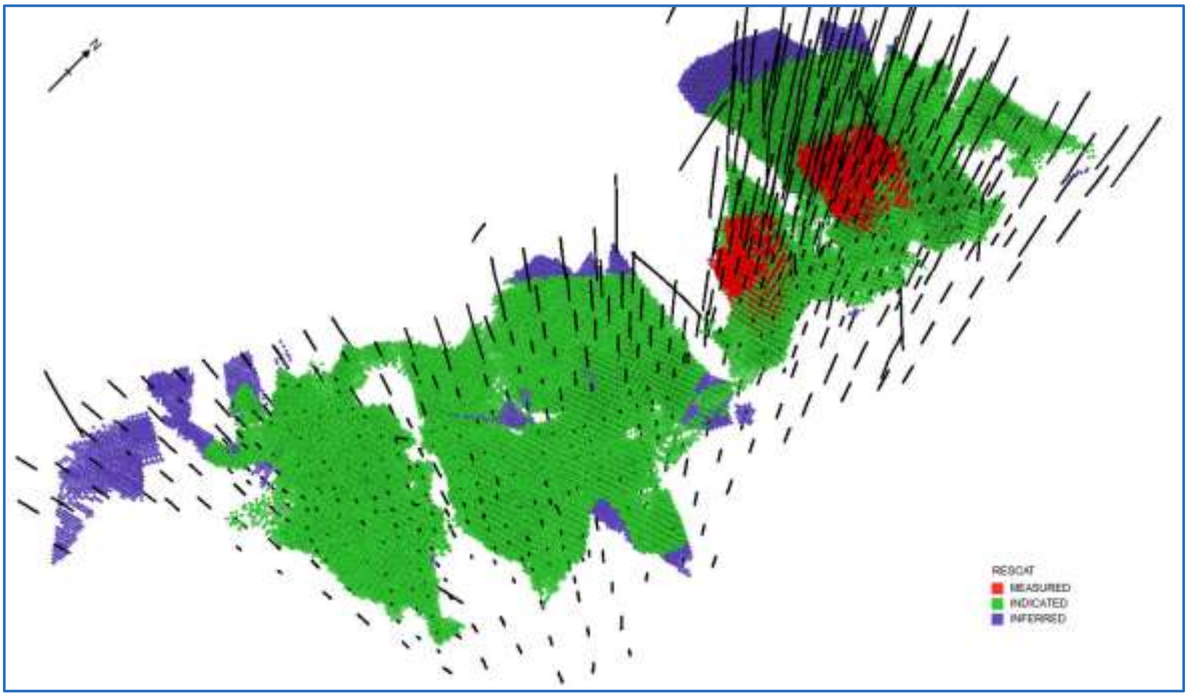


Figure by OreWin, 2019.
View from above looking approximately north.
Low-grade mineralisation shell excluded.
Some cells obscured by overlying cells.

14.10.2 Comparison to Previous Classification Method

A comparison was made to the PFS16 Mineral Resource.

The PFS16 classification was based on the number of composites used to estimate into a cell and the average distance between the cell centre and all of the composites used to estimate the gold grade into that cell.

The following steps were used to apply the classification coding:

1. All cells with a gold estimate were initially coded as Inferred ('conf' = 3).
2. Cells coded as Inferred were upgraded to Indicated ('conf' = 2) via one of the following two paths:
    a. If the gold grade estimate was based on four or more composites ('au_num' >= 4) and if the average distance to the closest composite was 75 m or less ('avedist' <= 75), (note: a maximum of three composites from each drillhole was permitted, therefore 'au_num' >= 4 equates to data from at least two different drillholes used to inform the estimate), or
    b. If the model cell was coded as one of the sulphide mineralised units (MPY, MPM, ERH and TRS), and if three composites were used to inform the gold estimate ('au_num' = 3), and if the average distance to the closest composite was 75 m or less ('avedist' <= 75). This step (2b) was established so that contiguous mineralisation in the narrow, high-grade sulphide zones could be considered as Indicated.
3. Cells were coded as Measured ('conf' = 1) if they had a gold grade estimated using the maximum permitted number of composites ('au_num' = 10), and the average distance to the closest composite was 35 m or less ('avedist' <= 35).

PFS16 resources were reported at NSR ($/t) cut-offs that were specific to oxide or sulphide material types, within a floating cone open pit based on the following metal prices: $1,200/oz gold, $18.00/oz silver, $3.00/lb copper, and $1.20/lb zinc.

14.10.3 Classification Method Comparison

The PFS16 and PFS19 methods of resource classification are different, with the former effectively based solely on thresholds applied to numeric information generated during the estimation process, and the latter being driven by geological characteristics and continuities, evaluated against sampling intensity, and supplemented by data quality information and estimation output.

The two methods have produced very different outcomes in relation to the assignment of Measured material, which are discussed in more detail below.

14.11 Mineral Resource Report

14.11.1 PFS19 Classified Tonnes and Grade Estimates

For consistency, the PFS19 resource is reported using cut-offs based on calculations of Net Smelter Return (NSR). This method is considered to be appropriate for polymetallic deposits such as Gediktepe. Separate NSR cut-offs are applied to each of the oxide and sulphide zones.

CIM guidelines required that a Mineral Resource must have: "reasonable prospects of economic extraction". To meet this requirement, the classified resource has been constrained to those model cells falling within an optimised pit shell that was developed using the metal price parameters used for the determination of Ore Reserves but inflated by 14%, and where all categories of material (including Inferred) have been considered in the pit optimisation.

The PFS19 Measured, Indicated, and Inferred Mineral Resources are shown in Table 14.23. Measured plus Indicated Mineral Resources are combined in Table 14.24.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Some of the terminology used in these tables has been chosen specifically to maintain consistency with PFS16 terminology. The relationship between the terms in the table and descriptions in foregoing sections in this report are as follows:

Table	Report text
Low Oxide	Low-grade Mineralisation Shell: Weathered (MISW)
Diss. Sulphide	Low-grade Mineralisation Shell: Fresh (MISF)

Table 14.23 Gediktepe PFS19 Mineral Resources – All Classifications

MEASURED		Tonnes (kt)	Grade					Metal			
			Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Oxide	Gossan	–	–	–	–	–	–	–	–	–	–
	Low Oxide	–	–	–	–	–	–	–	–	–	–
	Total Oxide	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**
Sulphide	Massive Pyrite	3,999	0.67	25.1	1.01	1.83	0.34	86	3,221	40	73
	Diss. Sulphide	–	–	–	–	–	–	–	–	–	–
	Enriched	–	–	–	–	–	–	–	–	–	–
	Total Sulphide	**3,999**	**0.67**	**25.1**	**1.01**	**1.83**	**0.34**	**86**	**3,221**	**40**	**73**
Total Measured		**3,999**	**0.67**	**25.1**	**1.01**	**1.83**	**0.34**	**86**	**3,221**	**40**	**73**

INDICATED		Tonnes (kt)	Grade					Metal			
			Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Oxide	Gossan	2,562	2.79	67.6	0.10	0.10	0.48	230	5,571	3	2
	Low Oxide	112	0.83	36.9	0.22	0.18	0.21	3	132	0	0
	Total Oxide	**2,674**	**2.71**	**66.3**	**0.10**	**0.10**	**0.47**	**233**	**5,703**	**3**	**3**
Sulphide	Massive Pyrite	17,049	0.83	30.4	0.87	1.92	0.38	454	16,681	148	327
	Diss. Sulphide	5,588	0.40	15.9	0.43	0.87	0.18	71	2,853	24	48
	Enriched	907	1.19	45.7	3.14	2.61	0.22	35	1,331	28	24
	Total Sulphide	**23,544**	**0.74**	**27.6**	**0.85**	**1.69**	**0.33**	**560**	**20,865**	**200**	**399**
Total Indicated		**26,217**	**0.94**	**31.5**	**0.78**	**1.53**	**0.34**	**792**	**26,568**	**203**	**402**

INFERRED		Tonnes (kt)	Grade					Metal			
			Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Oxide	Gossan	12	1.09	22.4	0.08	0.08	0.15	0	9	0	0
	Low Oxide	11	0.78	21.0	0.40	0.21	0.09	0	7	0	0
	Total Oxide	**23**	**0.95**	**21.8**	**0.23**	**0.14**	**0.12**	**1**	**16**	**0**	**0**
Sulphide	Massive Pyrite	2,847	0.52	20.0	0.77	1.15	0.27	47	1,832	22	33
	Diss. Sulphide	111	1.01	26.3	0.43	1.39	0.26	4	94	0	2
	Enriched	–	–	–	–	–	–	–	–	–	–
	Total Sulphide	**2,958**	**0.53**	**20.2**	**0.76**	**1.16**	**0.27**	**51**	**1,926**	**22**	**34**
Total Inferred		**2,981**	**0.54**	**20.3**	**0.76**	**1.16**	**0.27**	**51**	**1,941**	**23**	**34**

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Effective Date of Mineral Resource is 5 March 2019.
3. Mineral Resources are estimated at NSR cut-offs of $20.72/t for oxide and $17.79/t for sulphide.
4. Mineral Resources have been constrained using an optimised pit shell, to reflect reasonable prospects of economic extraction.
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6. Mineral Resources are inclusive of Mineral Reserves, except for mining losses and grade dilution, which are determined through re-blocking of the resource model after declaration of the Mineral Resource.
7. Mineral Resources are quoted on a 100% project basis.
8. Totals may not match due to rounding.

Table 14.24 Gediktepe PFS19 Mineral Resources – Measured plus Indicated Only

MEASURED + INDICATED		Tonnes (kt)	Grade					Metal			
			Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Oxide	Gossan	2,562	2.79	67.6	0.10	0.10	0.48	230	5,571	3	2
	Low Oxide	112	0.83	36.9	0.22	0.18	0.21	3	132	0	0
	Total Oxide	**2,674**	**2.71**	**66.3**	**0.10**	**0.10**	**0.47**	**233**	**5,703**	**3**	**3**
Sulphide	Massive Pyrite	21,047	0.80	29.4	0.89	1.90	0.37	539	19,903	188	400
	Diss. Sulphide	5,588	0.40	15.9	0.43	0.87	0.18	71	2,853	24	48
	Enriched	907	1.19	45.7	3.14	2.61	0.22	35	1,331	28	24
	Total Sulphide	**27,542**	**0.73**	**27.2**	**0.87**	**1.71**	**0.33**	**645**	**24,086**	**241**	**472**
Total Measured + Indicated		**30,216**	**0.90**	**30.7**	**0.81**	**1.57**	**0.34**	**878**	**29,790**	**243**	**475**

Notes:
1 CIM definitions were followed for Mineral Resources.
2 Effective Date of Mineral Resource is 5 March 2019.
3 Mineral Resources are estimated at NSR cut-offs of $20.72/t for oxide and $17.79/t for sulphide.
4 Mineral Resources have been constrained using an optimised pit shell, to reflect reasonable prospects of economic extraction.
5 Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6 Mineral Resources are inclusive of Mineral Reserves, except for mining losses and grade dilution, which are determined through re-blocking of the resource model after declaration of the Mineral Resource.
7 Mineral Resources are quoted on a 100% project basis.
8 Totals may not match due to rounding.

14.11.2 Comparison of PFS16 and PFS19 Resource Estimates

The summary Mineral Resource tonnes and grades estimates for both PFS16 and PFS19 are presented in Table 14.25.

Comparisons of PFS16 versus PFS19 resource estimates for individual material types and classification categories show some marked differences.

The overall direction of change in the as-reported Mineral Resource tonnes and grades estimates from PFS16 to PFS19 is downwards. For combined Measured and Indicated resources, the magnitude of the changes in the grades is from –2% to –10% (relative) and there is an overall drop in tonnage of –16% in PFS19. This equates to an across-the-board reduction in contained metal in the PFS19 Measured plus Indicated Mineral Resource of between –18% to –25%.

In assessing the causes of these differences between the two study results, the following factors should be considered:

- Additional drilling has been undertaken since PFS16 – specifically Phases 4 and 5 of the drilling were completed.
- A different independent consultancy was commissioned for the PFS19 resource modelling study.
- Refinements to the geological and mineralogical interpretations were completed for use in PFS19.

- While PFS19 applied broadly similar estimation techniques, using similar geological interpretations, the manner in which the low-grade gossan and disseminated / transition sulphide material were estimated was markedly different.
- The methods used to assign resource classification categories to the estimates are different.
- Both sets of reported tonnes and grades used NSR cut-offs, however the intervening changes in metal prices, recoveries, and costs applied meant that the cut-off values for the oxide and sulphide materials are significantly different.
- For the same reasons responsible for the NSR differences, the optimised pits used to constrain the reported estimates are different.

Table 14.25 Comparison between PFS16 and PFS19 Mineral Resource Estimates Reported at Respective 2016 and 2019 NSR Cut-offs

Weathering Zone	Resource Version	Cut-off (NSR $/t)	Classification	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Oxide	PFS16 ϕ	11.70	Measured	1,722	2.65	67	0.12	0.16
			Indicated	2,110	2.56	71	0.18	0.35
			Inferred	*213*	*1.57*	*63*	*0.13*	*0.17*
	PFS19	20.72	Measured	–	–	–	–	–
			Indicated	2,674	2.71	66	0.10	0.10
			Inferred	*23*	*0.95*	*22*	*0.23*	*0.14*
Sulphide	PFS16 ϕ	15.67	Measured	12,027	0.78	29	1.00	1.89
			Indicated	20,180	0.77	30	0.85	1.95
			Inferred	*1,685*	*0.81*	*32*	*0.98*	*1.80*
	PFS19	17.79	Measured	3,999	0.67	25	1.01	1.83
			Indicated	23,544	0.74	28	0.85	1.69
			Inferred	*2,958*	*0.53*	*20*	*0.76*	*1.16*
Total PFS16		**M+I ***		**36,039**	**0.97**	**34**	**0.83**	**1.75**
Total PFS19		**M+I ***		**30,217**	**0.91**	**31**	**0.80**	**1.57**

* Totals do not include Inferred
ϕ PFS16 Mineral Resource estimates taken from IMC, 2016

To enable a comparison of the two generations of model on a more-constant and current basis, the two models were reported within the 2018 resources pit shell using 2018 NSR cut-off parameters and based on NSR calculated using 2019 metal prices. The percentage difference of Measured plus Indicated is shown in Table 14.26.

Table 14.26 Percentage Difference between 2016 vs. 2018 Model Inventories when Reported on a Like-For-Like Basis

Both reported within the 2018 Resource Pit Shell at 2019 NSR Cut-offs, with NSR calculated on Updated Metal Prices

Weathering Zone	Cut-off (NSR $/t)	Classification	Percentage Difference				
			TONNAGE	GRADE			
				Au	Ag	Cu	Zn
Oxide	20.72	M+I	14%	–5%	–10%	6%	6%
Sulphide	17.79	M+I	9%	11%	8%	12%	8%
Overall Differences		**M+I**	**10%**	**7%**	**5%**	**11%**	**8%**

Weathering Zone	Cut-off (NSR $/t)	Classification	Percentage Difference				
			TONNAGE	CONTAINED METAL			
				Au	Ag	Cu	Zn
Oxide	20.72	M+I	14%	8%	2%	21%	20%
Sulphide	17.79	M+I	9%	21%	18%	22%	18%
Overall Differences		**M+I**	**10%**	**17%**	**15%**	**22%**	**18%**

Notes:
Calculations do not include Inferred
Difference calculated as (2018–2016)/2016
NSR calculated using $1,315/oz Au, $18.00/oz Ag, $3.20/lb Cu, and $1.10/lb Zn

While at face value the PFS16 and PFS19 Mineral Resource estimate reports appear to indicate that the quantum and/or tenor of mineralisation has diminished in PFS19, when reported on a like-for-like present-day basis, the 2018 Measured plus Indicated inventory is in fact larger than its 2016-equivalent.

As Table 14.26 shows, when Measured plus Indicated is reported using the same NSR cut-offs, with NSR calculated on the same basis, and within the same 2018 resource pit shell, the 2018 model reports overall higher tonnes and grades relative to the 2016 model, with the magnitude of changes for grades ranging between 5% and 11% (relative) and an overall increase in tonnage of 10%. This equates to an across-the-board increase in contained metal of between 15% and 22% in the 2018 model.

Changes in the NSR cut-offs used in the two generations of model are a significant contributor to the differential between the two reports, as are changes in metal prices, which have in-turn changed the resource pit. Underpinning these after-modelling factors, and further intensifying the differences in the reports, are significant changes to the modelling of the mineralisation and the methods for estimating the resources.

In terms of the confidence in the estimates, the substantial reduction in the proportion of Measured material in 2018, relative to 2016 is directly related to the very different methods used to assign classification categories. While the 2016 approach, which applies thresholds to outputs from the estimation process, is not unprecedented for resource classification, the exclusion of any explicit consideration of variations in geological continuity across the Gediktepe deposit presents the risk of failing to acknowledge and keep track of potentially material uncertainties in the resource estimates.

14.12 Conclusions and Recommendations

The Gediktepe project is a polymetallic deposit that exhibits significant primary variability of mineralisation styles, as illustrated within the sulphide zones where massive pyrite, enriched, and disseminated (transitional) mineralisation are marked by their individual characteristics. This variability is further complicated by the tendency for different metals of interest (Au, Ag, Cu, Zn, and Pb) to show individual and characteristic distributions, both spatially and in grade tenor.

These fundamental variations have been further augmented by, firstly, the considerable effects of weathering, which has leached and redistributed the metals differentially, and, secondly, by post-mineralisation faulting.

The combination of these factors manifests in a complex mineral deposit that presents significant challenges for both geological definition and sampling activities, as well as for the evaluation of Mineral Resources.

Under these circumstances, the diligent work undertaken by Polimetal in documenting, interrogating, interpreting, and modelling the Gediktepe deposit has assisted greatly in developing a dataset and conceptual model that are detailed and robust.

The suitability of the fundamental geological work is a consequence of, amongst other considerations:

- A systematic programme of drilling and sampling that has resulted in a relatively consistent spatial distribution of drillholes, despite some topographical and other practical challenges.
- The dominant use of diamond core drilling, utilising relatively high-volume PQ diameters to depths as far as practical.
- Rigorous logging, sampling, and assaying procedures, using certified laboratories and incorporating routine QA/QC practices and analyses.
- A recognition of key relationships between mineralisation types and grade characteristics.
- Diligent interpretations of lithologies, mineralisation, and weathering boundaries.

The output of this work, which has progressed through several phases (exploration, PEA, PFS) is therefore considered suitable as input into the resource modelling and grade estimation that forms the basis for PFS19.

Notwithstanding these efforts, it is inevitable that a deposit of this nature will retain variable, and at times material levels of uncertainty resulting in lower confidence. Some of these uncertainties are inherent to the geological characteristics of the mineralisation, others to limitations of the methods of resource definition, and yet others are related to the limitations of methods of resource evaluation.

These uncertainties are not evenly distributed throughout the deposit, and any associated project risks need to be assessed, not only with regard to their likely (financial) magnitude, but also with respect to their locations in space (across the deposit) and time (within the mining schedule).

The resource classification categories assigned to the Gediktepe estimates (Measured, Indicated, Inferred) have, at a global scale, identified different levels of confidence (uncertainty) across the deposit, and this is sufficient for feasibility assessment. However, these categories do not necessarily reflect variations in confidence at a more local resolution, which may impact on the shorter term effectiveness, and hence profitability, of eventual mining.

It is recommended that additional targeted actions be taken to identify particular areas of significance, but lower confidence. The "targeted" approach is to ensure that the refinement actions are effective, without undue costs in time and expenditure.

Three key factors that can be applied in assessing relevant variations in economic significance and confidence across the Gediktepe deposit have been identified. These are the mapping of:

- The intrinsic value of metals within local volumes.
- The local levels of geological confidence.
- The timing of individual volumes within the mining schedule.

Spatial consideration of these factors will allow the identification of lower confidence but relatively high value areas that may be scheduled for mining early in the mine plan, thus enabling these areas to be targeted and prioritised for further assessment.

Through a process that progresses from higher priority to lower, geological confidence in targeted areas may be raised through one or a combination of the following actions.

- Additional, focussed drilling.
- Selected resampling and assaying.
- Review of local geological interpretations.
- Refinement of resource modelling and grade estimation procedures.

If these activities prove to be successful in raising confidence in the estimates, then revisions to resource classifications may be justified in some areas.

15 MINERAL RESERVE ESTIMATES

15.1 Mineral Reserve Statement

The Gediktepe Mineral Reserve reported according to the CIM guidelines is summarised in Table 15.1.

Table 15.1 Gediktepe PFS19 Mineral Reserves

Classification	Tonnage (kt)	Grade				Contained Metal			
		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (koz)	Ag (koz)	Cu (kt)	Zn (kt)
Oxide									
Proven	–	–	–	–	–	–	–	–	–
Probable	2,755	2.34	56.7	–	–	207	5,020	–	–
Proven & Probable	2,755	2.34	56.7	–	–	207	5,020	–	–
Sulphide									
Proven	3,620	0.68	26.7	1.03	1.93	79	3,105	37	70
Probable	14,960	0.89	33.1	0.89	1.99	429	15,903	133	298
Proven & Probable	18,580	0.85	31.8	0.92	1.98	509	19,008	170	368

Notes:
1 CIM definitions were followed for Mineral Reserves.
2 Effective Date of Mineral Reserve is 5 March 2019.
3 Mineral Reserves were reported using a Net Smelter Return (NSR) based on metal prices of $1,300/oz Au, $18.5/oz Ag, $3.30/lb Cu, and $1.28/lb Zn, smelter terms for treatment and refining charges and transport including ocean freight for sulphide ore concentrates.
4 Cut-offs applied were: oxide ore $20.67/t and sulphide ore $17.74/t. Additionally, enriched mineralisation with a Cu/Zn grade ratio < 0.75 is considered to be waste.
5 Metal prices used for economic analysis to demonstrate the Mineral Reserve are Au $1,315/oz, Ag $18.0/oz, Cu $3.20/lb and Zn $1.10/lb.
6 Reported Mineral Reserves incorporate and include mining losses and grade dilution that are not reported in the Mineral Resource.
7 Only Measured Mineral Resources (and dilution) were used to report Proven Mineral Reserves and only Indicated Mineral Resources (and dilution) were used to report Probable Mineral Reserves.
8 Mineral Reserves are a subset of, not additive to, the Mineral Resources and are quoted on a 100% project basis.
9 Totals may not match due to rounding.

15.2 NSR Reporting Cut-off

Due to its polymetallic nature, the oxide and sulphide portions of the Mineral Reserve are quoted at an NSR cut-off based on metal prices, metal recoveries, plus on and off-site processing costs. This parallels the pit optimisation approach for the project, which is discussed in Section 16.2. For the pit optimisation, Polimetal selected metal prices of $1,300/oz Au, $18.5/oz Ag, $3.30/lb Cu, and $1.28/lb Zn. The pit shells produced from this optimisation were used for pit design work.

At the time of creating the mine schedules and the economic analysis to support PFS19, the various parameters used to define NSR and the associated ore cut-offs were updated based on revised metallurgical parameters, cost estimates, and long-term metal price forecasts. The metal prices used in the economic analysis to demonstrate the Mineral Reserve are

$1,315/oz Au, $18.0/oz Ag, $3.20/lb Cu, and $1.10/lb Zn.

Cut-offs applied were: oxide ore $20.67/t and sulphide ore $17.74/t. Additionally, enriched mineralisation with a Cu/Zn grade ratio ≥ 0.75 is considered to be ore.

The following tables summarise the prices, costs and other estimation parameters adopted for the Mineral Reserve reporting and ore definition in the mining schedule used for financial modelling. Table 15.2 summarises metal prices.

Table 15.2 Mineral Reserve Metal Prices

Metal	Unit	Price (US$)
Copper	lb	3.20
Gold	troy oz	1,315
Silver	troy oz	18.0
Zinc	lb	1.10

Oxide ore parameters are summarised in Table 15.3.

Table 15.3 Oxide Ore Parameters

	Unit	Value
Recovery		
Gold	%	90.16
Silver	%	70.65
Treatment and Refining Charges		
Gold Payable	%	99
Gold Refining and Freight	$/troy oz	5.133
Silver Payable	%	98
Silver Refining and Freight	$/troy oz	1.602
Royalty		
Gold Royalty	%	3.9
Silver Royalty	%	2.6
Process and G&A Costs		
Total Process and G&A	$/t ore	25.64

For the financial model schedule and the reporting of the Mineral Reserve, more-detailed metallurgical analysis led to the assignment of updated recovery parameters to the main types of sulphide mineralisation; being massive pyrite, enriched, and disseminated. Metal recovery assumptions plus expected Cu and Zn concentrate grades for each ore type are detailed in Table 15.4.

Table 15.4 Ore Recovery Parameters

Parameter	Value/Formula
Oxide	
Gold Recovery	Fixed at 90.16%
Silver Recovery	Fixed at 70.65%
Massive Pyrite – Copper Concentrate	
Concentrate Grade	Fixed at 30% Cu
Copper Recovery	(10.342 x % Cu Feed Assay) + 57.492
Gold Assay in Concentrate	(4.7196 x g/t Au feed assay) + (7.3198 x (g/t Au feed assay)^2)
Silver Assay in Concentrate	(11.475 x g/t Ag feed assay) – (0.1127 x (g/t Ag feed assay)^2)
Zinc Recovery	% Cu feed assay x ((10.342x% Cu feed assay)+57.492) x ((0.9852 x % Zn feed assay) + 0.2705) / % Zn feed assay / % Cu concentrate assay
Lead Recovery	15.278 – (15.917 x % Pb feed assay)
Arsenic Recovery	% Cu feed assay x ((10.342x% Cu feed assay)+57.492) x ((0.8518 x % As feed assay) + 0.0266) / % As feed assay / % Cu concentrate assay
Massive Pyrite – Zinc Concentrate	
Concentrate Grade	Fixed at 58% Zn
Zinc Recovery	(0.5181 x % Zn feed assay) + 77.379
Gold Assay in Concentrate	(2.293 x g/t Au feed assay) – (0.6249 x (g/t Au feed assay)^2)
Silver Assay in Concentrate	(4.7899 x g/t Ag feed assay) – (0.0364 x (g/t Ag feed assay)^2)
Copper Recovery	(9.3369 x % Cu feed assay) + 1.0891
Lead Recovery	10.414 + (10.944 x % Pb feed assay)
Arsenic Recovery	% Zn feed assay x ((0.5181x% Zn feed assay)+77.379) x 0.05 / % As feed assay / % Zn concentrate assay
Enriched – Copper Concentrate	
Concentrate Grade	Fixed at 32.9% Cu
Copper Recovery	Fixed at 67.7%
Gold Recovery	Fixed at 10%
Silver Recovery	Fixed at 10%
Zinc Recovery	Fixed at 29.5%
Lead Recovery	Fixed at 45.5%
Arsenic Recovery	Fixed at 50%

Parameter	Value/Formula
Enriched – Zinc Concentrate	
Concentrate Grade	Fixed at 50% Zn
Zinc Recovery	Fixed at 56.4%
Gold Recovery	Fixed at 10%
Silver Recovery	Fixed at 10%
Copper Recovery	Fixed at 11.9%
Lead Recovery	Fixed at 13.8%
Arsenic Recovery	Fixed at 6%
Disseminated – Copper Concentrate	
Concentrate Grade	Fixed at 25.8% Cu
Copper Recovery	(14.576 x % Cu feed assay) + 60.396
Gold Assay in Concentrate	(33.038 x g/t Au feed assay) – (14.246 x (g/t Au feed assay)^2)
Silver Recovery	(0.0895 x (g/t Ag feed assay)^2) – (0.3866 x g/t Ag feed assay)
Zinc Recovery	% Cu feed assay x ((14.576x% Cu feed assay)+60.396) x 7.6 / % Zn feed assay / % Cu concentrate assay
Lead Recovery	Fixed at 40%
Arsenic Recovery	% Cu feed assay x ((14.576x% Cu feed assay)+60.396) x 0.47 / % As feed assay / % Cu concentrate assay
Disseminated – Zinc Concentrate	
Concentrate Grade	Fixed at 49.5% Zn
Zinc Recovery	(4.6259 x % Zn feed assay) + 67.751
Gold Recovery	Fixed at 10%
Silver Recovery	Fixed at 20%
Copper Recovery	% Zn feed assay x ((4.6259 x % Zn feed assay) + 67.751) x 3.9 / % Cu feed assay / % Zn concentrate assay
Lead Recovery	Fixed at 18.1%
Arsenic Recovery	% Zn feed assay x ((4.6259 x % Zn feed assay) + 67.751) x 0.68 / % As feed assay / % Zn concentrate assay

On-site processing costs associated with concentrator treatment of sulphides at a rate of 2.275 Mtpa are summarised in Table 15.5.

Table 15.5 Sulphide Ore Site Costs

	Unit	Value
Process and G&A Costs		
Total Sulphide Processing and G&A	$/t ore	22.19

Table 15.6 summarises parameters associated with the transport, treatment, and refining of the copper and zinc concentrates.

Table 15.6 Sulphide Ore Concentrate Parameters

	Copper Concentrate	Zinc Concentrate
Primary Metal Payable	Lesser of: 96.5%, or Cu content less 1%	Lesser of: 85%, or Zn content less 8%
Gold Payable	Lesser of: 90%, or Au content less 1 g/t	65% after 1 g/t deduction
Silver Payable	Lesser of: 90%, or Ag content less 30 g/t	65% after 93.3 g/t deduction
Treatment Charge	$90.00/dry tonne	$296.00/dry tonne *
Refining Charge – Cu	$0.09/lb	–
Refining Charge – Au	$5.00/oz	–
Refining Charge – Ag	$0.50/oz	–
Moisture Content	12%	12%
Ocean Freight	$30.00/wet tonne	$30.00/wet tonne
Port, Warehouse, and Handling	$18.75/wet tonne	$18.75/wet tonne
Inland Freight	$12.00/wet tonne	$12.00/wet tonne
Customs and Insurance	$1.06/wet tonne	$1.06/wet tonne

* Price participation at Zn > $1.00/lb applied

15.3 Comparison with 2016 PFS Mineral Reserve

The penultimate Mineral Reserve estimate was completed as part of the 2016 PFS. Table 15.7 compares the 2016 PFS estimate with the current version.

In all areas, the 2019 Mineral Reserve has lower tonnage and mineable metal grades than that estimated in 2016, coupled with a reduction in reporting confidence.

The main causes of the changes observed is a corresponding reduction in the Mineral Resource tonnage, grade, and classification confidence since that reported in 2016, change in cut-off grade from the change in processing method and operating costs and the mining dilution methodology.

In the 2016 Mineral Reserve estimate, mining dilution was treated simplistically and assumed to be already incorporated into the Mineral Resource model estimates (2.5 m vertical cells). The 2019 estimate has specifically allowed for ore loss and mining dilution using a resource re-blocking process to simulate expected mining selectivity as described in Section 16.2.1. This approach has reduced ore grade and contained metal relative to the in situ Mineral Resource estimate.

Table 15.7 Mineral Reserve Comparison

	Proven					Probable				
	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
2019 Oxide	–	–	–	–	–	2,797	2.35	56.7	–	–
2019 Sulphide	3,620	0.68	26.7	1.03	1.93	14,913	0.89	33.0	0.89	2.00
2016 Oxide	1,456	2.98	74.7	–	–	1,767	2.93	80.3	–	–
2016 Sulphide	10,425	0.84	31.0	1.04	2.05	11,267	1.00	39.3	0.93	2.63

	Total Proven + Probable				
	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
2019 Oxide	2,797	2.35	56.7	–	–
2019 Sulphide	18,533	0.85	31.8	0.92	1.99
2016 Oxide	3,223	2.95	77.7	–	–
2016 Sulphide	21,692	0.93	35.3	0.99	2.35

16 MINING METHODS

16.1 Introduction

Polimetal commissioned preliminary mine planning work for the open pit as part of the ongoing feasibility study work. OreWin reviewed this work and verified that it is reasonable and suitable for use in PFS19. The results are described in this Section.

Open pit mining is planned to be carried out on 2.5 m flitches using small excavators (3–4 m^3 capacity) and trucks. Drilling and blasting will be required. All mining services will be performed by a suitably qualified and experienced Turkish mining contractor. It is currently anticipated that the same mining contractor will provide initial construction services, particularly construction of the tailings storage facility (TSF).

Grade control to determine material types and ore boundaries will be performed based on blasthole sampling and assaying, and under the control of the mine geologists. Feed to the process plants is expected to be a combination of both direct tipping and reclaim from ROM stockpiles to ensure optimal feed to the process plant, particularly for sulphides.

16.2 Open Pit Mining

16.2.1 Diluted Mining Model

The Gediktepe resource model has parent cells for grade estimation of 10 m (E) x 10 m (N) x 2.5 m (RL). Where necessary, to honour geological boundaries, parent cells were permitted to split further; down to a minimum size of 5 m (E) x 5 m (N) x 0.5 m (RL) sub-cells. The orebody is moderately dipping and narrow in some areas.

The mining model used as the basis for mine planning needs to reflect the expected ore loss and dilution associated with the mining method. A re-blocking or regularisation approach was selected to simulate ore loss and dilution. Re-blocking is a simple method that is not software-specific.

Six alternative SMU's (selective mining unit) sizes were assessed. The 5 m x 5 m x 5 m SMU was selected as the basis for the mining model. Calculated ore loss and dilution associated with this SMU size is summarised in Table 16.1.

Table 16.1 Dilution and Ore Loss – 5 m x 5 m x 5 m SMU

Category	Cu (%)	Au (g/t)	Ag (g/t)	Zn (%)	Tonnage (kt)	Tonnage (%)
In Situ Resource	0.73	0.89	30.5	1.56	30,955	
Dilution	0.06	0.04	1.6	0.08	4,316	13.9
Mining Loss	0.72	0.58	21.3	0.96	2,212	7.1
Diluted Resource	0.65	0.80	27.3	1.41	33,059	106.6

16.2.2 Geotechnical Analysis

Several geotechnical studies are currently underway, and the results are pending, hence preliminary pit slope design recommendations for PFS19 are based on the data collected and analysed to date, and on the results of previous studies by external consultants. Polimetal has commissioned a pit slope design study, which is to include the logging of core from geotechnical drillholes and obtaining and analysing orientation measurements where possible. Laboratory testing on samples of core is currently being performed. Polimetal has commissioned work to develop a groundwater model to assess the level of water drawdown in pit slopes during mining. It is assumed that the data collected to date and the results from the previous studies are representative of conditions throughout the pit area.

16.2.2.1 Initial Recommendations

On the eastern side of the pit, if the bench faces are cleanly developed and scaled along the foliation such that the bench face slope is formed by the foliation at an average angle of 40°, rockfall hazards will be mostly removed and a minimum 6 m-wide catch bench would likely provide adequate rockfall protection in most cases. Leaving a 6 m-wide catch bench in the slope at 10 m vertical intervals would result in an inter-ramp slope angle of 29°. In the overburden, it is recommended that 5 m-high production benches with bench faces cut at 45° and a minimum 5.7 m-wide catch bench be developed at 5 m vertical intervals (single benches). This bench configuration results in an inter-ramp slope angle of approximately 25°.

On the west side of the pit, where the structural conditions are more favourable, bench face angles in phase slopes will mostly be limited by rock quality and the mining methods used to develop steep bench faces in highly fractured rock. It is recommended excavating bench faces at 63.5° in this sector. For phase slopes, where trim blasting to a free face is not used and bench faces are formed by cushion blasting in conjunction with standard production blasting, single benching (10 m-high benches) is recommended. Assuming 6.5 m-wide catch benches are left at 10 m vertical intervals results in a 41° inter-ramp slope. For final slopes, where cushion blasting is used in conjunction with trim blasting to a free face and scaling, double benching can be accomplished by stacking two 10 m-high production benches so that an 8.5 m catch bench is left in the slope at 20 m vertical intervals. This results in a 47° inter-ramp slope.

In the overburden, the geotechnical drillhole data indicates that the depth of highly weathered rock conditions on the west side of the pit is less than approximately 10 m. It is recommended to excavate the first bench at a bench face angle of 45° and leave a 6 m-wide catch bench on top of sound bedrock at the crest of the pit.

It was assumed that effective depressurisation of all pit slopes will be feasible, and that groundwater will not be a control on stability. Achieving this may require that drainage enhancements such as wells and horizontal drains be installed in less-permeable geotechnical units and where locally perched groundwater occurs in pit slopes.

16.2.2.2 Initial Pit Design Review

The configuration of the recommended 47° west wall inter-ramp slope (20 m bench stack height with an 8.5 m catch berm) was reviewed in the context of Turkish practices and regulations regarding maximum bench stack heights. After discussion with relevant parties, and technical assessment of the effectiveness of a narrower catch berm, an alternative west wall inter-ramp slope configuration of a 15 m-high stack height with a 6.5 m catch berm was adopted for both intermediate pit stages and for the ultimate pit. This revised configuration achieves the initial 47° inter-ramp slope target.

After review of the initial pit eastern (footwall) inter-ramp slope design, slight flattening (2° to 3°) was recommended to achieve acceptable factors of safety (FOS) for the rock types intercepted. Additionally, the southern portion of the eastern pit slope incorporates a permanent creek diversion. Review of the risks around this critical infrastructure recommended that:

- The berm the diversion is located on should be wider, and
- The overall local pit slope should be reduced in order to achieve a FOS of 1.5 (1.2 for standard slope design) to ensure longevity of this critical infrastructure.

These design changes to the east wall, which increase mine waste quantities, are incorporated into the PFS19 mine design.

16.2.3 Pit Optimisation Parameters

Initial pit optimisation was performed using simple parameters and elevated metal prices to constrain in situ Mineral Resources to report within a potentially economic open pit volume.

In May 2018, pit optimisation work commissioned by Polimetal used only Measured and Indicated mineralisation and treated enriched mineralisation (and associated dilution) as waste. The resulting shells were used as the basis for the ultimate pit and intermediate pit stage designs used at that time. Subsequently, process parameters were developed and enriched mineralisation with a Cu/Zn grade ratio ≥ 0.75 was considered for processing as ore.

The following tables summarise the parameter set adopted for the optimisation used for pit design.

The prices adopted for the pit optimisation are shown in Table 16.2

Table 16.2 Optimisation Metal Prices

Metal	Unit	Price (US$)
Copper	lb	3.30
Gold	troy oz	1,300
Silver	troy oz	18.5
Zinc	lb	1.28

Optimisation mining costs were based on averaged budget pricing from local contractors, including the current mining contractor for the Alacer Çöpler mine. Costs used are based on a reference cost at the 1220 RL of $1.48/t, increased by $0.01/t for each 5 m reduction in elevation.

Oxide ore parameters are summarised in Table 16.3.

For the optimisation, concentrator metal recoveries into the separate copper and zinc concentrates are assumed to be the same for all mineralisation types. This excludes enriched mineralisation (and associated dilution) that was treated as waste. Metal recovery of all ore types and expected Cu and Zn concentrate grades are detailed in Table 16.4.

On-site processing costs associated with concentrator treatment of sulphides at a rate of 2.275 Mtpa are summarised in Table 16.5

Table 16.3 Oxide Ore Parameters

	Parameter	Unit	Value
Recovery			
	Gold	%	88
	Silver	%	64.4
Treatment Charge / Refining Charge			
	Gold Payable	%	99
	Gold Refining and Freight	$/troy oz	5.133
	Silver Payable	%	98
	Silver Refining and Freight	$/troy oz	1.602
Royalty			
	Gold Royalty	%	3.9
	Silver Royalty	%	2.6
Process and G&A Costs			
	G&A Cost	$/t ore	5.50
	TSF Sustaining Cost	$/t ore	0.43
	Oxide Process Consumables	$/t ore	9.94
	Oxide Process Personnel	$/t ore	4.80
	Total Process and G&A	$/t ore	20.67

Table 16.4 Sulphide Ore Recovery Parameters

Metal Recovery to Concentrates *		
	Cu Concentrate	**Zn Concentrate**
Copper	60.0%	7.0%
Gold	17.2%	15.7%
Silver	12.3%	21.5%
Zinc	3.5%	81.0%
Lead	20.0%	11.5%
Cu and Zn Conc. Grades	Cu = 30.0%	Zn = 51.5%

* Excludes enriched treated as waste

Table 16.5 Sulphide Ore Site Costs

Process and G&A Costs	**Unit**	**Amount**
G&A Cost	$/t ore	5.50
TSF Sustaining Cost	$/t ore	0.43
Sulphide Process Consumables	$/t ore	8.90
Sulphide Process Personnel	$/t ore	2.91
Total Sulphide Process and G&A	$/t ore	17.74

Table 16.6 summarises parameters associated with the transport, treatment, and refining of the copper and zinc concentrates.

Pit slopes applied in the optimisation were based on geotechnical study recommendations.

Table 16.6 Sulphide Ore Concentrate Parameters

	Copper Concentrate	Zinc Concentrate
Primary Metal Payable	Lesser of: 96.5%, or Cu content less 1%	Lesser of: 85%, or Zn content less 8%
Gold Payable	Lesser of: 90%, or Au content less 1 g/t	65% after 1 g/t deduction
Silver Payable	Lesser of: 90%, or Ag content less 30 g/t	65% after 93.3 g/t deduction
Treatment Charge	$90.00/dry tonne	$296.00/dry tonne *
Refining Charge – Cu	$0.09/lb	–
Refining Charge – Au	$5.00/oz	–
Refining Charge – Ag	$0.50/oz	–
Moisture Content	12%	12%
Ocean Freight	$30.00/wet tonne	$30.00/wet tonne
Port, Warehouse, and Handling	$18.75/wet tonne	$18.75/wet tonne
Inland Freight	$12.00/wet tonne	$12.00/wet tonne
Customs and Insurance	$1.06/wet tonne	$1.06/wet tonne

* Price participation at Zn > $1.00/lb applied

16.2.4 Pit Optimisation Results

The pit optimisation was run using Whittle optimisation software. This produces a series of theoretical pit shells for a range of revenue factors that are effectively applied to the assumed commodity prices. Thus, the shell associated with the revenue factor of one equates to a break-even pit where the marginal cost of production (defining the shell limit) matches the revenue generated.

The breakeven shell does not equate to the shell that maximises project value after considering associated cash flows.

In the case of Gediktepe, there is a major step-out in the potential pit shell to the north–west at revenue factors approaching one. Due to the large waste stripping hurdle to develop this shell, mining costs are incurred several years earlier than revenue resulting in a reduction in NPV relative to the smaller shells. For this study, shell 23, which corresponds to a revenue factor of 0.84, was adopted as the basis for the ultimate pit design.

Figure 16.1 demonstrates the large increase in tonnage between shell 23 and 24, and the relatively small (un-discounted) incremental cash flow associated with this shell expansion.

Figure 16.1 Pit Optimisation Results

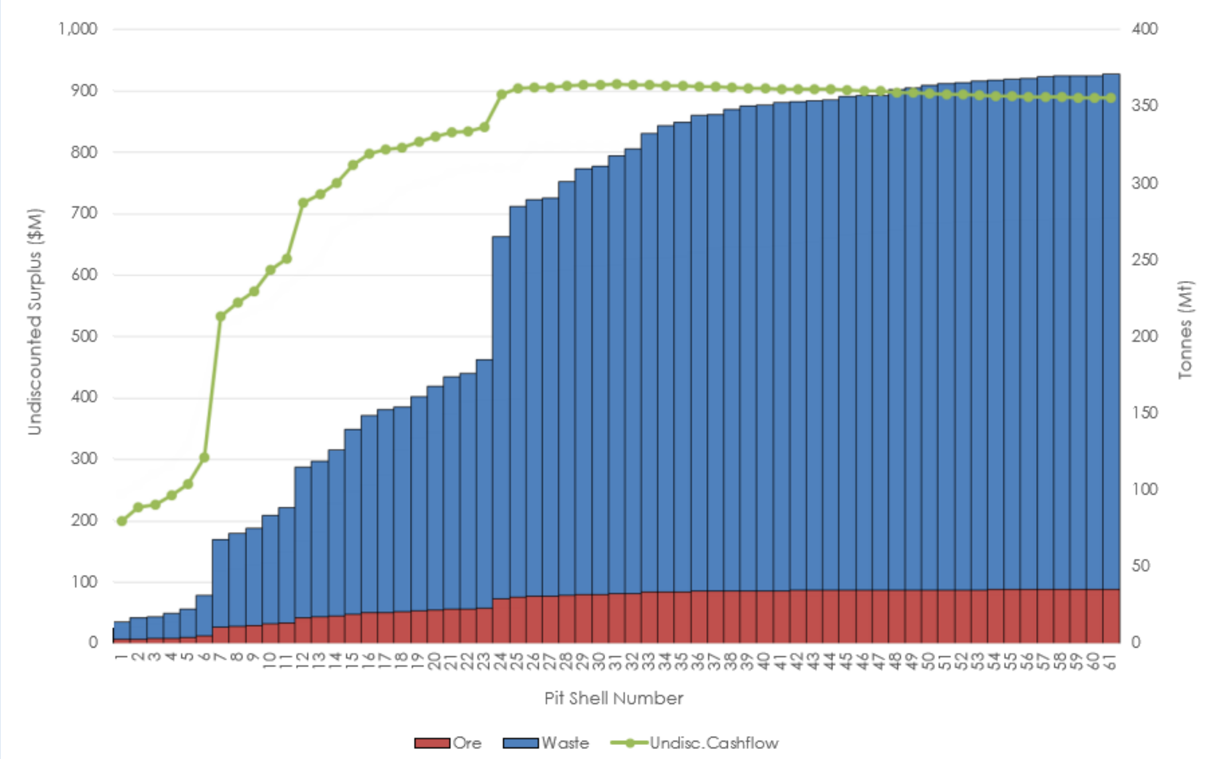


Figure from Polimetal, 2018.

As an aid to pit shell selection, the optimisation tool also provides estimates of "best case" and "worst case" discounted cash flows associated with successive shells. This notional cash flow excludes capital costs. The best case assumes that individual shells can be mined to completion before starting the next shell. The worst case assumes "top-down" mining to pit limits without any deferral of waste mining. The actual NPV is generally between these two theoretical cases. Figure 16.2 shows the undiscounted, best case discounted and worst case discounted cash flows for increasing pit tonnages. This shows that the optimum tonnage where the mid-way line flattens out is in the 150–170 Mt range.

Figure 16.2 Pit Optimisation Operating Surplus

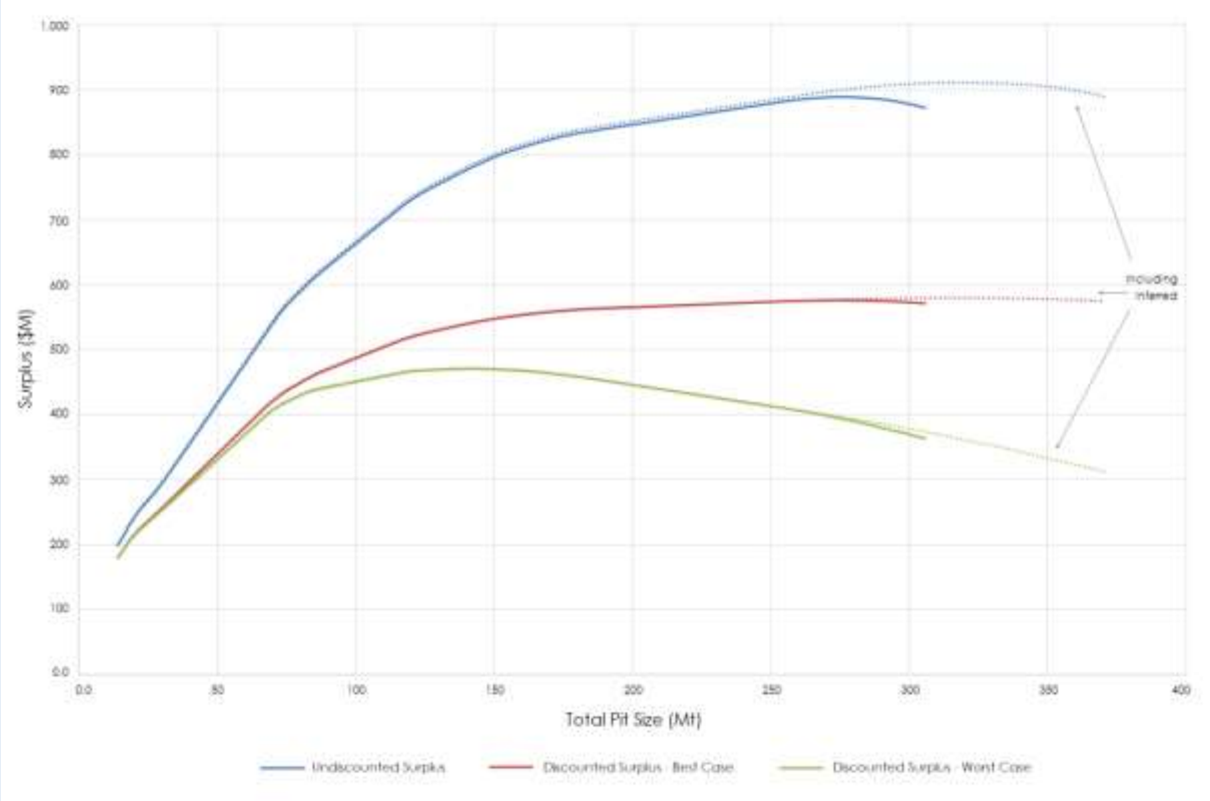


Figure from Polimetal, 2018.

The quantities associated with the shells of interest are summarised in Table 16.7. The selected pit shell, 23, has a total tonnage of 162.5 Mt.

Table 16.7 Pit Shell 23 Inventory

Ore (Mt) *	Cu (%)	Au (g/t)	Ag (g/t)	Zn (%)	Waste (Mt)	Total (Mt)
22.6	0.64	0.95	31.7	1.56	139.9	162.5

* Oxide and sulphide

In addition to the Measured and Indicated base case, other sensitivity optimisation runs were performed to test changes since PFS16 and to determine the potential impact if Inferred mineralisation can be converted to Measured or Indicated resource. A further run simulated the impact of lower copper recoveries into concentrate.

16.2.5 Pit Optimisation Verification

OreWin performed a pit optimisation to verify the ultimate pit design. It was found that the revenue factor 0.84 matched the pit design well, while the revenue factor 1 pit was larger than the design in the northern end. This leads to the conclusion that the pit design is reasonable.

The inputs used by OreWin for the optimisation are shown in Table 16.8. OreWin optimisations for revenue factor 0.84 and revenue factor 1 pits compared to the pit design are shown in Figure 16.3 through Figure 16.6.

Table 16.8 OreWin Optimisation Inputs

Description	Unit	Value
Mining Cost – Ore	$/t	1.25
Mining Cost – Waste	$/t	1.12
Incremental Mining Cost Increase Below 1,220 m	$/5 m bench	0.019
Metal Price		
Cu	$/lb	3.20
Zn	$/lb	1.10
Au	$/oz	1,315
Ag	$/oz	18.00
Processing and G&A Cost		
OXID	$/t	24.45
MSPY	$/t	21.81
DISS	$/t	21.81
MSEN	$/t	21.81

Figure 16.3 Pit Design with Optimisations – Plan View

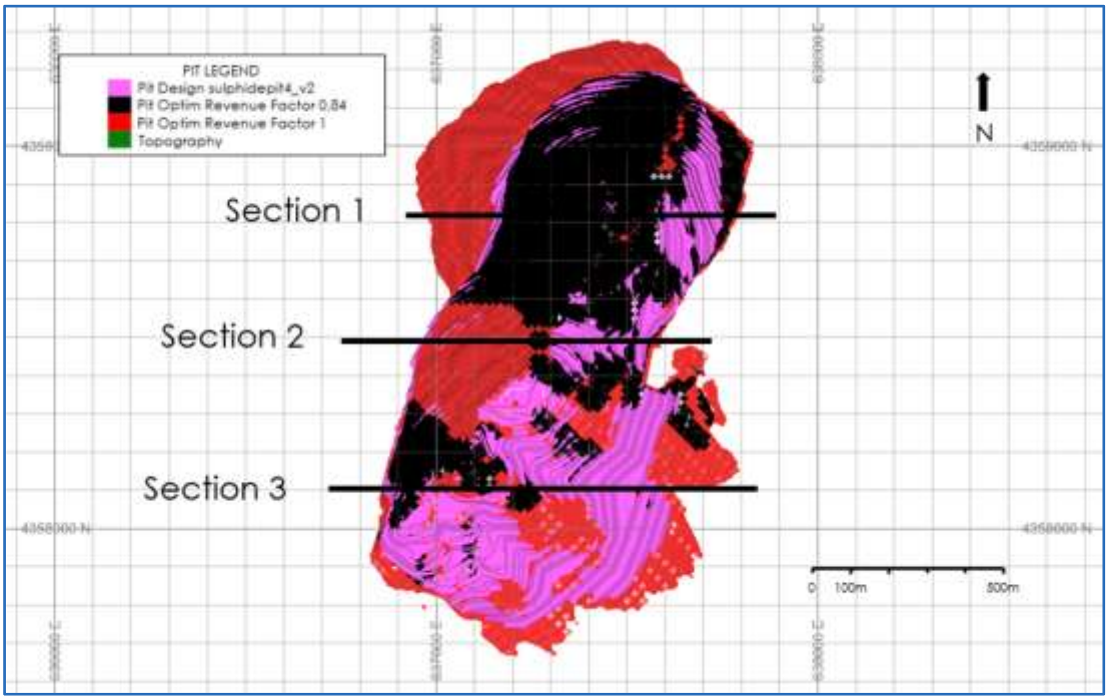


Figure by OreWin, 2019.

Figure 16.4 Pit Design with Optimisations – Section 1

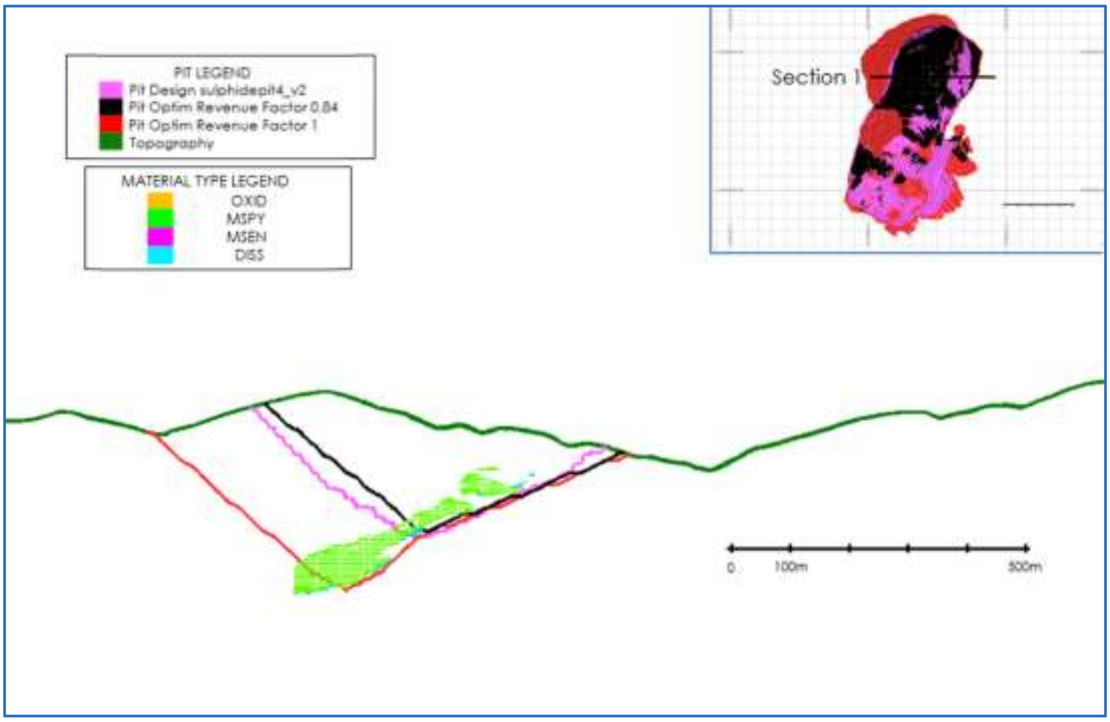


Figure by OreWin, 2019.

Figure 16.5 Pit Design with Optimisations – Section 2

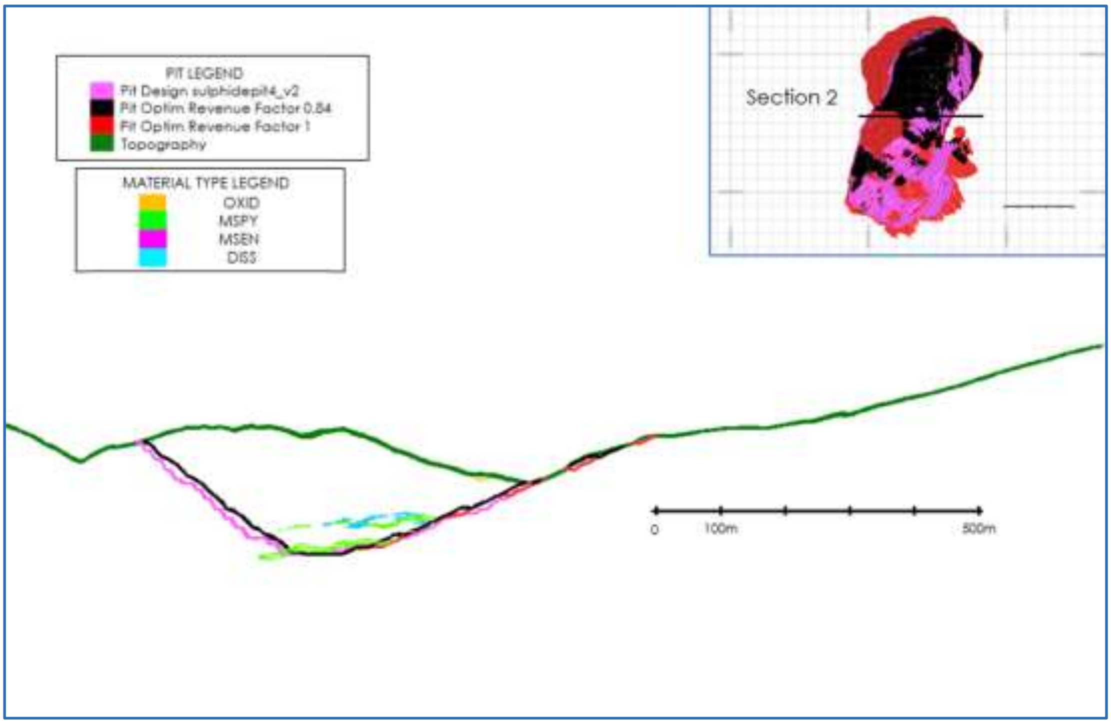


Figure by OreWin, 2019.

Figure 16.6 Pit Design with Optimisations – Section 3

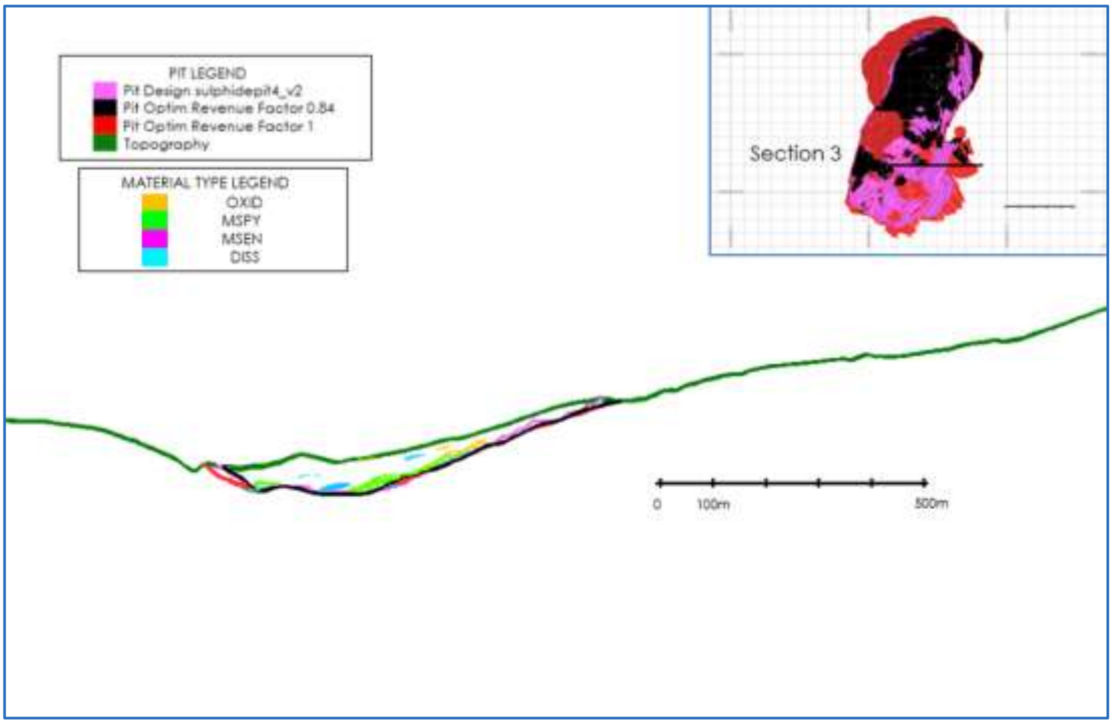


Figure by OreWin, 2019.

The material movement against operating cash flow can be seen in Figure 16.7. It is important to note that the operating cash flow here excludes capital costs.

Figure 16.7 Material Movement vs. Operating Cash Flow *

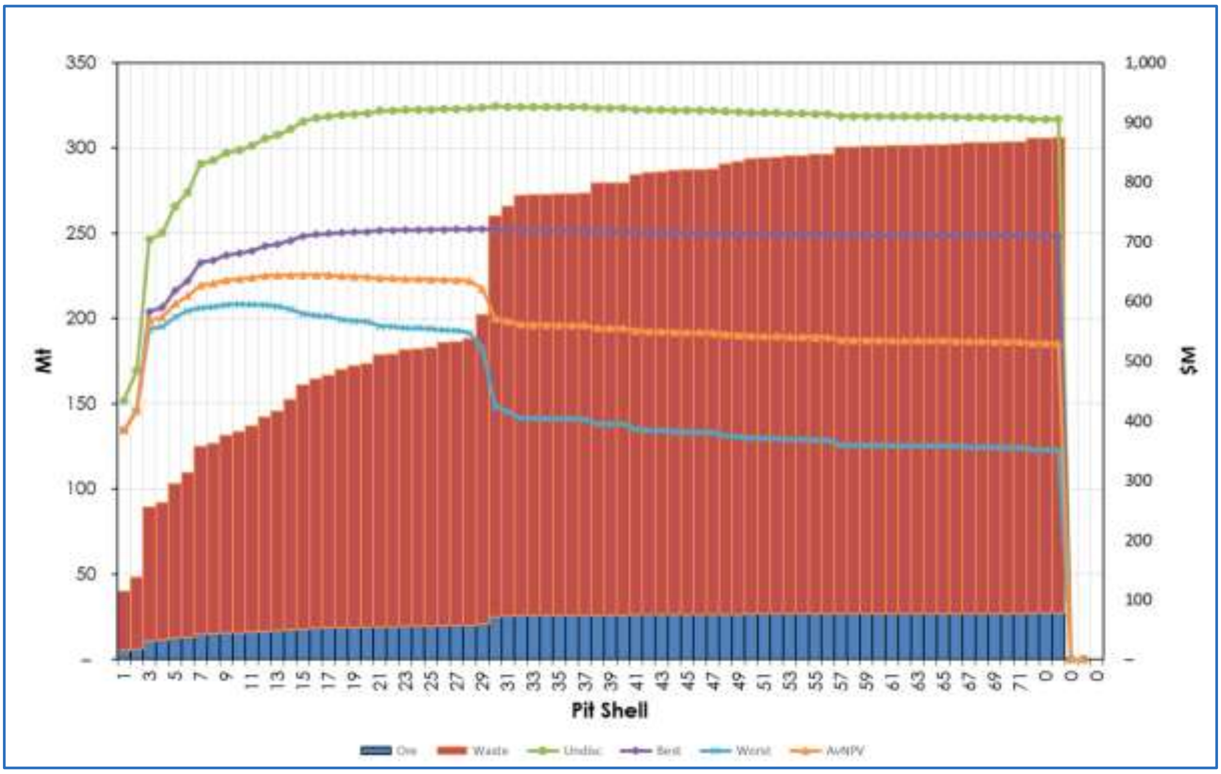


Figure from Polimetal, 2018.
* Operating cash flow excludes capital costs

16.2.6 Ultimate Pit and Stage Designs

An ultimate pit design was prepared based on shell 23. Intermediate mining stage designs were completed based on selected lower revenue factor optimisation shells.

The initial ultimate pit was reviewed to tighten compliance with the pit shell quantities. Subsequent geotechnical review resulted in additional slope configuration changes, flattening pit walls, particularly on the east wall to ensure the integrity of the proposed lined channel to accommodate the creek diversion along the eastern footwall of the pit.

The initial mining stages based on optimisation shells were further sub-divided based on geometry into logical sub-areas to provide the maximum scheduling flexibility for both ore and early construction waste. The initial stages are focussed on oxide ore due to its profitability, and to minimise the processing overlap between oxide and sulphide treatment phases. Strip ratios in some of these early stages are elevated due to the need to deliver construction waste from the mine during the pre-strip period.

Figure 16.4 shows the ultimate pit design.

Figure 16.8 Ultimate Pit Design

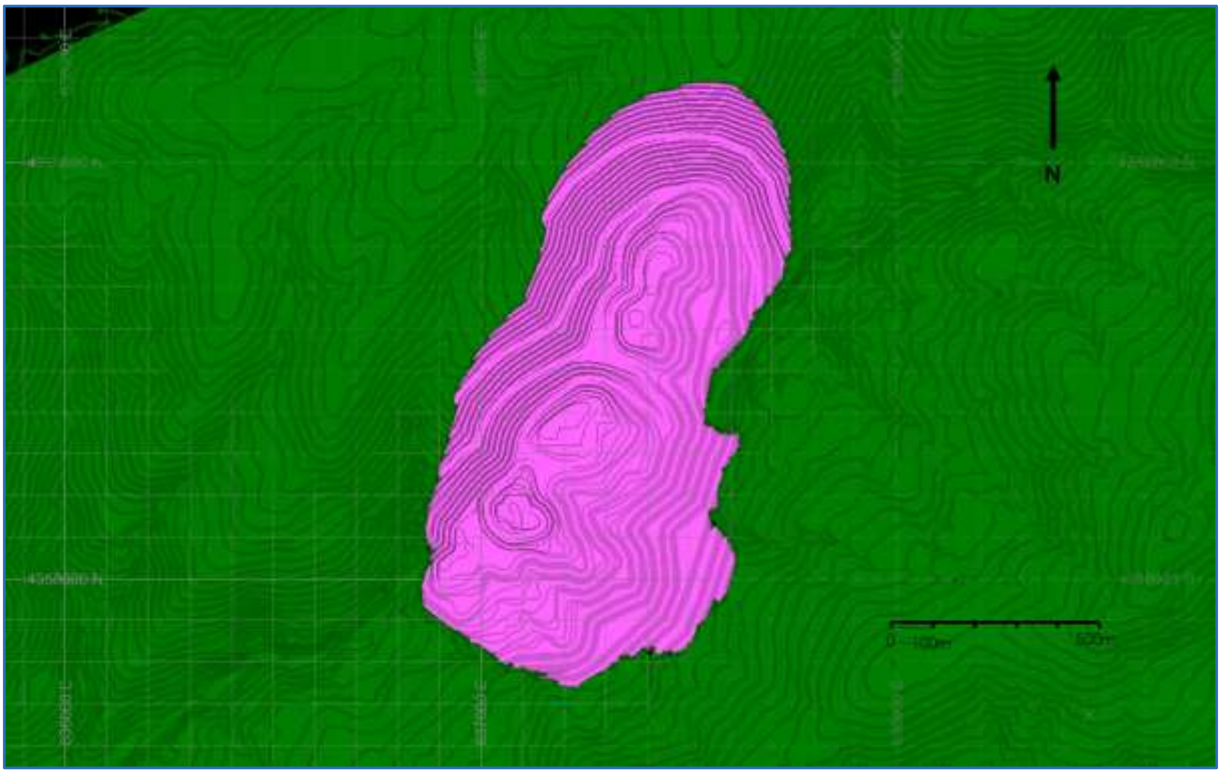


Figure from Polimetal, 2018.

16.2.7 ROM Stockpile

A run-of-mine (ROM) ore stockpile area is required immediately adjacent to the primary crusher to allow blending of sulphides to the concentrator. A number of stockpiles are required to separate ore types and facilitate concurrent reclaim to the crusher and building of new stockpiles. Additionally, the layout should allow dumping into the crusher by both trucks (direct tip) and by a front-end loader reclaiming from stockpiles.

The initial design concept incorporating up to eight small radial ‘finger’ stockpiles is shown in Figure 16.9

Figure 16.9 ROM Stockpile Schematic

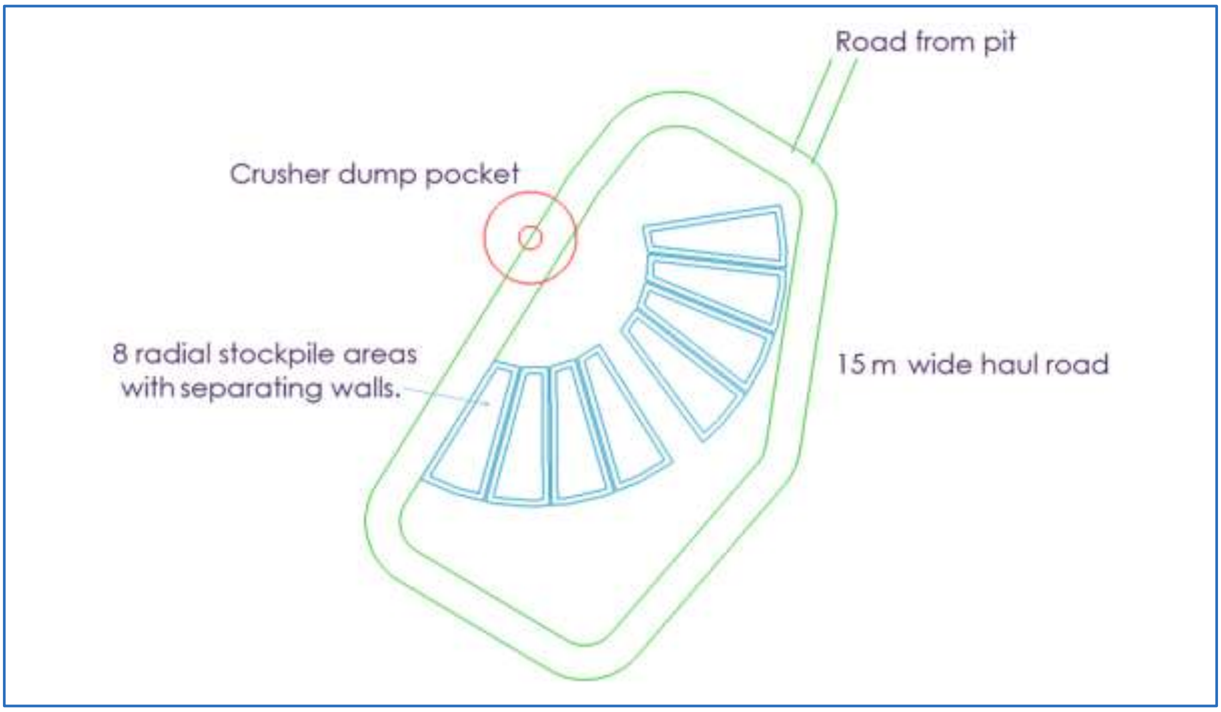


Figure from Polimetal, 2018.

16.2.8 Topsoil Stockpiles

Suitable topsoil from the mine area will be recovered prior to waste stripping and stored in an approved stockpile configuration designed to maintain soil viability for future use in mine rehabilitation. Initial areas proximal to the mine selected for topsoil storage are shown in Figure 16.10.

Figure 16.10 Mine Topsoil Stockpile Areas

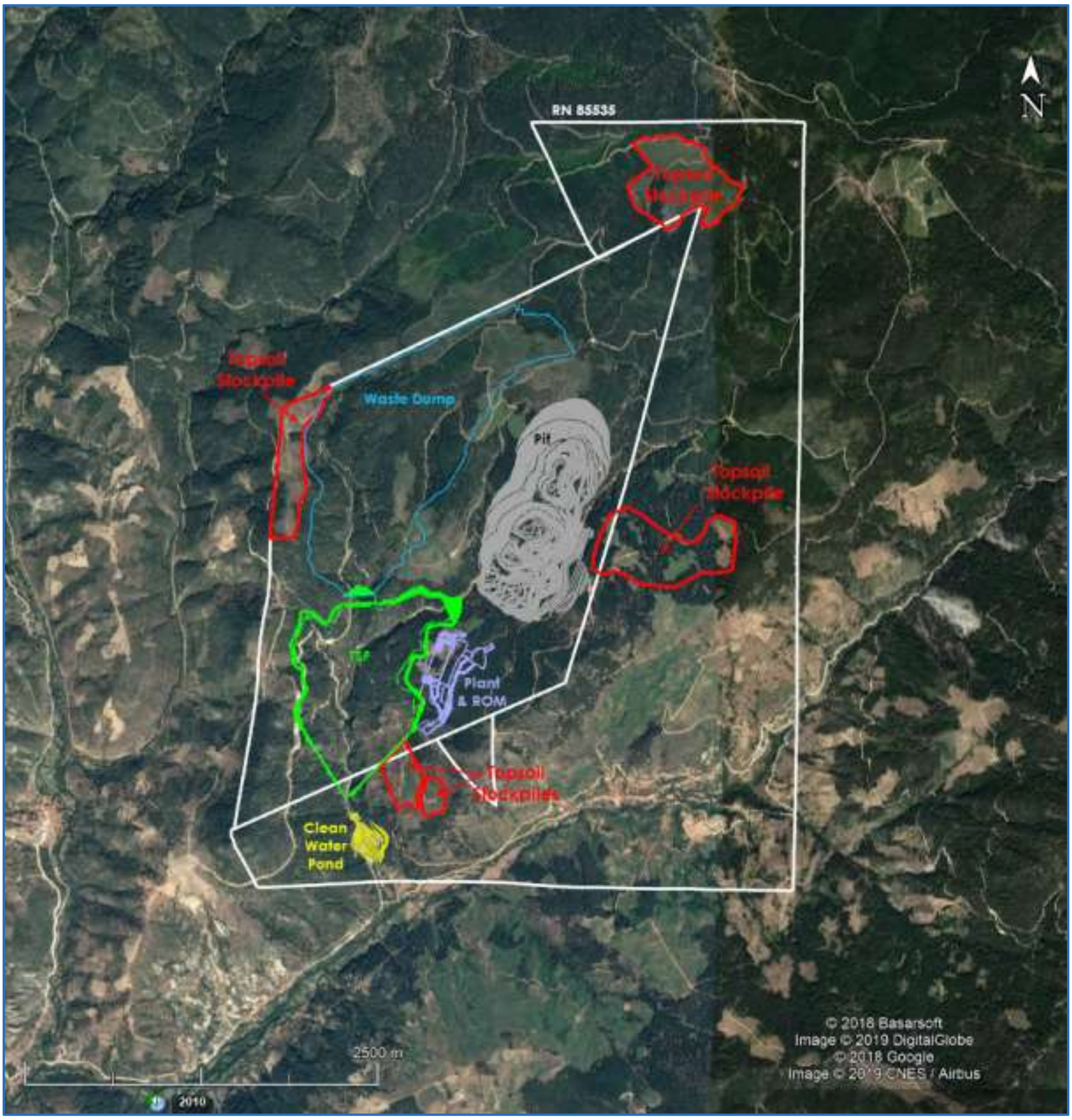


Figure by OreWin, data from Polimetal, 2018.

16.2.9 Waste Rock Dump

The initial waste rock dump located to the east of the mine up-dip of the mineralisation trend was relocated to the west after geotechnical drilling identified unfavourable foundation conditions in the preliminary eastern location.

16.3 Mining Personnel

The owner's mining team will initially support oxide mining in the first two years of production and then transition to supporting sulphide ore mining to the concentrator. Table 16.9 shows the planned make-up of the owner's team once steady state sulphide production is achieved.

Table 16.9 Owner's Mining Team

Owner's Mining Staff	Number
Mine Manager	1
Mine Production Superintendent	1
Planning Superintendent	1
Production Engineer	2
Mine Planning Engineer	3
Chief Surveyor	1
Surveyor	1
Surveyor Helper	2
Rock Mechanics Engineer	2
Chief Geologist	1
Resource and Database Geologist	2
Grade Control Geologist	2
Grade Control and Shift Supervisor	4
Sampler	8
Clerk	1

16.4 Production Schedule

Initial mining and process schedules based on producing a standard Cu concentrate for sale resulted in elevated lead in the copper concentrate. After further metallurgical testing and review of concentrate markets, a two-tier copper concentrate strategy was adopted. The majority of copper concentrate is standard specification suitable for developing smelter letters of intent and long-term sales contracts. The remaining copper concentrate is complex with lower copper and higher lead and zinc. While this complex concentrate is saleable on the spot market, albeit with some penalties, it would generally not support long-term sales contracts.

The two-tier copper concentrate strategy introduces considerable flexibility into the processing schedule and allows the opportunity to incorporate the high-grade enriched mineralisation (excluded from initial pit optimisation) into the production schedule and the Mineral Reserve.

Due to the polymetallic nature of the orebody, it is not feasible to assign a grade-based cut-off for ore definition. The ore cut-off is established based on NSR and operating costs associated with each mining model cell to determine which volumes generate revenue. This is the same process used in the pit optimisation.

At the time of scheduling for the PFS cash flow modelling there were significant changes to the initial pit optimisation parameters that influence ore definition. The more significant changes included:

- A lower metal price expectation.
- Modified concentrate metal recoveries by ore type, including the inclusion of the majority of the enriched mineralisation as ore.
- Modified processing costs based on new testwork and revised reagent consumptions and costs.

These changes reduced pit ore quantities due to a higher cut-off but this was offset by inclusion of higher grade enriched mineralisation. These modified parameters are included in Section 0, Mineral Reserve, supporting the Mineral Reserve NSR reporting cut-off that was also adopted for mine scheduling.

Mine and process scheduling was carried out on a monthly basis for the first five years (including a one-year pre-strip) and quarterly for the remainder of the mine life. It was guided by a linear programming tool to facilitate the required ore blending outcomes. The detailed period resolution was essential to verify the practicality of planned processing during the transition between oxide and sulphide mining, while honouring the requirement to minimise residence time of sulphide ore on ROM stockpiles.

During the period when both oxide and sulphide ore are available for treatment, parcels of ore will be campaign treated. This will coincide with the ramp-up period for the sulphide ore and will allow development of operating knowledge for the treatment of the sulphides allowing time to analyse data during oxide campaigns. Processing of sulphide ore will be undertaken whenever two weeks of ore supply is available (50–85 kt during ramp-up), which will generally allow treatment of sulphide ore for over three weeks. The preference for sulphide ore treatment will minimise ageing or oxidation effects. The oxide ore can be treated over shorter timeframes (one week). There will be a changeover period of one to two days to empty the coarse ore bin, flush the grinding circuit, and run down the tailings thickener. ROM pad capacity will also dictate changeover frequency.

The equipment used by both flowsheets includes the crushing plant, coarse ore bin and reclaim system, SAG mill, tailing thickener, water circuits, and some reagents.

Skilled operators will be required for the sulphide flotation process and oxide elution and gold room operations – other operators will remain unchanged despite the different ore treatment.

In addition to ore mining targets, waste mining in the pre-strip and initial years targeted minimum quantities of suitable waste to construct the clean water pond and the TSF to manage mine area run-off and ensure tailings storage availability at the commencement of oxide ore processing.

Figure 16.11 and Figure 16.12 show, respectively, total mining and ore mining by annual period.

Figure 16.11 Total Tonnage Mined by Period

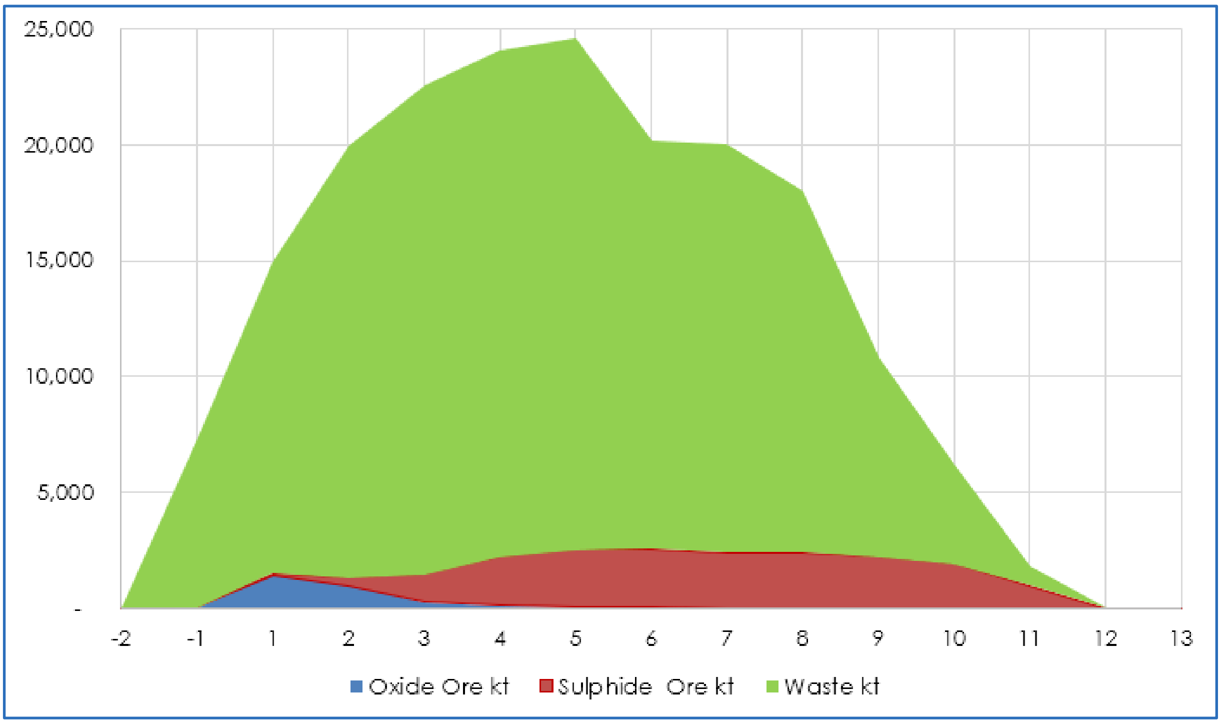


Figure by OreWin, 2019

Figure 16.12 Ore Mined by Period

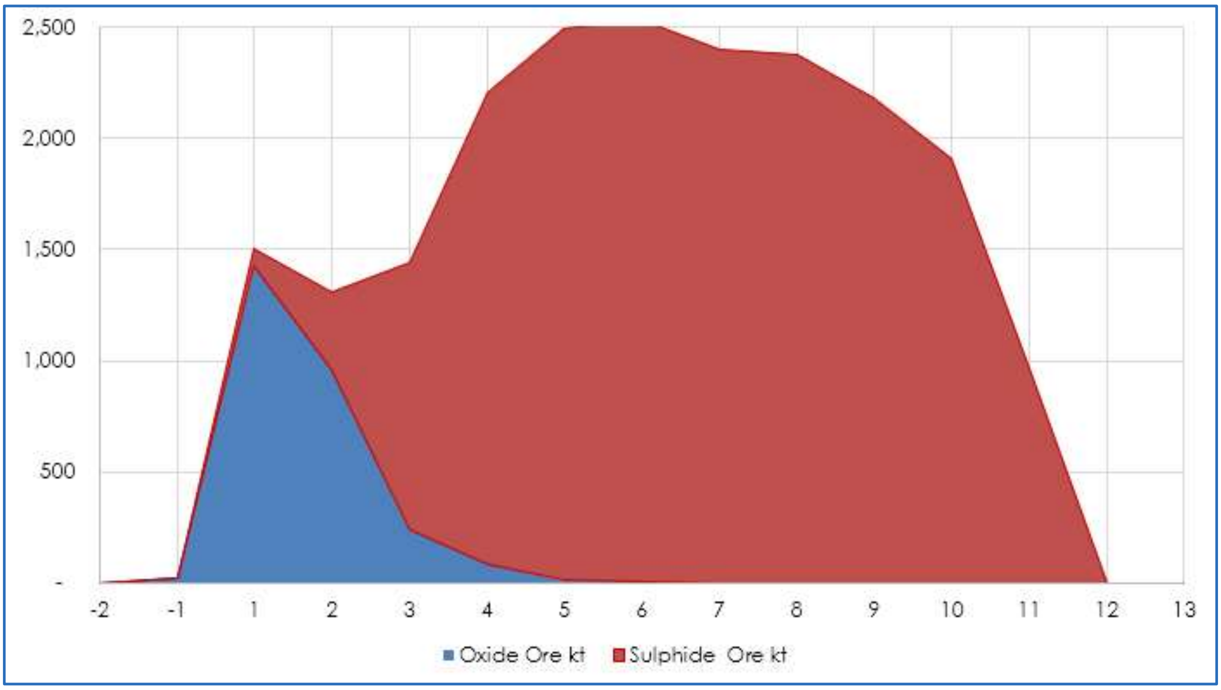


Figure by OreWin, 2019

Figure 16.13 and Figure 16.14 show the oxide and sulphide processing. The design oxide throughput at full capacity is 1,096 ktpa while the corresponding sulphide concentrator throughput is 2,378 ktpa. A portion of the lower value oxide ore mined is displaced from treatment by higher value sulphide ore and stored in a long-term stockpile for processing at the end of the mine life.

In order to manage metal recovery and the proportion and characteristics of the complex copper concentrate, inclusion of the higher grade enriched mineralisation was capped at 10% of the sulphide tonnage processed within any schedule period. After metallurgical review of the initial scheduling results, the enriched controls were expanded to exclude any enriched mineralisation with a Cu/Zn ratio of < 0.75 (approximately 20% of the available enriched tonnage), from processing.

Enriched material with a Cu/Zn ratio of < 0.75 is excluded from the Mineral Reserve.

Figure 16.13 Oxide Processing

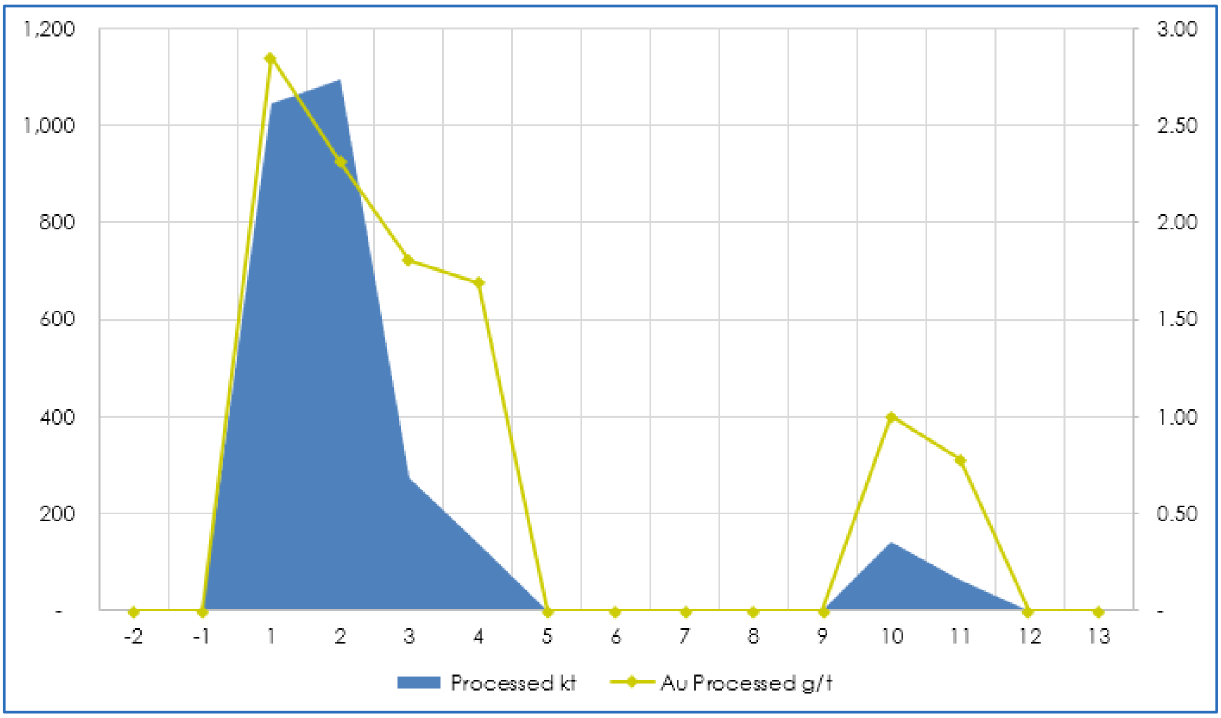


Figure by OreWin, 2019.

Figure 16.14 Sulphide Processing

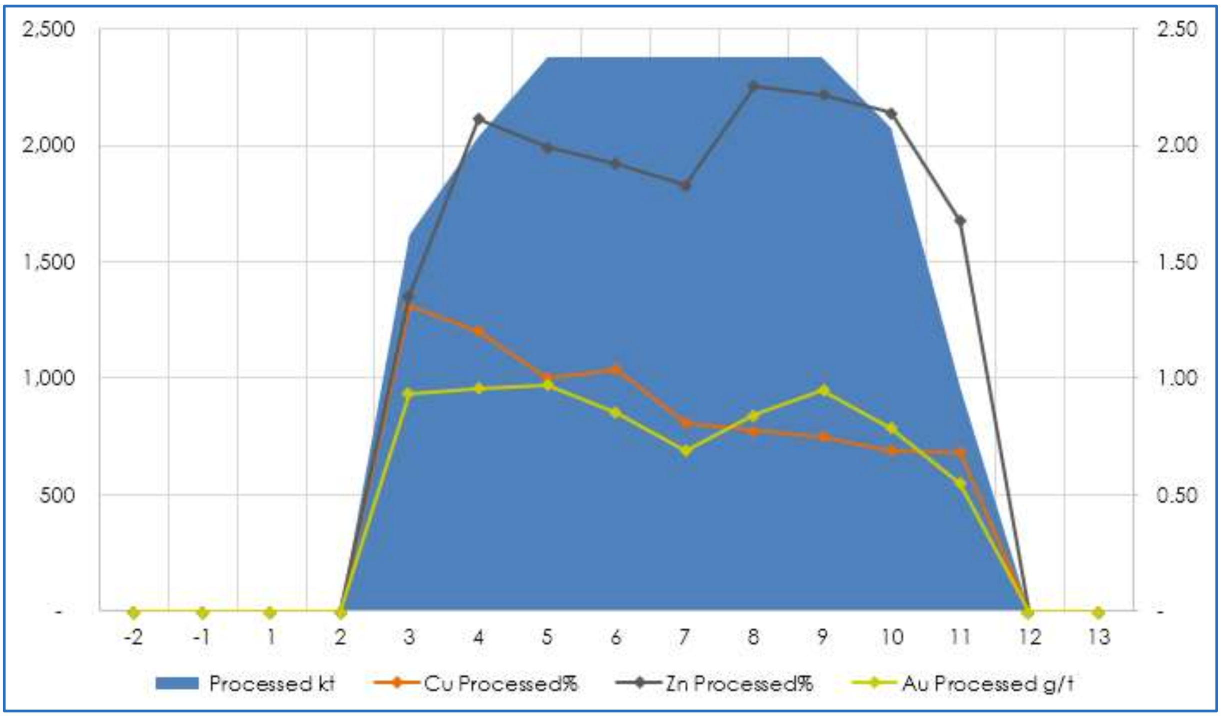


Figure by OreWin, 2019.

The processing schedules for both oxide and sulphide ores incorporate ramp-up rates with a monthly resolution advised by GRES. Oxide and sulphide ramp-up to design throughput is achieved over 4 and 5 months respectively. In addition to throughput, metal recovery factors for oxide and sulphide are also ramped up to design over a 3 to 9-month period. These recovery factors recognise the challenges associated with commissioning a new concentrator with complex ore types, including a major feed contribution from stockpiled ore that may be partially oxidised.

Ramp-up assumptions incorporated in the processing schedule are summarised in Table 16.10.

Table 16.10 Process Ramp-Up Assumptions

	% of Design Throughput and Recovery								
Month from Start-up	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**
Oxide Throughput	70%	85%	90%	100%					
Oxide Recovery Factor	70%	85%	90%						
Sulphide Throughput	60%	70%	80%	90%	100%				
Cu Recovery Factor	50%	60%	70%	75%	80%	85%	90%	95%	100%
Zn Recovery Factor	70%	75%	80%	85%	90%	95%	100%		

An important aim of the mining and processing scheduling was to minimise sulphide ore residence time on stockpiles due to the expected recovery reduction associated with ore oxidation after mining. Oxide mineralisation includes a long-term, low-grade stockpile of approximately 200 kt. Due to the lower value, the treatment of this mineralisation is deferred to the end of the mine life.

Sulphide ore stockpiles peak at about 500 kt towards the end of the mine life. Future scheduling work will aim to reduce this inventory to minimise the potential for oxidation, particularly the stockpiles of enriched ore.

Copper concentrate production commences at the beginning of year-3, initially averaging more than 50 ktpa before declining as feed grades drop after year-6, as shown in Figure 16.15. The combined copper concentrate grade averages approximately 29%, with minor variations depending on the ore type make-up of the concentrator feed and the mix between standard and complex copper concentrates.

The complex copper concentrate includes Pb and/or Zn grades that are greater than the standard copper concentrate rejection limits. The presentation of complex copper concentrate is estimated by applying a variable, period based, Pb (in copper concentrate) cut-off to separate standard and complex copper concentrates.

Zinc concentrate production also commences at the beginning of year-3 and averages between 60–70 ktpa, as shown in Figure 16.16. The zinc concentrate averages approximately 56% zinc content, varying over a small range depending on the ore types fed to the concentrator.

Figure 16.15 Annual Copper Concentrate Production

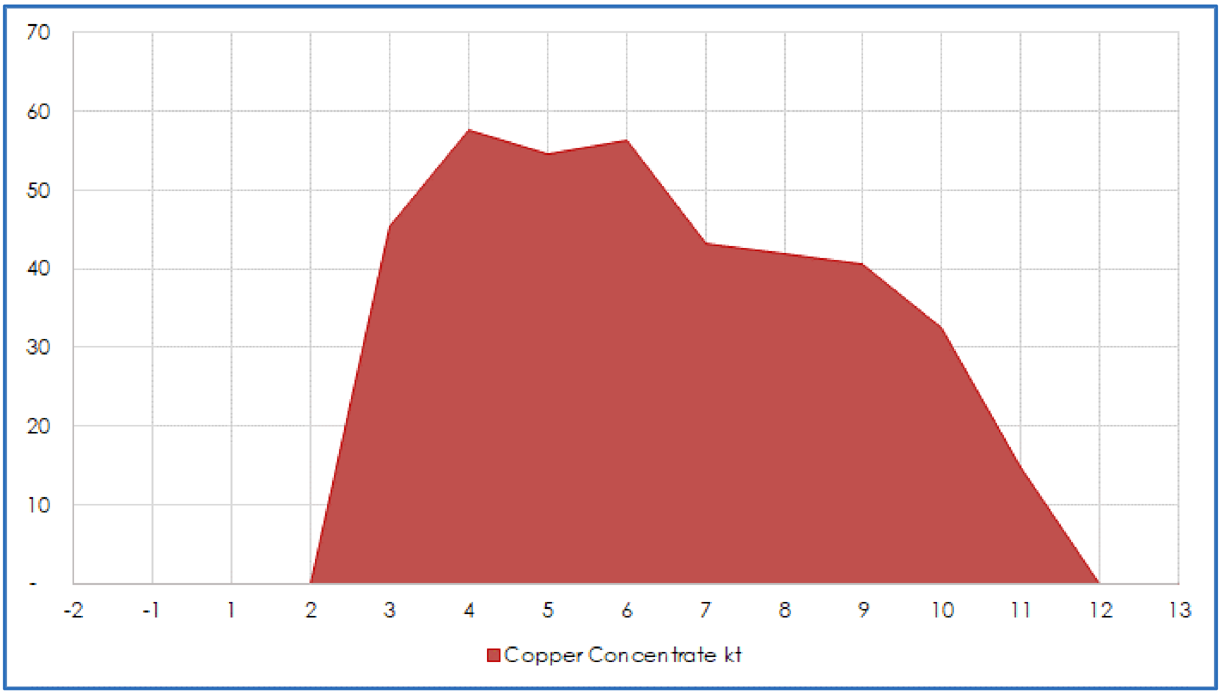


Figure by OreWin, 2019.

Figure 16.16 Annual Zinc Concentrate Production

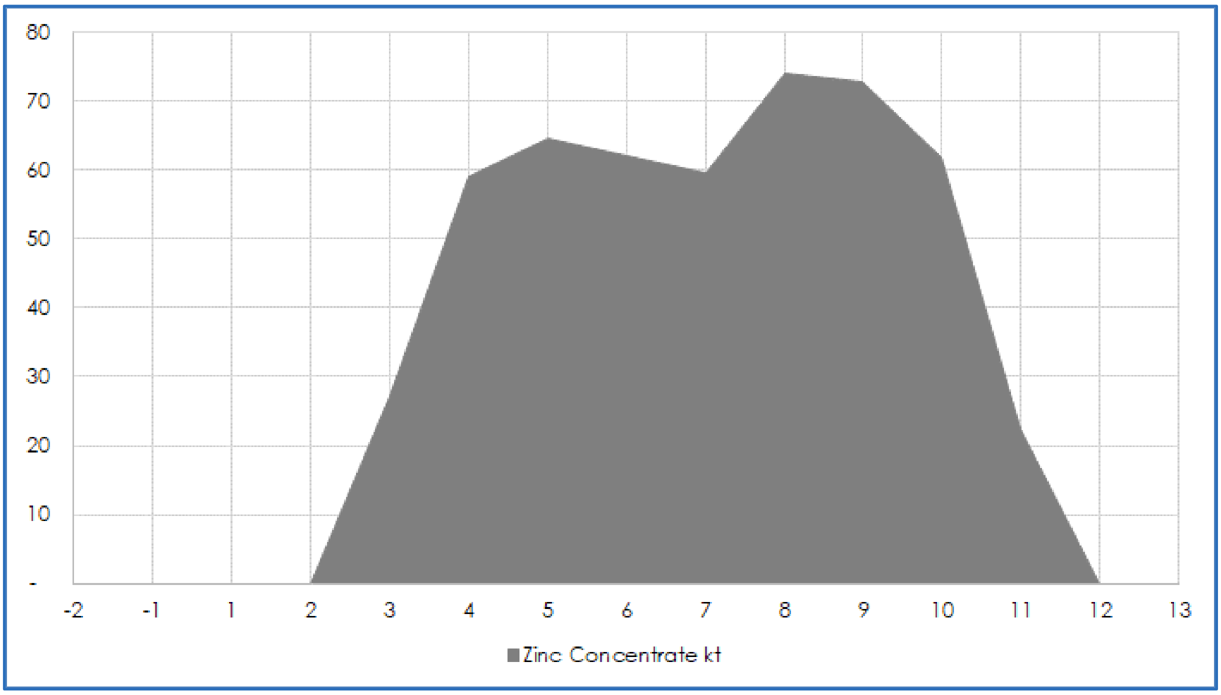


Figure by OreWin, 2019.

The recovered doré and metal in concentrate production is shown in Figure 16.17 through Figure 16.20. Table 16.11 shows the annual mining, processing, and metal production quantities. Table 16.12 includes predicted concentrate quantity and quality for the three concentrate products.

The scheduled gold and silver is initially produced as doré from oxide treatment, and then as by-product from the separate copper and zinc concentrates produced by the sulphide concentrator.

Figure 16.17 Annual Copper Production

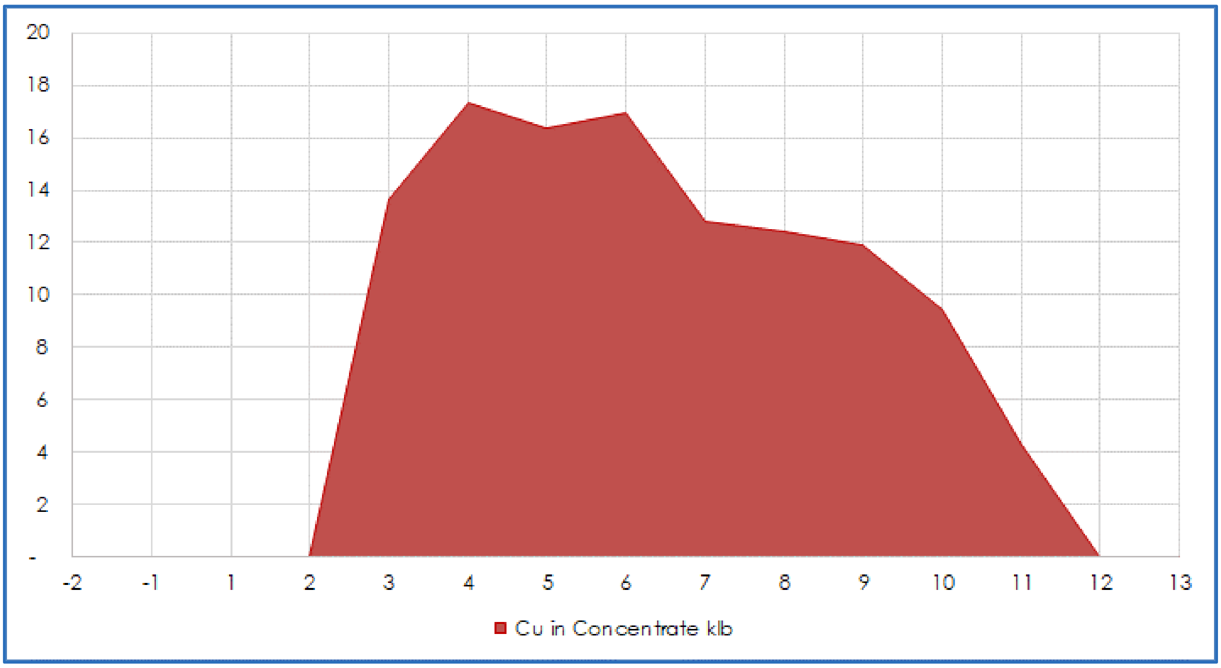


Figure by OreWin, 2019.

Figure 16.18 Annual Zinc Production

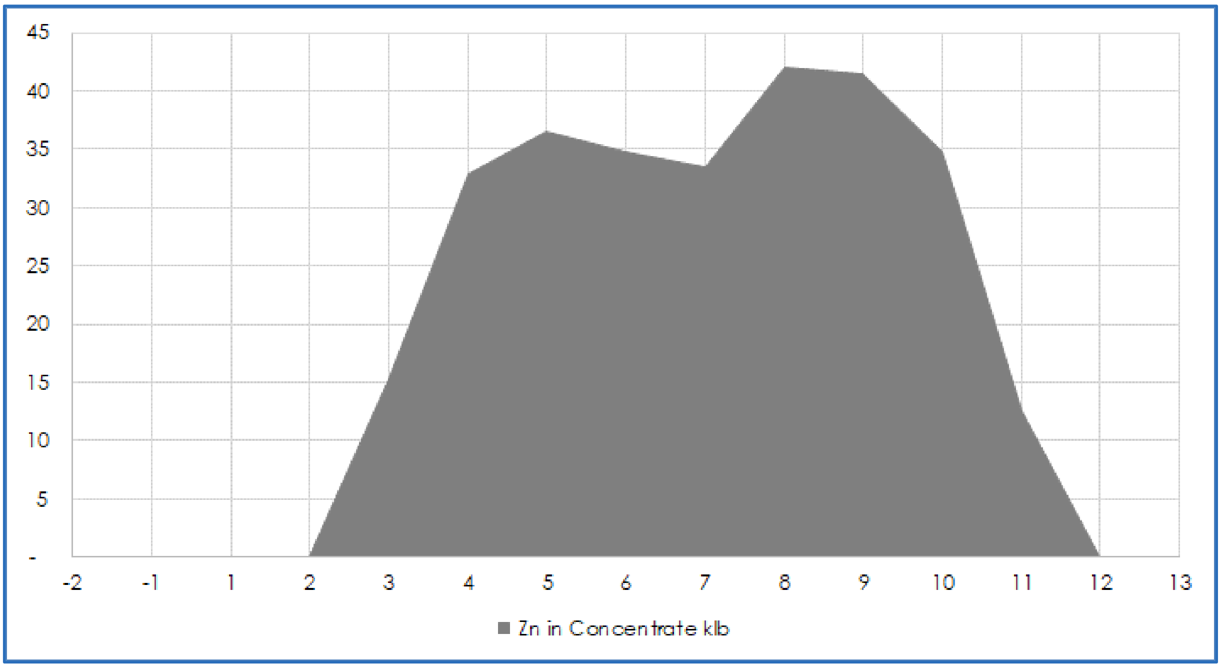


Figure by OreWin, 2019.

Figure 16.19 Annual Gold Production

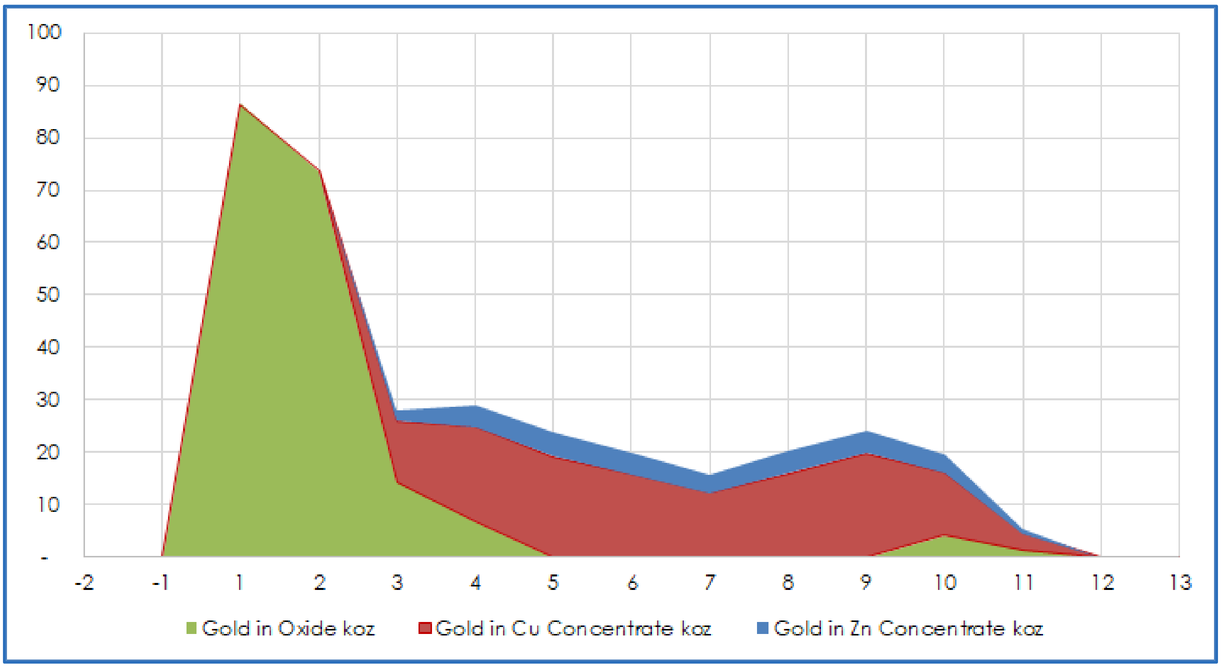


Figure by OreWin, 2019.

Figure 16.20 Annual Silver Production

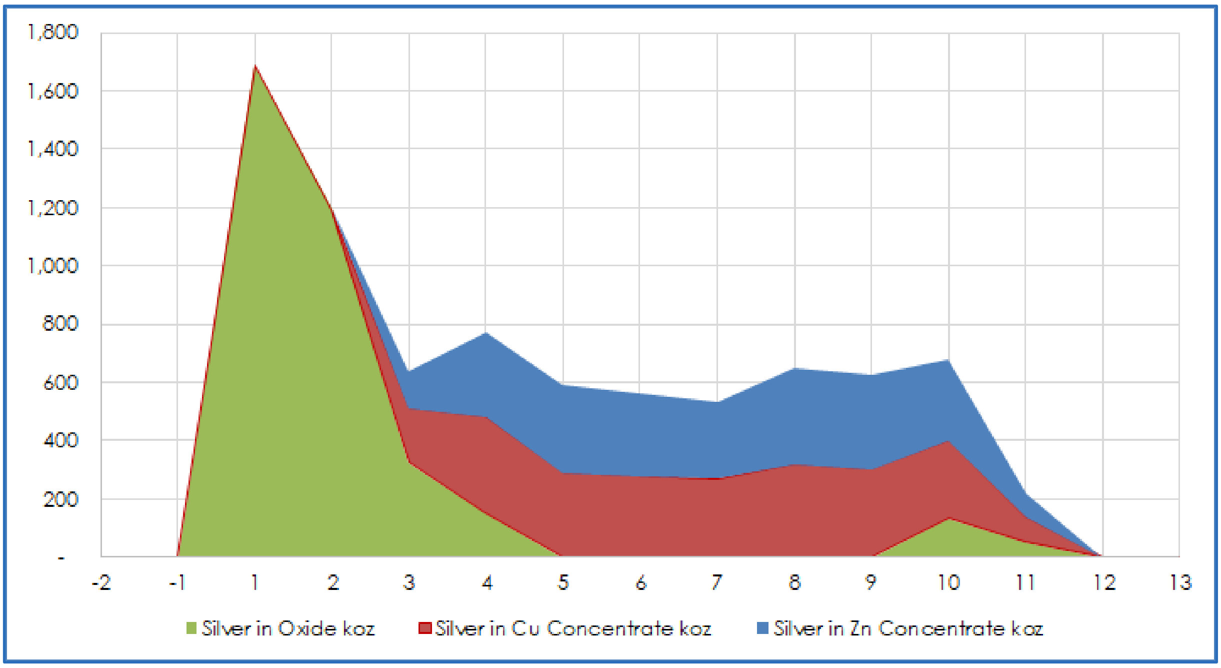


Figure by OreWin, 2019.

Table 16.11 Annual Production Quantities

Production	Units	Totals / Year	–1	1	2	3	4	5	6	7	8	9	10	11
Mine Production														
Oxide Ore	kt	2,755	23	1,426	956	242	86	12	9	1	–	–	–	–
Oxide Grade – Au	g/t	2.34	0.94	2.39	2.41	1.93	2.53	0.90	0.51	1.41	–	–	–	–
Oxide Grade – Ag	g/t	56.7	31.0	61.8	49.6	57.0	60.6	33.1	44.6	77.9	–	–	–	–
Sulphide Ore	kt	18,580	1	75	351	1,196	2,119	2,481	2,522	2,399	2,377	2,184	1,912	962
Sulphide Grade – Cu	%	0.92	0.32	0.72	0.91	1.47	1.20	0.98	1.00	0.79	0.78	0.76	0.70	0.68
Sulphide Grade – Zn	%	1.98	1.39	0.96	1.19	1.43	2.12	1.95	1.94	1.84	2.25	2.25	2.14	1.67
Sulphide Grade – Au	g/t	0.85	0.59	1.00	0.91	0.93	0.96	0.95	0.85	0.68	0.85	0.98	0.78	0.55
Sulphide Grade – Ag	g/t	31.8	14.7	37.0	28.8	33.9	37.9	31.9	30.0	28.2	33.3	35.5	31.0	20.0
Weathered Waste	kt	26,449	4,095	7,406	3,952	3,929	3,096	1,879	1,557	534	1	–	–	–
Fresh Waste	kt	142,757	3,223	6,056	14,706	17,171	18,803	20,220	16,097	17,120	15,621	8,612	4,305	823
Total Material	kt	190,541	7,342	14,964	19,965	22,538	24,105	24,592	20,184	20,054	18,000	10,796	6,217	1,785
Process Plant Production														
Oxide Ore	kt	2,755	–	1,046	1,096	274	137	–	–	–	–	–	141	61
Oxide Grade – Au	g/t	2.34	–	2.85	2.32	1.80	1.69	–	–	–	–	–	1.00	0.78
Oxide Grade – Ag	g/t	56.7	–	71.2	48.0	52.4	48.5	–	–	–	–	–	41.5	37.0
Sulphide Mill Ore	kt	18,580	–	–	–	1,618	2,041	2,378	2,378	2,378	2,378	2,378	2,072	962
Sulphide Grade – Cu	%	0.92	–	–	–	1.31	1.20	1.00	1.04	0.81	0.78	0.75	0.69	0.68
Sulphide Grade – Zn	%	1.98	–	–	–	1.35	2.11	1.99	1.92	1.83	2.25	2.22	2.14	1.67
Sulphide Grade – Au	g/t	0.85	–	–	–	0.93	0.95	0.97	0.85	0.69	0.85	0.95	0.78	0.55
Sulphide Grade – Ag	g/t	31.8	–	–	–	32.9	37.8	32.6	29.8	28.0	33.3	34.8	31.3	20.0
Metal Recovered to Doré														
Gold	koz	187	–	86	74	14	7	–	–	–	–	–	4	1
Silver	koz	3,547	–	1,690	1,195	326	151	–	–	–	–	–	133	51

Production	Units	Totals / Year	–1	1	2	3	4	5	6	7	8	9	10	11
Metal in Sulphide Concentrate														
Copper Concentrate	kt	387	–	–	–	45	58	54	56	43	42	41	32	15
Copper	klb	253,870	–	–	–	30,104	38,143	36,098	37,371	28,272	27,319	26,221	20,866	9,476
Gold	koz	128	–	–	–	12	18	19	16	12	16	20	12	3
Silver	koz	2,329	–	–	–	185	332	291	278	268	318	302	268	87
Zinc Concentrate	kt	503	–	–	–	27	59	65	62	60	74	73	62	22
Zinc	klb	625,585	–	–	–	33,582	72,564	80,479	76,729	73,637	92,589	91,443	76,802	27,760
Gold	koz	31	–	–	–	2	4	4	4	3	4	4	3	1
Silver	koz	2,272	–	–	–	124	290	299	282	263	333	325	278	79

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Table 16.12 Annual Concentrate Production

Production	Units	Total	–1	1	2	3	4	5	6	7	8	9	10	11
Standard Copper Concentrate														
Tonnage	kt	357.9	–	–	–	45.4	51.7	51.2	56.3	43.1	41.9	36.1	19.7	12.5
Cu Grade	%	29.9	–	–	–	30.1	30.2	30.2	30.1	29.8	29.5	29.5	29.9	29.5
Au Grade	g/t	10.1	–	–	–	8.0	9.3	10.6	8.8	8.8	11.9	15.2	11.3	6.1
Ag Grade	g/t	174	–	–	–	127	159	153	154	194	236	211	187	173
Zn Grade	%	3.91	–	–	–	3.32	4.88	3.67	4.28	4.64	3.94	3.28	2.65	2.50
Pb Grade	%	1.66	–	–	–	0.98	1.31	1.45	1.71	2.27	2.41	1.86	1.14	1.93
As Grade	%	0.18	–	–	–	0.18	0.18	0.23	0.23	0.20	0.14	0.12	0.07	0.11
Complex Copper Concentrate														
Tonnage	kt	28.6	–	–	–	–	6.0	3.3	–	–	–	4.5	12.7	2.2
Cu Grade	%	28.1	–	–	–	–	28.0	28.2	–	–	–	27.8	28.2	28.0
Au Grade	g/t	13.0	–	–	–	–	13.9	16.0	–	–	–	14.7	11.8	8.7
Ag Grade	g/t	358	–	–	–	–	357	366	–	–	–	388	366	245
Zn Grade	%	4.94	–	–	–	–	5.17	5.05	–	–	–	5.18	4.75	4.73
Pb Grade	%	5.67	–	–	–	–	5.31	5.81	–	–	–	5.70	5.83	5.42
As Grade	%	0.24	–	–	–	–	0.26	0.26	–	–	–	0.26	0.22	0.23
Zinc Concentrate														
Tonnage	kt	503.4	–	–	–	27.1	59.1	64.6	62.1	59.6	74.0	72.9	61.7	22.3
Zn Grade	%	56.4	–	–	–	56.1	55.7	56.5	56.1	56.0	56.8	56.9	56.5	56.4
Au Grade	g/t	1.9	–	–	–	2.3	2.1	2.2	2.1	1.8	1.8	1.8	1.7	1.3
Ag Grade	g/t	140	–	–	–	142	153	144	141	137	140	138	140	110
Cu Grade	%	2.21	–	–	–	2.85	2.65	2.17	2.30	2.30	2.08	1.87	1.95	2.22
Pb Grade	%	2.53	–	–	–	1.62	2.07	2.41	2.25	2.34	2.89	3.23	3.01	1.73
As Grade	%	0.12	–	–	–	0.13	0.15	0.13	0.14	0.13	0.10	0.10	0.11	0.12

17 RECOVERY METHODS

17.1 Introduction

The oxide processing facility has been designed to treat 1.095 Mtpa of oxide ore for approximately two years and will be followed by processing 2.4 Mtpa of sulphide ore over a total mine life of approximately 11 years. The project will therefore be installed and commissioned in two stages:

- Stage 1 oxide ore – comprising a two-year period for processing gold and silver ore, which will be treated in a single stage semi-autogenous grinding (SAG) mill circuit, followed by sodium cyanide leaching, carbon-in-pulp (CIP) and elution and electrowinning techniques to recover the gold and silver; and,
- Stage 2 sulphide ore – the oxide processing plant will be expanded to process copper and zinc-bearing ore by flotation. A 5.5 MW secondary grinding ball mill will be added to the grinding circuit. Sequential flotation will be employed to produce separate copper and zinc concentrates for export.

The major unit operations of the oxide and sulphide process flowsheets have been tested at bench scale, along with specialist vendor testwork as required.

During the period when both oxide and sulphide ore are available for treatment, parcels of ore will be campaign treated. This will coincide with the ramp-up period for the sulphide ore and will allow development of operating knowledge for the treatment of the sulphides allowing time to analyse data during oxide campaigns. Processing of sulphide ore will be undertaken whenever two weeks of ore supply is available (50–85 kt during ramp-up), which will generally allow treatment of sulphide ore for over three weeks. The preference for sulphide ore treatment will minimise ageing or oxidation effects. The oxide ore can be treated over shorter timeframes (one week). There will be a changeover period of one to two days to empty the coarse ore bin, flush the grinding circuit, and run down the tailings thickener. ROM pad capacity will also dictate changeover frequency.

17.2 Oxide Ore Recovery Methods

17.2.1 PFS16 Oxide Process Flowsheet

The oxide metallurgical testwork and processing methods during PFS16 focussed on treating the oxide ores by heap leach. A simplified flowsheet of the PFS16 heap leach circuit has been given in Figure 17.1.

The run-of-mine (ROM) ore was to be crushed in three crushing stages to produce a product with a P_{100} size of 19 mm. The crushed ore was to be discharged onto a conveyor feeding an agglomerating drum. The ore was then agglomerated with cement, lime, and sodium cyanide solution. The agglomerated ore would discharge to a conveyor to transfer the ore to the heap leach pad. At the heap leach pad, grass hopper conveyors and a stacker were to be utilised to stack the ore.

The ore would be leached for 45 days with the cyanide solution applied at an irrigation rate of 12.2 L/h/m^2. The pregnant solution from the leach pad was to be collected in a solution pond and then pumped to a Merrill Crowe (zinc precipitation) circuit where the cyanide-soluble gold and silver in the pregnant solution would be precipitated using zinc dust. The precipitate, with high amounts of elemental copper and zinc, was to be leached in a batch leach circuit with sulphuric acid. The leach residue was then filtered with the filtrate utilised within the circuit or disposed of to the tailings pond. The acid leach residue containing gold and silver was then smelted in the furnace to produce a doré bar.

The PFS19 flowsheet has been changed from the three-stage crush, heap leach flowsheet to a single-stage crush, grind, and tank leach flowsheet. The key drivers for the flowsheet change are:

- Site conditions – geotechnical investigations in 2018 identified low-strength, highly weathered schist that would be unsuitable for locating the heap leach pad and ponds. The layout options were limited and restricted to avoid the areas of highly weathered schist resulting in higher site preparation costs for heap leaching compared to the smaller footprint for an agitation leach process.
- Concerns with high-slump and lower percolation during heap leaching of high-clay ores and ability to achieve projected recoveries. Up to 60 tpd (20 kg/t) cement addition would be necessary for high-clay material.
- Materials handling concerns with oxide and clay material packing in cone crushers, low screening efficiency and blocking of chutes and transfer points.
- Better control and reaction to peaks of copper ions in solution with agitation leach as final CIP plant will include a cold cyanide wash for copper removal. High cyanide-soluble copper (CN^{sol} Cu) levels would result in high zinc dust consumption and increase copper and zinc levels in barren solution necessitating high-bleed stream flows.
- More effective use of equipment in an integrated oxide / sulphide project. For example, the single mill in the oxide circuit will be suitable for the SAG mill duty for the sulphides treatment, offsetting additional capital for the sulphide plant and total project.
- Higher gold (7%) and silver (16%) extractions from agitated tank leaching compared to heap leaching.
- Lower gold inventory in CIP tanks than in heaps.
- Avoids close out costs for detoxifying the heap.

Disadvantages of the PFS19 tank leach–CIP flowsheet include higher capital and operating costs and reduced flexibility in mining schedule (oxide material can be stockpiled and campaign crushed to the heap leach pad without impacting sulphide plant operation during the change-over mining period).

Figure 17.1 PFS16 Flowsheet for Oxide Ore Processing

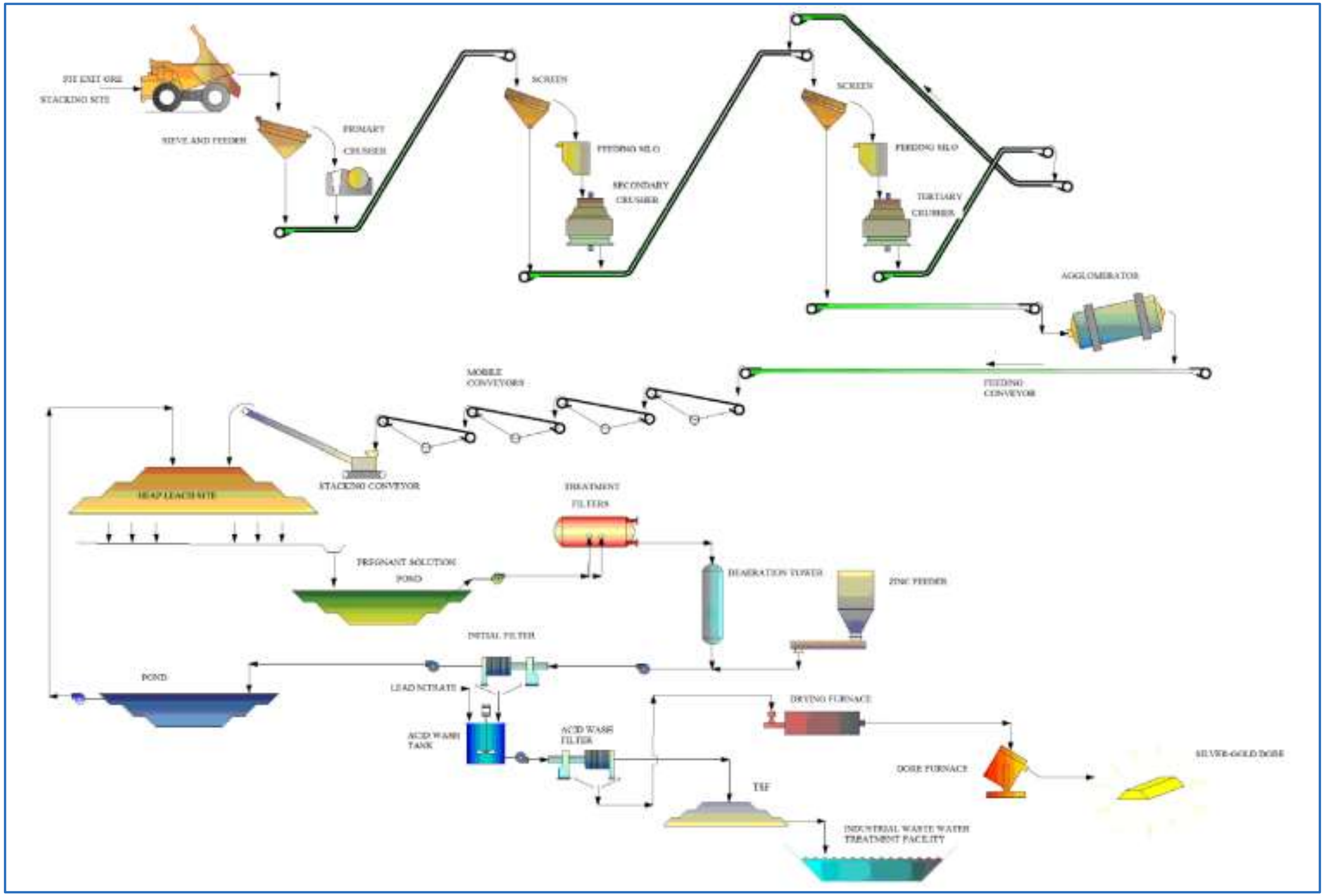

Figure from Polimetal, 2019.

17.2.2 PFS19 Oxide Process Flowsheet

Figure 17.2 provides a simplified flow diagram of the updated PFS19 oxide processing facility. The main differences from the PFS16 flowsheet are the inclusion of grinding and the application of tank leaching instead of heap leaching to extract the precious metals from the oxide ore.

The updated flowsheet has been divided into two key areas:

- Crushing – the crushing circuit will receive run-of-mine (ROM) ore from the mining operation and crush it in a single stage to a size suitable for SAG milling.
- Process plant – this will include the grinding, leaching and recovery circuits. Oxide ore will be ground in a single stage SAG mill before utilising chemical dissolution, extraction, and electrowinning to recover the gold and silver. Residual tails from the process plant will be detoxified prior to disposal in the tailings storage facility.

Various utility and plant infrastructure such as water, reagents supply and distribution, air services, fuel, power supply and distribution, roads, communications, and site buildings will support the project.

The discussion of the process plant is supported by reference to the following information:

- Process flow diagrams (PFDs);
- Piping and instrumentation diagrams (P&IDs);
- Process design criteria (PDC);
- Mass balance;
- Mechanical equipment list (MEL);
- Process control philosophy; and
- Metallurgical testwork, described in Section 13.

The PDC have been derived primarily from the metallurgical testwork completed on the master composites at ALS Metallurgy Pty Ltd in Balcatta, Western Australia, from February through November 2018.

17.2.2.1 Merrill Crowe vs. CIP Trade-off Study

A trade-off study was conducted to compare the capital and operating costs and the process risks associated with a Merrill Crowe / CIP (hybrid) flowsheet and a leach / CIP flowsheet. The methodology adopted for the trade-off was:

- Review of the CN^{sol} Cu in the master composite and variability samples;
- Comment on the expected precious metal recoveries for each flowsheet;
- Complete capital cost estimates for each flowsheet in US dollars (USD) to an accuracy of +/–30%;
- Complete operating cost estimates for each flowsheet in USD to an accuracy of +/–30%;
- Identify the process risks and opportunities for each flowsheet; and,
- Complete a differential cost benefit analysis.

The key risks identified include:

- Micronised particles of iron oxide in suspension, identified during the metallurgical testwork programme, will impact the filtration capacity of equipment in the Merrill Crowe circuit;
- Cyanide-soluble zinc and iron species loading onto the activated carbon in the CIP circuit will affect elution and acid washing, and accumulate on the carbon;
- Acid digestion of the Merrill Crowe zinc precipitate had not been tested. The evolution of hydrogen gas during digestion will have an impact on the design considerations for this plant area, particularly as the acid digest circuit located inside the gold room building;
- Copper (and zinc) recovered as a sulphide precipitate from the Merrill Crowe barren solution and the zinc precipitate acid digest liquor, with sodium hydrosulphide (NaHS) will potentially produce hydrogen sulphide (H_2S) gas, which will impact the design for this plant area, particularly as the copper recovery circuit would be located inside the gold room.

The trade-off study concluded that the CIP flowsheet with precious metal recovery by electrowinning would offer a better project return. The CIP flowsheet was selected as the recovery method for the Gediktepe oxide ore.

Figure 17.2 PFS19 Flowsheet for Oxide Ore Processing

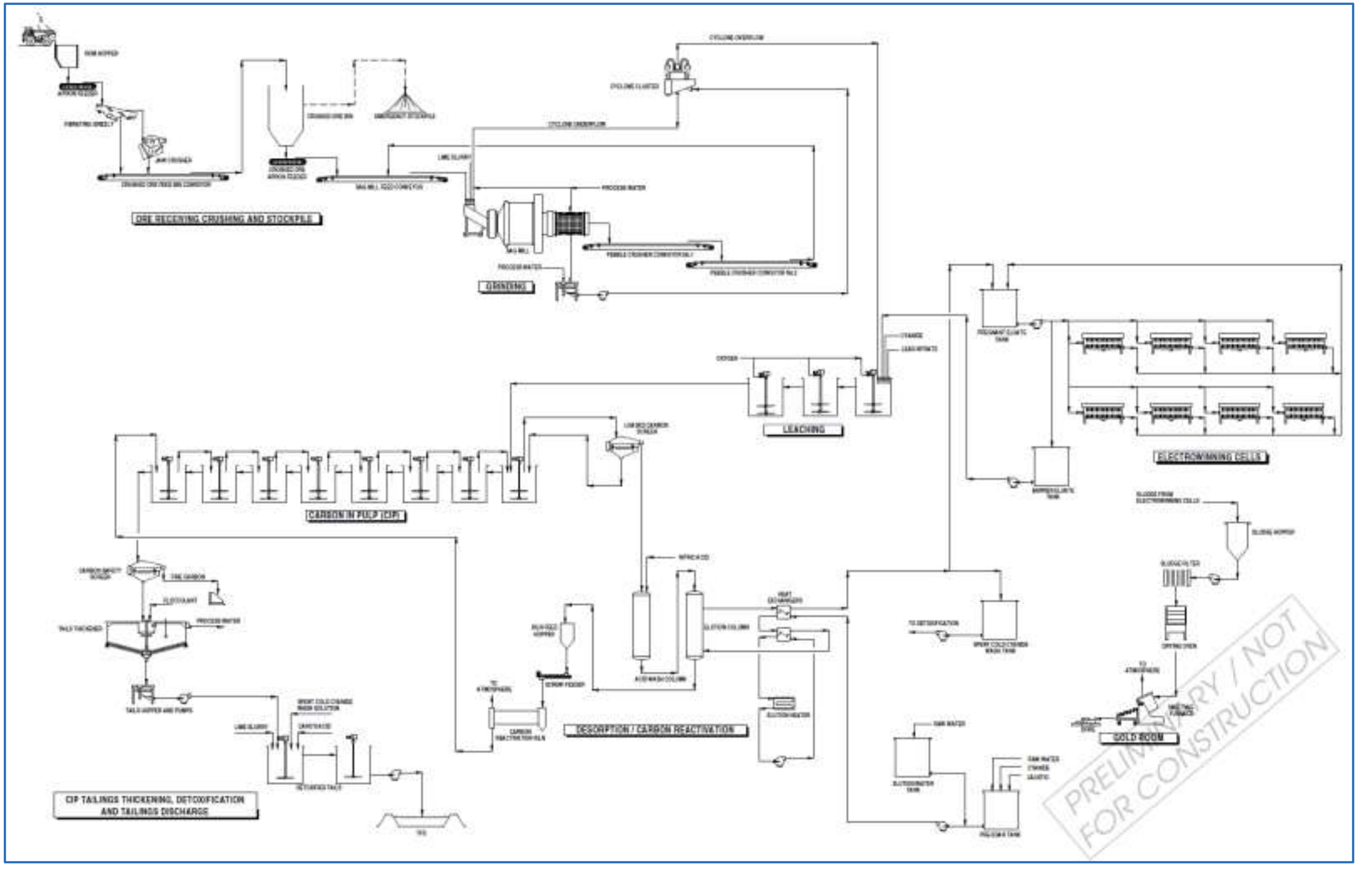

Figure from GRES, 2019.

17.2.3 Updated Process Design Basis

17.2.3.1 Oxide Plant Design Basis

The process plant design has been based on the key parameters as outlined in Table 17.1.

Table 17.1 Oxide Process Design Basis

Parameter	Unit	Value	Comments
Operating Hours – Crushing Plant	hours	6,570	MTTF and MTTR analysis
Operating Hours – Process Plant	hours	8,059	MTTF and MTTR analysis
Plant Throughput	Mtpa	1.095	Basis of design
Milling Circuit Product Grind Size	µm	125	Testwork
Feed Grade			
Gold	g/t	2.78	Mining and processing schedule – median value
Silver	g/t	62.27	Mining and processing schedule – median value
Copper	%	0.121	Mining and processing schedule – median value
Metallurgical Recoveries			
Gold	%	92.45	Median value, mass balance
Silver	%	78.34	Median value, mass balance
Water Consumption	m^3/t feed	0.51	Mass balance
Reagent Consumption			
Sodium Cyanide	kg/t feed	2.30	Mass balance
Oxygen	kg/t feed	0.18	Testwork
Lime	kg/t feed	5.59	Testwork

17.2.3.2 Materials of Construction

Due to the elevated pH and minimal presence of chlorides, the flowsheet is considered to be non-corrosive. High-density polyethylene (HDPE) piping and carbon steel with rubber lining have been selected as the main materials of construction (MoC).

For specific reagents, such as sulphuric acid, nitric acid, and hydrogen peroxide, appropriate MoC, such as, carbon steel, stainless steel, fibre reinforced plastics, and polyvinyl chloride (PVC), will be applied.

17.2.3.3 Crushing

Operational Availability

The crushing circuit, incorporating ROM ore handling, crushing, and storage, has been designed on the basis of 6,570 operating hours per annum, or 75% operational availability. This has been derived from first principles, taking into consideration planned and unplanned downtime events.

The crushing circuit will consist of a ROM ore bin, variable speed apron feeder, primary crusher and a crushed ore bin. The oxide processing circuit will be commissioned first, with the sulphide circuit commissioned in year-3. The sulphide ore processing rate is higher than that for the oxide ore and as such the crushing circuit has been designed to accommodate the sulphide ore processing rate.

Processing Rate

The dry feed rate to the crushing circuit has been calculated on the basis of the annual treatment rate and operational availability. The calculation is detailed below:

Hourly process rate	= Annual oxide ore processing rate ÷ annual operating hours
	= 2,372,500 ÷ 6,570
	= 361 tph

While processing oxide ore, the crushing circuit will have a nominal capacity of 167 tph, however the circuit has been designed to accommodate the sulphide ore processing capacity of 6,500 tpd, or 361 tph.

ROM Feed Distribution

No ROM samples were sized. The particle size distributions (PSDs) for the ROM has been derived from the BRUNO crushing simulation software and based on a 750 mm top size.

Bulk Handling Properties

No bulk handling properties of the ROM ore have been measured. The design of the ore bins, chutes, conveyors and stockpiles have been based on GRES internal database of similar ores and experience with the oxide material at Alacer's Çöpler operation.

Primary Crushing

Limited ore characterisation testwork has been completed on the oxide ore. Only one crushing work index (CWi) test has been conducted on a gossan sample for the PFS16. No uniaxial compressive strength (UCS) testing has been completed.

The abrasion index (Ai) applied is the same as that determined for the PFS16, which had a value of 0.1182. This value is indicative of low abrasivity and coupled with the low crushing work index and low strength of the coarse material as evidenced by the low Drop Weight Index (DWi) range of 2.05 to 2.64 kW/m^3, primary size reduction in a jaw crusher is considered appropriate. The JKMRC breakage characteristics indicate the gossan and clay-like gossan materials are soft.

17.2.3.4 Process Plant

Operational Availability

The process plant, consisting of areas from milling onwards, has been designed on the basis of 8,059 operating hours per annum, or 92% operational availability. The process plant will use standard industry equipment with the plant uptime derived from first principles, taking into consideration scheduled and unplanned downtime events. The design will accommodate a reasonable level of redundancy in the form of standby pumps, tank bypassing and surge capacity at critical points in the circuit.

The mismatch between the crushing circuit and process plant operational availabilities will be managed by the 25 hours surge capacity provided in the crushed ore bin.

Processing Rate

The nominal processing rate to the process plant has been calculated as follows:

Process rate = Annual high-grade ore throughput ÷ annual operating hours

= 1,095,000 ÷ 8,059

= 136 tph solids

Grinding

Ore characterisation parameters applied were determined using the PFS16 comminution studies. Further comminution testwork is planned on the variability ore types and lithologies as part of the upcoming ore variability testwork.

Ore Characterisation

The SAG mill comminution (SMC) tests conducted on gossan and disseminated gossan samples returned JK DWi values ranging from 2.05–2.64 kWh/m^3. These values are indicative of low ore competency and are in the bottom 20% of the JKTech database. The oxide ore is expected to offer little resistance to crushing or a ball charge, and unlikely to provide competent ore media in a SAG mill. The specific gravity of the tested material ranged from 2.73 to 2.85.

The A*b (rock breakage parameters) values for three samples ranged from 103.2 to 139.3, again indicating the samples were softer than most in the JKTech database.

Bond rod mill work index testing was conducted on one sample only, with a value of 11.5 kWh/t. Bond ball mill work index testing was completed by RDI in 2015, SGS in 2016, and

ALS in 2018, and ranged from 6.2 kWh/t to 11.6 kWh/t (median value of 8.9 kWh/t).

Primary Grinding Mill Sizing Criteria

Leach testwork indicated gold and silver extractions were insensitive to grind size and likely due to high ore porosity. A coarse target product size of 125 µm has been selected for design, which can be achieved by a single stage SAG mill operating in closed circuit with a cluster of hydrocyclones for classification. The design parameters for the oxide grinding mill have been summarised in Table 17.2, with the mill operating conditions summarised in Table 17.3.

Table 17.2 Basis for Primary Grinding Mill Design

Description	Units	Oxide Ore	Comments
Feed Rate	tph	136	Basis of design
Ore SG		3.014	Testwork
Feed Size (F_{80})	mm	86.5	Feed Size (F_{80}), BRUNO simulation
Product Size (P_{80})	µm	125	Target product size (P_{80})
Rock Breakage parameter		103.2	Testwork

Table 17.3 Primary Grinding Mill Selection

Description	Units	Oxide Ore	Comments
Nominal Size (inside shell diam. x length)	m	6.4 x 4.26	Based on sulphide ore
Motor Power Draw	kW	1,633	JKSimMet simulation
Installed Motor Power	kW	3,000	Based on sulphide ore
Specific Energy	kWh	12.0	Calculated

The notional mill sizing of 6.4 m (inside shell diameter) with an effective grinding length of 4.26 m has been selected with a 3,000 kW variable voltage variable frequency (VVVF) drive. A VVVF drive motor has been specified to provide turndown flexibility, which will be advantageous while processing the oxide ore.

17.2.3.5 Leaching and Adsorption

Leaching Criteria

The leach parameters have been based on the process conditions derived by bench-scale testwork completed at ALS from May through August 2018, namely:

- An initial sodium cyanide concentration of 1,000 mg/L and maintaining a sodium cyanide concentration of 500 mg/L;
- Lead nitrate addition of 100 g/t; and
- 17 hours retention time.

Based on bench-scale testwork, a total sodium cyanide input of 2.50 kg/t of feed has been determined to achieve gold and silver leach extractions ranging from 84.4% to 93.5% and 68.6% to 82.2% respectively.

The leach circuit design will comprise three mechanically agitated leach tanks arranged in a configuration that will facilitate the bypassing of one leach tank at a time to allow for maintenance while continuing to operate the remaining leach tanks. The size of the leach tanks have been based on extracting the majority of the cyanide-soluble gold and silver in the leach circuit, (see Figure 17.3 and Figure 17.4), whilst minimising the co-extraction of copper and zinc. The use of multiple tanks will ensure the mean residence time of the leach feed solids approaches seventeen hours, by minimising short-circuiting.

Figure 17.3 Leach Test Results – Gold

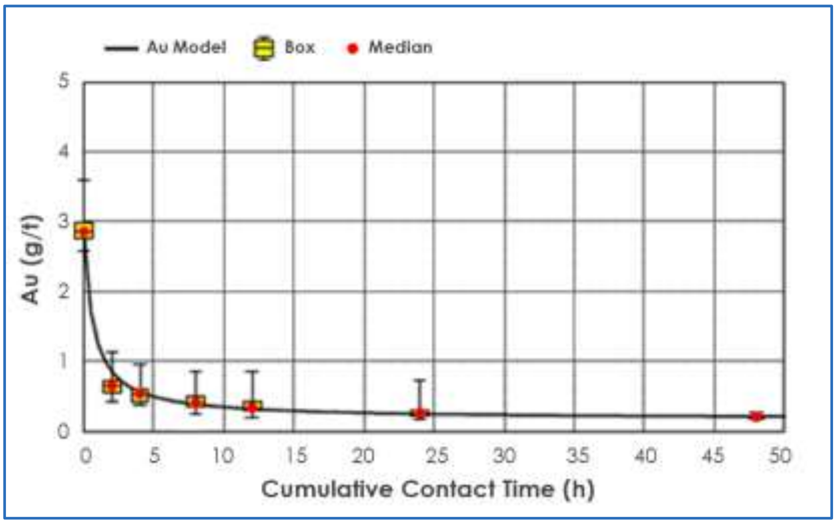

Figure from GRES, 2019

Figure 17.4 Leach Test Results – Silver

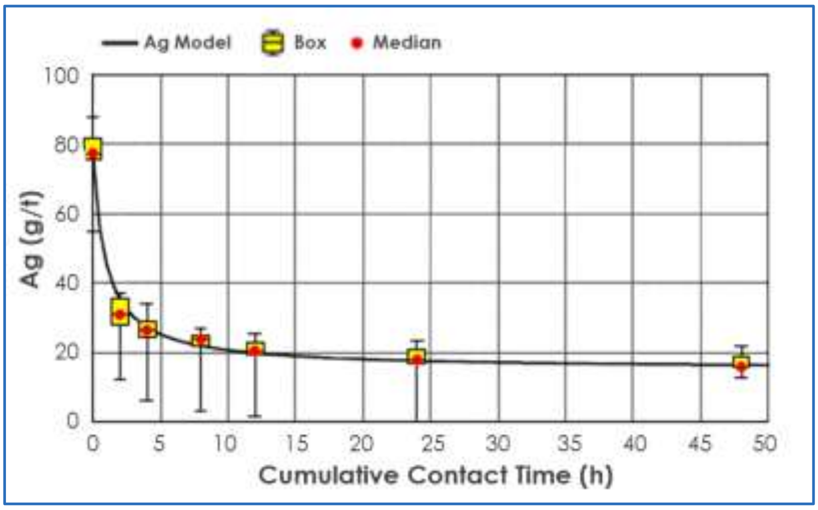

Figure from GRES, 2019

Adsorption Criteria

A continuous CIP circuit with carbon transfer by upstream airlifts has been selected for the design. The slurry will cascade from tank to tank via in-tank interstage screens. This configuration is considered typical, simple to operate and maintain. The size of the CIP tanks has been based on the outcomes of the leach / CIP bench scale testwork completed at ALS in 2018.

Eight stages of CIP contact have been recommended and designed to allow for sufficient residence time and redundancy to cope with process disruptions (having one CIP tank offline for maintenance). Tank sizing has accounted for 30% slurry back mixing as a result of the carbon being pumped counter-current to the flow of slurry.

The resultant CIP design criteria are summarised in Table 17.4

Table 17.4 Adsorption Design Criteria

Parameter	Value	Comments
Number of Contact Stages	8	
Tank Live Volume per Stage (m^3)	615	
Overall Slurry Residence Time (h)	29.6	
Overall Carbon Residence Time (days)	5.8	17.5 hours per stage
Slurry Back-mixing	30%	During carbon transfer
Precious Metal Recovery	99.4% for Au 99.1% for Ag	Mass balance, % of leached gold and silver
Carbon Loading	1,101 g Au/t 20,175 g Ag/t	Mass balance (includes barren carbon grade)

In-Tank Screens

The in-tank screens will allow the slurry to pass through while retaining the carbon in each CIP tank. The in-tank screen (one per CIP tank) will be mechanically swept to prevent blinding and to maintain a high screen flux. The in-tank screens selected for the project are typical and applied throughout the gold industry.

Screening

Various screening duties will be critical for the effective operation of a CIP circuit, namely:

- Feed slurry screening will be used to remove oversize solids debris that may be the same size or larger than the in-tank screen aperture, which would otherwise build-up within the circuit. A screen aperture size of around 800 µm has been selected. A linear vibrating screen, which provides a high capacity per unit area, has been selected for this duty;
- Carbon safety screens will be used to capture any carbon excursions from the last CIP contact stage. The screen aperture size selected will be the same as for the in-tank screens and the duty will be to prevent significant carbon loss in the event that the final in-tank screen fails. Similarly, a linear screen has been selected for this duty;
- The loaded carbon from the first CIP tank will be pumped to a screen for washing and dewatering. The aperture of this screen will be 800 µm. A linear screen has been selected for this duty.

The qualitative characteristics of the screens selected in the leach and adsorption circuit are summarised in Table 17.5

Table 17.5 Leach and Adsorption Screens

Description	Duty	Type	Cut Size (µm)
Leach Feed Trash Screen	Feed slurry screening to prevent the ingress of oversize material and debris.	Vibrating	800
CIP In-tank Screen	Retain carbon in the CIP tanks while allowing slurry to progress down the train.	Interstage screen – cylindrical swept wedge wire screens	1,000
Carbon Safety Screen	Capture lost resin in the barren slurry in the event of an excursion from RIP system.	Vibrating	1,000
Loaded Carbon Screen	Drain, wash and dewater the loaded carbon.	Vibrating	800

Activated carbon mesh size of 6 x 12 (3.35 mm x 1.7 mm).

Carbon Transfer

Carbon breakage will occur due to physical stress on the carbon. Physical stress will be induced by the attrition between the carbon and equipment. This will occur in the vicinity of the agitator blades and around sharp bends in transfer piping. The design intent will, therefore, be to minimise carbon breakage by:

- Using airlifts for transfer between carbon between the CIP tanks;

- Water eductors to transfer carbon to and from the acid wash and elution columns;
- The plant layout will minimise carbon transfer distances, and using long radius bends in carbon transfer piping; and
- Rubber lined tank agitators and baffle plates in the CIP adsorption tanks.

Carbon Inventory

The overall carbon inventory in the circuit has been estimated as shown in Table 17.6.

Table 17.6 Overall Carbon Inventory

Item	Carbon (tonnes)	Comments
CIP Circuit	47	Based on 10 g/L carbon concentration
Elution Circuit	8	One cycle per day
Regeneration Circuit	8	One regeneration cycle per day
Total	**63**	

17.2.3.6 Elution

The elution circuit will comprise a number of processing steps to elute impurities and precious metals from the loaded carbon, all of which will be performed in fixed bed columns, namely:

- Acid wash column
  - Dilute nitric acid will be used to remove impurities such as calcium, magnesium and minor quantities of copper, zinc, and iron co-loaded onto the carbon. This column will be a mild steel rubber lined vessel;
- Elution column
  - An eluent solution comprising 3% sodium cyanide and 3% sodium hydroxide solution to strip the precious metals from the loaded carbon.

A standard Anglo American Research Laboratories (AARL) strip circuit has been selected. The loaded carbon will be fed into the top of the column in batches. The eluent solution will be supplied from the bottom of the column, flowing upwards and out for the top of the column via strainers.

Column Sizing Criteria

The bed volume (BV) for the acid wash and elution columns has been based on a daily loaded carbon treatment rate of 8 t. The following design parameters have been applied:

- A carbon bed depth-to-column diameter ratio of 5:1 to 6:1 for both the acid wash and elution columns;
- Internal top and bottom wedge wire screens with 800 µm apertures;

- Operating temperature range of 115°C to 125°C due to high carbon silver loading. For this operating temperature range, Mobiltherm 603 or equivalent should be used as the boiler thermal fluid.

Electrowinning Criteria

Eluted precious metals will be recovered by eight electrowinning cells operating in parallel. The following parameters have been applied to size the electrowinning cells:

- Cathode dimensions of 1,000 mm x 1,000 mm;
- Stainless steel wire cathodes (125 µm or 152 µm diameter wire);
- 18 cathodes per cell;
- Gold and silver current efficiency of 12%;
- Gold and silver barren eluate grade of 5 g/L and < 20 mg/L respectively;
- An allowance for copper electrowinning;
- 12 to 14 hour plating time;
- Current density < 20 A/m^2.

17.2.3.7 Tailings

Thickener Criteria

Sizing parameters for the tails thickener have been based on batch dynamic thickening testwork completed by Outotec on a master composite leach tail sample.

For the tails thickener a specific settling rate of 1.0 $t/m^2/h$ has been applied to obtain a required thickener diameter of 14 m. However, as the tails thickener will be used in the sulphide circuit to dewater flotation tails, a 23 m diameter thickener has been selected for the tails thickening duty. The tails thickener will be of the high rate style.

The oxide tailings will achieve a thickener underflow density of between 55% and 60% solids.

Cyanide Detoxification

A WAD cyanide level in the tailings discharge to the TSF target of less than 5 ppm has been applied for design. Testwork using Caro's acid (hydrogen peroxide and sulphuric acid) achieved levels of 3 ppm WAD cyanide.

17.2.3.8 Water Management

The raw water consumption in the process plant will be driven by the volume of water that reports to the TSF as part of the plant tails. Water surge for the process plant will be provided by various dams and tanks as outlined in Table 17.7.

Table 17.7 Water Surge Capacities

Storage	Storage Volume (m^3)	Estimated Live Capacity (hours)
Non-contact Water Dam	3,200	30
Raw Water Tank	1,000	12
Process Water Dam	11,950	10

17.2.4 Process Risks

Process risks and the control measures identified include:

- Limited ore characterisation testwork has been conducted. The primary grinding mill has been sized to accommodate the harder sulphide ore and therefore the primary mill will be oversized for the softer oxide ore. The primary grinding mill will have a variable speed drive, which will allow operational flexibility.
- Additional variability testing will be necessary to gain a greater understanding the extent of CN^{sol} Cu throughout the deposit.
- CN^{sol} Zn and Fe-species loading onto activated carbon in the CIP circuit might be difficult to elute during acid washing, reducing the activity of the carbon. Acid wash and elution testwork is to be completed.

17.2.5 Process Description – Oxide Treatment

17.2.5.1 Crushing

The crushing circuit has been designed for a maximum treatment rate of 6,500 dry tonnes per day (tpd).

The ROM ore will be loaded into the ROM bin by a front-end loader or direct tipped by 25 t dump trucks. A 750 mm static grizzly will be fitted to the ROM bin to protect it, and all downstream processing equipment from oversize material. The static grizzly will be inclined and hinged to allow easy removal of oversize material or in case of a blockage or hang-up.

Mining will be required to supply ore at a P_{100} of 750 mm to minimise grizzly cleaning requirements. Any oversize ore will be scalped from the screen and stockpiled adjacent to the ROM bin.

The ROM ore will be drawn from the ROM bin at a controlled rate by a variable speed apron feeder and discharged onto the vibrating grizzly feeder equipped with 90mm bar spacing's. Oversize material from the vibrating grizzly feeder will discharge into the primary crusher. The undersize material from the vibrating grizzly feeder will gravitate onto the primary crusher discharge conveyor. The primary crusher is a 160 kW Metso C-120 single toggle jaw crusher single with a 1,200 mm by 870 mm gape and will operate with a closed side setting (CSS) of 80 mm to give a product with a P_{80} < 90 mm. The crusher product will discharge onto the primary crusher discharge conveyor and be transferred to the crushed ore bin feed conveyor that then discharges into the crushed ore surge bin. A dust collector will be positioned at the end of the primary crusher discharge conveyor for dust control and management.

The speed of the variable speed apron feeder will be controlled by a PID controller to maintain an overall circuit throughput rate as measured by the weightometer on the crushed ore bin feed conveyor.

Process spillage in the crushing area will be pumped to the primary mill discharge hopper by a sump pump.

17.2.5.2 Crushed Ore Reclaim

Under normal operating conditions, the rate of crushed ore into the 4,200 t capacity (sulphide ore) crushed ore surge bin will exceed the rate of withdrawal of ore to the milling circuit. The crushed ore surge bin will be designed to allow the withdrawal of excess crushed material for stockpiling and future reclamation via the emergency feed bin and emergency feed bin belt feeder. The emergency feed bin will be fed with a front-end loader during periods of crusher downtime.

The ore from the crushed ore surge bin will be withdrawn at a controlled rate by two variable speed reclaim belt feeders operating in parallel that will discharge onto the primary mill feed conveyor. A weightometer will indicate the instantaneous and totalised mill feed tonnage and will be used to control the speed of the reclaim belt feeders.

A dust collector will be positioned at the top of the surge bin for dust control and management. Process spillage in the reclaim area will be pumped to the primary mill discharge hopper by a sump pump.

17.2.5.3 Grinding and Classification

The milling circuit will consist of a single stage SAG mill in closed circuit with a hydrocyclone cluster. Process water and lime slurry will be added to the mill feed chute to control the mill discharge density and slurry pH respectively.

The primary SAG mill will be of the grate discharge type with a diameter of 6.4 m (inside shell) and an effective grinding length of 4.23 m. The mill will operate with a nominal ball load of 6% by volume and an operating critical speed ranging from 60% to 78%. The SAG mill will be powered by a 3,000 kW motor with variable speed capability. The mill power draw and product size will be controlled by the periodic addition of grinding media. Media addition to the mill will be a field operator task.

Slurry passing through the mill discharge grate will flow to the discharge trommel equipped with a spray bar. The slurry will be separated from oversize pebbles and undersized mill balls, with the washed oversize material exiting the trommel onto the pebble transfer conveyor and recycled back to the SAG mill feed chute via the pebble recycle conveyor. Trommel screen undersize material will discharge into the primary mill discharge hopper.

Process water is also added to the primary mill discharge hopper. The added process water serves two purposes:

- To dilute the discharge slurry prior to being pumped to the classifying hydrocyclone cluster, and;
- Control the level in the primary mill discharge hopper.

The combined slurry will be pumped from the primary mill discharge hopper to the primary mill cyclone cluster by the variable speed primary mill cyclone feed pumps. The cyclone underflow stream will be returned to the primary SAG mill for further grinding, while the cyclone overflow (target P_{80} size of 125 µm) will be directed to the tank leach trash screen to prevent the introduction of oversize material and debris into the leaching and adsorption circuit. Oversize material from the trash screen will report to a trash bin, whilst the trash screen underflow will report to the leach feed distribution box.

Process spillage in the milling area will be controlled by two sump pumps.

17.2.5.4 Leach and Adsorption

The leach and adsorption circuit will consist of three 1,370 m^3 agitated leach tanks and eight 615 m^3 agitated carbon in pulp (CIP) tanks.

Trash screen underflow will report to the leach feed distribution box. Sodium cyanide solution, lime slurry and lead nitrate solution are also added to the leach feed distribution box. Lime slurry will be added to ensure the circuit pH is maintained at or above the target pH setpoint of 11.0. Sodium cyanide will be added to achieve an initial sodium cyanide concentration in the first leach tank of 1,000 mg/L and a lead nitrate addition of 100 g/t. Oxygen gas (> 99% purity), will be added to each leach tank via the leach tank agitator shaft.

Slurry discharging from the last leach tank will flow by launder arrangement to the first of eight CIP adsorption tanks.

The eight adsorption tanks, providing a total residence time of 31 hours, will be interconnected with launders and slurry will sequentially flow through each tank. Each tank will be fitted with a dual impellor mechanical agitator to ensure uniform mixing. The tanks will also be equipped with mechanically swept woven wire intertank screens to retain the carbon inventories. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for tank, agitator or screen maintenance.

Barren carbon will be returned to the circuit at CIP tank 8 and will advance counter current to the slurry flow by pumping slurry and carbon from tank 8 to tank 7 and so forth using recessed impellor pumps. The intertank screen in tank 7 will retain the carbon and the slurry will flow by gravity back to tank 8. This counter current process will be repeated until the carbon reaches tank 1, the first CIP tank. A recessed impellor pump will be used to transfer slurry containing loaded carbon to the loaded carbon screen mounted above the acid wash column. The loaded carbon will be washed and dewatered on the loaded carbon screen prior to reporting to the acid wash column. The associated slurry and wash water will return to CIP tank 1.

Slurry from the last CIP tank, CIP tank 8, will gravitate to the vibrating carbon safety screen.

17.2.5.5 Precious Metals Recovery – Elution

Carbon will be advanced at a rate of 8 tpd. The target loaded carbon grade is 1,101 g Au/t and about 20,175 g Ag/t. The loaded carbon will be eluted in a standard AARL elution circuit.

Acid Wash

Acid washing of the loaded carbon will be conducted by diluting concentrated nitric acid with raw water to a concentration of 3% w/w in the column. Nitric acid has been selected to minimise the circulation and build-up of chlorides in the circuit. During acid washing the dilute solution of nitric acid will be pumped through the column in an up-flow direction to remove contaminants, predominantly carbonates, from the loaded carbon. This process improves the elution efficiency and has the beneficial effect of reducing the risk of calcium–magnesium 'slagging' within the carbon during the regeneration process.

After acid washing, the carbon bed will be rinsed with raw water. Four bed volumes of raw water will be pumped through the column to displace any residual acid from the carbon. Dilute nitric acid and rinse water will be directed to the cyanide detoxification circuit.

Cold Cyanide Wash

Following acid washing, the loaded carbon will be hydraulically transferred to the elution column, where the loaded carbon will be cold cyanide washed. The cyanide wash stage will be conducted by diluting concentrated sodium cyanide solution with raw water to a concentration of 5% w/v. During cold cyanide washing the dilute cyanide solution will be pumped through the column in an up-flow direction, to remove cyanide-soluble contaminants, predominantly copper, from the loaded carbon. The process aids in improving the carbon activity and will minimise copper contamination of the eluate.

After the cold cyanide washing the carbon bed will be rinsed with raw water. Four bed volumes of water will be pumped through the column to displace any residual cyanide from the carbon. The spent cold cyanide wash liquor will be directed to the spent cold cyanide wash tank and then pumped at a controlled rate to the cyanide detoxification circuit.

Pre-Soak and Elution

Strip solution containing 3% sodium cyanide and 3% sodium hydroxide (caustic) will be pumped from the pre-soak tank through heat exchangers into the base of the elution column.

The loaded carbon will be soaked in the cyanide / caustic solution to condition the gold and silver for elution. The carbon will then be eluted by hot elution water passed through the column, with the pregnant eluate directed to one of two electrolyte tanks. At the end of the elution process a cooling stage will be deployed to cool the column contents.

Carbon Reactivation Kiln

The barren carbon will be transferred from the elution column to the kiln dewatering screen. The carbon regeneration kiln will be a horizontal LPG fired rotary type unit with a nominal capacity of 400 kg/h.

The barren carbon will be hydraulically transferred to the kiln dewatering screen from the elution column. The dewatered barren carbon will discharge into the kiln feed hopper. From here, the barren carbon enters the kiln and will be re-activated at a temperature of 700°C. The re-activated carbon will discharge from the kiln into the carbon quench tank. From here, the regenerated carbon will be hydraulically transferred to the barren carbon screen.

17.2.5.6 Precious Metals Recovery – Electrowinning

The electrowinning circuit has been designed to treat the pregnant eluate containing silver and gold in a sodium cyanide-based solution.

Electrowinning

Pregnant eluate will be pumped from the pregnant eluate tank through eight electrowinning cells operating in parallel. Each 1,000 mm by 1,000 mm cell will contain eighteen stainless steel wire cathodes. A rectifier will supply current to each cell to enable electrowinning of the precious metals to the cathode surface. Eluate overflowing each cell will report back to the pregnant eluate tank and will be continuously recirculated through the electrowinning cells until the residual gold and silver grade in the barren solution is below 5 ppm and 20 ppm respectively, at which time the solution will be diverted to the barren eluate tank (and then pumped, in a controlled manner to the leach circuit. The electrowinning process requires approximately 12 to 14 hours.

Electrowinning Cell Harvesting

Gold and silver sludge will be harvested from the electrowinning cells daily. A high-pressure water gurney will be used to dislodge and remove the gold and silver sludge from the stainless steel wire cathode. The sludge will flow by gravity to the sealed sludge holding tank and then pumped to the sludge filter. The filtered sludge will be manually collected and transferred to the retort oven. Filtrate from the sludge filter will be recycled to the sludge holding tank until the filtrate is clear. Thereafter, the filtrate will be directed to the gold room sump pump and then pumped to the leach circuit.

17.2.5.7 Precious Metals Recovery – Gold Room

Filtered sludge from the electrowinning circuit will be transferred manually to the retort oven for the retorting and collection of mercury. Volatilised mercury will be condensed and captured by the mercury condenser. The non-condensable gasses will then pass through a carbon column to ensure the vent gas emitted to the atmosphere meets the required standard.

The dry retorted precious metal sludge will be smelted with fluxes in the LPG fired smelting furnace to produce doré bars. Slag from smelting operations will be returned manually to the primary mill feed box. Fumes generated during smelting will be vented to the atmosphere.

17.2.5.8 Major Reagents

The process plant will be supported by various reagents, all of which have been detailed below.

Hydrogen Peroxide

Hydrogen peroxide (60% H_2O_2) will be delivered to the plant by truck in a 20 m^3 iso-container. Hydrogen peroxide will be used in the cyanide detoxification circuit.

Lime Mixing and Distribution

Hydrated lime will be delivered to the site in bulk by road tanker. The road tankers will be pneumatically unloaded directly to the lime silo. Lime will be metered from the silo by rotary valve and screw feeder and discharged into the mechanically agitated lime mixing tank. The lime slurry is then pumped from the mixing tank to the agitated lime slurry storage tank and distributed throughout the plant on a ring main system.

A dust collector will collect dust during loading of lime into the storage silo and discharge that dust into the lime silo.

Sodium Cyanide Mixing and Distribution

Sodium cyanide (NaCN) briquettes will be supplied to the plant in bulka bags. The sodium cyanide briquettes will be dissolved in an agitated mixing tank with raw water and sodium hydroxide and then transferred to a storage tank. From there the sodium cyanide solution will be pumped to the leach and adsorption circuit and the elution circuit.

Sodium Hydroxide (Caustic)

Sodium hydroxide (50% NaOH) will be delivered to the plant by truck in a 20 m^3 iso-container. Sodium hydroxide will be used in the elution circuit for loaded carbon stripping.

Nitric Acid

Nitric acid will be delivered to the plant by truck in a 15 m^3 iso-container. Nitric acid will be used in the elution circuit for loaded carbon acid washing.

Sulphuric Acid

Concentrate sulphuric acid (98% H_2SO_4) will be delivered to the plant by truck in a 15 m^3 iso-container and will be stored in a single 30 m^3 carbon steel tank providing a total capacity of five days. Sulphuric acid will be used in the cyanide detoxification circuit.

Flocculant

Flocculant will be used in the tails thickener and will be supplied to the plant in 25 kg bags. The stock solution will be made-up in a vendor supplied mixing plant and then stored in a 12 m^3 storage tank. From here the flocculant will be pumped to the various consumers by dedicated flocculant supply pumps.

Lead Nitrate

Lead nitrate will be delivered to the plant by truck in a 15 m^3 iso-container. Lead nitrate will be used in the leach circuit.

17.2.5.9 Water Services

Raw Water Storage and Distribution

Raw water from the non-contact water pond will be filtered via sand filters and directed to the enclosed potable water tank to supply potable water to the plant, the raw water tank and the fire water tank.

Raw water will be used in the following areas:

- Water make-up to the process water pond;
- Reagent make-up;
- Elution circuit and gold room;
- Pump gland seal water;
- Potable water;
- Safety showers; and
- Fire water.

Raw water will be distributed by the raw water pumps.

Potable Water and Safety Shower Water Storage and Distribution

Raw water will be used to supply potable water to the process plant and used for safety shower and eye wash stations. Safety shower/eye wash water will be reticulated throughout the plant to supply the strategically located safety shower and eye wash stations. The safety shower/eye wash water will be distributed by the multi-staged electric safety shower pump. A diesel safety shower pump will be installed to supply safety shower water during an electrical power outage.

Potable water will be distributed by the potable water pumps via an ultraviolet sterilisation unit.

Gland Water Distribution

Raw water will be used as gland water. The gland water will be reticulated throughout the plant by the multi-staged gland water pumps.

Process Water Storage and Distribution

Process water (tails thickener overflow and decant return water) will be stored in the process water pond. Process water will predominately be used in the grinding circuit.

Fire Water Storage and Distribution

Fire water for the process plant will be drawn from the fire water tank. The fire water pumping system will contain:

- An electric jockey pump to maintain fire ring main pressure;
- An electric fire water delivery pump to supply fire water; and
- A diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump.

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

17.2.5.10 Cyanide Detoxification

Slurry from the last CIP tank, CIP tank 8, will gravitate to the vibrating carbon safety screen to recover any fine carbon passing through the intertank screens or overflowing tanks. Screen oversize will report to the fine carbon bin located at ground level. Screen underflow will gravitate to the carbon safety screen undersize hopper and will be pumped to the tails thickener by the variable speed CIP tails pumps.

Flocculant will be added to the thickener feed well to aid settling and improve overflow clarity. Thickened slurry at 55% solids w/w will gravitate to the tails hopper and then be pumped by the variable speed underflow pumps to the cyanide detoxification feed distribution box. The tails thickener overflow will gravitate to the process water pond.

The purpose of the cyanide detoxification circuit will be to achieve a WAD cyanide level in the plant tailings that complies with the International Cyanide Management Code (ICMC).

Various plant spillage, acid wash effluent and spent cold cyanide wash solution from the elution circuit are also added to the circuit via the cyanide detoxification feed distribution box. Cyanide detoxification will be achieved using the Caro's Acid process, which destroys WAD cyanide by oxidising the cyanide. The Cyanide detoxification circuit will consist of two mechanically agitated tanks installed in series.

The WAD cyanide level in the last tank will be monitored by the cyanide WAD analyser.

The detoxified slurry is then pumped to the TSF.

Detoxified slurry and other process plant effluents will be pumped to the TSF. Water recovered from the TSF will be pumped to the process water pond by the TSF water return pump.

17.2.5.11 Air Services

Oxygen

Oxygen for the leach circuit will be supplied from a vendor installed package. The oxygen will discharge to the oxygen receiver and then be distributed to the leach and CIP tanks.

Plant and Instrument Air

Plant and instrument air will be supplied to the plant by duty / standby air compressors. Plant air will be filtered and dried through an instrument air drier to produce instrument air with a low dew point.

Plant and instrument air will be stored in dedicated air receivers from where it will be distributed to the various plant areas.

Low Pressure Blower Air

Low pressure air will be supplied by duty / stand-by low pressure blowers. All low pressure air will be used by the airlifts for carbon transfer in the CIP Circuit.

17.2.6 Process Control

The process plant control system would be a programmable logic controller-based (PLC) system. The user will interact via standard personal computers running Citect SCADA software to provide control. The process facility will be controlled from the centrally located main control room in the plant area.

4-20 mA analogue I/O signals will predominantly be associated with the process instrumentation and control, including flow, pressure, density and the control of modulating valves and actuators, and variable speed drives.

Digital I/O will generally be based on 24 VDC hardwired signals, typically associated with the status and control of drives, valves and actuators and mechanical plant.

In each area the I/O associated with the MCC will be installed in one or more tiers of the MCC and will be hard wired to the starter modules within the MCC. The digital and analogue I/O associated with the process instrumentation will be wired to process control cubicles (PCCs).

Three visual display units (VDUs) will be installed within control room to provide operator interfaces. These units will present the operator with graphical process information in the form of trends, mimic pages, alarm summaries, logs and reports. This interface will also enable the operator to start and stop equipment, control variable speed drives and alter process set-points.

The adjustment of controller parameters will be made from the controller face plate and it will be possible to password protect this adjustment to prevent unauthorised adjustments. Display screens will be configured for the trending of individual or related parameters and a number of alarm pages will be developed to allow the setting of alarm points attached to various parameters. All analogue input signals including outputs from flow, pressure, temperature and weighing instruments will be displayed appropriately on mimic pages. A short-term trend plot for each input and output from the system can be provided where required on the mimic pages.

The analogue and digital I/O associated with the plant instrumentation will be cabled to one or more PCC within the plant areas. These units will be located within the area switch-rooms and will house the PLC racks, instrumentation power supplies and communication hardware. Communications at the interface between these units and control system will be via ethernet and will be by fibre optic or copper cable as appropriate.

17.3 Sulphide Process Design and Description

17.3.1 Introduction

The PFS16 flowsheet for treatment of the sulphide ore has been refined for PFS19, as shown in Figure 17.5. The flowsheet includes primary crushing, two stage grinding, separate flotation of talc / silicate minerals, copper, and zinc concentrates, regrind and concentrate thickening and filtration circuits.

The differences in the updated flowsheet from that proposed in PFS16 are:

- A single crushing stage with a semi-autogenous primary grinding mill (SAG – that used for the oxide treatment) followed by a secondary grinding ball mill to generate a flotation feed P_{80} size of 38 µm. The grinding circuit will include a pebble crusher to handle slow grinding, coarse material from the SAG mill and a sizing screen to control the transfer size to the ball mill, both operating in closed circuit with the SAG mill. The comminution circuit proposed in PFS16 incorporated three stages of crushing and two stages of ball milling for P_{80} size of 45 µm.
- A crushed ore storage bin has been included to minimise long-term storage of plant feed in 'dead' stockpile to minimise oxidation and aging effects in flotation.
- Stirred bead mills (IsaMills) have replaced the overflow ball mills in the regrind duties where the product sizes have changed from a P_{95} of 20 µm for the copper regrind to a P_{80} size of 15 µm and for the zinc regrind a P_{90} size of 20 µm has been changed to a P_{80} size of 20 µm.
- The cleaner stage for both copper and zinc has been revised to a cleaner and cleaner scavenger arrangement with the cleaner scavenger tailing open circuited to zinc flotation feed for the copper and final tail for the zinc cleaner scavenger tail.
- Treatment of process water using activated carbon has been included to reduce the residual reagent content of the recycled water and thereby prevent inadvertent recovery of copper and zinc into the pre-float circuit concentrate.

Figure 17.5 PFS19 Flowsheet for Sulphide Ore Processing

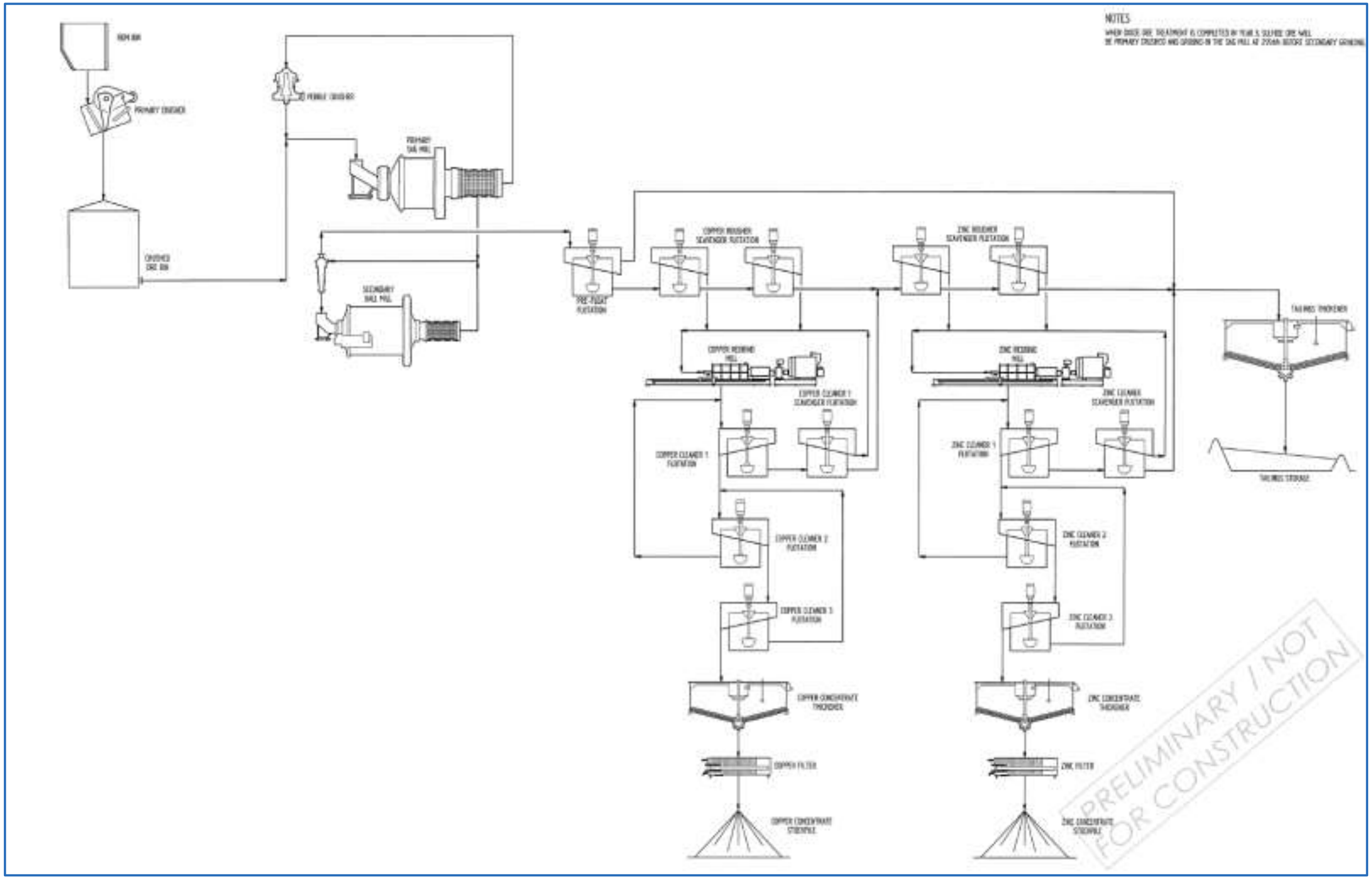

Figure from GRES, 2019.

17.3.2 Metallurgical Testwork Outcome Considerations

The process plant design has been based on the key parameters as outlined in Table 17.8.

Table 17.8 Sulphide Circuit – Design Parameters

Description	Units	Design Value	Comments
Plant Throughput	Mtpa	2.37	
Annual Operating Hours – Concentrator	h	8,059	
Daily Throughput	tpd	6,500	Project requirement
Milling Rate	tph	295	
Grind Product Size D_{80}	µm	38	Testwork assessment
Copper Regrind Size D_{80}	µm	15	Testwork assessment
Zinc Regrind Size D_{80}	µm	20	Testwork assessment
Feed Assay			
Copper	% Cu	0.8	Mine design value 15 May 2018
Zinc	% Zn	1.68	Mine design value 15 May 2018
Copper Concentrate			
Copper Grade	% Cu	30	Based on LCT median 28.7% Cu
Copper Recovery	%	70	Based on LCT median 58%
Transportable Moisture Limit	% moisture	13.3	Testwork Bureau Veritas
Zinc Concentrate			
Zinc Grade	% Zn	51.5	Based on LCT median 51% Zn
Zinc Recovery	%	81	Based on LCT median 82%
Transportable Moisture Limit	% moisture	13.2	Testwork Bureau Veritas

The metallurgical balance and flotation circuit equipment selection has been based on median values achieved in the locked cycled flotation testing. The concentrate production rate and grade used the maximum locked cycle performance as a check on the capacity of the equipment to handle the higher concentrate rates and the expected short-term maximum head grades from the mine.

The aspects identified in testwork that impact on the performance and design of the processing plant have been addressed in the following manner:

- Feed preparation – fine grinding to a particle size P_{80} of 38 µm was required to provide adequate liberation of the minerals for their separation in a sequential flotation circuit.

- Feed preparation – the different flotation behaviour of the three main lithologies requires control of the feed blend to avoid high levels of disseminated and enriched material specifically to limit the lead content of the feed to less than 0.25% Pb in disseminated material and the copper-to-zinc ratio in the enriched material to less than 0.75:1.
- Feed preparation – due to the propensity of the feed to oxidise with a detrimental impact on flotation performance, a maximum two to four week feed supply on the ROM pad has been targeted in the operating schedule. Blending fingers will be used to minimise fluctuating head grades (copper-to-zinc ratio, lead, pyrite).
- Pre-float – a pre-float circuit will remove a portion of silicate minerals, which are naturally floating to minimise silica levels in the copper (and zinc) concentrate.
- Pulp chemistry – to minimise loss of base metals into the pre-float concentrate due to inadvertent flotation from residual reagents in the recycled process water, the process water will be treated using activated carbon to remove these chemicals (and some metallic ions). Any effect of metal ions in tailing dam return water will be addressed by returning this water stream to the tailing thickener to use the residual high pH from the zinc circuit to raise the pH and precipitate metallic ions.
- Pulp chemistry – an anti-scalant will be dosed into the process water to minimise gypsum precipitation onto mineral particle surfaces, equipment surfaces and inside pipes. The sulphate levels in the site water have been measured at 2,000 ppm.
- Pulp chemistry – mild steel grinding media will be used in the milling circuit to create a reducing pulp redox potential in the flotation feed, which has been shown in testwork as necessary to effect the copper–zinc and chalcopyrite–pyrite separations.
- Regrind size reduction – fine grinding technology will be used in regrind applications to increase liberation with reduction in particle size to a D_{80} of 15 µm for the copper circuit and a D_{80} of 20 µm for the zinc circuit.
- Copper–zinc selectivity – in addition to the pulp redox potential, zinc sulphate will be dosed into the feed and copper cleaner circuit to depress sphalerite in the copper flotation stage.
- Copper–lead selectivity – the main contributor to lead reporting into copper concentrate is inclusions of fine galena within chalcopyrite and pyrite grains in the disseminated material. In addition to the blending strategy, sodium cyanide has been shown to reduce pyrite recovery in other complex sulphide operations and due to the inclusions of galena in the pyrite, reduced pyrite recovery is expected to manifest as reduced lead recovery and therefore addition of cyanide into the copper regrind and cleaning circuit has been included in the design.
- Pyrite selectivity – additions of SMBS and sodium sulphide will be used for depression of pyrite in the copper circuit. Lime will be used to adjust and maintain pH in the slurry at levels sufficient to depress the pyrite in the zinc circuit. Starvation levels of collector will also be used in the copper and zinc circuits to minimise inadvertent collection of the iron sulphides.
- Pyrite selectivity – cleaner circuits are designed for open circuit operation to avoid build-up of circulating loads of pyrite. Cleaner scavenger cells have been included to limit loss of the respective copper and zinc metal to cleaner tail.

17.3.3 Crushing

The crushing circuit used for oxide ore treatment will be used for treatment of the sulphide ore. The equipment has been designed for the 6,500 tpd capacity required for the sulphide ore based on a 75% utilisation of the crushing plant over a 24 hour period. The crushing plant will operate on a three, eight hour shift basis.

The crushing circuit was designed using the Metso Bruno simulation package and data base parameters due to the absence of ROM size distributions. Apart from the Mic and t_{10} values determined in the SMC tests, the only specific crushing breakage parameters (unconfined compressive strength, crushing work index, crushability index) measured were crushing work indices for a massive pyrite composite (3.2 kWh/t) and a disseminated sample (18.8 kWh/t). However, the lack of resistance to impact breakage indicated by the relatively high A*b values calculated from the SMC tests (median A*b of 90), the low Mic crushing energies (1.8 kWh/t to 4.6 kWh/t) and the relatively high t_{10} values suggest that the rock will break readily in crushing with low power consumption. The description of the ore also supports this assumption and that a high crushability index (> 45%) could be used in the Metso Bruno model for simulation of the crushing circuit.

Mine production delivered to the ROM pad will be stored in a number of separate stockpile fingers according to ore type and grade to facilitate blending of the feed to the crushing plant. Stockpiled material will be reclaimed by a front-end loader. The ROM bin has been designed to accept direct tipping of material if the feed blend and ore delivery schedule permits.

Crushed rock will be conveyed to a crushed ore storage bin of 4,200 t capacity that will provide 14 hours of milling. A door will provide access for a backhoe or small loader to remove rill material for additional short-term emergency feed or stockpiling when required. This door will also allow equipment access to empty the bin in the event of failure of the reclaim system and freezing or fusing of the crushed ore.

17.3.4 Grinding and Classification

The SMC parameters indicate the ore is of moderate hardness compared to other material in the JKMRC data base and support the low Bond work index values measured on all samples from the deposit – the highest ball mill work index was 11.9 kWh/t measured on a disseminated sample using a closing screen of 75 µm.

The 80th percentile values have been used for calculations to select the grinding power requirements.

The JKMRC equation for SAG mill feed size (F_{80} = 0.2 x crusher closed side setting (mm) x (DWi)0.7) indicated that a crushing plant product D_{80} of 80 mm would be produced at a crusher closed side setting of 100 mm.

A two stage, SAG and ball mill grinding circuit has been proposed to reduce the crushed material to a P_{80} of 38 µm for feed to flotation. The mill selection has been based on a JKSimMet simulation and an SMC specific energy calculation to determine the grinding power requirements and mill sizes. The SAG mill used for the oxide feed treatment (6.4 m diameter by 4.26 m EGL, powered by a VVVF 3,000 kW motor) will be supplemented with a ball mill, 6.1 m diameter by 7.3 m long powered by a 5,500 kW motor. A length-to-diameter ratio of less than 1.3 has been selected to minimise overgrinding (generation of fine particles less than 5 µm). A summary of results of the calculations is given in Table 17.9.

Table 17.9 Summary of Sulphide Grinding Circuit Design

Method	Units	Value (80th Percentile)	Design Value Disseminated)
Treatment Rate	tph	295	295
Feed Size F_{80}		80	90
Product Size P_{80}		38	38
SMC Rock Breakage Parameters			
A*b		84	65
DWi	kWh/m³	5.3	5.3
Specific Comminution Energy	kWh/t	7.9	8.6
SAG Mill			
Size (diameter x EGL)	m	6.4 x 4.26	6.4 x 4.26
Installed Motor Power	kW	3,000	3,000
Power Drawn	kW	2,120	2,190
Ball Mill			
Size (diameter x EGL)	m	6.1 x 7.3	6.1 x 7.3
Installed Motor Power	kW	5,500	5,500
Power Drawn	kW	4,300	4,790
Circuit Specific Energy	kWh/t	21.8	23.7

The JKSimMet simulations highlighted the importance of minimising the top size in the feed to the ball mill. Consequently, to optimise grinding efficiency, a vibrating screen has been installed to classify the SAG mill product to return material coarser than 0.75 mm to the SAG mill and provide a feed size to the ball mill circuit with a D_{80} of 0.25 mm. Therefore, the SAG mill will operate in closed circuit with a pebble crusher, a short head cone crusher to reduce the size of coarse material discharged from the SAG mill, and a screen 3.0 m wide by 8.5 m long.

The ball mill will operate in closed circuit with a cluster of 250 mm diameter cyclones. The cyclones have been designed with a circulating load of 300% (expected to be lower) and will have an overflow density of 30% solids to achieve the flotation feed size D_{80} of 38 µm. An operating pressure of 110 kPa will be required. The cyclone overflow will report to a horizontal, vibrating trash screen 2.4 m wide by 4.8 m long. The trash screen will be fitted with polyurethane screen panels having an aperture of 1.0 mm. Cyclone underflow will be directed to the ball mill feed chute.

Water will be added to the SAG mill feed chute, classifying screen feed and ball mill cyclone feed hopper to attain desired densities. The classifying screen feed pumps and cyclone feed pumps will operate in a duty and standby configuration and each pump will be equipped with a variable speed drive.

Oversize trash will gravitate directly to a trash bin. Undersize product from the trash screen will gravitate to the pre-float feed box. A launder sampler will be located on the trash screen underflow and the sample will be pumped to the on-stream analyser (OSA) for elemental and density analysis.

Two sump pumps will be located in the grinding circuit to return spillage back to the process.

Ball charging to the mills will be facilitated by use of a kibble arrangement. Balls will discharge into the ball mill feed chute via an impingement box.

17.3.4.1 Concentrate Regrinding

The specific grinding energy required for the concentrate regrind duties was determined by generating signature plots in tests conducted to grind rougher concentrate samples using bead mills. A regrind P_{80} size of 15 µm was used in the flotation testwork for copper rougher concentrate and a P_{80} size of 20 µm for zinc rougher concentrate.

The continuous test results have been used as the basis for design and IsaMills selected for the regrind duty (Table 17.10).

Table 17.10 Summary of Data from Regrinding Tests

Description	Feed Size F_{80} (µm)	Product Size P_{80} (µm)	Media Size (mm)	Media Type	Specific Energy	
					Copper kWh/t	Zinc kWh/t
SMD Test	38.3	14.7	3.0	Kings CM270	17.7	
SMD Test	45.9	22.5	3.0	Kings CM270		11.5
Netzsch Continuous Test	27.7	15	3.0	Magotteaux	31.2	
Netzsch Continuous Test	26.5	20	3.0	Magotteaux		20.7
Design					31.2	20.7

17.3.5 Flotation

17.3.5.1 Introduction

The copper and zinc minerals will be recovered sequentially in the flotation circuit (Table 17.11), which will comprise a pre-float stage to remove naturally floating silicates, a copper roughing / scavenging and three stage cleaning circuit with regrinding of the copper rougher / scavenger concentrate, and a similarly configured zinc flotation circuit. The first cleaning stage in each circuit will include a cleaner scavenger bank to allow open circuit operation of the cleaning stages – copper cleaner scavenger tailing will report to the zinc rougher feed and zinc cleaner scavenger tailing will report to final tailing.

The flowsheet reflects the locked cycle testing procedure. Flotation times, reagent additions and stream assays used in the locked cycle tests have formed the basis for sizing and selection of equipment. Flotation times have been scaled-up using standard factors of two to three times the bench-scale tests for middlings / scavenger duties – a one to one ratio has been used for the fast-floating rougher flotation component. For cleaning duties, froth surface area and typical froth carry rates (tonnes of mineral per unit area of cell surface) have been used – the cleaner flotation times will therefore have scale-up factors of over four relative to the laboratory times where scraping is used to assist removal of froth (high carry rate compared to plant equipment).

Table 17.11 Summary of Flotation Circuit Design

Description/Stage	Units	Copper Circuit	Zinc Circuit
Pre-Float			
Feed Rate	tph	295	
Feed Density	% solids	30	
Flotation Time	minutes	8.4	
Scale-up Factor (from laboratory)		1.7	
Number and Size of Cells		3 off 40 m^3	
Rougher / Scavenger			
Feed Rate	tph	292	287
Feed Density	% solids	29	25
Flotation Time	minutes	17	19.3
Scale-up Factor (from laboratory)		1.9	1.8
Number and Size of Cells		6 off 40 m^3	8 off 40 m^3
Froth Carry Rate (max. head grade)	t/m^2h	0.8 (0.9)	0.5 (0.8)
Cleaner 1			
Feed Rate		60	44
Feed Density		20	16
Flotation Time	minutes	17	13.2
Scale-up Factor (from laboratory)		4.3	4.4
Number and Size of Cells		4 x 20 m^3	3 x 20 m^3
Froth Carry Rate (max head grade)	t/m^2h	1.1 (1.2)	0.8 (1.2)

Description/Stage	Units	Copper Circuit	Zinc Circuit
Cleaner 1 Scavenger			
Feed Rate	tph	40	27
Feed Density	% solids	16	11
Flotation Time	minutes	15	14
Scale-up Factor (from laboratory)		7.3	4.8
Number and Size of Cells		3 x 20 m³	3 x 20 m³
Froth Carry Rate (max head grade)	t/m²h	0.8 (0.8)	0.4 (0.5)
Cleaner 2			
Feed Rate	tph	29	22
Feed Density	% solids	25	20
Flotation Time	minutes	16	18
Scale-up Factor (from laboratory)		5.2	6.2
Number and Size of Cells		5 x 5 m³	6 x 5 m³
Froth Carry Rate (max head grade)	t/m²h	1.8 (1.8)	1.0 (1.3)
Cleaner 3			
Feed Rate	tph	14	12.6
Feed Density	% solids	25	26
Flotation Time	minutes	18	23
Scale-up Factor (from laboratory)		9	23
Number and Size of Cells		3 x 5 m³	3 x 5 m³
Froth Carry Rate (max head grade)	t/m²h	1.1 (1.7)	(1.5)

The depressants sodium sulphide, sodium metabisulphite, zinc sulphate, and when required, sodium silicate, are added into the grinding circuit to enable access to surfaces exposed during the breakage and attrition of particles.

Tank cells have been selected for the flotation duties due to:

- The ability to install froth crowding, which enables operation using deeper froths in scavenger and low mass pull duties leading to improved control of froth depth;
- The minimisation and equalisation of froth carry distances to the concentrate launders;
- The even air dispersion compared to ‘square’ cells.

Alternative cell types for cleaning duties such as column, Jameson or Woodgrove cells, can be considered in detailed design to reduce plant footprint size.

Cell surface area values have been taken from standard vendor specifications – the differences for cells of the same volumetric capacity are achieved using different launder and crowding arrangements. The effective flotation cell volume has been determined by assuming 15% is air or froth – a lower value of 10% has been used for scavenger cells.

The launders will be provided with water sprays to assist movement of the concentrate, disrupt the froth bubbles and provide dilution of the concentrate prior to the subsequent flotation stage.

Dart plug valves have been chosen for pulp level control in the cells based on the flexibility to cater for larger pulp flow variations than pinch valves. Each discharge will be fitted with two darts operating in a 'master-slave' manner. Low pressure air will be added down the agitator shaft in each cell and controlled using individual automatic control valves coupled with air flowmeters.

17.3.5.2 Pre-Float

Trash screen undersize gravitates to the feed box of the pre-float flotation circuit that comprises three, forced air tank cells each with a volume of 40 m³. Frother (MIBC) is the only reagent added in the pre-float stage to remove silicate gangue minerals and where minimising loss of copper, zinc, and precious metals into the pre-float concentrate is essential as this stream reports to final tail. The pre-float tailings will discharge by dart valve arrangement and subsequently be pumped to the copper feed conditioning tank.

When not required to remove silicates (low non-sulphide gangue feed blends), the pre-float cells will be used as aerating conditioning tanks prior to copper flotation.

17.3.5.3 Copper Flotation

The copper rougher / scavenger circuit will be configured as two rougher 40 m³ cells discharging via dart plug valves into the four 40 m³ scavenger cells installed in two, two cell arrangements complete with discharge dart valves.

The copper minerals will be recovered from the pyrite, sphalerite, galena and remaining non-sulphide gangue at natural pH pulp conditions throughout the copper circuit. Sodium aerofloat will be the main copper sulphide collector. Collector will be added in a stage-wise manner to avoid excess levels, which will result in the inadvertent flotation of the pyrite and sphalerite.

Depressants sodium sulphide, sodium metabisulphite and zinc sulphate will be used to minimise the flotation of the other minerals. Depressants for the rougher / scavenger stage will be added in the grinding circuit. Sodium silicate will be added as a dispersant for the non-sulphide gangue when required. The depressants will also be added into the regrind mill to maximise the rejection of pyrite and sphalerite through the copper cleaner scavenger tailing. Sodium cyanide will also be added in the regrind circuit to improve depression of pyrite (and hence lead) in the copper cleaning stage.

Rougher / scavenger concentrate at 5% Cu to 10% Cu will be pumped to 150 mm diameter dewatering cyclones to enable control of the feed density from the cyclone underflow to the regrind mill. The flow rate to the IsaMill will be controlled to a set flow rate, which will be maintained by recirculating a portion of the discharge via 'chunk' valve.

The regrind feed cyclone overflow will be combined with the regrind mill discharge and pumped to the 10 m³ copper cleaner feed conditioning tank. The four 20 m³ copper cleaner cells (two cells – dart valves – two cells – dart valves configuration) will precede three 20 m³ cleaner scavenger cells (one cell – dart valves – two cells – dart valves configuration). The concentrate from the cleaner cells will be pumped to the copper cleaner 2 cells for further upgrading while the concentrate from the cleaner scavenger cells will be recirculated to the copper regrind circuit. The tailing from the cleaner scavenger cells will be open-circuited by pumping via the OSA and a flowmeter (combined data will calculate the mass loss of copper for flotation control purposes) to the zinc flotation feed conditioning tanks.

Five 5 m³ tank cells (configured as one cell, two cells, two cells separated by dart valves) will act as the copper cleaner 2 stage with the tailing discharge flowing directly into the first cell of the copper cleaner bank. The cleaner 2 concentrate will be pumped to the three 5 m³ copper cleaner 3 cells (configured as one cell then two cells separated by dart valves). The cleaner 3 tailing will pass via dart valves into the first copper cleaner 2 cell. Concentrate from the cleaner 3 cells at 28% Cu to 30% Cu will form the final copper concentrate that will be pumped via the OSA to the copper concentrate thickener feed hopper.

Provision has been made for the future installation of a flotation column, which can act as a fourth stage of copper cleaning if required when treating high-lead or low-copper feed blends.

17.3.5.4 Zinc Flotation

The zinc circuit flowsheet is the same as the copper circuit flowsheet. Tailing from the copper rougher / scavenger cells and the cleaner scavenger cells will be conditioned in two stages – lime will be dosed into the first, agitated conditioning tank to increase the pulp pH to 11.5 followed by addition of copper sulphate into the feed of the second 40 m³ conditioning tank. The zinc rougher / scavenger circuit will be configured as two rougher 40m³ cells (one cell – discharge dart valves – one cell) discharging via dart plug valves into the six 40 m³ scavenger cells installed in three, two cell arrangements complete with discharge dart valves. The number of cells could be reduced by selecting larger cells, however, 40 m³ cells have been selected to minimise the different cell sizes in the plant and number of spare parts.

The zinc mineral sphalerite will be recovered from the mainly pyrite gangue at high pulp pH conditions throughout the zinc circuit using lime dosing provided at each stage. High pH conditions depress the flotation of pyrite. Copper sulphate will be dosed into the pulp to activate the hydrophilic sphalerite and reverse the depression required in the copper circuit. Under these conditions, the sphalerite flotation rate was shown to be high in the testwork.

Sodium iso-propyl xanthate will be the collector used. It will be added in a stage-wise manner to avoid excess levels, which will result in the inadvertent flotation of the pyrite and residual copper and lead minerals into the zinc concentrate, and to minimise residual (excess) xanthate in the process water, which if returned to the pre-float circuit would increase loss of copper.

Rougher / scavenger concentrate at 12% Zn to 25% Zn will be pumped to 150 mm diameter dewatering cyclones to enable control of the feed density from the cyclone underflow to the regrind mill. The flow rate to the M5000 IsaMill, identical in size to the copper regrind mill, will be controlled to a set flow rate that will be maintained by recirculating a portion of the discharge via 'chunk' valve.

The regrind feed cyclone overflow will be combined with the regrind mill discharge and pumped to the two 12 m³ zinc cleaner feed conditioning tanks where lime and copper sulphate will be added respectively. The three 20 m³ zinc cleaner cells (one cell – dart valves – two cells – dart valves configuration) will precede three 20 m³ cleaner scavenger cells (one cell – dart valves – two cells – dart valves configuration). The concentrate from the cleaner cells will be pumped to the zinc cleaner 2 cells for further upgrading while the concentrate from the cleaner scavenger cells will be recirculated to the zinc regrind circuit. The tailing from the cleaner scavenger cells will be open-circuited by pumping via the OSA and a flowmeter (combined data will calculate the mass loss of zinc for flotation control purposes) to the final flotation tail hopper.

Six 5 m³ tank cells (configured as two cells, two cells, two cells separated by dart valves) will act as the cleaner 2 stage with the tailing discharge flowing directly into the first cell of the zinc cleaner bank. The cleaner 2 concentrate will be pumped to the three 5 m³ zinc cleaner 3 cells (configured as one cell then two cells separated by dart valves). The cleaner 3 tailing will pass via dart valves into the first zinc cleaner 2 cell. Concentrate from the cleaner 3 cells at 51% Zn to 56% Zn will be final zinc concentrate, which will be pumped via the OSA to the zinc concentrate thickener feed hopper.

The number of zinc cleaner cells also could be reduced with selection of alternative sizes but an increase in spares will result.

On Stream Analysis System

A twelve-stream Courier analyser system will be installed in the flotation circuit to monitor the performance of the flotation circuits. Slurry samples from nominated streams will be directed to the Courier for multi-element analysis. Analytical results from the Courier will be displayed and recorded on a monitor in the plant control room and shift composite sub-samples will be collected for metallurgical accounting purposes.

The following streams will be measured on line by the Courier:

- Flotation feed;
- Pre-float concentrate;
- Copper rougher feed;
- Copper rougher / scavenger concentrate;
- Copper scavenger tail;
- Copper final concentrate;
- Copper cleaner 1 tail;
- Zinc rougher / scavenger concentrate;
- Zinc scavenger tail;

- Zinc final concentrate;
- Zinc first cleaner tail; and
- Flotation final tail.

A particle size analyser Outotec PSI 500 will also be installed to measure the particle size distribution of the flotation feed, copper regrind product, and zinc regrind product streams.

17.3.6 Concentrate Thickening and Filtration

Frothbuster technology has been included in the design to minimise build-up of froth on the surface of the concentrate thickeners. Final copper concentrate, filtrate from the copper concentrate filter and spillage from the copper concentrate areas will be pumped to the copper concentrate Frothbuster, which will operate at 100 kPa to de-aerate the pulp prior to discharge into the feedwell of the copper concentrate thickener.

The copper concentrate thickener has been sized using the standard industry solids settling flux rate of 0.25 t/m² despite testwork results showing much higher rates were achieved. A 7 m diameter high-rate concentrate thickener fitted with an auto-dilution feed system has been selected, which has the capacity to handle the concentrate tonnage produced from a high feed grade of 1.5% Cu. Flocculant will be mixed into the feed slurry to increase the solids settling rate and maintain clear thickener overflow water. The copper concentrate slurry will be thickened to 60% solids (w/w) and will then be pumped to a 150 m³ agitated concentrate storage tank by one of two peristaltic type pumps in a duty / stand-by configuration. The copper concentrate thickener overflow will gravitate to the combined thickener overflow tank from where it will be pumped to the process water treatment circuit

The copper concentrate thickener area will be provided with a sump pump to aid clean up and will pump to the copper Frothbuster feed hopper.

The copper concentrate storage tank will have capacity to store up to 23 hours of copper concentrate production. A single duty filter feed pump will pump the copper concentrate to a plate and frame type pressure filter, which has been selected due to the fineness of the concentrate. A narrow chamber depth has been used for sizing based on the testwork results to realise the target concentrate moisture content of 12%. A filter fitted with 1.5 m by 1.5 m plates providing 29 chambers has been selected for the copper concentrate duty. The filter will operate automatically with control effected using a dedicated vendor supplied PLC and operator interface system linked to the plant control system. The filtrate will be collected in the Frothbuster feed hopper via a blowdown vessel.

The filter cake will be discharged onto the floor of the concentrate storage shed from where it will be loaded into containers on trucks while parked on a weighbridge. The trucks will then transport the concentrate to the port. Containers will be unloaded or stored on a hard stand at the port awaiting shipment in bulk carriers.

The zinc concentrate dewatering circuit will be identical to that of the copper circuit. The zinc concentrate thickener will also be 7 m diameter. The 150 m³ capacity thickened filter feed storage tank however will have a nominal 18 hour capacity down to 10 hours when treating head grades of 2.6% Zn.

The size of the zinc filter plates and chamber depth has been chosen to be the same as that of the copper filter. Thirty-nine chambers will be required in the zinc concentrate filter.

17.3.7 Tailings Disposal

The combined flotation tailing (pre-float concentrate, zinc rougher / scavenger tailing and zinc cleaner scavenger tailing will be pumped to the 23 m diameter high-rate thickener, which was used for oxide CIP tailing. The tailing thickener has been sized for the design sulphide tailing tonnage calculated by the mass balance, using the results of testwork (Patterson & Cooke, 2018), which gave a 0.75 t/m^2h settling flux – an additional 15% safety factor has been added to allow for periods of higher tailing tonnages.

Flocculant will be added to the thickener feed line and feed well to assist settling suspended solids. The thickener will be equipped with a bed level device to measure bed level and provide a process variable signal for the flocculant addition control.

A bed pressure sensor will be fitted to the thickener base to measure the bed pressure. There will be two underflow lines that will discharge into the tailings thickener underflow hopper. Underflow density will be controlled to 65% solids by varying the valve opening of the duty underflow line. The thickened underflow pulp will then be pumped to the tailings storage facility.

Thickener overflow will report to water treatment circuit. An area sump pump will return any spillage back to the process.

The tailings storage facility (TSF) will be a valley filled land form with tailings discharged via several point discharges along the dam wall. Supernatant water recovered from the tailings storage facility will be pumped from the dam to the to the process water dam in the plant.

17.3.8 Water Treatment

A multimedia filtration plant will treat 24,000 m^3/d of process water using powder activated carbon to remove residual flotation reagents and other organics from the process water.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The project is a greenfield site and consists of an open pit mine with a process plant and ancillary facilities situated 38 km away from Bigadiç. Some infrastructure that has been used for exploration activities exists on site; however, existing infrastructure is limited and could be considered for low level pioneering activities. The facilities that have been installed on site to date include:

- Accommodation complex that includes messing facilities – not commissioned yet;
- Main site office (decommissioned primary school);
- Core cutting and storage / logging areas;
- Power supply and distribution;
- Stores; and
- First aid station.

The proposed mining area is accessible by an existing paved road. The proposed plant area is accessible only by a gravel road.

The infrastructure that will adjacent to the orebody will be limited to:

- A water control weir and diversion pipelines;
- A mine servicing area, mine production offices, first aid station, mine worker ablutions (including sewage treatment), and fuel facility;
- The mine waste dump and contact water catchment dam;
- Gatehouse with a first aid clinic;
- Site access road;
- Access roads linking the TSF, and administrative areas with the process plant;
- The ROM pad;
- The processing plant;
- Concentrate storage;
- Accommodation village and associated catering facilities;
- Plant maintenance workshops;
- Warehouse;
- Administration building;
- Laboratory;
- Reagents storage building;
- Main fuel farm;

- Communications;
- Site change house;
- Security facilities and infrastructure;
- Water treatment system;
- Sewage treatment facilities;
- Tailings storage facility;
- Water treatment plant;
- Water control weirs; and
- Mines rescue / firefighting building and equipment.

Due to the project's close proximity to Bigadiç and surrounding towns the infrastructure to support the operations, including power supply and site access, is readily available.

A general arrangement of the site is included in Figure 18.1.

18.2 Workshop and Stores

The Plant workshop will be contained within a single pre-engineered clad, steel framed building, located adjacent to the eastern side of the Plant Warehouse. The building will be 36 m long by 12 m wide, complete with internal walls, 4 m long front awnings, offices, three 5 m height x 4 m wide double sliding doors complete with personnel access doors. The workshop will incorporate electrical, mechanical and welding bays, and each bay will have its own sliding door and personal access door. High bay lighting will be included with skylight roof sheeting, roof and wall vents. The floors will be concrete, and each doorway will include a 6 m wide concrete apron.

The Plant warehouse will be contained within a single pre-engineered clad, steel framed building, 36 m long by 12 m wide including eave roof, complete with internal walls, 4 m overhang awnings, two office, kitchen, single toilet, store racking, tool store, two 5 m height x 6 m wide chain wheel operated roller doors and two 5 m height x 4m wide double sliding doors and four personnel access doors. The warehouse incorporates a tool store with heavy duty shelving, an open area for non-waterproof and non-sunproof large equipment, pallet racking and office area under a mezzanine floor. The floor will be concrete, and each doorway will include a 6 m wide concrete apron.

A fenced compound area 72 m long x 12 m wide will be installed at the rear of the warehouse and workshop with 6 m wide swing gate at the entry and exit to enable secure storage of large bulk items and unloading of semi-trailers.

18.3 Main Change Room

A change room complex, to service the site, will be established to the east of the Administration Building as the first building of the non-process infrastructure compound. The change room building will be 21 m long x 14 m wide will be contained in a pre-fabricated clad, steel framed building, complete with non-slip vinyl floorcovering.

The building shall consist of male and female change room structures separated by a 3 m wide covered breezeway. The change room lockers are allocated for employees working in a job classified as 'dirty' and the number of lockers, shower and ablution cubicles are designed to accommodate approximately 180 personnel across the entire operation. This change room is designed based on the local employees who drive in and out each day and need to change prior to and after each shift. The shower cubicles are only provided for the special occasions for the employees with extra cleaning during the shift. There will be sufficient vanity bars for both male and female change area. A 3 m x 3 m cleaners room will be located in the corner of the female change area.

A pedestrian foot path alongside the light vehicle access road will be connected to the front gate house to enable employees to park their cars in the front gate and walk to the change room to get ready to work and vice versa back home.

18.4 Security Gatehouse and Control Points

The security gatehouse will be located where the main access road into the plant passes through the perimeter fence system. There will be an automatic sliding gate at this location for all vehicles and trucks in and out and of the plant site. A swipe card system will operate for both vehicles and personnel with continuous monitoring by a security guard. The security gatehouse building will consist of a gatehouse, security office, community office, induction room, and ablution blocks. Security guards and officers will utilise the gatehouse and security office to manage the site security system with CCTV and issue site access swipe cards. The induction room will utilised be for the new starters and temporary contractors to conduct inductions prior to accessing site. A carpark for private vehicles will be located near the main gate house with access via a pedestrian foot path to get to the change room and administration building. The ablution block will include a small change area with lockers for the security guards.

18.5 Administration Buildings

The office complex to service the site will be a two storey pre-fabricated building located between the change room and dry mess. The administration building will be approximately 800 m^2 each storey. The office complex will accommodate approximately 100 management personnel including administration, human resources, health safety and environment, payroll, procurement, and processing. Each workstation will be provided with a desk and chair, electrical, data and communication outlets. There will be a small kitchenette, large meeting room and toilets at each building level. The single offices are dedicated to the senior management, while the joint offices and open area are for the general employees.

The administration office complex will be installed during plant construction and will be separate from the mining contractor's temporary facilities, used for process plant, mine access construction and permanent mine office.

Figure 18.1 Site General Arrangement

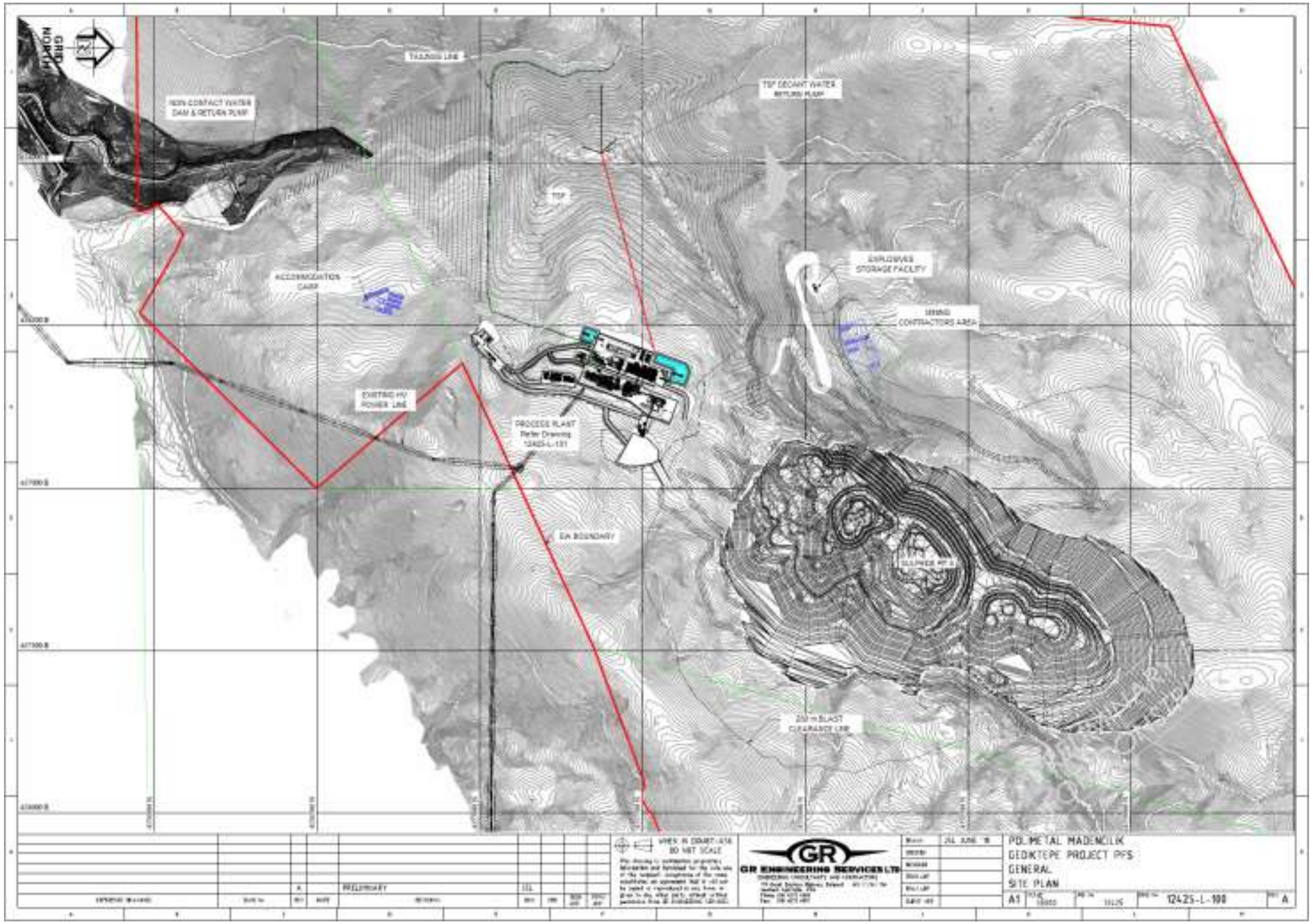

Figure from GRES, 2019.

18.6 Operations Accommodation Camp

Based on Alacer experience at the Çöpler site, an allowance has been made for a 500-person accommodation camp proximal to the mine site. Amongst other benefits, it will ensure continuity of operations when road access is difficult due to weather conditions.

The proposed camp is a mix of single and married accommodation and includes a kitchen / dining area plus associated social and recreational facilities. Cost estimates are based on recent Çöpler mine rates provided by Alacer.

It is recommended that Polimetal undertake traffic and accommodation surveys and trade-off studies to determine if there is suitable local accommodation in the area or if it is practical to transport personnel from the regional centre of Balıkesir.

18.7 Kitchen and Dry Mess

The messing facilities will be a pre-fabricated building located between the administration building and prayer room. The building will consist of kitchen and dining area to accommodate all the processing plant personnel. The kitchen will comprise a cooking area, food storage including fridge and freezer area, food preparation and cleaning area, kitchen office and crib room for kitchen staff. There will be a truck parking bay and double door at the back of the kitchen to accommodate the bulk food delivery. The restaurant facilities will provide the seats and meals production as outlined in Table 18.1.

Table 18.1 Restaurant Capacity Summary

Description	Normal Operation	Final Allowance
Seats Available	120	168
Cooking capacity/shift	200	300

The cooking facilities will be required to produce meals on a two-shift basis, seven days per week.

18.8 Prayer Room and Ablution Building

The prayer room and ablution building will be a pre-fabricated building located between the kitchen and emergency response team (ERT) buildings. The prayer room will be 12 m x 6 m with two entries and exits; these two entry and exits are to allow separate male and female access into and from the room. A screen will be installed in the prayer room to separate the male and female users. There will be sufficient ablutions rooms in the breezeway between the prayer room and ablution blocks. There will be two double entry and exit doors on both side of the breezeway. A 1.5 m-long awning will be overhung over the ablution block and a 3 m x 3 m cleaners store room will be in the corner of the ablution block.

18.9 Emergency Response Team Building and Induction Room

The ERT building and induction room will be a pre-fabricated building located next to the prayer room as the last building of the NPI complex. Being the last building will provide sufficient space for the ambulance and emergency response vehicle. The building will consist of treatment room, medical store room, data room, disabled bathroom unit, office, safety and medical area, and induction room. There will be a double door with ramp and canopy at the treatment room to enable the paramedic to push a patient in and out of the area. The induction room will also be used for the employee induction, training or assembly purposes.

18.10 Laboratory

The Laboratory will be a pre-engineered building. The building will divide into a wet area and dry area. There will be concrete floor and floor drain for the wet area, and a roller door to accommodate the sample and equipment transportation. The dry area will include balance room, thermogravimetric analyser (TGA) room, fusion room, XRF room, office with small kitchenette, and a bathroom. A breezeway will be located between the dry and wet area with all double doors for internal and external access. The laboratory will be located next to the workshop.

18.11 Main Control Room

The main control room will be a pre-fabricated building located at the north side of the grinding building. The main control room will be the centralised control hub from crushing circuit to both the oxide and sulphide circuits. There will be two main access doors for the building with eight main control stations to cover each part of processing plants, and there will be another four smaller control stations in the 12 m x 12 m main control area. The main control building consists of a 6 m x 3 m server room to store all the critical communication equipment inside with full time air conditioning, small office, kitchenette, and toilets.

18.12 Fuel storage

The light vehicle diesel fuel storage will be located adjacent to the workshop and stores area. A single 50,000L double-skin tank will be installed with single diesel unloading and single refuelling bowser, which will be placed on level compacted drained ground alleviating the need for concrete bunding and slab areas. This refuelling station will only be for the processing plant light vehicles.

19 MARKET STUDIES AND CONTRACTS

The Gediktepe project is currently planned to produce the following products:

- Gold and silver doré from the cyanide leaching of the oxide resource
- Copper flotation concentrates
- Zinc flotation concentrates

The metallurgical testing to date indicates that the gold / silver doré will be of marketable quality, as will the zinc concentrate. The copper concentrate, however, will be sold under two different qualities: a 'standard' grade concentrate typically > 20% Cu, < 7% Zn, and < 2.5% Pb, and a 'complex' grade concentrate with higher penalty elements typically > 20% Cu, < 10 %Zn, < 6% Pb. The complex concentrate should not exceed 50% of the annual production.

Modelling and metallurgical testing have shown that mercury and arsenic levels in concentrates will generally be below smelter rejection levels.

No specific contracts for delivery of doré or concentrates have been finalised at this time. However, Polimetal, through its marketing consultants, have contacted a number of smelters and trading organisations and have obtained estimated product shipment and treatment charges that have been used in the financial modelling of the project.

Standard grade copper concentrates and zinc concentrates should be the subject of letters of intent (LOI's). The complex grade copper concentrates are similar to products sold by other Turkish producers, such as Cayeli, but may have to be sold on the spot market.

The specific smelting and refining terms used in the financial modelling are described in the following.

Gold and Silver Doré Metal

The product from the cyanide leaching will be a high-silver doré product typically assaying 6% Au and 90% Ag:

- Gold 99% payable + $5.133/payable oz for refining and freight
- Silver 98% payable + $1.602/payable oz for refining and freight

Copper Concentrate – Standard

- Minimum 50% of annual production of copper concentrate
- Copper pay for lesser of 96% or 1 unit deduction
- Gold pay for lesser of 90% or 1 g/t deduction from Au content
- Silver pay for lesser of 90% or 30 g/t deduction from Ag content
- Lead penalty $4.50 for each 1% above 0.5% (rejection above 2.5% Pb)
- Zinc penalty $1.50 for each 1% above 3% (rejection above 7.0% Zn)

- Arsenic penalty $2.50 for each 0.1% above 0.2% (rejection above 0.5% As)
- Treatment charge $90/dmt/concentrate
- Refining charge copper $0.09/lb payable Cu
- Refining charge gold $5.00/payable oz Au
- Refining charge silver $0.50/payable oz Ag

Copper Concentrate – Complex

- Maximum 50% of annual production of copper concentrate
- Copper pay for lesser of 96% or 1 unit deduction
- Gold pay for lesser of 90% or 1 g/t deduction from Au content
- Silver pay for lesser of 90% or 30 g/t deduction from Ag content
- Lead penalty $4.50 for each 1% above 0.5% (rejection above 6.0% Pb)
- Zinc penalty $1.50 for each 1% above 3% (rejection above 10.0% Zn)
- Arsenic penalty $2.50 for each 0.1% above 0.2% (rejection above 0.5% As)
- Treatment charge $95/dmt/concentrate
- Refining charge copper $0.09/lb payable Cu
- Refining charge gold $5.00/payable oz Au
- Refining charge silver $0.50/payable oz Ag

Zinc Concentrate

- Minimum 49% Zn, max 5% Cu.
- Zinc pay for lesser of 85% or 8 unit deduction
- Gold pay for 65% after 1 g/t deduction from Au content
- Silver pay for 65% after 93.31 g/t deduct from Ag content
- Copper penalty: none, subject to rejection above 5.0% Cu
- Lead penalty $1.50 for each 1% above 3.%, (rejection 5% Pb)
- Arsenic penalty $1.50 for each 0.1% above 0.2% (rejection 0.5% As)
- Treatment charge $296/dmt concentrate

19.1 Freight and Port Charges

Gold / silver doré bars will be shipped to refiners, most likely in Europe. An allowance of 1.3 cents/oz payable gold and 2 cents/oz payable silver has been included in the refining charges.

A port study (Polimetal, 2017) was reviewed by Lydia and its consultant (CLK Logistics) and forms the basis for the costs associated with delivering Gediktepe flotation concentrates to third party smelters.

The study team visited three of the ports in Figure 19.1 and recommended the selection of Aliaga on the Mediterranean coast and Gemlik on the Sea of Mamara as being the most suitable options for trucking to, storing, and loading Gediktepe concentrates for shipment to third party smelters.

The Gemlik port is located 269 km from the project and roughly four hours' drive, mainly on paved roads. (Figure 19.2). Glencore and Trafigura currently use the Gemlik port for exporting lead and zinc concentrates to Europe and China.

The Aliaga port region is located 234 km from the project and roughly four hours' drive, mainly on paved roads (Figure 19.3). This port has several separately owned facilities for storing and loading concentrates and is currently used by several mining companies. Concentrates can be handled either in bulk or bagged and shipped out in containers.

A flow diagram showing the alternative transport and freight costs is shown in Figure 19.4.

Figure 19.1 Location of Gediktepe Project Site and Gemlik, Bandirma, and Aliaga Ports

Figure from Polimetal, 2019.

Figure 19.2 Road Access Gediktepe to Gemlik Port Site

Figure from Polimetal, 2019.

Figure 19.3 Road Access Gediktepe to Aliaga Port Site

Figure from Polimetal, 2019.

Figure 19.4 Alternative Transport and Freight Costs

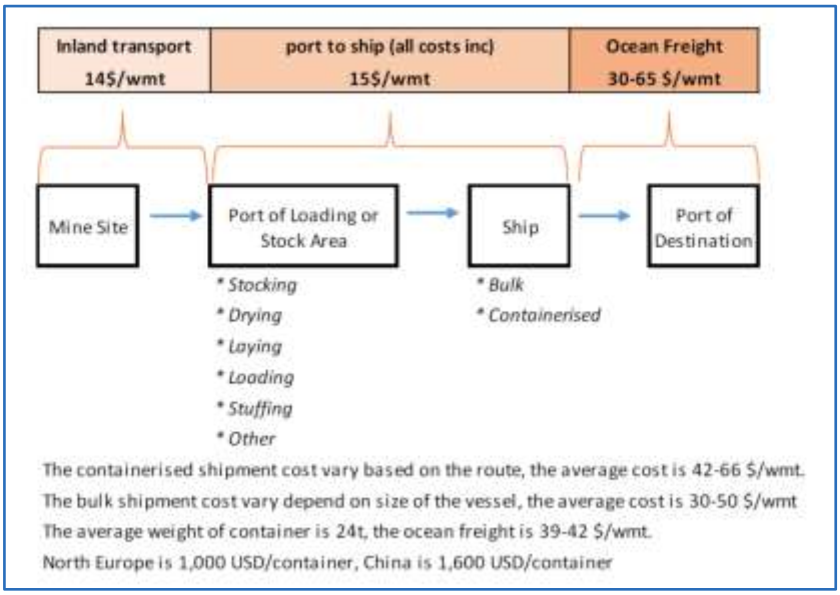

Figure from Polimetal, 2019.

For the financial model, it was assumed that copper and zinc concentrates containing 12% moisture will be transported in bulk to smelters in Europe with the following charges:

- Sea freight $30/wmt
- Port charges $13.50/wmt
- Inland transport $12/wmt
- Open warehousing $1.25/wmt
- Material handling $4/wmt
- Insurance $0.06/wmt
- Custom clearance $1.0/wmt
- Inspection $3.29/wmt
- Insurance 0.15% of CIF

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Baseline Studies

20.1.1 Baseline Water Quality

Environmental baseline studies were started at the project site on 15 September 2013 with Topçuoğlu Madencilik San. ve Tic. Ltd. Şti. (Topçuoğlu). For water quality purposes, samples from five developed stand pipes (fountains) and three creeks were taken and measured for temperature, pH, EC & TDS values. Also, samples were sent to ALS (Prague), and were also analysed for soluble metal, total soluble metal, cyanide content, and major ion concentrations. Additionally, Piper & Schoeller diagrams were drawn and assessed.

After Topçuoğlu completed the initial environmental baseline studies, Golder Associates (Türkiye) Ltd. Şti. (Golder) carried out further baseline studies on site during December 2013, March 2014, and June 2014. In that work, Golder completed the following site specific studies and desktop studies:

- Selected the location of the meteorological station (MS) at project site. After selecting the installation location, approval from Turkish State Meteorological Service (TSMS) was obtained and construction started in October 2014.
- Water quality sampling and evaluation has been done. Five water reservoirs, 22 fountains, 11 water springs, and 10 surface water locations have been identified.
- Water samples were analysed for temperature, pH, electrical conductivity, salinity, total soluble solids, oxidation–reduction potential, soluble oxygen, and flow amounts.

20.1.2 Acid Rock Drainage

A geochemical characterisation programme was implemented to assess the environmental stability of ore and waste rock (WR) in terms of its acid rock drainage and metal leaching potential. This test programme selected representative samples from exploration drill core and included the following components:

- Mineralogical analysis
- Whole rock analysis
- Acid–base accounting (ABA)
- Net acid generating (NAG) test
- Short-time leaching (STL) test
- NAG leach test

Major findings of the mineralogical analyses were as follows:

- A large component of the samples (from 10% to 66%) consists of quartz, which is considered environmentally inert.
- Eight out of 12 samples were found to contain a carbonate (calcite) concentration of 0.5% to 29.8%.
- One sample was found to contain a dolomite concentration of 5.9%.
- The main sulphur mineral is pyrite. Nine samples were found to contain a pyrite concentration of 0.1% to 0.8%, one sample was found to contain a marcasite concentration of 2.3%. The massive pyrite sample contains 85.5% pyrite.
- One sample was found to contain a hematite concentration of 37.7%, and one sample was found to contain an ankerite concentration of 5.2%.
- Five samples were found to contain a magnetite concentration of 1.4% to 4.7%.

The results of the geochemical characterisation are summarised in Table 20.1 from Golder's environmental baseline study report.

Assessment of the relative proportions of rock types in the PFS19 mine plan indicates that the planned mine waste is collectively PAG. The PFS16 waste was assessed as being overall NPAG – the different outcome is due to different proportions of rock types in the PFS19 pit as a result of the changes in the geological resource and waste cell model and the larger in-pit volume.

The PFS19 waste management strategy is to deposit sulphide-bearing PAG waste rock in a separate facility so the main waste dump will remain NPAG. Marble and dacite rocks that were tested for sulphide content are available in the EIA boundary and will be used to neutralise the PAG waste rock placed in a separate dump. Details of this design will be defined in the waste rock management plan, which will be completed before waste rock mining begins.

20.1.3 Flora and Fauna

Additional information regarding the site environmental conditions include:

- Land usage, protection zones, and archaeological status were assessed.
- Fauna and flora studies were performed in May 2014 and October 2014 by Prof. Dr. Hayri Duman (Gazi University, Science Faculty, Biology Dept.) and Doç. Dr. Zafer Ayaş (Hacettepe University, Science Faculty, Biology Dept.).
- A socio-economic assessment was done as a desktop study.

After Golder completed its site-specific studies, SRK Danışmanlık ve Mühendislik A.Ş. (SRK) was selected to carry on environmental baseline studies and completed the Environmental Impact Assessment (EIA) report according to Turkish Environmental Regulations.

Table 20.1 Geochemical Characterisation Results

Sample Name	Lithological Code	Alteration Code	Type	NNP	NPR	NAG	General Assessment
DGS-1	CL SER SC	OX	WR	UNCERTAIN	PAG	NON-PAG	NON-PAG
DGS-2	CL SER SC	OX	WR	NON-PAG	NON-PAG	NON-PAG	NON-PAG
DGS-3	CL SER SC	SUL	WR	UNCERTAIN	NON-PAG	NON-PAG	NON-PAG
DGS-4	CL SER SC	SUL	WR	NON-PAG	NON-PAG	NON-PAG	NON-PAG
DGS-5	CL SER SC	SUL	WR	UNCERTAIN	NON-PAG	NON-PAG	NON-PAG
DGS-6	CL SER SC	SUL	WR	NON-PAG	NON-PAG	NON-PAG	NON-PAG
DGS-7	CL SER SC	SUL	WR	NON-PAG	NON-PAG	NON-PAG	NON-PAG
DGS-8	CL SER SC	SUL	WR	PAG	PAG	PAG	PAG
DGS-9	FAULT ZONE	OX	Ore + WR	UNCERTAIN	UNCERTAIN	NON-PAG	NON-PAG
DGS-10	FAULT ZONE	SUL	WR	NON-PAG	NON-PAG	NON-PAG	NON-PAG
DGS-11	FAULT ZONE	SUL	WR	PAG	PAG	PAG	PAG
DGS-12	FE SCH	OX	Ore	UNCERTAIN	PAG	NON-PAG	UNCERTAIN
DGS-13	FE SCH	OX	WR	UNCERTAIN	PAG	PAG	PAG
DGS-14	FE SCH	OX	Ore	UNCERTAIN	PAG	PAG	PAG
DGS-15	GOSSAN	OX	Ore	PAG	PAG	NON-PAG	PAG
DGS-16	GOSSAN	OX	Ore	PAG	PAG	NON-PAG	PAG
DGS-17	GOSSAN	OX	WR	UNCERTAIN	PAG	NON-PAG	PAG
DGS-18	MASSIVE PY MAG ZONE	SUL	Ore	PAG	PAG	PAG	PAG
DGS-19	MASSIVE PY MAG ZONE	SUL	Ore	PAG	PAG	PAG	PAG
DGS-20	MASSIVE PY MAG ZONE	SUL	Ore	PAG	PAG	PAG	PAG
DGS-21	PY CL SER SC	SUL	WR	PAG	PAG	PAG	PAG
DGS-22	PY CL SER SC	SUL	WR	PAG	PAG	PAG	PAG
DGS-23	PY CL SER SC	SUL	WR	PAG	PAG	PAG	PAG
DGS-24	QF CL SC	OX	WR	UNCERTAIN	NON-PAG	NON-PAG	NON-PAG
DGS-25	QF CL SC	OX	Ore	UNCERTAIN	PAG	NON-PAG	UNCERTAIN
DGS-26	QF CL SC	SUL	WR	PAG	PAG	PAG	PAG
DGS-27	QSC	SUL	WR	UNCERTAIN	NON-PAG	NON-PAG	NON-PAG
DGS-28	QSC	SUL	WR	UNCERTAIN	PAG	PAG	PAG
DGS-29	QSC	SUL	WR	UNCERTAIN	PAG	PAG	PAG
DGS-30	QSC	SUL	WR	PAG	PAG	PAG	PAG

Notes: WR = Waste rock, PAG = Potentially acid generating, NNP = Net neutralisation potential, NPR = Neutralisation potential ratio, NAG = Non-acid generating

20.2 Environmental Impact Assessment Studies and Reporting

The EIA addresses the specific requirements of the Turkish regulatory system. The EIA seeks public feedback, but formal stakeholder engagement is limited. It is a proscriptive process that requires meeting specified numerical standards.

To support the project final feasibility study, an Environmental and Social Impact Assessment (ESIA) will be performed that meets the minimum Turkish standards but also meets International guidelines. The ESIA process is more risk based and places more emphasis on social issues.

The following EIA studies were performed;

- Meteorological data of Dursunbey MS was compiled,
- Fauna and flora studies were performed in April, June, and July 2015 by Prof. Dr. Hayri Duman (Gazi University, Science Faculty, Biology Dept.) and Doç. Dr. Zafer Ayaş (Hacettepe University, Science Faculty, Biology Dept.).
- A Hydrobiology study was performed in November 2015 by Prof. Dr. Aydin Akbulut (Hacettepe University, Science Faculty, Biology Dept.).

The EIA boundary was defined based on the PFS16 mine plan and facilities layout, and all the EIA studies focussed within the red line boundary on Figure 20.1.

20.2.1 Flora, Fauna, and Hydrobiology Studies

Nineteen endemic flora species (one local, six regional, and 12 widespread) were identified during the flora studies, and seeds of those species collected by Prof. Dr. Hayri Duman. They were sent to the Turkey Seed Gene Bank of Ministry of Food, Agriculture and Livestock, Ankara on 11 November 2015.

During fauna studies, photo-traps and Sherman traps were used and no endemic fauna species were identified in the project area.

A hydrobiology study also determined that no aquatic life was identified within the project area at the time of the study.

20.2.2 Protected Areas

There are no protection areas in close proximity to the project area.

Figure 20.1 Map of Environmental Impact Assessment (EIA) Boundary

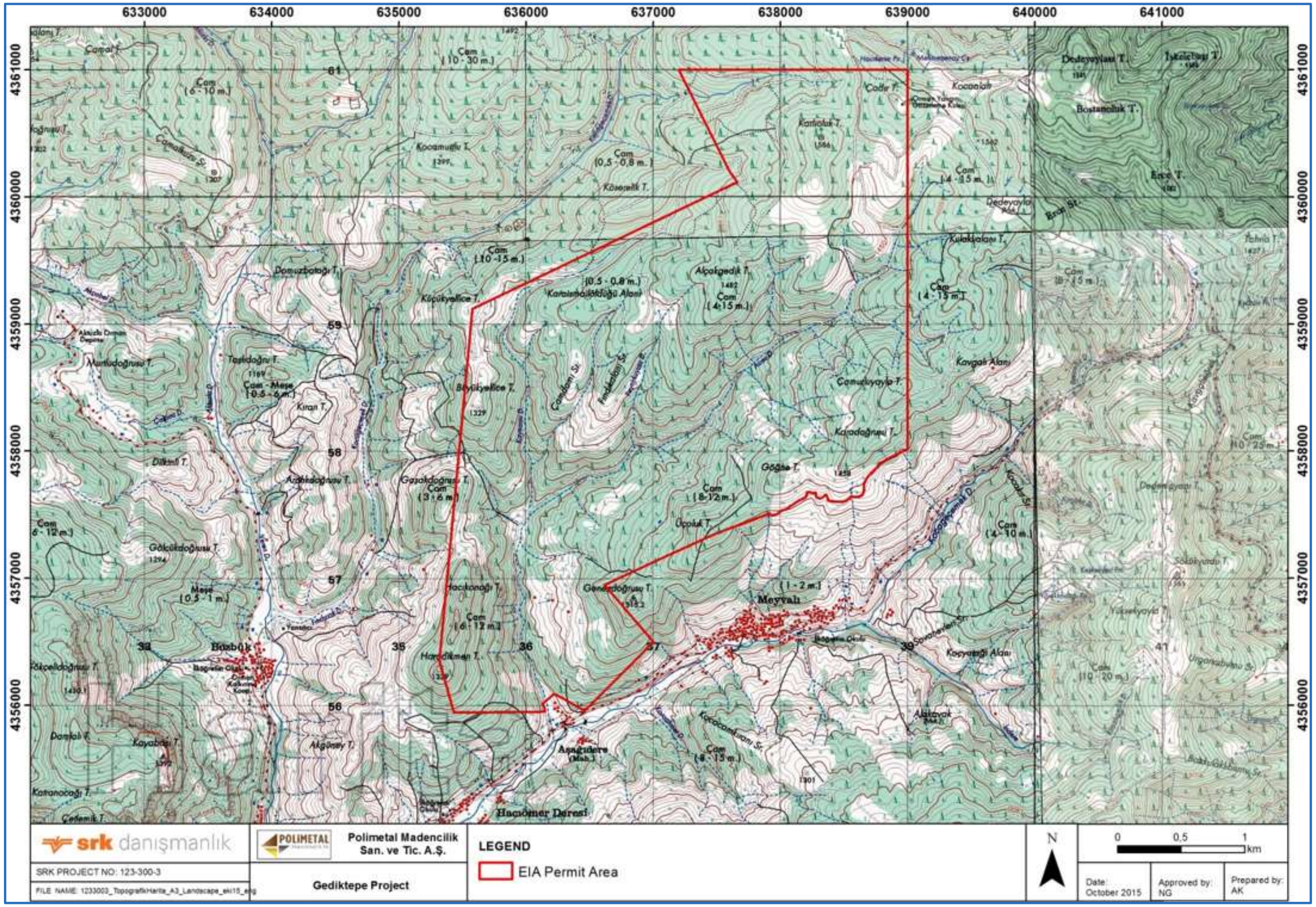

Figure from Polimetal, 2019.

On August 11, 2015, a public participation meeting was held, as a part of legal requirement of Turkish EIA Regulation, to inform locals and the public about the planned mining operation. Approximately 120 individuals participated in the meeting.

During the meeting, local people stated that they supported the project and requested that Polimetal address the following items:

- create local job opportunities,
- provide high quality water to the local villages, and
- construct a by-pass road around the Hacıömerderesi Village.

All of the above requests were addressed in the EIA report. New village water supply pipelines will be constructed from the water sources that are outside of the EIA boundary. Engineering and construction of the village by-pass road has been completed.

20.2.3 Hydrology and Hydrogeology Studies

The regional and project area hydrology and characterised catchment basins were studied. Figure 20.2 shows the drainage catchment basins that were defined.

Figure 20.2 Gediktepe Drainage Catchment Basins

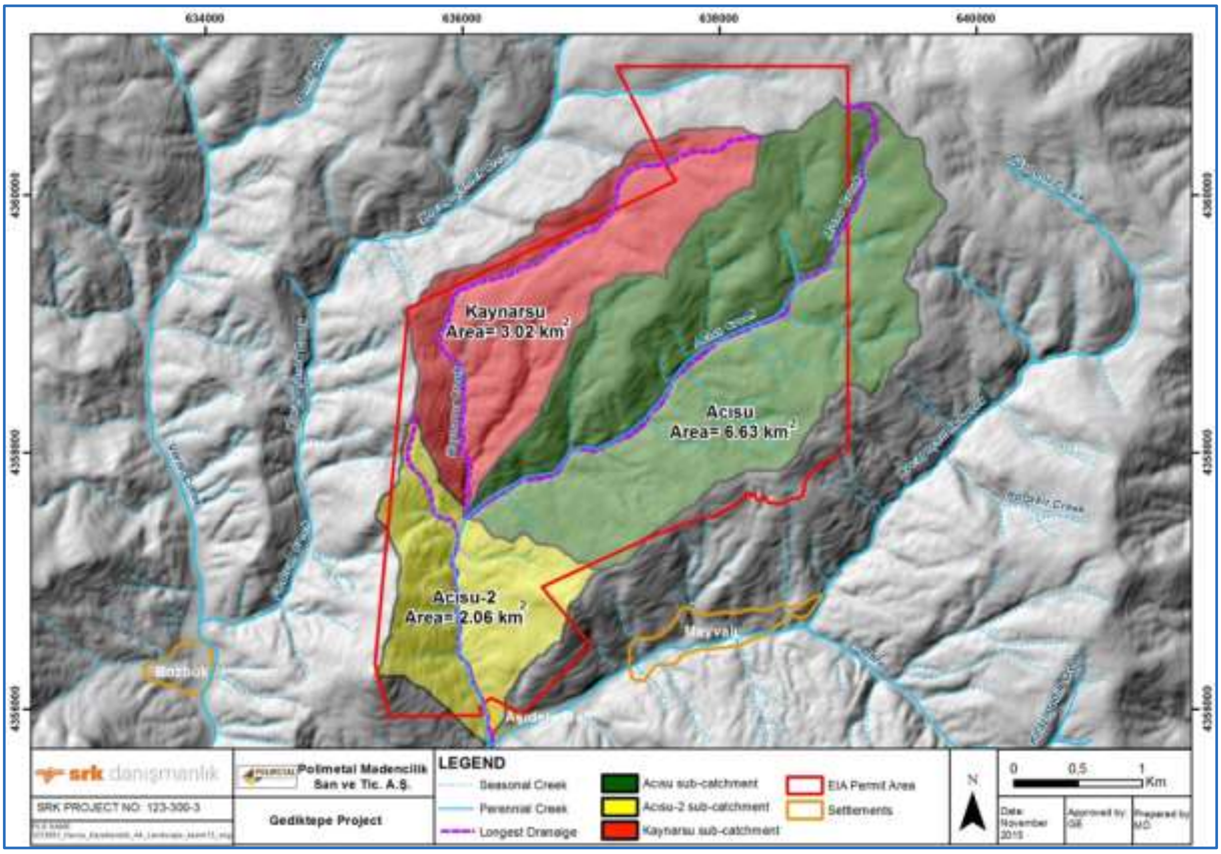

Figure from Polimetal, 2019.

Two weirs were constructed: 1) between the pit and waste dump area, and 2) at the flume location of the TSF to measure flow rates. Flow rates from these weirs were used for Hydrograph analysis and conceptual water balance was calculated and completed. Also, hydrochemical properties and quality of surface water resources were measured and determined. Figure 20.3 shows the locations of the weirs.

Figure 20.3 Gediktepe Locations of Weirs for Flow Rate Measurements

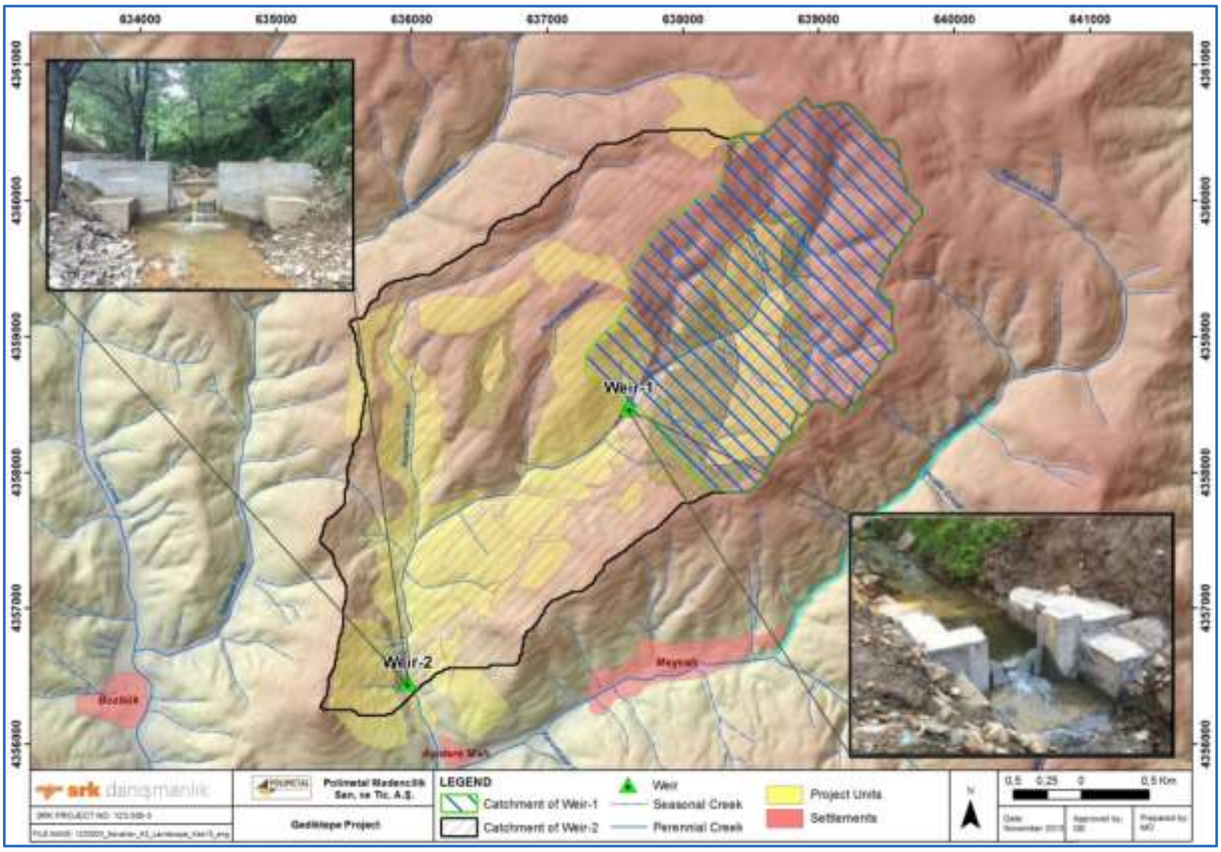

Figure from Polimetal, 2019.

Figure 20.4 shows the locations and the summarised test results from the 13 wells. Figure 20.4 shows the static water levels within the wells over time.

A three-dimensional calibrated underground water flow model was established based on the aquifer test results and static water levels. A stylised illustration of the flow model is presented in Figure 20.5.

In order to provide a constant water supply to the mine, inclusive of the dry season, a water dam with 690,000 m^3 capacity has been designed at the south side of the TSF.

Table 20.2 Water Observation Wells Locations and Summary Test Results

Hole	Easting (m)	Northing (m)	RL (m)	Drilled Depth (m)	Casing Depth (m)	Drilling Diameter (in.)	Flow Amount (l/s)	Drawdown (m)	Pumping Time (hr)	Test Type	Hydraulic Conductivity (m/s)
GTMW-01A	636,698	4,357,838	1,127	112	109	14 (0–32 m); 10 (32–112 m)	0.23	28.57	72	Pump	4.2E-08
GTMW-01B	636,709	4,357,842	1,128	32	30	7					
GTMW-02A	637,592	4,358,509	1,239	182	170	10	0.22	66.32	72	Pump	8.2E-09
GTMW-03	636,943	4,359,026	1,396	330	107	10 (0–104 m); 7 (104–330 m)					
GTMW-03B	636,940	4,359,020	1,399	325	270	10 (0–104 m); 7.5 (104–325 m)	2.04	43.04	72	Pump	1.60E-07
GTMW-04A	637,526	4,357,895	1,292	56	39	10	1.51	16.92	20.5	Pump	1.30E-06
GTMW-04B	637,515	4,357,909	1,290	206.5	126.5	10 (0–59 m); 4.7 (59–206.5 m)	0.1	117	4	Airlift	5.3E-08
GTMW-06	636,399	4,358,312	1,263	122	121	10	0.4	48	2	Build Up	5.4E-08
GTMW-08A	636,809	4,357,213	1,233	70	61.5	10	0.62	26.35	38	Pump	1.4E-07
GTMW-09	636,099	4,356,108	985	92	92	10	0.45	24.99	26	Pump	7.8E-08
GTMW-11	635,960	4,359,176	1,278	116	96	10	12.2	26	39	Pump	1.7E-06
GTMW-14	638,980	4,359,748	1,443	86	84	10	4.33	63.53	72	Pump	6.7E-07
GTMW-15	638,508	4,358,198	1,446	88	88	10	0.26	39.96	15	Pump	1.3E-07
W1							25	41.7	7.5	Pump	3.4E-06

Figure 20.4 Static Water Levels in Wells March – December 2015

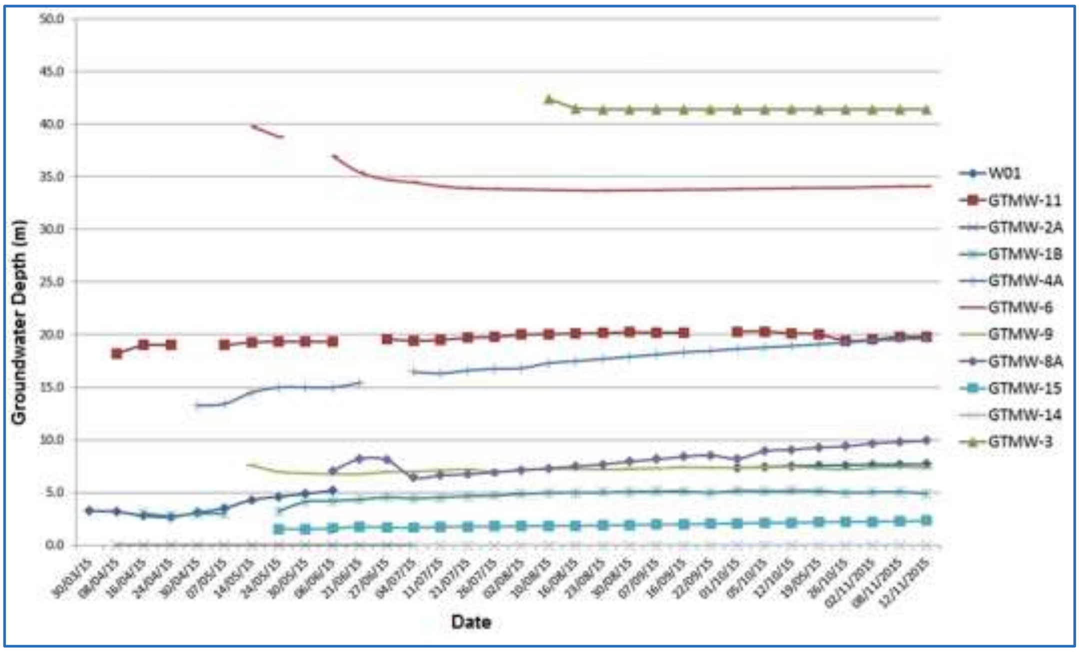

Figure from Polimetal, 2018

Figure 20.5 Illustration of Groundwater Flow Model

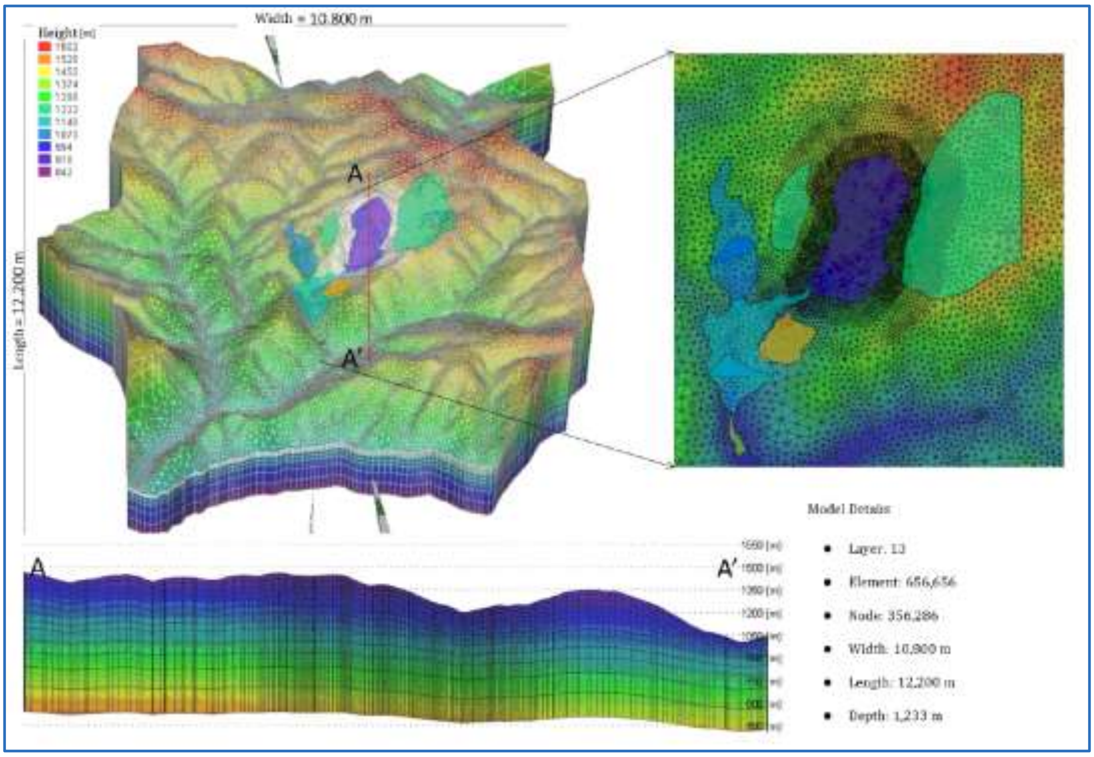

Figure from Polimetal, 2018

20.2.4 Acid Rock Drainage and Metal Leaching

A gap analyses for acid rock drainage (ARD) and metal leaching was undertaken, resulting in the selection of new samples for testing. All samples were selected from diamond drill core; locations shown in long-section in Figure 20.6. These samples were sent to SGS Canada for static testing and kinetic testing. Additionally, rock samples, water samples, soil and sediment samples were also collected.

The analysis focussed on selecting samples to represent all ore and waste lithologies in the mine. Table 20.3 is a list of the 55 static test samples, which are comprised of: 12 gossan samples, 25 chlorite–sericite schist samples, five quartz–feldspar schist samples, seven quartz schist samples, three fault zone samples, and three samples of massive pyrite.

Approximately 36% of the waste rock in the PFS19 pit is quartz–feldspar schist and chlorite–sericite schist. Total sulphide amount of these rocks is lower than 0.1%, which is accepted as inert waste rock according to Turkish regulation.

Marble and dacite rocks that were tested for sulphide content are available in the EIA boundary and will be used to neutralise the PAG waste rock placed in a separate dump. Details of this design will be defined in the waste rock management plan, which will be completed before waste rock mining begins.

Table 20.3 ARD / ML Analysis Sample List

Static Test Sample No.	Kinetic Test Sample No.	Sample Type	Lithology/ Mineralogy Zone	Weath. Zone	ABA	Total Rock	Static NAG	NAG Sol^n Analysis	Leach-ate Analysis	Kinetic Test	XRD	BHID	Start (m)	Finish (m)
GT-Ore-1	HC 1	Ore	Gossan	Ox	X	X	X	X	X	X		DRD-033	3.0	5.0
GT-Ore-2					X	X			X			DRD-020	42.0	43.6
DGS-9		Waste+Ore	Gossan	Ox	X	X	X		X			DRD-019	9.1	15.0
DGS-12		Ore	Gossan	Ox	X	X	X		X		X	DRD-001	31.9	37.5
DGS-13		Waste	Gossan	Ox	X	X	X	X	X			DRD-012	52.0	56.0
DGS-14		Ore	Gossan	Ox	X	X	X	X	X			DRD-062	1.8	5.0
DGS-15					X	X	X		X			DRD-013	12.4	17.4
DGS-16					X	X	X		X		X	DRD-015	23.8	30.0
DGS-17		Waste	Gossan	Ox	X	X	X		X		X	DRD-008	5.5	9.5
GT-WR-11					X	X	X		X			DRD-023	4.0	8.0
GT-WR-21					X	X	X		X			DRD-005	2.0	6.0
GT-WR-22					X	X	X	X	X		X	DRD-015	0.0	3.8
GT-WR-2	HC 3	Waste	Chl–Ser Schist	Ox/Sul	X	X	X	X	X	X	X	DRD-116	5.7	7.8
GT-WR-3		Waste	Chl–Ser Schist	Ox	X	X	X		X			DRD-006	17.5	21.0
DGS-1					X	X	X		X		X	DRD-048	8.3	13.7
GT-WR-7	HC 4	Waste	Chl–Ser Schist	Sulph	X	X	X	X	X	X		DRD-022	111.0	115.0
DGS-6	DGS-6				X	X	X	X	X	X	X	DRD-069	29.2	35.0
GT-WR-10	HC 6				X	X	X	X	X	X		DRD-122	24.0	28.0

Static Test Sample No.	Kinetic Test Sample No.	Sample Type	Lithology/ Mineralogy Zone	Weath. Zone	ABA	Total Rock	Static NAG	NAG Sol^n Analysis	Leach-ate Analysis	Kinetic Test	XRD	BHID	Start (m)	Finish (m)
GT-Ore-3	HC 2	Ore	Chl–Ser Schist	Sulph	X	X	X	X	X	X	X	DRD-012	95.0	98.5
DGS-22	DGS-22	Waste	Chl–Ser Schist	Sulph	X	X	X	X	X	X	X	DRD-012	109.2	117.0
DGS-2					X	X	X		X			DRD-003	35.0	43.0
DGS-3					X	X	X		X			DRD-024	103.5	109.5
DGS-4					X	X	X		X			DRD-015	46.5	54.5
DGS-5					X	X	X		X			DRD-048	50.0	55.0
DGS-7					X	X	X		X			DRD-020	128.5	136.5
DGS-8					X	X	X	X	X		X	DRD-043	19.0	27.0
GT-WR-1					X	X	X		X			DRD-013	27.0	29.5
GT-WR-4					X	X						DRD-031	21.0	23.2
GT-WR-5					X	X						DRD-038	52.5	56.5
GT-WR-6					X	X						DRD-039	28.8	31.7
GT-WR-8					X	X						DRD-073	175.6	179.5
DGS-26					X	X	X	X	X		X	DRD-066	43.5	50.8
GT-WR-18					X	X	X	X	X			DRD-009	71.0	75.4
GT-WR-19					X	X						DRD-071	53.0	56.0
GT-WR-20					X	X						DRD-096	88.0	91.0
DGS-21					X	X	X	X	X			DRD-070	37.9	42.0
DGS-23					X	X	X	X	X			DRD-020	90.5	98.5

Static Test Sample No.	Kinetic Test Sample No.	Sample Type	Lithology/ Mineralogy Zone	Weath. Zone	ABA	Total Rock	Static NAG	NAG Sol^n Analysis	Leach-ate Analysis	Kinetic Test	XRD	BHID	Start (m)	Finish (m)
GT-WR-12	HC 5	Waste	Qtz–Felds Schist	Ox	X	X	X	X	X	X	X	DRD-016	26.5	29.5
DGS-24					X	X	X		X			DRD-012	2.0	10.0
GT-WR-13				-	X	X	X		X			DRD-027	22.0	27.5
GT-WR-9				Sulph	X	X	X		X			DRD-073	50.0	54.0
GT-WR-14					X	X	X	X	X			DRD-157	18.3	21.0
GT-WR-15	HC 7	Waste	Qtz Schist	Sulph	X	X	X	X	X	X		DRD-002	78.5	84.5
DGS-29	DGS-29	Waste	Qtz Schist	Sulph	X	X	X	X	X	X	X	DRD-002	68.5	76.5
GT-WR-16					X	X	X					DRD-116	65.0	67.0
GT-WR-17					X	X	X	X	X			DRD-040	49.0	55.0
DGS-27					X	X	X		X		X	DRD-004	148.5	153.5
DGS-28					X	X	X	X	X			DRD-014	30.4	38.4
DGS-30					X	X	X	X	X			DRD-002A	34.0	40.0
DGS-10		Waste	Fault	Sulph	X	X	X		X			DRD-017	50.0	55.5
DGS-11					X	X	X	X	X		X	DRD-022	195.0	203.0
DGS-25		Ore	Fault	Ox	X	X			X			DRD-004	31.0	34.0
DGS-18		Ore	MPy	Sulph	X	X	X	X	X		X	DRD-043	43.0	49.0
DGS-19					X	X	X	X	X			DRD-062	6.8	11.6
DGS-20					X	X	X	X	X			DRD-012	83.0	88.0

Figure 20.6 Long-Section showing ARD Sample Locations (looking west)

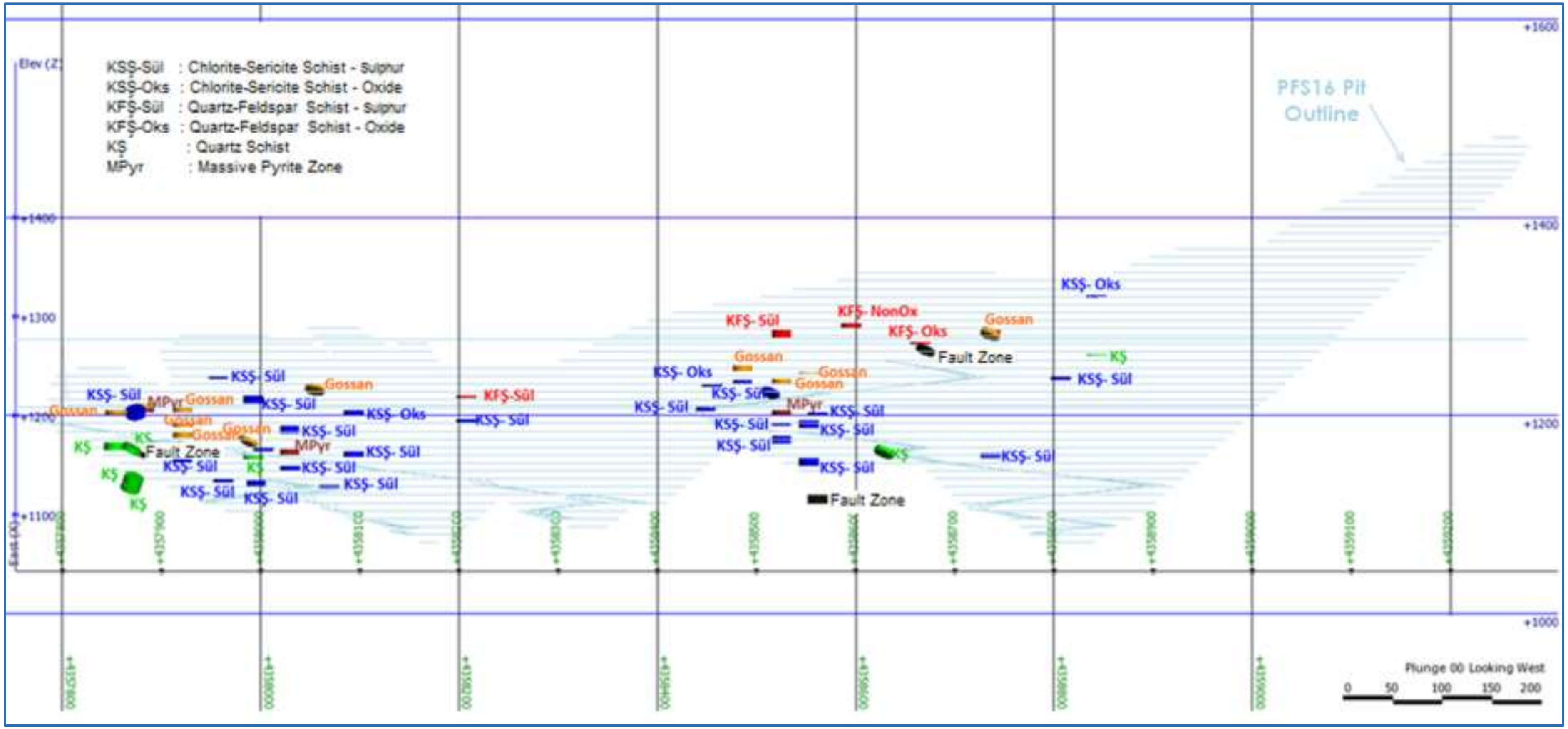

Figure from Polimetal, 2019.

Table 20.4 shows ta list of the kinetic test sample tests. For the EIA report, the 38-week results of kinetic tests were utilised. Polimetal continued kinetic testing on samples HC2, HC4, HC7, DGS22, and DGS29 to see the net acid potential and soluble metal in these samples after longer periods of time.

Table 20.4 List of Kinetic Sample Tests

Kinetic Test Sample No.	Static Test Sample No.	Sample Type	Lithology/ Mineralogy Zone	Weath Zone	BHID	Start (m)	Finish (m)	Test Duration (weeks)
HC 1	GT-Ore-1	Ore	Gossan	Ox	DRD-033	3	5	38
HC 2	GT-Ore-3	Ore	Chl–Ser Schist	Sulph	DRD-012	95	98,5	> 40
HC 3	GT-WR-2	Waste	Chl–Ser Schist	Ox/Sul	DRD-116	5,7	7,8	38
HC 4	GT-WR-7	Waste	Chl–Ser Schist	Sulph	DRD-022	111	115	> 40
HC 5	GT-WR-12	Waste	Qtz–Felds Schist	Ox	DRD-016	26,5	29,5	38
HC 6	GT-WR-10	Waste	Chl–Ser Schist	Sulph	DRD-122	24	28	38
HC 7	GT-WR-15	Waste	Qtz Schist	Sulph	DRD-002	78,5	84,5	> 40
DGS-6	DGS-6	Waste	Chl–Ser Schist	Sulph	DRD-069	29,2	35	38
DGS-22	DGS-22	Waste	Chl–Ser Schist	Sulph	DRD-012	109,2	117	> 40
DGS-29	DGS-29	Waste	Qtz Schist	Sulph	DRD-002	68,5	76,5	> 40

The results of kinetic tests at week 38 are summarised in Figure 20.7 through Figure 20.11, which show pH, EC, SO_4 concentration, and total Ficklin metals concentration.

Based on the static and kinetic test results, it was concluded that:

- HC4 sample (sulphide zone chlorite–sericite schist), HC7, and DGS29 samples (quartz schist) were identified as potentially acid generating,
- HC1 (gossan), HC3 (oxide zone chlorite–sericite schist), HC5 (oxide zone quartz feldspar schist), HC6 (sulphide zone chlorite–sericite schist), and DGS6 (sulphide zone chlorite–sericite schist) samples were identified as not acid generating.

Figure 20.7 Leachate Weekly pH Change

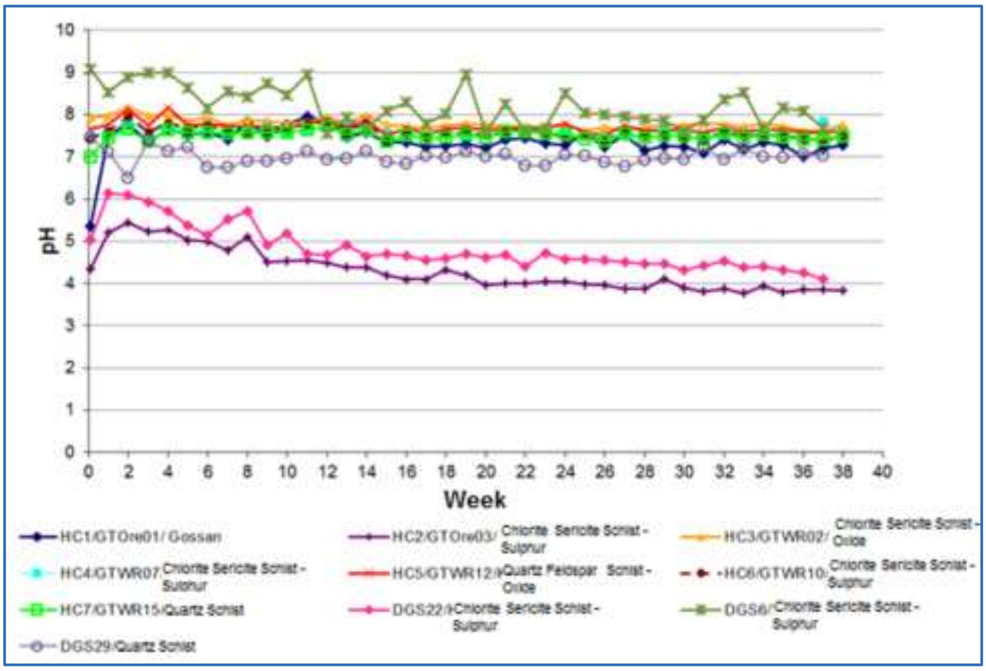

Figure from Polimetal, 2018

Figure 20.8 Leachate Weekly EC Change

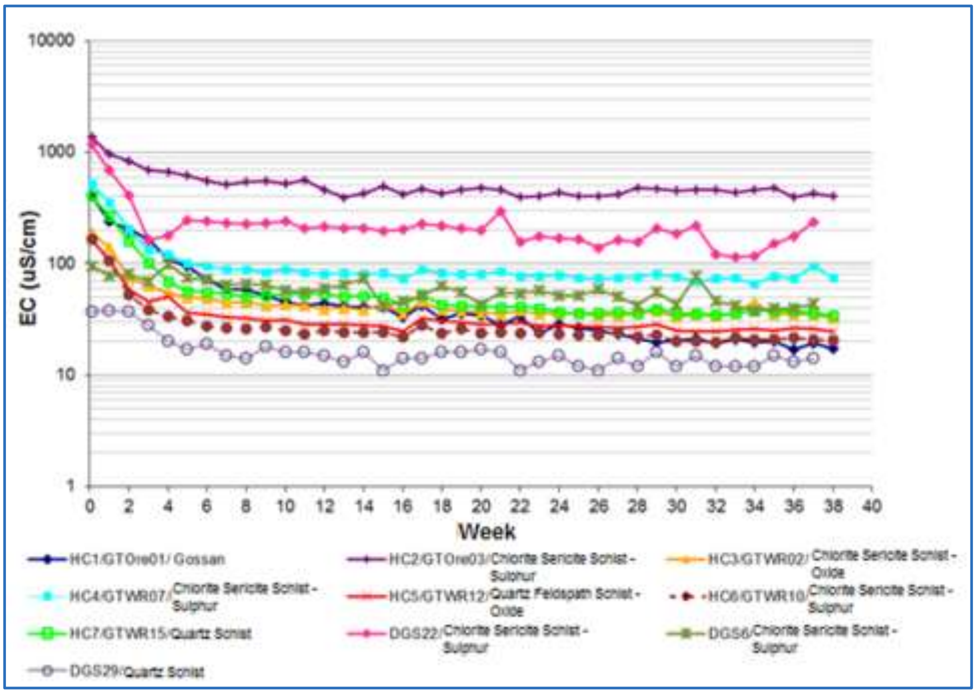

Figure from Polimetal, 2018

Figure 20.9 Leachate Weekly SO_4 Concentration Change

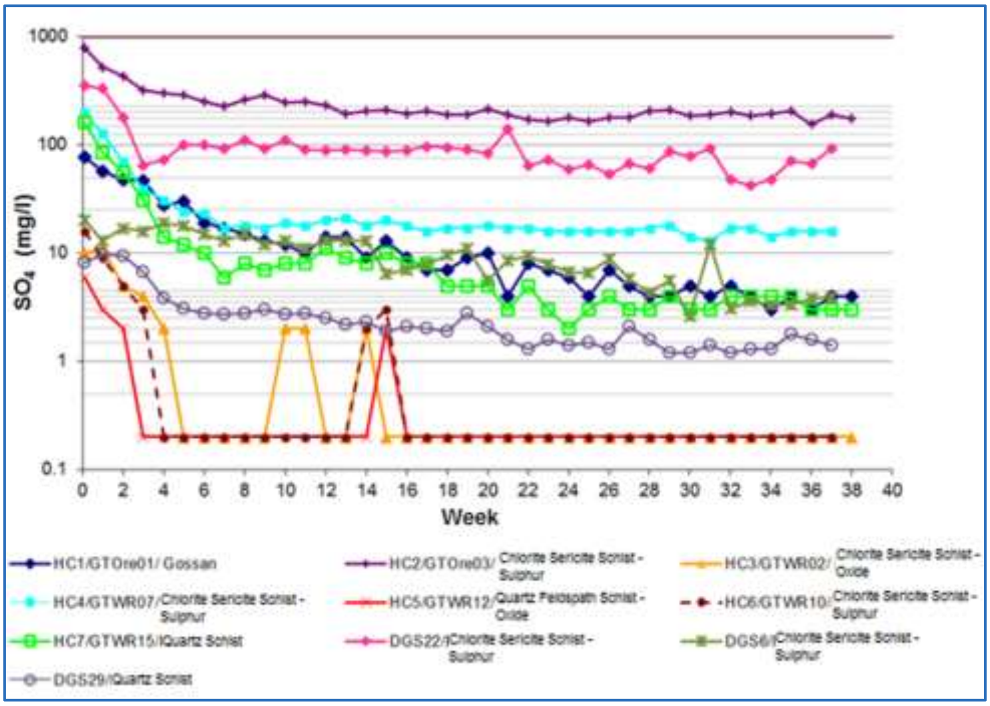

Figure from Polimetal, 2018

Figure 20.10 Leachate Weekly Total Ficklin Metals Concentration Change

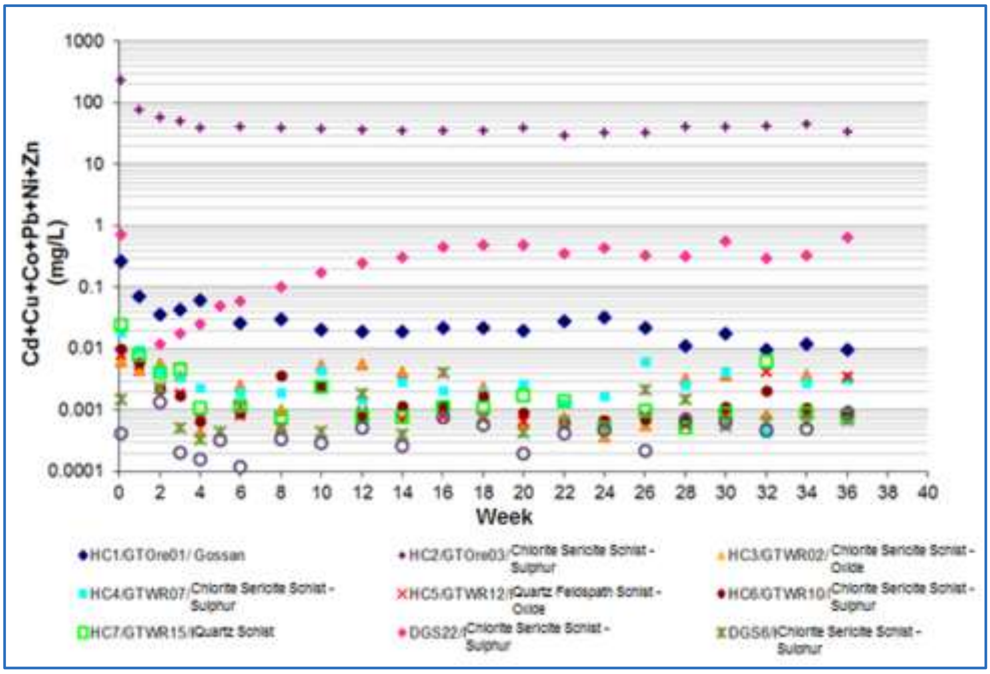

Figure from Polimetal, 2018

OreWin

Figure 20.11 Leachate Weekly Total Ficklin Metals Graph

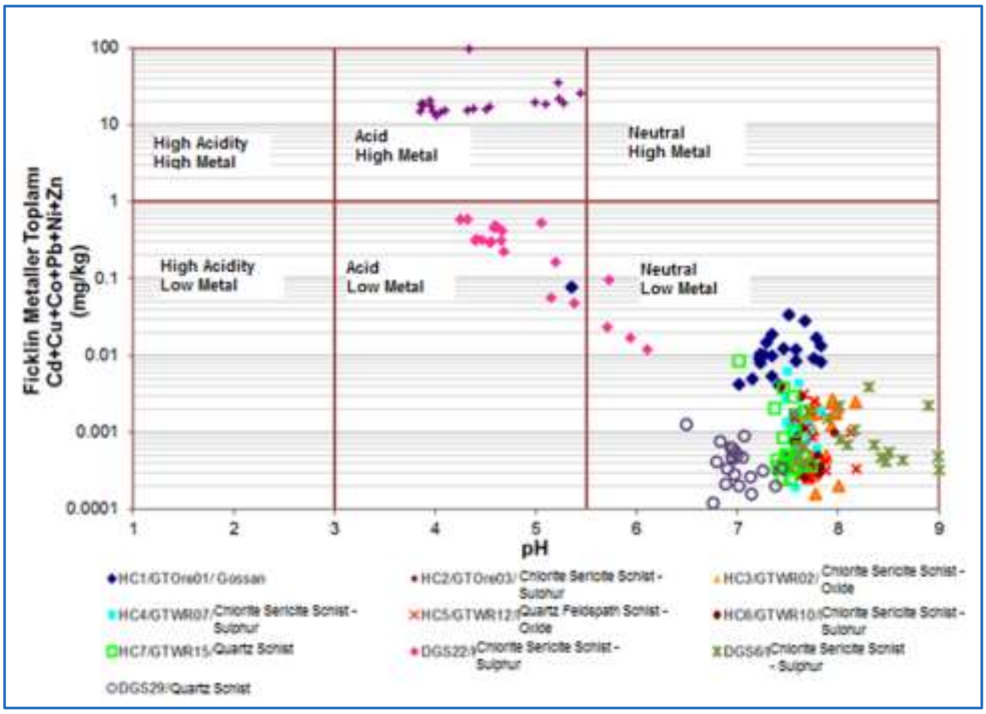

Figure from Polimetal, 2018

20.2.5 PFS19 Preliminary Waste Strategy

PFS16 assumed that all waste could be incorporated into a single, non-acid generating waste dump based on a favourable overall ratio between neutralising potential and acid generating potential. Additional work has since been performed to support the waste dump strategy for PFS19, including assessment of metals leaching.

The PFS19 mine waste schedule constituents are characterised in terms of weathering, sulphur content, and lithology. Based on these material sub-types, waste is initially sub-divided as being suitable (generally < 0.1% sulphur and stipulated lithology) or unsuitable as construction material for the clean water pond and TSF embankments. Over the life-of-mine, approximately 36 Mt of suitable waste is directed to these construction activities.

The waste remaining after satisfying construction requirements is directed to the mine waste dump(s). This waste dump material stream has been sub-divided into PAG and NPAG, with PAG material including all waste sourced from MISZ (low-grade mineralisation shell). PAG waste totals 17.6 Mt or approximately 13% of the remaining mine waste.

Annual presentation and allocation of mine waste is summarised in Figure 20.12.

Figure 20.12 Mine Waste Disposal

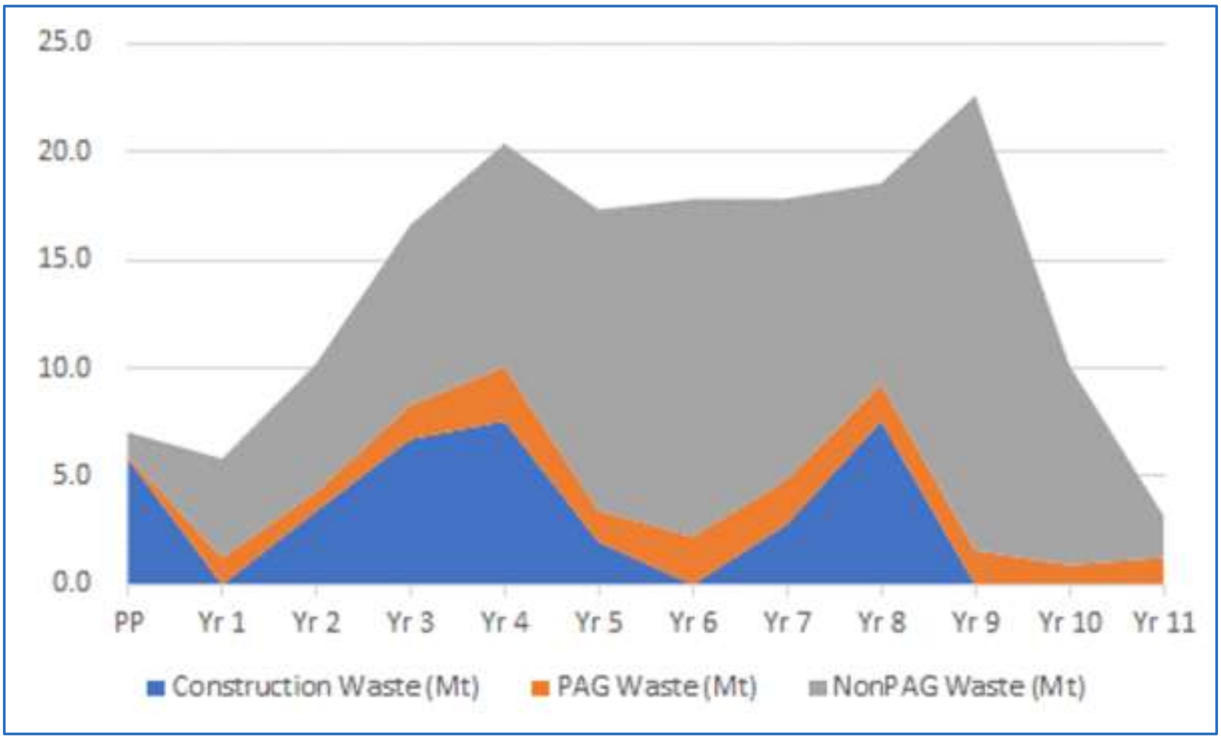

Figure from Polimetal, 2019.

The PFS19 waste dump strategy assessed predicted water seepage run-off quality relative to applicable Turkish effluent guidelines. Three different waste rock dump (WRD) compositions incorporating all (13%), partial (2%), or nil PAG waste were assessed for water quality. This analysis showed that, to meet all metals seepage quality criteria, all PAG material should be excluded.

In practical terms, rather than combining all waste in a single dump, the recommended strategy is to place the bulk of the waste that is not problematic, in terms of ARD or long-term metals leaching, into a single dump, with all PAG material managed in a separate facility.

The preliminary assessment in PFS19 is that this separate PAG dump will incorporate a liner and underdrainage for solution management, and will also incorporate dosing with limestone as a management technique.

20.2.6 Emissions

PM_{10} parameters were measured at the site and an air quality model was created for PM_{10} (Figure 20.13 and Figure 20.14). In addition, HC, CO, NO, CO_2 and SO_2 emissions were calculated and found that they were below the legal limits so air quality models for those parameters were not prepared. The PM10 modelling at nearby villages is shown in Table 20.5.

Table 20.5 Air Quality of Modelling of PM_{10} at Nearby Villages

	Baseline (µg/m³)	24 Hours		Annual	
		Increase (µg/m³)	Total (µg/m³)	Increase (µg/m³)	Total (µg/m³)
Limit Value		**50**		**40**	
Hacıömerderesi Village	18.9	7.3	26.2	2.6	21.5
Meyvalı Village	14.9	14.3	29.3	5.4	20.3

Based on this air quality modelling, daily and annual PM_{10} concentrations will stay below legal limits at Hacıömerderesi and Meyvalı villages.

Figure 20.13 24 Hours PM_{10} Concentration Dispersion

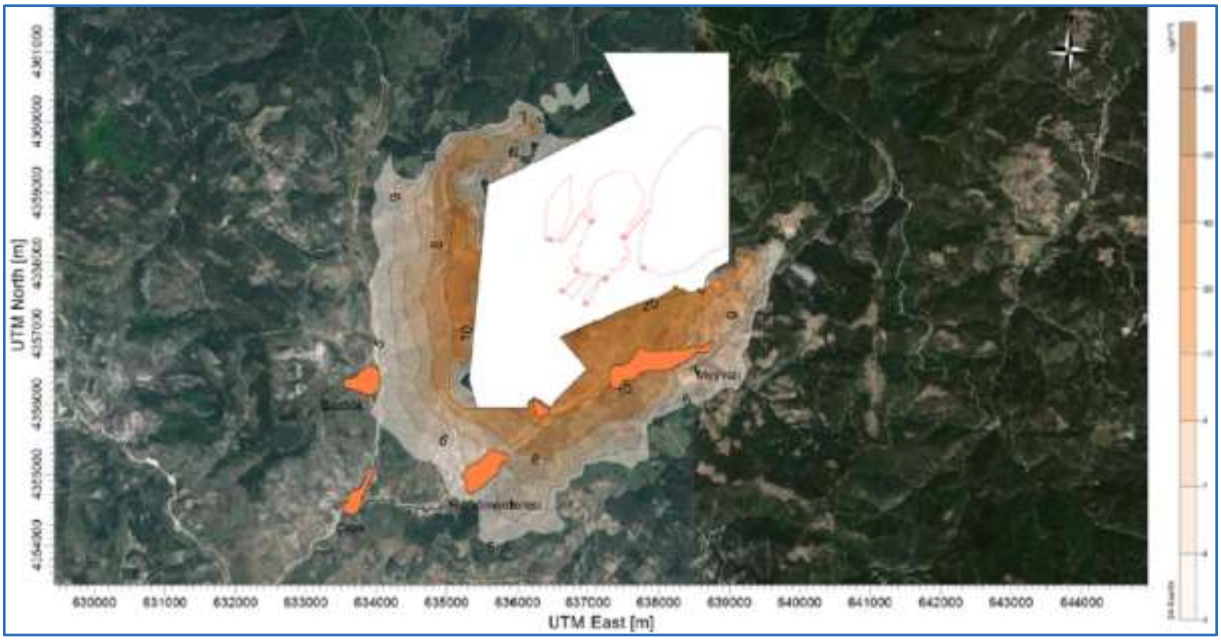

Figure from Polimetal, 2018

Figure 20.14 Annual PM_{10} Concentration Dispersion

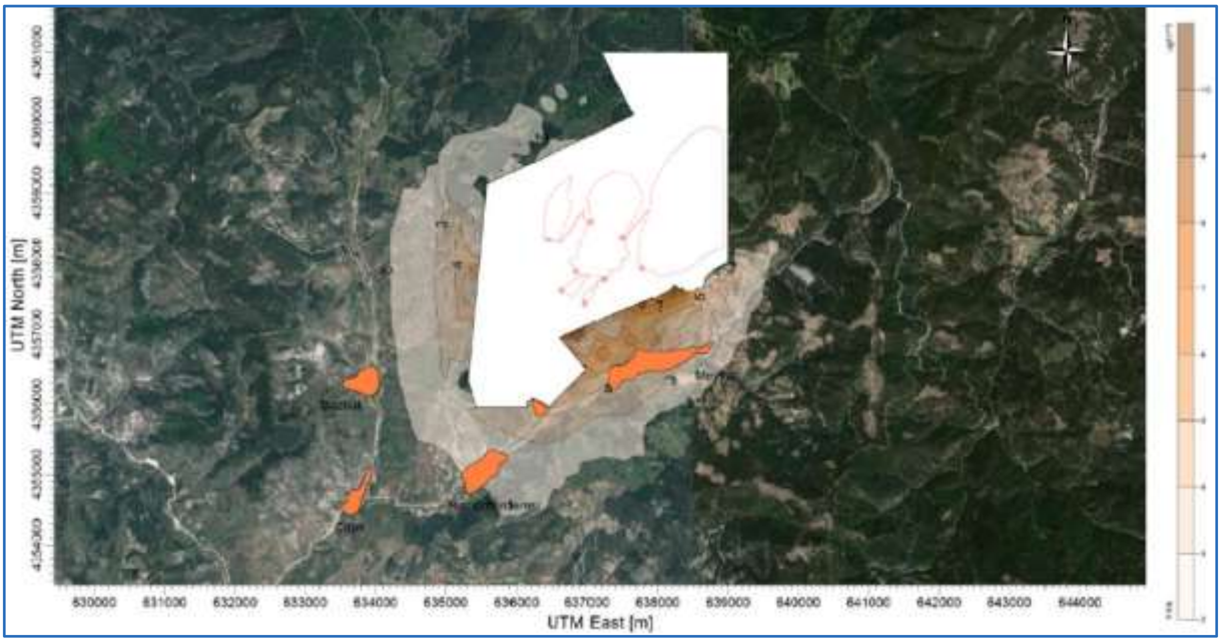

Figure from Polimetal, 2018

20.2.7 EIA Status

Based on the studies that are summarised in this section, the EIA report was compiled and submitted to the Ministry of Environment and Urbanisation on 15 December 2015. The first evaluation commission meeting was held with the participation of 18 government institutions on 13 January 2016.

Additional information was requested by the Water and Sewage Administration of Balıkesir Municipality. A revised EIA report was re-submitted in late-February 2016 and the EIA positive certificate for the operation was received on 1 July 2016.

The EIA report will be compiled when the project design is finalised at the end of the project feasibility studies.

20.3 Permitting

Most of the project area falls into forest land and will need forestry permits from the General Directorate of Forestry and Prime Ministry. The project will require a total 370.4 ha of forest permit area over the life of the mining operation.

It is expected that the following additional permits will be required:

- EIA revision
- Forest permits,
- Explosive usage and storage permits

- Environmental permit (including emission and water discharge permits)
- Environmental permit for tailings storage facility
- Explosive transportation permit
- Highway connection permit
- Village road usage permit
- Underground water usage permit
- Water usage permit
- Waste regular storage permit
- Private security permit
- Radio permit
- Permit for non-agricultural use
- Temporary storage permit for hazardous waste

There may be other permits that are not foreseen at this time, and some of the above may become unnecessary as more planning and detail is completed at the project.

20.4 Land Ownership

Approximately 90.8% of the project area belongs to the Forest Department and the remainder is private land owned by the Municipality, Treasury, and individuals (1,068,313.4 m^2). To date, 756,265 m^2 of private lands have been purchased by Polimetal, and the purchasing process still continues.

Figure 20.15 shows land ownership within the EIA boundary.

Figure 20.15 Land Ownership within the EIA Boundary

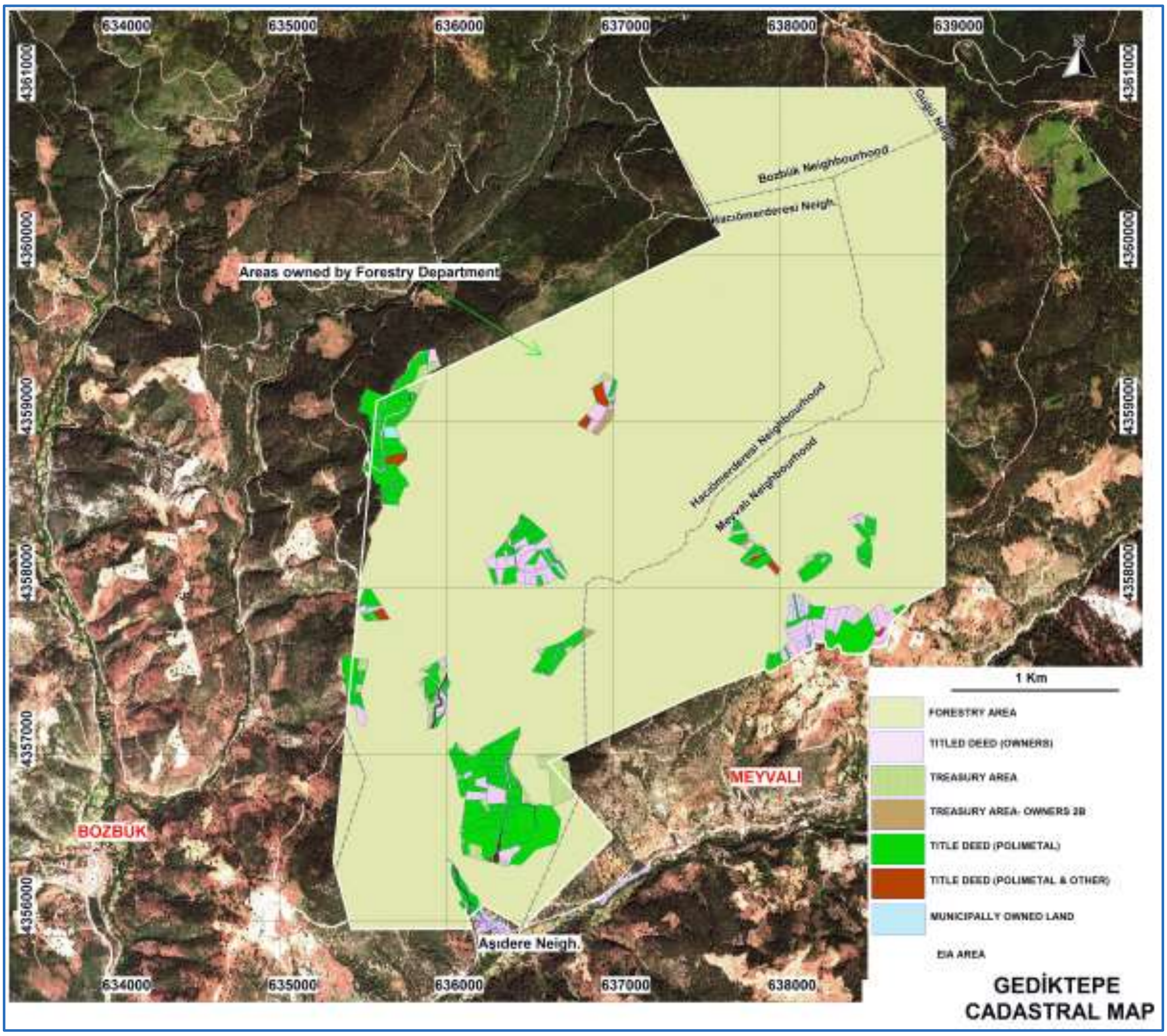

Figure from Polimetal, 2019.

20.5 Social and Community Impact

Polimetal has been drilling on the site since 2012 and has the support of the local community. The camp is established and is currently used by all project-related groups.

During the exploration period, local community and all officials were informed about the status and development of the project. During exploration drilling, around 100 local people were employed from the villages of Meyvalı and Hacıömerderesi.

Polimetal opened a liaison office at Hacıömerderesi village and a dedicated public and community relations officer was employed to contact and to inform all households and stakeholders. This office has now closed.

The local community is accustomed to mining activities in the region: the government has been operating one of the country's biggest open pit boron mines 57 km south of the Gediktepe project location, and approximately 40 km north of the project, a private company is operating an open pit lead, zinc, and copper mine and flotation plant.

Manpower for the project will be sourced from the local community depending on the requirements of the job. Considering the current local income level, the Gediktepe project will add value to the local community by employment, local contracting opportunities, local purchasing, community development programmes, and, transportation.

The Turkish State Water Works (DSI) has designed and planned to construct a potential water storage pond, which would be located within the footprint of the TSF, for local irrigation purpose. Because of this conflict, Polimetal has applied to the GDMPA to take a public welfare decision in favour of Gediktepe project. GDMPA personnel visited the site, and took ore samples and all the project details, and the public welfare decision was made and the DSI's water storage pond was cancelled with the approval of three ministers (Minister of Forest and Water Works, Minister of Energy and Natural Resources, and Minister of Development) in October 2015.

20.6 Closure

The details of project closure are still being finalised and require additional information for design. The collection of that information is in progress or is incorporated into the project execution plan.

Estimated closure and reclamation costs for the tailings storage facility have been provided. Those costs are included in the project financial analysis as late-stage capital expenditures in year-11.

Rehabilitation costs have been included for placing and spreading topsoil in disturbed areas and replanting seedlings.

The salvage value of equipment and scrap metal recovered from the process plant are expected to cover the cost of decommissioning of the plant facilities.

A summary of the estimated closure and reclamation cost are summarised in Section 21.

21 CAPITAL AND OPERATING COSTS

21.1 Summary

The capital and operating cost estimates include:

- Development and operation of an open pit mine
- Construction and operation of an oxide processing plant to produce gold and silver doré
- Construction and operation of a sulphide processing plant to produce copper and zinc concentrates with by-product gold and silver by flotation, with subsequent transport to European smelters for treatment, and
- All associated support infrastructure and utilities to construct and operate the mining and processing project.

The base capital and operating cost estimates have been developed by various parties contracted to Polimetal. Due to the different rates of scope development in different project areas, and different inherent risks, individual capital and operating costs have different levels of accuracy. Application of capital contingency factors appropriately reflects this accuracy spread.

All cost estimates are presented in United States dollars (US$) and, in the majority of cases are based on prices that were current in the fourth quarter, 2018. Where cost estimates are based on earlier data, appropriate escalation has been applied.

The United States dollar to Turkish Lira exchange rate adopted for the estimates is 6.0 TL/US$.

Total project initial, expansion and sustaining capital costs are shown in Table 21.1.

Table 21.1 Project Capital Costs

Capital Costs	Unit	Initial	Expansion	Sustaining	Total
Plant	US$M	44.4	53.2	2.9	100.5
Infrastructure (TSF)		19.6	–	15.3	34.9
Site Investigation and Project Eng.		10.0	–	–	10.0
Private Land Purchase (incl. above)		–	–		–
154 kVa PTL for Concentrator		0.9	–	3.4	4.3
Clean Water Pond		5.7	–		5.7
Operations Accommodation Camp		10.9	–		10.9
Water Diversion Structures		–	–	2.1	2.1
PAG Waste Dump		6.7	–		6.7
Concentrate Handling Port Facilities				1.0	1.0
Rehab. and Closure				22.7	22.7
EPCM		9.4	9.0		18.4
Owners EPCM Management Team		9.4	4.5		13.9
Pre-production Mining		25.9	–		25.9
Contingency		21.2	3.8	9.5	34.5
Capital Costs		164.1	70.6	56.9	291.6
Contingency (% Direct Costs)		17%	7%	20%	15%

Sunk costs for project infrastructure are excluded from the estimates. These include the village bypass road, field camp, initial power line, land acquisition and all other exploration and feasibility study costs to the end of 2018.

Table 21.2 shows the breakdown of estimated life-of-mine project operating costs.

Table 21.2 Project Operating Costs

	Total (US$M)	Breakdown Unit	Unit Cost (US$)
Mine			
Owner's Staff	40.2	$/t total moved	0.21
Mining Cost	270.0	$/t total moved	1.42
Mine	310.2	$/t total moved	1.63
Process			
Oxide Direct Cost	57.4	$/t ore Oxide	20.85
Sulphide Mill Direct Cost	369.3	$/t ore Sulphide	19.88
Process	426.8	$/t ore	20.08
Administration			
Sitewide G&A	43.8	$/t ore	2.06
Site Camp Costs	41.4	$/t ore	1.94
Land Usage / Forestry Fee	22.4	$/t ore	1.05
License and Compliance Fees	0.6	$/t ore	0.03
Administration	108.3	$/t ore	5.07
Overall Operating Cost	**845.2**	**$/t ore**	**39.62**

Excluding pre-stripping, estimated mine life operating costs total $845.2M.

21.2 Capital Costs

21.2.1 Process Plants

Processing plant capital estimates have been developed by GRES from first principles with costs for all major mechanical and electrical equipment supported by budget pricing. The estimate is characterised as being to an Association for the Advancement of Cost Engineering Class 2 estimate with a level of accuracy within –10% to +15%.

Costs include purchase, freight and installation costs. Contingency plus EPCM and owner's costs are collected separately. The estimate assumes all new equipment.

The oxide plant is initial capital while sulphide plant expenditure is classed as expansion capital occurring after production start-up. Sustaining capital in this category is based on a percentage of the original install cost.

The sulphide concentrator commences operation in year-3 of production and uses comminution and associated equipment installed for oxide processing.

21.2.2 Tailings Storage Facility

The tailings storage facility (TSF) estimate has been compiled by ENSU, a Turkish engineering company engaged by Polimetal to design and cost the TSF and the clean water pond. The initial TSF lift is required at the start of production to provide for oxide processing tailings and to capture run-off from upstream areas that are disturbed during establishment of the mine. The TSF embankment is built in a series of downstream raises using selected mine waste. Sustaining capital is for the cost of deferred embankment stages.

Cost estimates use detailed quantities and market unit rates, which are generally at a discount to the rate reports published by government organisations. Unit rates have been verified by comparison with current contract rates from the Alacer Çöpler project.

21.2.3 Site Investigations and Project Engineering

This item captures ongoing Polimetal costs estimated to be required to complete the feasibility study and support a project approval decision. It includes allowances for:

- Salary and associated costs for Polimetal Gediktepe team.
- Additional metallurgical testwork and associated drilling for metallurgical samples.
- Additional drilling to improve the orebody confidence classification and confirm geotechnical conditions.
- Completion of all the study work required to finalise the feasibility study and secure project funding, including more detailed front end engineering design (FEED).

21.2.4 154 kVa Power Transmission Line

An additional power line has been identified as being required to support the increased power demand when the sulphide processing plant comes on line in production year-3.

It is assumed that a 26 km overhead 154 kVa powerline will be constructed in production year-2 to supply 25 MW to the site. As well as the power line, the capital cost estimate includes provision of a new sub-station plus design, fees, and land expropriation.

The cost arrangement includes the power utility owning the line and reimbursing the project for the line cost by means of a discount on power tariffs over a five-year period, commencing in year-3. This discount is shown as an adjusted capital cost rather than in the cost of power and associated operating costs.

21.2.5 Clean Water Pond

The clean water pond is located downstream of the TSF and is constructed at the start of the project to capture non-contaminated water diverted around the area disturbed by clearing for mine operations, and to provide local water supply needs (location shown in).

21.2.6 Operations Accommodation Camp

Based on Alacer experience at the Çöpler site, an allowance has been made for a 500-person accommodation camp proximal to the mine site. Amongst other benefits, it will ensure continuity of operations when road access is difficult due to weather conditions.

The proposed camp is a mix of single and married accommodation and includes a kitchen / dining area plus associated social and recreational facilities. Cost estimates are based on recent Çöpler mine rates provided by Alacer.

21.2.7 Water Diversion Structures

This estimate is a deferred cost that will apply when the existing creek running through the mine area is re-routed into a lined channel on a purpose-built berm on the eastern footwall of the open pit.

21.2.8 PAG Waste Dump

Approximately 18 Mt of potentially acid generating (PAG) waste has been identified in the open pit. The current technical advice is that the appropriate management approach is to separate this material in a purpose built, lined dump, with associated under-drainage, collection and diversion facilities and structures.

The location and specific design of this facility is not finalised. Pending design confirmation, and since the proposed facility has common design and cost elements with the TSF, ENSU TSF unit rates have been used to build up an estimate of the initial cost of a suitably sized facility to accommodate this material.

21.2.9 Concentrate Handling Port Facilities

The sulphide concentrator will produce separate copper and zinc concentrates. Ore type variations are expected to result in significant short-term variations in the quality of the copper concentrate. The overall control strategy is to separate the copper concentrate into standard and complex batches for sale. In order to manage this process in the short term and assemble concentrate batches within specifications, significant blending is expected to be required at the selected port facility.

Capital provision has been made to construct six separate, 10 kt capacity, covered concentrate storage bays at the port to facilitate the blending and ship-loading process.

21.2.10 Rehabilitation and Closure

The rehabilitation and closure cost estimate has three main components:

- TSF closure (estimated by ENSU),
- PAG dump closure (assumed to be double the cost of the PFS16 heap leach pad closure), and
- Haulage, placement, spreading and seeding of stockpiled topsoil on disturbed areas.

Costs are assigned at the end of the mine life.

21.2.11 EPCM

The EPCM allowance in the initial and expansion capital relates specifically to the pre-contingency estimates made by GRES for the capital cost of the oxide and sulphide processing plants. The estimate includes project management, engineering, and drafting, and EPCM supervision and management. No other EPCM allowance has been made.

21.2.12 Owner's EPCM Team

A significant owner's EPCM team allowance has been included, based in part on the Alacer experience at the Çöpler project.

The allowance in the initial capital estimate is the same as estimated by GRES for EPCM related items. The scope of the owner's EPCM team will encompass all of the site activities required to prepare the project for production.

The expansion owner's team is significantly reduced, 50% of the GRES sulphide plant allowance, recognising the reduced owner EPCM scope in this expansion phase.

21.2.13 Pre-Production Mining

Mining costs to develop the pit for ore mining and deliver suitable construction waste to the TSF embankment prior to oxide plant start of production are capitalised.

21.2.14 Contingency

The overall contingency estimate of 17% of direct costs excludes EPCM and Owner EPCM team costs. This number rolls up the lower plant contingency estimated by GRES with higher contingencies (up to 30%) for other capital items where the design / scope and/or the costs are not as well defined.

The lower contingency of 7% for expansion capital directly reflects the GRES sulphide plant estimate.

21.3 Operating Costs

21.3.1 Mining Costs

Open pit mining is to be carried out on a contract basis using a local Turkish contractor. It is anticipated that the selected mining contractor will also carry out the pre-mine civil construction works including infrastructure earthworks, and construction of the clean water pond and TSF.

Budget pricing based on preliminary mine plan designs and quantities has been secured by Polimetal from local contractors, including the incumbent at the Çöpler mine. Contract mining costs will cover drilling and blasting plus loading and hauling of both ore and waste plus establishment and maintenance of all mine area haul roads.

In addition, these costs also include topsoil stripping and stockpiling and an allowance of 2% of the contract value for unanticipated works carried out under day rates for tasks not included in the contract rates.

These overall direct contract costs (excluding pre-stripping) are estimated to be $270.0M or $1.42/t mined.

In addition to the above direct costs, owner's costs associated with the mining operation (excluding the pre-strip period) total $40.2M or $0.21/t mined. These costs incorporate:

- The owner's mining team of $1.52Mpa (steady state sulphide treatment)
- Grade control costs ($0.49/t of ore)
- Rehandle and crusher feed charges at the primary crusher ($0.83/t of ore for 45% of oxide ore and 85% of sulphide ore)

21.3.2 Processing Costs

Oxide and sulphide plant processing costs have been developed from first principles by GRES. Major components of the operating costs include labour, power, fuel, reagents and maintenance costs. Operating costs are based on the planned production rates and the results of the ore types and the various testwork programmes detailed elsewhere in this report.

21.3.2.1 Oxide Processing

Life-of-mine cost averages are different due to allocation of process staff costs during those periods when both oxide and sulphide ores are treated, and also as a result of reduced tonnages during plant ramp up periods.

21.3.2.2 Sulphide Processing

The life-of-mine unit cost is higher due to:

- Lower throughput associated with ramp-up rates in the first year of operation,
- Additional operating costs related to gear up of plant staff during year-2 prior to the start of production in year-3, and
- Allocation of process staff costs at the end of the process life when residual oxide stockpiles are processed.

21.3.3 Administration Costs

Administrative costs consist of general office staff salaries, maintenance, supplies and general, camp costs, plus land use and forestry fees totalling $108.3M over the life of the mine.

22 ECONOMIC ANALYSIS

All dollars in PFS19 are US dollars.

22.1 Financial Summary Results

This PFS is for the construction and operation of an open pit mine, oxide (CIP) and sulphide (concentrator) processing facilities, and associated infrastructure. The initial project mines oxide ore to produce gold and silver doré on site from CIL processing, After the first two years of production, treatment transitions to processing of polymetallic sulphide ore in a concentrator (by flotation) to produce separate copper and zinc concentrates for sale outside of Turkey.

The oxide treatment rate is 1.096 Mtpa while the sulphide treatment rate is 2.378 Mtpa. The combined treatment life of the project is approximately 11 years. Total oxide ore is 2.7 Mt and sulphide ore totals 18.8 Mt. Oxide ore processing is focussed in the first two years of operation. Sulphide processing commences in year-3 of processing and treats 18.6 Mt of copper / zinc ore over a nine-year period. Concentrator products include separate copper and zinc concentrates, both with by-product silver and gold credits. The long-term metal price assumptions used in the base case economic analysis are detailed in Table 22.1.

Table 22.1 Economic Analysis Metal Prices

Metal	Unit	Price (US$)
Copper	lb	3.20
Gold	troy oz	1,315
Silver	troy oz	18.0
Zinc	lb	1.10

The analysis calculates annual cash flows over the life of the mine and incorporates, Turkish taxes, permit and license fees, and government royalties on metal sales. The analysis is based on 2018 fourth quarter US dollars and a Turkish to US$ exchange rate of 6.0.

The base case economic analysis returns an after tax Net Present Value (NPV) at an 8% discount rate of $186.1M. It has an after tax internal rate of return (IRR) of 27% and a payback period of 4.1 years. Financial results are summarised in Table 22.2.

Table 22.2 Financial Results

	NPV	
	Before-Tax	After-Tax
	US$M	US$M
Undiscounted	420.4	412.0
5%	258.4	252.5
8%	**191.0**	**186.1**
10%	154.8	150.5
15%	86.8	83.5
IRR	27%	27%
Peak Funding	**–164.1**	
Payback (Years)	4.09	4.12

Table 22.3 summarises life-of-mine production, processing and concentrate quantities.

Table 22.3 Life-of-Mine Production and Processing Quantities

Life-of-Mine Production	Unit	Quantity
Oxide Ore	kt	2,755
Oxide Grade – Au	g/t	2.34
Oxide Grade – Ag	g/t	56.7
Sulphide Ore	kt	18,580
Sulphide Grade – Cu	%	0.92
Sulphide Grade – Zn	%	1.98
Sulphide Grade – Au	g/t	0.85
Sulphide Grade – Ag	g/t	31.8
Weathered Waste	kt	26,449
Fresh Waste	kt	142,757
Total Material	kt	190,541
Copper Concentrate	kt	387
Zinc Concentrate	kt	503

Life-of-mine metal production is summarised in Table 22.4. Metal smelting and refining losses associated with concentrate treatment have not been deducted from these totals.

Table 22.4 Life-of-Mine Metal Production

Copper in Concentrate	**kt**	**115**
Copper in Concentrate	**kt**	**284**
Gold		
Oxide	koz	187
Copper Concentrate	koz	128
Zinc Concentrate	koz	31
Total Gold	**koz**	**345**
Silver		
Oxide	koz	3,547
Copper Concentrate	koz	2,329
Zinc Concentrate	koz	2,272
Total Silver	**koz**	**8,148**

22.2 Turkey Fiscal Environment

Turkish Mining Law 3213, established in June 1985, sets out the principles and procedures with regard to exploring, operating, enjoying rightful ownership of, and renunciation of mines. The mining law has been amended from time-to-time. Secondary legislation related to the mining law includes 'Regulation on the Implementation of Mining Activities' and 'Regulation on Mining Activity Permits'.

Other legislation related to mining activities address environmental law (EIA), licensing, health and safety, air, soil and water pollution, hazardous waste, wildlife protection, and rehabilitation.

All minerals are owned by the State.

Exploration and mining activities are carried out under a licensing system including Exploration licenses, Operations licenses, and Operating permits. License-holder responsibilities vary according to the type of license and the minerals being mined, but will generally include:

- Payment of annual license fees,
- Payment of annual royalties,
- Submission of technical and financial reports, and
- Making timely application for permits to the relevant state institutions.

In general, all mining activities are exempt from VAT and customs duty on imported machinery and equipment.

Mining is supported by a favourable tax regime and government infrastructure provided by the General Directorate of Mineral Research and Exploration. Investment support of mining activities includes investment incentive programmes that reduce / defer corporate income tax and royalty discounts, depending on the type of commodity and the location of the mine.

22.3 Model Assumptions

22.3.1 Revenue Assumptions

Oxide ore is treated on site to produce gold and silver metal doré. The sales revenue is based on the study metal prices in Table 22.1, after adjusting for metal payability and refining charges, as detailed in Section 19.

Gross revenue from oxide gold and silver sales totals $305.4M.

Sulphide ore is processed on site to produce separate copper and zinc concentrates that are: transported by road to a port, blended as required, assembled into shipment batches, and shipped in bulk to suitable smelters. The concentrate revenue is calculated by determining the value of the payable metal according to the Table 22.1, assumptions and then deducting smelting, refining, freight and port charges. These charges are detailed in Section 19.

The gross value of payable metal in concentrates is $1,575M. The net smelter return, after deducting smelting and refining charges, penalties, land and sea freight and port charges, is estimated to be $1,290M.

For cash flow purposes, concentrate revenue and associated off-site realisation charges are deferred by two months to simulate probable cash flow delays associated with land transport, port blending, batch shipping and smelter payment terms. No working capital delay is applied to doré sale revenue.

22.3.2 Taxation

Taxable revenue for corporate tax purposes is defined as metal revenue minus operating expenses, including royalties, depreciation and depletion. The applicable Turkish tax rate is 20% of taxable income. Turkish investment incentives are modelled to reduce the tax payable in the initial production years.

Based on Alacer's advice, a capital investment incentive of 50% is applied to 70% of the total project capital expenditure, resulting in an investment incentive of $82.1M.

22.3.3 Royalties, Depreciation, and Depletion

Royalties are payable to the Turkish government at a rate based on the commodity and the metal price, as detailed in Table 22.5. These are new base royalty rates advised by Alacer as being scheduled to be made law in 2019. The PFS19 financial analysis base case prices fall in the ranges shown in bold in the table.

Table 22.5 Base Royalty Rate by Commodity and Metal Price

Rate (%)	Gold ($/oz)	Silver ($/oz)	Copper ($/t)	Zinc ($/t)
1	< 800	< 10	< 5,000	< 1,000
2	801–900	11–12	5,001–5,300	1,001–1,175
3	901–1,000	13–14	5,301–5,600	1,176–1,350
4	1,001–1,100	15–16	5,601–5,900	1,351–1,525
5	1,101–1,200	**17–18**	5,901–6,200	1,526–1,700
6	1,201–1,300	19–20	6,201–6,500	1,701–1,875
7	**1,301–1,400**	21–22	6,501–6,800	1,876–2,050
8	1,401–1,500	23–24	**6,801–7,100**	2,051–2,225
9	1,501–1,600	25–26	7,101–7,400	2,226–2,400
10	1,601–1,700	27–28	7,401–7,700	**2,401–2,575**
11	1,701–1,800	29–30	7,701–8,000	2,576–2,750
12	1,801–1,900	31–32	8,001–8,300	2,751–2,925
13	1,901–2,000	33–34	8,301–8,600	2,926–3,100
14	2,001–2,100	35–36	8,601–8,900	3,101–3,275
15	> 2,100	> 37	> 8,901	> 3,276

The PFS19 financial analysis base case prices fall in the ranges shown in bold.

Based on commodity type, license-holders can obtain a royalty discount depending on the commodity. Table 22.6 shows the royalty discount for the Gediktepe products and the resultant effective royalty rate.

Table 22.6 Royalty Rate by Commodity and Metal Price

Royalty	Gold	Silver	Copper	Zinc
Base Royalty (%)	7.0	5.0	8.0	10.0
Incentive Reduction	40%	40%	50%	50%
Effective Royalty (%)	4.2	3.0	4.0	5.0

Based on the advice from Alacer, the additional royalty previously payable for mining on Forestry land is eliminated in the 2019 royalty law changes.

Royalty payable on metal sales is estimated after all applicable deductions are made from gross sales revenue. Alacer has advised that these deductions include:

- Ore processing costs at the mine, including administration overheads, but excluding mining and related expenses,
- All off-site doré and concentrate realisation charges, including land and sea transport, port handling, smelting and refining, and penalties, and
- Applicable depreciation and amortisation (D&A) charges (estimated at 80% of total D&A for the project).

Application of deductions from revenue that are not metal-specific are allocated pro-rata, based on the relative metal gross revenue share.

Estimated life-of-mine project royalties are $38.0M.

Depreciation is calculated using the declining balance method starting with the first year of ore production. The initial and sustaining capital use an 11-year life and 20% rate, except for structures, which use a 50-year life and 4% rate. Any remaining asset value at the last year of production is fully depreciated at that time.

Depletion of land and concession costs is applied at the rate the resource is mined.

22.3.4 Results

The key financial results of the study are summarised in Table 22.2.

Figure 22.1 presents the undiscounted, after-tax cash flow modelled for the project. A summary of total project initial and deferred capital costs is shown in Table 22.7. Table 22.8 shows the breakdown of estimated life-of-mine project operating costs.

Cash costs have been calculated using a gold equivalent ounce (AuEq) method, which is a non-IFRS[1] measure (with no standardised definition under IFRS) that converts non-gold production into gold equivalent ounces. Calculation of AuEq converts payable metals into revenue. For PFS19, the AuEq calculation uses the following metal prices to calculate gross revenue: Au = $1,315/oz, Ag = $18/oz, Cu = $3.20/lb, and Zn - $1.10/lb. This total revenue is then divided by the gold price of $1,315/oz to give the AuEq. The AuEq is shown in Table 22.9.

Table 22.10 shows the annual mining and processing production quantities.

The base case cash flow is detailed in Table 22.10.

[1] International Financial Reporting Standards

Figure 22.1 Undiscounted After-Tax Cash Flow (US$M)

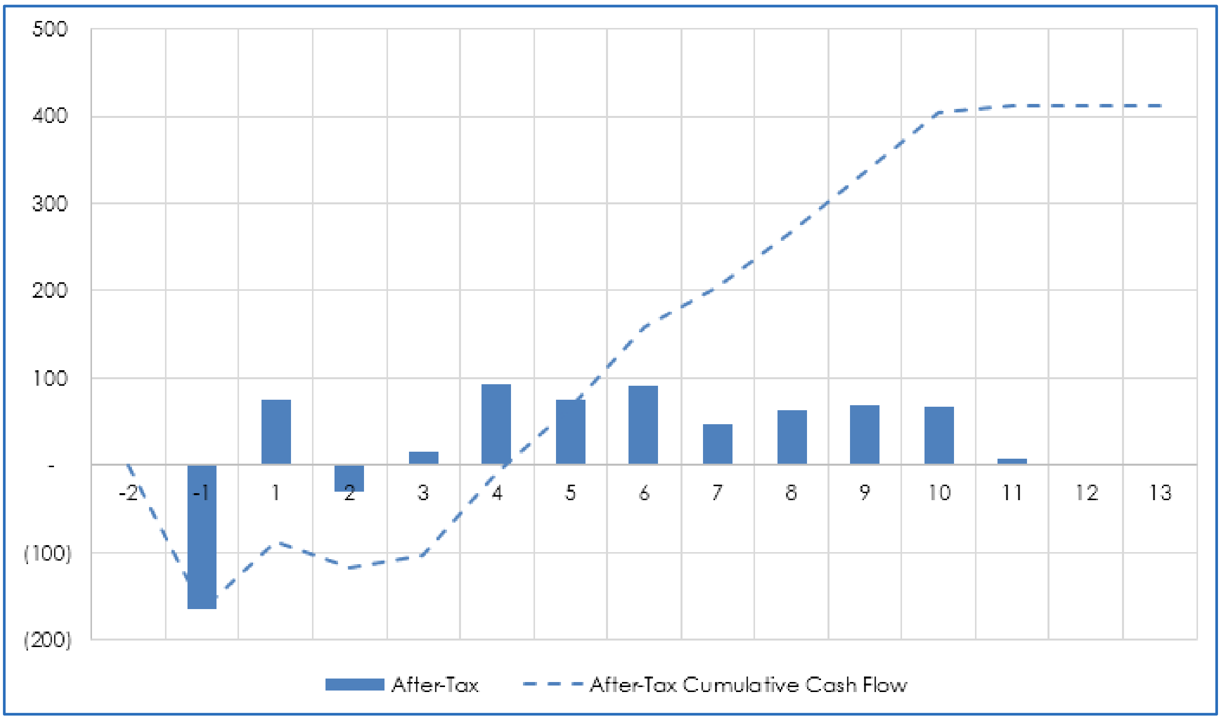

Figure by OreWin, 2019.

Table 22.7 Summary of Project Capital Costs

Capital Costs	Initial	Expansion	Sustaining	Total
	US$M			
Plant	44.4	53.2	2.9	100.5
Infrastructure	53.8	––	21.8	75.6
Closure	––	––	22.7	22.7
EPCM	9.4	9.0	––	18.4
Owner's EPCM Management Team	9.4	4.5	––	13.9
Pre-Production Mining	25.9	-	––	25.9
Contingency	21.2	3.8	9.5	34.5
Capital Costs	164.1	70.6	56.9	291.6

Table 22.8 Project Operating Costs

	Total (US$M)	Breakdown Unit	Unit Cost (US$)
Mine			
Owner's Staff	40.2	$/t total moved	0.21
Mining Cost	270.0	$/t total moved	1.42
Mine	310.2	$/t total moved	1.63
Process			
Oxide Direct Cost	57.4	$/t ore Oxide	20.85
Sulphide Mill Direct Cost	369.3	$/t ore Sulphide	19.88
Process	426.8	$/t ore	20.08
Administration			
Sitewide G&A	43.8	$/t ore	2.06
Site Camp Costs	41.4	$/t ore	1.94
Land Usage / Forestry Fee	22.4	$/t ore	1.05
License and Compliance Fees	0.6	$/t ore	0.03
Administration	108.3	$/t ore	5.07
Overall Operating Cost	**845.2**	**$/t ore**	**39.62**

Table 22.9 Gross Revenue by Metal and AuEq

Source	Gross Revenue (US$M)	AuEq (koz)
Oxide – Au	242.8	185
Oxide – Ag	62.6	48
Total Oxide	**305.4**	**232**
Sulphides – Cu	783.9	596
Sulphides – Au in Cu Conc.	149.3	114
Sulphides – Ag in Cu Conc.	35.1	27
Total Cu Concentrate	**968.4**	**736**
Sulphides – Zn	584.9	445
Sulphides – Au in Zn Conc.	12.6	10
Sulphides – Ag in Zn Conc.	8.9	7
Total Zn Concentrate	**606.4**	**461**
Gross Revenue	**1,880.2**	**1,430**

Based on metal prices: Au = $1,315/oz, Ag = $18/oz, Cu = $3.20/lb, and Zn - $1.10/lb
AuEq is Payable ounces = Gross Revenue / Gold Price

Table 22.10 Annual Production Quantities

Production	Units	Totals / Year	–1	1	2	3	4	5	6	7	8	9	10	11
Mine Production														
Oxide Ore	kt	2,755	23	1,426	956	242	86	12	9	1	–	–	–	–
Oxide Grade – Au	g/t	2.34	0.94	2.39	2.41	1.93	2.53	0.90	0.51	1.41	–	–	–	–
Oxide Grade – Ag	g/t	56.7	31.0	61.8	49.6	57.0	60.6	33.1	44.6	77.9	–	–	–	–
Sulphide Ore	kt	18,580	1	75	351	1,196	2,119	2,481	2,522	2,399	2,377	2,184	1,912	962
Sulphide Grade – Cu	%	0.92	0.32	0.72	0.91	1.47	1.20	0.98	1.00	0.79	0.78	0.76	0.70	0.68
Sulphide Grade – Zn	%	1.98	1.39	0.96	1.19	1.43	2.12	1.95	1.94	1.84	2.25	2.25	2.14	1.67
Sulphide Grade – Au	g/t	0.85	0.59	1.00	0.91	0.93	0.96	0.95	0.85	0.68	0.85	0.98	0.78	0.55
Sulphide Grade – Ag	g/t	31.8	14.7	37.0	28.8	33.9	37.9	31.9	30.0	28.2	33.3	35.5	31.0	20.0
Weathered Waste	kt	26,449	4,095	7,406	3,952	3,929	3,096	1,879	1,557	534	1	–	–	–
Fresh Waste	kt	142,757	3,223	6,056	14,706	17,171	18,803	20,220	16,097	17,120	15,621	8,612	4,305	823
Total Material	kt	190,541	7,342	14,964	19,965	22,538	24,105	24,592	20,184	20,054	18,000	10,796	6,217	1,785
Process Plant Production														
Oxide Ore	kt	2,755	–	1,046	1,096	274	137	–	–	–	–	–	141	61
Oxide Grade – Au	g/t	2.34	–	2.85	2.32	1.80	1.69	–	–	–	–	–	1.00	0.78
Oxide Grade – Ag	g/t	56.7	–	71.2	48.0	52.4	48.5	–	–	–	–	–	41.5	37.0
Sulphide Mill Ore	kt	18,580	–	–	–	1,618	2,041	2,378	2,378	2,378	2,378	2,378	2,072	962
Sulphide Grade – Cu	%	0.92	–	–	–	1.31	1.20	1.00	1.04	0.81	0.78	0.75	0.69	0.68
Sulphide Grade – Zn	%	1.98	–	–	–	1.35	2.11	1.99	1.92	1.83	2.25	2.22	2.14	1.67
Sulphide Grade – Au	g/t	0.85	–	–	–	0.93	0.95	0.97	0.85	0.69	0.85	0.95	0.78	0.55
Sulphide Grade – Ag	g/t	31.8	–	–	–	32.9	37.8	32.6	29.8	28.0	33.3	34.8	31.3	20.0
Metal Recovered to Doré														
Gold	koz	187	–	86	74	14	7	–	–	–	–	–	4	1
Silver	koz	3,547	–	1,690	1,195	326	151	–	–	–	–	–	133	51

Production	Units	Totals / Year	–1	1	2	3	4	5	6	7	8	9	10	11
Metal in Sulphide Concentrate														
Copper Concentrate	kt	387	–	–	–	45	58	54	56	43	42	41	32	15
Copper	klb	253,870	–	–	–	30,104	38,143	36,098	37,371	28,272	27,319	26,221	20,866	9,476
Gold	koz	128	–	–	–	12	18	19	16	12	16	20	12	3
Silver	koz	2,329	–	–	–	185	332	291	278	268	318	302	268	87
Zinc Concentrate	kt	503	–	–	–	27	59	65	62	60	74	73	62	22
Zinc	klb	625,585	–	–	–	33,582	72,564	80,479	76,729	73,637	92,589	91,443	76,802	27,760
Gold	koz	31	–	–	–	2	4	4	4	3	4	4	3	1
Silver	koz	2,272	–	–	–	124	290	299	282	263	333	325	278	79

Table 22.11 Cash Flow

Cash Flow	Totals/ Year	$'000												
		–1	1	2	3	4	5	6	7	8	9	10	11	12
Gross Income Sales														
Oxide	305,367	–	142,374	116,842	24,381	11,400	–	–	–	–	–	7,667	2,703	–
Sulphides	1,574,800	–	–	–	103,595	216,494	214,706	221,113	176,676	199,513	187,936	167,802	86,963	–
Gross revenue	1,880,167	–	142,374	116,842	127,976	227,894	214,706	221,113	176,676	199,513	187,936	175,468	89,666	–
Doré refining	6,516	–	3,093	2,250	585	271	–	–	–	–	–	230	88	–
Transport	64,889	–	–	–	5,287	8,513	8,685	8,628	7,488	8,454	8,278	6,860	2,698	–
Copper conc. treatment	34,931	–	–	–	4,083	5,219	4,919	5,065	3,879	3,775	3,679	2,978	1,333	–
Zinc conc. treatment	149,018	–	–	–	8,036	17,491	19,134	18,369	17,642	21,902	21,578	18,261	6,605	–
Refining charge	23,676	–	–	–	2,736	3,548	3,348	3,427	2,623	2,584	2,510	2,021	879	–
Concentrate insurance	2,045	–	–	–	190	282	283	278	228	254	254	200	77	–
Sales cost	281,075	–	2,791	2,262	17,437	34,360	36,956	33,981	31,949	37,221	33,903	32,812	17,403	–

Cash Flow	Totals/ Year	$'000												
		–1	1	2	3	4	5	6	7	8	9	10	11	12
Penalties	3,907	–	–	–	118	453	610	425	463	458	388	748	244	–
Government royalty on ore	37,974	–	3,349	2,567	2,908	4,863	4,578	4,607	3,372	4,226	4,324	3,178	2	–
Net revenue	1,557,210	–	136,233	112,013	107,512	188,218	172,563	182,100	140,893	157,609	149,321	138,730	72,017	–
Operating Cost														
Mine	327,867	17,659	23,036	29,249	34,874	38,099	39,202	32,979	36,152	35,086	22,312	14,046	5,173	–
Process	428,389	1,634	22,969	23,483	42,074	43,377	45,397	45,397	45,397	45,397	45,397	45,077	22,791	–
G&A	114,868	6,596	7,993	8,556	9,674	10,282	10,227	10,751	10,500	10,500	10,500	10,912	8,378	–
Operating cost	845,235	–	53,998	61,288	86,622	91,758	94,826	89,127	92,049	90,983	78,208	70,035	36,341	–
Net operating income pre-tax	711,975	–	82,235	50,725	20,890	96,461	77,737	92,973	48,844	66,626	71,113	68,695	35,676	–
Capital Cost														
Initial and expansion	234,700	164,137	–	70,563	–	–	–	–	–	–	–	–	–	–
Sustaining	56,851	–	5,665	9,412	5,537	1,921	1,921	(161)	1,919	2,447	367	367	27,455	–
Capital cost	291,551	164,137	5,665	79,975	5,537	1,921	1,921	(161)	1,919	2,447	367	367	27,455	–
Depreciation	291,551	–	32,827	27,395	37,911	31,436	25,533	20,811	16,616	13,677	11,431	9,218	64,696	–
Depletion	30,000	6,000	4,800	3,840	3,072	2,458	1,966	1,573	1,258	1,007	805	3,221	–	–
Tax	8,389	–	772	390	–	849	1,005	1,412	619	1,039	1,178	1,125	–	–
Operating cash flow after-tax	703,586	–	81,463	50,335	20,890	95,611	76,732	91,562	48,224	65,587	69,935	67,570	35,676	–
Before-tax cash flow	420,423	(164,137)	76,571	(29,251)	15,353	94,540	75,816	93,135	46,925	64,179	70,745	68,328	8,221	–
Before-tax cum. cash flow		(164,137)	(87,567)	(116,817)	(101,465)	(6,925)	68,891	162,026	208,951	273,129	343,875	412,203	420,423	420,423
After-tax	412,035	(164,137)	75,798	(29,640)	15,353	93,690	74,811	91,723	46,306	63,140	69,568	67,203	8,221	
After-tax cum. cash flow		(164,137)	(88,339)	(117,979)	(102,627)	(8,936)	65,875	157,598	203,903	267,043	336,611	403,814	412,035	412,035

22.3.1 Sensitivity

The economic sensitivity of the project was evaluated with respect to initial capital costs, operating costs and metal prices between +/–30% of base case values. Changes in metal prices is also indicative of relative changes in metal recoveries and/or the processed head grades.

Figure 22.2 and Figure 22.3 show the initial capital and operating cost sensitivities. These indicate that the project NPV is about twice as sensitive to operating costs as it is to the initial capital cost.

Further breakdown of the operating costs reveals that the project NPV is equally sensitive to changes in mining and sulphide processing costs and less sensitive to changes in oxide processing and G&A costs. Figure 22.4 through Figure 22.6 show mining cost and oxide / sulphide processing cost sensitivities. Figure 22.7 shows sensitivity to G&A costs.

Metal price sensitivity was assessed by individual payable metal to determine relative impact and ranking. Project NPV was most sensitive to changes in copper price, followed by zinc and then gold. Figure 22.8 through Figure 22.11 show the metal price sensitivities with an NPV at 8% discount rate. Sensitivities of NPV with a 5% discount rate are shown in Figure 22.12 to Figure 22.21.

Figure 22.2 Sensitivity to Initial Capital Costs – 8% Discount Rate

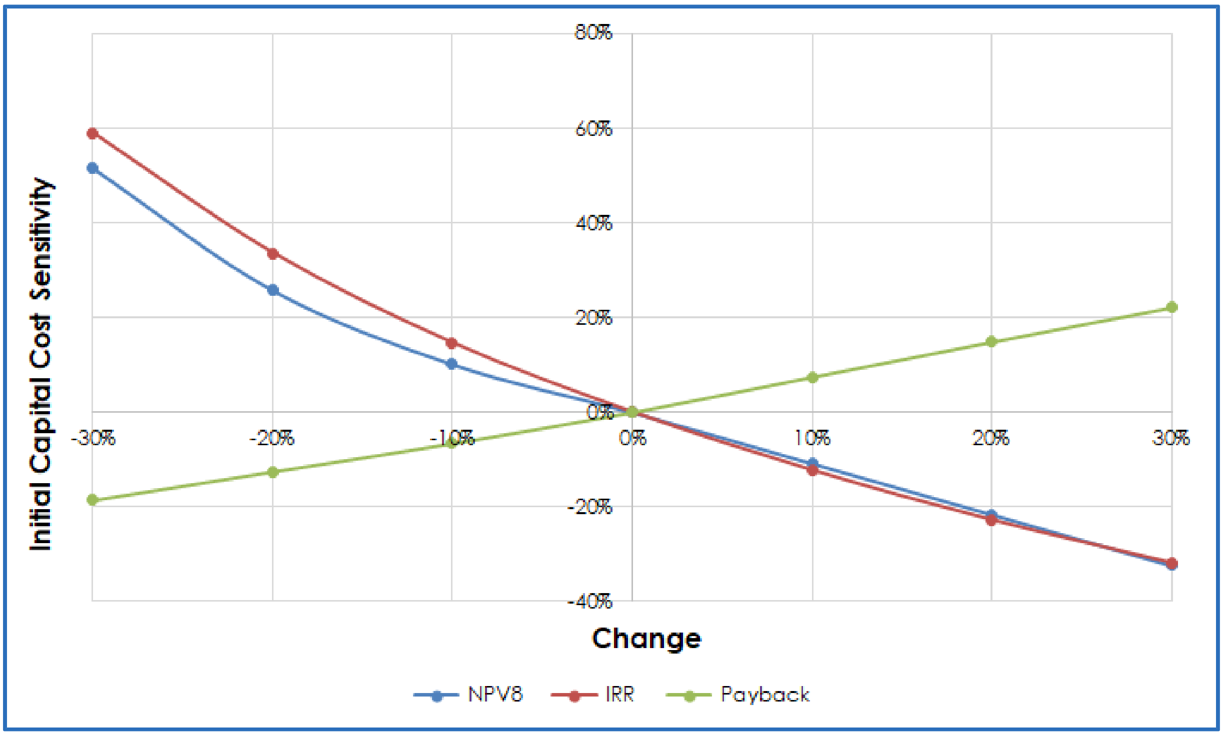

Figure by OreWin, 2019.

Figure 22.3 Sensitivity to Operating Costs – 8% Discount Rate

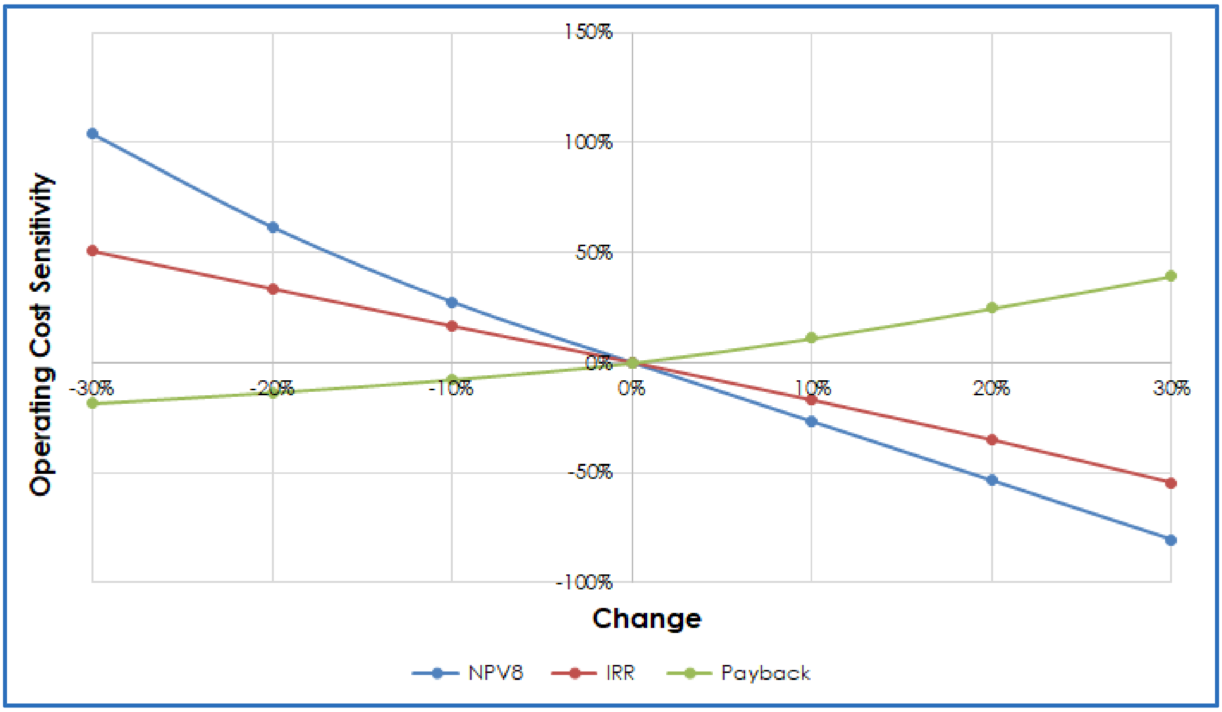

Figure by OreWin, 2019.

Figure 22.4 Sensitivity to Mining Operating Costs – 8% Discount Rate

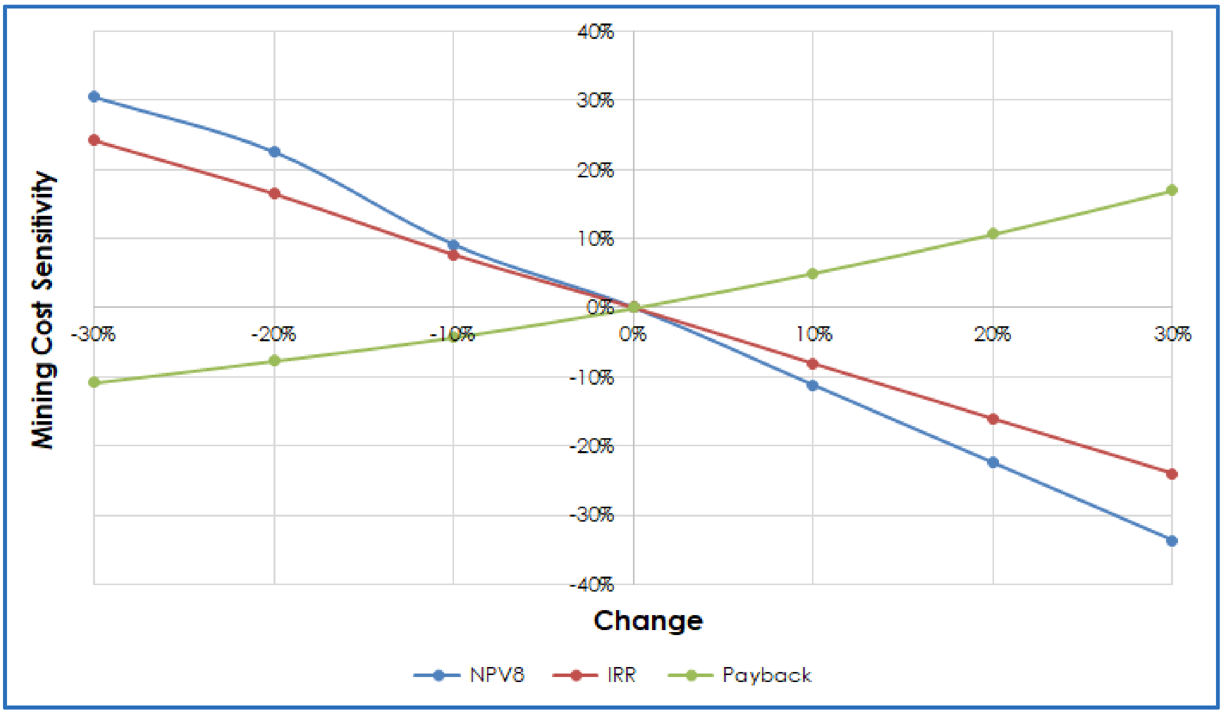

Figure by OreWin, 2019.

Figure 22.5 Sensitivity to Oxide Processing Operating Costs – 8% Discount Rate

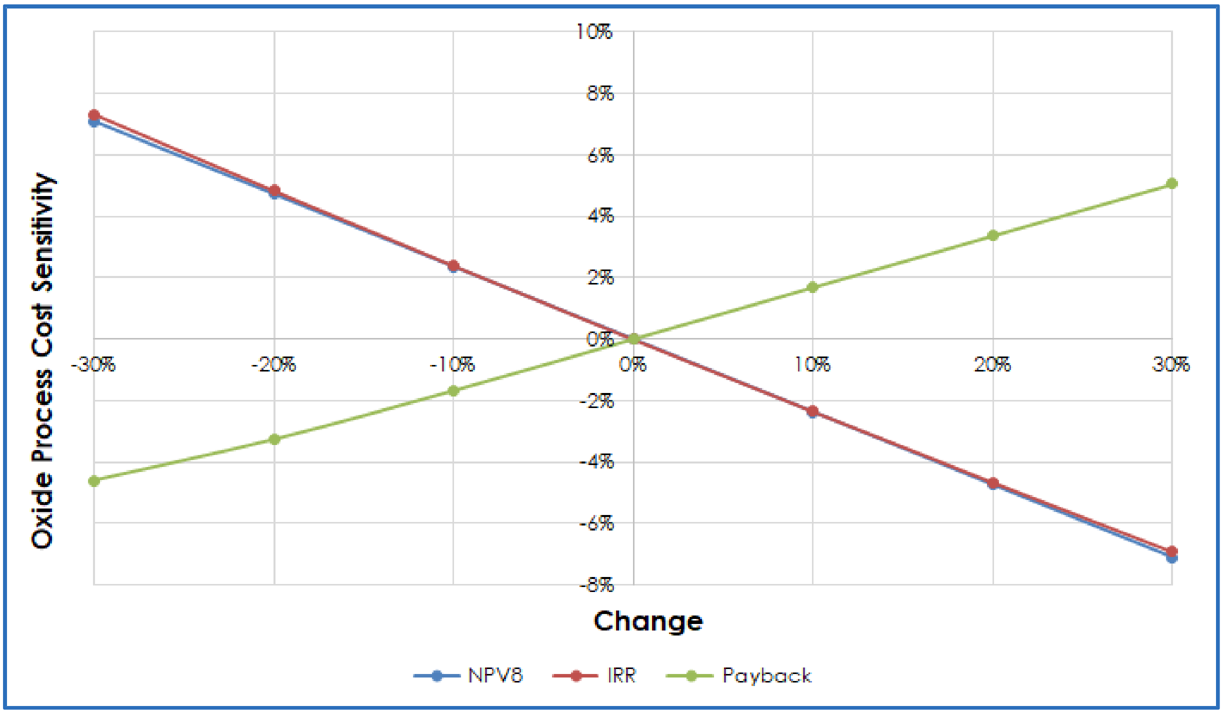

Figure by OreWin, 2019.

Figure 22.6 Sensitivity to Sulphide Processing Operating Costs – 8% Discount Rate

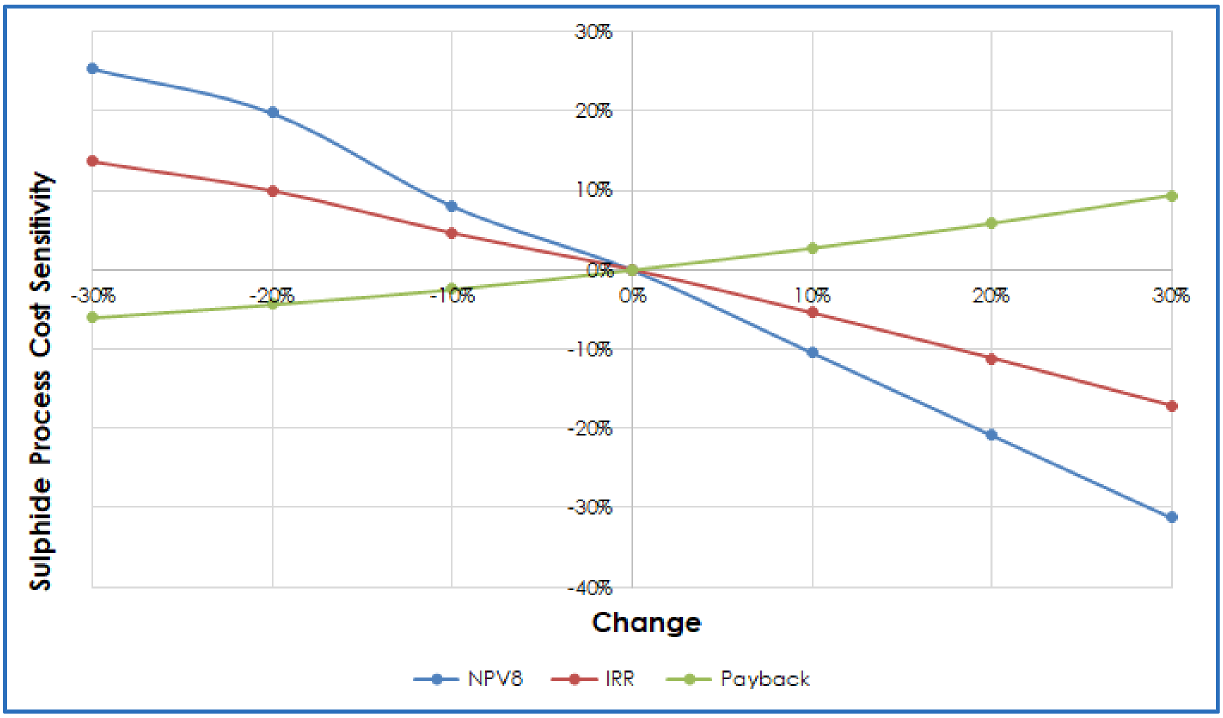

Figure by OreWin, 2019.

Figure 22.7 Sensitivity to G&A Operating Costs – 8% Discount Rate

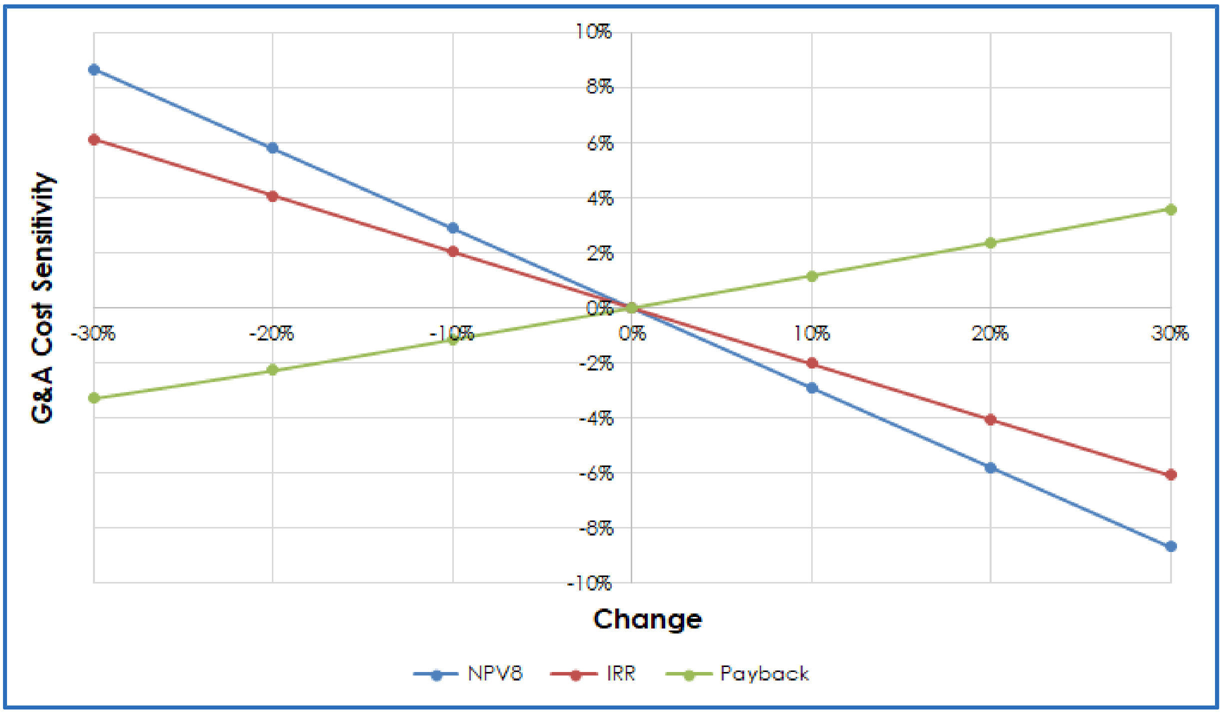

Figure by OreWin, 2019.

Figure 22.8 Sensitivity to Copper Price – 8% Discount Rate

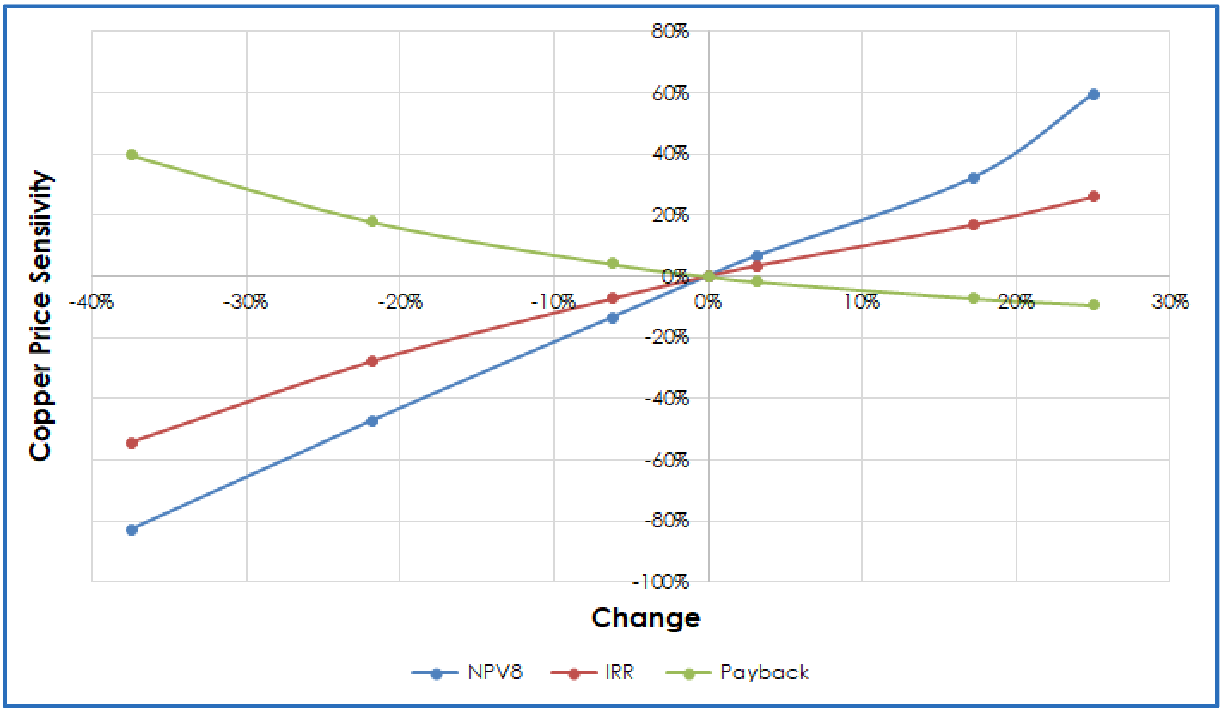

Figure by OreWin, 2019.

Figure 22.9 Sensitivity to Zinc Price – 8% Discount Rate

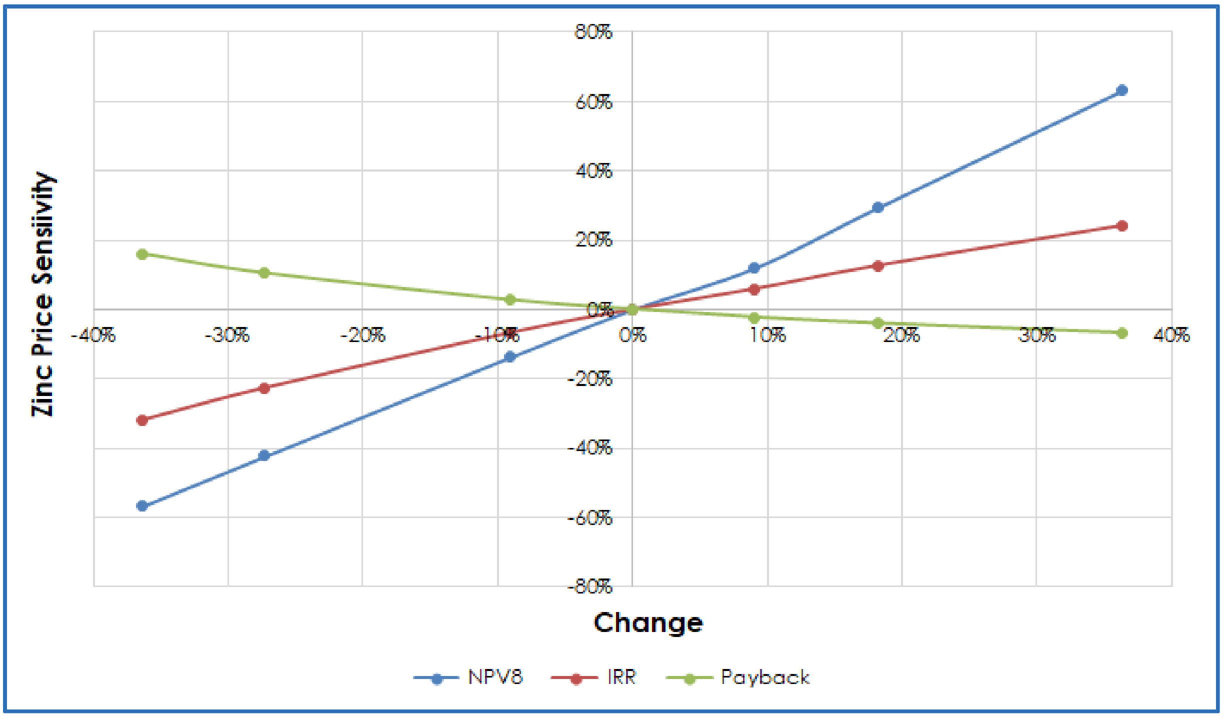

Figure by OreWin, 2019.

Figure 22.10 Sensitivity to Gold Price – 8% Discount Rate

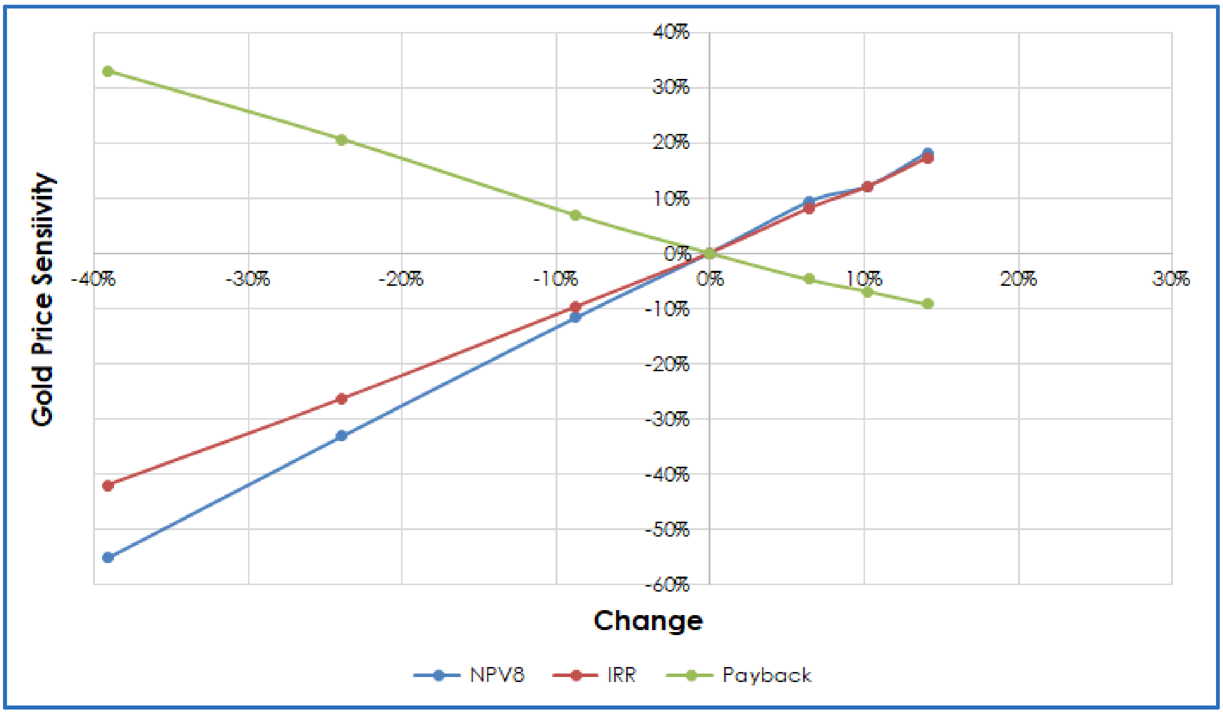

Figure by OreWin, 2019.

Figure 22.11 Sensitivity to Silver Price – 8% Discount Rate

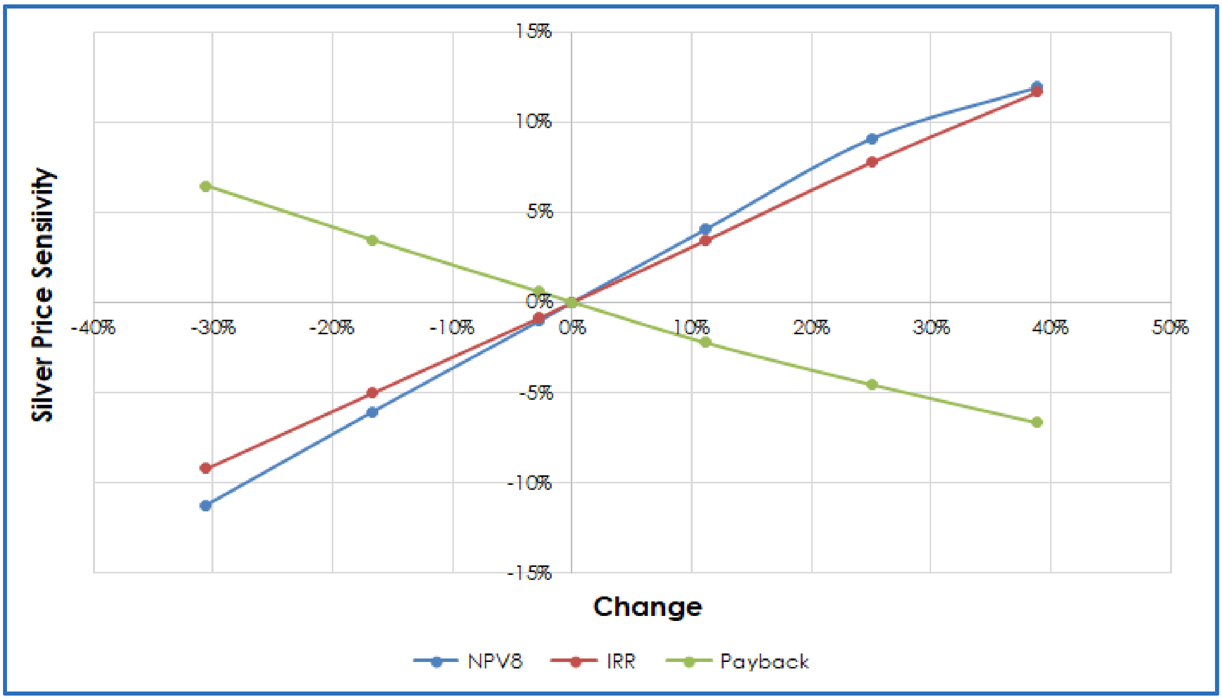

Figure by OreWin, 2019.

Figure 22.12 Sensitivity to Initial Capital Costs – 5% Discount Rate

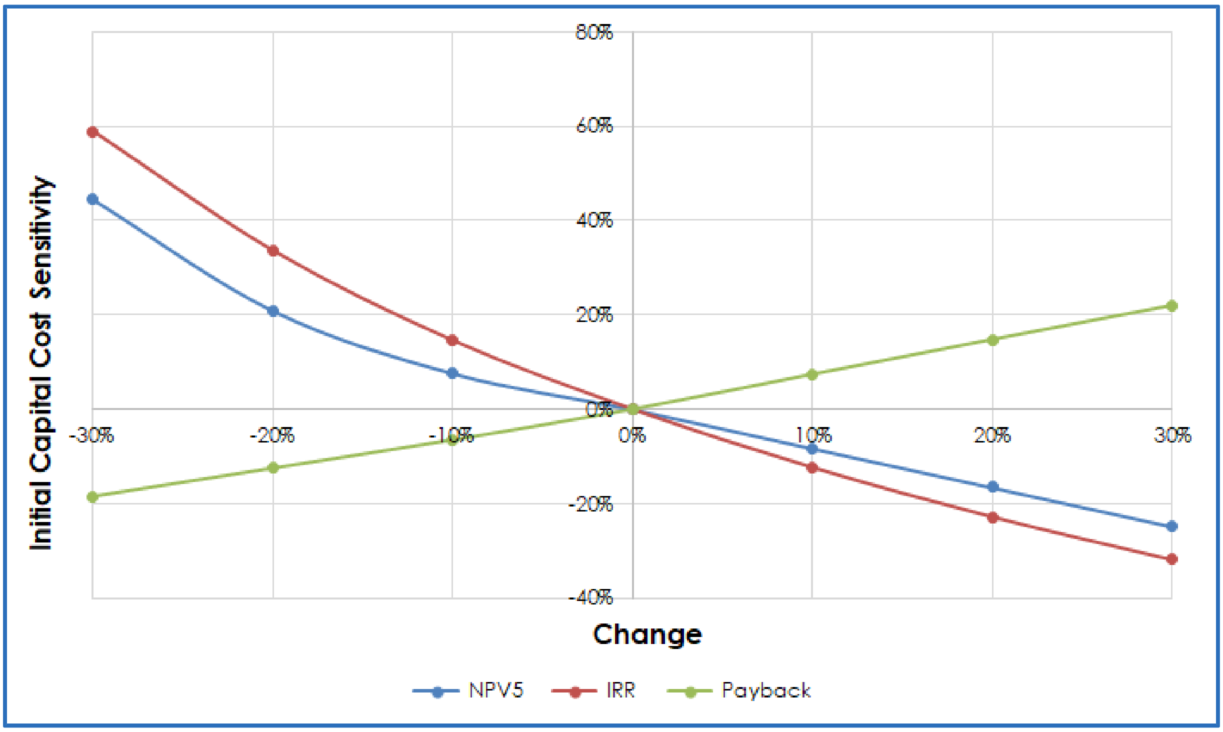

Figure by OreWin, 2019.

Figure 22.13 Sensitivity to Operating Costs – 5% Discount Rate

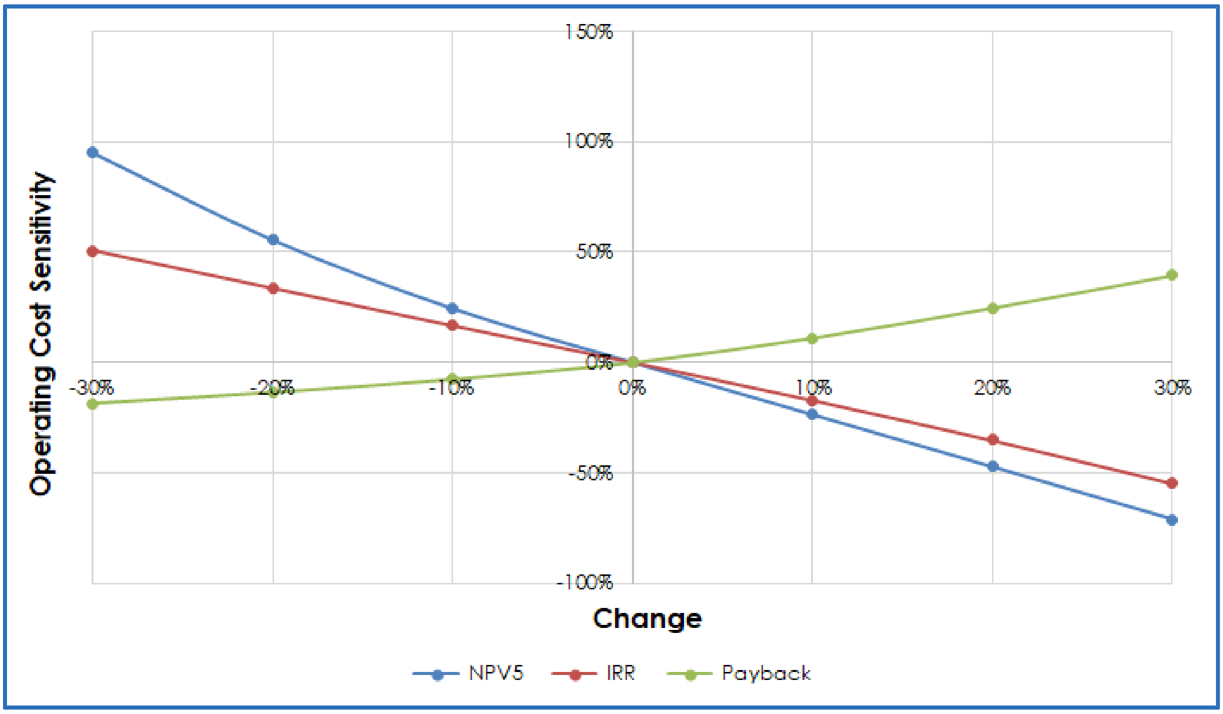

Figure by OreWin, 2019.

Figure 22.14 Sensitivity to Mining Operating Costs – 5% Discount Rate

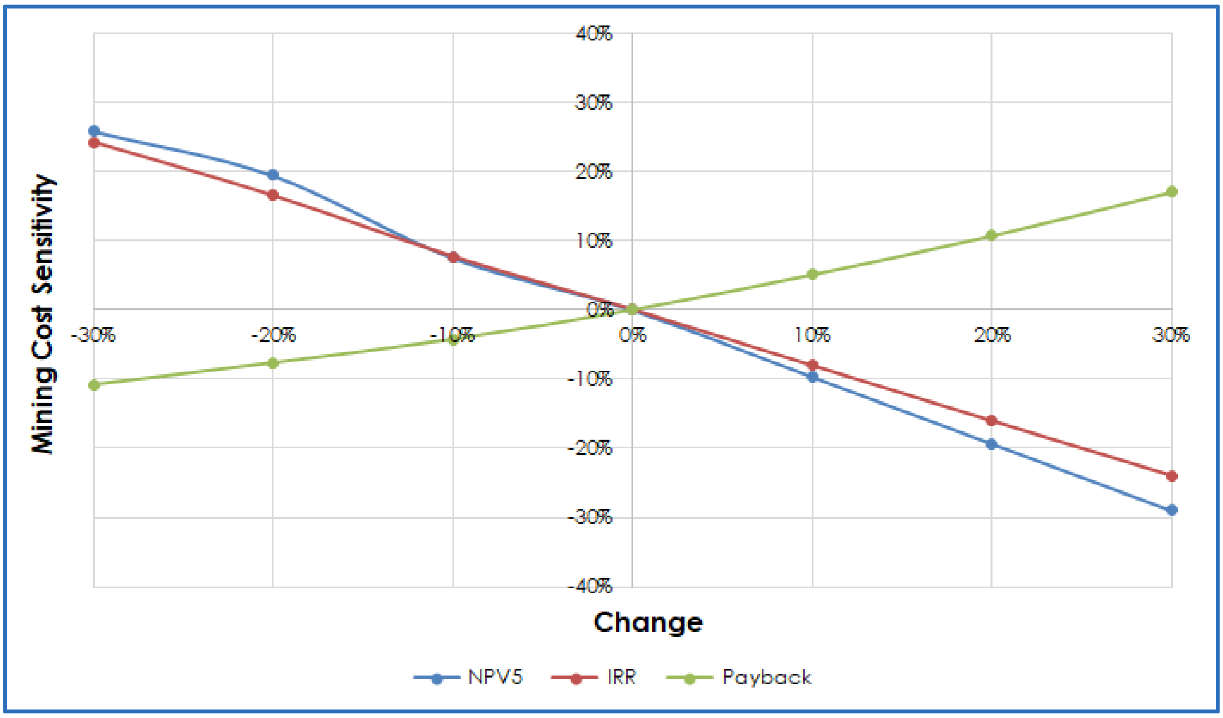

Figure by OreWin, 2019.

Figure 22.15 Sensitivity to Oxide Processing Operating Costs – 5% Discount Rate

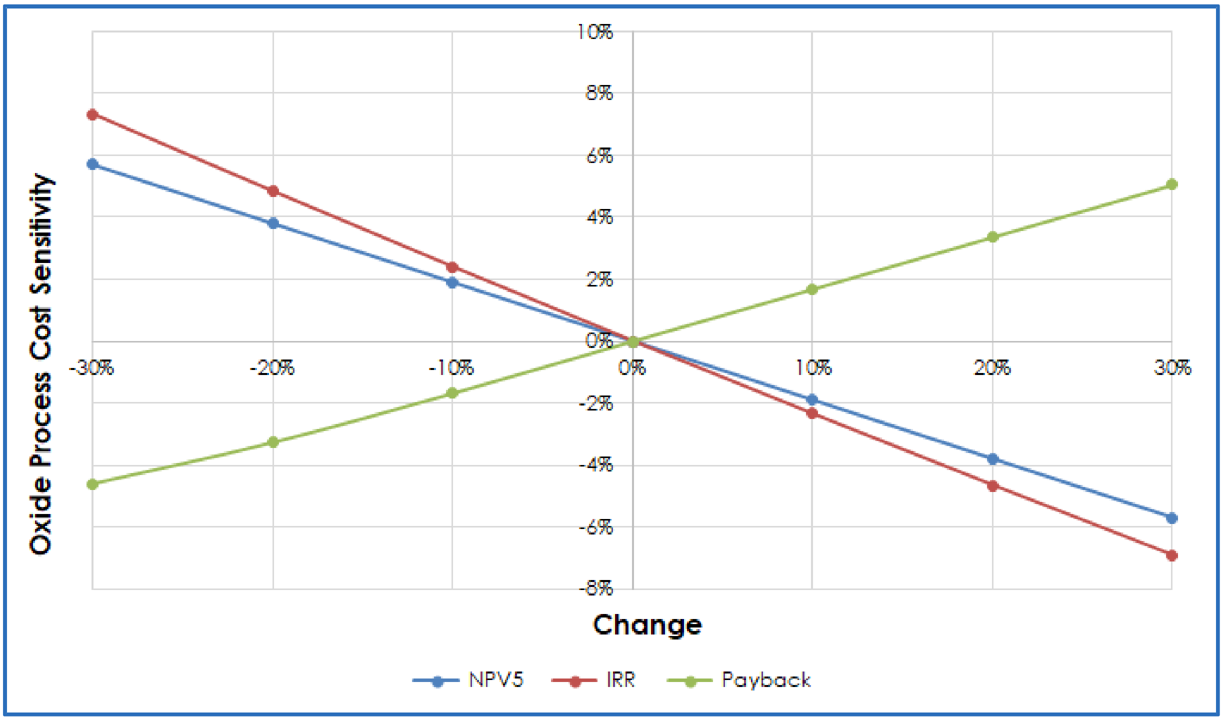

Figure by OreWin, 2019.

Figure 22.16 Sensitivity to Sulphide Processing Operating Costs – 5% Discount Rate

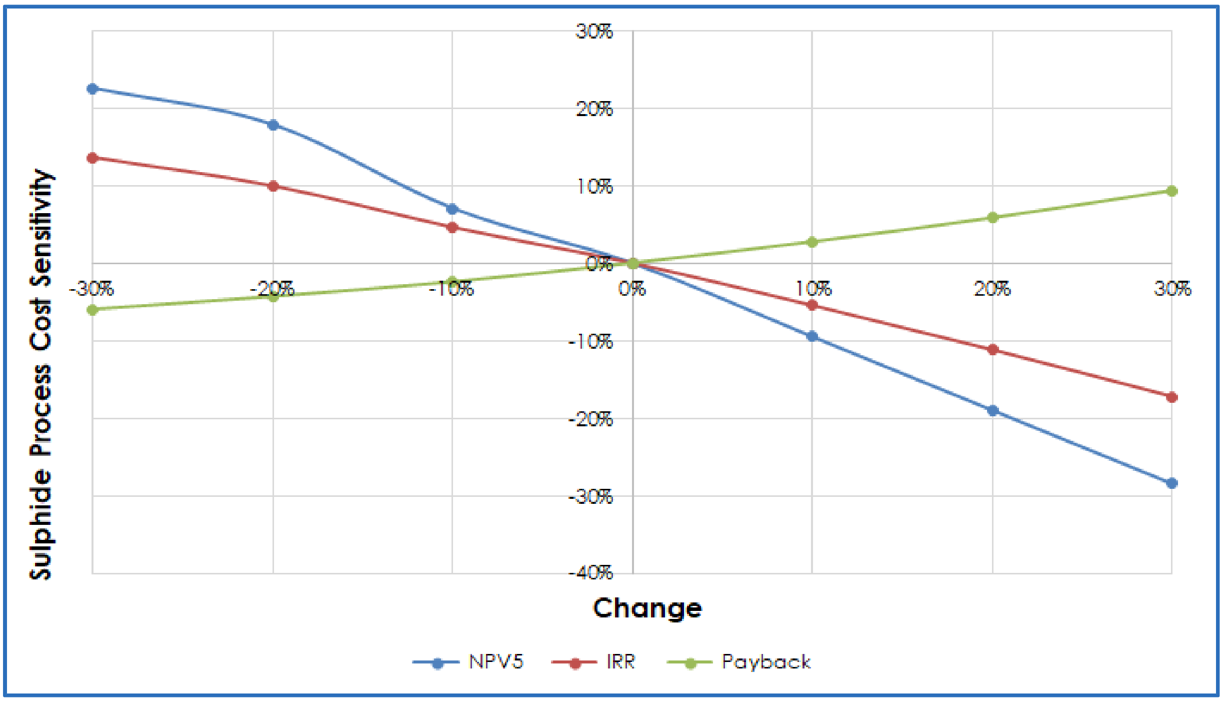

Figure by OreWin, 2019.

Figure 22.17 Sensitivity to G&A Operating Costs – 5% Discount Rate

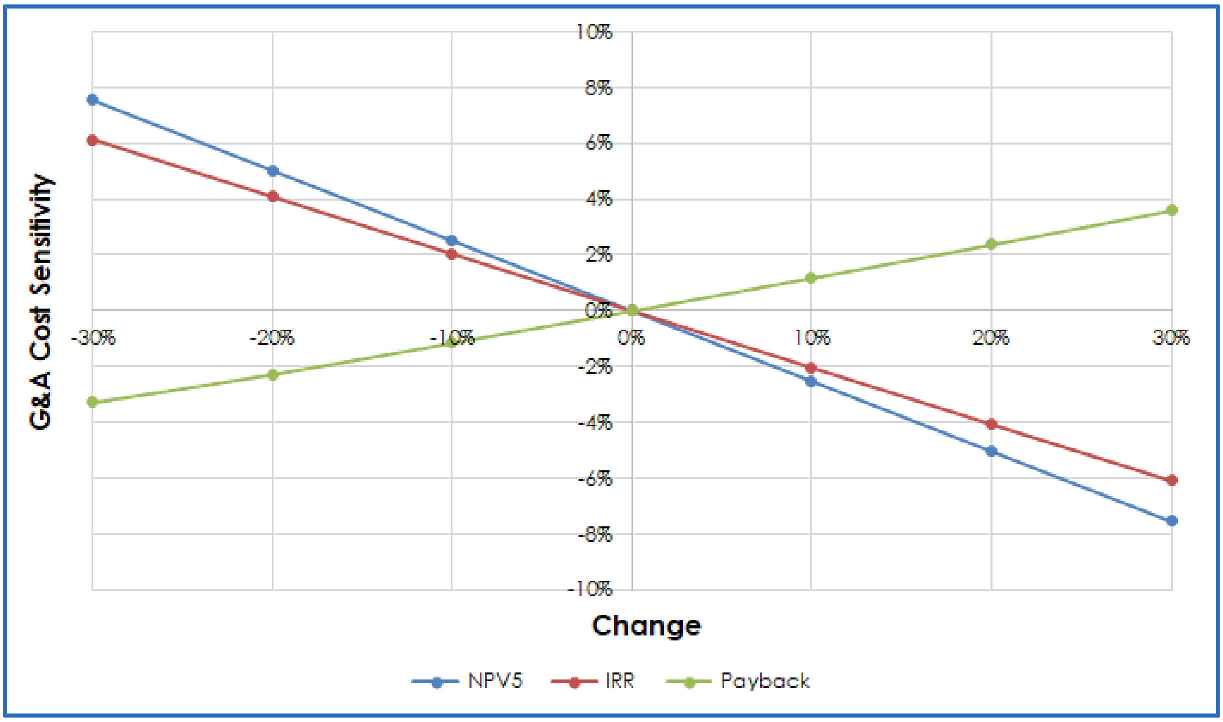

Figure by OreWin, 2019.

Figure 22.18 Sensitivity to Copper Price – 5% Discount Rate

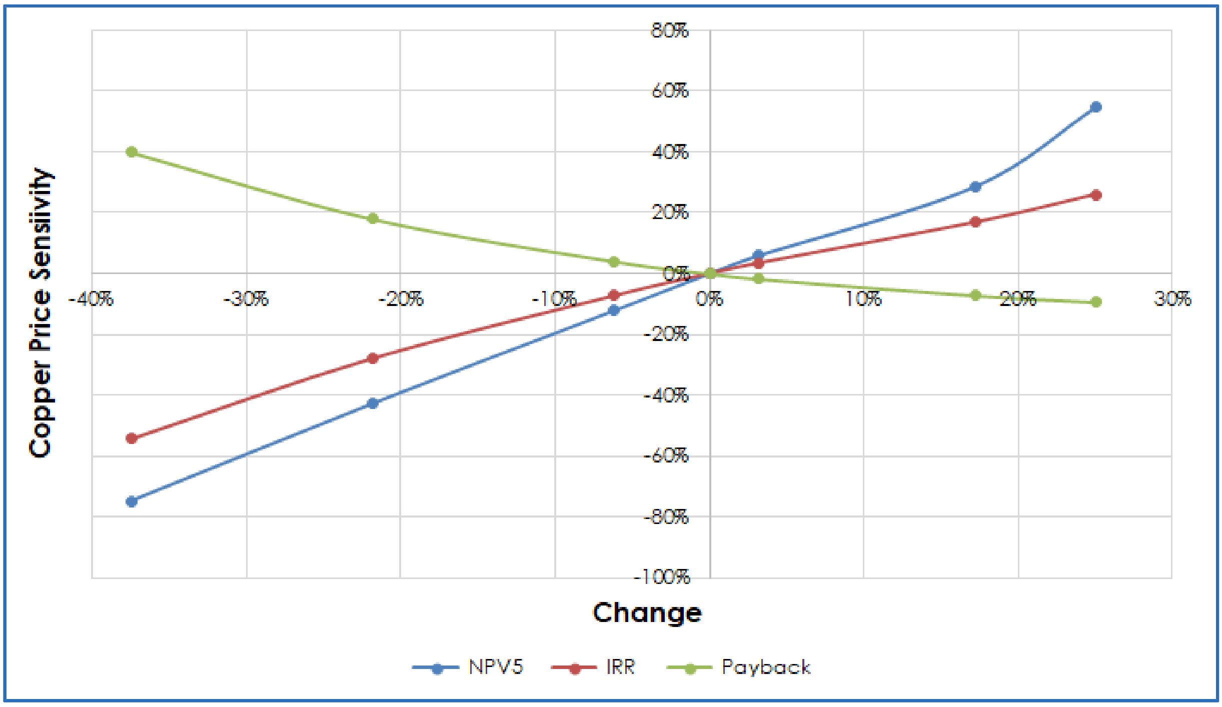

Figure by OreWin, 2019.

Figure 22.19 Sensitivity to Zinc Price – 5% Discount Rate

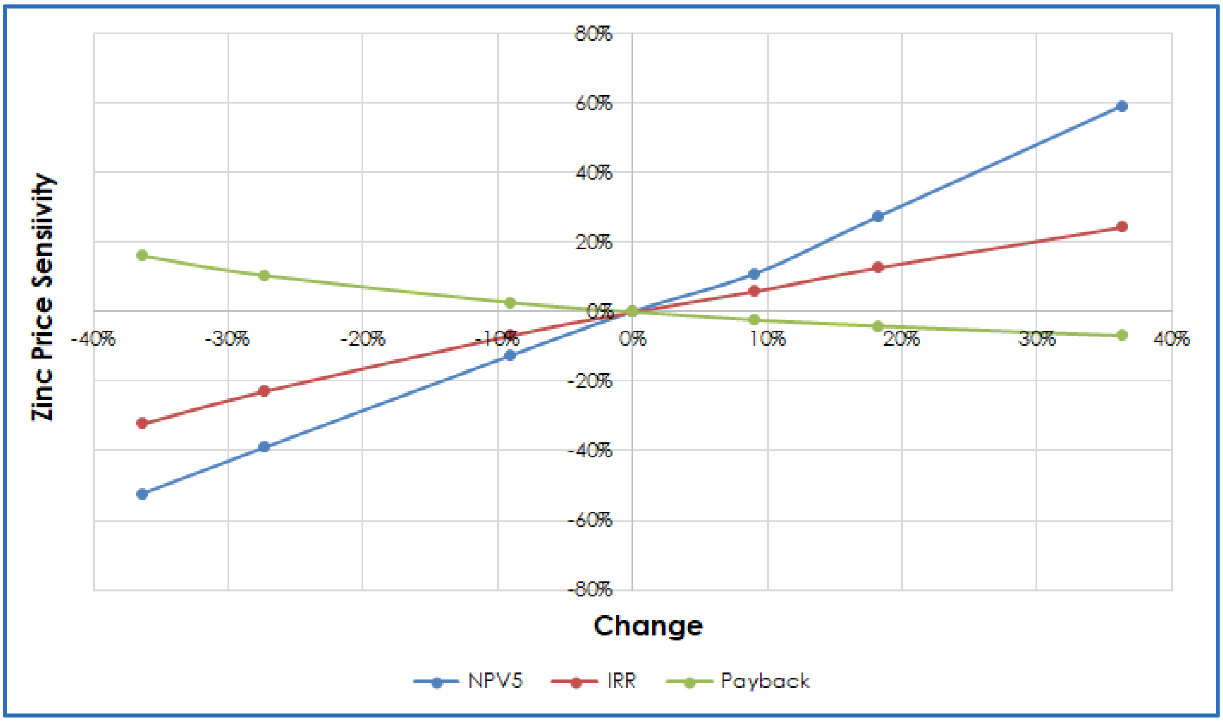

Figure by OreWin, 2019.

Figure 22.20 Sensitivity to Gold Price – 5% Discount Rate

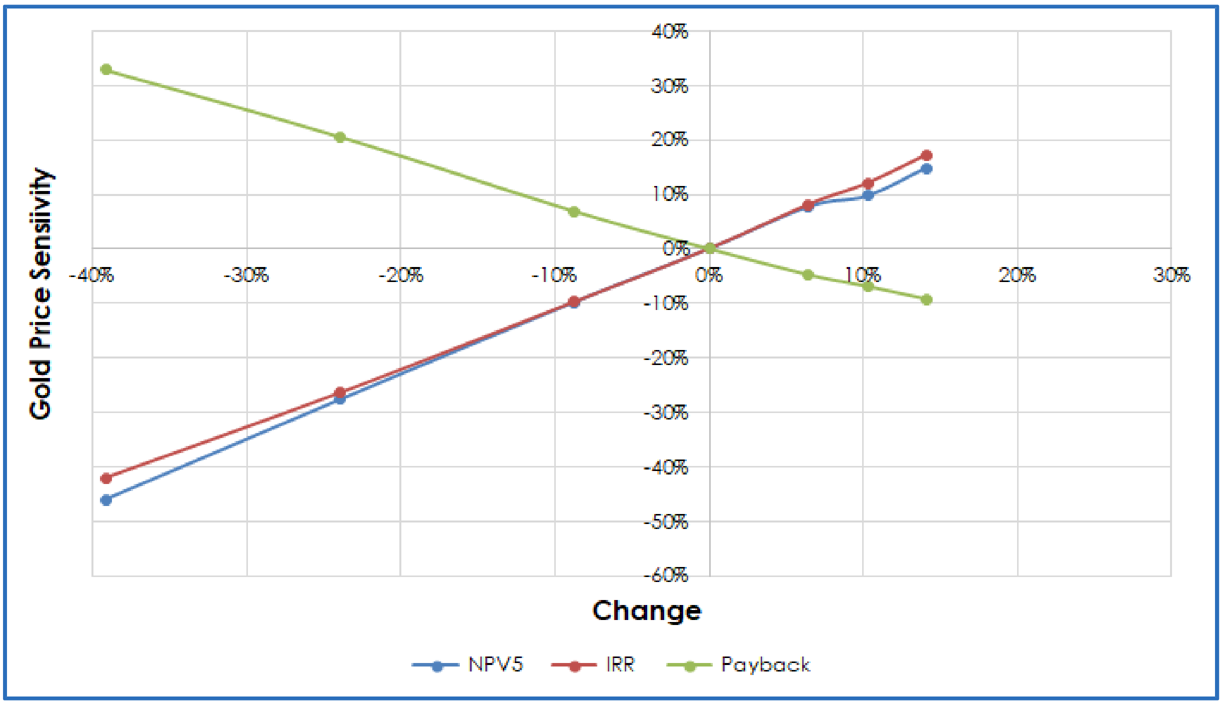

Figure by OreWin, 2019.

Figure 22.21 Sensitivity to Silver Price – 5% Discount Rate

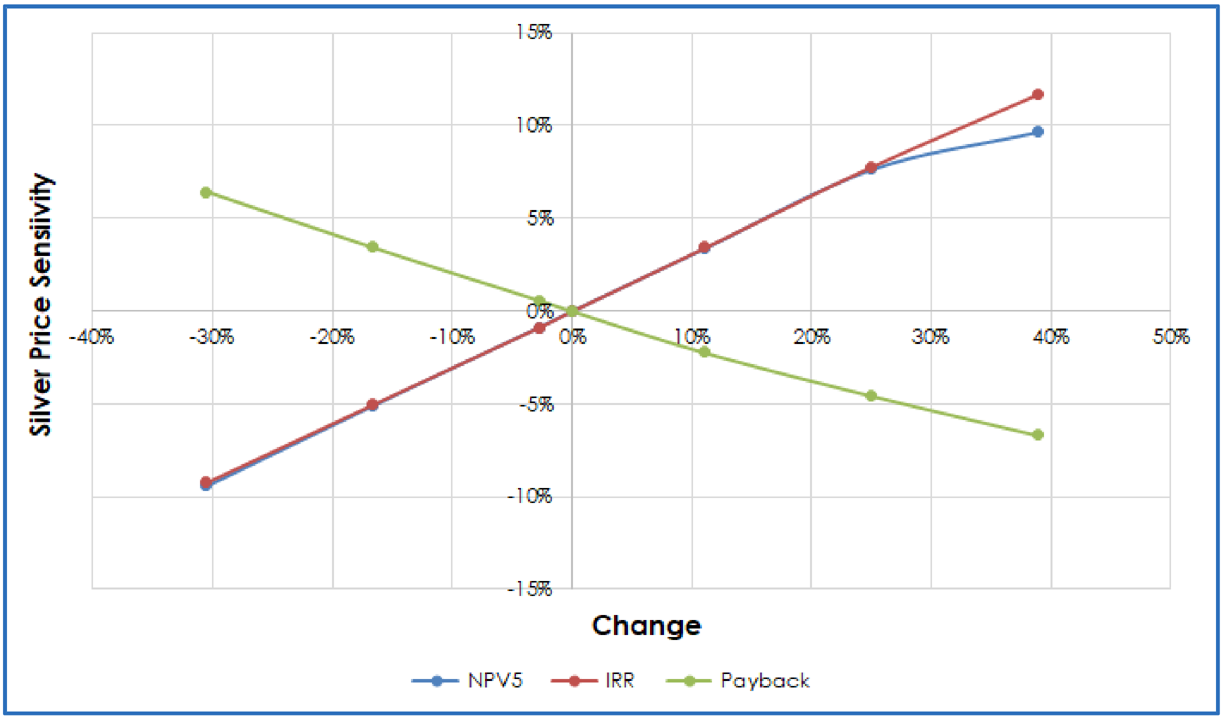

Figure by OreWin, 2019.

23 ADJACENT PROPERTIES

This section not used.

24 OTHER RELEVANT DATA AND INFORMATION

This section not used.

25 INTERPRETATION AND CONCLUSIONS

PFS19 is at a prefeasibility level of accuracy and has estimated Mineral Resources and Mineral Reserves suitable for a PFS. The economic analysis to support the Mineral Reserves shows a positive business case. There are risks associated with developing the project that should to be analysed in future studies with a view to reducing the risks. The key risk areas are as follows.

Mineral Resources

The resource classification categories assigned to the Gediktepe estimates (Measured, Indicated, and Inferred) have, at a global scale, identified different levels of confidence (uncertainty) across the deposit, and this is considered sufficient for prefeasibility assessment. However, these categories do not necessarily reflect variations in confidence at a more-local resolution, which may impact on the shorter term effectiveness, and hence profitability, of eventual mining.

The uncertainty in the mineralogical interpretations may necessitate that sampling for grade control be close-spaced and of a high degree of accuracy. A detailed plan in regard to grade control measures is required. To arrive at the most appropriate grade control strategy, studies into the accuracy and practicality of the various available measures should be undertaken, including, but not limited to, blasthole sampling, RC drillhole sampling, trenching, grab sampling, and portable XRF sampling, as well as methods for obtaining accurate and meaningful mapping data from already-mined benches. The feedback of this information into the grade control model in a timely and accurate way will be very important to ensure that knowledge in regard to the tenor and type of mineralisation that is due to be imminently exposed is available in a usable form when required.

Mining

Ore production will require blending of the enriched ore type and management of stockpiles to minimise oxidation. This will require a coordinated approach to the management and operation of the ore feed from the mine to the mill. The design of the PAG dump needs to be investigated in more detail and this may identify cost reductions while still maintaining a suitable PAG management methodology.

The pit optimisation and design have been prepared using different costs and price assumptions. The sensitivity analysis suggests that the PFS19 pit design is reasonable and suitable for the study. When the pit optimisation and economic analysis are more closely aligned there may be changes to the final pit design, although the shape of the pit is not likely to be drastically different because the geometry of the deposit is unlikely to change significantly. The metal prices used to define the optimisation pit shell are higher than the long-term forecasts used in the economic analysis; any changes to the shape of the pit from a change in metal prices could be offset by lower than expected mining contractor costs.

Process and Metallurgical Testwork

The main risk is that processing of enriched ore presents challenges due to the pre-activation of zinc in situ, resulting in a relatively high proportion of zinc reporting to the copper concentrate. This may be further affected by the weathering of mined ore in stockpiles prior to feeding to the mill. The mining schedule in PFS19 is based on a blending constraint that limits the enriched ore feed to the mill at < 10% but will require enriched stockpiles of up to 40 kt in some months of year-4. If stockpile residence time is not properly managed for both enriched and general sulphide ores the proportion of lower value complex copper concentrate could increase.

Infrastructure

PFS19 has identified the infrastructure requirements for Gediktepe. There remains some options for the location and operation of the infrastructure. These need to be further defined to finalise. They include:

- Surface infrastructure layout
- PAG waste dump location
- Workshops
- Camp options
- Closure plan

26 RECOMMENDATIONS

PFS19 has identified a positive business case and it is recommended that the assessment of the Gediktepe project be continued to a feasibility study level in order to increase the confidence of the estimates and progress the project development. There are a number of areas that need to be further examined and arrangements that need to be put in place to advance the development of the Gediktepe project. The key areas for further work are as follows.

26.1 Mineral Resources

It is recommended that additional work be undertaken in an effort to reduce the uncertainty in the current mineralogical model. This may involve some or all of the following activities:

- Additional, focussed drilling.
- A short-range variability study to attempt to better understand the grade distributions.
- Selected resampling and assaying.
- Review of local geological and mineralogical interpretations.
- Refinement of resource modelling and grade estimation procedures and parameters.

The uncertainty in the mineralogical interpretations may necessitate that sampling for grade control be close-spaced and of a high degree of accuracy. A detailed plan in regard to grade control measures is required. To arrive at the most appropriate grade control strategy, studies into the accuracy and practicality of the various available measures should be undertaken, including, but not limited to, blasthole sampling, RC drillhole sampling, trenching, grab sampling, and portable XRF sampling, as well as methods for obtaining accurate and meaningful mapping data from already-mined benches. The feedback of this information into the grade control model in a timely and accurate way will be very important to ensure that knowledge in regard to the tenor and type of mineralisation that is due to be imminently exposed is available in a usable form when required.

26.2 Mining

The key areas for mining and mine planning that should be addressed in the feasibility study include:

- Update and revise the open pit and waste dump designs based on updated process parameters, pit slopes, metal prices and costs to align the pit optimisation and pit design.
- Assessment of the critical path pre-production forestry and construction activities (clean water pond and TSF) is required to confirm the required lead time prior to commencement of oxide processing.
- Detailed ore production schedules, including the stockpile movements and strategies, should be prepared in the feasibility study to identify the requirements for processing enriched ores and ensure stockpile residence time for sulphide ore is minimised.
- Prepare detailed designs and schedules for the waste dumps, including the PAG dump. Detailed specifications for the PAG dump should be prepared for the dump design,

management, and closure.

- Investigate the possibility of encapsulating the PAG within cells in the main waste dump.
- Obtain updated mining contractor budget pricing based on the final feasibility study mine plan and schedules.

26.3 Process and Metallurgical Testwork

The following testwork is recommended to be carried out for the feasibility study:

Oxide samples

- Variability testing of samples with a range of precious metal head grade, cyanide-soluble copper content, silver-to-gold ratios, spatial and depth locations, and mine schedule composites.
- Investigation of acid washing and elution conditions for removal of copper and zinc, and recovery of gold and silver from loaded carbon.
- Effect of low temperature (climate) on leach extractions and adsorption efficiency.
- Optimisation of leach conditions (cyanide concentration, pulp density, and dissolved oxygen levels).

Sulphide samples

- Variability testing of samples from each ore type with a range of head grade, copper-to-zinc ratios, lead content, spatial and depth locations, and mine schedule composites.
- Investigate the influence of copper-to-zinc ratio on the behaviour of the enriched ore and blends of enriched ore with other sulphide ore types.
- Process water treatment parameters for removal of residual reagent using activated carbon.

26.4 Infrastructure

The following is recommended to be carried out for the feasibility study:

- Optimise surface infrastructure layout.
- Prepare detailed closure planning and costing.
- Prepare a detailed project implementation schedule to cover all the activities from pre-production of the oxide plant through to the post commissioning period of the sulphide plant.
- An on-site camp has been assumed for PFS19. It is recommended that Polimetal undertake traffic and accommodation surveys and trade-off studies to determine if there is suitable local accommodation in the area or if it is practical to transport personnel from the regional centre of Balıkesir.

27 REFERENCES

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