TECHNICAL REPORT

PREFEASIBILITY STUDY

GEDIKTEPE PROJECT, BALIKESIR PROVINCE, TURKEY

Prepared for:

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and

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1.0 SUMMARY

Polimetal Madencilik San. Ve Tic A.S. (Polimetal) assembled a team of consultants to complete a Preliminary Feasibility Study (PFS) for the Gediktepe mining project in Western, Turkey. The project plan utilizes open pit mining to produce gold and silver by heap leaching of oxide mineralization followed by gold, silver, zinc, and copper production by flotation of sulfide mineralization.

The planned production rate for oxide heap leaching is 3,000 tonnes per day (tpd) for three years. Processing of sulfide ore starts in Year 3 at an average rate of 4,500 tpd and ramps up to 6,500 tpd producing a copper concentrate and a zinc concentrate.

The project team was comprised of the three consulting companies listed below and the engineering staff at Polimetal. Polimetal provided input regarding infrastructure costs and owner's costs.

- Resource Development Inc. (RDi) for process testing and design
- SRK Consulting (U.S.) Inc. (SRK) for pre-feasibility level heap leach pad and tailing storage facility designs.
- Independent Mining Consultants, Inc. (IMC) for resource modeling, mine planning

The deposit is in the Balikesir province, roughly 42 km straight line distance southeast of the town of Balikesir. It is about 17 km south-southwest of the town of Dursunbey. A location map is provided in Section 4.0.

The Gediktepe project is a massive sulfide deposit hosted in metamorphic schist units. The upper portion of the deposit has been oxidized by surface and ground water. The oxide zone is nearly devoid of base metals. The sulfide zone is polimetallic with economic values of zinc, copper, gold and silver. The major economic minerals are sphalerite and chalcopyrite. Pyrite is ubiquitous.

The data base for this project reflects all drilling completed through August 5, 2015. The mineral resource is based on 487 holes that were drilled by Polimetal. Reverse circulation drilling was utilized for 184 holes and the remaining 303 holes were by diamond drilling. A nearest neighbor comparison of the two drill types demonstrated that both types of drill data are acceptable for estimation purposes. IMC has reviewed and verified the drill hole data, including the QAQC information. As a result of the review and verification, IMC and the qualified person, John Marek, find that the drill hole data is acceptable for the determination of mineral resources and mineral reserves.

The mineral resources were established using a computer based block model to estimate the in-ground mineralization. The component of that mineralization that has reasonable prospects of economic extraction was estimated using the floating cone algorithm. The economic and process input information to the floating cone are summarized in Sections 14 and 15.

The Qualified Person for the mineral resources is John Marek of IMC. The mineral resource could change as additional drilling is completed and more detailed process recovery information becomes available. Metal prices could materially change the resources in either a positive or negative way. Table 1-1 summarizes the mineral resources. The stated mineral resources include the mineral reserve.

Table 1-1
Gediktepe, Mineral Resources, 1 June 2016

			Head Grades				Contained Metal			
Material Type Classification	NSR Cutoff $/t	Tonnages ktonnes	Au gm/t	Ag gm/t	Cu %	Zn %	Au koz	Ag koz	Cu klb	Zn klb
Oxides										
Measured	$11.70	1,722	2.645	66.5	0.12	0.16	146.4	3,680		
Indicated	$11.70	2,110	2.561	71.0	0.18	0.35	173.7	4,817		
Meas+Ind.	$11.70	3,832	2.599	69.0	0.15	0.26	320.2	8,497		
Inferred	$11.70	213	1.574	63.1	0.13	0.17	10.8	432		
Sulfides										
Measured	$15.67	12,027	0.777	28.5	1.00	1.89	300.4	11,030	263,824	501,133
Indicated	$15.67	20,180	0.773	30.1	0.85	1.95	501.5	19,506	378,158	867,540
Meas+Ind.	$15.67	32,207	0.774	29.5	0.90	1.93	802.0	30,536	641,982	1,368,673
Inferred	$15.67	1,685	0.807	31.7	0.98	1.80	43.7	1,719	36,256	66,866
Oxides+Sulfides										
Measured	11.70/15.67	13,749	1.011	33.3	0.89	1.67	446.9	14,710	263,824	501,133
Indicated	11.70/15.67	22,290	0.942	33.9	0.79	1.80	675.3	24,323	378,158	867,540
Meas+Ind.	11.70/15.67	36,039	0.968	33.7	0.82	1.75	1,122.1	39,033	641,982	1,368,673
Inferred	11.70/15.67	1,898	0.893	35.3	0.88	1.62	54.5	2,151	36,256	66,866

Notes:
Mineral resources include the mineral reserve
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
The Qualified Person for the Mineral Resource is John Marek, RM-SME
Summation errors are due to rounding
Metal Prices Used: Gold: $1,200/oz. Copper: $3.00/lb. Zinc: $1.20/lb. Silver: $18/oz.
Tonnages are reported in 000's of metric tonnes
Contained precious metal reported in 000's troy ounces, contained base metal reported in 000's of lbs.
Contained copper and zinc not reported for oxides. No recovery potential is expected for these metals in the oxide zone
Copper and zinc grades are reported in the oxide zone because they have an impact on process plant design and costs
Floating cone inputs used to define Resource:
-Mining Cost=$1.47/tonne
-G&A Costs=$4.78/tonne ore
-Oxide Processing: $6.92/tonne, Sulfide Processing: $10.89/tonne
-Pit Slope Angle: 48°

The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be realized or that they will convert to mineral reserves. The contained copper and zinc within the oxide zone are not presented on the statement of mineral resources because there is no process planned to produce those metals in the oxide zone. The grades of copper and zinc are shown because their presence has an impact on the design of the oxide process plant and oxide processing costs

The Gediktepe deposit will be mined by conventional open pit hard rock mining methods. Polimetal currently plans to utilize a contract mining company to move the ore and waste from the mine. Compared with typical mining practices in North America, Turkish contractors generally utilize small back hoe loading units with relatively small haul trucks. The mine geometries have been designed with 12 meter haul roads and minimum mining widths of ~70 meters.

The Gediktepe PFS plan produces oxide mineralization to a heap leach facility at the rate of 3,000 tpd for just over 3 years. After that period, the minor oxide material that is incurred during sulfide mining will be wasted.

The minor sulfide ores that are incurred in Preproduction and Year 1 are wasted. Sulfide ore that is incurred in Year 2 is stockpiled for processing with fresh ore in Year 3. Both oxide and sulfide ores are processed in Year 3. The crushing circuit is sufficiently large that both oxide and sulfide feed material can be crushed through the circuit on a short term campaign basis. A tripper is planned downstream of crushing so that oxide material goes to agglomeration and then on to the heap leach pad by conveyor. The tripper can also send the sulfide feed to the grinding circuit followed by floatation.

The PFS mine production schedule is summarized on Table 1-2. Total mined material ramps up to 18,500,000 tonnes per year (52,857 tpd) inclusive of both ore and waste. The mine and plant are assumed to operate 350 days per year. Table 1-3 summarizes the feed to the planned process plant and illustrates the recovered metal that is planned for production.

As a result of the mine and process plans that are summarized on Tables 1-2 and 1-3, the total of all proven and probable category material that is planned for processing constitutes the mineral reserve that is presented on Table 1-4. Metal prices used for determining the mineral reserve are about 17% lower than metal price inputs defining the mineral resource. Alternative metal prices were used in the financial analysis presented later. The Qualified Person for the mineral reserve is John Marek of IMC. The mineral reserve could change as more drilling and engineering is completed. Metal prices could materially change the mineral reserve in a positive or negative way. Changes to operating costs could also impact the statement of reserves.

The payable copper and zinc metal within the oxide zone are not presented on the statement of mineral reserves because there is no process planned to produce those metals in the oxide zone. The grades of copper and zinc are shown because their presence has an impact on the design of the oxide process plant and oxide processing costs.

The contained mercury and arsenic in the ore is reported on Tables 1-2 and 1-3 as a check to understand the impact of those elements on processing and marketing of concentrates. Process testing to date indicates that there is minimal risk of smelter penalties due to either mercury or arsenic. Mercury and arsenic are not shown on the statement of reserves as they have no economic impact positive or negative on the project.

The metallurgical test work indicates that the oxide ores will be treated by heap leach and the sulfide ore will be floated to produce two concentrates for zinc and copper. The process flowsheets and associated recoveries are discussed in Section 17.

The flowsheet for the heap leach process is shown on Figure 17-1. The run-of-mine ore will be crushed in three crushing stages to produce a product with P_{100} of 19 mm. A 19 mm opening screen can be used on the feed to the secondary crusher to remove the finished product. The secondary crusher product will be recycled back to the screen. The crushed ore will be discharged onto a conveyor which will convey the ore to the agglomerating drum.

The flowsheet for processing sulfide ore is presented on Figure 17-2. The run-of-mine sulfide ore will be crushed in 3 stages and ground to P_{80} of 325 mesh using ball mills in the comminution circuit.

Following the grind, a pre-float will be applied to remove talc and fibrous silicates. That will be followed by copper flotation and then zinc flotation to produce two concentrates. Each of the two circuits will incur regrind and 4 stages of cleaning to produce the final concentrates.

The project will require the development of a number of infrastructure items in order to operate. The current approach that utilizes a combination of oxide heap leaching followed by sulfide flotation will require both heap leach and tailing storage facilities.

The major infrastructure items are:

1) Heap Leach Pad (HLP): A PFS level design of a heap leach facility was completed by SRK that has 3.6 million tonnes of capacity. The HLP is located immediately southwest of the pit and process plant.
2) Tailing Storage Facility(TSF): A PFS level design of a tailing storage facility was completed by SRK with a capacity of 22 million tonnes of flotation tailing. The selected facility is located southwest of the pit and lower in the valley. The TSF is planned for 3 phases of expansion over the mine life using downstream construction.
3) Waste Storage Facility (WSF): The waste storage facility is east of the pit and will be discussed in Section 16 regarding the mine plan.
4) Water Supply: A water supply system will be required for the project. The water supply system will include a freshwater pond and a water treatment plant. On site testing for water resources is now underway at the project site.
5) Power Transmission Line: A power supply system has been planned by Polimetal that incorporates a new power line from Dursunbey to site following an existing power line route. Cost estimates were developed by Polimetal working with the local Turkish power authorities.
6) Bypass road construction: A bypass road will be constructed so that mine traffic will not have to travel through the town of Haciömerderesi.
7) Mine Buildings: A mine camp will be constructed southwest of the project site. Mine site offices, laboratories, warehouses etc. will also be erected southwest of the mine pit.

Figure 1-1 illustrates the overall end of mine life general arrangement drawing. The pit, plant area, waste storage, heap leach pad, and tailing facilities are shown in their final configuration as a result of this PFS mine schedule.

Table 1-2
Gediktepe Mine Production Schedule

Mined Material																				
	CUTOFF	Oxide Mined Material								CUTOFF	Sulfide Mined Material									
Years	NSR $/tonne	ORE ktonnes	NSR $/tonne	Gold gm/t	Silver gm/t	Copper %	Zinc %	Mercury ppm	Arsenic ppm	NSR $/tonne	ORE ktonnes	NSR $/tonne	Gold gm/t	Silver gm/t	Copper %	Zinc %	Mercury ppm	Arsenic ppm	WASTE ktonnes	TOTAL ktonnes
PreProd	$15.16	92	$34.63	1.25	32.3	0.37	0.86	2.7	1,486										257	349
Y 1	$15.16	886	$68.17	2.15	68.4	0.22	0.50	5.6	1,091	$14.55	138	$51.68	0.81	26.9	0.76	1.99	2.1	551	4,740	5,764
Y 2	$15.16	1,048	$117.65	3.73	85.9	0.13	0.21	7.5	1,566	$14.55	379	$49.92	0.77	28.2	0.79	1.72	1.9	632	9,068	10,495
Y 3	$15.16	1,048	$94.92	2.99	76.7	0.10	0.10	6.1	1,420	$14.55	1,193	$65.40	0.90	34.5	1.25	1.66	2.3	575	9,317	11,558
Y 4	$15.16	149	$104.18	3.05	111.1	0.14	0.16	6.6	1,409	$14.55	2,275	$78.33	0.95	30.8	1.67	1.64	1.8	656	10,576	13,000
Y 5	$15.16	71	$55.61	1.92	36.1	0.10	0.07	4.3	1,770	$14.55	2,275	$80.68	0.97	38.3	1.42	2.62	2.1	681	16,154	18,500
Y 6	$15.16	14	$67.98	2.40	32.3	0.09	0.12	4.2	1,415	$14.55	2,275	$63.44	0.95	40.0	0.77	2.95	3.3	789	16,211	18,500
Y 7	$15.16									$14.55	2,275	$58.15	0.87	36.1	0.75	2.55	2.4	556	16,225	18,500
Y 8	$15.16									$14.55	2,275	$67.46	1.18	44.8	0.81	3.00	2.8	625	16,225	18,500
Y 9	$15.16									$14.55	2,275	$65.73	1.06	42.9	0.81	2.93	2.5	582	15,090	17,365
Y 10	$15.16	20	$27.38	0.98	32.1	0.20	0.07	1.9	706	$14.55	2,275	$61.62	1.00	31.6	0.99	2.05	2.1	767	8,308	10,603
Y 11										$14.55	2,275	$48.68	0.72	27.9	0.73	1.84	1.4	634	3,756	6,031
Y 12										$14.55	1,920	$51.37	0.65	25.6	0.79	2.00	1.2	597	1,479	3,399
Y 13																				
Y 14																				
Y 15																				
TOTAL	$15.16	3,328	$92.34	2.92	76.4	0.15	0.26	6.3	1,383	$14.55	21,830	$63.90	0.92	35.3	0.98	2.35	2.2	650	127,406	152,564

Sulfide Material Mined in Year 1 and Oxide Material Mined in Years 5-10 assumed to be waste

Table 1-3
Gediktepe Process Production Schedule

	Heap Leach Material												Mill Feed Material															
	Oxide Feed Material								Contained Metal		Payable Metal		Sulfide Feed Material								Contained Metal				Sulfide Payable Metal			
Years	ORE ktonnes	NSR $/tonne	Gold gm/t	Silver gm/t	Copper %	Zinc %	Mercury ppm	Arsenic ppm	Gold Ozx1000	Silver Ozx1000	Equation Au Kozs	44% Ag Kozs	ORE ktonnes	NSR $/tonne	Gold gm/t	Silver gm/t	Copper %	Zinc %	Mercury ppm	Arsenic ppm	Gold Ozx1000	Silver Ozx1000	Copper Lbs x 1000	Zinc Lbs x 1000	23% Au Kozs	19% Ag Kozs	67% Cu Mlbs	69% Zn Mlbs
PreProd																												
Y 1	978	$65.01	2.06	65.0	0.24	0.53	5.3	1,129	64.8	2,045.1	50.8	901.9																
Y 2	1,048	$117.65	3.73	85.9	0.13	0.21	7.5	1,566	125.8	2,893.6	105.3	1,276.1																
Y 3	1,048	$94.92	2.99	76.7	0.10	0.10	6.1	1,420	100.9	2,584.0	83.4	1,139.6	1,572	$61.66	0.87	33.0	1.14	1.68	2.2	589	43.9	1668.4	39,367	58,112	10.1	334.4	26.29	40.26
Y 4	149	$104.18	3.05	111.1	0.14	0.16	6.6	1,409	14.6	532.3	12.2	234.8	2,275	$78.33	0.95	30.8	1.67	1.64	1.8	656	69.3	2254.7	83,809	82,254	15.4	421.8	55.97	56.98
Y 5													2,275	$80.68	0.97	38.3	1.42	2.62	2.1	681	70.6	2803.8	71,220	131,406	15.5	509.4	47.56	91.03
Y 6													2,275	$63.44	0.95	40.0	0.77	2.95	3.3	789	69.5	2922.4	38,469	147,958	15.8	542.9	25.69	102.50
Y 7													2,275	$58.15	0.87	36.1	0.75	2.55	2.4	556	63.6	2638.0	37,616	127,896	14.6	497.8	25.12	88.60
Y 8													2,275	$67.46	1.18	44.8	0.81	3.00	2.8	625	85.9	3273.7	40,475	150,465	20.3	633.1	27.03	104.23
Y 9													2,275	$65.73	1.06	42.9	0.81	2.93	2.5	582	77.3	3141.4	40,575	146,954	17.9	603.5	27.10	101.80
Y 10													2,275	$61.62	1.00	31.6	0.99	2.05	2.1	767	72.8	2312.0	49,854	102,818	17.2	438.7	33.29	71.23
Y 11													2,275	$48.68	0.72	27.9	0.73	1.84	1.4	634	52.4	2044.3	36,463	92,285	12.1	389.4	24.35	63.93
Y 12													1,920	$51.37	0.65	25.6	0.79	2.00	1.2	597	39.8	1581.1	33,567	84,657	8.8	278.3	22.42	58.65
Y 15																												
Total	3,223	$93.66	2.95	77.7	0.15	0.27	6.3	1,378	306.2	8,055.1	251.6	3,552.3	21,692	$63.98	0.93	35.3	0.99	2.35	2.2	650	645.1	24,639.9	471,416	1,124,806	147.7	4,649.4	314.80	779.21

Sulfide Material Mined in Year 1 and Oxide Material Mined in Years 5-10 assumed to be waste

Sulfide Payable Recoveries include both Process Plant Recovery and Smelter Payable Estimates.

Table 1-4
Gediketepe Mineral Reserves, 1 June 2016

Classification	Cutoff NSR $/Tonne	Oxide Mineral Reserves: Oxide Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Payable Metal: Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	15.16	1,456	2.98	74.7	0.12	0.17	118.0	1,541.4		
Probable	15.16	1,767	2.93	80.3	0.18	0.35	133.6	2,010.9		
Proven+Probable	15.16	3,223	2.95	77.7	0.15	0.27	251.6	3,552.3		

Classification	Cutoff NSR $/Tonne	Sulfide Mineral Reserves: Sulfide Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Payable Metal: Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	14.55	10,425	0.84	31.0	1.04	2.05	64.3	1,924.6	160.2	326.6
Probable	14.55	11,267	1.00	39.3	0.93	2.63	83.4	2,724.8	154.6	452.6
Proven+Probable	14.55	21,692	0.93	35.3	0.99	2.35	147.7	4,649.4	314.8	779.2

Classification	Cutoff NSR $/Tonne	TOTAL MINERAL RESERVES: Total Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Payable Metal: Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	15.16/14.55	11,881	1.11	36.3	0.93	1.82	182.3	3,466.0	160.2	326.6
Probable	15.16/14.55	13,034	1.26	44.9	0.83	2.32	217.0	4,735.6	154.6	452.6
Proven+Probable	15.16/14.55	24,915	1.19	40.8	0.88	2.08	399.3	8,201.7	314.8	779.2

Notes:

Mineral Reserve Based on Metal Prices of:
$1,000/oz Gold, $15.00/oz Silver, $2.50/lb Copper, $1.00/lb Zinc
Payable Metal is not shown for copper and zinc in the oxide zone because there is no plan to recover copper or zinc from the oxide zone. Their grades are shown because copper and zinc have an impact on the design of the oxide process and oxide process costs.
The Qualifed Person for the Mineral Reserve is John Marek, RM-SME
Pit slope angles are 48 degrees in fresh rock and 42 degrees in weathered rock
Ktonnes are 1000 metric tonnes
Mlbs are millions of pounds of copper and zinc metal
Kozs are 1000 troy ounces of gold and silver.

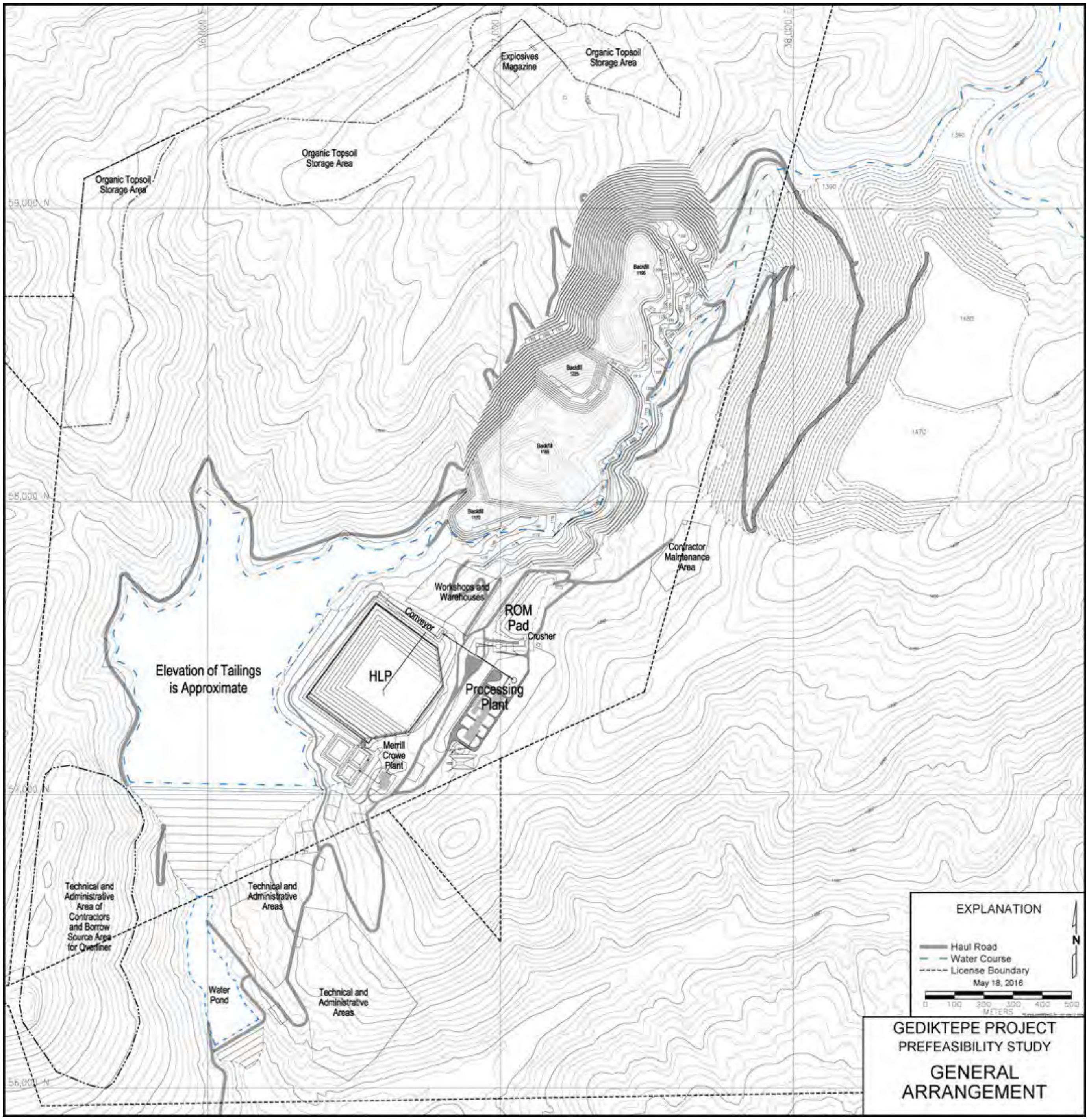


Figure 1-1
General Arrangement at End of PFS Mine Life

Operating Costs

Operating costs for each component of the project were estimated by the project team. Those costs were combined into the financial analysis that is summarized in Section 22.

The operating costs include the costs of mining, processing, and G&A costs. The average operating costs over the life-of-mine by category are provided on Table 1-5.

Mine operating costs are based on a budgetary quote from a Turkish contract mining company. Supervisory, engineering, and ore control costs from Polimetal staff have been added to the contract mining costs. Mine road construction, and topsoil removal are included in the mine operating costs.

Process operating costs were estimated from first principals by RDi. Concentrate treatment and refining costs are based on current typical costs provided by commodity traders.

General and Administrative (G&A) costs were estimated from first principals and include all mine site costs not included in mining or processing costs. This cost covers administration costs and staff, camp costs, employee transportation, government permits, and other necessary expenses.

All costs (operating and capital) are presented in 4th quarter 2015 U.S. Dollars. Costs in Turkish Lira were converted to U.S. Dollars at the exchange rate of: 3.00 Turkish Lira / U.S. Dollar.

Table 1-5
Gediktepe Operating Cost by Category

OPCOST Category	Unit Cost	Units	Total Cost ($000's)
Mining	1.45	$/tonne material	221,126.5
Oxide Ore Processing	9.51	$/tonne ore	30,640.8
Sulfide Ore processing	11.88	$/tonne ore	257,678.7
Site Wide G&A	7.45	$/tonne ore	185,661.4
		Total:	695,107.5

The average mining cost per tonne ore processed is: $8.87 per tonne. This equates to an average operating cost of $25.83/tonne of oxide ore and $28.20/tonne of sulfide ore.

Capital Cost

Capital costs have been estimated by the three consultants (RDi, SRK and IMC) and the Polimetal staff.

Due to the use of a mining contractor, there are no capital costs for mine mobile equipment. The mine preproduction stripping is shown as a capital cost.

The initial process plant costs during preproduction are for the construction of the oxide processing facilities. The large sustaining plant capital cost shown in Years 1 and 2 is the capital cost for the construction of the sulfide process plant.

Infrastructure costs include the heap leach pad, and the tailing storage facility that is required for the PFS mine schedule. All infrastructure items on page 1-4 are included.

Table 1-6 summarizes the project capital costs.

Table 1-6
Estimated Project Capital Costs, 4th Quarter 2015 USD x 1000

Cost or Income Item		Totals Project Life Costs x1000	Capital Costs in Years Shown, USD x 1000										
			Preprod -2	Preprod -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 13
Capital Costs													
Initial Capital Costs													
Plant		$ 46,381.2	$ -	$ 46,381.2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure		$ 41,972.5	$ 6,089.30	$ 35,883.16	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Site Investigation and Proj. Eng.		$ 6,900.0	$ 6,100.0	$ 800.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Private Land purchase		$ 1,600.0	$ -	$ 1,600.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-Production Mining		$ 3,153.9	$ -	$ 3,153.9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Contingency	Avg. 20%	$ 19,711.2	$ 2,210.2	$ 17,501.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Subtotal		$ 119,718.7	$ 14,399.5	$ 105,319.2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital Costs													
Plant		$ 81,052.7	$ -	$ -	$ 27,058.3	$ 49,994.4	$ -	$ -	$ 2,000.0	$ -	$ 2,000.0	$ -	$ -
Infrastructure		$ 23,336.5	$ -	$ -	$ 3,587.7	$ 6,233.3	$ 2,180.4	$ 1,434.8	$ 2,282.3	$ 3,686.4	$ 659.9	$ 3,271.7	$ -
Site Investigation and Proj. Eng.		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation		$ 17,661.7	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,685.5	$ -	$ -	$ -	$ 14,976.2
Contingency	Avg. 22%	$ 26,779.8	$ -	$ -	$ 5,751.2	$ 11,076.4	$ 683.8	$ 401.9	$ 1,969.3	$ 1,047.9	$ 557.3	$ 947.6	$ 4,344.3
Subtotal		$ 148,830.7	$ -	$ -	$ 36,397.3	$ 67,304.1	$ 2,864.1	$ 1,836.7	$ 8,937.2	$ 4,734.3	$ 3,217.1	$ 4,219.3	$ 19,320.5
Total		$ 268,549.4	$ 14,399.5	$ 105,319.2	$ 36,397.3	$ 67,304.1	$ 2,864.1	$ 1,836.7	$ 8,937.2	$ 4,734.3	$ 3,217.1	$ 4,219.3	$ 19,320.5

The accuracy of the capital estimate is expected to be in the range of + 20% to -15% of the actual project cost for each of the project cost centers except the HLP, TSF and Sulfide Plant costs. The accuracy of the capital cost estimate for those three items is less accurate and expected to be in the range of +30 to -20% of the actual project costs. The applied contingencies by capital cost center are provided in Table 1-7.

Table 1-7
Capital Cost Center Applied Contingency

Cost Area	Conting.
Pre-Production Mining	25%
Oxide Plant	12%
Sulfide Plant	18%
Private Land Purchase	25%
Site Investigation and Engineering	25%
Non TSF/HLP Infrastructure	25%
Non TSF/HLP Reclamation	25%
HLP Construction	32%
TSF Construction	30%
HLP Reclamation	34%
TSF Reclamation	32%

Financial Analysis

The financial analysis utilized the base case capital and operating costs that are summarized in Section 21. Additional assumptions in the economic analysis are:

1) Base case metal prices of: $1,250 /oz Gold, $18.25 /oz Silver, $2.75/lb Copper, and $1.00/lb Zinc.

2) Sensitivity tests were performed for: metal prices/head grade, operating costs and capital costs.

3) Turkish tax rates and incentives have been incorporated into the analysis.

4) Discounting is started at the beginning of project construction, and end of year discounting is applied.

5) Contingency is applied to capital costs and is variable based on the relative risks assessed by each of the contractors in each area of costing. On average, contingency is in the range of 20 to 22%.

The base case results indicate that the after tax $NPV_{5\%}$ of the combined oxide and sulfide project is $475.2 million, the internal rate of return is 46.5% and that the payback period of the initial capital cost is 2.5 years.

Figure 1-2 summarizes the project cash flows over time.

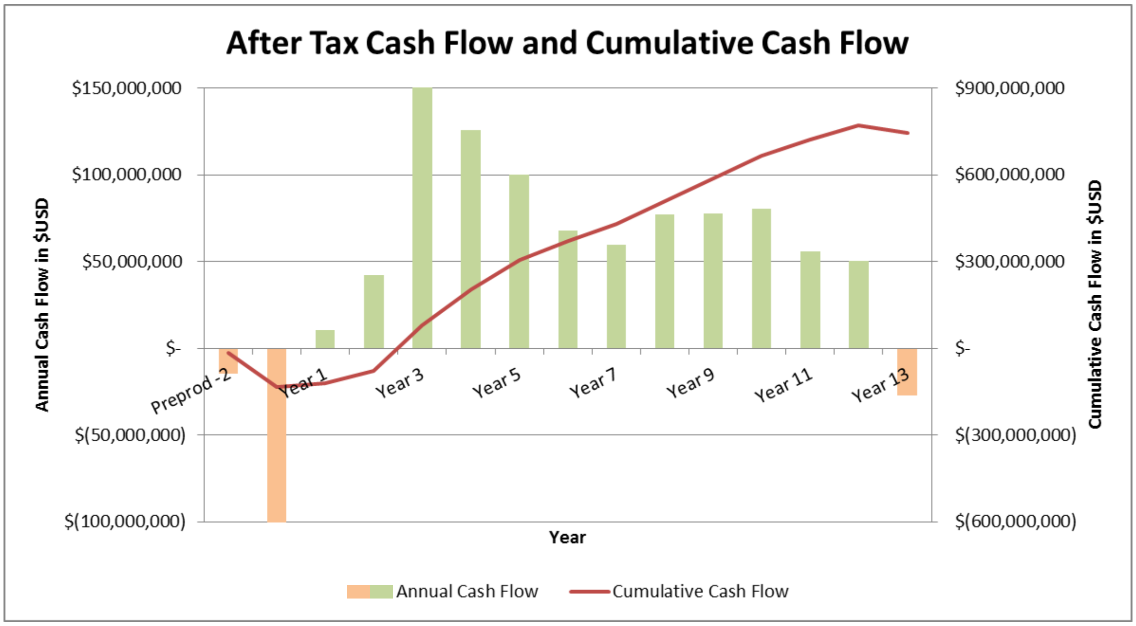


Figure 1-2
Project Cash Flows over Time for the Base Case Metal Prices

The project is robust to changes in metal prices (which corresponds to changes in recovery or changes in head grade), operating costs and capital costs. Figure 1-3 summarizes the internal rate of return versus changes in metal price, operating costs, and capital costs. The project is most sensitive to changes in metal price. Figure 1-4 illustrates the response of the project's Net Present Value at a 5% discount rate as the metal price, operating costs, and capital costs are varied.

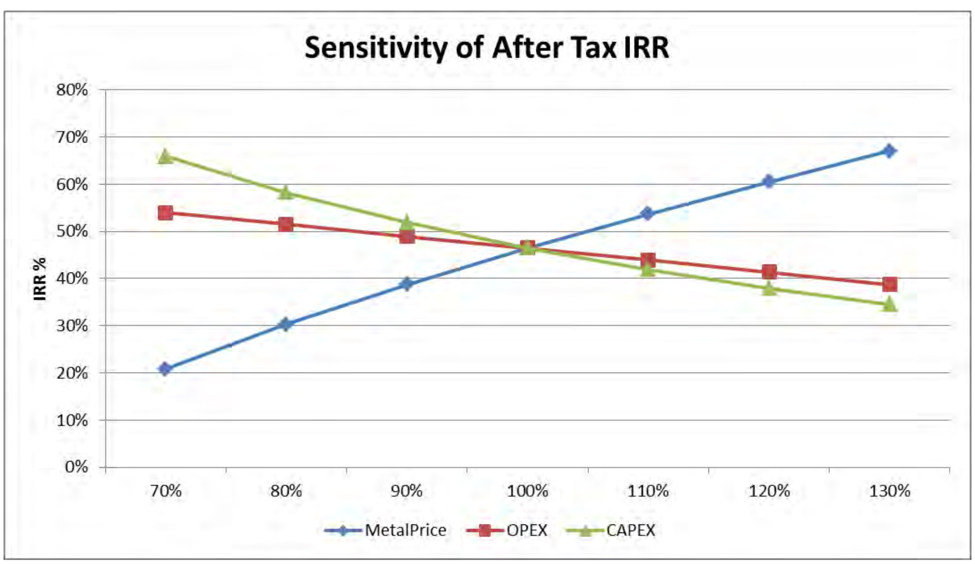


Figure 1-3
Project IRR Sensitivity to Input Parameters

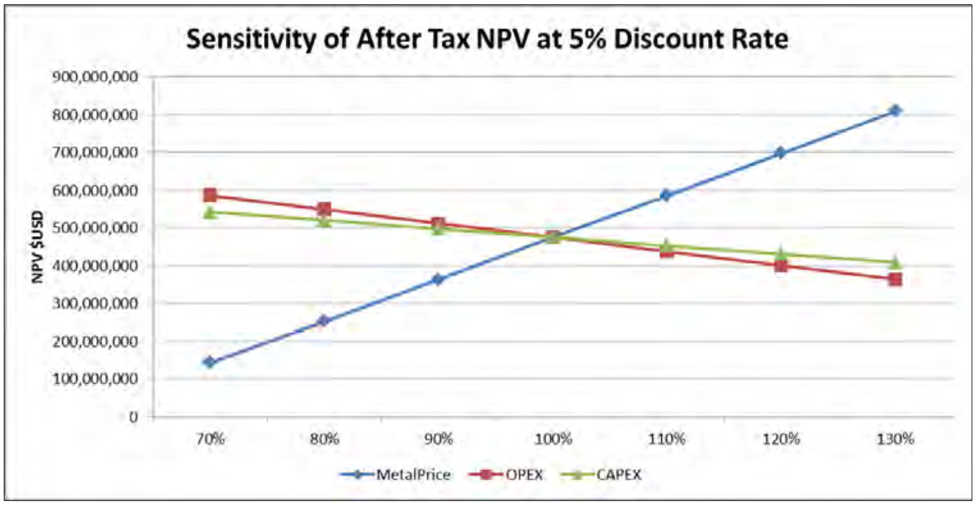


Figure 1-4
Project $NPV_{5\%}$ Sensitivity to Input Parameters

In response to current volatility of the metal markets Polimetal desired to present a sensitivity of the project economics at metal prices more conservative than the base case prices used. The metal prices used in this conservative evaluation are: $950/oz. Au, $13.50/oz. Ag, $2.25/lb Cu, and $0.80/lb Zn. The economic indicators at these metal prices are an after tax NPV5% of $243.8 million and a 28.9% IRR.

Conclusions and Recommendations

This prefeasibility study indicates that the Gediktepe project is an economically robust project over a wide range of metal price assumptions and project cost estimates. Processing testing that was completed during the last year has developed a flow sheet and approach for the sulfides that produces marketable concentrates for both copper and zinc at reasonable process recoveries.

The heap leach component of the project can be quickly moved toward production with financial commitment to geotechnical data collection and additional metallurgical testing followed by more detailed engineering of the heap leach facility and oxide process plant design.

The development of sulfide mining and processing can be established during the oxide production period and consequently has several years available to complete: preproduction stripping, detailed testing, detailed engineering, and construction.

There are a number of tasks that are recommended for continued development of the project. A specific list is presented in Section 26. Some of the major items are:

1) Drill hole data QAQC for the sulfide zone of the deposit should be improved with more checks, and standards as drilling continues.

2) Complete the geotechnical investigation and design for all of the project infrastructure items.

3) Complete detailed site wide water balance for input to design..

4) Continue the current efforts to obtain environmental permits as time and engineering warrants.

5) Continue process metallurgical testing to provide final details for plant design.

The costs and timing of these tasks have been addressed in the estimated project capital cost in Section 21 and execution schedule in Section 24.

2.0 INTRODUCTION

Polimetal Madencilik San. Ve Tic A.S. (Polimetal) assembled a team of consultants to complete a Preliminary Feasibility Study (PFS) for the Gediktepe mining project in Western, Turkey. The current project plan utilizes open pit mining to produce gold and silver by heap leaching of oxide mineralization followed by production of gold, silver, zinc, and copper by flotation of sulfide mineralization.

The project team was comprised of:

Resource Development Inc. (RDi) for process testing and design
SRK Consulting (U.S.), Inc. (SRK) for pre-feasibility level heap leach pad and tailing storage facility design
Independent Mining Consultants, Inc. (IMC) for resource modeling, mine planning
Polimetal Engineering Staff for selected infrastructure items.

The report authors wish to thank the Polimetal staff Hakan Hassoy, Firuz Alizade, Oguz Karamercan and the Polimetal geology department who provided input and guidance critical to the completion of this study.

This document has been prepared in accordance with the guidelines provided in NI43-101 and conforms to Form 43-101 F1 for Technical Reports. The mineral resource definitions conform to the Appendix to the Companion Policy 43-101 CP, CIM – Definitions Adopted by CIM Council, June 20, 2011.

The sources of information used in the preparation of this report include:

1) Personnel inspection of the property
2) Technical information provided by Polimetal
3) Drill hole and geologic data collected by Polimetal
4) Metallurgical test results performed by RDi and other metallurgical labs on samples collected by Polimetal
5) Information provided by Polimetal including geotechnical reports on pit wall stability.
6) Prefeasibility level designs developed by SRK for the heap leach pad and tailing storage facilities.
7) Technical and economic information developed by the project team.
8) Information provided by other experts with specific knowledge and expertise in their fields as described in Section 3.0 of this report.
9) Basic engineering mass balances, drawings, and cost estimmates completed by GR Engineering Services., Ltd. (GRE).
10) Additional information obtained from public domain sources.

The technical team and qualified persons for this report are:

Table 2-1
Technical Team and Qualified Persons

QP Name	Company	Qualification	Site Visit Date	Responsible Chapters
John Marek	IMC	SME Registered Member	July 7, 2014	1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 15 16, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27
Deepak Malhotra	RDi	SME Registered Member	August 10, 2015	13, 17, 21
Terry Mandziak	SRK	PE	August 28-29 2014	Parts of 1, 18, 21

Metric units are used throughout this report as the standard. Tonnes mean metric tonnes. Ktonnes means 1,000 metric tonnes. Standard metric abbreviations are used. Units are occasionally spelled out for clarity. Tonnes per day are referred to as tpd. Occasional references in text of mtpd are used, which also means metric tonnes per day.

Metal production for precious metals will be summarized in Troy Ounces and base metal production in Imperial Pounds as is the custom for marketing and sales. Economics have been calculated in U.S. Dollars. When Turkish lira are required as input, the conversion that has been applied is 3.00 Turkish Lira to the Dollar.

3.0 RELIANCE ON OTHER EXPERTS

This Technical Report relies on reports and information from legal and technical experts who are not Qualified Persons as defined by NI 43-101. The Qualified Persons responsible for preparation of this report have reviewed that information and the conclusions provided and have determined that they conform to industry standards, and are acceptable for use in this report.

IMC and the consulting team have not reviewed the ownership documents or license documentation for the Gediktepe project. We have relied on information provided by Polimetal as outlined in Section 4.0.

IMC has relied on the slope stability geotechnical work completed by Fugro Sial Geosciences Consulting & Engineering Ltd. for open pit stability analysis. IMC holds the opinion that the analysis is appropriate for incorporation into this Prefeasibility Technical Report.

Sections 6 through 9 were initially written by Polimetal Project Geologist Onur Ozgur and translated by Deniz Ekin Karabulut. John Marek has reviewed, confirmed, and edited that work and has taken responsibility as the qualified person for those sections.

Section 20 regarding Environmental Studies, Permitting, and Social or Community Impact was written by Project Manager Oguz Atil Karamercan & Chief Environmental Engineer Fehmi Alemdar of Polimetal.

Basic engineering work and process plant capital cost estimation was completed during the period of November 2015 through April 2016 by GR Engineering Services., Ltd. from Belmont, Western Australia (GRE). This work included process mass balances for the process circuits and 68 drawings providing initial design takeoffs for the process plants. This information was provided to the project team in the form of spreadsheets and drawing files. The specific references to those files are presented in Section 27 (References) and the information is available upon request.

Deepak Malhotra of RDi reviewed the work by GRE and utilized that information as guidance in the preparation of estimated process plant capital.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Gediktepe project is located in Western, Turkey in the Balikesir province. The project site is roughly 42 km straight line distance southeast of the town of Balikesir and about 17 km south-southwest of the town of Dursunbey. The project coordinates are:

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

Figure 4-1 illustrates the general location within Turkey and the Balikesir Province.

4.1 Project Ownership

Polimetal Mining Industry and Trade Inc., otherwise known as Polimetal Madencilik Sanayi ve Ticaret A.Ş. (Polimetal), was formed in 2011 as a joint venture company between Lidya Mining (Lidya Madencilik San. ve Tic. A.Ş.) (50%) and Alacer Gold (50%). Gediktepe mining licenses are held by Lidya Mining (50%) and Alacer Gold (50%).

The property consists of two operational licenses and an exploration license

	License Number (IR)	Area (Hectare)
Operating License	20054077	657.87
	200700250	480.88
Exploration License	201400291	829.12

Operation License - 20054077

On 1 July 2005, the exploration license of Gediktepe was acquired from the General Directorate of Mining Affairs (GDMA) by tender on behalf of Yeni Anadolu Mineral Madencilik San. Tic. Ltd. Sti. (YAMAS). The license area covers 657.87 hec.

That exploration license was changed to an operation license on 23 June 2011 and is valid for 10 years. The License was transferred to Polimetal Madencilik San. ve Tic A.S. (Polimetal) from YAMAS on 26 July 2011.

An Environmental Impact Assessment permit application was submitted and the EIA Permit (not required) was granted on 14 March 2012. A Forest Permit was granted on 11 October 2013 and Workplace Opening and Working Permit (GSM) was obtained on 24 October 2013. After obtaining all necessary permits; the operation permit was acquired on 28 July 2014.

Figure 4-1
Project Location

Exploration License - 201400291

On 17 September 2014, the exploration license which is on east side of 20054077 was acquired by Polimetal from GDMA by auction tender. The license area covers 829.12 hec.

Operation License - 200700250:

An exploration license was transferred to operational stage on 13 May 2014 by the previous owner Hakki Musa Nogay. The license area covers 480.88 hec. Polimetal purchased the operation license from Hakki Musa Nogay during June of 2014. Transferring of the license from Hakki Musa Nogay to Polimetal was completed on 18 November 2015.

The locations of the licenses are shown on Figure 4-2.

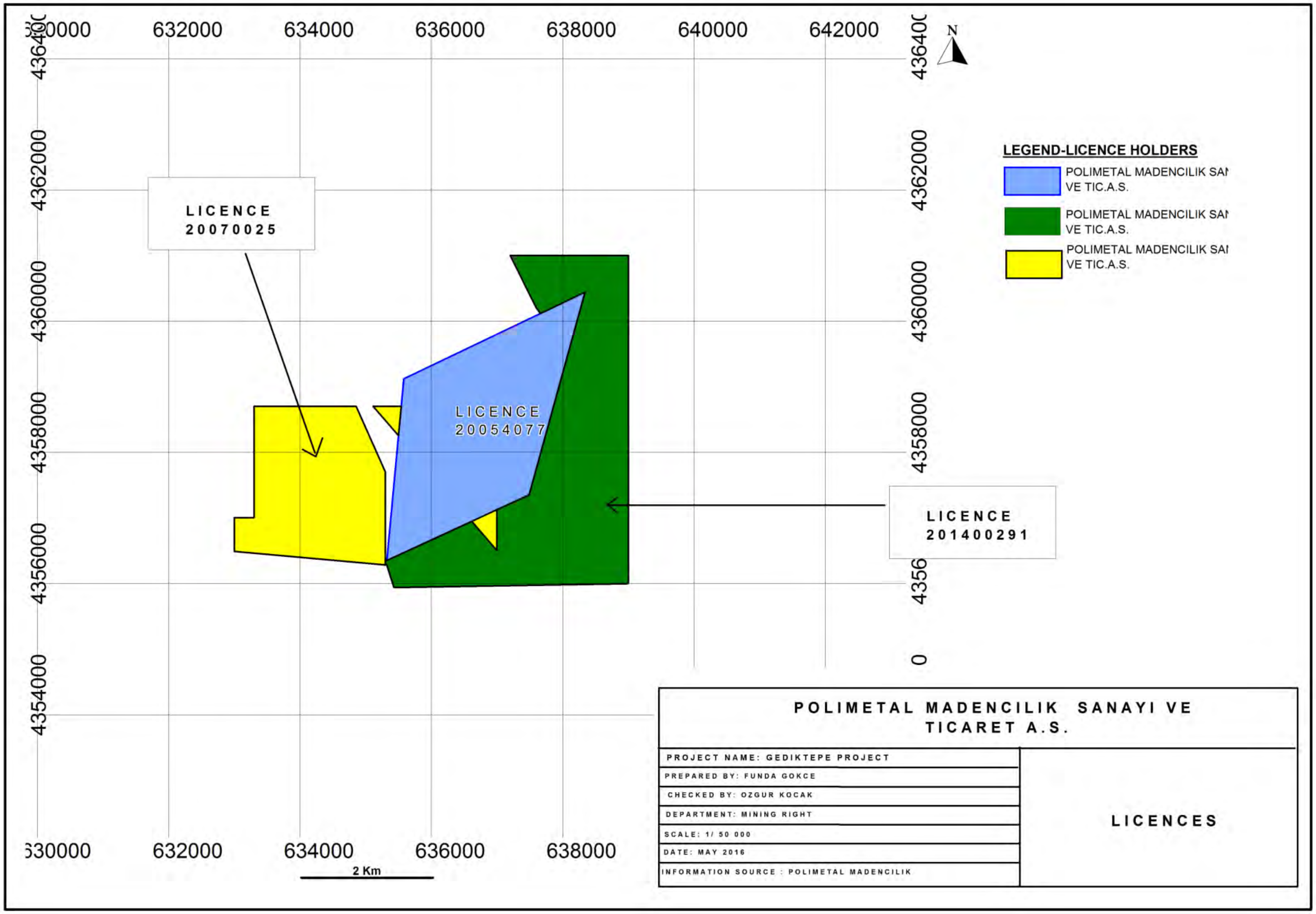


Figure 4-2
Gediktepe Licenses
The Grid Lines are 2 km Apart

4.2 Royalties or Encumbrances

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

A forestry permit should be granted for any forest land that will be used in the project. To obtain the permit, a permit application should 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 Operation Chief of Forestry Dept., Regional Management of Forestry Dept., General Management of Forestry Dept. and Prime Ministry, respectively.

The cost of forest permit depends on the location of the project, type of project (operating a mine, infrastructure or power line, etc.), type of forest and the frequency 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 re-forestation fee and a deposit will be paid one time. After reclamation of the used area, the deposit will be paid back to Polimetal.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, and PHYSIOGRAPHY

The Gediktepe project is located in Western, Turkey in the Balikesir province. The project site is roughly 42 km straight line distance southeast of the town of Balikesir and about 17 km south-southwest of the town of Dursunbey.

The project is 97 km by road from Balikesir via Turkish highway D565 through Bigadic. The last segments of the road are not paved but are generally improved gravel roads.

The nearest airport is Balikesir Koca Seyit Airport serving Balikesir and Edremit which is 183 km from site. There is also airline services to Izmir, Turkey which is also a major shipping port. Izmir is 219 km from the Gediktepe project site.

The elevation at the project site ranges from 1,100m to 1,475m.

Figure 4-1 in the previous section illustrates the location of the project on the map of Turkey and the location relative to the nearest cities.

5.1 Climate

The climate is hot and arid during the summer and warm during the fall. There is occasional snow during the winter but no accumulation. Spring is often the rainy period. Lowest temperatures are -7°C, with the highest at +38°C and an annual average of +14.8°C. Wind generally blows from the North and North-East. A meteorological station has been installed at site as part of the environmental base line data collection.

Average annual precipitation at nearby Dursunbey is 541 mm, with the heaviest precipitation during December, January and February which average 71 mm per month. Average evaporation from the Dursunbey data is 943mm per year with the highest average monthly evaporation of 190mm experienced during July.

5.2 Local Resources

The villages of Hacıömerderesi, Meyvalı, Çatak, and Bozbük are within a few kilometers of the project site. They are generally agricultural villages with relatively minor infrastructure to support a mining project. The current exploration field office is in Hacıömerderesi where several old buildings are used for the camp and offices.

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. About 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.

5.3 Infrastructure

The project will need to develop a water supply as there is no developed system in the area capable of supporting the project. The water supply system will include construction of a freshwater pond and a water treatment plant.

Local electric power is not sufficient for the project and a power line is planned that will extend from the town of Dursunbey to the project site.

5.4 Physiography

The terrain at Gediktepe is mountainous with steep erosional valleys. Elevations in the project area range from 1,100 meters to 1,400 meters above sea level. Coniferous trees cover most of the project site with occasional open meadows in areas of less steep terrain.

Figure 5-1 is a topographic map of the area which illustrates the immediate terrain in the project area. The contour interval is 10m indicating the steep nature of the valleys. The two mining licenses that contain the mine plan are shown for reference relative to Figure 4-2.

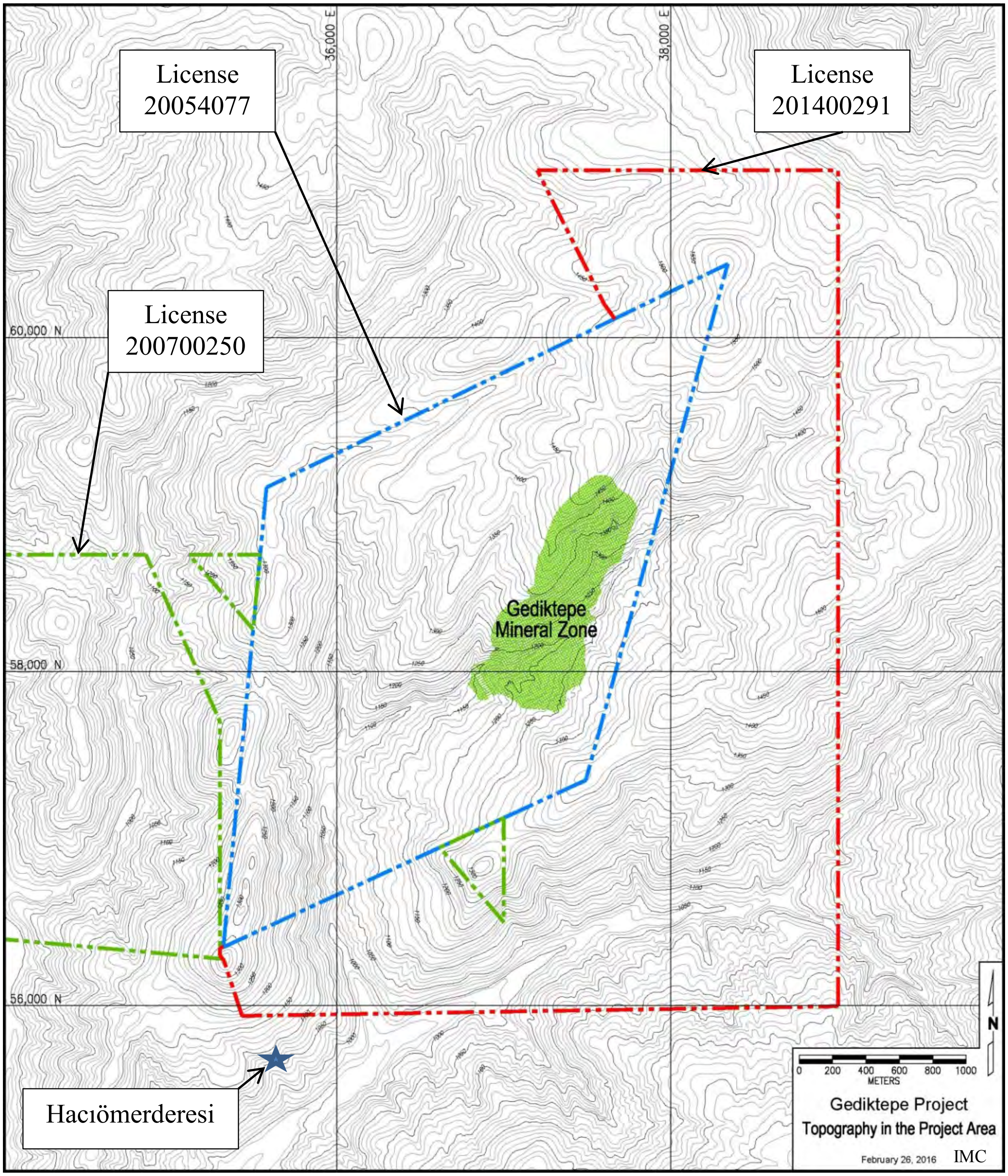


Figure 5-1
Topography in the Project Area at 10m Contour Intervals
The Licenses that Contain the Planned Operation are Shown
Coordinates are Truncated UTM
Green Area Illustrates the Location of the Mineralization

6.0 HISTORY

The initial exploration company that found Gediktepe was Alacer Gold Company. They obtained the first exploration license in 2005. That license was described in Section 4.0 as license number 20054077 which constitutes the central area of the project.

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

Permit applications were submitted at different times for grass roots exploration, drilling and a meteorological station. A summary of the permits and the timing are summarized below for License Number 20054077.

1) An EIA Permit (not required) 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.

2) For Phase 1 Drilling; an EIA Permit (not required) was obtained on 14 March 2012 and on 18 June 2013 for 234 drill locations & forestry permit for 234 drill locations was obtained on 11 October 2013.

3) For Phase 2 Drilling; the EIA Permit (not required) was obtained on 18 December 2013 and on 04 February 2014 for 139 drill locations & the forestry permit for those 139 drill locations was obtained on 02 September 2014.

4) For Phase 3 Drilling; the EIA Permit (not required) was obtained on 02 April 2014 for 268 drill locations & the forestry permit for 264 drill locations was obtained on 02 September 2014.

5) For Phase 4 Drilling; the EIA Permit (not required) was obtained on 27 June 2014 for 344 drill locations. The Prime Ministry approval has been waiting for the forestry permit for 175 drill locations.

6) For the Meteorological Station; the EIA Permit (not required) was obtained on 03 February 2014 & the forestry permit was obtained on 02 September 2014.

7) The revised project operation was submitted to the General Directorate of Mining Affairs (GDMA) on 25 September 2014 to enlarge the operation permit area and to change the annual production & processing capacity to accommodate as much as 2,375,000 tonnes of run of mine ore.

8) The EIA Permit (not required) was obtained on 27 June 2014 for 242 drill and trench locations. The forestry permits for 17 drill and trench locations were received on 13 November 2015. The permit process continues with the Prime Ministry approval waiting for the forestry permit of another 61 drill and trench locations planned for Stage 2 geotechnical investigations.

9) EIA application for oxide and sulfide mining & processing was submitted on July 9th, 2015 and a public participation meeting was held on August 11^{th}, 2015. The EIA report was submitted to Ministry of Environment & Urbanization on December 15, 2015.

7.0 GEOLOGIC SETTING AND MINERALIZATION

This section was originally prepared by Onur Ozgur a Project Geologist for Polimetal. It has been reviewed and edited by John Marek in sufficient detail that John Marek is the qualified person for this Section.

7.1 Regional Geology

The Gediktepe Project is located inside the Afyon zone which is one of the main tectonic units of Turkey. The Afyon zone is located between Menderes Massive to the south and the Taysoni zone to the north. It is a belt consisting of generally low grade weathered metamorphic rocks (Figure 7-1).

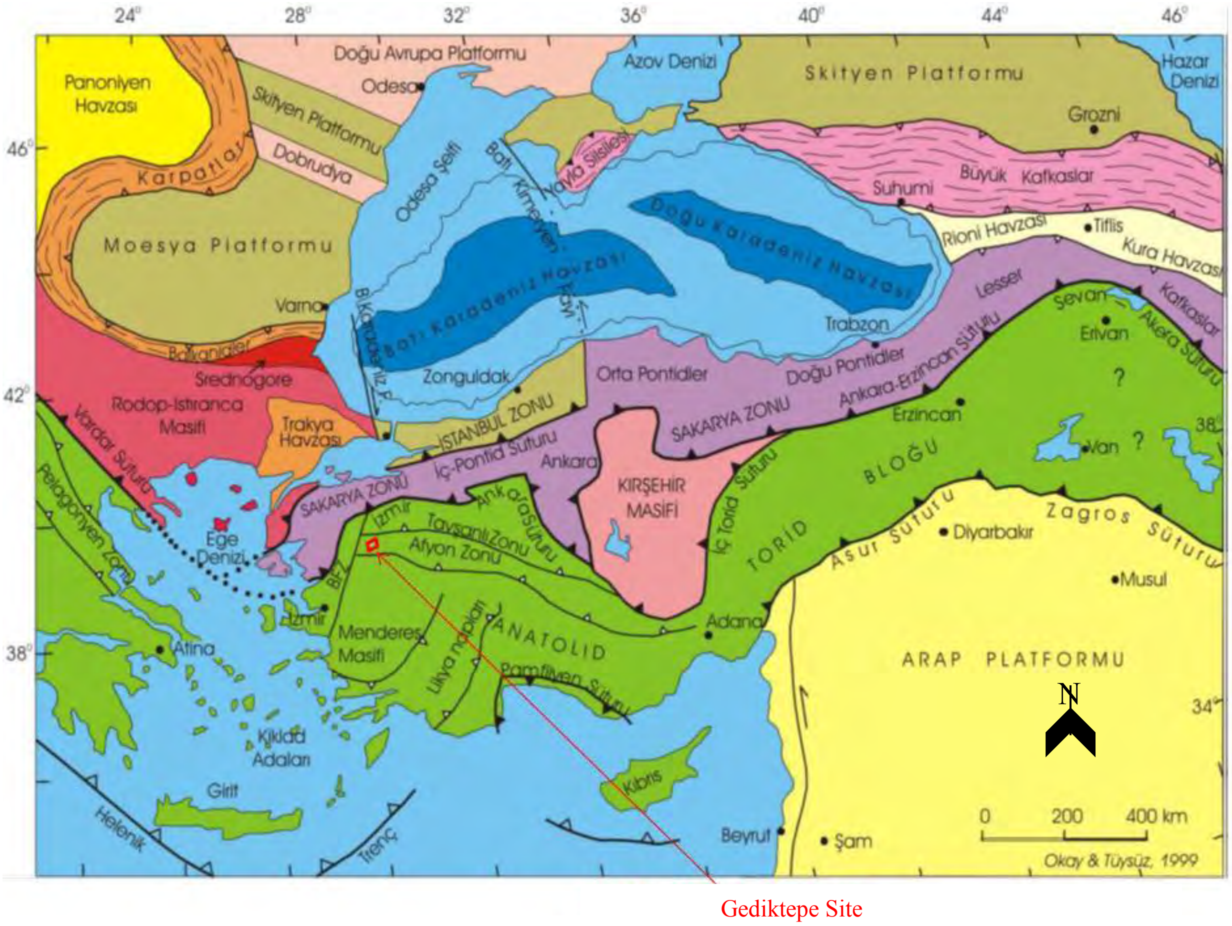


Figure 7-1
Tectonic Map of Turkey (Okay and Tüysüz, 1999)

The local geology at the Gediktepe project consists of Paleozoic and Upper Paleozoic aged metamorphics. The metamorphics are generally composed of gneiss, schist, mica schist, phyllite, amphibolite, marble and quartzite with different degrees of metamorphism. These metamorphics are stratigraphically overlain by Triassic aged carbonates and an upper Cretaceous aged ophiolitic mélange. The upper Cretaceous aged ophiolitic mélange consists of flysch facies units where olistostromal blocks and ophiolite sections are located.

Magmatic rock intrusions developed later between the Oligocene-Lower Miocene, due to extensional features in western Anatolia. Those intrusions cut the Paleozoic aged metamorphic and Upper Cretaceous aged ophiolitic rocks and settled in the region. These intrusions have been called Alaçam Mountains granites where they outcrop in an arc shaped geometry over a nearly 30 km² area (Figure 7-2)

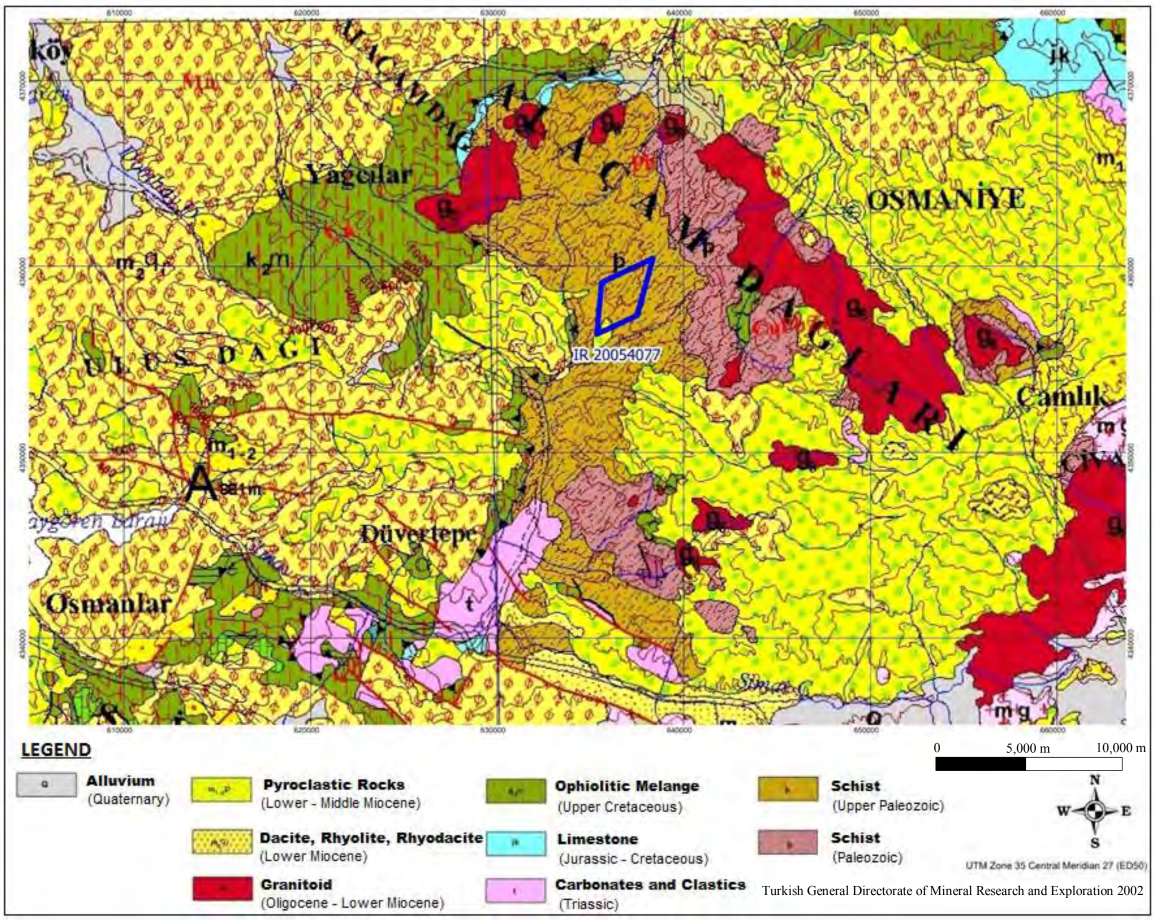


Figure 7-2
Regional Geology Surrounding Gediktepe

The Alaçam Mountain granites are comprised of granite porphyries and aplitic dykes. They create hornfelsic belts where they intruded Paleozoic aged metamorphic rocks. Skarn formations are abundant at the contacts of recrystallized limestone blocks of Upper Cretaceous aged ophiolitic mélange.

Lower Miocene aged volcanic rocks are stratigraphically above the Paleozoic aged metamorphics, Cretaceous aged ophiolitic mélanges, and the Oligocene – Lower Miocene aged magmatic rocks. Lower Miocene aged volcanic rocks are composed of andesite and dacite composite intrusions, domes, lava flows, dykes and volcanogenic sedimentary rocks.

Volcanic rocks, surrounding the Lower-Middle Miocene aged Alaçam Mountains, outcrop over an area of hundreds square kilometers from Bigadiç to Simav and from Dursunbey to Düver Hill. Ignimbrite is one of the volcanic rocks and has a felsic character. It is composed of dacite and rhyolites. Ignimbrites have the widest distribution among the felsic volcanic rocks. Thickness of ignimbrites is as much as 350-400 m around the Alaçam Mountains.

In some areas, these units are overlain by; Pliocene aged terrestrial sediments and Quaternary aged alluvial deposits sourced from the local metamorphics, ophiolitic mélange, granitoids, and felsic volcanic rocks.

The regional geology at Gediktepe is composed of the Upper Paleozoic aged metamorphics, and the Lower-Middle Miocene aged intrusive (dacites).

Figure 7-3 is a stratigraphic column of the Gediktepe project area. Mineralization at Gediktepe is hosted in the Paleozoic units that are illustrated at the bottom of the stratigraphic column.

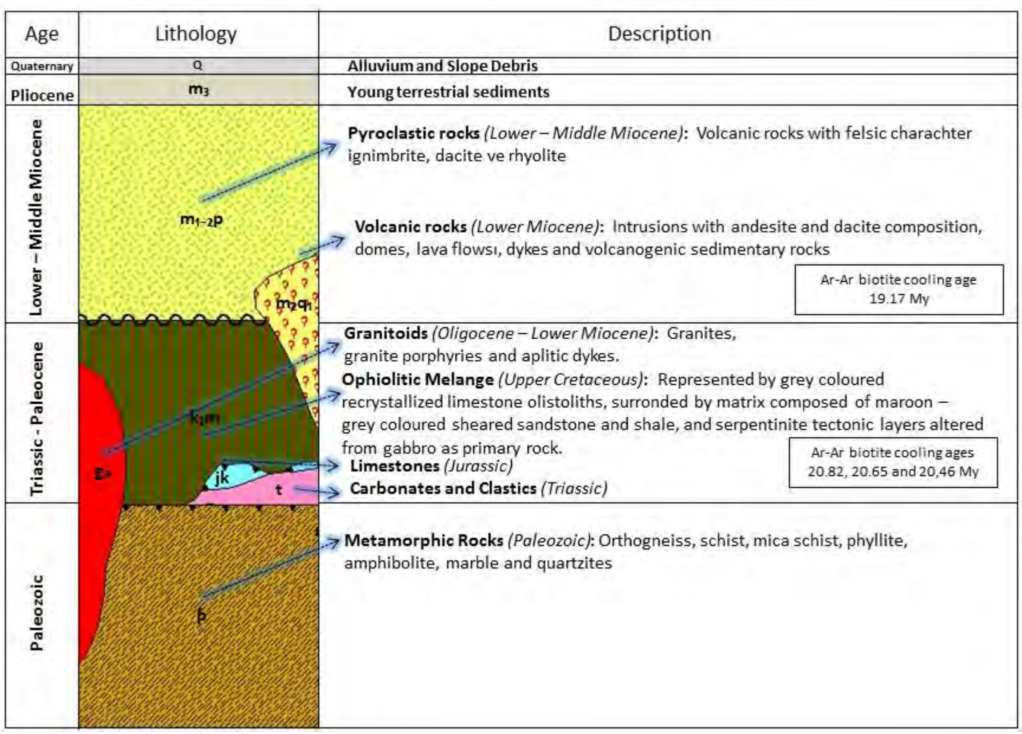


Figure 7-3
Regional Stratigraphic Column

7.2 Deposit Geology

Upper Paleozoic aged metamorphics are the most common units at the Gediktepe Site. Stratigraphically the metamorphics are sequenced from top to bottom as:

Quartz-Feldspar Schist
Chlorite-Sericite Schist
Quartz Schist.

The second most common rocks are the Lower-Middle Miocene volcanics. Those volcanics, are observable around Karadikmen hill, southwest of Gediktepe, contain altered Dacite-Rhyodacite characterized by lava flows and pyroclastics. The youngest units on site are the ore bearing Gossan, Ferricrete, along with Talus, Colluvium and Alluvium that are weathering products of the host rock.

Each of the major units will be discussed in the following paragraphs. Figure 7-17 at the end of this section illustrates the host rock and ore type geometries.

<u>Quartz Schist (Upper Paleozoic)</u>

Quartz Schist is the lowest unit stratigraphically at Gediktepe. It can be observed in outcrop in the southern part of the site from Üçoluk hill to the Aşı stream and in the northeast from Alçakgedik Hill to again the Aşı stream at southeast.

Macroscopically the Quartz Schist is beige-grey, beige-light green colored unit that contains large quartz porphyroblasts. Feldspar, chlorite, muscovite and sericite are other minerals that are observable in unit.

Figure 7-4
Quartz-Schist in Outcrop

Figure 7-5
Quartz-Schist Core Photograph

Chlorite-Sericite Schist (Upper Paleozoic)

Chlorite-Sericite Schist is the main ore host at Gediktepe. The gold and silver of the oxide zone and the copper-zinc of the sulfide zone are contained within this unit. It is observed at Fındıkalanı Ridge, Çamdamı Ridge, Karaismailöldüğü, and northwest of Göğne Hill in the license area.

The color of the unit, varies between green to dark green due to its mafic mineral bands. Macroscopic investigation shows good schistosity. The orientation of the unit is generally N10-30E with a dip of 20-40^0 NW.

The rock composition, from less abundant to high abundant is: quartz, calcite, chlorite, muscovite-sericite. In some places, one can observe euhedral disseminated pyrite minerals.

When the chlorite-sericite schists contain disseminated pyrite more than 15-25% by volume, they are called Pyrite-Chlorite-Sericite Schist by the logging geologists. Disseminated pyrite minerals show an arrangement parallel to schistosity planes and appear as pyrite bands. There are numerous occurrences where locally intense sulfide mineralization has obliterated the protolith. This material is logged as Massive Pyrite, or Massive Pyrite Magnetite.

Figures 7-6 through 7-8 illustrate the unit in outcrop and in core.

Figure 7-6
Chlorite-Sericite Schist in Outcrop

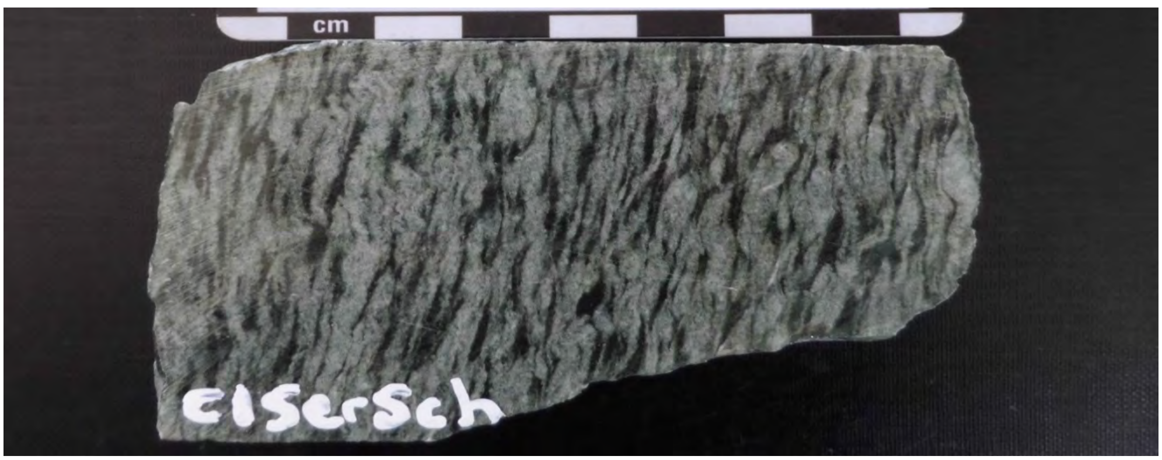


Figure 7-7
Chlorite-Sericite Schist Core Photograph

Figure 7-8
Chlorite-Sericite Schist Altered to Pyrite Chlorite-Sericite Schist

Petrographic analysis indicates that, the rock has been intensely chloritized, epidote altered, silicified, carbonitized and mineralized. Fractures and spaces between individual crystals of cataclastic structured epidote are filled with quartz, calcite and chlorite. The largest euhedral epidote crystal size reaches up to 1mm.

Quartz-Feldspar Schist (Upper Paleozoic)

The Quartz-Feldspar Schist is beige-light green color, and is observable over a wide area of the Gediktepe area. It is the primary waste capping over the deposit and it generally contains almost no sulfides.

Macroscopically, it consists of (2-4mm) feldspar and quartz porphyroblasts, and also can be differentiated from other metamorphic rocks by its weak schistosity characteristic. Chlorite and sericite minerals coating feldspar and quartz porphyroblasts are other rock component minerals.

As a result of examination of thin section analysis, the Quartz-Feldspar Schist contains high amounts of feldspar minerals (orthoclase, plagioclase) and lesser quartz porphyroblasts. It has a well-developed schistosity. Porphyroblast fragments can reach up to 4-5mm sizes and are composed of interlocked crystals.

Figure 7-9
Quartz-Feldspar Schist in Outcrop

Figure 7-10
Quarts-Feldspar Schist Core Photograph

7.3 Mineralization

Mineralization at Gediktepe is related with schist that is metamorphosed under green schist facies conditions. The mineralization is thought to be deposited syngenetically in sedimentary units and metamorphosed to schist. Green schists generally are comprised of actinolite + chlorite + albite + epidote minerals.

The mineralization is largely contained within the Chlorite-Sericite Schists and is divided into the following types:

- Oxide
  - Gossan
- Sulfide
  - Massive Pyrite Magnetite MPM
  - Massive Pyrite MPY
  - Enriched Enr
  - Transition Sulfide Tran

The combined oxide and sulfide zones cover an area of 1,300 m NE-SW, by 450 m NW-SE, with a thickness between 50 and 100m. The deposit generally strikes NE-SW and dips NW.

<u>Oxide Mineralization</u>

The upper portion of the Gediktepe deposit has been weathered, leached, and oxidized by naturally occurring acidic surface water and ground water. The natural acidity is due to the presence of sulfides, particularly pyrite.

Within the oxidized zone, the sulfide mineralization has been nearly completely leached out leaving the gold and silver relatively intact. Relic "lenses" or zones of high gold mineralization can been seen in the oxide zone. There is some evidence that gold mineralization has been transported downward chemically or mechanically as there is often an increase in gold grade just above the oxide –sulfide contact.

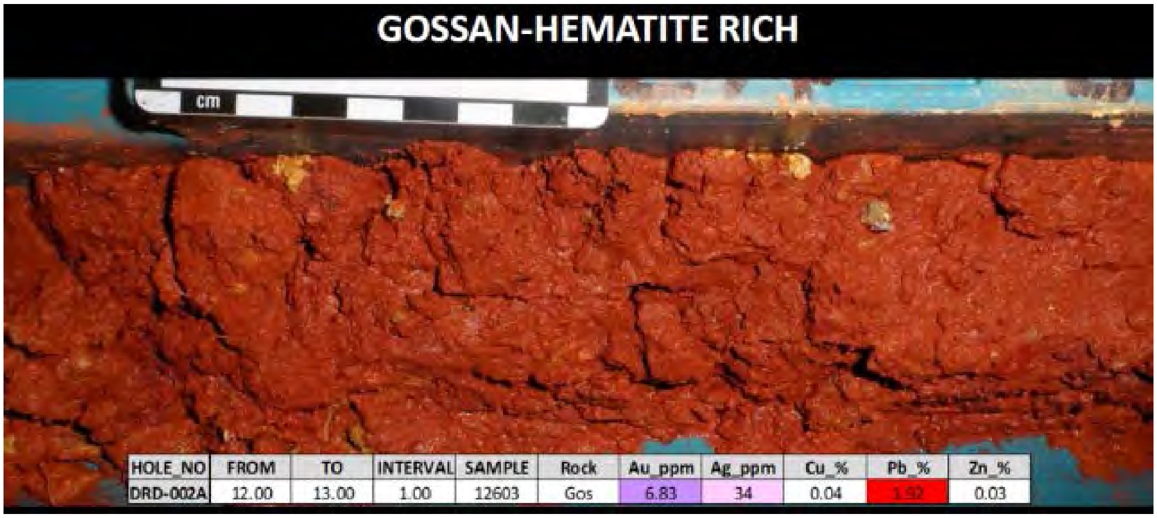
The bottom of oxidation is generally abrupt with rapid changes of metal grade between adjacent assay intervals. Copper and zinc grades are typically less than 0.10% within the oxide zone, but they increase to typical values of 1.4% Zn and 0.80% Cu immediately below the oxide horizon. Gold and silver follow the opposite trend. Gold can be in the range of 3.0 gm/t in the oxide zone and often less than 0.5 gm/t immediately below in the sulfide zone.

The Gediktepe oxide type mineralization is indicated by yellow to red leach zones of intense iron oxide Gossan. Near surface, a leached cap has occurred which is typically low grade in all economic metals. This material often forms a few meters of ferricrete.

The gossan oxide is a hematite rich and highly oxidized unit. It can vary in texture from a breccia with unoxidized clasts to a consistent nearly soil like material where oxidation has obliterated the original texture of the protolith. It hosts locally high values of gold and silver. In general, the upper portion of the gossan zone is low in precious metal grade with abrupt increases from waste to ore grade as one scans down a drill hole. That boundary has been incorporated into the block model as a population boundary between low grade and high grade gossan.

Figures 7-11 and 7-12 illustrate the appearance of the gossan ore zone.

Figure 7-11
Gossan Breccia



HOLE_NO	FROM	TO	INTERVAL	SAMPLE	Rock	Au_ppm	Ag_ppm	Cu_%	Pb_%	Zn_%
DRD-002A	12.00	13.00	1.00	12603	Gos	6.83	34	0.04	1.92	0.03

Figure 7-12
Hematite Rich Gossan

<u>Sulfide Mineralization</u>

The massive pyrite and massive pyrite magnetite are the primary ore types within the sulfide zone. The MPM+MPY mineralization forms lenses or veins that average 20m wide and contain pyrite, sphalerite, tetrahedrite, tenantite, chalcopyrite, and galena. Magnetite is present in both units but it is particularly evident in the MPM. Other sulfide minerals include pyrite, sphalerite, tetrahedrite, tenantite, chalcopyrite, and galena. Within these unites the mineralization has completely over printed the shistosity of the host units.

Figures 7-13 and 7-14 are illustrations of the MPM and MPY Units.

Figure 7-13
Massive Pyrite (MPY)

Figure 7-14
Massive Pyrite-Magnetite (MPM)

Two additional forms of sulfide mineralization have been identified at Gediktepe:
1) Enriched, and 2) Transition Sulfides.

There are thin zones of chalcocite rich secondary enrichment that are located centrally in the deposit. Copper grades are elevated in the enriched zone compared to the surrounding sulfides. The enriched zones are limited in thickness and area and are a minor component of the mill feed. Roughly 2.3% of the sulfide mineral reserve is enriched. Figure 7-15 is an illustration of the enriched sulfide. The black matrix in the photo is chalcocite rich.

Figure 7-15
Enriched Sulfide

Transition sulfide is material that is more disseminated than the MPM and MPY units. The occurrence is similar mineralization to MPM and MPY but it is less massive and it is intermixed with the protolith host. It can appear as veinlets of mineralization or as dissemination within the host schist. The grades for all economic metals are lower than that for MPM and MPY. Transition sulfide is often in contact with the massive units and it can be just above or just below on section.

The term transition indicates that the intensity of mineralization or grade is in transition from waste to the higher grade massive ores that are nearby.

Figure 7-16
Transition Sulfide

Figure 7-18 is a Northwest-Southeast cross section through the deposit that illustrates the geometry of the major ore hosts at Gediktepe. The location of the section relative to the drilling is shown on Figure 7-17.

7.4 Deposit Structure

Structural details are not well mapped at this time due to the extensive ground cover and weathered surface outcrop. However, mineralization and rock type offsets in the drill holes indicate that the mineralization is offset by a series of steeply dipping northwest – southeast striking faults.

The tabular ore bodies, particularly in the sulfide zone dip gently to the west. As one moves northeasterly in the deposit each ore zone becomes more shallow. However, there are several places in the deposit where that trend stops abruptly between the northwest-southeast drill sections. At these locations, the tabular ore zones drop downward to the northeast indicating post mineral offset of the mineral zone.

This offset geometry can be interpreted to occur 3 to 4 times across the deposit as one moves from southwest to northeast. Within the model each tabular body or lens is treated independently and is modeled as described as an abrupt break and offset in the mineralization.

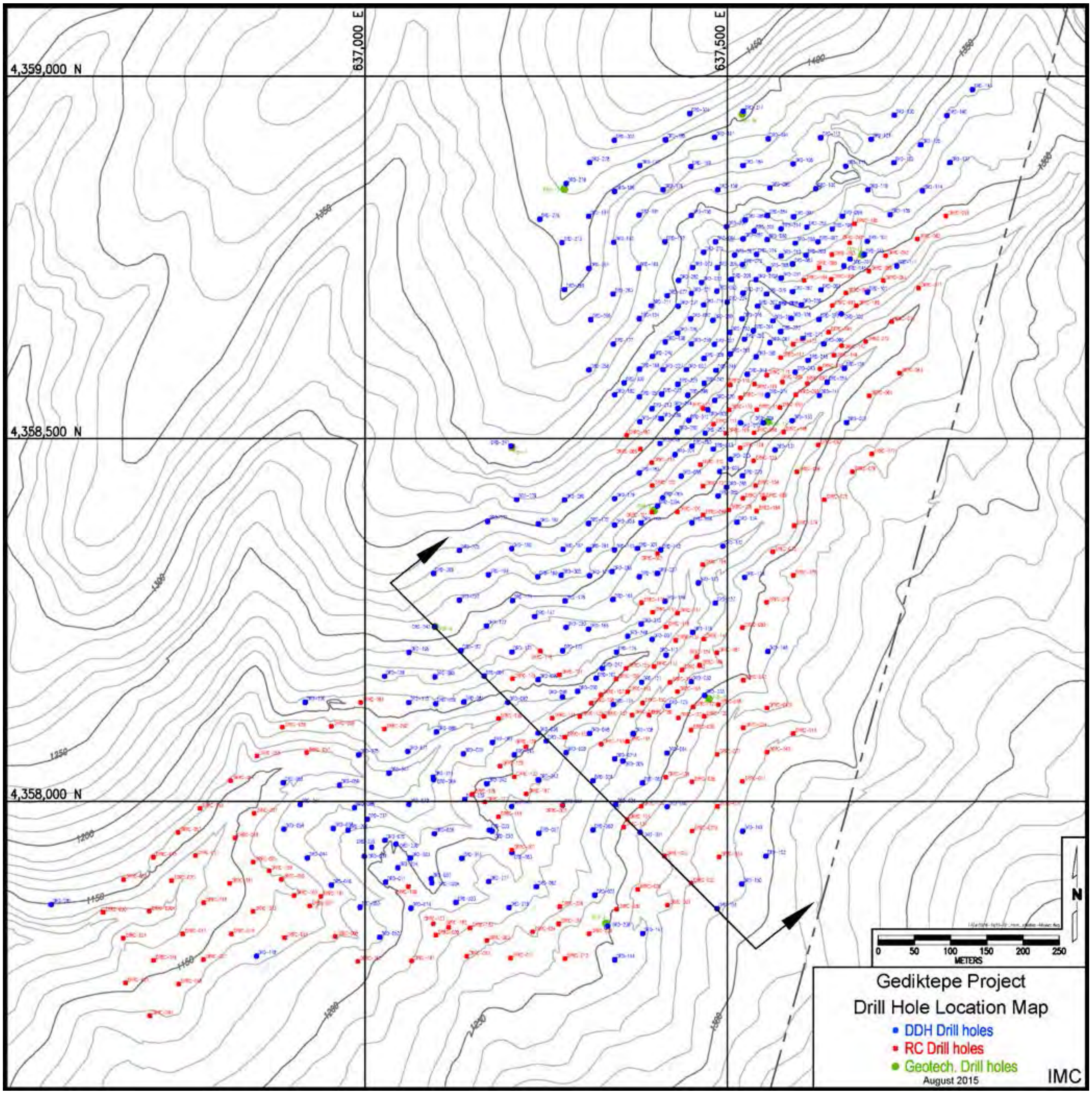


Figure 7-17
Gediktepe Drilling and Section Location for Figure 7-18

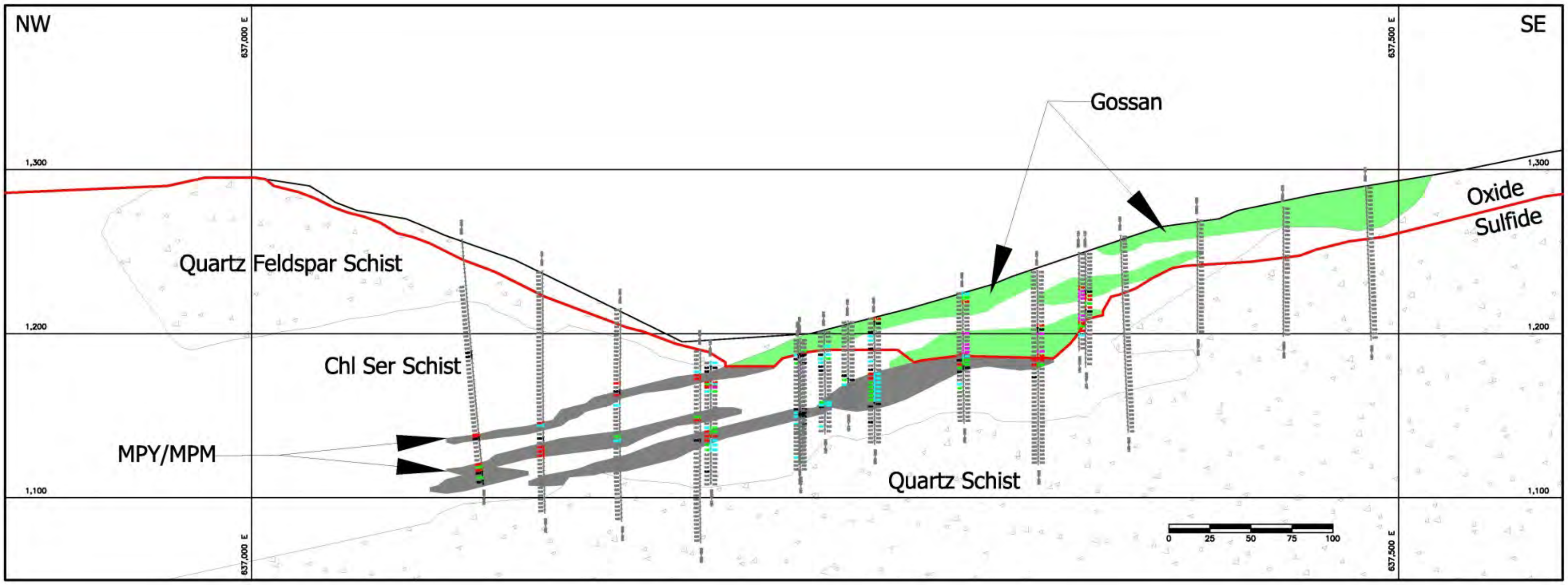


Figure 7-18
Summary Cross Section Through the Gediktepe Deposit, Looking Northeast
Green Illustrates Oxide Gossan Zone
Grey Illustrates the Sulfide MPY/MPM Zones

8.0 DEPOSIT TYPE

The Gediktepe deposit is a skarn type Massive Sulfide or MS type of metal sulfide ore deposit. These deposits are created by volcanic-associated hydrothermal events in submarine environments. They are typically high in copper, zinc, and other sulfide minerals.

Figure 8-1 is an illustration of this type of deposit which explains the syngenetic occurrence of sulfide mineralization at Gediktepe. Subsequent weathering and oxidation have also occurred at Gediktepe forming the oxide and gossan portion of the deposit.

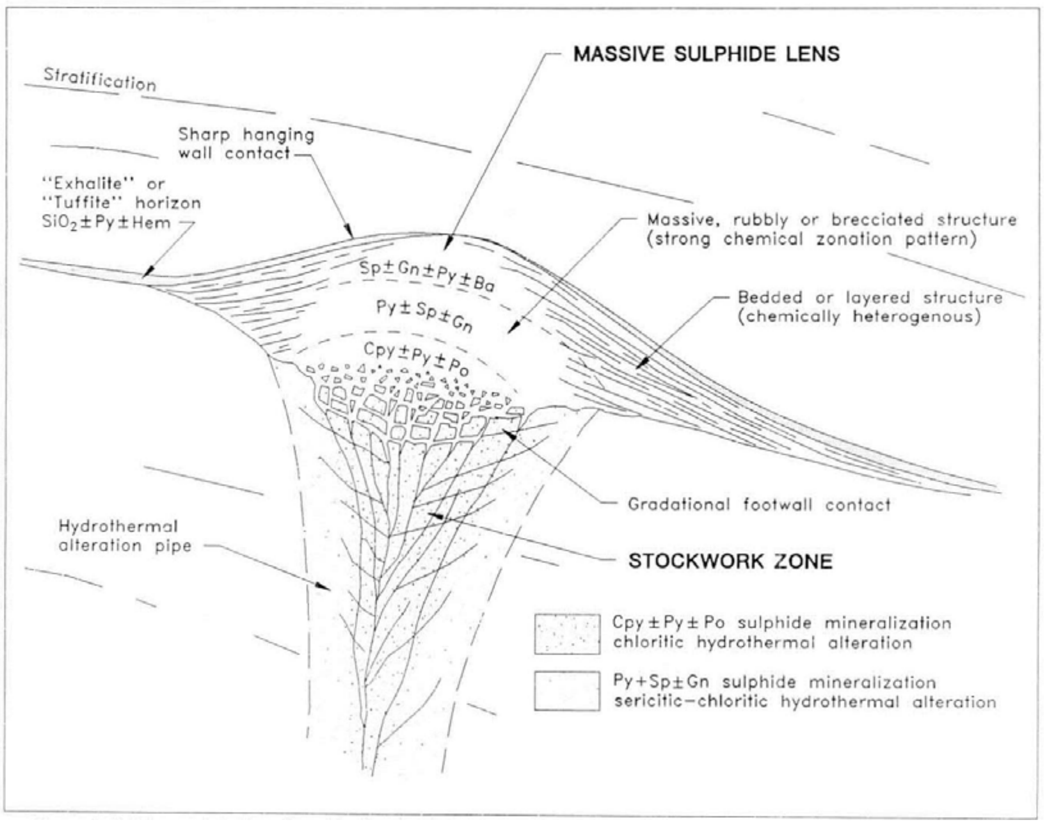


Figure 8-1
Vertical Section of an Ideal Convex MS deposit (Lydon, 1984)

9.0 EXPLORATION

This section summarizes the exploration methods that were applied at Gediktepe outside of the drill programs. Drilling will be summarized in Section 10.0

9.1 Geochemical Studies

There were several surface geochemical sampling programs completed between 2012 and 2014. Early work was completed by Alacer prior to the entering a joint venture agreement with Lidya.

Initial stream channel sediment sampling was completed by Alacer. The results of the work indicated the presence of a gold anomaly with associated base metals.

Later efforts included a regular 100m grid of soil samples over the entire area of the first license 20054077. The soil sampling shows a strong gold, copper, lead, and zinc anomaly that directly overlies the Gediktepe mineralization.

Rock chip sampling was completed where outcrops were available. As with the soil samples, high values for economic minerals were directly over where the deposit was later drilled.

Table 9-1 summarizes the amount of geochemical sampling and analysis that has been completed at Gediktepe.

Table 9-1
Summary Geochemical Sampling at Gediktepe

Summary Count of Geochemical Sampling			
Company	Rock	Soil	Stream
Alacer	240	289	20
Polimetal	113	497	0
Total	353	786	20

9.2 Geophysical Studies

Two types of ground based geophysical studies were completed at Gediktepe: 1) magnetic and 2) induced polarization (IP).

A magnetic survey was completed at Gediktepe during August of 2013. A total of 112.2 km of survey line was analyzed over 32 cross sections. Sections were oriented north-south and cover the entire area of the initial license #20054077.

The resulting map of magnetic anomalies indicates the zones where magnetite is present within the deposit. In addition, it illustrates potential structural features in the area.
The IP survey was completed in parallel with the magnetometer study. There were 22 IP section lines oriented NW to SE for a total of 41.6 km of line. Most of the initial license #20054077 was covered with the study.

The results of the magnetometer survey and IP study were analyzed collectively. In general, the results indicate that low resistivity combined with high magnetic response coincides with the high grade zones of mineralization.

10.0 DRILLING

All exploration drilling at site has been conducted by Polimetal. Phase 1 was an 11 hole core program that was completed during 2013. Phase 2 started later in 2013 and continued through early 2014. Phase 2 utilized both diamond core (DD) and reverse circulation (RC) drilling. Phase 3 also applied DD and RC methods and was completed in 2015. Core is predominately PQ with a few HQ holes.

The drill hole data for this estimate of mineral resources was provided to IMC on 5 August 2015 and represents all Resource data available from Gediktepe at that time. No additional Resource drilling has been completed up to the effective date of this report

The summary of exploration drilling on site through August of 2015 is as follows:

Table 10-1
Summary of Exploration Drilling Through August of 2015

Drilling Phase	Number of Core Holes	Number of RC Holes	Meters Drilled
1	11		1,528.5
2	143		17,114.8
		81	6,790.0
3	149		26,061.1
		103	6,042.0
Total	303	184	57,536.4

As of August 2015, IMC received a total of 487 drill holes at Gediktepe amounting to 57,536.4m of drilling. Figure 10-1 illustrates the drill hole locations and the drill hole type within the data base. Three pairs of diamond versus reverse circulation twins were drilled as reported in Section 12.

The project coordinate system within which the drill holes are surveyed is UTM European Zone 35. The magnetic declination in the area is +4.78 degrees. Hole collars were surveyed after drilling was completed by a local surveying firm. 255 of the 303 diamond holes have downhole survey data available. Down hole surveys were performed with a Devico reflex device. RC drill holes were not down hole surveyed. Eight holes of the initial 11 hole program were angle holes. The rest of the holes are vertical or sub-vertical. The average deviation of the surveyed holes is less than 1 degree per 100m.

Fugro Sial drilled 9 nearly vertical oriented core geotechnical holes for slope stability investigations. The locations of these holes are also shown on Figure 10-1.

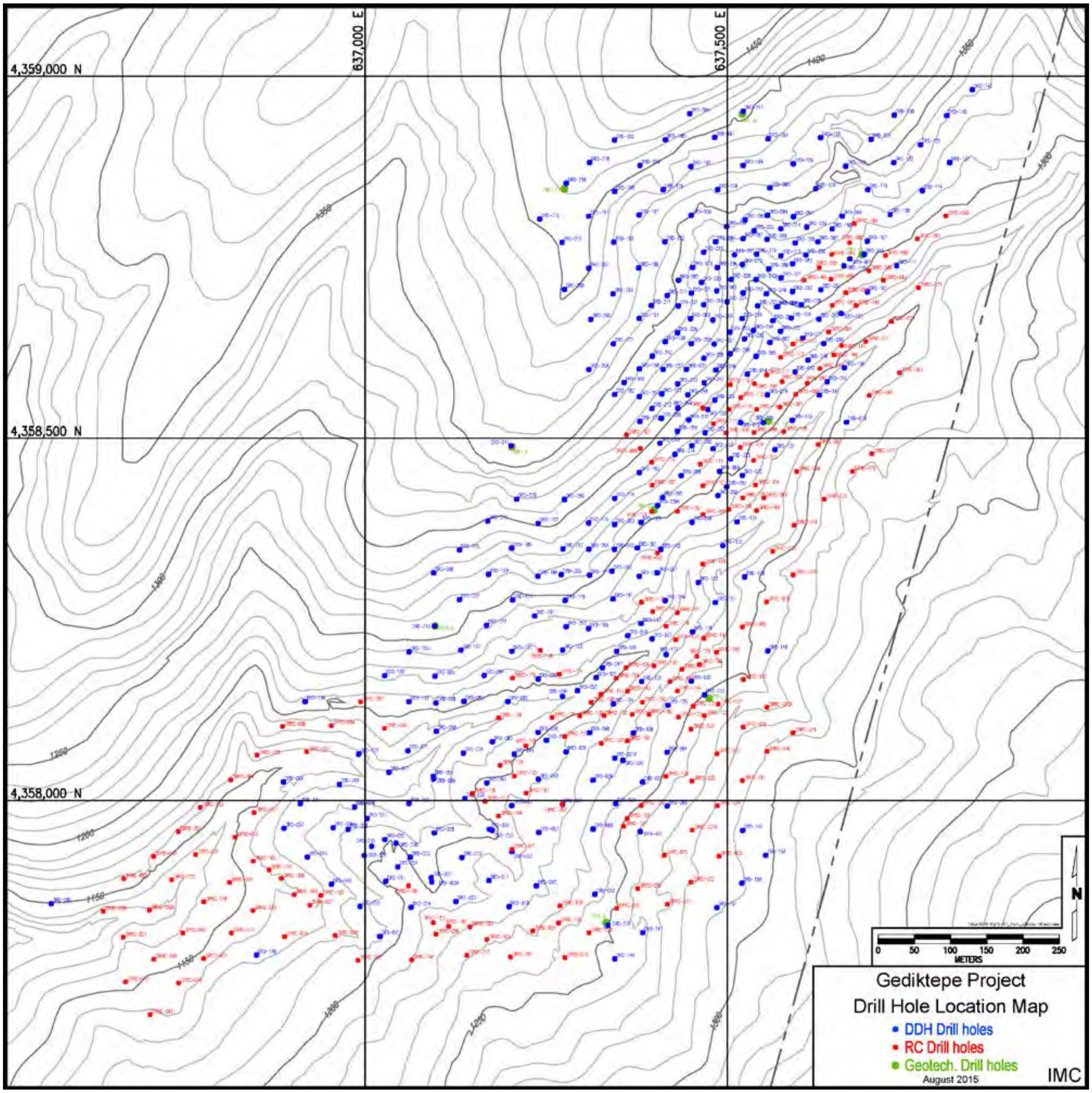


Figure 10-1
Gediktepe Drill Hole Location Map
Blue = DDH Drill Holes
Red = RC Drill Holes
Green=Geotechnical Drill Holes

11.0 SAMPLE PREPARATION ANALYSIS AND SECURITY

Assay information for the Gediktepe deposit is collected by two drilling methods: 1) diamond drilling and 2) reverse circulation (RC) drilling. Diamond drilling is predominately PQ diameter with some HQ diameter drilling. RC drilling has been used on the outside edges of the deposit to define extensions or set limits and it has been used for infill in some zones of the deposit.

Phase 1 drilling during 2013 was the first 11 drill holes in the deposit. All were core holes and all were prepared and assayed at SGS. Phase 2 drilling had 224 holes while the most recent Phase 3 drilling had 252 drill holes. All Phase 2 and Phase 3 drilling, including core and RC were prepared and assayed at ALS.

The first 11 holes were transported by SGS to Ankara. The remaining holes have been transported to ALS in Izmir.

Both the SGS lab in Ankara and the ALS lab in Izmir are ISO-9001:2008 certified.

IMC and John Marek (QP for this section) hold the opinion that the Gediktepe sample preparation and security procedures are appropriate for providing reliable data for the calculation of mineral resources and mineral reserves.

11.1 On Site Sample Preparation

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

Density measurements are completed by Polimetal geology staff on samples of the whole core at the logging facility. Density samples are dried for 6 hours and then coated to prevent moisture from entering the sample. They are then weighed in air and in water.

Core is split using a diamond saw. Half is bagged for shipment to the assay lab, and the remaining half is stored in the core tray for historic reference. Assay intervals are nominally 1.5 to 2m long but they can reduce to as short as 0.40 m in the ore zones. Transportation from Gediktepe to the respective labs was the responsibility of Polimetal. Once samples are delivered to the lab, the lab logs the sample into their system and confirms transfer and possession of the sample.

Polimetal inserts standards, duplicates, and blanks into the sample shipments by Polimetal. Duplicates are additional splits of the core.

Reverse circulation samples are collected using a rotary splitter at the drill rig. Chip samples are collected for rock type and geologic logging including lithology, alteration, mineralization and structures. Appoximately 55% of the RC sample intervals were 2m long.

The other 45% of samples are shorter with the shortest being 1m in length. Weights of RC samples are typically about 3 kg.

As with core, standards, blanks, and duplicates are submitted with RC samples. RC duplicates are second splits from the drill rig.

11.2 SGS Sample Preparation and Analysis

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

1) The samples were logged in and weighed on arrival.
2) The samples were dried and crushed by SGS protocol CRU24
3) Pulps were prepared. The lab certificates from SGS did not list the pulp protocol, but the nominal pulp criteria for the AA and ICP analysis at SGS is 75 micron
4) Gold was assayed by protocol FAA303, a fire assay with AA finish on a 30 gm aliquot.
5) Copper and silver were assayed by protocol AAS42S, which is an AA finish.
6) All other metals were assayed by protocol ICP40B which is a four acid digestion and multi-element ICP procedure.

11.3 ALS Sample Preparation and Analysis

The ALS sample preparation and assay procedures were applied to the Phase 2 and Phase 3 drilling for both core and RC samples.

1) The samples were logged in and weighed on arrival.
2) The core samples were dried and crushed by ALS protocol CRU-31 with 70% less than 2mm. RC samples are not crushed but are dried before splitting
3) Samples are split with a riffle splitter before pulping.
4) Pulps were prepared with ALS protocol PUL-32 where 1000 kg is reduced to 85% passing 75 micron.
5) Gold was assayed by protocol Au-AA25, a fire assay with AA finish on a 30 gm aliquot.
6) All other metals were assayed by protocol ME-ICP61 which is a four acid digestion to report 33 elements by ICP methods.

11.4 Data Base Assembly

The certificate information was sent to Polimetal electronically. Electronic copies of the certificates are stored and were provided to IMC for verification. Polimetal employees in Ankara maintain a master assay database using “Datashed” database software. Assay data was transferred to IMC in .csv files.

12.0 DATA VERIFICATION

Polimetal collects quality control and assurance (QAQC) information as part of their data handling and assaying procedures. IMC acquired that information and completed a statistical analysis of the results. In addition, IMC completed comparisons of the certificates of assay versus the drill hole data base to confirm the computer data base that is maintained by Polimetal.

The following analysis was completed by IMC

1) A comparison of the assay data base against the original certificates of assay
2) Analysis of inserted standards
3) Analysis of inserted blanks
4) Analysis of duplicate assays
5) Analysis of check assays
6) Comparison of diamond drilling results to reverse circulation (RC) drill results.

During the site visit by John Marek in 2014, the locations of about 12 drill holes were spot checked against the recorded coordinates and map locations.

The data verification by IMC was completed in two iterations. The first was completed in December of 2014 that addressed the data base that was complete as of 9 July 2014. The second iteration was completed during September of 2015 and addresses the data that was available on 5 August 2015.

Drilling was started in late 2013 and continued through mid-2015 for inclusion into this statement of mineral resources. The first 11 holes of the 2013 drilling program were called Phase 1. Those holes were assayed at SGS laboratories. There are 224 Phase 2 drill holes and 252 holes in Phase 3. Phase 2 and 3 were all assayed at ALS Chemex.

As a result of the data verification work that is presented in this section, IMC and John Marek (QP for this section) hold the opinion that the drill hole data base for Gediktepe is acceptable for the calculation of mineral resources and mineral reserves.

12.1 Certificates vs the Data Base

IMC compared the Polimetal data base against PDF scans of the certificates of assay as a check on the integrity of the data base. The following thirty drill holes were selected at random (by IMC) from the July 2014 data set:

DRD-004 DRD-008 DRD-013 DRD-025 DRD-041 DRD-048 DRD-039
DRD-061 DRD-066 DRD-072 DRD-082 DRD-092 DRD-098 DRD-099
DRD-102 DRD-111 DRD-116 DRD-121 DRD-131 DRD-141 DRD-151
DRRC-002 DRRC-009 DRRC-019 DRRC-027A DRRC-038
DRRC-046 DRRC-060 DRRC-069 DRRC-079

Thirty drill holes are about 13% of the 235 drill holes drilled in Phase 1 and Phase 2 of drilling prior to July 2014. Within that list, holes DRD-004 and DRD-008 were part of the Phase 1 drilling that were assayed at SGS. The remaining 28 holes were assayed at ALS Chemex.

Twenty six drill holes (10%) of Phase 3 holes that were drilled in 2014 and 2015 were selected at random (by IMC) from the August 2015 Gediktepe database for certificate checks. These drill holes were:

DRD-160	DRD-174	DRD-190	DRD-199	DRD-216	DRD-225
DRD-227	DRD-239A	DRD-250	DRD-266	DRD-280	DRD-296
DRD-303	DRRC-086	DRRC-100	DRRC-112	DRRC-115	DRRC-126
DRRC-136	DRRC-137	DRRC-138	DRRC-149	DRRC-162	DRRC-163
DRRC-180	DRRC-188				

The results of the certificate check for the two selected holes that were assayed at SGS are summarized on Table 12-1.

Table 12-1
SGS Lab, Certificates of Assay vs the Data Base

Hole Name	Metal Analysis	No. of Assays	No. of Certs	Percent Available	No. of Errors	Error Rate %	Description of Errors
DRD-004	Silver	221	221	100%	3	1.4%	Values are mean of original assay and lab duplicate assay
	Gold	221	221	100%	2	0.9%	Values are mean of original assay and lab duplicate assay
	Copper	221	221	100%	12	5.4%	Values are mean of original assay and lab duplicate assay
	Lead	221	221	100%	11	5.0%	10 values are mean, 1 value of >10000 ppm was set to 1
	Sulfur	221	221	100%	3	1.4%	Values are mean of original assay and lab duplicate assay
	Zinc	221	221	100%	14	6.3%	Values are mean of original assay and lab duplicate assay
DRD-008	Silver	82	82	100%	0	0.0%	Values are mean of original assay and lab duplicate assay
	Gold	82	82	100%	16	19.5%	2 values are mean, 14 values near detection limit , shifted by 1 sample
	Copper	82	82	100%	6	7.3%	Values are mean of original assay and lab duplicate assay
	Lead	82	82	100%	5	6.1%	Values are mean of original assay and lab duplicate assay
	Sulfur	82	82	100%	4	4.9%	Values are mean of original assay and lab duplicate assay
	Zinc	82	82	100%	5	6.1%	Values are mean of original assay and lab duplicate assay

Most of the differences between the value in the database and the certificate value are because the data base utilized the calculated average of the initial assay and the lab duplicate assay. The average of the original value and the duplicate value should not be used in the primary data base. The original value should be the basis for the mineral resource calculations. Using an average value results in a sub-set of the data base with too low of a variance when compared to the typical data set. Duplicate values are meant to check the original data, not modify it.

One result for lead (which is in ppm in the database) was entered as 1% rather than 10,000 ppm.

The averaging procedure that was applied for the first 11 holes is not recommended, however, it is not a reason to reject the SGS sourced data and the averaging procedure has not biased the assay data. There are eleven drill holes out 487 in the database which were assayed by SGS as their primary lab, about 2.3% of the assay data.

There are a total of 54 holes in the set of certificates that were checked by IMC that were assayed at ALS Chemex. Of those holes, 9 were found where the data base and the certificate were not a match.

About half of the checked certificates have had the silver data rounded to the nearest PPM in the data base rather than the lab result that reports to the nearest 0.10 PPM. This is not an error or a bias, but the rounding is not consistent. Some holes have applied it and others have not.

The nine holes with differences are summarized on Table 12-2.

In holes DRD-190 and DRRC-086 the average gold grade was used in the 3 intervals. In drill hole DRD-303 one interval had silver assay values that did not match the certificate. Drill hole DRRC-162 in the Polimetal assay data base was missing 14 zinc assays. The certificate data was added to the IMC version of the data base for completeness before resource estimation.

In drill hole DRRC-136, the wrong assay columns for silver, copper, lead, sulfur and zinc were entered for the entire drill hole. This column mismatch occurred because the check assay gold grades were incorrectly entered as the silver values, the remaining element columns following silver did not match either. Drill hole DDRC-136 was reentered into the data base by IMC from the certificates and corrected prior to estimation of mineral resources.

Table 12-2
ALS Chemex Lab Assays, Certificates of Assay vs the Data Base

Holeid	Analysis	No. of Assays	No. of Certs	Available %	No. of Errors	% in error	Description of Errors
DRD-013	Silver	66	66	100%	1	1.5%	Sample No. 11646 used assay method AA46m instead of IG62
	Gold	66	66	100%	0	0.0%	
	Copper	66	66	100%	0	0.0%	
	Lead	66	66	100%	0	0.0%	
	Sulfur	66	66	100%	0	0.0%	
	Zinc	66	66	100%	0	0.0%	
DRD-025	Silver	47	20	43%	0	0.0%	Missing some certififcates in the provided files
	Gold	47	20	43%	0	0.0%	
	Copper	47	20	43%	0	0.0%	
	Lead	47	20	43%	0	0.0%	
	Sulfur	47	20	43%	0	0.0%	
	Zinc	47	20	43%	0	0.0%	
DRD-098	Silver	49	49	100%	18	36.7%	11 samples set to 0.0, Certificates are low values
	Gold	49	49	100%	0	0.0%	
	Copper	49	49	100%	0	0.0%	
	Lead	49	49	100%	0	0.0%	
	Sulfur	49	49	100%	0	0.0%	
	Zinc	49	49	100%	0	0.0%	
DRD-121	Silver	63	53	84%	0	0.0%	Missing some certififcates in the provided files
	Gold	63	53	84%	0	0.0%	
	Copper	63	53	84%	0	0.0%	
	Lead	63	53	84%	0	0.0%	
	Sulfur	63	53	84%	0	0.0%	
	Zinc	63	53	84%	0	0.0%	
DRD-190	Silver	80	80	100%	0	0.0%	Silver grades are rounded to integers, no decimals
	Gold	80	80	100%	2	2.5%	2 intervals have Au check assays, averages used
	Copper	80	80	100%	0	0.0%	
	Lead	80	80	100%	0	0.0%	
	Sulfur	80	80	100%	0	0.0%	
	Zinc	80	80	100%	0	0.0%	
DRD-303	Silver	40	40	100%	1	2.5%	One Assay stored as 101 ppm certif has 100ppm
	Gold	40	40	100%	0	0.0%	
	Copper	40	40	100%	0	0.0%	
	Lead	40	40	100%	0	0.0%	
	Sulfur	40	40	100%	0	0.0%	
	Zinc	40	40	100%	0	0.0%	
DRRC-086	Silver	55	55	100%	0	0.0%	Silver grades are rounded to integers, no decimals
	Gold	55	55	100%	1	1.8%	1 intervals has Au check assays, averages used
	Copper	55	55	100%	0	0.0%	
	Lead	55	55	100%	0	0.0%	
	Sulfur	55	55	100%	0	0.0%	
	Zinc	55	55	100%	0	0.0%	
DRRC-136	Silver	39	39	100%	0	0.0%	Aluminum Grade in Silver column
	Gold	39	39	100%	39	100.0%	
	Copper	39	39	100%	39	100.0%	Iron Grades in Copper Column
	Lead	39	39	100%	39	100.0%	Sulfur Grade in Lead Column
	Sulfur	39	39	100%	39	100.0%	Antimony (Sb) grades in sulfur column
	Zinc	39	39	100%	39	100.0%	No values, all no assay
DRRC-162	Silver	14	14	100%	0	0.0%	Silver grades are rounded to integers, no decimals
	Gold	14	14	100%	0	0.0%	
	Copper	14	14	100%	0	0.0%	
	Lead	14	14	100%	0	0.0%	
	Sulfur	14	14	100%	0	0.0%	
	Zinc	14	14	100%	14	100.0%	14 intervals with missing zinc, Certifcate is Available

12.2 Standards

Polimetal inserts known standards into the sample stream for laboratory assay. Two of the standards are for gold, providing confirmation at 0.63 and 3.84 gm/t respectively. The third standard is a base metal standard with certified gold value also. The names of the standards and their values are:

Name	Source	Certified Value
G907-4	Geostats Pty Ltd	3.84 gm/t gold
G910-8	Geostats Pty Ltd	0.63 gm/t gold
GBM398-1	Geostats Pty Ltd	0.183 gm/t Gold
		1.482 % Copper
		2.030 % Zinc
		2.667 % Lead
		5.100 gm/t Silver

Multi-element neutron activation analysis was run on all samples for matrix identification. Neutron activation values are not certified values and were not used in the IMC analysis of the standards results.

During Phase 1 drilling, all assays were submitted to SGS the laboratory. During that period, the gold standards of G907-4 and G910-8 were submitted as part of the sample stream. The total number of standard submissions during the Phase 1 drilling amounts to 44 samples out of 1082 samples (1 in 25).

Standard Submitted to SGS in Phase 1 Drilling

Standard	Number	SGS Average	Max	Min	Certified Value
G907-4	23	3.952 gm/t gold	4.09	3.86	3.840 gm/t gold
G910-8	21	0.592 gm/t gold	0.65	Trc	0.630 gm/t gold

The following summarizes the review of Standards submitted to ALS Chemex data to date. Phases 1, 2 and 3 standards results have been combined. The table below summarizes the results of the ALS submissions

Standards Submitted to ALS Chemex in All Phases of Drilling

Standard	Number	ALS Average	Max	Min	Certified Value
G907-4	415	3.846 gm/t Au	4.20	0.19	3.845 gm/t Au
G910-8	913	0.595 gm/t Au	3.95	Trc	0.630 gm/t Au
GBM389-1	243	0.207 gm/t Au	3.81	0.01	0.183 gm/t Au
	244	5.020 gm/t Ag	5.90	0.25	5.100 gm/t Ag
	243	1.503 % Cu	1.69	0.004	1.482 % Cu
	241	2.030 % Zn	2.24	0.005	2.030% Zn

The minimum value of G907-4 is nearly identical to the certified value of GBM389-1, implying that at least one sample swap has occurred. The maximum value for G910-8 at 3.61 is likely a swap with G907-4 and the trace result for G910-8 is most likely a swap of a blank.

Table 12-3 summarizes the results of the Polimetal inserted certificates. Of the 1,572 submitted standards about 2% are more than 10% different from the original value. Other than the low gold value of GBM389-1, there does not appear to be an issue with biased reporting of standards. The low gold value of GBM389-1 would not have a measureable impact on the estimate of mineral resources.

The greatest concern regarding the use of standards is the small number of inserted standards for base metals. IMC received roughly 240 assay values for copper, zinc, and silver standards out of 31,495 total assays, this is less than one base metal assay per drill hole. It is understood that the base metal component of the oxide portion of the deposit is minimal, but most of the tonnage within the mineral resource is sulfide and a significant portion of the economic value of sulfides is in the base metals.

IMC obtained 1,571 total gold standards out of the assay data base of 31,495 assays. This amounts to an average insertion rate of 1 out of every 20 samples. This is more typical of the rate that is expected on standards insertion.

With only 244 base metal standards used, there is little to assure the unbiased assay of copper, zinc, and silver within the deposit.

Table 12-3 summarizes the results of the standards that were sent to IMC with data base transfer which is dated 5 August 2015.

Table 12-3
Summary of Submitted Standards Results, ALS Chemex

Standard Name	Metal	Number	Certified Value PPM	Mean of Reported Results	Number Out of Tolerance By 10%	Bias Negative is Low
G907-4	Gold	415	3.840	3.846	7	0.17%
G910-8	Gold	913	0.630	0.595	10	-5.48%
GBM389-1	Gold	243	0.183	0.207	14	12.91%
GBM389-1	Silver	244	5.100	5.020	9	-1.58%
GBM389-1	Copper	243	14,823	15,030	9	1.40%
GBM389-1	Zinc	241	20,295	20,273	9	-0.11%

12.3 Blanks

Blank samples were inserted into the sample stream by Polimetal prior to shipment of the samples to the assay labs. They are typically inserted as the last sample at the end of a drill hole to assure no carryover of values.

In total there are 1,134 blanks that were inserted into the sample stream which results in an average insertion rate of 1 out of 27.8 samples for all phases of drilling to date.

The following numbers of blanks assays were reported from labs along with the number of samples that reported high enough grade to reflect economic mineralization.

	Number of Inserted Blanks	Number Above Trace Level		Number Above Ore Value	
Gold	1,134	0.005 ppm	30	0.05 ppm	1
Silver	1,126	0.25 ppm	17	1.00 ppm	0
Copper	1,134	50 ppm	32	500 ppm	0
Zinc	1,134	50 ppm	82	500 ppm	0

The out of tolerance percentage is small; there is only 1 gold value that approaches minimum interesting economic gold grades. One drill hole DRD-100 has the sample numbers accidently placed in the silver blanks column. These 3 intervals have been removed from the blanks review.

In summary, the blank insertion results are acceptable and indicate that there is little occurrence where blank samples report as unacceptably high.

12.4 Duplicate Assays

Polimetal procedure is to periodically submit split core or RC coarse rejects for duplicate analysis. The duplicate assays are resubmitted to the same lab where the original is run. Consequently, the duplicates are not an independent check of the assays. The intent of the duplicate is to confirm the consistency of sample preparation and assay procedure.

IMC has identified 1,219 duplicate assays within the Polimetal data base which equates to duplicates run every 25th interval.

IMC has analyzed the duplicate results against the original assays with XY scatter plots. The scatter plots show good correlation between the original and the duplicate. The duplicate assay data supplied to IMC were not capped and all high grade assays were included for review.

Table 12-4 summarizes a comparison of the original assay versus the duplicate results. Low grade trace values are removed from the analysis in order to focus on the grade range of economic values. Statistical hypothesis tests have been applied to compare the two populations

Table 12-4
Comparison of Original vs Duplicate Assays

Metal	Number of Duplicates	Cutoff Grade for Precision	Original Mean	Duplicate Assay Mean	Hypothesis Test Results at 95% Conf			
					T-Test	Paired T	Binomial	KS
Gold	1205	0.001 gm/t	0.192 gm/t	0.192 gm/t	Pass	Pass	Pass	Pass
Silver	1205	0.10 gm/t	7.653 gm/t	7.510 gm/t	Pass	Pass	Fail	Fail
Copper	1190	0.01 % Cu	0.148 %	0.148 %	Pass	Pass	Pass	Pass
Zinc	1198	0.001 % Zn	0.291 %	0.295 %	Pass	Pass	Pass	Pass

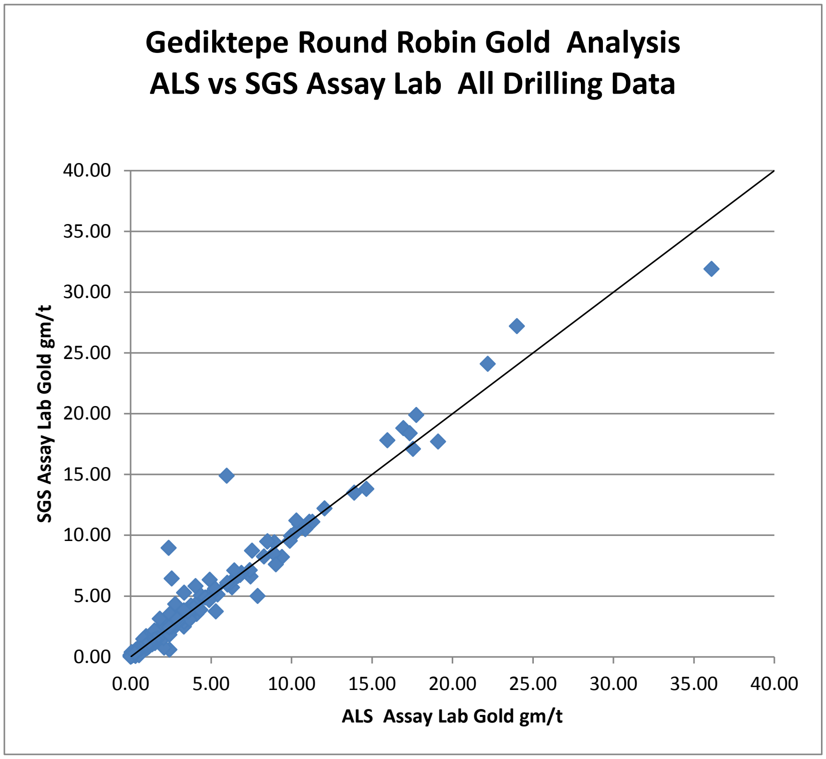
In summary, the duplicate results confirm the original samples and their assay values. This data set includes all values of the duplicate assays and they have not been capped.

12.5 Round Robin Third Party Check Assays

Polimetal provided additional QAQC information several months after the original data transfer which constituted a set 269 check assays by third party assay laboratories. Polimetal selected 269 prepared pulps and sent them to Acme labs and SGS labs for independent assay. IMC prepared X-Y plots of the original ALS assays versus the Acme and SGS labs.

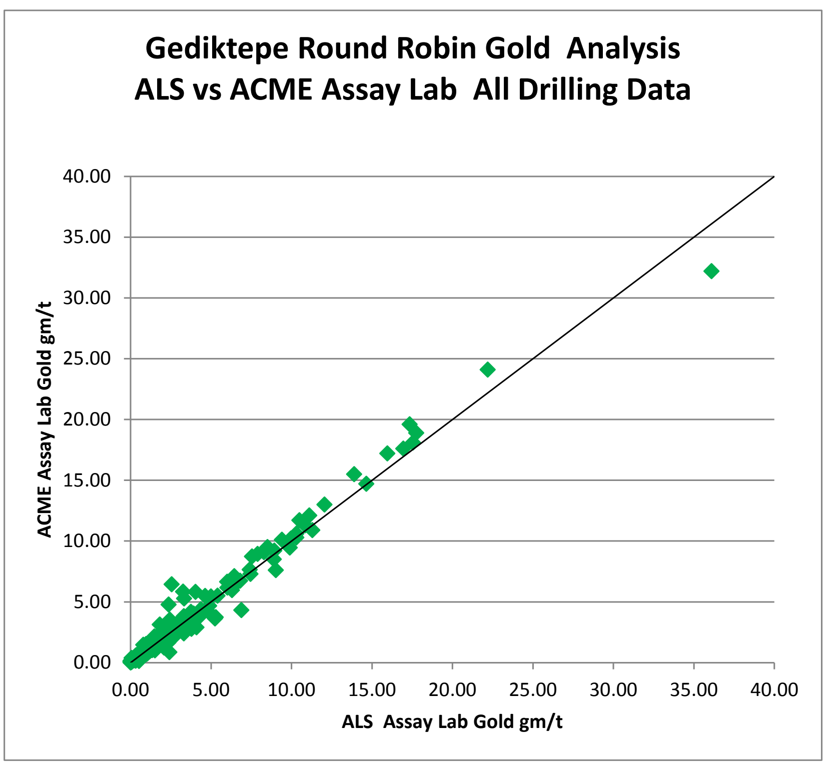
Figure 12-1 summarizes the results of the checks by lab showing the number and mean grade of each of the assay labs gold results. Only gold has been analyzed during the round robin results.

Figure 12-1
Round Robin Check Assay for Gold



Gold Check Assays
ALS vs SGS

Number of checks 269
Mean ALS= 2.885 gm/t
Mean SGS = 3.007 gm/t



Gold Check Assays
ALS vs ACME

Number of checks 265
Mean ALS = 2.740 gm/t
Mean ACME = 2.842 gm/t

Results of the round robin third party checks on gold assays are acceptable. Results from both SGS and Acme reported slightly higher values than ALS Chemex above about 15 gm/t. The high grade value that was reported by ALS as more than 35 gm was reported back as a similar high result by both check labs.

Unfortunately the only available round robin check assays are for gold, the number is not large. IMC recommends that third party check assays be completed by resubmitting pulps on a 1 in 20 basis for assay of gold, silver, copper, and zinc.

As with the standards, there is minimal verification of the base metal results at Gediktepe.

12.6 Diamond Drilling Compared to Reverse Circulation

The Gediktepe deposit has been drilled primarily with diamond core drilling (DD). The early drill program applied reverse circulation (RC) drilling to the outside fringes of the deposit and diamond core drilling (DD) to the center core of the deposit. As additional in-fill drilling was completed, the majority of the in-fill drilling was DD, but some of the infill drilling completed in 2014-2015 included RC holes.

A comparison was made between the DD and RC results on a nearest neighbor basis to understand if the RC data was reliable relative to diamond. Individual assay intervals were used in the nearest neighbor analysis.

IMC utilized a nearest neighbor procedure where DD assays were compared to the nearest RC data within the oxide and sulfide populations. The portion of the Gediktepe deposit that is the subject of this study has now been infill drilled to 25m grid. As a result, most drill hole assays are approximately 25 meters from the nearest assay in another drill hole. The maximum sample spacing for the nearest neighbor comparison was set at 25m in order to mirror the new drill hole spacing.

Table 12-5 summarizes the results of the analysis. When selecting the data, the trace and zero values were not included in the paired data sets in order to make the comparison reasonable.

Table 12-5
Diamond Drilling Assays Compared to Reverse Circulation Assay
Cutoff Grades Applied to Remove Zero Values from the Analysis

Metal	Population	Cutoff Grade	Separation Meters	Number of Pairs	DDH Mean	Reverse Circ Mean	Hypothesis Test Results at 95% Conf			
							T-Test	Paired T	Binomial	KS
Gold	Oxide	0.05 gm/t	25	133	2.540 gm/t	1.411 gm/t	Fail	Fail	Fail	Fail
	Sulfide	0.05 gm/t	25	256	0.611 gm/t	0.547 gm/t	Pass	Pass	Pass	Fail
Silver	Oxide	1.00 gm/t	25	199	40.508 gm/t	40.122 gm/t	Pass	Pass	Pass	Pass
	Sulfide	1.00 gm/t	25	630	12.014 gm/t	10.799 gm/t	Pass	Pass	Pass	Pass
Copper	Oxide	0.01 %	25	193	0.108 %	0.103 %	Pass	Pass	Fail	Pass
	Sulfide	0.01 %	25	989	0.348 %	0.364 %	Pass	Pass	Pass	Pass
Zinc	Oxide	0.01 %	25	180	0.066 %	0.104 %	Fail	Fail	Pass	Pass
	Sulfide	0.01 %	25	1172	0.575 %	0.389 %	Fail	Fail	Fail	Fail

In almost all cases, the RC average grade is equal to or lower than the DDH grade. This may be due to the location of the RC drilling, which is mainly on the Southern and Eastern fringe of the deposit. Most of the hypothesis tests pass at the 95% confidence level.

Three sets of twin holes have been drilled at Gediktepe; the pairs are given:

DRRC-062,DRD-142 DRRC-001,DRD-053 DRRC-002,DRD-051

The assays in these holes are included in the nearest neighbor analysis above. When only the twinned hole data is used in the nearest neighbor analysis, the average grades were more similar and all of the hypothesis tests pass at the 95% confidence level.

IMC completed the same type of analysis using the composited drill hole data that was used for block grade estimation. The results indicated the same trend. Within the oxide zone, the RC drilling has a lower gold grade than the DD drilling. The RC data in the oxide or Gossan zones appears to be low biased compared to DD. Inclusion of the RC data is likely producing a conservative estimate of gold grade in the gossan-oxide zone.

In all other zones of the deposit, the RC drilling is more similar to the DD information and appears to be statistically reliable. All RC data was incorporated into the estimation of mineral resources and mineral reserves even though its incorporation is likely conservative.

Downhole plots of assays from the twinned RC and DD holes were reviewed to check for downhole contamination in RC drilling. Two of the 3 plots did not show noticeable downhole contamination in the RC holes. Minimal downhole contamination at sub-economic grades was potentially observed in the 3rd plot.

12.7 QAQC Recommendations

The following procedural changes are suggested for future drill programs at Gediktepe.

1) More base metal standards should be inserted and a range of standard values should be used covering a range of copper and zinc grades. Standards should be inserted on a 1 in 20 or 1 in 25 basis.

2) IMC recommends that a regular and thorough check assay procedure be implemented where a third party check is run every 20 samples.

3) It is recommended that the silver grades in the assay data base be carried to one decimal place.

4) When more than one assay is available for an interval, they should not be averaged. The first assay should be used and the second stored as a duplicate or check.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

A sufficient amount of testwork has been performed by several testing laboratories for the Gediktepe prospect to support a Pre-Feasibility Study for the oxide and sulfide ores. The primary objective of the testwork was to develop process flowsheets for treating both oxide and sulfide ores from the prospect. A brief review of the metallurgical testwork is presented in the following sections based on the review of the following reports:

1. Metallurgical Testing of Oxide Samples from Gediktepe Prospect, Turkey, RDi Report, January 13, 2015
2. Metallurgical Testing of Sulfide Samples from Gediktepe Prospect, Turkey, RDi Report, June 2, 2015
3. Optimization of Gediktepe Cu-Zn Sulfide Flotation Conditions, Hacettepe Mineral Technologies, Ankara, Turkey, August 5, 2015
4. Report on Oxide Ore Metallurgical Test Programme Update, SGS Minerals Services UK Limited, February 1, 2016
5. Gold Deportation and Qemscan Study on Two Metallurgical Samples from the Polimetal Madencilik Copper-Zinc-Lead Deposit, Turkey, SGS Canada Inc., February 8, 2016
6. The Mineralogical Characteristics of Eight Feed Samples from Turkey, SGS Canada Inc., February 4, 2016

13.1 Oxide Ores

The majority of the testwork was performed by Resource Development Inc. (RDi). Once the process flowsheet was established, confirmation heap leach testwork was performed at SGS Minerals Services UK Limited.

13.1.1 Feed Preparation and Characterization

There are two ore types constituting the oxide ore, namely Gossan and disseminated. Drill core rejects from about 30 holes were received at RDi to produce the two oxide composites. The two composites were blended in the following proportions to produce a master oxide composite for the metallurgical study: 35% Gossan and 65% disseminated oxide.

All the samples constituting each of the two composites (Gossan and disseminated oxide) were mixed together, blended thoroughly and split into two parts. One part was saved and the other part was stage crushed to P_{100} of 6 mesh. The crushed samples were thoroughly blended and split into 1-kg and 10-kg charges.

A 1-kg charge of each composite was pulverized to 150 mesh and representative splits taken out for head analyses. The test results, given in Tables 13-1 and 13-2, indicated the following:

- The Gossan composite assayed 4.447 g/t Au, 97.5 g/t Ag, 0.574% Pb, 0.0956% Zn, 0.1014% Cu, 1.10% S_{Total} and 0.11% $C_{Organic}$.

- The disseminated oxide composite assayed 1.671 g/t Au, 59.9 g/t Ag, 0.294% Pb, 0.0992% Zn, 0.1236% Cu, 2.96% S_{Total} and 0.11% $C_{Organic}$.

- The Gossan composite was fully oxidized (<0.01% $S_{Sulfide}$) whereas the disseminated oxide sample had only 24.7% of sulfur as sulfate sulfur.

- Both composites contained almost all carbon as organic carbon.

- Gossan composite contained much higher amounts of precious metals than the disseminated oxide composite.

Table 13-1
Head Analyses of Two Oxide Composite Samples

Element	Sample	
	Gossan Ore	**Disseminated Oxide**
Au, g/t	4.447 (4.162, 4.731)	1.671 (1.755, 1.589)
Ag, g/t	97.5 (97.2, 97.8)	59.9 (60.2, 59.6)
Pb, %	0.574	0.294
Zn, %	0.0956	0.0992
Cu, %	0.1014	0.1236
S_{Total}, %	1.10	2.96
$S_{Sulfide}$, %	<0.01	2.23
$S_{Sulfate}$, %	1.10	0.73
C_{Total},%	0.11	0.13
$C_{organic}$,%	0.11	0.11
$C_{inorganic}$,%	<0.01	0.02

Table 13-2
ICP Analyses of Two Oxide Composite Samples

Element	Gossan Ore	Disseminated Oxide
Percent		
Al	1.61	4.45
Ca	0.05	0.10
Fe	26.96	14.16
K	0.59	1.64
Mg	0.44	1.16
Na	0.08	0.42
Ti	0.07	0.07
ppm		
As	1705	462
Ba	16034	5360
Bi	333	97
Cd	42	21
Co	<1	<1
Cr	42	46
Cu	1045	1292
Mn	174	307
Mo	7	1
Ni	7	6
Pb	6485	3004
Sr	75	29
V	45	60
W	<10	<10
Zn	1006	800

Following the preparation of the master oxide composite, it was also submitted for head analyses.

The results, given in Tables 13-3 to 13-5, indicated the following:

- The master composite assayed 2.403 g/t Au, 67.9 g/t Ag, 0.426% Pb, 0.0874% Zn, 2.20% S_{Total} , 1.37% $S_{Sulfide}$ and 0.14% $C_{Organic}$.

- ICP analyses indicated that the composite assayed 723ppm As and 1208ppm Cu.

- The metallic assays indicated that some coarse gold may be present in the sample. However, there was no coarse silver present in the sample.

Table 13-3
Head Analyses of Oxide Master Composite

Element	Assay
	Oxide Composite
Au, g/t	2.403 (2.410, 2.396)
Ag, g/t	67.9 (68.2, 67.6)
Pb, %	0.426
Zn, %	0.0874
S_{Total}, %	2.20
$S_{Sulfide}$, %	1.37
$S_{Sulfate}$, %	0.83
C_{Total},%	0.16
$C_{organic}$,%	0.14
$C_{inorganic}$,%	0.02

Table 13-4
ICP Analyses of Master Oxide Composite

Element	Oxide Composite
Percent	
Al	3.32
Ca	0.09
Fe	17.86
K	1.27
Mg	0.90
Na	0.34
Ti	0.07
ppm	
As	723
Ba	6319
Bi	182
Cd	29
Co	<1
Cr	27
Cu	1208
Mn	268
Mo	3
Ni	<5
Pb	4224
Sr	33
V	57
W	<10
Zn	880

Table 13-5
Metallic Assays for the Oxide Master Composite

Sample	Minus 150 Mesh		Plus 150 Mesh		Cal Feed Grade, g/t
	Weight, gms	Assays g/t	Weight, gms	Assays g/t	
GOLD VALUES					
Oxide	952.4	2.986, 2.372, 2.921, 2.389	40.58	4.054	2.724
SILVER VALUES					
Oxide	952.4	78.286, 70.046, 69.227, 81.801	40.58	61.505	74.2951

The assay-by-size data, given in Table 13-6 indicates that gold and silver minerals are distributed in all size fractions in the same proportion as the weight of the sample.

Table 13-6
Distribution of and Silver by-Size in Oxide Master Composite

Size (mesh)	Assay g/t		Distribution %		
	Au	Ag	Wt.	Au	Ag
+10	2.76	68.8	25.7	27.6	24.7
10 X 14	2.18	62.6	13.3	11.3	11.6
14 X 100	2.39	63.0	31.7	29.5	27.8
100 X 400	2.59	74.6	12.6	12.7	13.1
-400	2.93	98.4	16.6	18.9	22.7

13.1.2 Mineralogical Evaluation

The two composite samples, namely Gossan and disseminated oxide, were mineralogically evaluated to determine the bulk mineralogy with an emphasis on gold and silver mineralogy. The highlights of the study indicated the following:

- Iron oxide in the form of goethite dominated the two oxide composites and occurred as fine grained, granular material and large masses. Both samples carried goethite pseudomorphs after pyrite. Magnetite showed advanced replacement by hematite.

- An extensive search failed to identify discreet silver mineralogy as a sulfide or native silver.

- A few 2 to 3 micron size grains of gold were seen in granular iron oxide and appeared to be liberated.

13.1.3 Comminution Studies

The comminution studies were undertaken on the master composite and the individual oxide composites. The tests included abrasion index, crushability work index, Bond's ball mill work indices and SMC testing. The test results indicated the following:

- The abrasion index for the master oxide composite was 0.1184 which indicated that the ore is abrasive.
- The crushability work index for the Gossan sample was 7.4 kwh/mt.
- The Bond's ball mill work index at 100 mesh grind size was 10.41 kwh/mt for Gossan composite and 9.69 kwh/mt for disseminated composite.

13.1.4 Gravity Tests

The objective of the gravity tests was to determine if one could recover free gold from the ore in a concentrate which could be directly smelted. Rougher and cleaner gravity tests were performed at P_{80} of 48 and 100 mesh using the flowsheet given in Figure 13-1. The test data, summarized in Table 13-7, indicated that the concentrate was too low for direct smelt and the recoveries of gold and silver were also very low. Hence, gravity concentration process will not work for this prospect.

Table 13-7
Gravity Concentration Test Results for Oxide Master Composite

Product	**Assay g/t**		**Distribution %**		
	Au	**Ag**	**Wt.**	**Au**	**Ag**
P_{80} 48=Mesh Grind (T-1)					
Gemeni Concentrate	15.77	96.8	1.7	10.7	2.5
Gemeni Tailing	3.80	83.7	4.8	7.5	6.2
Cal. Knelson Concentrate	6.87	87.1	6.5	18.1	8.7
Knelson Tailing	2.16	63.6	93.5	81.9	91.3
Cal. Feed	2.47	65.1	100.0	100.0	100.0
P_{80} 100=Mesh Grind (T-2)					
Gemeni Concentrate	12.0	92.4	2.6	12.7	3.4
Gemeni Tailing	2.61	58.2	4.1	4.4	3.4
Cal. Knelson Concentrate	6.23	71.4	6.6	17.2	6.9
Knelson Tailing	2.13	68.7	93.4	82.8	93.1
Cal. Feed	2.40	68.9	100.0	100.0	100.0

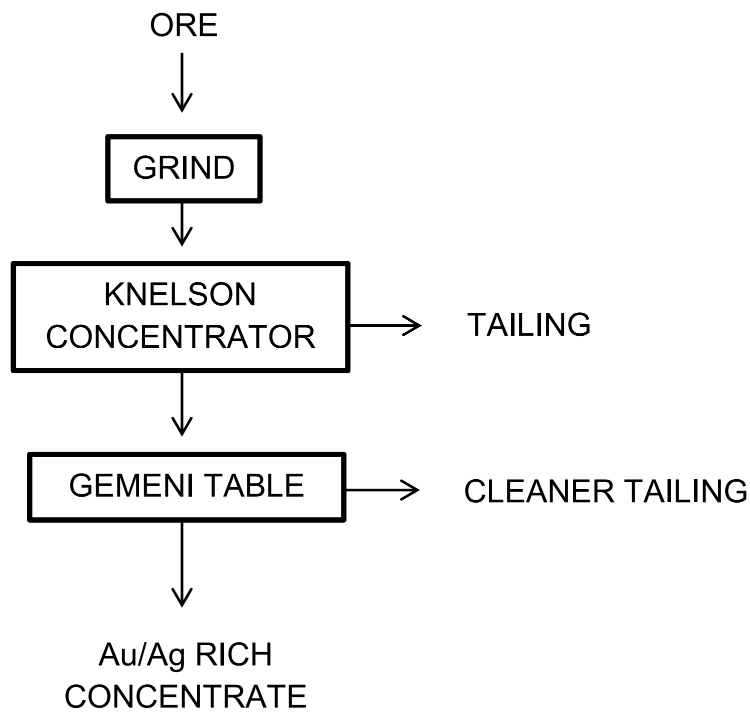


Figure 13-1
Gravity Concentration Test Flowsheet

13.1.5 Cyanidation Leach Tests

A series of cyanidation leach tests were performed on the master composite to evaluate the effect of process parameters on gold and silver extraction. The process parameters evaluated were grind size, leach time, carbon-in-leach (CIL), cyanide concentration, leach pulp density and addition of lead nitrate.

The selected test results are summarized in Tables 13-8 and 13-9. The test results indicate the following:

- The gold extraction ranged from 87.1% to 90.8% and the silver extraction ranged from 59.9% to 66.2% in 72 hours of leach time for primary grind of P_{80} of 65 to 200 mesh.

- The majority of the gold and silver were leached in 48 hours of leach time.

- The ore did not exhibit preg-robbing properties.

- The addition of lead nitrate had no effect on the leach time of gold extraction but improved silver extraction by 2% to 6%.

- A cyanide concentration of 0.75 g/L or higher needs to be maintained in the leach circuit to optimize gold and silver extraction.

- Rheology tests indicated that leaching could be undertaken at 45% to 50% solids.

Table 13-8
Cyanidation-Leach Test Results for Oxide Composite

Parameter	**Grind, P_{80} mesh**									
	65 (T-1)		**100 (T-2)**		**150 (T-3)**		**200 (T-4)**		**150 (T-5)**	
	Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**
Extraction %										
6 hrs.	71.3	29.4	74.4	28.2	57.4	14.3	47.1	6.7	-	-
24 hrs.	81.9	52.8	83.7	54.0	85.3	55.0	86.1	57.8	-	-
48 hrs.	86.3	58.7	87.6	58.5	88.5	61.2	89.3	64.9	-	-
72 hrs.	87.1	59.9	88.4	62.0	89.3	63.4	90.8	66.2	-	-
Carbon, g/t	-	-	-	-	-	-	-	-	90.0	64.9
Residue, g/t	0.32	26.40	0.28	26.20	0.27	24.00	0.23	22.40	0.27	26.8
Cal. Feed, g/t	2.43	65.80	2.41	68.96	2.52	65.49	2.52	66.20	2.67	76.5
Reagent Consumption, kg/t										
NaCN	1.368		1.383		1.803		1.983		2.313	
Lime	5.597		5.602		5.498		5.550		5.587	

Table 13-9
Effect of NaCN Concentration on Leaching of
Master Composite Oxide Ore (P80 = 100 Mesh)

Parameters	**NaCN Concentration, g/L**									
	1.0 (maintained) (T-2)		**1.0 (decay) (T-15)**		**0.75 (maintained) (T-13)**		**0.75 (decay) (T-14)**		**0.5 (maintained) (T-12)**	
Extraction %	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**
6 hrs.	74.4	28.2	62.9	20.2	56.6	12.9	45.9	10.8	40.4	1.9
24 hrs.	83.7	54.0	67.1	28.5	82.6	53.1	50.3	9.3	69.4	27.6
48 hrs.	87.6	58.5	73.6	35.6	85.3	62.0	52.5	9.4	81.2	51.7
72 hrs.	88.4	62.0	76.8	39.1	87.9	64.7	52.5	9.6	85.6	58.4
Residue, g/t	0.28	26.20	0.60	39.40	0.31	23.40	1.36	59.20	0.34	27.40
Cal. Feed, g/t	2.41	68.96	2.58	64.70	2.51	66.23	2.87	65.48	2.38	65.85
Reagent Consumption, kg/t										
NaCN	1.383		1.325		1.414		1.014		1.229	
Lime	5.602		5.715		5.216		5.530		5.586	

13.1.6 Coarse Ore Cyanidation Tests

Bottle roll cyanidation tests were performed on coarse ore to determine amenability of heap leach process for extraction of precious metals. The tests were performed on 18 mm, 12.5 mm, 6.35 mm and 6 mesh material. The test results, summarized in Table 13-10, indicated that the oxide ore is amenable to heap leach process. Gold and silver extractions of 79% and 49.3% respectively were obtained at a crush size of 18 mm. These results were confirmed in the static bucket leach tests.

Table 13-10
Cyanidation Test Results for Coarse Size Ore

Parameters	**Crush Size, ins**									
	P_{100} = 18 mm (T-17)		**P_{100} = 12.5 (T-16)**		**P_{100} = 12.5 mm (T-18)**		**P_{100} = 6.35 mm (T-19)**		**P_{100} = 6 mesh (T-6)**	
Extraction %	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**
6 hrs.	56.9	28.9	76.1	29.5	61.2	29.1	71.3	26.6	61.4	33.3
24 hrs.	68.4	38.8	81.6	34.8	71.7	38.9	78.9	34.3	75.4	47.6
48 hrs.	71.8	44.4	85.1	39.6	75.1	43.2	83.2	38.1	79.1	52.9
72 hrs.	75.0	46.8	87.9	40.6	77.6	45.7	85.4	40.5	83.3	53.6
96 hrs.	79.0	49.3	88.5	41.6	79.3	48.2	87.3	42.3	-	-
120 hrs.	-	-	89.9	42.4	-	-	-	-	-	-
144hrs.	-	-	88.5	42.8	-	-	-	-	-	-
Residue, g/t	0.57	30.20	0.24	38.0	0.70	40.40	0.38	53.20	0.43	29.0
Cal. Feed, g/t	2.71	59.59	2.11	66.38	3.38	77.93	3.03	92.23	2.43	62.47
Reagent Consumption, kg/t										
NaCN	0.780		0.420		0.907		1.027		1.561	
Lime	6.079		8.170		6.961		7.204		4.971	

13.1.7 Column Leach Tests

Two column leach tests were performed for a leach period of 45 days. One test had a P_{100} of 18 mm and the other had a P_{100} of 12.5 mm. The ore was agglomerated with 2.25 kg/t of cement and 1.81 kg/t of lime and loaded into 100 mm diameter columns. The material was cured for two days and 1 g/L NaCN solution applied at the rate of 0.2 L/min/m^2.
The column leach tests, summarized in Table 13-11, indicated gold extraction of ± 87.5% and silver extraction of ± 50%. The NaCN consumption was ± 2 kg/mt. The pregnant solution analyses, given in Table 13-12, indicated that significant amount of copper was also leached.

Based on these results, it is reasonable to conclude that the oxide ore should be heap leached. This is based on the fact that gold extraction is almost the same in the heap leach process and in the agitated cyanide leach process.

Table 13-11
Summary of Column Leach Test Results

Test No.	Crush Size P_{80}, mm	Leach Time Days	Extraction %		Residue, g/t		Cal. Feed g/t		Reagent Consumption, Kg/t	
			Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
1	18	45	87.5	50.6	0.38	39.0	3.04	78.95	2.02	6.729
2	12.5	45	87.7	48.4	0.41	41.0	3.30	79.53	1.916	6.773

Table 13-12
Pregnant Solution Analyses for the Oxide Column Leach Tests

Element	P_{100} = 18 mm Column (1)			P_{100} = 12.5 Column (2)		
	Days 1-5	Days 20-25	Days 40-45	Days 1.5	Days 20-25	Days 40-45
Al	0.1	0.1	0.4	0.2	0.1	0.8
As	<0.1	0.1	0.1	0.0	0.1	0.1
Ca	371	1.3	1.0	399	2.3	1.4
Cu	94.4	11.0	4.0	41	11.0	5.5
Fe	<0.1	0.9	0.2	<0.1	0.7	0.7
K	17.2	4.6	6.1	16.9	3.1	4.9
W	0.2	0.6	0.3	0.1	0.5	0.5
Zn	16.9	3.0	1.3	15.3	3.1	1.6

Note: Values for Ba, Bi, Cd, Co, Cr, Mg, Mn, Mo, Ni, Pb, Sr, Ti and V were <0.1 ppm.

13.1.8 Confirmation Heap Leach Test work at SGS Minerals Services UK Limited

SGS received 200 individually packaged samples, weighing from 400 grams to 4500 grams, for the confirmation test work. These samples represented the following ore types: gossan, low gossan, high gold and high base metals.

Composite samples prepared for test work included gossan, low gossan, high gold, high base metal, low base metal, middle, south and north zones. The head analyses of these composites is given in Table 13-13 and the screened metallic assays for the two gossan composites are given in Tables 13-14 and 13-15. Metallic assays indicate the presence of coarse gold in the low gossan sample.

Table 13-13
Head Analyses of Oxide Composites

Sample	1024A Gossan	2024A Low Gossan	3006A High Au	4006A High BM	5006A Low BM	6006A Middle	7006A North	8006A South
(%)								
Cu	0.085	0.047	0.12	0.36	0.031	0.11	0.064	0.13
Pb	0.38	0.16	0.44	0.27	0.26	0.29	0.3	0.36
Zn	0.098	0.071	0.16	0.15	0.052	0.13	0.052	0.087
As	0.217	0.054	0.196	0.106	0.06	0.171	0.094	0.135
Cd	0.0001	<0.0001	0.0004	0.0003	<0.0001	0.0002	<0.0001	<0.0001
Ni	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Co	<0.001	0.001	<0.001	0.001	0.001	<0.001	<0.001	0.001
Mn	0.016	0.031	0.016	0.031	0.005	0.029	0.024	0.029
Bi	0.0187	0.0042	0.0248	0.0116	0.005	0.0213	0.0105	0.0165
Sb	0.0327	0.0113	0.0529	0.0111	0.0109	0.0215	0.0136	0.0304
Hg	0.0004	0.0002	0.0038	0.0004	0.0002	0.0004	0.0004	0.0006
Te	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Se	0.0023	0.0008	0.0036	0.0037	0.003	0.0025	0.0034	0.0043
SiO2	29.66	56.59	24.89	36.27	47.24	36.53	49.62	44.16
Al	1.65	3.57	1.13	2.27	3.55	2.09	2.55	2.21
Fe	21.47	5.44	23.79	15.77	5.32	17.97	10.58	12.97
Mg	0.48	0.63	0.41	0.95	1.2	0.56	0.38	0.36
Cr	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Ca	2.27	1.52	1.13	0.97	1.44	1.56	1.95	1.03
Na	0.007	0.01	0.007	0.007	0.007	0.006	0.01	0.008
K	0.16	0.21	0.13	0.12	0.21	0.2	0.23	0.16
S(tot)	2.52	0.56	4.53	6.92	1.52	2.87	0.92	2.59
S(sol)	0.33	0.12	0.36	0.36	0.15	0.35	0.22	0.2
C(tot)	0.26	0.24	0.23	0.22	0.2	0.24	0.23	0.26
S(org)	0.04	0.05	0.05	0.04	0.05	0.05	0.05	0.06
Cl	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
F	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
(mg/kg)								
Au	3.57	0.87	10.28	1.37	1.19	2.2	2.14	3.95
Ag	44.2	29	181.8	124.1	13.4	35.7	66.8	120.3

Table 13-14
Screened Metallic Result for the Gossan Sample

Fraction	Wt. %	Assay Au g/t	Assay Ag g/t	Dist. Au %	Dist. Ag %
Oversize	2.44	4.41	39.00	2.83	2.37
Undersize Undersize duplicate	97.56	3.82 3.75	40.20 40.30	97.17	97.63
Sample	100.00	3.80	40.22	100.00	100.00

Table 13-15
Screened Metallic Result for the Low Gossan Sample

Fraction	Wt. %	Assay Au g/t	Assay Ag g/t	Dist. Au %	Dist. Ag %
Oversize	1.11	8.13	17.90	7.86	0.67
Undersize Undersize duplicate	98.89	0.98 1.16	30.20 29.10	92.14	99.33
Sample	100.00	3.80	40.22	100.00	100.00

Each composite sample was cyanide leached at a fine grind of P_{100} of 75 micrometers for 48 hours. The test results are summarized in Table 13-16. The test results indicate the following:

- Gold and silver extractions ranged from 86% to 96% and 47% to 78% respectively for a leach time of 48 hours.
- The copper extraction was extremely variable and ranged from 7% to 53% whereas the zinc extraction was low (i.e. <3.5%).
- The cyanide consumption was reasonable for all samples (<0.8kg/t) except for high base metal composite.

These results confirmed that the oxide portion of the deposit was amenable to direct cyanidation for gold and silver extraction.

Table 13-16
Cyanidation Bottle Roll Leach Tests for Composite Samples at P100 of 75 microns
(Leach Time: 48 hours)

Sample	Extraction %				Cal. Head				Reagent Consumption, Kg/t	
	Au	Ag	Cu	Zn	Au, g/t	Ag, g/t	%Cu	%Zn	NaCN	CaO
Gossan	88.84	51.09	6.99	2.75	3.63	40.17	0.08	0.10	0.88	4.62
Low Gossan	86.57	72.03	7.17	2.04	1.08	29.49	0.05	0.07	0.44	3.56
High Gold	90.68	54.17	23.92	3.41	11.21	166.06	0.11	0.17	1.51	4.29
High Base Metals	89.87	70.41	38.74	1.18	1.53	111.35	0.33	0.15	7.17	2.32
Low Base Metals	94.98	47.65	7.05	2.67	1.39	13.94	0.03	0.05	0.50	1.72
Middle Zone	95.95	75.34	19.32	3.43	2.01	32.25	0.10	0.12	1.52	4.68
North Zone	92.58	78.39	19.99	1.93	2.22	58.04	0.06	0.05	0.81	2.70
South Zone	88.06	72.83	53.55	3.23	3.56	114.11	0.12	0.09	0.31	2.74

Coarse ore cyanidation tests were performed in duplicate with the two gossan composite at varying crush sizes. The test at 3.35mm was performed as a single test. The test results, summarized in Tables 13-17 and 13-18, indicate the following:

- Good gold extraction (± 80%) was obtained for both composites at a relatively coarse size of P_{100} of 25mm.
- The gold extraction for the gossan composites did not appear to be size dependent. However, silver extraction improved slightly with finer crush size for the gossan sample.
- Both copper and zinc extractions were independent of the crush size.
- The cyanide consumption was higher for the tests at coarser size which is contradictory to what one would expect.

Table 13-17
Coarse Ore Cyanidation for Gossan Ore at Varying Crush Sizes
(Leach Time: 336 hours)

Crush Size, P_{100} mm	**Extraction %**				**Cal. Head**				**Reagent Consumption, Kg/t**	
	Au	**Ag**	**Cu**	**Zn**	**Au, g/t**	**Ag, g/t**	**% Cu**	**% Zn**	**NaCN**	**Lime**
25	77.37	34.19	6.13	1.91	3.52	41.62	0.084	0.103	2.67	1.65
19	79.90	40.16	6.78	10.13	3.70	42.29	0.071	0.052	2.43	1.58
16	79.46	37.93	5.50	2.14	3.88	40.24	0.079	0.099	2.69	1.68
12.5	80.52	35.69	5.31	2.37	3.98	44.59	0.075	0.096	0.90	3.24
6.3	86.89	37.64	7.22	2.52	3.85	45.70	0.075	0.108	1.08	3.53
3.35	85.58	42.44	7.43	2.68	3.92	42.50	0.077	0.105	0.88	3.72

Table 13-18
Coarse Ore Cyanidation for Low Gossan Ore at Varying Crush Sizes
(Leach Time: 336 hours)

Crush Size, P_{100} mm	Extraction %				Cal. Head				Reagent Consumption, Kg/t	
	Au	Ag	Cu	Zn	Au, g/t	Ag, g/t	%Cu	%Zn	NaCN	Lime
25	84.21	36.86	6.76	2.30	1.01	34.42	0.043	0.064	2.47	0.91
19	85.47	40.85	6.00	2.30	0.92	28.02	0.047	0.069	2.19	1.02
16	85.74	41.21	7.39	3.04	1.16	35.01	0.047	0.067	0.42	2.21
12.5	84.45	43.66	6.68	2.48	1.00	32.40	0.047	0.065	0.55	2.07
6.3	84.75	51.03	5.20	2.24	1.05	29.71	0.046	0.069	0.34	2.35
3.35	81.74	53.26	5.22	2.34	1.07	32.20	0.048	0.068	0.51	2.32

Coarse ore bottle roll tests were performed on the other composites at a crush size of 19mm in duplicate for 42 days (1008 hrs.). The test results, summarized in Table 13-19 indicated gold recovery between 84% to 94% and silver recovery between 30% and 48%.

Table 13-19
Coarse Ore Cyanidations for Variability Samples
(Leach Time: 1008 hours)

Sample	Extraction %				Cal. Head				Reagent Consumption Kg/t	
	Au	Ag	Cu	Zn	Au g/t	Ag g/t	% Cu	% Zn	NaCN	Lime
High Au	89.06	47.94	27.63	7.00	8.65	184.48	0.10	0.16	1.24	4.22
High BM	83.62	30.04	29.92	1.80	1.47	130.69	0.25	0.13	2.87	2.22
Low BM	90.93	33.66	6.49	4.31	1.09	14.75	0.03	0.05	0.14	2.51
Middle	94.51	74.09	27.67	5.40	2.09	70.06	0.09	0.11	4.19	2.44
North	88.42	59.45	18.71	3.67	2.78	89.00	0.05	0.05	3.64	2.04
South	87.15	69.61	36.05	5.95	4.71	215.30	0.09	0.08	4.20	3.11

A total of five column tests were performed at SGS, U.K. The process conditions for these tests are given in Table 13-20. The first four columns were run for 107 days and the fifth column was run for 95 days. The solution at a cyanide concentration of 1.0 g/L NaCN was pumped to the top of the column at 12 l/hr/m2(0.2l/min/m2) The pH of the exiting solution for the columns 1 to 4 were high ranging between 11.5 and 12.8 because of high amounts of cement used for agglomeration. High pH such as this can impair the rate of leaching.

Table 13-20
Test Conditions for Column Tests at SGS, UK

Column No.	Sample	Crush Size P_{100}, mm	Process Parameters
1	Gossan	6.3	Agglomerated with 3kg/t hydrated lime and 30kg/t cement
2	Gossan	19	Agglomerated with 3kg/t hydrated lime and 20kg/t cement
3	Low Gossan	6.3	Same as column 1
4	Low Gossan	19	Same as column 2
6	Blend North, Middle and South Zones	19	1.81kg/t hydrated lime and 7kg/t cement

The leach test results are summarized in Table 13-21. The test results indicated the following:

- The gold extraction ranged from 77.3% to 90.7% and the silver extraction ranged from 38.9% to 63.6%. The gold extraction for columns 4 and 6 at 45 days was 79% and 86.9% respectively.
- The copper extraction was low in all tests except for the blended material where the copper extraction was 25.5%.
- The Merrill Crowe testing of the pregnant solution from one of the columns resulted in 98.6% of gold and 98.7% of silver in the precipitate.
- The NaCN consumption in column testing ranged between 0.55kg/t and 1.84kg/t. The higher consumption was for the blended material.

Table 13-21
Summary of Column Leach Test Results Performed at SGS UK

Column No.	Sample	Crush Size P_{100}, mm	Leach Time, Days	Extraction %				Cal. Head			
				Au	Ag	Cu	Zn	Au g/t	Ag g/t	% Cu	% Zn
1	Gossan	6.3	45	71.4	34.1	-	-	-	-	-	-
		6.3	107	80.5	41.9	5.3	3.3	3.6	43.8	0.09	0.10
2	Gossan	19	45	65.9	28.7	-	-	-	-	-	-
		19	107	79.2	38.9	6.1	2.5	3.7	42.2	0.09	0.10
3	Low Gossan	6.3	45	63.4	33.2	-	-	-	-	-	-
		6.3	107	77.3	48.0	1.4	2.6	0.9	30.3	0.05	0.07
4	Low Gossan	19	45	79.0	36.6	-	-	-	-	-	-
		19	107	85.9	49.4	2.1	3.1	1.0	37.2	0.04	0.07
6	Blend North, Middle and South Zones	19	45	86.9	59.2	-	-	-	-	-	-
		19	95	90.7	63.6	25.5	3.5	2.8	66.2	0.08	0.09

Based on an estimated feed consisting of a blend of 70% of gossan and 30% of low gossan to the heap, the blended gold and silver extractions were estimated to be 81.2% and 42.1% respectively. The NaCN consumption was estimated to be 0.6kg/t. For mine planning purposes and for input into the financial model, a gold recovery function dependent on head grade was used to estimate gold recovery. Details are found in Section 15. The recovery function is similar to the blend and summary results stated above.

13.2 Sulfide Ores

Initially the testwork on the sulfide ores was undertaken by RDi. Limited testwork performed at Hacettepe Mineral Technologies Inc. produced a flowsheet which could produce saleable-grade copper and zinc concentrates. Additional locked-cycle tests at SGS confirmed that the process flowsheet was technically viable.

13.2.1 Feed Preparation and Characterization

There are three ore types constituting the sulfide ore, namely massive pyrite, massive pyrite magnetite and disseminated sulfide. Drill core rejects from about 38 holes were received at RDi to produce the three sulfide composites. The three composites were blended in the following proportion to prepare a master sulfide composite for the metallurgical study as suggested by the client: 33% massive pyrite, 10% magnetite rich pyrite and 57% disseminated sulfide.

All the composites constituting each of the three composites were mixed together, blended thoroughly and split into two parts. One part was saved and the other part was stage crushed to P_{100} of 6 mesh. The crushed samples were thoroughly blended and split into 1-kg and 10-kg charges.

A 1-kg charge of each composite was pulverized to 150 mesh and representative samples taken out for head analyses. The test results, given in Tables 13-22 and 13-23, indicated the following:

- The massive pyrite composite assayed 1.228 g/t Au, 27.7 g/t Ag, 0.1548% Pb, 1.848% Zn, 1.074% Cu, and 38.00% $S_{Sulfide.}$

Table 13-22
Head Analyses of Three Sulfide Composite Samples

Element	Sample		
	Massive Pyrite	Magnetite Rich Pyrite	Disseminated Sulfide
Au, g/t	1.228 (0.898, 1.557)	0.744 (0.823, 0.665)	0.502 (0.508, 0.494)
Ag, g/t	27.7 (27.4, 28.0)	24.4 (24.2, 24.6)	18.7 (18.4, 19.0)
Pb, %	0.1548	0.327	0.1816
Zn, %	1.848	2.240	1.172
Cu, %	1.074	0.648	0.652
S_{Total}, %	39.35	39.97	15.98
$S_{Sulfide}$, %	38.00	38.30	14.75
$S_{Sulfate}$, %	1.35	1.68	1.23
C_{Total},%	0.33	0.61	0.50
$C_{organic}$,%	0.27	0.29	0.07
$C_{inorganic}$,%	0.06	0.32	0.43

Table 13-23
ICP Analyses of Three Sulfide Composite Samples

Element	Massive Pyrite	Magnetite Rich Pyrite	Disseminated Sulfide
Percent			
Al	0.44	0.20	5.03
Ca	0.15	0.97	1.51
Fe	40.88	42.87	17.86
K	0.03	0.01	1.61
Mg	0.64	0.79	2.29
Na	0.03	0.01	0.17
Ti	0.02	0	0.05
ppm			
As	740	419	494
Ba	175	173	393
Bi	141	119	67
Cd	123	123	69
Co	19	17	9
Cr	75	68	59
Cu	12618	7091	6768
Mn	413	697	986
Mo	<1	<1	<1
Ni	2	1	12
Pb	1812	3497	1823
Sr	17	25	51
V	12	8	108
W	<10	<10	<10
Zn	19368	23880	12574

- The magnetite rich pyrite composite assayed 0.744 g/t Au, 24.4 g/t Ag, 0.327% Pb, 2.24% Zn, 0.648% Cu, and 38.30% $S_{Sulfide}$.

- The disseminated sulfide composite assayed 0.502 g/t Au, 18.7 g/t Ag, 0.1816% Pb, 1.172% Zn, 0.652% Cu, and 14.75% $S_{Sulfide}$.

Following the preparation of the master sulfide composite, it was also submitted for head analyses. The results, given in Tables 13-24 to 13-26, indicate the following:

- The sulfide master composite assayed 0.573 g/t Au, 23.8 g/t Ag, 0.185% Pb, 1.358% Zn, .854% Cu (ICP) and 25.18% $S_{Sulfide}$.

- The composite sample contained 549ppm As which could cause problems if it concentrated in copper or zinc concentrates.

- The metallic assays indicated that portion of both gold and silver may be present as coarse particles in the composite sample.

Table 13-24
Head Analyses of Sulfide Master Composite

Element	**Assay**
	Sulfide Composite
Au, g/t	0.573
Ag, g/t	23.8
Pb, %	0.185
Zn, %	1.358
S_{Total}, %	26.53
$S_{Sulfide}$, %	25.18
$S_{Sulfate}$, %	1.35
C_{Total},%	0.45
$C_{organic}$,%	0.12
$C_{inorganic}$,%	0.33

Table 13-25
ICP Analyses of Sulfide Master Composite

Element	Sulfide Composite
Percent	
Al	3.07
Ca	0.94
Fe	28.29
K	0.94
Mg	1.63
Na	0.15
Ti	0.04
ppm	
As	549
Ba	179
Bi	94
Cd	89
Co	12
Cr	29
Cu	8540
Mn	815
Mo	<1
Ni	<5
Pb	1936
Sr	24
V	68
W	<10
Zn	15391

Table 13-26
Metallic Assays for the Sulfide Master Composite

Sample	Minus 150 Mesh		Plus 150 Mesh		Cal Feed Grade, g/t
	Weight, gms	Assays g/t	Weight, gms	Assays g/t	
GOLD VALUES					
Sulfide	940.5	0.871, 0.648, 0.700, 0.724	55.24	8.275	1.154
SILVER VALUES					
Sulfide	940.5	18.522, 15.572, 12.3432, 13.062	55.24	57.7324	17.2523

13.2.2 Mineralogical Evaluation

The three composite samples, namely massive sulfide, magnetite rich pyrite and disseminated sulfide composites were mineralogically evaluated to determine the bulk mineralogy with an emphasis on gold and silver mineralogy. The highlights of the study indicated the following:

- The sulfide mineralogy is dominated by pyrite which occurs in the range of 1 to 150 microns. Large grains commonly carry inclusions of magnetite and other sulfides.

- Chalcopyrite is seen as liberated fragments but it more commonly occurs as aggregates with pyrite and sphalerite. Grain size is very fine (2 to 50 microns). Chalcopyrite commonly shows mild to strong alteration to covellite and chalcocite.

- A few free galena particles were observed but majority of galena is seen as small inclusions in sphalerite and most commonly in pyrite.

- No free silver or gold particles were seen in the sulfide composites.

13.2.3 Comminution Studies

The comminution studies were undertaken on the master sulfide composite and individual sulfide composites. The tests included abrasion index, crushability work index, Bond’s ball mill work indices and SMC testing.

The test results indicated the following:

- The abrasion index for the master sulfide composite was 0.1166 which indicated that the ore is abrasive.

- The crushability work index for massive pyrite was 3.2 Kwh/mt and for disseminated sulfide was 18.8 Kwh/mt.

- The Bond’s ball mill work index at 100 mesh grind size were 5.65, 9.79 and 6.29 kwh/mt for massive pyrite-magnetite, disseminated sulfide and massive pyrite, respectively.

13.2.4 Rougher Flotation Tests

Flotation testing was initiated with the objective of producing two concentrates, namely copper concentrate containing gold and silver and zinc concentrate.

The first series of tests were performed using a standard suite of reagents to first depress zinc minerals and float copper minerals. The test results, given in Table 13-27, indicated that zinc minerals did not get depressed and floated along with copper and pyrite. Since pyrite is the major impurity in the ore and it did not get rejected, the weight recovery was significant in these tests.

Table 13-27
Different Flotation Process Results (Float Cu/Pb, Depress Zn)

Test No.	Primary Grind P_{80}, mesh	Cu/Pb Collector	Pb/Cu Rougher Conc. Recovery %						Zn Rougher Conc.Recovery %					
			Wt.	Au	Ag	Pb	Cu	Zn	Wt.	Au	Ag	Pb	Cu	Zn
1	65	SIPX	54.2	76.2	81.9	78.4	79.4	86.2	6.9	13.0	14.8	11.5	15.4	8.7
2	100	SIPX	58.9	92.0	90.7	86.1	92.1	93.7	2.4	3.6	5.4	4.5	4.4	2.0
3	150	SIPX	58.2	92.9	86.8	86.4	93.0	93.7	1.9	3.2	7.3	3.7	3.3	1.6
4	200	SIPX	56.1	88.4	87.4	85.2	91.3	90.4	3.2	3.9	7.0	3.5	3.9	4.0
5	150	3418A	28.0	57.8	60.9	61.3	75.6	80.3	31.6	32.9	29.0	27.8	20.1	14.6
6	150	AP242	27.0	60.7	66.8	67.5	87.9	86.4	32.9	33.5	24.3	21.6	9.2	8.2
7	150	AP238	40.3	69.7	75.0	76.4	88.1	85.7	19.4	23.8	15.0	13.1	7.7	8.8
8	150	3418A	24.1	55.4	60.8	63.6	75.8	86.6	35.2	38.6	28.9	24.4	19.6	8.0
25	200	AP7279	40.6	76.0	66.2	-	82.9	78.9	17.9	20.3	19.7	-	11.3	15.1

Note: Flotation Time: 5 mins for Pb/C
: 5 mins for Zn

The second series of tests were performed with the primary objective of floating Cu, Zn, Pb, Au and Ag minerals while rejecting a majority of the pyrite in the rougher flotation step. Based on the mineralogy of the samples, a finer primary grind was needed. Hence, a finer grind and pH of 12 was evaluated in this series of tests. The test results, given in Table 13-28, indicated that a simple reagent suite consisting of collectors AP3894 and AP404 and frother MIBC recovered ± 16% of weight, 88.6% of copper, 90.6% of zinc, 57.7% of gold and 72.2% of silver in six minutes of flotation time. The primary grind was P_{80} of 325 mesh and flotation pulp pH was 12.

Table 13-28
Rougher Bulk Flotation Test Results

Test No.	Primary Grind P_{80}, mesh	Flotation Time, min	Collectors	pH	Rougher Concentrate Recovery %						Grade				
					Wt.	Au	Ag	Pb	Cu	Zn	Au g/t	Ag g/t	% Pb	% Cu	% Zn
9	150	9	PAX/ AP404	7.4	57.2	93.6	73.3	82.8	88.2	90.8	0.90	32.8	0.275	1.356	2.2
12	150	8	AP3894	12.0	18.0	55.3	55.8	68.1	82.0	88.8	2.14	83.0	0.712	3.966	7.183
13	325	8	AP3894	12.0	15.1	78.9	57.4	72.3	87.5	90.4	7.98	116.2	0.89	4.93	8.220
14	200	8	AP3418A	12.0	21.8	65.0	54.2	76.1	87.9	90.8	2.48	80.1	0.696	3.513	6.013
15	200	8	AP3894	12.1	18.9	52.9	65.0	71.9	85.8	89.9	1.88	92.5	0.755	3.89	6.67
16	100	10	AP3894/AP404	12.0	27.6	71.8	71.2	-	88.7	91.7	1.88	53.3	-	2.82	4.92
17	200	10	AP3894/AP404	12.0	25.2	50.2	74.9	-	91.9	93.5	1.85	67.5	-	3.29	5.56
18	325	10	AP3894/AP404	12.0	21.4	68.2	76.5	-	92.4	93.9	2.10	91.0	-	3.69	6.23
19	200	10	AP3894	12.0	24.4	69.0	66.6	-	91.0	93.4	2.14	62.9	-	3.28	5.67
20	325	6	AP3894/AP404	12.0	16.1	57.7	72.2	-	88.6	90.6	1.92	111.2	-	4.98	8.35
26	325	10	AP3418A/AP404	12.0	22.8	90.0	76.9	-	91.7	92.1	1.53	90.4	-	3.63	6.12

13.2.5 Bulk Cleaner Flotation Tests

The bulk rougher concentrate was generated using optimum process parameters in a one-cubic-foot flotation machine for cleaner flotation tests using 10-kg charges. The rougher concentrate assayed 4.95% Cu, 8.35% Zn, 1.92 g/t Au, 111.2 g/t Ag and 38.65% S_{total}.

Two first-cleaner flotation tests were performed with/without regrind to generate Kinetic data in order to determine process parameters for the first-cleaner flotation. The test results, summarized in Table 13-29, indicated that regrind was beneficial for obtaining good concentrate grade and four minutes of flotation time was sufficient to float a majority of the zinc and an acceptable amount of the copper.

Using the process parameters determined for first-cleaner flotation, flotation kinetics for second-stage cleaner were determined. The test data, given in Table 13-30, indicated that a bulk concentrate assaying 8.80% Cu, 25.27% Zn, 4.68 g/t Au, and 181 g/t Ag can be produced in the open-circuit test.

Table 13-29
First Bulk Cleaner Flotation Kinetics with/without Regrind

Product	Cumulative Flotation Time, min	Cumulative Assay					Cumulative Recovery %					
		Au g/t	Ag g/t	% Pb	% Cu	% Zn	Wt.	Au	Ag	Pb	Cu	Zn
Test No. 21 No Regrind												
Conc. 1	1	4.66	125.5	1.39	6.46	28.8	19.1	37.9	26.7	34.5	25.6	65.4
Conc. 2	2	4.50	164.3	1.40	7.09	25.5	27.4	52.5	41.2	50.0	40.4	83.1
Conc. 3	3	3.69	147.9	1.13	6.89	16.65	46.5	73.1	63.0	68.6	66.6	92.1
Conc. 4	4	3.15	139.5	1.05	6.56	13.97	57.3	76.9	73.4	78.0	78.2	95.2
Tail	-	1.27	67.9	0.40	2.46	0.94	42.7	23.1	26.6	22.0	21.8	4.8
Cal Feed	-	2.35	109.1	0.77	4.81	8.40	100.0	100.0	100.0	100.0	100.0	100.0
Test No. 22, 10 Min Regrind												
Conc. 1	1	3.61	163.7	1.35	6.73	35.70	14.3	18.3	21.7	24.9	20.0	63.6
Conc. 2	2	4.01	182.0	1.46	7.92	30.85	20.9	29.9	35.3	39.5	34.5	80.5
Conc. 3	3	4.28	190.5	1.46	9.02	22.86	31.7	48.4	56.0	60.2	59.7	90.6
Conc. 4	4	4.22	186.4	1.42	9.03	20.10	37.2	56.0	64.4	68.6	70.1	93.5
Tail	-	1.97	61.2	0.39	2.28	0.83	62.8	44.0	35.6	31.4	29.9	6.5
Cal Feed	-	2.81	107.8	0.77	4.79	8.01	100.0	100.0	100.0	100.0	100.0	100.0

Table 13-30
Bulk Cleaner 2 Flotation Kinetic with/without Regrind

Product	Flotation Time, min	Assay					Recovery %					
		Au g/t	Ag g/t	% Pb	% Cu	% Zn	Wt.	Au	Ag	Pb	Cu	Zn
Test No. 23 No Regrind												
Cl 2 Conc. 1	0.5	4.18	135.7	1.61	4.09	38.20	8.6	14.1	10.7	17.7	7.4	39.6
Cl 2 Conc. 2	0.5	4.95	164.2	1.58	6.56	28.30	6.8	13.1	10.2	13.6	9.3	23.0
Cl 2 Conc. 3	1	4.40	206.6	1.60	8.35	21.30	5.9	10.1	11.1	12.0	10.3	15.0
Cl 2 Conc. 4	1	4.73	199.7	1.51	10.34	10.96	2.7	5.0	4.9	5.2	5.8	3.6
Comb. Cl 2 Conc.	3	4.51	168.3	1.59	6.53	28.20	24.0	42.3	36.8	48.5	32.8	81.2
Cl 2 Tail		3.27	123.0	0.89	6.19	3.56	18.1	23.1	20.3	20.4	23.5	7.7
Cal. Cl 1 Conc.	4	3.95	148.8	1.29	6.39	17.60	42.1	65.3	57.1	68.9	56.2	88.9
Cl. 1 Tail		1.54	81.4	0.42	3.62	1.59	57.9	34.7	42.9	31.1	43.8	11.1
Cal Ro. Feed	-	2.56	109.8	0.79	4.79	8.33	100.0	100.0	100.0	100.0	100.0	100.0
Test No. 24, 10 Min Regrind												
Cl 2 Conc. 1	0.5	4.78	161.8	1.21	7.18	33.60	14.2	26.0	19.2	21.9	22.2	61.6
Cl 2 Conc. 2	0.5	4.43	201.2	1.43	10.16	23.70	6.0	10.2	10.1	10.9	13.3	18.4
Cl 2 Conc. 3	1	4.94	212.2	1.58	11.52	11.32	4.3	8.1	7.6	8.6	10.7	6.2
Cl 2 Conc. 4	1	4.24	188.2	1.36	10.14	5.12	2.5	4.0	3.9	4.3	5.4	1.6
Comb. Cl 2 Conc.	3	4.68	181.0	1.33	8.80	25.27	26.9	48.4	40.8	45.7	51.5	87.8
Cl 2 Tail		3.20	158.0	0.97	5.71	3.62	5.2	6.4	6.9	6.5	6.5	2.4
Cal. Cl 1 Conc.	4	4.44	177.2	1.27	8.30	21.75	32.1	54.8	47.7	52.1	58.0	90.2
Cl. 1 Tail		1.74	91.9	0.55	2.84	1.11	67.9	45.2	52.3	47.9	42.0	9.8
Cal Ro. Feed	-	2.60	119.3	0.78	4.59	7.74	100.0	100.0	100.0	100.0	100.0	100.0

13.2.6 Copper-Zinc Separation Test

The two approaches to the separation of copper and zinc minerals are:

- Depress zinc minerals and float copper.
- Depress copper minerals and float zinc.

Both of these approaches were evaluated. The depression of copper minerals using NaHS and floating zinc minerals was found to be the most successful approach. In addition, this approach was tested on rougher concentrate, bulk first-cleaner flotation and bulk second-cleaner flotation. The greatest success was obtained with the second-cleaner flotation concentrate. The results are summarized in Table 13-31.

Table 13-31
Zinc and Copper Separation Using NaHS to Depress Copper

Product	Recovery %					Grade			
	Wt.	Zn	Cu	Au	Ag	% Zn	% Cu	Au g/t	Ag g/t
Test No. 28									
Zn Conc.	37.0	73.3	20.4	15.9	27.0	45.22	4.35	1.97	119.0
Cu Conc.	63.0	26.7	79.6	84.1	73.0	9.65	9.98	6.09	188.2
Bulk Conc.	100.0	100.0	100.0	100.0	100.0	22.80	7.90	4.57	162.6
Test No. 31									
Zn Conc.	41.4	78.7	23.9	20.5	30.4	42.83	4.42	2.32	120.9
Cu Conc.	58.6	21.3	76.1	79.5	69.6	8.17	9.91	6.37	195.4
Bulk Conc.	100.0	100.0	100.0	100.0	100.0	22.52	7.64	4.70	164.6

The test results indicate that marketable-grade zinc concentrate, assaying ± 45% Zn and ± 4.4% Cu, was produced in the separation process. The tailings containing ± 80% of copper, ± 84% of gold and ± 73% of silver will go to the copper upgrading circuit. It will be combined with the bulk cleaner 1 tailing and reground and upgraded in the copper flotation circuit. The copper concentrate was low in copper and could not be marketed.

13.2.7 Optimization of Cu-Zn Sulfide Flotation Test Conditions at Hacettepe Mineral Technologies Inc., (HMT) Ankara

Based on the review of RDi testwork, HMT reevaluated the sequential copper and zinc flotation process using alternative reagents. The initial scoping testwork was performed using the same master composite prepared by RDi. The optimum flotation parameters, given in Table 13-32, included the addition of meta bi-sulfite, zinc sulfate, sodium sulfide and sodium silicate in the grinding mill. The pulp pH was 6.5 to 7.0. Aeration was required to increase the pulp redox potential in order to enhance collector adsorption. The pulp pH was maintained at pH 6.5 to 7 during all stages of copper flotation. A mixture of sodium Aerofloat (alkyl dithiophosphate) and Aero 8761 (monothiophosphate) was used as a collector for copper. Following copper rougher flotation, the pH was adjusted to 11.5 with lime and sphalerite activated with copper sulfate. Aerofloat 7279 (thinocarbanate) was used as collector to float zinc. The metallurgical results are summarized in Table 13-33. The recoveries of copper and zinc in copper rougher concentrate were 70% and 19% respectively. After regrinding and four stages of cleaner flotation in open-circuit, a copper concentrate, assaying 30% Cu and 7.36% Zn, was produced with 20.22% copper recovery and 2.75% zinc recovery. The zinc circuit produced a concentrate assaying 53.32% Zn and 4.91% Cu and recovered 56.67% of zinc.

Table 13-32
Summary of the Optimum Flotation Conditions

	MBS	ZnSO4	Na2S	Na-Silicate	CuSO4	Collector	pH
Grinding	3 kg/t	500 g/t	250 g/t	500 g/t			7.48
Aeration							7.08
Cu Conditioner			.			50 g/t NaAF 50 g/t A8761 20 g/t MIBC	
Cu Regrind	3 kg/t	1.5 kg/t	350 g/t	350 g/t			6.8
Cu Cl1						40 g/t NaAF 40 g/t 8761	
Cu Cl Scav.						40 g/t NaAF 40 g/t 8761	7
Zn Cond. 1					300 g/t		11.5 (lime)
Zn Cond. 2						20 g/t A7279	
Zn Regrind							11.5 (lime)
Zn Cl Cond.					300 g/t	30 g/t A7279	11.5

Table 13-33
Summary of Metallurgical Results of Sequential Cu-Zn Flotation

	Wt (%)	Grade %		Recovery %	
		Cu	Zn	Cu	Zn
Cu Cl Conc.	0.51	29.89	7.36	20.22	2.75
Cu Rougher Conc.	13.19	4.05	1.95	70.76	18.81
Zn Cl Conc.	1.45	2.55	53.32	4.91	56.67
Zn Rougher Conc.	8.5	1.03	10.82	11.57	67.42
Feed	100.0	0.75	1.36	100.0	100.0

Additional open-cycle tests were performed to optimize copper and zinc recoveries. These tests were performed with new master sulfide composite samples. The head analyses of these composites are given in Table 13-34.

The mineralogical analyses indicated that chalcopyrite was the main copper mineral followed by enargite and covellite. Sphalerite was the main zinc mineral. Adequate degree of liberation (70%) for separation of the two minerals (Cu and Zn) could be achieved at particle size finer than 38 micrometers.

Table 13-34
Head Analyses of the Master Sulfide Composites

Element	Sample			
	Master Sulfide Composite (April 2015)	Master Sulfide Composite (July 2015)		Master Sulfide Composite (No enriched ore)
		HMT	SGS	
% Cu	0.9	0.61	0.75	0.82
%Fe	27.2	25.81	28.5	25.47
%Pb	0.22	0.23	0.27	0.27
%Zn	2.02	1.41	1.91	1.56
%S	-	-	29.6	-
Au, g/t	-	-	0.64	-
Ag, g/t	-	-	25.3	-

Open-cycle tests were performed to select and optimize the reagents for the process conditions. These variables included collector types, primary grind, pulp density and depressants. When the optimum parameters were employed for the locked-cycle test, the results indicated that recycling cleaner tailings negatively affected the grade of the concentrate. Therefore, eight locked-cycle tests were performed to refine the process parameters. The refinement included addition of a pre-flotation step to recover talc which was diluting the copper concentrate.

The process flowsheet which produced both copper and zinc marketable-grade concentrates with reasonable recoveries is given in Figure 13-2. The process parameters are given in Table 13-35 and the results of the average of last three cycles for the LCT-8 are given in Table 13-36. The test results indicate the following:

- The copper concentrate assaying 25.03% Cu and 1.45% Zn, recovered 58.3% of the copper.

- The zinc concentrate assaying 54.2% Zn, 2.95% Cu and 2.07% Pb, recovered 84.9% of zinc.

- All the products were not analyzed for gold and silver. However, since gold is tied up with pyrite and pyrite is rejected in both copper and zinc concentrates, the recoveries of precious metals are expected to be low.

- The solids were recycled in the test. However, process water was not recycled in the test.

Table 13-35
Flotation Process Conditions for Locked -Cycle Test 8

	MBS	ZnSO4	Na2S	Na-Silicate	NaCN	CuSO4	Collector & Frother	pH
Grinding	3 kg/t	500 g/t	500 g/t	500 g/t				7.08
Pre-Flotation							9 g/t MIBC	
Cu- Flotation			.				50 g/t NaAF 70 g/t A8761 12 g/t MIBC	6.88
Cu Regrind	2 kg/t	2 kg/t	1 kg/t		250 g/t			6.8
Cu Cl1				200 g/t			40 g/t NaAF 40 g/t 8761	
Cu Cl Scav.							50 g/t NaAF 50 g/t A8761	
Zn Cond. 1 & 2						500 g/t	30 g/t + 20g/t A7279	11.5 (lime)
Zn Regrind				320 g/t				11.5 (lime)
Zn Cl Cond.						400 g/t	40g/t A7279	11.5
Zn Cl Scav.							25g/t A7279	
Py Flotation							50g/t KAX	

Table 13-36
Summary of Locked-Cycle (LCT-8) Test Results

Product	Assays%				Distribution %				
	Cu	Zn	Pb	Fe	Wt.	Cu	Zn	Pb	Fe
Cu Cleaner 4 Conc.	25.03	1.45	0.90	17.9	1.3	58.3	1.1	6.9	0.9
Zn Cleaner 4 Conc.	2.95	54.2	2.07	3.7	2.7	13.8	84.9	31.8	0.4
Pre-Float Conc.	0.24	0.54	0.09	6.4	0.6	0.2	0.2	0.3	0.2
Zn Cleaner Scav. Tails	0.75	2.39	0.85	32.4	3.9	5.1	5.4	19.0	5.0
Rougher Tails	0.14	0.16	0.08	25.9	91.4	22.5	8.4	42.0	93.5
Combined Tails	0.17	0.25	0.11	26.2	95.4	37.6	13.8	61.0	98.5
Cal Feed	0.58	1.72	0.18	25.3	100.0	100.0	100.0	100.0	100.0

13.2.8 Locked-Cycle Testing of Process Flowsheet at SGS, UK

A confirmatory locked-cycle test with the master composite was performed at SGS, UK facilities to confirm that the process flowsheet and process parameters developed at HMT, Turkey would consistently produce marketable-grade concentrates.

The test flowsheet was the same as given in Figure 13-2 for the locked-cycle test and the reagent additions were similar to those reported in Table 13-33. The test results averaged for the last three cycles are presented in Table 13-37.

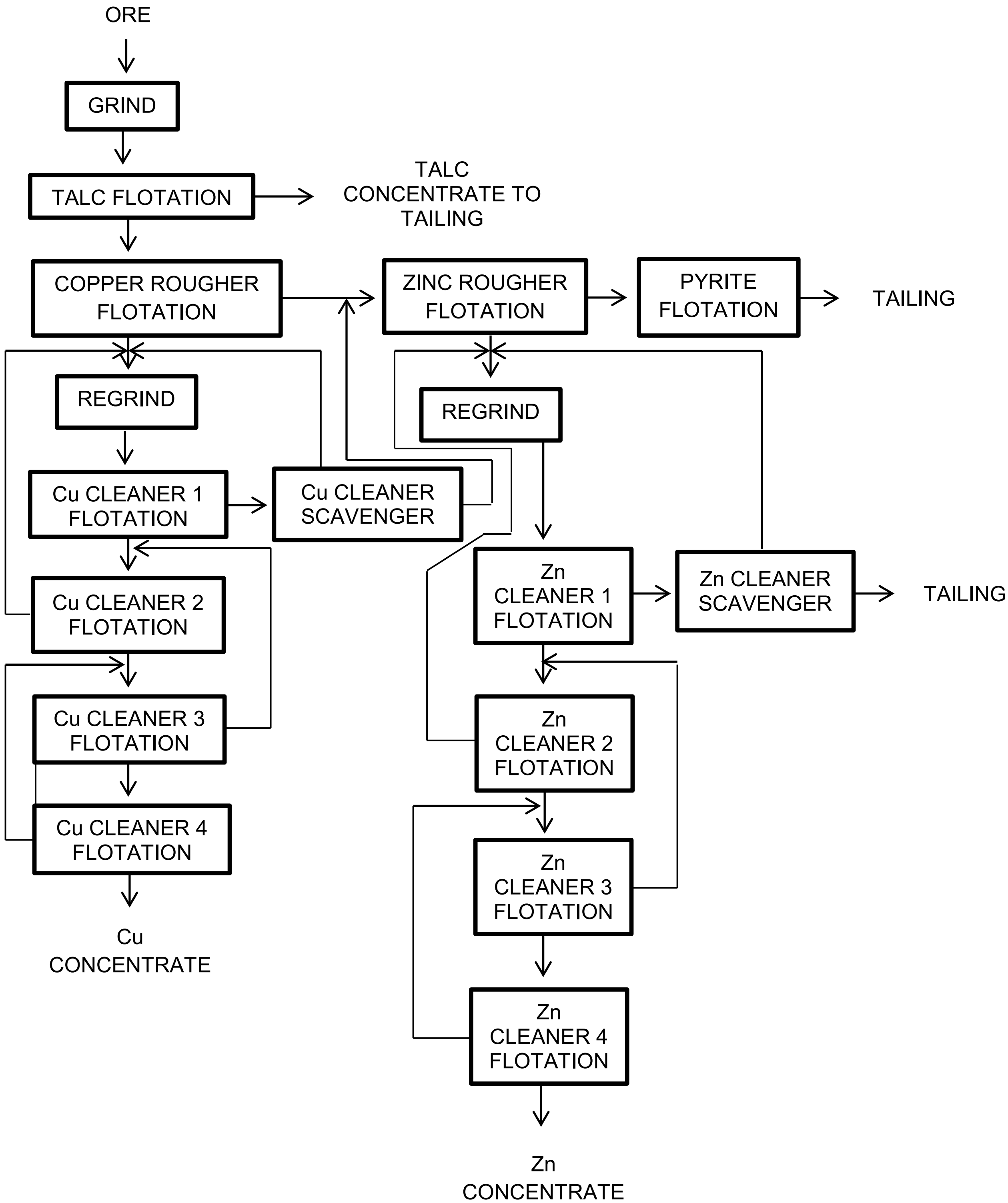


FIGURE 13-2
LOCKED-CYCLE TEST FLOWSHEET

The test results indicate the following:

- The copper concentrate assaying 30.95% Cu, 2.49% Zn, 4.34 g/t Au and 108.5 g/t Ag recovered 69.2% of copper, 17.2% of gold and 12.3% of silver in the LCT. The zinc concentrate assaying 54.3% Zn, 2.71% Cu, 3 g/t Au and 143.5 g/t Ag recovered 81.5% of zinc, 15.7% of gold and 21.5% of silver.

Table 13-37
Summary of Locked-Cycle Test Results without Recycling of Process Water at SGS, UK with Master Composite Sample (LCT)

Product	Assay							Distribution %							
	% Cu	% Zn	% Pb	% Fe	% S	Au g/t	Ag g/t	Wt.	Cu	Zn	Pb	Fe	S	Au	Ag
Cu Cleaner 4 Conc.	30.95	2.49	1.30	24.79	31.55	4.34	108.5	2.0	69.2	2.8	11.0	1.7	2.2	17.2	12.3
Zn Cleaner 4 Conc.	2.71	54.3	2.69	5.94	34.26	3.00	143.5	2.6	8.1	81.5	30.3	0.5	3.2	15.7	21.5
Pre-Float Conc.	1.09	1.61	0.30	16.59	15.11	0.51	26.0	0.7	0.8	0.6	0.9	0.4	0.4	0.7	1.0
Zn Cleaner Scav. Tail	0.47	0.94	0.34	41.63	38.34	0.82	30.0	12.6	6.6	6.8	18.1	18.0	17.1	20.5	21.5
Rougher Tail	0.16	0.18	0.11	28.31	26.21	0.28	9.3	82.1	15.2	8.4	39.8	79.4	76.9	45.9	43.7
Combined Tail	0.20	0.28	0.14	30.08	27.83	0.35	12.1	94.7	21.9	15.1	57.8	97.4	94.2	66.4	65.7
Cal Feed	0.89	1.76	0.23	29.25	27.99	0.50	17.6	4.59	100.0	100.0	100.0	100.0	100.0	100.0	100.0

These results confirm that marketable-grade copper and zinc concentrates can be produced by employing the process parameters and flow sheet developed for the project. The recoveries obtained in the LCT locked-cycle test will be used to determine the project economics for the PFS. Process water was not recirculated in this test. A water treatment plant has been added to the project for the PFS to treat recycled process water. The additional on-going test work is being directed to determine the effect of recycling process water on copper and zinc metallurgy. This test work along with geo-metallurgy testing will be completed for the Feasibility Study.

14.0 MINERAL RESOURCE

The mineral resource at Gediktepe was developed using a computer based block model of the deposit. The block model was assembled based on the drill hole data base and interpreted geology from Polimetal after review and verification of that information by IMC. Mineral resources were estimated using the block model and the floating cone open pit software to establish the component of the deposit with reasonable prospects of economic extraction. John Marek, of IMC acted as the qualified person for the development of the block model and the estimation of mineral resources.

Some features of the 2015 model are:

1) The block size is 10m x 10m in plan with a 2.5m high block. Open pit mining will be planned on 5m benches with the option of splitting the bench into two 2.5 meter benches with a backhoe to follow high grade zones with reduced dilution.

2) Gediktepe personnel supplied IMC with detailed wire frames of the project lithology and main ore bearing units which were incorporated into the model and the data base. Geology was assigned to the model on a whole block basis.

3) An oxide – sulfide distinction in the assay data base was defined by Gediktepe geologists. An oxide – sulfide surface was interpreted by IMC and assigned to the model on a whole block basis.

4) There is a substantial change of grade populations at the oxide – sulfide contact. The contact is a hard boundary during grade estimation, even in gold and silver.

The final statement of mineral resources is presented at the end of this section and reflects material that is inside of a computer generated pit (floating cone). The purpose of using a floating cone is to provide some assurance that the mineral resource has "reasonable prospects of economic extraction" as required by CIMM best practices. The economic assumptions that were used for that pit are also summarized.

14.1 Model Location

The Gediktepe block model is rotated 45 degrees to align with the strike of the orebody and the orientation of the drilling. The following Table 14-1 and Figure 14-1 summarize the model location.

Table 14-1
Gediktepe Model Area – Block Corners

Gediktepe August 2015 Model Area - Block Corners				
	South	West	North	East
Easting	637,050.13	636,095.54	637,509.75	638,464.34
Northing	4,357,246.50	4,358,201.09	4,359,615.31	4,358,660.71
Elevation Range		1,015.00	1,487.50	
Model Rotation, Primary Axis =		45	degrees	
Model		200	Blocks in 45 Bearing	
Size		135	Block in 135 Bearing	
Block Size 10m x 10 m 2.5m high		234	Levels	

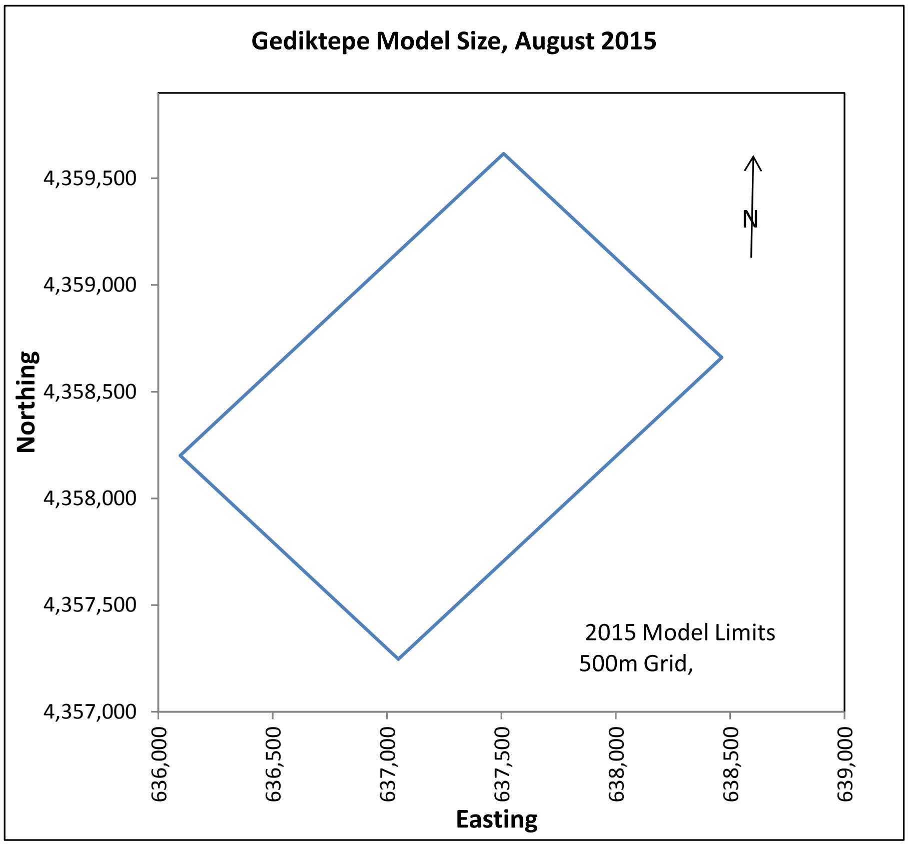


Figure 14-1
Gediktepe IMC Block Model Location

14.2 Data Base

The drill hole data was provided to IMC by Polimetal on 5 August 2015 and reflects the available data on that date.

Both diamond drilling and reverse circulation drilling (RC) have been utilized for the assembly of the model. IMC has compared the results of the two drill types and has concluded that the two sources can be commingled (Section 12).

Table 14-2 summarizes the number of assays from each drill method, and Table 14-3 summarizes the basic statistics of the drill hole data base.

Table 14-2
Number of Assays Used in the Block Model

Number of Assays Used in the Block Model, August 2015 Data Transfer			
Economic Elements	Diamond Drilling 303 holes, 44,704 m	Reverse Circulation Drilling 184 holes, 12,832 meters	Total Drilling 487 holes, 57,536 m
Gold	24,282	8,012	32,294
Silver	24,282	8,012	32,294
Copper	24,282	8,012	32,294
Zinc	24,282	7,796	32,078
Lead	24,282	8,012	32,294
Mercury	2,091	357	2,448
Arsenic	24,282	8,013	32,295

Table 14-3
Basic Statistics of Assay Data

Basic Statistics of August 2015 Assay Data				
Economic Element	Number of Assays	Mean Grade	Standard Deviation	Maximum Value
Gold ppm	32,294	0.241	1.443	149.95
Silver ppm	32,294	8.694	53.755	5,689
Copper %	32,294	0.20	0.60	13.42
Zinc %	32,078	0.34	1.21	32.20
Lead %	32,294	0.08	0.37	10.75
Mercury ppm	2,408	2.515	16.733	780
Arsenic ppm	32,295	189.03	600.54	10,000

There are equal numbers of assays for gold, silver, and copper at Gediktepe because all drill intervals have been assayed. Only a few zinc assays are missing from the total. This simplifies a number of later tasks regarding grade estimation and the determination of confidence classification. Mercury was assayed in the oxide and sulfide ore zones, but was not consistently assayed in the low grade areas of the deposit.

Figure 14-2 illustrates the drill hole locations and the drill hole types. It also shows the location of the cross section that is used to illustrate the geologic modeling of the deposit (Figure 14-3).

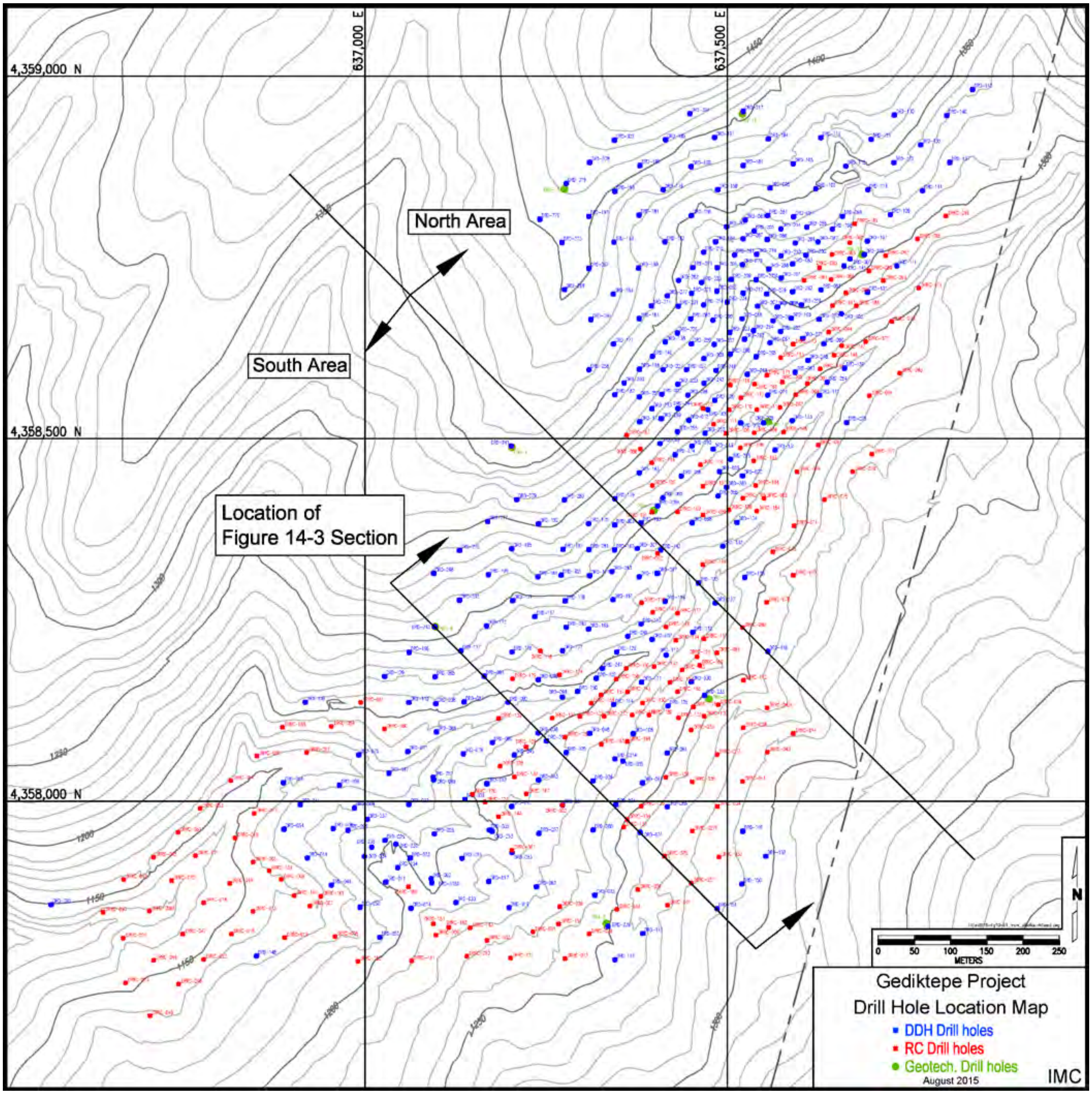


Figure 14-2
Drill Hole and Location Map
Blue = DDH Holes, Red = RC Holes

14.3 Block Model Assembly Procedures

Geology and Data Populations

IMC was provided with the geologic logging in the assay data base as well as interpreted wire frames that were completed by the Polimetal geology staff. IMC reviewed the geologic interpretations and found them to be a reliable estimate of the logged geology. Further statistical work indicated that the logged and interpreted geology also defined the grade domains within the deposit.

The geologic interpretation is a combination of the protolith rock type and the ore types. Figure 14-3 is a cross section through the Gediktepe deposit at the location shown on Figure 14-2 (Section 500 NE). That section illustrates the geologic features that were assigned to the model.

Rock type and ore type codes were assigned to the model variable “geol_0815”:

Ore Types		
Gossan	= Code 10	
Low Grade Gossan	= Code 101	Interpreted by IMC
Massive Pyrite	= Code 20	
Massive Pyrite Magnetite	= Code 21	
Enriched	= Code 50	
Transition Sulfide	= Code 60	
Protolith Rock Types		
Quartz Feldspar Schist	= Code 32	
Chlorite-Sericite Schist	= Code 33	
Quartz Schist =	= Code 31	

The IMC review of the Gossan wire frame indicated that it contained both high grade and low grade oxidized mineralization. There was often a distinct grade boundary between that high grade and low grade within each drill hole.

IMC scanned all of the drill holes in the oxidized zone and selected a grade boundary based on a gold grade of roughly 0.10 gm/t or when there was a substantial increase in grade between assay intervals as one scanned down the drill hole. Those drill hole picks were used to generate a surface that was the top of high grade gossan mineralization. That interpreted surface was used to assign a code to the model to break the gossan into two components: Low Grade Gossan (Code 101) and High Grade Gossan (Code10).

An oxidation surface was assigned to the model by IMC. The logged contact between oxide and sulfide was extracted from the drill hole data base. Those points were used to interpret a surface and assign the code to the model on a nearest whole block basis.

Oxidation coding was stored in the variable “oxide”:

Oxide	= Code 1
Sulfide	= Code 2

The oxide code was modified to respect the wire frame interpreted ore types. For example, if a block was coded as oxide from the process defined above, and the ore type was 20, 21, 50, or 60, the block was recoded as “sulfide”. The reverse process was also applied so that sulfide blocks coded as Gossan by the wire frames were corrected and assigned as “oxide”.

Figure 14-3 illustrates the geometry of the deposit and mineralization. The Low Grade Gossan (101) is not shown on this particular section.

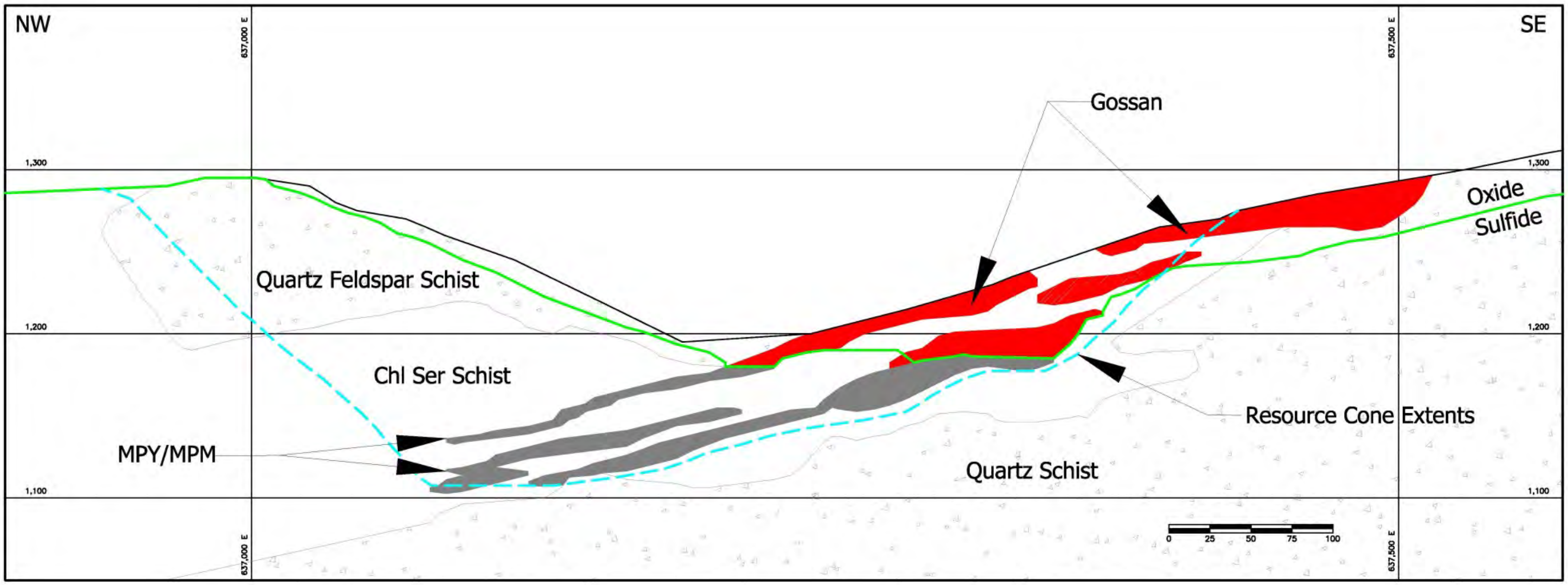


Figure 14-3
Cross Section 500 NE, Looking Northeast
Illustrating IMC Interpretation

A full suite of cross sections every 50m were plotted by IMC throughout the ore body to compare the wire frame interpretation with the logged rock type and grade. Review of these sections prompted the addition of the Low Grade Gossan zone to the block model.

Within the sulfide zone, the copper and particularly zinc assays indicated that the high grades generally occur in the MPY or MPM units. Boundary analysis of the contained assays and composites indicate that these two units can be combined into one population for estimation.

The protolith or country rock contains minor mineralization. The Chlorite-Sericite Schist (CSS-33), Quartz Feldspar Schist (QF-32), and the Quartz Schist (QSCH- 31) were combined for grade estimation, separated by the oxidation surface. The country rocks above the sulfide bound are labelled as 'Oxides – Not Gossan'. The three combined protolith units in the sulfides are titled 'Not High Grade Sulfides'.

The grade population boundaries were established by the wire frames, and the additional surfaces discussed on the previous page. The seven populations' boundaries that were used for grade estimation are summarized below. They are stored in a variable called "geol_0815".

Grade Populations:	Lith Code: "geol_0815"
Gossan High Grade	10
Gossan Low Grade	101
NOT Gossan, in Oxide	31,32,33 and oxide = 1
MPY/MPM	20 and 21
Enriched Zone	50
Transition Zone	60
NOT High Grade Sulfide	31,32,33 and oxide = 2 = sulfide

The boundary analysis was completed using capped assays as described later in text. During boundary analysis, assays are treated as paired data across the domain boundaries being tested. Statistical hypothesis tests were applied to confirm grade change at hard boundaries. The null hypothesis that the average difference between the assays equals zero was rejected with a 95% confidence. The null hypothesis was rejected regardless of the domains being compared leading to the decision to make the domain boundaries hard search boundaries. All seven population boundaries were treated as "hard" boundaries during grade estimation by ordinary linear kriging. The same seven boundaries were applied to the four economic metals of: gold, silver, copper, and zinc. The same populations were used to estimate the grades for the ancillary metals of lead, mercury, and arsenic. Examples of the grade breaks are shown graphically in contact plots in Figures 14-4 through 14-6. These graphs show: 1) the average grade of the assays inside and outside of the high grade gossan, and 2) the average grade of the assays inside and outside of the MPM/MPY boundaries at increasing separation distances.

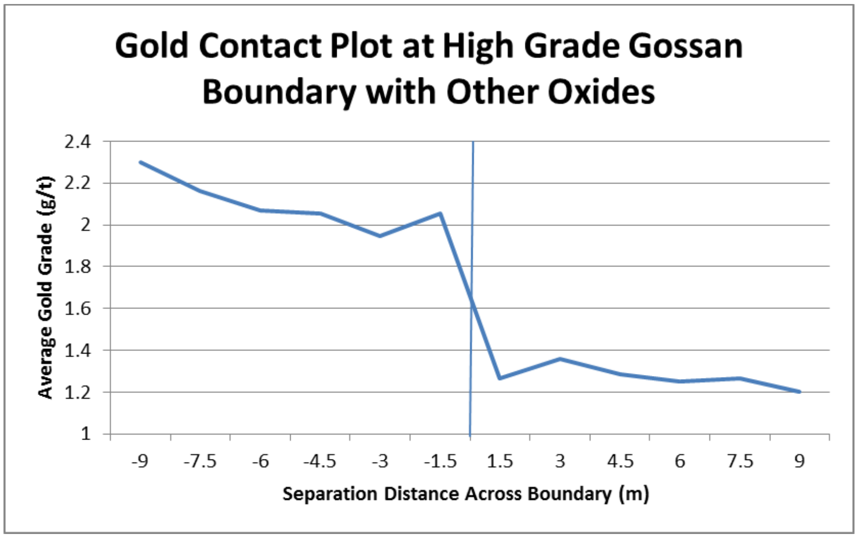


Figure 14-4
Plot of Average Gold Assay Grades Paired Across High Grade Gossan Boundary by Separation Distance

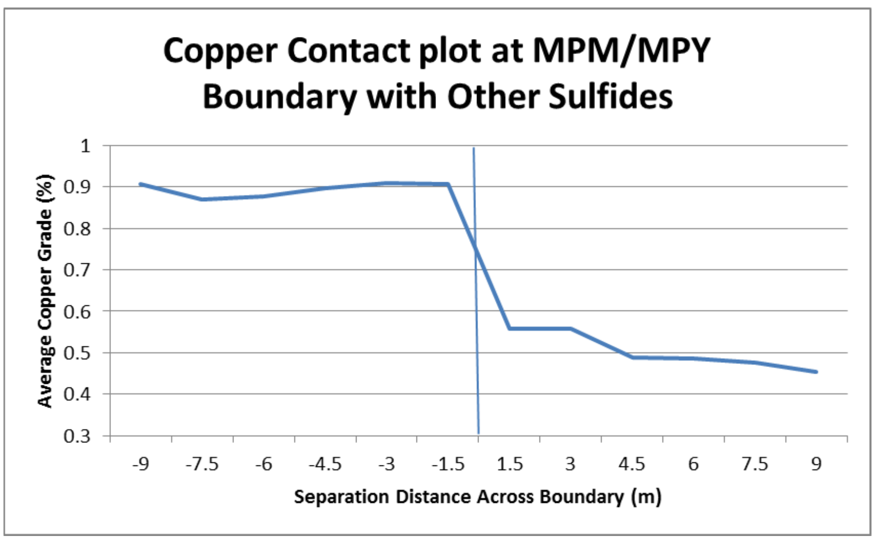


Figure 14-5
Plot of Average Copper Assay Grades Paired Across MPM/MPY Boundary by Separation Distance

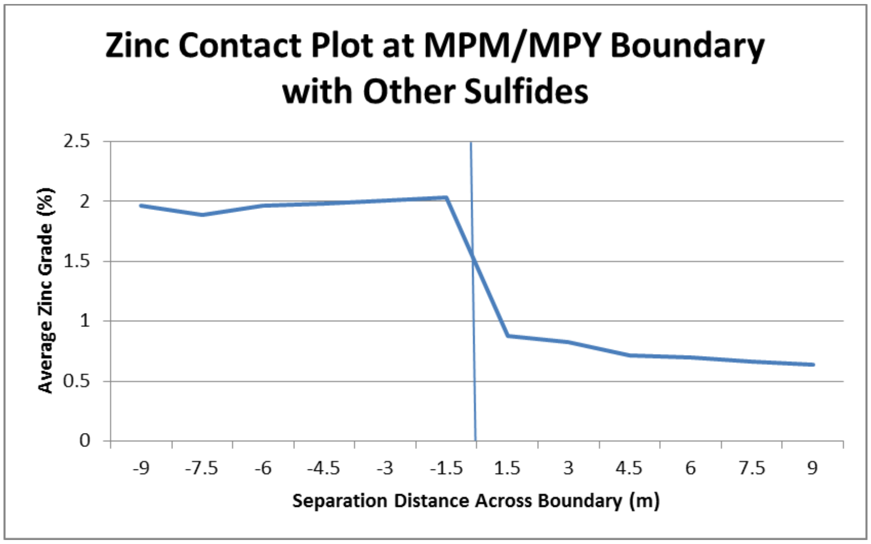


Figure 14-6
Plot of Average Zinc Assay Grades Paired Across MPM/MPY Boundary by Separation Distance

At Gediktepe, there is typically a substantial change in gold grade between the oxide zone and the sulfide zone. Some of the high grade oxide zones lie immediately above the sulfide contact. At the contact, there is an abrupt reduction in gold grade as one scans downward from the oxide to the sulfide. A plot of the capped gold assay grades down diamond core hole DRD-26 is shown in Figure 14-7 to provide an example of the change in gold grades at the oxide sulfide boundary.

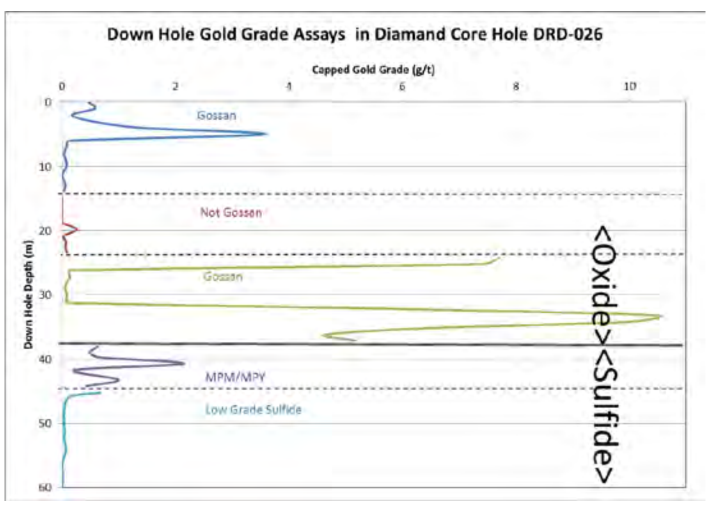


Figure 14-7
Plot of Gold Grade Down Drill Hole DRD-026

Copper and zinc react in the opposite sense in that there is little base metal in the oxide zone but substantial copper and zinc in the sulfide zone. As a result the oxide-sulfide contact is a hard boundary for grade estimation in the 'country rock or protolith' at Gediktepe.

Grade Capping

Assays were capped prior to compositing for grade estimation. A different cap was applied to each metal in each of the domains described in the previous section. Caps were applied to the logged lithology rather than the wire frame codes. Consequently, there appears to be an extra population called "Pyrite" within the logged data. Assays coded as Pyrite are predominately contained in the Chlorite Schist host rock or the "Not HG Sulfide" population.

The assay information was sorted into the seven domains and cumulative frequency plots were completed for each domain. The plots indicate the high grade outliers within each distribution. Those high grade values were replaced with a cap value that reflects the grade level above which the samples were considered as population outliers.

The number of capped assays was generally small. For example, there are 9 gold assays total in the oxide high grade zone that are above 10.0gm/t before capping. Six of these occur in intervals logged as Gossan and have been capped at 30 gm/t. The remaining 3 assay intervals fall outside of the high grade gossan. For these intervals the cap applied in this zone changed 3 assays to 10 gm/t.

Table 14-4 illustrates the cap values that were applied to each metal in each of the domains. The Pyrite domain is actually part of the Not HG Sulfide Domain.

Table 14-4
Cap Values Applied to Assays Prior to Compositing

Cap Values for August 2015 Assay Data Base									
Population		Oxide Code	Rock Type Code	Gold gm/t	Silver gm/t	Zinc %	Copper %	Mercury ppm	Arsenic ppm
Oxide	Gossan	1	10	30.000	1000.0	2.00	0.60	50.0	8000.0
	LG Gossan + Not Gossan	1	Not 10	10.000	100.0	0.50	0.65	50.0	8000.0
Sulfide	MPY	2	20	5.200	170.0	15.00	5.00	50.0	8000.0
	MPM	2	21	6.000	170.0	15.00	3.00	50.0	8000.0
	Enriched	2	50	2.400	80.0	10.00	10.00	50.0	8000.0
	Transition Sulfide	2	60	2.000	100.0	6.00	3.00	50.0	8000.0
	Pyrite	2	30	2.600	100.0	6.00	1.40	50.0	8000.0
	Not HG Sulfide	2	Not 20,21,50,60	2.100	32.0	6.00	1.20	50.0	8000.0

Only a few assays were cut. The percentage of assay caps for the primary economic metals were: Gold 0.23%, Silver 0.54%, Zinc 0.14%, and Copper 0.28%.

The capped assay values were stored in separate variables within the IMC version of the data base. For example, the un-modified gold assay was stored in a variable labeled as "au". The capped gold value was stored in a variable labeled as "au_cap". In this way, all of the original information is preserved.

From this point forward, the cap values were utilized in compositing and for grade estimation.

Compositing

Prior to grade estimation, the assay data was composited to nominal 2.5m long intervals that respect the Polimetal wireframe geologic boundaries described below. The assay data base was ‘tagged’ with the wire frame code that intersected each assay interval. The composite procedure applied to Gediktepe is designed to respect the distinct grade changes at the domain boundaries.

Composite method:

1) Compositing was applied to the “capped” assay values.

2) The assay intervals were individually coded with the population codes that were presented in the previous subsection. Where possible, they were coded by the wire frame or surface interpretations directly, rather than from the model blocks.

3) Within each drill hole, the length of the assay interval within a domain was determined and a composite length that was approximately 2.5m was applied.

4) The composite intervals start and stop at the domain boundaries. Consequently, there is no averaging across domain boundaries.

5) Composites range in length between 0.40 meters and 3.3 meters. The average composite length is 2.465 meters.

6) Short composites (minimum 0.4m) were allowed due to the narrow width of some of the high grade zones. If short intervals were not allowed, some of the higher grade zones would not be represented for grade estimation.

Table 14-5 summarizes the base composite statistics that were used for block grade estimation. Gold, silver and copper have been consistently sampled however; there are slightly fewer zinc assays and composites than for the other elements.

Table 14-5
Basic Statistics of Composite Values

Basic Statistics of August 2015 Composites Used for Block Model Estimation											
	Rock - Ore Code	Number Au, Ag, Cu	Gold gm/t	Std Dev Gold	Silver gm/t	Std Dev Silver	Copper %	Std Dev Copper	Number Zn	Zinc %	Std Dev Zinc
Oxides											
Gossan High Grade	10	438	2.283	3.766	61.90	111.76	0.13	0.22	427	0.17	0.56
Gossan Low Grade	101	389	0.559	1.366	19.27	45.61	0.07	0.12	380	0.11	0.42
NOT Gossan	31 32 33	891	0.213	1.035	7.30	33.71	0.05	0.15	869	0.05	0.12
Sulfides											
MPM - MPY	20-21	1,768	0.750	1.038	28.14	32.48	0.91	0.79	1,760	1.83	2.29
Enriched Zone	50	74	1.061	0.618	38.71	24.92	3.73	2.15	74	2.64	2.31
Transition Zone	60	329	0.284	0.467	12.29	21.12	0.40	0.39	323	0.70	1.29
Not HG Sulfides	31 32 33	16,564	0.027	0.119	1.29	4.51	0.05	0.15	16,481	0.08	0.32

Block Grade Estimation

Block grade estimation utilized ordinary linear kriging, respecting the seven domain boundaries as stated previously. Whole blocks values were used in this estimation run due to the size of the model blocks, 10m x 10m horizontally and 2.5m high. The 2.5m block height is a good match to the thickness of the high grade mineralized zone. No block partials or sub-blocking was used in this model.

Table 14-6 summarizes the grade estimation parameters. The domains are as discussed earlier in this section.

Grade estimation within the NOT Gossan and NOT High Grade Sulphide zones incorporated a high grade limit to minimize the smearing of the few moderate grade samples that occur. For example, zinc composites within the sulfide zone but not in MPY/MPM, that were greater than 0.30% were limited to a 12.5m radius. Composites that were in the oxide zone and not in the Gossan and had gold grades greater than 1.2 gm/t were limited to 12.5m radius.

The entire deposit was also divided into North and South zones to accommodate a rotation of the mineralized zone orientations between the NE and SW areas of the deposit. The boundary was established at block column 105. This is nearly the location of the L800 NE cross section. Northeast of that line, the zone strike is generally N15E. Southwest of that line, the zone strike is generally N45E. The line between the North and South zones is indicated on Figure 14-2.

The North and South areas were not boundaries for grade estimation, but indicate when the search ellipse was changed from a strike of N45E (Southwest) versus N15E (Northeast).

All grade estimation runs utilized a maximum of 10 composites and a minimum of 1 composite. A maximum of 3 composites per drill hole was also a limit. Any block that utilized 4 or more composites would have utilized at least 2 drill holes in the estimation process.

At a later stage, IMC was requested to populate the model with arsenic and mercury values. Arsenic was completely assayed and was estimated with the same procedures and domains as were applied to gold, silver, zinc, and copper. Mercury is under assayed compared to the other elements. There are generally sufficient mercury values to populate 83% of the blocks in the high grade sulfide of MPM and MPY. In order to populate the blocks that were not estimated, the average of the kriged block values was assigned to the unassigned blocks in each domain. Table 14-7 summarizes the default values for mercury

Table 14-6
Block Grade Estimation Parameters

Oxide Zone											
Oxide Zone Area	North Area, Gold, Silver, Copper, Zinc					South Area, Gold, Silver, Copper, Zinc					
	Orientation Strike and Dip	Nugget Total Sill	Major Meters	Semi Maj Meters	Minor Meters	Orientation Strike and Dip	Nugget Total Sill	Major Meters	Semi Maj Meters	Minor Meters	12.5m HG Limit Grade
Gossan High Grade	N15E 20 NW	0.10 1.00	150	75	6	N45E 20 NW	0.10 1.00	150	75	6	Au = none Ag = none Cu = none Zn = none Pb = none
Gossan Low Grade	N15E 20 NW	0.10 1.00	150	75	6	N45E 20 NW	0.10 1.00	150	75	6	Au = none Ag = none Cu = none Zn = none Pb = none
NOT Gossan in Oxide (Lith = PYS 30, QSC 31, QCS 32, CSS 33)	N15E 20 NW	0.10 1.00	150	75	6	N45E 20 NW	0.10 1.00	150	75	6	Au = 1.20 gm/t Ag = 75 gm/t Cu = 0.30% Zn = 0.40% Pb = 1.0%

Sulfide Zone											
Sulfide Area	North Area, Gold, Silver, Copper, Zinc					South Area, Gold, Silver, Copper, Zinc					
	Orientation Strike and Dip	Nugget Total Sill	Major Meters	Semi Maj Meters	Minor Meters	Orientation Strike and Dip	Nugget Total Sill	Major Meters	Semi Maj Meters	Minor Meters	12.5m HG Limit Grade
Massive Pyrite (MPY 20) AND Massive Pyrite Magnetite (MPM 21)	N15E 20 NW	0.10 1.00	150	75	6	N45E 20 NW	0.10 1.00	150	75	6	Au = none Ag = none Cu = none Zn = none Pb = none
Enriched Zone (ERH 50)	N15E 20 NW	0.10 1.00	150	75	6	N45E 20 NW	0.10 1.00	150	75	6	Au = none Ag = none Cu = none Zn = none Pb = none
Transition Zone (TRS 60)	N15E 20 NW	0.10 1.00	150	75	6	N45E 20 NW	0.10 1.00	150	75	6	Au = none Ag = none Cu = none Zn = none Pb = none
NOT High Grade Sulfide (Lith = PYS 30, QSC 31, QCS 32, CSS 33)	N15E 20 NW	0.10 1.00	150	75	6	N45E 20 NW	0.10 1.00	150	75	6	Au = 0.25 gm/t Ag = 10 gm/t Cu = 0.90% Zn = 0.30% Pb = 2.0%

For ALL Domains and Two Areas:

Maximum composites = 10
Minimum Composites = 1
Maximum composites per drill hole = 3.

Table 14-7
Default Values for Mercury

Oxide vs Sulfide Assignments	High Grade Gossan 10	Low Grade Gossan 101	Massive Pyrite 20	Magnetite Pyrite 21	Enriched Zone 50	Transition Zone 60
OXIDE						
Total Number of Blocks	6,603	6,556				
Kriged Blocks	4,366	3,021				
Mean Mercury (ppm)	6.472	2.226				
# Blocks Assigned	2,237	3,535				
SULFIDE						
Total Number of Blocks			17,114	14,308	590	4,241
Kriged Blocks			14,676	11,242	417	1,792
Mean Mercury (ppm)			2.175	1.446	2.033	1.423
# Blocks Assigned			2,438	3,066	173	2,449

Density

Density was assigned to the block model based on the specific gravity tests that were collected by Polimetal. The average in-situ dry density values were applied to each rock type. Blocks within the model with undefined rock type were assigned the average of the Hanging Wall Quartz Schist (QSCH-31) and Footwall Quartz Feldspar (QF-32)

The densities for all materials including waste are summarized below on Table 14-8.

Table 14-8
Density Assignments

Model Lith Code	Lithologic Unit	Average Density	Number of Samples	Block Weight Ktonnes 10x10x2.5
10	Gossan	2.526	298	0.6315
20	Massive Pyrite	4.327	467	1.0818
21	Massive Pyrite - Magnetite	4.371	391	1.0928
31	Hanging Wall Quartz Schist	2.673	511	0.6683
32	Footwall Qtz Feldspar Schist	2.678	744	0.6695
33	Clorite Sericite Schist	2.736	1405	0.6840
50	Enriched Zone	4.071	33	1.0178
60	Transition Sulfide PYS30 + TRS60 Averaged	3.194	483	0.7985
No Code	Default Average of 31 and 32	2.676	1255	0.6690

Number of samples reflect the amount of supporting data in each unit.

Classification

The block classification was assigned based on the number of composites used to estimate a block, and the average distance between the block center and all of the composites used to estimate the block.

The gold estimate was used to set the classification in all domains. This is possible because equal numbers of drill intervals were assayed for gold, silver, and copper. Only slightly fewer zinc assays were completed.

To set the confidence codes in the August 2015 Gediktepe block model, the following steps were used:

1) All estimated blocks for gold were initially coded as inferred (conf=3).

2) Blocks were coded as indicated (conf=2) if there was a gold grade estimated which used four or more composites (au_num >=4) which equates to at least 2 drill holes, and the average distance to the closest composite was 75m or less (avedist <=75).

3) If model blocks are in the sulfide mineralized units MPY, MPM, ERH and TRS (geol_0815= 20, 21, 50, 60) and three composites were found for the estimation (au_num =3) and the average distance to the closest composite was 75m or less (avedist <=75) these blocks were also set to indicated (conf=2).

4) Measured blocks (conf=1) are those model blocks having a gold grade estimated using the maximum number of composites (au_num=10) and the average distance to the closest composite was 35m or less (avedist <=35).

The third step in assigning the confidence codes was established so that contiguous mineralization in the narrow sulfide high grade zones can be considered as Indicated model blocks. The fourth step of setting the confidence codes allows model blocks in areas with closely spaced drilling (less than 35m apart) be classified as Measured material.

Model Validation

A number of tests were completed to check or validate the model. Those tests included:

1) Visual comparison on plan maps and cross-sections of drill hole results versus model results.
2) A bias check of the model was completed by comparing the mean grade of composites, nearest neighbor estimation, and the final estimated model grade within each population domain.

3) Scan line plots down selected drill holes. Drill hole grades are compared to the block model grades and nearest neighbor grades to confirm that the model respected the estimation boundaries.
4) Swath plots by elevation were prepared that compared nearest neighbor results versus the final block grade estimates.
5) An IMC smearing check where the contained composite grade distribution within each domain is compared to the block grade distribution in each domain at selected cutoff grades.

The model was re-estimated using the Nearest Neighbor (NN) method to compare against the kriged (OK) resource model in order to test for bias. The same search ellipses and population boundaries were used for both methods. In Table 14-9, the average grades of the 4 economic minerals are tabulated in each domain at a zero cutoff for the OK model, the NN model, and also the average grade of the composites within each domain. The OK model shows very little bias compared to the NN model. The greatest bias found in an ore zone is a 6% higher average NN silver grade in the low grade gossan domain compared with the average OK silver grade.

Table 14-9
Number of Blocks/Composites and Average Grade in Each Estimation Domain
to Compare the NN Model, OK Model and Composites

	Number of Blocks and Avg. Grade by OK				Number of Blocks and Avg. Grade by NN				Number of Composites and Avg. Grade			
	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Gossan High Grade blks/comps	6,603	6,603	6,603	6,603	6,603	6,603	6,603	6,603	438	438	438	429
grd	2.206	59.3	0.13	0.196	2.216	59.4	0.13	0.186	2.283	62.3	0.13	0.174
Gossan Low Grade blks/comps	6,556	6,556	6,556	6,556	6,556	6,556	6,556	6,556	389	389	389	384
grd	0.363	12.3	0.08	0.126	0.376	13.1	0.08	0.124	0.559	19.2	0.07	0.109
Not Gossan, in Oxide blks/comps	58,969	58,969	58,969	58,969	58,969	58,969	58,969	58,969	891	891	891	883
grd	0.009	0.5	0.02	0.013	0.013	0.7	0.01	0.012	0.213	7.3	0.05	0.061
MPY/MPM blks/comps	31,422	31,422	31,422	31,422	31,422	31,422	31,422	31,422	1,768	1,768	1,768	1,760
grd	0.746	28.1	0.83	1.842	0.749	28.3	0.83	1.864	0.750	28.1	0.91	1.840
Enriched Zone blks/comps	590	590	590	590	590	590	590	590	74	74	74	74
grd	1.068	40.9	3.77	2.810	1.073	41.6	3.87	2.850	1.061	38.7	3.73	2.664
Transition Zone blks/comps	4,241	4,241	4,241	4,241	4,241	4,241	4,241	4,241	329	329	329	323
grd	0.321	12.7	0.40	0.743	0.318	12.6	0.40	0.749	0.274	12.3	0.40	0.736
Not High Grade Sulfide blks/comps	906,488	906,488	906,488	906,488	906,488	906,488	906,488	906,488	16,564	16,564	16,564	16,495
grd	0.007	0.4	0.01	0.013	0.009	0.5	0.01	0.014	0.027	1.3	0.05	0.080

Drill holes were "dipped" into the model to compare composite grades against the block grades that the composite was located in. This allowed a comparison between the OK and NN estimates against the composite grades on a localized basis. Plots of the gold grades and zinc grades down DRD-026 and DRRC-139 respectively are shown in Figures 14-8 and 14-9.

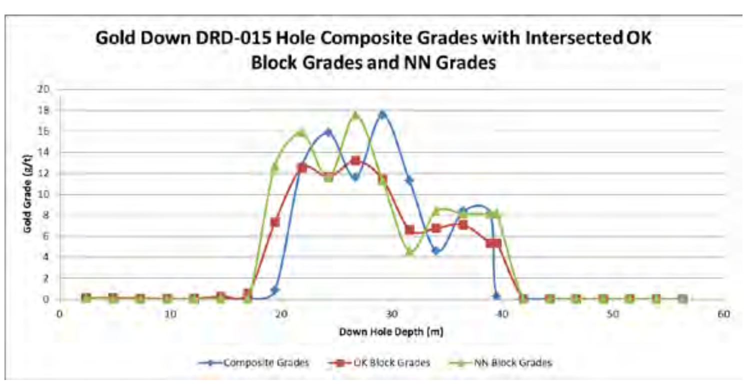


Figure 14-8
Down Hole Gold Grade Plot of: Composite Grades, OK Block Grades, NN Block Grades

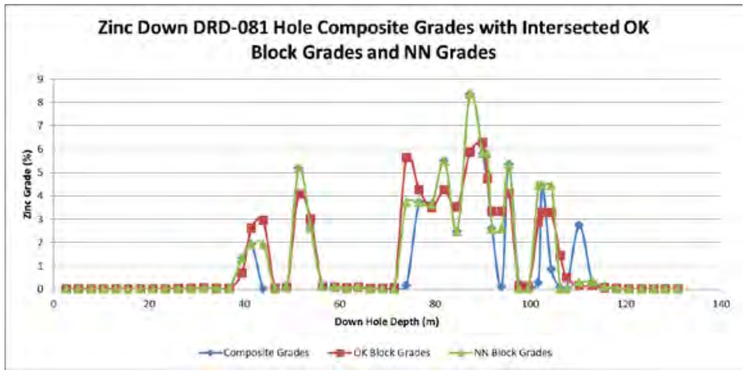


Figure 14-9
Down Hole Zinc Grade Plot of: Composite Grades, OK Block Grades, NN Block Grades

Swath plots comparing the OK model to the NN model were generated for the 4 economic metals and are provided in Figures 14-9 through 14-12. They show the average grades bench by bench stepping down through the model. The OK model and the NN model show similar grade profiles vertically through the model.

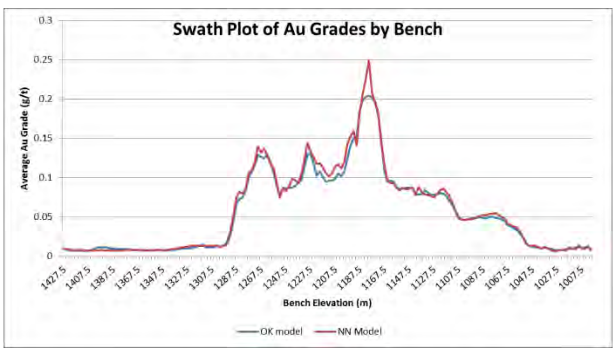


Figure 14-10
Vertical Swath Plot of Au Through Model

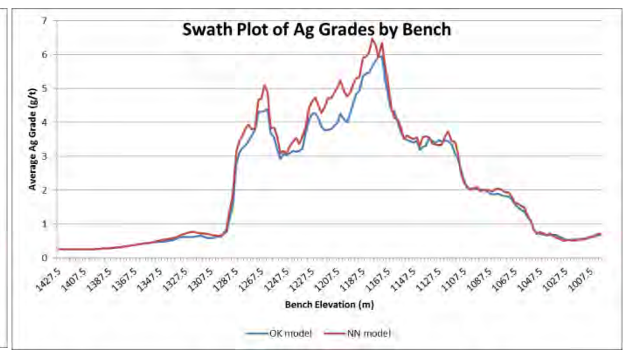


Figure 14-11
Vertical Swath Plot of Ag Through Model

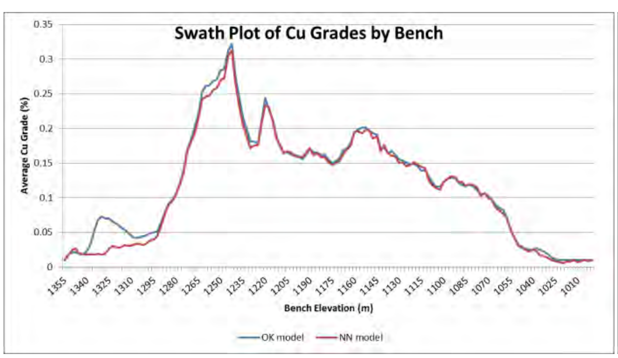


Figure 14-12
Vertical Swath Plot of Cu Through Model

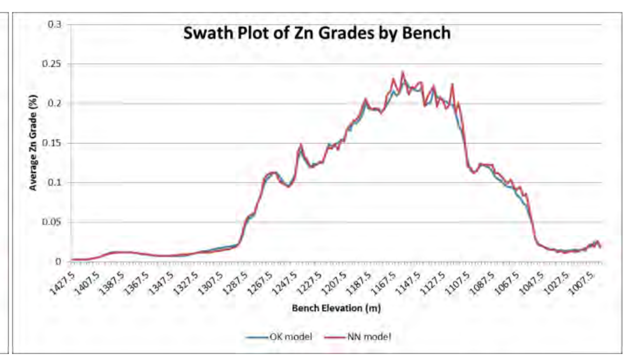


Figure 14-13
Vertical Swath Plot of Zn Through Model

The IMC smearing check compares the average block model grade at a given cutoff against the average grade of the contained composites. For example, a 0.50% total copper cutoff was applied to the model within a domain. The average grade of the model blocks contained within a the plus 0.50% Cu block grade volume and the tested domain boundaries are tabulated and compared against the average grade of the composites contained in the same volume. Comparison of averages is a check on potential model bias. The percentage of composites within the outline that are less than the selected block cutoff is a measure of the grade smearing of the model. The process is repeated at a number of cutoff grades.

The model grade should always be lower than the grade of the contained composites due to the search parameters used in block grade estimation. Table 14-10 summarizes the results for the primary economic domains and metals at Gediktepe. In all cases the block grades are appropriately less than the contained composite grades. The percentage of composites less than the tested cutoff are typical for ordinary kriging results.

Table 14-10
Comparison of Average Metal Grades in the Block Model and in the Composites in Volumes Defined as the Intersection of the Model Blocks Above the Stated Cutoff and the Blocks Within the Stated Domain

Gold in Gossan					
Cutoff Grade Tested gm/t	Percentage of Composites Less than Cutoff	Number of Composites Composites	Composite Grade gm/t	Number of Blocks	Block Grade gm/t
0.00	0.0%	438	2.28	6,447	2.26
1.00	27.3%	242	3.91	3,614	3.79
2.00	22.0%	159	5.45	2,429	4.90
3.00	21.4%	112	6.69	1,683	5.99
4.00	20.5%	78	8.10	1,251	6.87
5.00	24.1%	54	9.48	896	7.82
Copper in MPY/MPM					
Cutoff Grade Tested %	Percentage of Composites Less than Cutoff	Number of Composites Composites	Composite Grade %	Number of Blocks	Block Grade %
0.00	0.0%	1,768	0.91	31,060	0.84
0.50	12.9%	1,383	1.06	25,915	0.94
1.00	20.5%	488	1.67	7,465	1.42
1.50	21.3%	178	2.50	1,800	2.22
2.00	14.9%	87	3.30	794	2.88
2.50	16.9%	59	3.78	479	3.33
Zinc in MPY/MPM					
Cutoff Grade Tested %	Percentage of Composites Less than Cutoff	Number of Composites Composites	Composite Grade %	Number of Blocks	Block Grade %
0.00	0.0%	1,760	1.84	31,062	1.86
1.00	18.5%	977	3.00	19,509	2.73
2.00	18.4%	621	3.94	12,009	3.52
3.00	18.9%	370	4.99	6,796	4.32
4.00	15.4%	208	6.10	3,562	5.10
5.00	18.8%	101	7.27	1,483	6.02

Mining Dilution

Mining dilution was accounted for in the block estimation process and no additional factor was added or applied to the block model. Model blocks that are 2.5m high were used on a nearest whole block basis to estimate the boundaries of the high grade zones. Mining selection and resolution was not assumed to be finer than 2.5m.

Backhoe loading units are assumed in the operation in order to obtain 2.5m vertical resolution. Mining benches and blasting are generally planned on 5m benches. Visual grade control and the application of a unit like a Niton in the sulfide zones will be necessary to obtain the 2.5m selection. Within the oxide zone, additional blast hole samples may be required to split the 5m benches with upper and lower assays for 2.5m selection.

14.4 Mineral Resource

Mineral resources were developed using the floating cone algorithm. The floating cone algorithm utilizes estimated metal prices and production costs along with process recoveries and slope angles to generate a theoretical open pit geometry. For the resource cone, economic benefit was applied to measured, indicated, and inferred class mineralization within the block model.

The process recovery and smelter cost parameters for the resource cone are the same as applied to the mine plan schedules that are summarized on Tables 15-5 and 15-6 in the section on mineral reserves. For the estimation of resources, inferred mineralizaiton is allowed to contribute economic credit. Metal prices were altered upward slightly for the mineral resource compared to the mineral reserve. The metal prices for the mineral resource are: $1,200/oz gold, $18.00/oz silver, $3.00/lb copper, and $1.20 /lb zinc.

The estimate of mineral resources was completed early in the project life, prior to the calculation of the mineral reserve and financial analysis. As a result, the cost of processing inclusive of general and administrative costs is different from cost that was used to establish the mining plan in Sections 15 and 16 and the financial analysis of Section 22. Using the updated processing and G&A costs would not cause a material change of the mineral resource due to the abrubt grade change between mineralized and non-mineralized material in the Gediktepe deposit.

The mineral resource floating cone applied process + G&A Costs as follows:

Oxide Processing = $6.92/tonne + G&A $4.78/tonne = $11.70/tonne
Sulfide Processing = $10.89/tonne + G&A $4.78/tonne = $15.67/tonne

The slope angles are based on recent geotechnical work at Gediktepe by Fugro-Sial in July 2015. Weathered rock units are estimated at 42 degrees, and intact rock is estimated at 48 degree overall slopes. For simplicity, a 48 degree overall slope angle was used for the mineral resource since the weathered rock is generally on the shallow southeast side of the deposit and changing the angle would have little impact on the resource in that area.

The process recoveries and concentrate quality are based on metallurgical testing and judgement regarding the performance of a full scale plant.

The qualified person for the mineral resouce is John Marek of Independent Mining Consultants, Inc. The mineral resource will change as additional drilling is completed and as more detailed process recovery information becomes available. Metal prices could materially change the resources in either a positive or negative way.

The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be realized or that they will convert to mineral reserves.

The contained copper and zinc metal within the oxide zone are not presented on the statement of mineral resources because there is no process planned to produce those metals in the oxide zone. The grades of copper and zinc are shown because their presence has an impact on the design of the oxide process plant and oxide processing costs.
The mineral resource estimate for Gediktepe is summarized on Table 14-11. The stated mineral resources include the mineral reserve.

Table 14-11
Gediktepe Mineral Resources, 1 June 2016
Mineral Resources Include the Mineral Reserves

			Head Grades				Contained Metal			
Material Type Classification	NSR Cutoff $/t	Tonnages ktonnes	Au gm/t	Ag gm/t	Cu %	Zn %	Au koz	Ag koz	Cu klb	Zn klb
Oxides										
Measured	$11.70	1,722	2.645	66.5	0.12	0.16	146.4	3,680		
Indicated	$11.70	2,110	2.561	71.0	0.18	0.35	173.7	4,817		
Meas+Ind.	$11.70	3,832	2.599	69.0	0.15	0.26	320.2	8,497		
Inferred	$11.70	213	1.574	63.1	0.13	0.17	10.8	432		
Sulfides										
Measured	$15.67	12,027	0.777	28.5	1.00	1.89	300.4	11,030	263,824	501,133
Indicated	$15.67	20,180	0.773	30.1	0.85	1.95	501.5	19,506	378,158	867,540
Meas+Ind.	$15.67	32,207	0.774	29.5	0.90	1.93	802.0	30,536	641,982	1,368,673
Inferred	$15.67	1,685	0.807	31.7	0.98	1.80	43.7	1,719	36,256	66,866
Oxides+Sulfides										
Measured	11.70/15.67	13,749	1.011	33.3	0.89	1.67	446.9	14,710	263,824	501,133
Indicated	11.70/15.67	22,290	0.942	33.9	0.79	1.80	675.3	24,323	378,158	867,540
Meas+Ind.	11.70/15.67	36,039	0.968	33.7	0.82	1.75	1,122.1	39,033	641,982	1,368,673
Inferred	11.70/15.67	1,898	0.893	35.3	0.88	1.62	54.5	2,151	36,256	66,866

Notes:
Mineral resources include the mineral reserve
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
The Qualified Person for the Mineral Resource is John Marek, RM-SME
Summation errors are due to rounding
Metal Prices Used: Gold: $1,200/oz. Copper: $3.00/lb. Zinc: $1.20/lb. Silver: $18/oz.
Tonnages are reported in 000's of metric tonnes
Contained precious metal reported in 000's troy ounces, contained base metal reported in 000's of lbs.
Contained copper and zinc not reported for oxides. No recovery potential is expected for these metals in the oxide zone
Copper and zinc grades are reported in the oxide zone because they have an impact on process plant design and costs
Floating cone inputs used to define Resource:
- Mining Cost=$1.47/tonne
- G&A Costs=$4.78/tonne ore
- Oxide Processing: $6.92/tonne, Sulfide Processing: $10.89/tonne
- Pit Slope Angle: 48°

15.0 MINERAL RESERVES ESTIMATES

The mine plan and mineral reserve was developed by Independent Mining Consultants, Inc. (IMC) with John Marek acting as the Qualified Person for this section. The mine plan and mineral reserve were based on the block model that was summarized in Section 14 combined with economic evaluation and detailed mine planning.

Two ore types are produced from the Gediktepe mine plan:
1) an oxide ore that is heap leached on site, and
2) a sulfide ore that is processed on site to produce a salable copper concentrate and a salable zinc concentrate.

The mineral reserve is the total of all proven and probable category material that is planned for production. The mine plan that is presented in Section 16 details the production of that reserve. The final pit design and internal phase designs that contain the mineral reserve were guided by the results of the floating cone algorithm.

15.1 Floating Cones

The floating cone algorithm is a tool for phase design guidance. The algorithm applies approximate costs and recoveries along with estimated pit slope angles to establish theoretical economic breakeven pit wall locations.

Economic input applied to the cone algorithm is necessarily preliminary as it is one of the first steps in the development of the mine plan. The cone geometries should be considered as approximate as they do not assure access or working room. The important result of the cones is the relative change in geometry between cones of increasing metal prices. Lower metal prices result in smaller pits which provide guidance to the design of the initial phase designs. The changes in pit geometry as metal prices are increased indicate the best directions for the succeeding phase expansions to the ultimate pit. Measured and indicated class mineralization received economic credit. Inferred was treated as waste for mine planning.

Multiple floating cones were run at a range of metal prices. The targeted metal prices for mine planning were:

Au: $1,000/oz. Cu: $2.50/lb. Ag: $15.00/oz. Zn: $1.00/lb.

Those prices were factored upward and downward (revenue factors) from 0.40 to 1.40 of the base case. Figure 15-4 illustrates how the designed phases follow the progression of the floating cones as the metal prices are increased.

The process costs and recoveries used for input to the cones were originally based on earlier work at Gediktepe and subsequent floatation studies. These are provided in Tables 15-1 through 15-3. Treatment costs and refining charges (TCRC) are an average of smelter terms received from 2 separate concentrate traders. Net Smelter Return was calculated on a block by block basis to indicate the value of a block. Net Smelter Return was chosen for cutoff grade determination because Gediktepe is a polymetallic project with separate product streams that are combined to determine a block's overall value.

Table 15-1
Process Recoveries for Floating Cone Input

Oxide Ore			
Gold: 65.921% *(Au grade g/t)^.2314 (87.5% max)			
Silver: 45%			
Sulfide Ore			
Metal Recovery to Copper Concentrate:		Metal Recovery to Zinc Concentrate:	
Copper:	66%	Zinc:	84%
Gold:	32%	Gold:	0%
Silver:	17%	Silver:	17%

Table 15-2
TCRC Costs

Oxide Ore	
Gold Pay: 99%	Silver Pay: 98%
Gold Transport: $5.00/oz.	Silver Transport: $0.50/oz.
Sulfide Ore	
Copper Conc. Terms Cu Grade of 23.8%	Zinc Conc. Terms Zn Grade of 53.5%
Copper: Pay 96.5% after (1% ded. from Cu content) Gold: Pay 90% after (1 g/t ded. from Au content) Silver: Pay 90% after (30 g/t ded. from Ag content)	Zinc: Pay lesser of 85% of Zinc content or Zn content less 8% Silver: Pay 75% after (77.8 g/t deduct from Ag content)
Treatment Charge: $85.00/dry tonne Refining Charges: Cu: $0.085/lb Au: $4.50/oz Ag: $0.45/oz	Treatment Charge: $259.80/dry tonne escalator of $0.10/dollar zinc price above $1850/t (if Zn>$1850/t, treatment = $259.80 + (Zn pr-1850)*.1) de-escalator of $0.14/dollar zinc price below $1850/t (if Zn<$1850/t, treatment = $259.80 - (1850-Zn pr)*.14)
Assume conc. contains 9% water	Assume conc. contains 9% water
Ocean Freight: $40.00/wet tonne	Ocean Freight: $45.00/wet tonne
Port Charge: $10.00/wet tonne	Port Charge: $10.00/wet tonne
Land Freight: $14.10/wet tonne	Land Freight: $14.10/wet tonne
Insurance: 0.088% of CIF	Insurance: 0.11% of CIF

Table 15-3
Processing and G&A Costs

Cost Category	Cost $/tonne ore
PEA Direct Oxide Processing Cost:	4.20
Factored PEA Indirect Oxide Processing Cost:	<u>2.78</u>
Estimate of Oxide Processing Cost:	6.98
PEA Direct Sulfide Processing Cost:	11.15
Factored PEA Indirect Sulfide Processing Cost:	<u>2.50</u>
Estimate of Sulfide Processing Cost:	13.65
Factored PEA G&A Cost:	6.15

A mining cost of $1.47/tonne of material mined was used in the floating cones based on previous work in 2014 with an unpublished PEA. Floating cone slope angles were simplified to 48 degrees everywhere for these cone runs. When designing phases, shallower slopes were applied in weathered near surface material. Those weathered zones are sufficiently shallow that they would not impact the pit wall location in the floating cone.

15.2 Selection of Appropriate Guidance Cone

Floating cones of increasing size were evaluated to determine the pit geometry that would produce a robust mine schedule at the design metal prices of: Au: $1,000/oz. Cu: $2.50/lb. Ag: $15.00/oz. Zn: $1.00/lb. This was accomplished by floating a suite of cones at "revenue factors" between 0.4 and 1.4 and comparing the value of the increasing cone tonnages tabulated at the design metal prices. The factors were applied by multiplying the metal price inputs by the revenue factors. For example, at a revenue factor of 0.8, the metal prices used as input to the floating cones were: Au: $800/oz. Cu: $2.00/lb. Ag: $12.00/oz. Zn: $0.80/lb. The tonnages within each revenue factor pit were re-tabulated at the design metal prices and a value was assigned to each cone of:

NSR of ore grade material
– processing/G&A cost of ore grade material
<u>– mining cost of total material.</u>
= Value of Cone

The tonnages tabulated at the targeted design metal prices within each cone are given in Table 15-4.

Table 15-4
Tabulation of Tonnages within Floating Cone Geometries at $1,000/oz Gold Price

	Oxide at $13.13 NSR Cutoff				Sulfide at $19.80 NSR Cutoff						Total
Revenue	kt	NSR	au	ag	kt	NSR	au	ag	cu	zn	Material
Factors	ktonnes	$/t	g/t	g/t	ktonnes	$/t	g/t	g/t	%	%	ktonnes
0.40	3,300	91.73	2.897	75.9	3,692	85.93	1.162	41.2	1.74	2.42	21,366
0.50	3,449	89.34	2.826	74.2	9,673	74.92	1.113	42.6	1.23	2.72	51,263
0.60	3,519	88.08	2.789	73.4	12,604	69.96	1.023	39.6	1.15	2.57	66,999
0.70	3,526	87.98	2.785	73.3	17,966	66.18	0.994	37.8	1.05	2.49	116,297
0.80	3,534	87.82	2.780	73.2	20,815	64.10	0.957	36.6	1.02	2.43	145,931
0.90	3,538	87.74	2.778	73.2	21,985	63.03	0.940	35.9	1.00	2.39	157,753
1.00	3,539	87.72	2.778	73.2	23,520	61.75	0.917	35.0	0.98	2.33	177,756
1.10	3,538	87.74	2.778	73.2	24,385	60.88	0.901	34.3	0.98	2.29	188,402
1.20	3,540	87.70	2.770	73.1	29,447	56.77	0.826	31.5	0.95	2.06	250,704
1.30	3,542	87.68	2.776	73.1	29,645	56.68	0.825	31.4	0.94	2.05	256,691
1.40	3,540	87.70	2.777	73.1	29,686	56.66	0.824	31.4	0.94	2.06	257,652

A curve of the values contained within the floating cones is provided in Figure 15-1. There appears to be only marginal benefit of mining a pit larger than the 0.80 revenue factor pit at the design metal prices. IMC designed the final pit at Gediktepe to capture all of the ore within the 0.80 revenue factor cone.

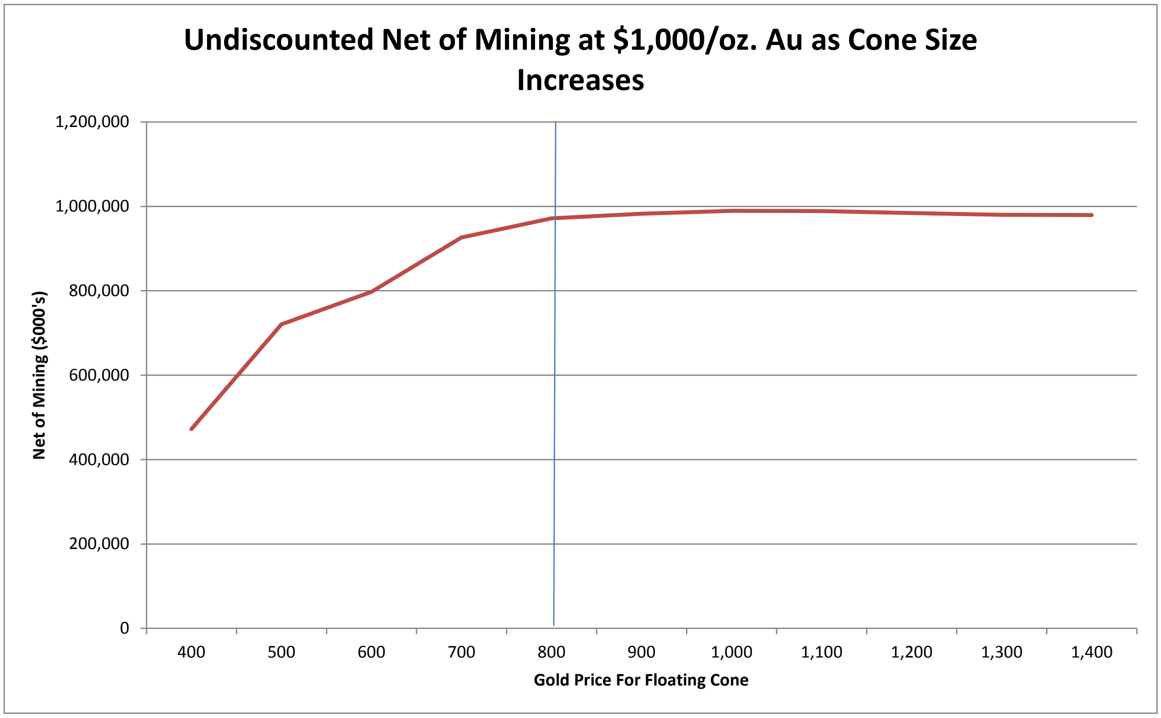


Figure 15-1
Value of Floating Cones when Evaluated at Design prices

15.3 Figures of Guidance Cones and Ultimate Pit Designs

The $800/oz. cone selected for pit design guidance is provided in Figure 15-2. For comparison, the ultimate pit is provided in Figure 15-3 at the same scale. Figure 15-4 illustrates how the designed phases follow the progression of the floating cone geometries as the metal prices are increased.

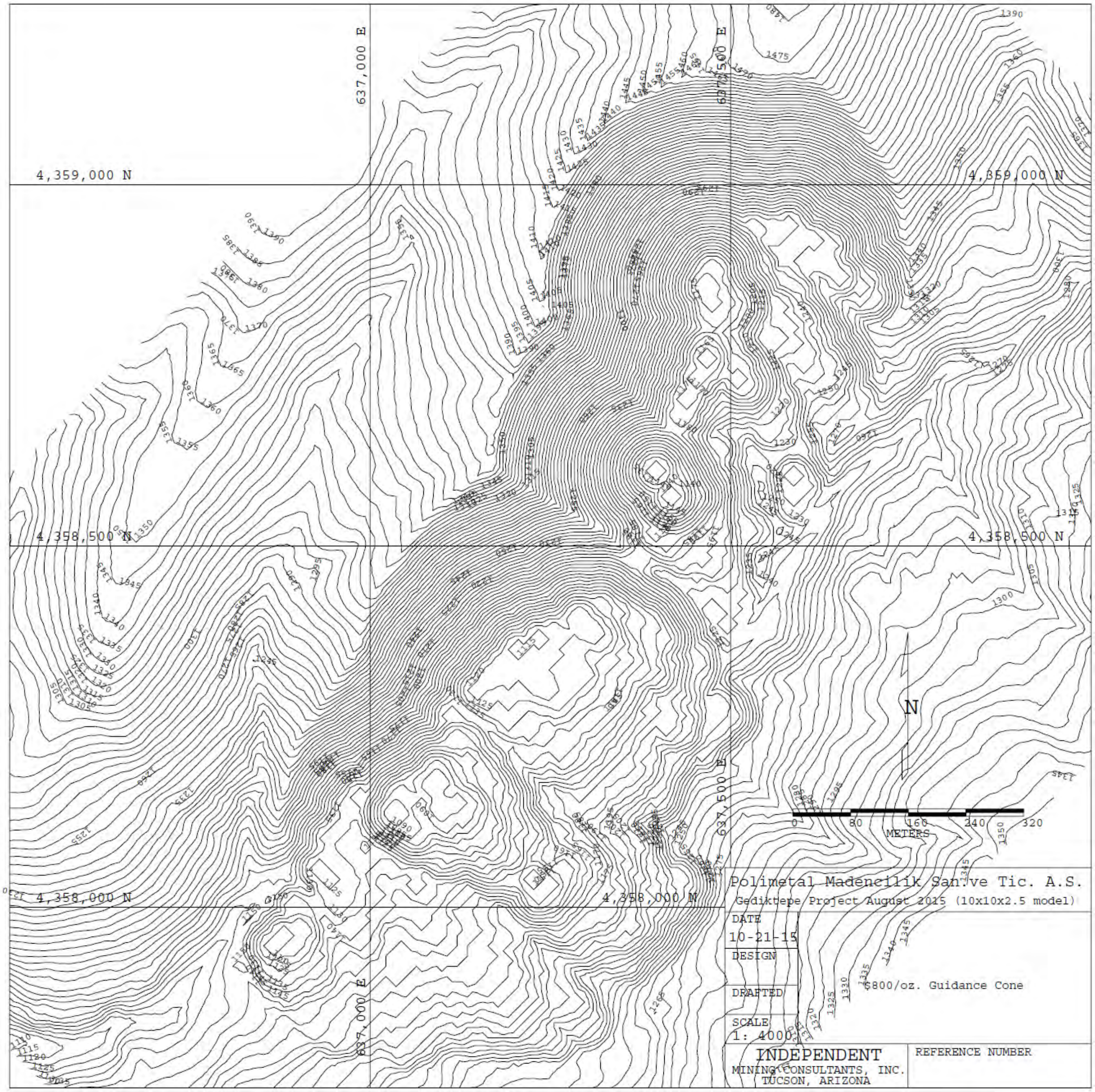


Figure 15-2
$800/oz Gold, $2.00/lb Cu, $0.80/lb Zn, and $12.00/oz Ag Cone
Selected for Ultimate Pit Design Guidance

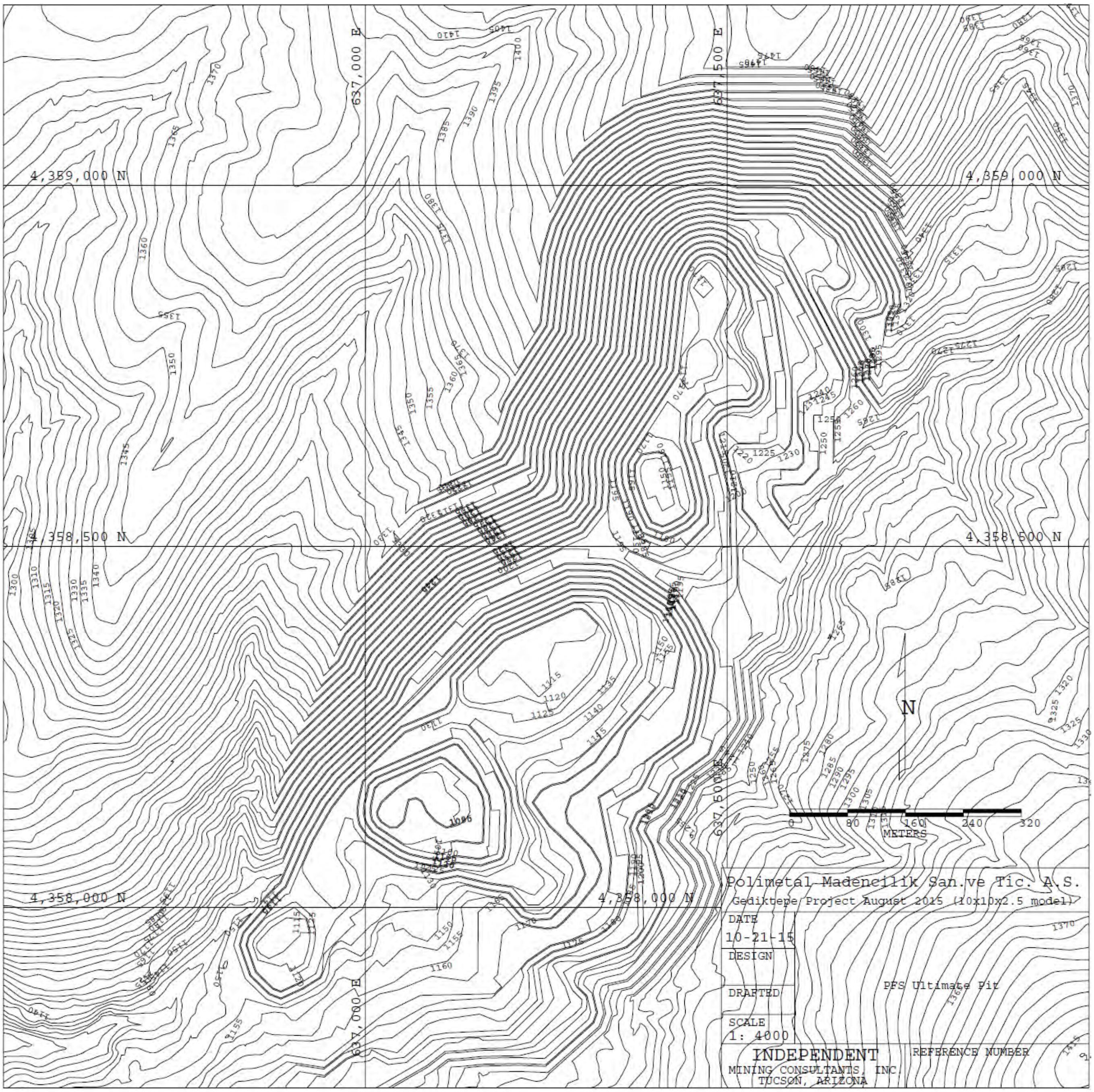


Figure 15-3
Ultimate Pit Design

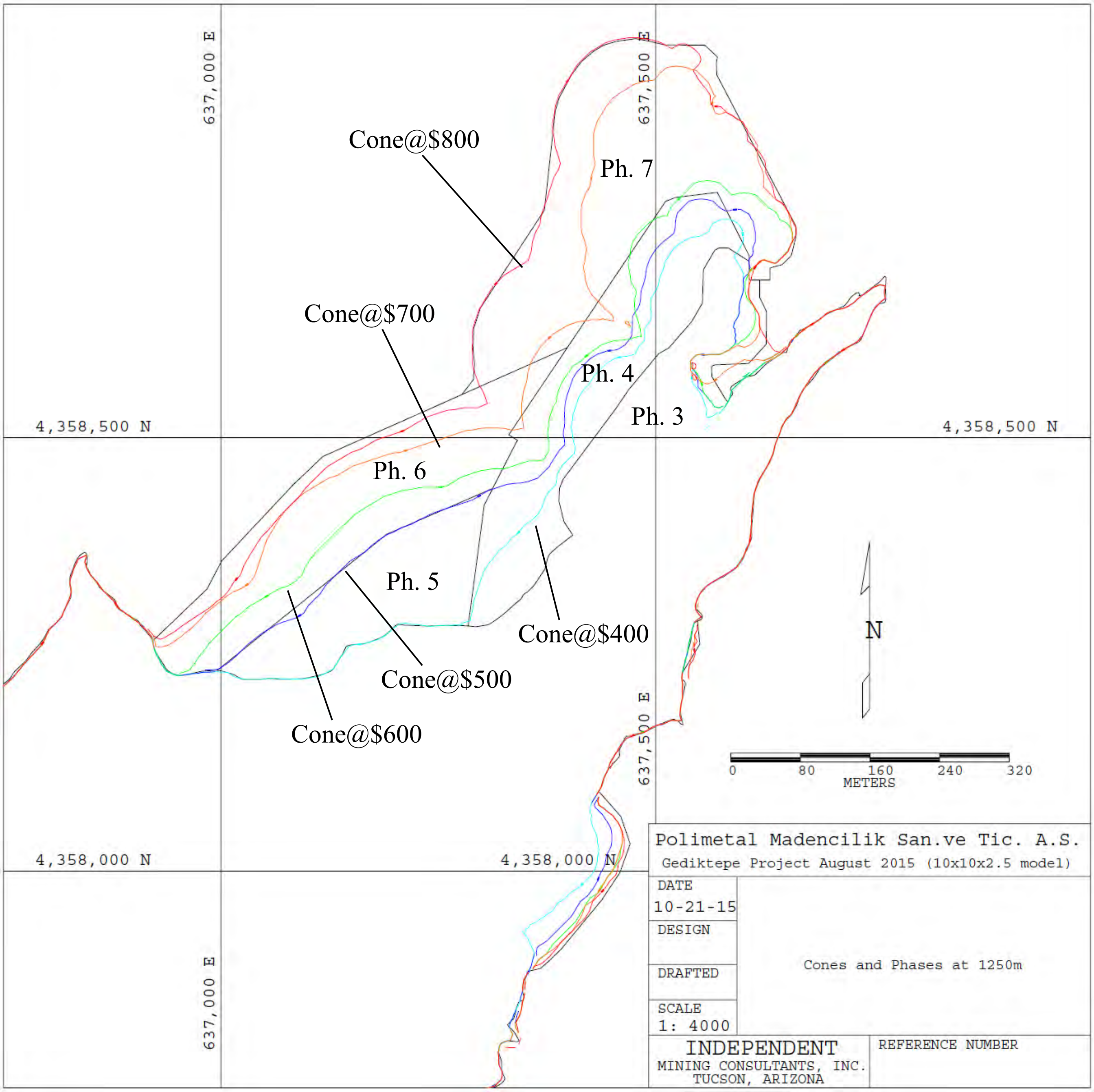


Figure 15-4
Mining Phases 3 through 7 and Floating Cones between $400/oz and $800/oz Gold Price
Sliced at Elevation 1250 m

15.4 Pre-Feasibility Updated Costs and Recoveries

Updated Pre-Feasibility recoveries and processing costs became available at the end of the Pre-Feasibility study work. The block model values were updated with these revised inputs for mine planning purposes. The final inputs used to calculate block Net of Process values are provided in Tables 15-5 through 15-7. Metal Prices of $1,000/oz. Gold, $2.50/lb Copper, $1.00/lb Zinc, and $15.00/oz. Silver were used.

IMC conducted a sensitivity check on the impact of incorporating the new inputs and determined that the existing phase designs are acceptable for use in Pre-Feasibility mine planning and no re-design of the final pit or phases was necessary.

Table 15-5
Process Recoveries

Oxide Ore			
Gold: 65.921% *(Au grade g/t)^.2314 (87.5% max)			
Silver: 45%			
Sulfide Ore			
Metal Recovery to Copper Concentrate:		Metal Recovery to Zinc Concentrate:	
Copper:	69.2%	Zinc:	81.5%
Gold:	17.2%	Gold:	15.7%
Silver:	12.3%	Silver:	21.5%

Table 15-6
TCRC Costs

Oxide Ore	
Gold Pay: 99%	Silver Pay: 98%
Gold Transport: $5.00/oz.	Silver Transport: $0.50/oz.
Sulfide Ore	
Copper Conc Terms Cu Grade of 30.95%	Zinc Conc Terms Zn Grade of 54.3%
Copper: pay lesser of 96.5% or Cu content less 1%	Zinc: Pay lesser of 85% of Zinc content or Zn content less 8%
Gold: pay lesser of 90% or Au content less 1 g/t	Gold: Pay 70% after (1g/t deduct from Au content)
Silver: pay lesser of 90% or Ag content less 30 g/t	Silver: Pay 70% after (93.31 g/t deduct from Ag content)
Treatment Charge: $85.00/ dry tonne	Treatment Charge: $259.80/dry tonne
Refining Charges:	escalator of $0.10/dollar zinc price above $1850/t
Cu: $0.085/lb	(if Zn>$1850/t, treatment = $259.80 + (Zn pr-1850)*.1)
Au: $5.00/oz	de-escalator of $0.14/dollar zinc price below $1850/t
Ag: $0.40/oz	(if Zn<$1850/t, treatment = $259.80 - (1850-Zn pr)*.14)
Assume conc. contains 9% water	Assume conc. contains 9% water
Ocean Freight: $40.00/wet tonne	Ocean Freight: $45.00/wet tonne
Port Charge: $10.00/wet tonne	Port Charge: $10.00/wet tonne
Land Freight: $14.10/wet tonne	Land Freight: $14.10/wet tonne
Insurance: 0.088% of CIF	Insurance: 0.11% of CIF

Table 15-7
Processing and G&A Costs

Cost Category	Cost $/tonne ore
Direct Oxide Processing Cost Estimate:	4.28
Indirect Oxide Processing Cost Estimate:	2.96
Estimate of Oxide Processing Cost:	7.24
G&A Cost for Oxide	7.92
Direct Sulfide Processing Cost Estimate:	9.18
Indirect Sulfide Processing Cost Estimate:	1.71
Estimate of Sulfide Processing Cost:	10.89
G&A Cost for Sulfide	3.66

15.5 Mineral Reserve Estimate

The Mineral Reserve is the sum of the Proven and Probable material that is scheduled to be processed in the mine plan that is presented in Section 16. The cutoff grade for material sent to the crusher is $15.16/tonne Net of Smelter for Oxides and $14.55/tonne Net of Smelter for Sulfides. These are "internal" cutoff grades because they correspond to the sum of the processing and G&A costs. The estimate of processing + G&A costs for oxides was $15.16/tonne and the estimate of processing + G&A costs for sulfides was $14.55/tonne. The Mineral Reserves are summarized in Table 15-8.

Just prior to completion of this study, the client added more personnel to the project G&A staffing and the process team amended the process operating costs. The net result was to add about $1.80 /tonne of oxide ore for G&A + processing costs and about $4.80 /tonne of sulfide ore for G&A + processing costs. IMC checked the impact of these cost increases on the mineral reserve that was based on the cutoffs of the previous paragraphs. The cost increases could potentially reduce the stated mineral reserve ore tonnage by as much as 3.2%.

IMC and John Marek hold the opinion that 3.2% is not a material change and that the stated reserves are robust to the higher estimated costs that result from the work that is summarized in this report. Financial analysis presented in Section 22 applies higher metal prices than were selected by the QP for mine design. Those higher metal prices would more than offset the 3.2% loss of reserve to the positive.

The qualified person for the mineral reserve is John Marek of Independent Mining Consultants, Inc. The mineral reserve could change as more drilling and engineering is completed. Metal prices could materially change the mineral reserve in a positive or negative way. Changes to project operating costs could also impact the statement of mineral reserves.

The payable copper and zinc metal within the oxide zone are not presented on the statement of mineral reserves because there is no process planned to produce those metals in the oxide zone. The grades of copper and zinc are shown because their presence has an impact on the design of the oxide process plant and oxide processing costs.

Table 15-8
Gediktepe Mineral Reserve, 1 June 2016

Classification	Cutoff NSR $/Tonne	Oxide Mineral Reserves					Payable Metal			
		Oxide Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	15.16	1,456	2.98	74.7	0.12	0.17	118.0	1,541.4		
Probable	15.16	1,767	2.93	80.3	0.18	0.35	133.6	2,010.9		
Proven+Probable	15.16	3,223	2.95	77.7	0.15	0.27	251.6	3,552.3		

Classification	Cutoff NSR $/Tonne	Sulfide Mineral Reserves					Payable Metal			
		Sulfide Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	14.55	10,425	0.84	31.0	1.04	2.05	64.3	1,924.6	160.2	326.6
Probable	14.55	11,267	1.00	39.3	0.93	2.63	83.4	2,724.8	154.6	452.6
Proven+Probable	14.55	21,692	0.93	35.3	0.99	2.35	147.7	4,649.4	314.8	779.2

Classification	Cutoff NSR $/Tonne	TOTAL MINERAL RESERVES					Payable Metal			
		Total Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	15.16/14.55	11,881	1.11	36.3	0.93	1.82	182.3	3,466.0	160.2	326.6
Probable	15.16/14.55	13,034	1.26	44.9	0.83	2.32	217.0	4,735.6	154.6	452.6
Proven+Probable	15.16/14.55	24,915	1.19	40.8	0.88	2.08	399.3	8,201.7	314.8	779.2

Notes:

Mineral Reserve Based on Metal Prices of:
$1,000/oz Gold, $15.00/oz Silver, $2.50/lb Copper, $1.00/lb Zinc
Payable Metal is not shown for copper and zinc in the oxide zone because there is no plan to recover copper or zinc from the oxide zone. Their grades are shown because copper and zinc have an impact on the design of the oxide process and oxide process costs.
The Qualifed Person for the Mineral Reserve is John Marek, RM-SME
Pit slope angles are 48 degrees in fresh rock and 42 degrees in weathered rock
Ktonnes are 1000 metric tonnes
Mlbs are millions of pounds of copper and zinc metal
Kozs are 1000 troy ounces of gold and silver.

16.0 MINING METHODS

The Gediktepe deposit will be mined by conventional open pit hard rock mining methods. Polimetal currently plans to utilize a contract mining company to move the ore and waste from the mine.

Compared with typical mining practices in North America, Turkish contractors generally utilize small back hoe loading units with relatively small haul trucks. The mine geometries have been designed with the assumption that mining will be completed by a Turkish contractor with 3-4m^3 backhoes and 35-40 tonne trucks.

The mineral reserve is the total of all measured and indicated (proven and probable) material that is planned for processing within the mine plan. The mineral reserve at Gediktepe is comprised of 2 ore types: oxides and sulfides.

Following a three month ramp up period that produces ore at an average of 2,200 tonnes per day, the Gediktepe PFS mine plan produces oxide mineralization to a heap leach facility at a rate of 3,000 tons per day for a little over 3 years. After that period, the minor oxide material that is incurred during sulfide mining will be sent to the waste storage facility.

Sulfide process feed starts during year 3 at a throughput rate of 4,500 tons per day. Sulfide ores that are incurred in year 1 are assumed to be waste while sulfide ore mined in year 2 is stockpiled to be fed to the crusher in year 3. In year 4, the sulfide throughput increases to 6,500 tons per day as the oxide material is exhausted.

The crushing circuit is sufficiently large that both oxide and sulfide feed material in year 3 can be crushed through the same circuit on a short term campaign basis. A tripper is planned downstream of crushing to send oxide material to the agglomerating drum and on to the heap leach pad by conveyor or send sulfide feed to the grinding circuit and flotation.

The mine production schedule is summarized on Table 16-1. Total material ramps up to 18,500,000 tonnes per year inclusive of both ore and waste (53,857 tpd). The mine and plant are assumed to operate 350 days per year. A graphical representation of the mine schedule material movements is provided in Figure 16-1.

Table 16-2 illustrates the process plant feed and recovered metal that results from the execution of the PFS mine plan and schedule on Table 16-1. With the exception of the oxide ore mined in pre-production and the sulfide ore mined in year 2, ore is planned to be processed in the same year that it is mined.

Table 16-1
Gediktepe PFS Mine Production Schedule

Mined Material																					
	CUTOFF	Oxide Mined Material								CUTOFF	Sulfide Mined Material										
Years	NSR $/tonne	ORE ktonnes	NSR $/tonne	Gold gm/t	Silver gm/t	Copper %	Zinc %	Mercury ppm	Arsenic ppm	NSR $/tonne	ORE ktonnes	NSR $/tonne	Gold gm/t	Silver gm/t	Copper %	Zinc %	Mercury ppm	Arsenic ppm	WASTE ktonnes	TOTAL ktonnes
PreProd	$15.16	92	$34.63	1.25	32.3	0.37	0.86	2.7	1,486										257	349
Y 1	$15.16	886	$68.17	2.15	68.4	0.22	0.50	5.6	1,091	$14.55	138	$51.68	0.81	26.9	0.76	1.99	2.1	551	4,740	5,764
Y 2	$15.16	1,048	$117.65	3.73	85.9	0.13	0.21	7.5	1,566	$14.55	379	$49.92	0.77	28.2	0.79	1.72	1.9	632	9,068	10,495
Y 3	$15.16	1,048	$94.92	2.99	76.7	0.10	0.10	6.1	1,420	$14.55	1,193	$65.40	0.90	34.5	1.25	1.66	2.3	575	9,317	11,558
Y 4	$15.16	149	$104.18	3.05	111.1	0.14	0.16	6.6	1,409	$14.55	2,275	$78.33	0.95	30.8	1.67	1.64	1.8	656	10,576	13,000
Y 5	$15.16	71	$55.61	1.92	36.1	0.10	0.07	4.3	1,770	$14.55	2,275	$80.68	0.97	38.3	1.42	2.62	2.1	681	16,154	18,500
Y 6	$15.16	14	$67.98	2.40	32.3	0.09	0.12	4.2	1,415	$14.55	2,275	$63.44	0.95	40.0	0.77	2.95	3.3	789	16,211	18,500
Y 7	$15.16									$14.55	2,275	$58.15	0.87	36.1	0.75	2.55	2.4	556	16,225	18,500
Y 8	$15.16									$14.55	2,275	$67.46	1.18	44.8	0.81	3.00	2.8	625	16,225	18,500
Y 9	$15.16									$14.55	2,275	$65.73	1.06	42.9	0.81	2.93	2.5	582	15,090	17,365
Y 10	$15.16	20	$27.38	0.98	32.1	0.20	0.07	1.9	706	$14.55	2,275	$61.62	1.00	31.6	0.99	2.05	2.1	767	8,308	10,603
Y 11										$14.55	2,275	$48.68	0.72	27.9	0.73	1.84	1.4	634	3,756	6,031
Y 12										$14.55	1,920	$51.37	0.65	25.6	0.79	2.00	1.2	597	1,479	3,399
Y 13																				
Y 14																				
Y 15																				
TOTAL	$15.16	3,328	$92.34	2.92	76.4	0.15	0.26	6.3	1,383	$14.55	21,830	$63.90	0.92	35.3	0.98	2.35	2.2	650	127,406	152,564

Notes:
Tonnages in red are assumed to be waste material.
Oxide ore mined in pre-production is fed to the crusher in year 1.
Sulfide ore mined in year 2 is fed to the crusher in year 3.

Table 16-2
Gediktepe PFS Crusher Feed Schedule
Oxide Ore to Heap and Sulfide Ore to Mill

	Heap Leach Material												Mill Feed Material															
	Oxide Feed Material								Contained Metal		Payable Metal		Sulfide Feed Material								Contained Metal				Sulfide Payable Metal			
Years	ORE	NSR	Gold	Silver	Copper	Zinc	Mercury	Arsenic	Gold	Silver	Equation	44%	ORE	NSR	Gold	Silver	Copper	Zinc	Mercury	Arsenic	Gold	Silver	Copper	Zinc	23%	19%	67%	69%
	ktonnes	$/tonne	gm/t	gm/t	%	%	ppm	ppm	Ozx1000	Ozx1000	Au Kozs	Ag Kozs	ktonnes	$/tonne	gm/t	gm/t	%	%	ppm	ppm	Ozx1000	Ozx1000	Lbs x 1000	Lbs x 1000	Au Kozs	Ag Kozs	Cu Mlbs	Zn Mlbs
PreProd																												
Y 1	978	$65.01	2.06	65.0	0.24	0.53	5.3	1,129	64.8	2,045.1	50.8	901.9																
Y 2	1,048	$117.65	3.73	85.9	0.13	0.21	7.5	1,566	125.8	2,893.6	105.3	1,276.1																
Y 3	1,048	$94.92	2.99	76.7	0.10	0.10	6.1	1,420	100.9	2,584.0	83.4	1,139.6	1,572	$61.66	0.87	33.0	1.14	1.68	2.2	589	43.9	1668.4	39,367	58,112	10.1	334.4	26.29	40.26
Y 4	149	$104.18	3.05	111.1	0.14	0.16	6.6	1,409	14.6	532.3	12.2	234.8	2,275	$78.33	0.95	30.8	1.67	1.64	1.8	656	69.3	2254.7	83,809	82,254	15.4	421.8	55.97	56.98
Y 5													2,275	$80.68	0.97	38.3	1.42	2.62	2.1	681	70.6	2803.8	71,220	131,406	15.5	509.4	47.56	91.03
Y 6													2,275	$63.44	0.95	40.0	0.77	2.95	3.3	789	69.5	2922.4	38,469	147,958	15.8	542.9	25.69	102.50
Y 7													2,275	$58.15	0.87	36.1	0.75	2.55	2.4	556	63.6	2638.0	37,616	127,896	14.6	497.8	25.12	88.60
Y 8													2,275	$67.46	1.18	44.8	0.81	3.00	2.8	625	85.9	3273.7	40,475	150,465	20.3	633.1	27.03	104.23
Y 9													2,275	$65.73	1.06	42.9	0.81	2.93	2.5	582	77.3	3141.4	40,575	146,954	17.9	603.5	27.10	101.80
Y 10													2,275	$61.62	1.00	31.6	0.99	2.05	2.1	767	72.8	2312.0	49,854	102,818	17.2	438.7	33.29	71.23
Y 11													2,275	$48.68	0.72	27.9	0.73	1.84	1.4	634	52.4	2044.3	36,463	92,285	12.1	389.4	24.35	63.93
Y 12													1,920	$51.37	0.65	25.6	0.79	2.00	1.2	597	39.8	1581.1	33,567	84,657	8.8	278.3	22.42	58.65
Y 15																												
Total	3,223	$93.66	2.95	77.7	0.15	0.27	6.3	1,378	306.2	8,055.1	251.6	3,552.3	21,692	$63.98	0.93	35.3	0.99	2.35	2.2	650	645.1	24,639.9	471,416	1,124,806	147.7	4,649.4	314.80	779.21

Sulfide Material Mined in Year 1 and Oxide Material Mined in Years 5-10 assumed to be waste

Sulfide Payable Recoveries include both Process Plant Recovery and Smelter Payable Estimates.

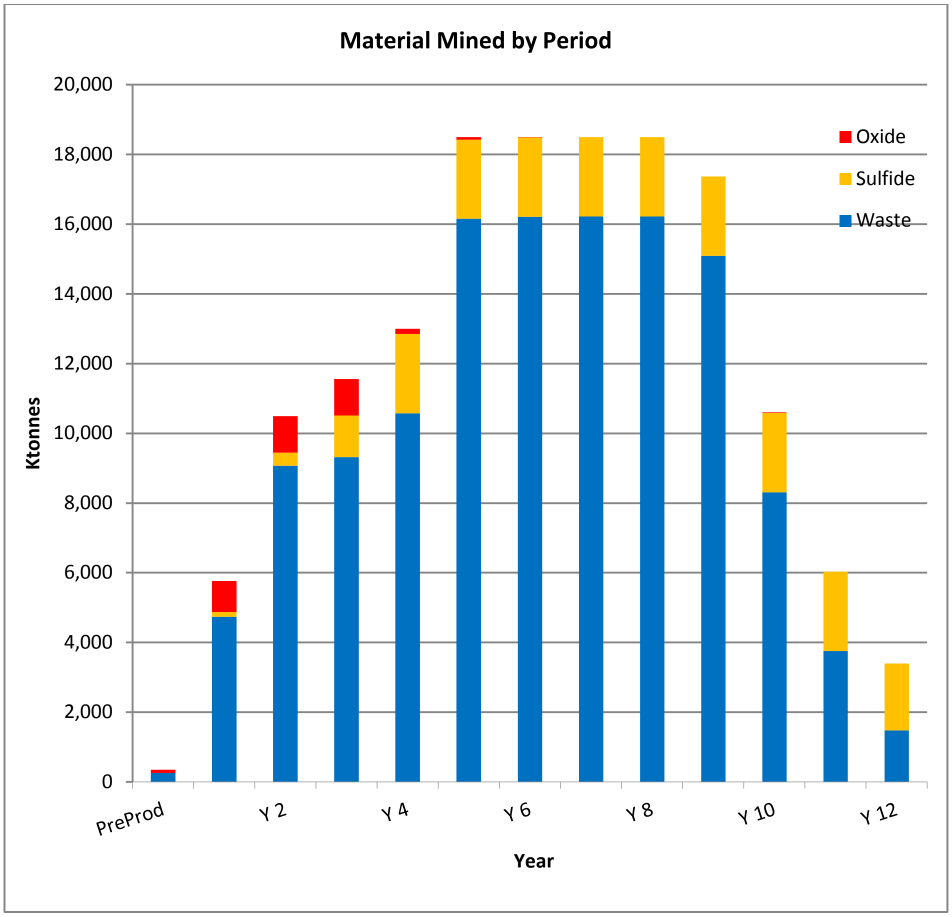


Figure 16-1
Graphical Representation of Gediktepe Mine Schedule

16.1 Floating Cones

The floating cone algorithm was used as a guide to determine the final pit geometry and intermediate phase design. The floating cone input parameters used to generate cones for pit design are provided in tables 15-1 through 15-3 in chapter 15. The design metal prices for the mine plan were: $1,000 /oz gold, $15 /oz silver, $1.00/lb zinc, and $2.50/lb copper.

The floating cone that was generated using metal prices at 80% of the base case prices above was used as a guide for the final pit design. This is explained in Section 15.2 of Chapter 15. Cones were also floated using metal price inputs between 40% and 80% of base case prices to provide an idea of how phase expansion should progress from highest value to lowest value pits.

16.2 Phase Designs

A total of seven phase or pushback designs were developed for the Gediktepe project. Phase designs are practical expansions of the mine excavation that incorporate haul road designs, operating room for equipment and all practical mining requirements.

Optimally, Phase 1 should represent the highest value zone of the deposit. Each subsequent expansion is a mineable width outward until the final pit design is reached in Phase 7 at Gediktepe. At any point in time, the mine schedule and mine plan is a combination of several phase designs until the ultimate pit geometry is achieved by mining out the final phase.

Phase designs incorporated the practical requirements of an operating mine. The design parameters for Gediktepe phase designs were:

Interramp Slope Angles	48 degrees in competent rock (From Fugro)
	42 degrees in weathered rock
Bench Height	5 meters
Standard Haul Road Width	12 meters
Maximum Haul Road Gradient	10%
Road width with Creek Diversion	18 meters.

The interramp slope angles were recommended by Fugro Sial during July of 2015. Their work included drilling 9 geotechnical boreholes, re-logging of 25 exploration holes, hydrology tests, fracture mapping and kinematic analysis of potential failure modes. The result of that work is the basis for the recommended 48 degree interramp angle in competent rock and 42 degree interramp angle in weathered rock.

Figures 16-2 through 16-10 illustrate the annual mine configuration that results from the seven phase designs being excavated according to the PFS mine plan.

16.3 Mine Production Schedule

The mine production schedule that is presented in Table 16-1 was based on the phase designs and the process plant feed rate for both oxide and sulfide material. Sufficient waste is moved during the mine life to assure continued release of the required process feed material.

The cutoff grades for the mine plan were based on early estimates of the process costs that were also applied to the floating cones. Since processing both sulfide ore and oxide ore produces a combination of economic metals, the cutoff grades are in terms of Net Smelter Return per tonne (NSR). Another way of understanding NSR is: NSR = Gross Income from Sales – off site processing charges.

In the case of the sulfide ores, NSR is truly Gross Income – smelting and transport charges. For oxide ore, the NSR value is the gross income from sales of the dore – refining and transport charges.

The design estimate of oxide process costs was $7.24/tonne of ore with a G&A cost of $7.92/tonne. Therefore, the internal cutoff for oxides was $15.16 NSR/tonne. For sulfide ore, the initial estimate of process costs was $10.89/tonne of ore with a G&A cost of $3.66/tonne. Consequently, the applied cutoff for sulfide mineralization was $14.55 NSR/tonne. The G&A cost for processing sulfides is estimated to be less than the G&A cost for processing oxides because the sulfides are processed at a greater throughput rate.

The inputs used to calculate the NSR values in the model on a block by block basis for use as a cutoff grade are identical to those used to define the mineral reserve. These inputs can be found on tables 15-5 through 15-6 in chapter 15.

No attempt was made to optimize cash flows by upward adjustment of cutoff grade at Gediktepe. There are abrupt grade changes between the high grade and low grade mineralization at Gediktepe. Any reasonable range of cutoff grades that might be applied would make little difference to process feed tonnage because the grade of the high grade material is generally well above the range of potential cutoff grades.

A 5m bench height was selected to match the expected equipment size and allow production of the narrow veins with backhoe excavators. To achieve the recommended 48 degree interramp slope angle, double benching will be utilized to result in a 10 meter face height with a catch bench width of 5.75 meters every 10 meters vertically.

The crusher location and plant location were established by Polimetal in order to accommodate project requirements while keeping the crusher and plant a sufficient distance from the pit to minimize the impact of blasting. A stockpile is planned adjacent to the primary crusher to accommodate blending of ore during sulfide processing and to stockpile oxide ore during the pre-production period.

Sulfide ore encountered during year 2 mining will be stockpiled uphill from the first phase of mining. It will be re-handled to the crusher in the first half of year 3 when sulfide ore processing begins. The stockpiled sulfide tonnage accounts for 1.4% of the project recovered copper. Consequently, no significant impact is expected on the project financials if recoveries are reduced due to oxidation while stockpiled for a year.

16.4 Waste Storage

The waste storage area is located immediately east of the pit. The area was selected after close communication between all project team members. Geotechnical guidance was provided by Fugro as a result of their site investigations.

Waste material will be placed in lifts from the bottom up. In this way some degree of compaction will be achieved to improve stability. Timber will be cleared as required and all organic material removed before stacking.

An overall slope angle of 2.5:1 or 21.8 degrees was applied due to the high seismic risk in the area. Haul roads up the face are 12m wide and flatten the overall dump slope.

Mine waste is required for the construction of the TSF dam. In years 1 through 8 of mining, 12.7 million tonnes of waste are required in the construction of the TSF dam. Only chlorite sericite schist and low sulfur quartz feldspar schist waste material have been allocated for TSF dam construction material because these waste types have been identified as non-acid generating material.

Late in the mine life, some waste material is stored back in the pit as illustrated on Figures 16-9 and 16-10. Those areas are complete and no further mining is planned making the area available for short hauls from the north end of the pit. Table 16-3 summarizes the mine waste destination by period.

Table 16-3
Mine Waste Destination by Period
(Ktonnes)

	Waste Dump	TSF			Pit Backfill	Total Waste
		Phase 1	Phase 2	Phase 3		
Period	ktonnes	ktonnes	ktonnes	ktonnes	ktonnes	ktonnes
Pre-Prod.	257					257
Year 1	1,419	3,459				4,878
Year 2	8,491	577				9,068
Year 3	8,317		1,000			9,317
Year 4	8,576		2,000			10,576
Year 5	14,850		1,375			16,225
Year 6	14,181			2,044		16,225
Year 7	13,987			2,238		16,225
Year 8	16,225					16,225
Year 9	7,000				8,090	15,090
Year 10					8,328	8,328
Year 11					3,756	3,756
Year 12					1,479	1,479
Total	93,303	4,036	4,375	4,282	21,653	127,649

16.5 Mine Staff Requirements

The mining contractor will be responsible for immediate supervision, equipment operation, and maintenance. However, Gediktepe will require staff to guide the contractor, plan the mine and maintain ore control over the project.

Polimetal has estimated the following mine department staff that will be required on the Gediktepe mine management team. This list reflects the mining department only and does not include the additional staff for the overall project or the process facility.

Table 16-4
Mine Department Staff

Personnel Title	# persons
Mine Manager	1
Mine Production Superintendent	1
Shift Engineers	4
Mine Planning Superintendent	1
Mine Planning Engineer	3
Blasting Engineer	1
Rock Mechanics Engineer	1
Map Engineer	1
Surveyor	1
Surveyor Helper	1
Mine Geology Manager	1
Geologist	2
Chief of Resource Geology	1
Resource Geologist	1
Geotechnical Engineer	1
Sampler	3
Grade Control Helper	2
SHE - Helper	3
Total Staff Personnel	29

There are additional staff members planned at Gediktepe that are not specifically assigned to the mine department. The additional overall project staff is described in the G&A subsection 21.1.3 of Section 21.0

16.6 Mine Ore Control Requirements

One of the challenges that will face the mine production group at Gediktepe is the assurance of careful ore control practices by the loading equipment. An assumption in the mining process is that the narrow "veins" that are shallow dipping can be selectively mined by further dividing the 5 meter bench into 2.5 meter bench increments when necessary to reduce dilution. In order to accomplish this level of control, a significant amount of sampling and assaying will be required along with the efforts of Polimetal mine staff to control the process.

At the very least, every blast hole near the mineralization must be assayed. In some cases, more than one assay will be required from a single blast hole. This will be accomplished by assaying the top half and the bottom half of the blast holes separately to distinguish the ore from the waste on 2.5 meter benches. For 5 meter benches where the top 2.5 meters and bottom 2.5 meters are not both ore or both waste, the bench will be mined in separate 2.5 meter passes. The logging geologist can generally select high grade material from the more disseminated material, but that is accomplished by the trained eye of the geologist. High grade and disseminated mineralization are both dark in color in the sulfide units and may be difficult for field personnel to distinguish.

During sulfide ore mining, a hand held Niton unit is expected to be used to provide additional guidance in splitting the bench into 2.5 meter benches when necessary. Sulfide ores contain high zinc grades with abrupt zinc grade changes at the boundary. The Niton is fairly reliable in making that determination. The absolute grade from the Niton is not important, but the units could be very helpful to distinguish ore from waste.

IMC has added $0.49/tonne of ore to the mining costs to reflect assay requirements of 2 samples per blast hole in ore benches. The mine staff includes geologists and samplers to maintain the ore control functions.

16.7 Mine Equipment Requirements

The Gediktepe deposit is planned to be mined by a Turkish contractor. No mining equipment was specified in the contractor's mining quote, but the cost breakdown accounted for the cost of drilling, loading, blasting, hauling and auxiliary costs. Additional contractor quotes were provided for topsoil stripping and hauling and road construction.

16.8 Mine Plan Drawings

Figures 16-2 through 16-10 illustrate the mine production schedule as shown on Table 16-1. The waste storage facilities are shown advancing in time just as the mine is shown being excavated.

Stacking of the HLP and the filling of tailings in TSF 1 are illustrated in an approximate manner. The impoundment structure for TSF 1 is shown over time as mine waste is delivered to the impoundment.

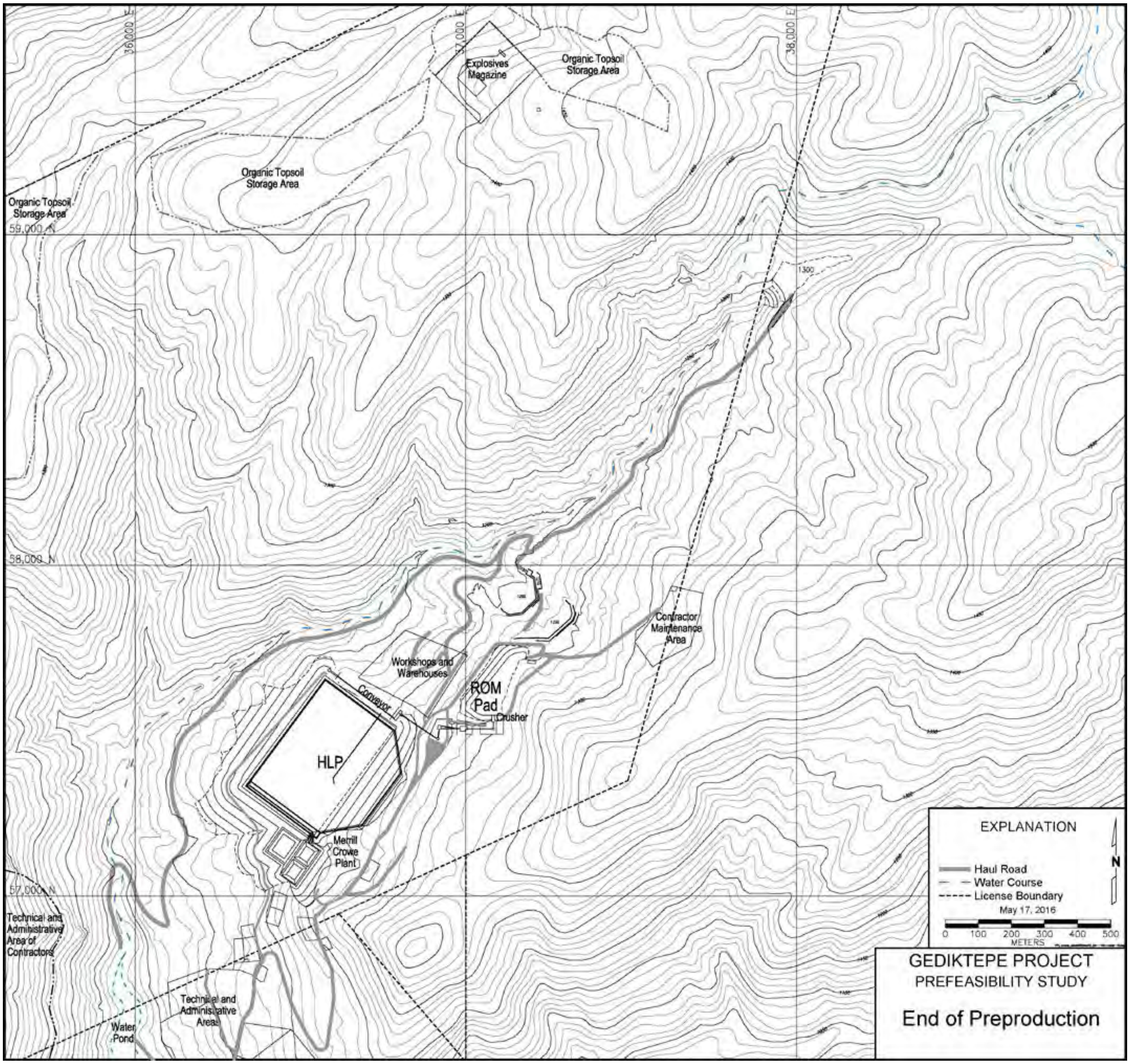


Figure 16-2
Annual Drawing End of Pre-Production

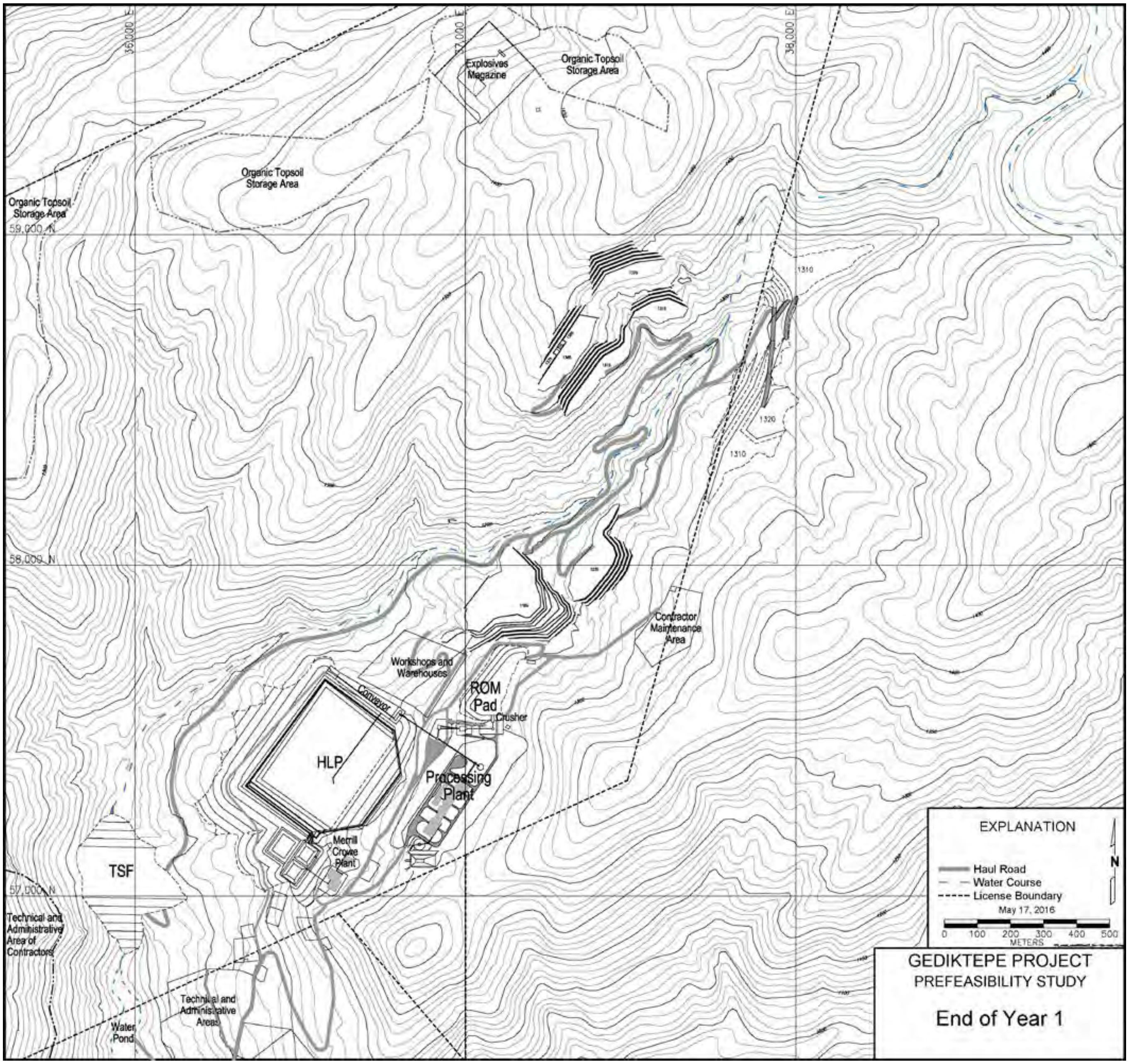


Figure 16-3
Annual Drawing End of Year 1

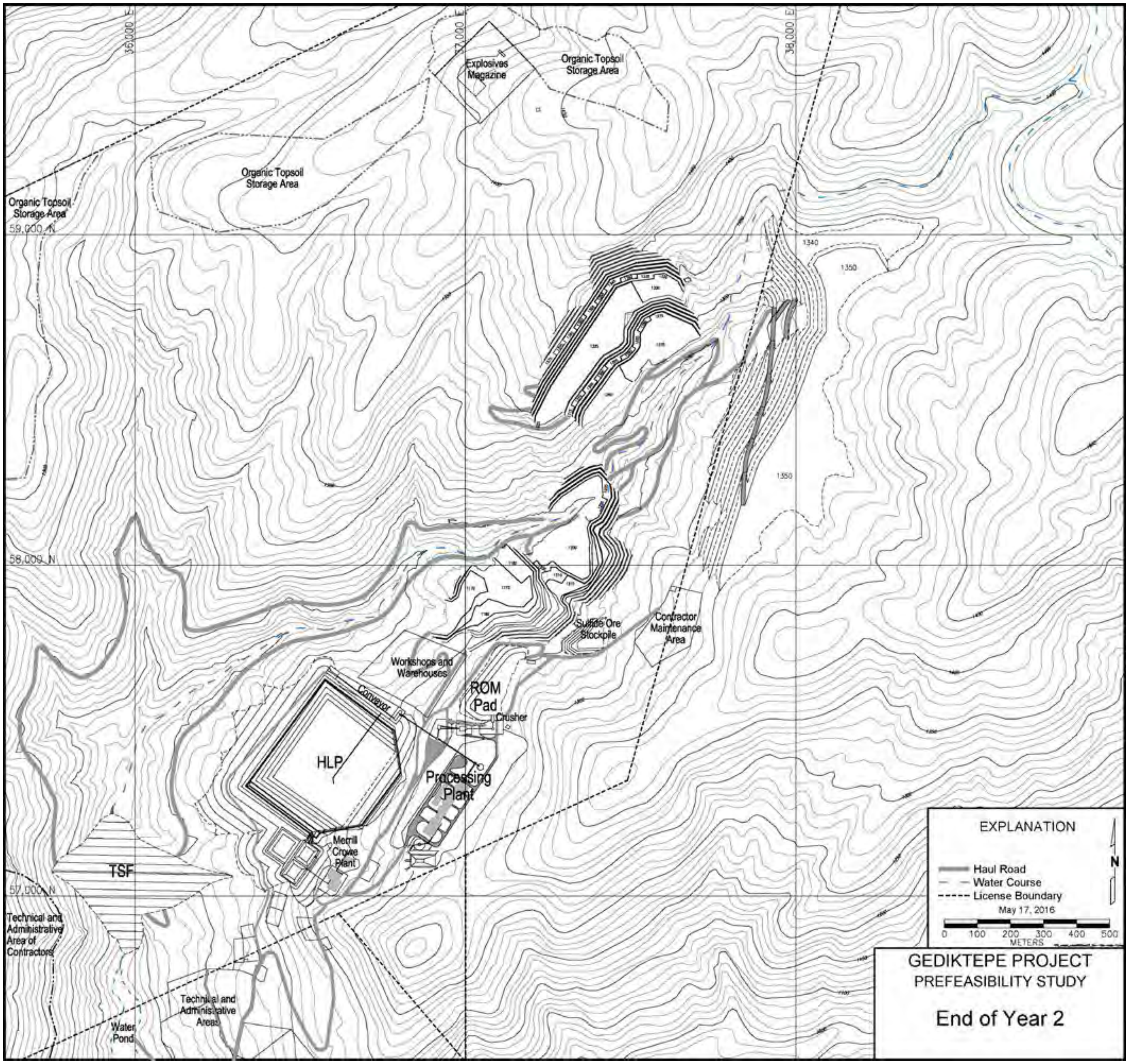


Figure 16-4
Annual Drawing End of Year 2

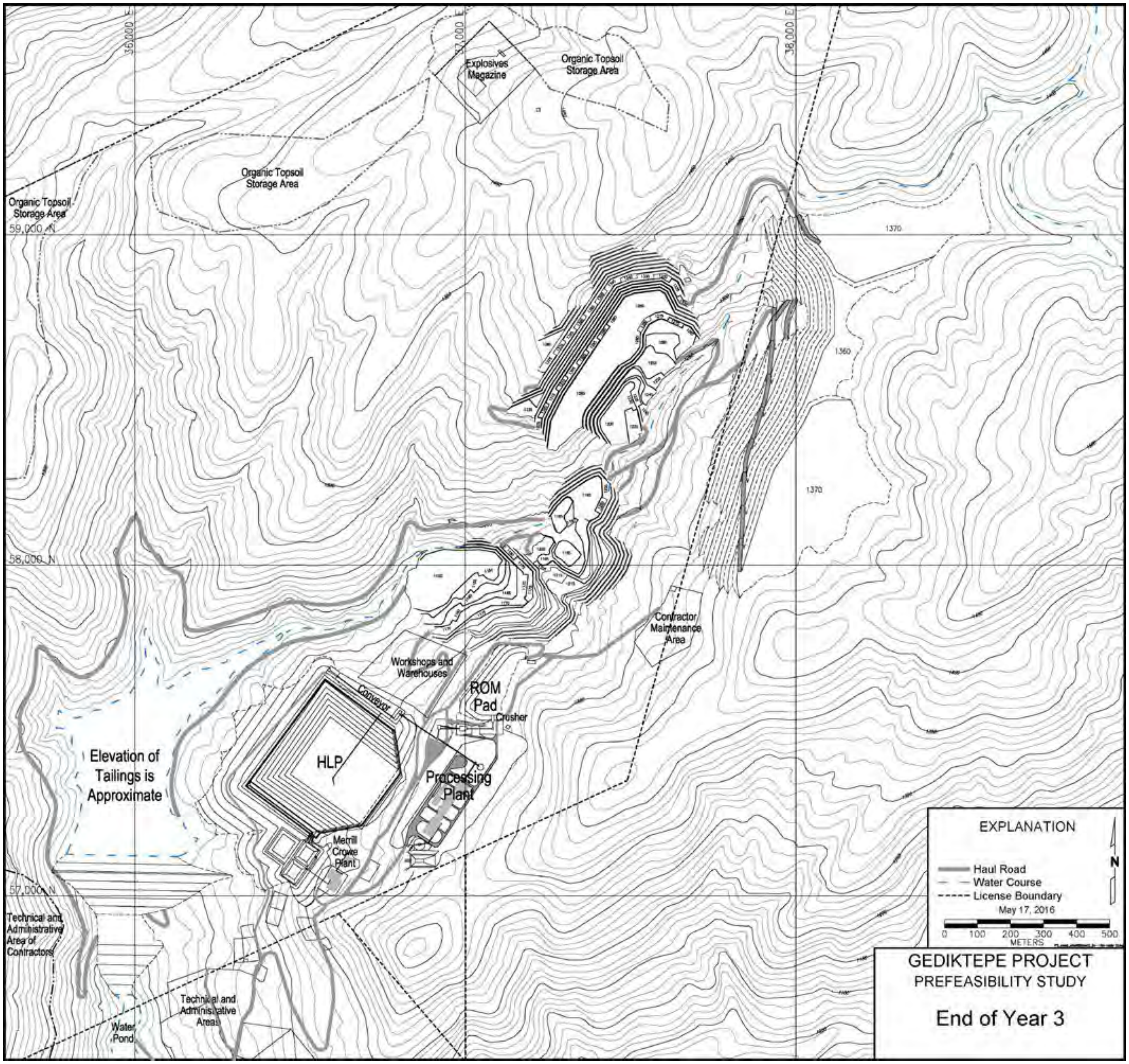


Figure 16-5
Annual Drawing End of Year 3

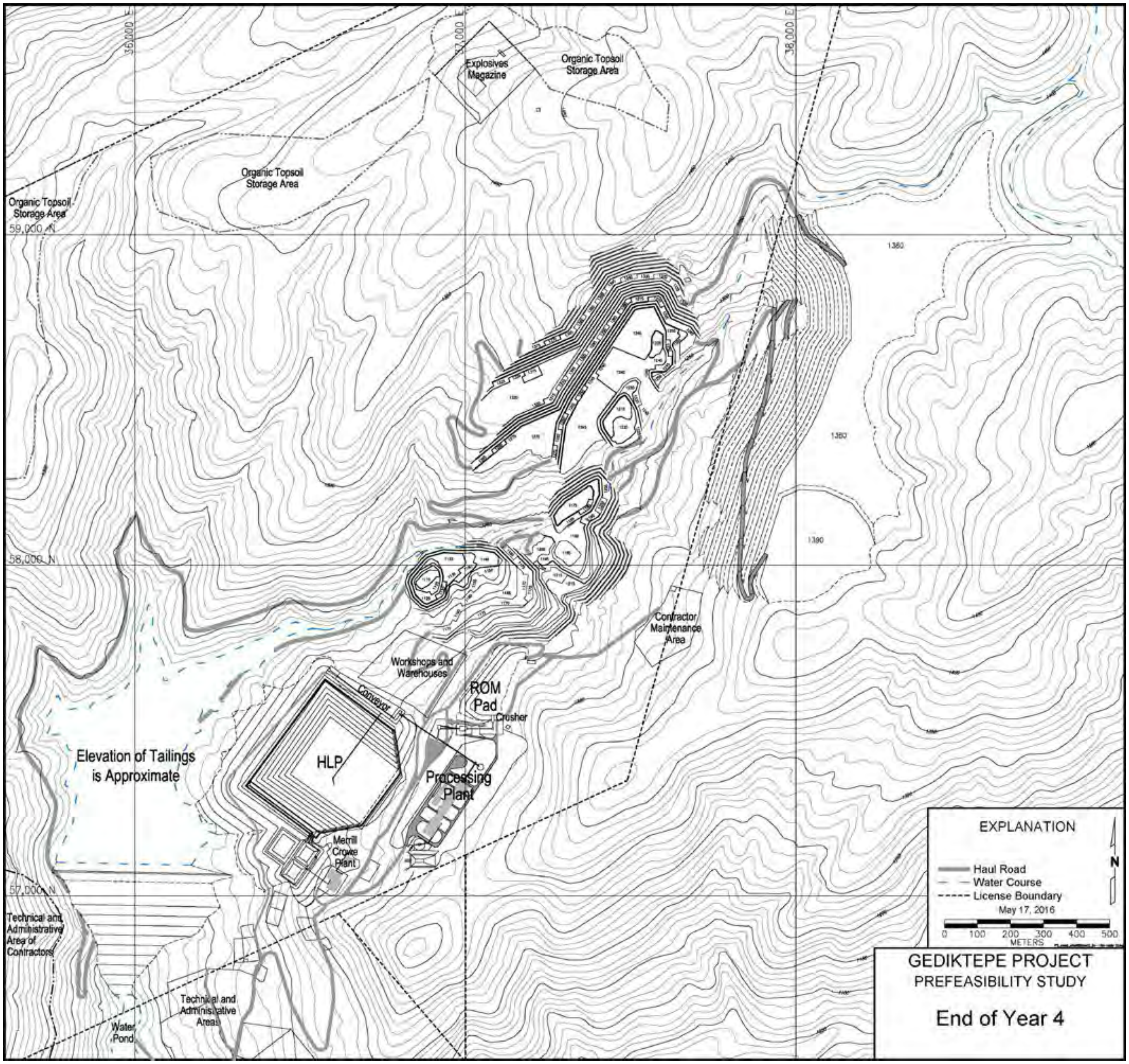


Figure 16-6
Annual Drawing End of Year 4

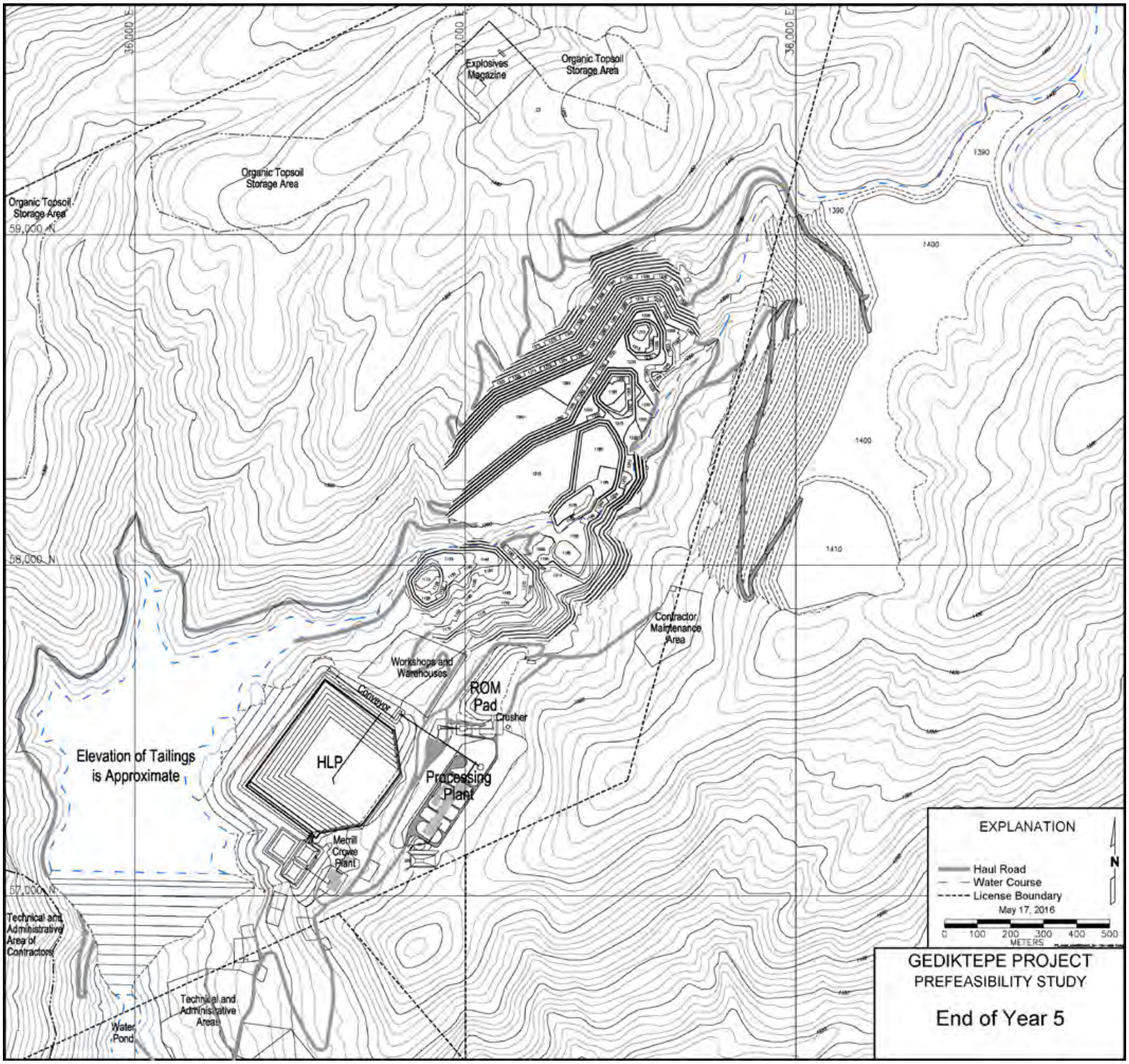


Figure 16-7
Annual Drawing End of Year 5

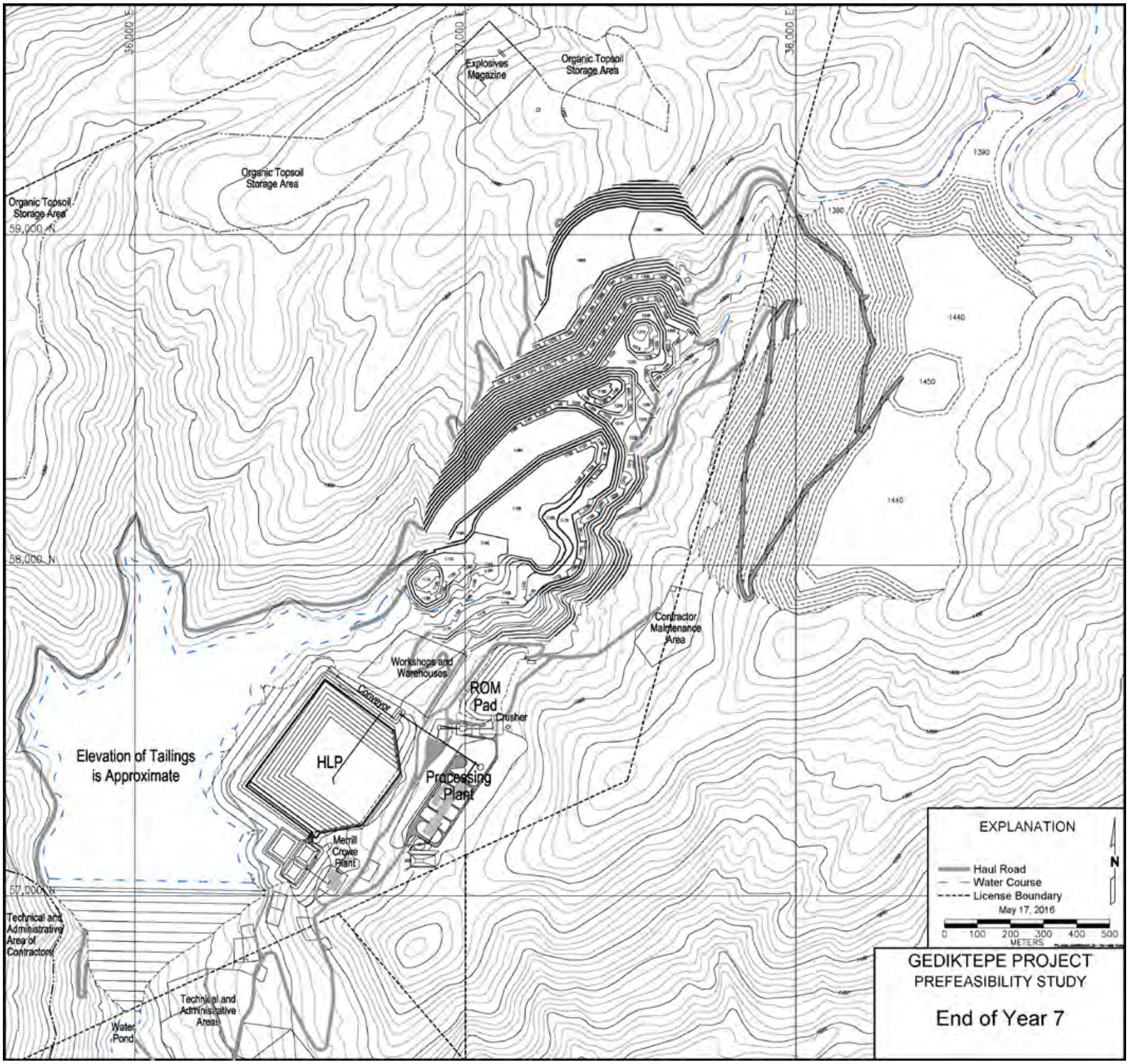


Figure 16-8
Annual Drawing End of Year 7

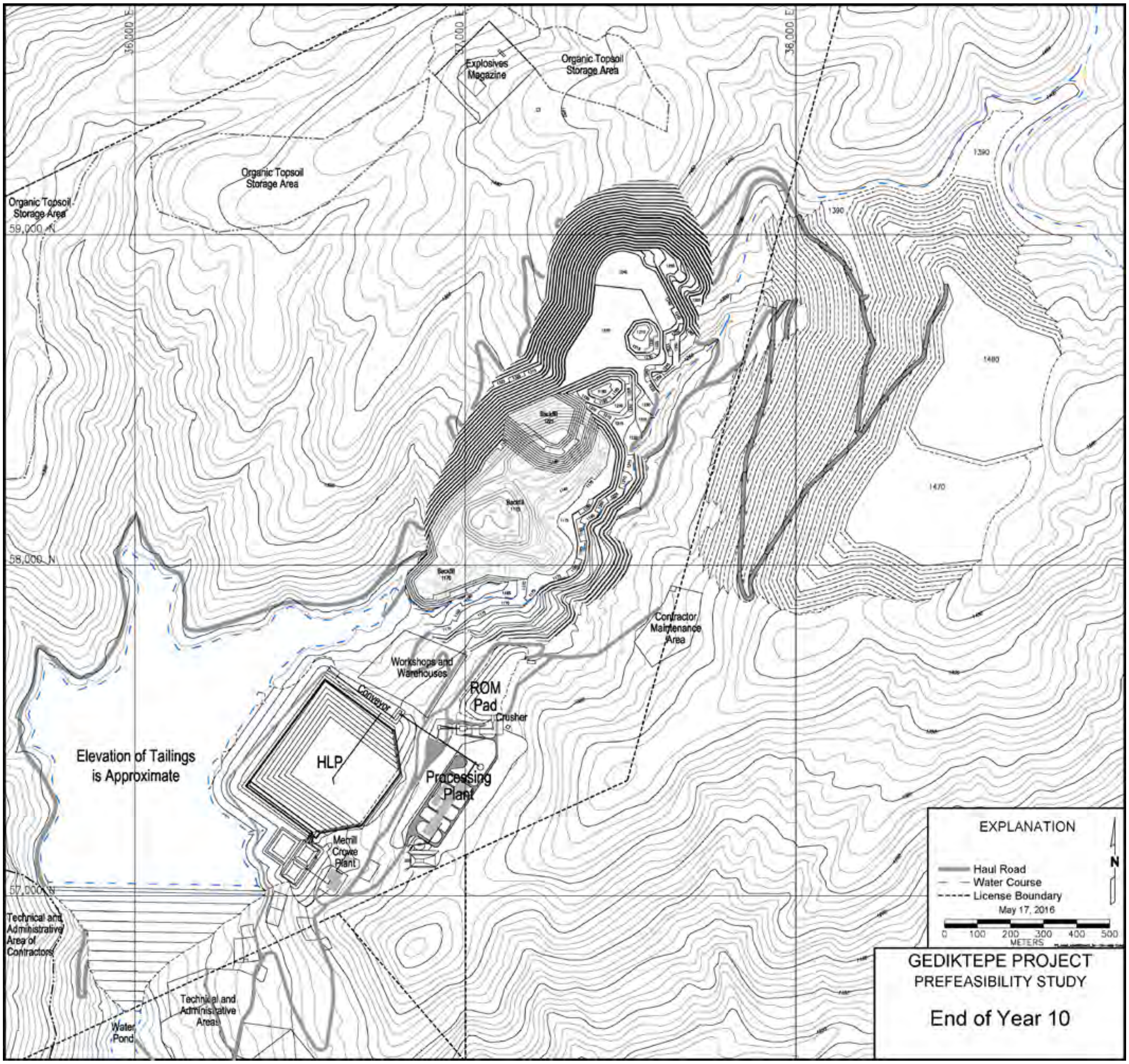


Figure 16-9
Annual Drawing End of Year 10

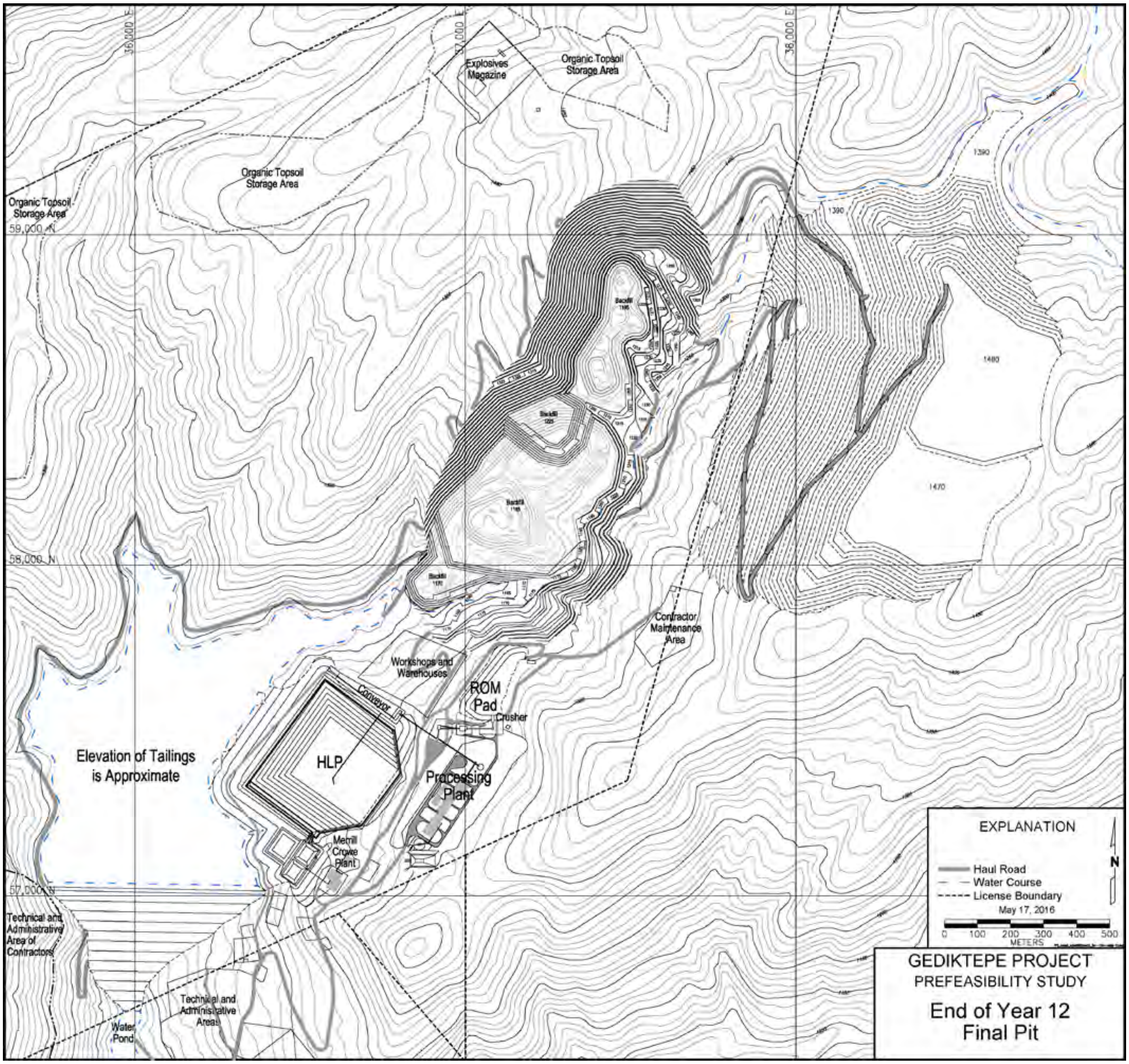


Figure 16-10
Annual Drawing End of Year 12

17.0 RECOVERY METHODS

The metallurgical test work indicated that the processing methods for the oxide and sulfide ore will be different. The oxide ore is amenable to heap leach as well as agitated leach with almost similar recoveries of precious metals. The process selected for treating oxide ores is heap leach. The sulfide ore will be floated to produce two concentrates, namely zinc concentrate and copper concentrate. Since sulfide ore processing will begin while oxide ore is still being processed, the crushing circuit is designed to treat both oxide and sulfide ores. The throughput capacity of the crushing circuit is 8,000 mtpd. The conceptual process flowsheets and associated recoveries are discussed in this section. The oxide plant is designed to process 3,000 mtpd and the sulfide plant is designed to process 6,500 mtpd.

17.1 Heap Leach Process Flowsheet for Oxide Ore

Based on the test work for the oxide ore, it became evident that the ore could be efficiently processed in a heap leach. This would result in lower capital cost while sacrificing very little in gold or silver recovery.

A simplified flow sheet for heap leach process is given in Figure 17-1. GR Engineering Services (GRE) completed basic processing plant engineering for the Heap Leach Oxide project producing an AACE class 3 level cost estimate. The design criteria for the plant (12256_Gediktepe PDC & Mass Balance – Oxide.xls) were provided by GRE services along with the process layout and flow sheets (Drawings: 12256-F-001 to 009; 12256-PI-001 to 011; 12256-L 001 to 010).

The run-of-mine ore will be crushed in three crushing stages to produce a product with P_{100} of 19 mm. A 19 mm opening screen is planned on the feed to the tertiary crusher to remove the finished product. The tertiary crusher product will be recycled back to the screen. The crushed ore will be discharged onto a conveyor which will convey the ore to the agglomerating drum.

The ore will be agglomerated with cement, lime and cyanide solution. The agglomerated ore will be discharged to a conveyor which will convey the ore to the leach pad. At the leach pad, grass hoppers and a stacker will be employed to stack the ore on the pad.

The ore will be leached with cyanide solution having a strength of 0.5 g/L and fed at a rate of 12.2 L/hr/m^2. The ore will be leached for 30 to 45 days and the pregnant solution collected in the pregnant solution pond. The pregnant solution will be processed using the Merrill Crowe process to recover gold and silver precipitate. In case the precipitate has high amounts of copper, it will be leached in a batch leach circuit with sulfuric acid and the residue filtered in a plate and frame filter press. The filtrate will be sent to the tailings pond in years 3 and 4 and to the clarifier backwash during years 1 and 2 when TSF construction is not complete. The residue containing gold and silver will be smelted in the furnace to produce a doré bar.

Based on the test work completed to date, the recoveries of precious metals projected for varying feed grades are given in Table 17-1.

Table 17-1
Projected Precious Metal Heap Leach Recoveries for Varying Feed Grades

Feed		**Recovery, %**	
g/t Au	**g/t Ag**	**Au**	**Ag**
3.0	79.0	84.5	47.6
0.5	17.1	62.0	37.0

Note: Discounted 3% for scale-up from laboratory to plant operation

Mine planning and financial modeling applied a variable recovery equation to the gold heap leach recovery and a fixed recovery of 45% to the silver heap leach recovery. The gold function closely models the results presented in Table 17-1 with an average recovery of 83% at average head grade of 2.95 gm/t. The fixed silver recovery of 45% is consistent with Table 17-1 and slightly conservative considering the 77 gm/t silver head grade to the leach pad.

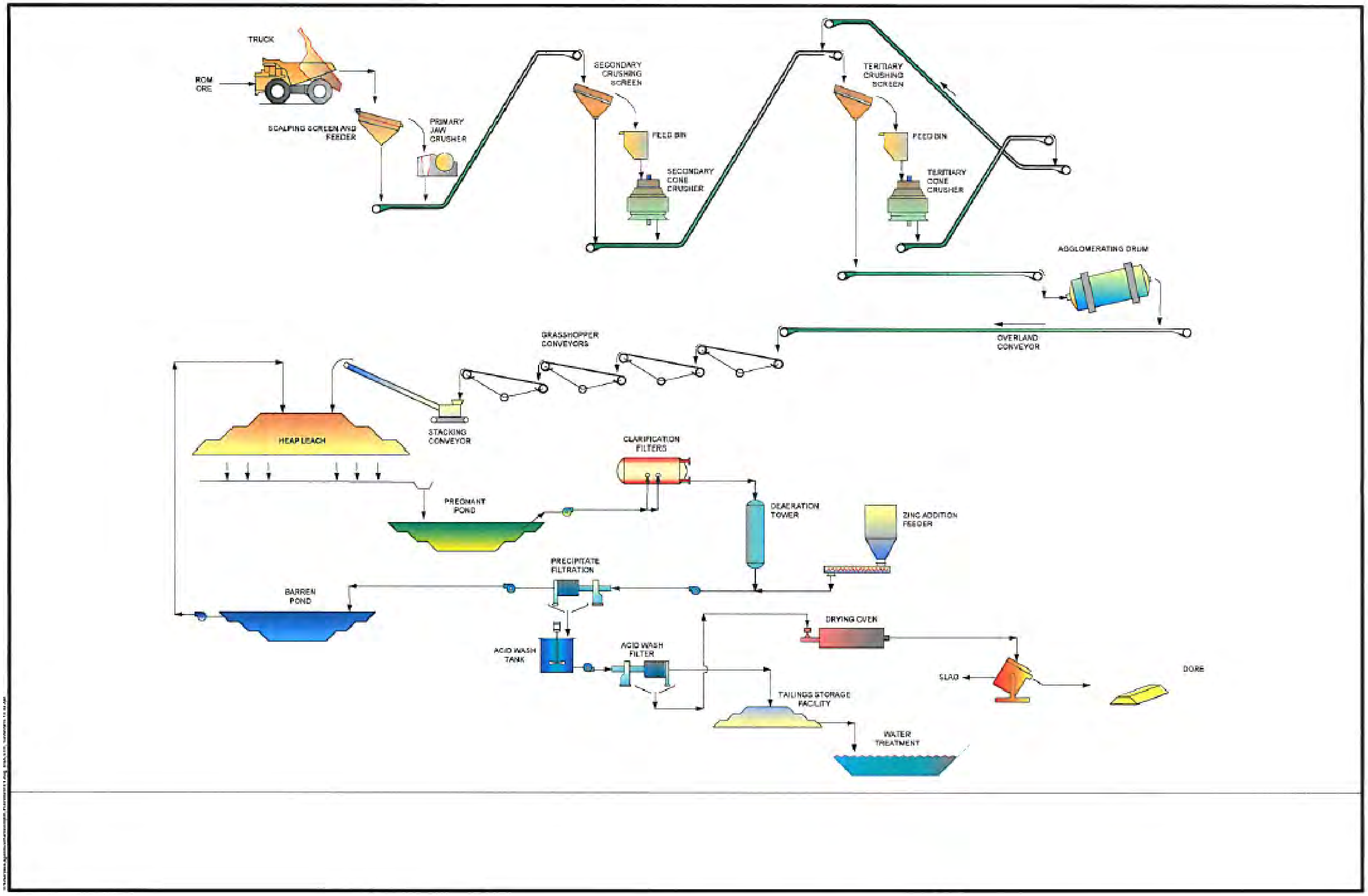


Figure 17-1
Flowsheet for Oxide Processing

17.2 Flotation Process Flowsheet for Sulfide Ore

A simplified version of the flow sheet for processing sulfide ore is provided on Figure 17-2. GR Engineering Services completed basic plant engineering for the Sulfide project producing an AACE class 4 level cost estimate. The design criteria for the plant (12256_Gediktepe PDC & Mass Balance – Sulphide Rev B.xls) were provided by GRE services along with the process layout and flow sheets (Drawings: 12256-F101 to 117; 12256 PI-101 to 118; 12256-L 001 to 010).

The plant was designed to process 6500 mtpd of massive sulfide ore. The run-of-mine ore will be processed in a three-stage crushing circuit to produce a product with a P_{80} of 12 mm. The crushing circuit will be the same as designed for the oxide ore heap leach process.

The fine crushed ore will be sent to the primary ball mill which will be operated in an open-circuit mode. Discharge of both primary and secondary ball mills are sent to the common sump. The ground slurry is pumped to the cyclones and cyclone underflow is sent to the secondary ball mill. The cyclone overflow, having a P_{80} of 45 microns, is the feed to the flotation circuit. The depressants, namely metabisulfite (MBS), zinc sulfate ($ZnSO_4$), sodium sulfide (Na_2S) and sodium silicate ($Na_2O(SiO_2)$) are added into the primary ball mill.

The cyclone overflow is fed to pre-flotation circuit which consists of two tank cells. MIBC is added to the feed box of the first cell. Talc is floated in these cells and sent to the tailings pond. The pre-flotation circuit tailings are sent to the copper conditioner where collector sodium aerofloat is added. It is also stage-added in the copper rougher flotation circuit.

The copper rougher concentrate is sent to the copper regrind cyclone sump where it is mixed with copper cleaner scavenger concentrate and copper cleaner 2 tailings. Depressants (MBS, $ZnSO_4$, Na_2S, $Na_2O(SiO_2)$ and NaCN) are added into the regrind mill. Copper regrind cyclone overflow, having a P_{80} of 20 micrometers, is sent to the copper cleaner conditioner where it is conditioned with the collector Aero 8761. Copper cleaner 1 concentrate is sent to second cleaner stage and the cleaner 1 tailing is sent to cleaner scavenger circuit. The concentrate is cleaned three more times in a counter-current circuit, namely, the tailings go back to the previous cleaner stage feed. The fourth cleaner concentrate is the final product and is sent to the thickener and thickener underflow to the filter.

The copper rougher tailings are sent to the zinc conditioner 1 where zinc is activated with copper sulfate ($CuSO_4$) and lime is added to adjust pH to 11.5. Collector Aero 7279 is added to zinc conditioner 2. Rougher zinc concentrate is floated in the rougher flotation circuit. The concentrate is sent to zinc regrind circuit. The zinc regrind and cleaner flotation circuit is identical to the copper circuit. The zinc cleaner 4 concentrate is sent to thickener and the thickener underflow to the concentrate filter.

The zinc rougher tailing and zinc cleaner scavenger tailings are combined and sent to tailing thickener. The thickener overflow is sent back as process water and the thickener underflow is pumped to the tailings pond. The process water will be treated in the water treatment plant before being returned to the process plant.

The overall material balance for the process based on locked cycle testing is shown on Table 17-2. The smelter penalty analyses for the copper and zinc concentrates are provided on Table 17-3. Additional optimization test work and engineering is ongoing at SGS UK. However, additional geo-metallurgical testing is recommended for final feasibility and advancement of the project.

Table 17-2
Summary of SGS Locked Cycle Test Results for Conceptual Sulfide Process Flowsheet without Process Water Recycle

Product	**Assays, % g/mt**							**% Distribution**							
	Cu	**Zn**	**Au**	**Ag**	**Pb**	**Fe**	**S**	**Wt.**	**Cu**	**Zn**	**Au**	**Ag**	**Pb**	**Fe**	**S**
Cu 4th Cl Conc.	30.95	2.49	4.34	108.5	1.30	24.79	31.55	2.0	69.2	2.8	17.2	12.3	11.0	1.7	2.2
Zn 4th Cl Conc.	2.71	54.3	3.00	143.5	2.69	5.94	34.26	2.6	8.1	81.5	15.7	21.5	30.3	0.5	3.2
Prefloat	1.09	1.61	0.51	26.0	0.30	16.59	15.31	0.7	0.8	0.6	0.7	1.0	0.9	0.4	0.4
Zn Cl Scav Tail	0.47	0.94	0.82	30.0	0.34	41.63	38.34	12.6	6.6	6.8	20.5	21.5	18.1	18.0	17.3
Ro Tail	0.16	0.18	0.28	9.3	0.11	28.31	26.21	82.1	15.2	8.4	45.9	43.7	39.8	79.4	76.9
Combined Tail	0.20	0.28	0.35	12.1	0.14	30.08	27.83	94.7	21.9	15.1	66.4	65.2	57.8	97.4	94.2
Cal. Head	0.89	1.76	0.50	17.6	0.23	29.25	27.99	100.0.	100.0.	100.0.	100.0	100.0	100.0	100.0	100.0

Note: Au and Ag are in g/mt

Table 17-3
Smelter Penalty Analysis of Copper and Zinc Final Concentrates

Element %	Concentrate	
	Copper	Zinc
Cu	30.39	2.84
Pb	1.23	2.77
Zn	2.78	51.52
As	0.037	0.132
Cd	0.005	0.0973
Ni	0.001	0.001
Co	<0.001	<0.001
Mn	<0.01	<0.01
Bi	0.0132	0.0259
Sb	0.0105	0.0204
Hg	0.0001	0.0017
Te	<0.0001	<0.0001
Se	0.0059	0.0045
SiO_2	4.05	2.04
Al_2O_3	0.86	<0.01
Fe	23.78	6.13
MgO	1.74	<0.01
Cr	0.01	<0.01
CaO	0.02	0.09
S	31.75	34.35
C	0.14	0.16
Cl	0.01	0.21
F	0.01	0.01
BaO	<0.01	<0.01
Ga	<0.01	<0.01
Ge	<0.01	<0.01
In	<0.01	<0.01
K2O	<0.01	<0.01
Mo	<0.001	<0.001
Na2O	<0.01	<0.01
Au, g/t	4.53	3.77
Ag, g/t	112.3	147.3

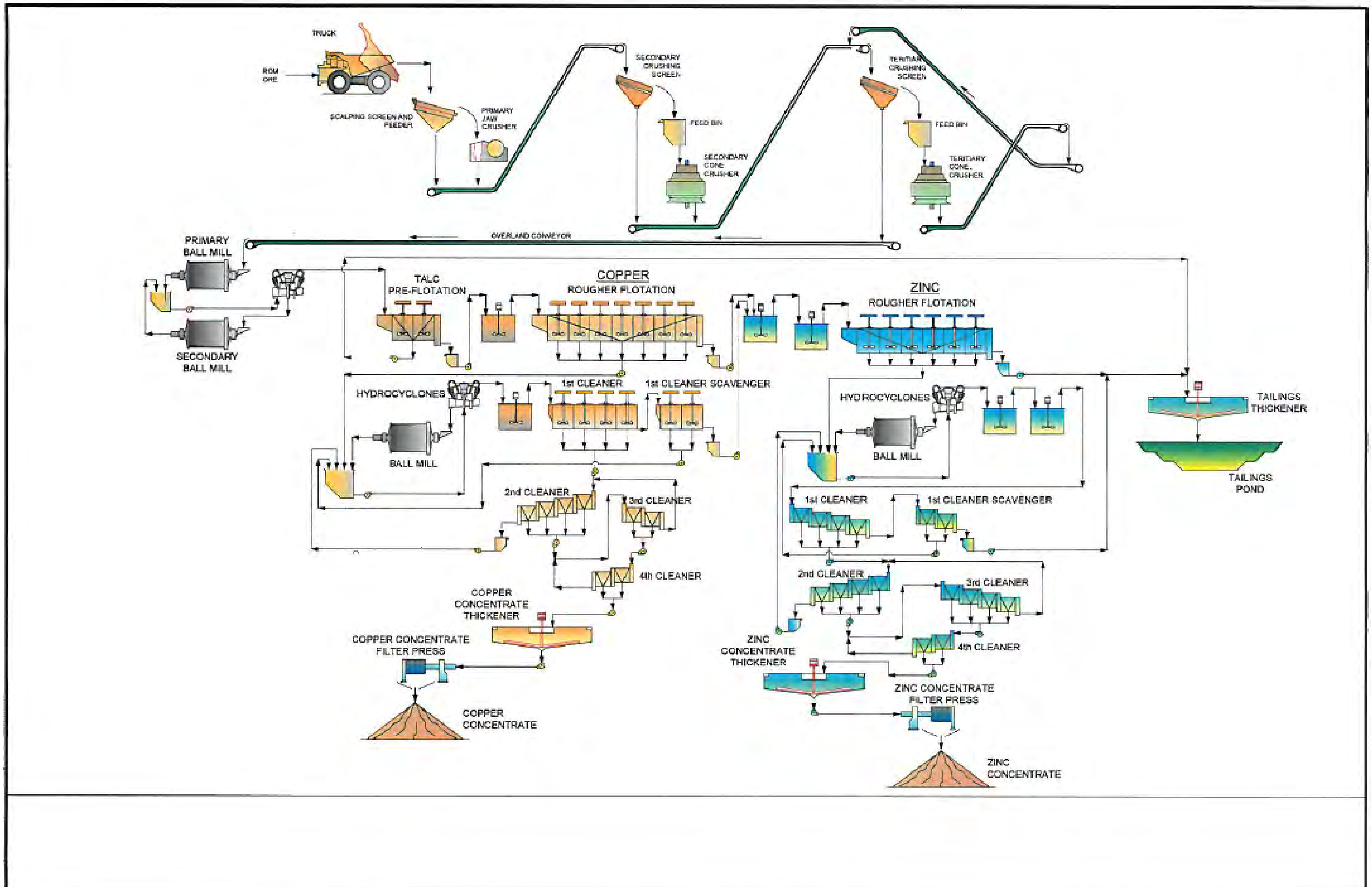


Figure 17-2
Flowsheet for Sulfide Processing

18.0 PROJECT INFRASTRUCTURE

The project will require the development of a number of infrastructure items in order to operate. The current approach to the project is a combination of oxide heap leaching followed by sulfide flotation. Therefore both heap leach facilities and tailing storage facilities will be required.

The following items will be discussed in the section:

1) Heap Leach Pad (HLP): The heap leach facility design was completed by SRK.
2) Tailing Storage Facility (TSF): A tailing storage facility design was completed by SRK.
3) Waste Storage Facility (WSF): The waste storage facility was designed by IMC and was discussed in Section 16 regarding the mine plan.
4) Water Supply: A water supply system will be required for the project. A quoted cost estimate for a freshwater pond and a water treatment plant were provided by Polimetal for inclusion into the PFS economic analysis.
5) Power Supply: A power supply system was planned by Polimetal that incorporates a new power line to the site. Cost estimates were developed by Polimetal working with the local Turkish power authorities. Those costs are included in the project PFS evaluation.
6) Access Road: An access road that bypasses the local village of Hacıömerderesi is designed and planned to be constructed during pre-production. Cost estimates were developed by Polimetal working with local engineering firms. Those costs are included in the project PFS evaluation.
7) Mine Site Buildings: A mine camp will be constructed close to the project site to house Polimetal staff because nearby villages are insufficient to house the workers that will be required at the mine site. Project technical and administrative buildings will also be required. Polimetal obtained quotes for pre-fabricated buildings that are included in the PFS economic evaluation.
8) Water Diversion Channels: SRK Turkey designed water diversion channels at the Mine Site for capturing and diverting both contact and non-contact water.

18.1 Heap Leach Pad

SRK developed a HLP design, sized to contain approximately 3.6 Mt of oxide ore with an ultimate height of 36m. An area for a dedicated HLP was identified in close proximity to the mine and process area to minimize handling of ore material, but was still outside of the proposed TSF footprint so as to have minimal impacts on storage capacity.

An underdrain system, comprised of a free draining granular material, will be installed in the area of any springs or seeps, to collect any groundwater. This underdrain will flow via gravity to a point outside of the HLP and solution pond limits, to protect groundwater and minimize any uplift pressures on the composite liner system.

For this study, SRK included a composite liner system that consists of the following (from bottom to top):

1) Prepared subgrade;
2) 500 mm thick low permeability Soil Liner compacted to achieve a maximum permeability of $1.0x10^{-9}$ meters per second (m/sec); and
3) 2.0 mm High Density Polyethylene (HDPE) geomembrane synthetic liner.

The solution collection system (SCS) for the HLP will consist of a solution collection piping network that is installed directly on the composite liner system and covered by an over liner material.

An external pond system was selected that consists of a pregnant, barren and storm pond to store high grade solution for processing, low grade solution to be applied onto the ore, and extra capacity for the design storm event, respectively. The pregnant and barren ponds will have a head on the liner system on a regular basis, and have been designed with a double liner system separated by a high permeability drainage layer that will allow solution that may leak from the upper primary geomembrane to be collected and removed via pumping, and minimize the head on the lower secondary geomembrane. The storm pond will contain solution on an infrequent basis and was designed with a composite liner system.

The calculated Factor of Safety (FOS) values for static stability analysis met the minimum FOS values, as identified in the design criteria. However, based on the material shear strengths and Peak Ground Accelerations assumed in the analysis, the FOS values for pseudo-static conditions were less than 1.0, indicating that there may be movement along the geomembrane/Soil Liner interface. Therefore, SRK has allowed for regrading and a stability buttress at closure in the cost estimate.

A diversion channel will be constructed around the HLP and solution ponds to divert non-impacted up gradient surface water around the facility. SRK delineated the up gradient watershed basin area and estimated the time of concentration for these basins. The diversion channel was sized for the peak flow rate (Q) for the 100-year, 24 hour rainfall event with no onsite detention or retention structures.

Figure 18-1 depicts the geometry of the HLP stacked with ore in the final configuration.

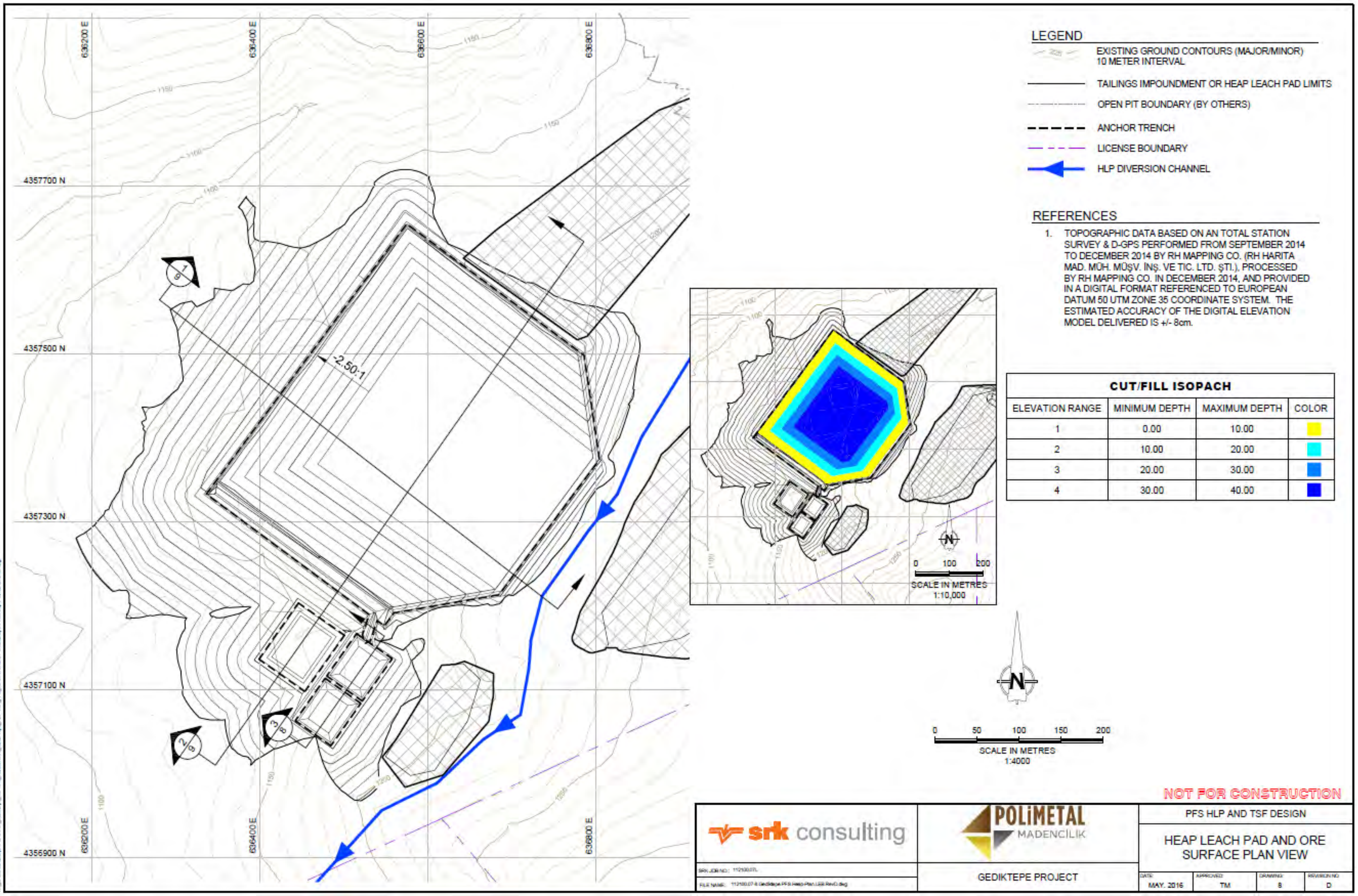


Figure 18-1
Ultimate Configuration of Oxide Heap Leach

18.2 Tailing Storage Facility

SRK designed a TSF, sized to contain approximately 22.0 Mt of mined sulfide material and be constructed in three phases. Considering the high seismic hazard potential of the project area, downstream embankment construction was selected in the PFS design to store the sulfide tailings with conventional tailings deposition.

An underdrain system, comprised of a free draining granular material, will be installed in the area of any springs or seeps, to collect any groundwater. This underdrain will flow via gravity to a point outside of the TSF, to protect groundwater and minimize any uplift pressures on the composite liner system.

The containment system for the TSF will include a liner system consisting of (from bottom to top):

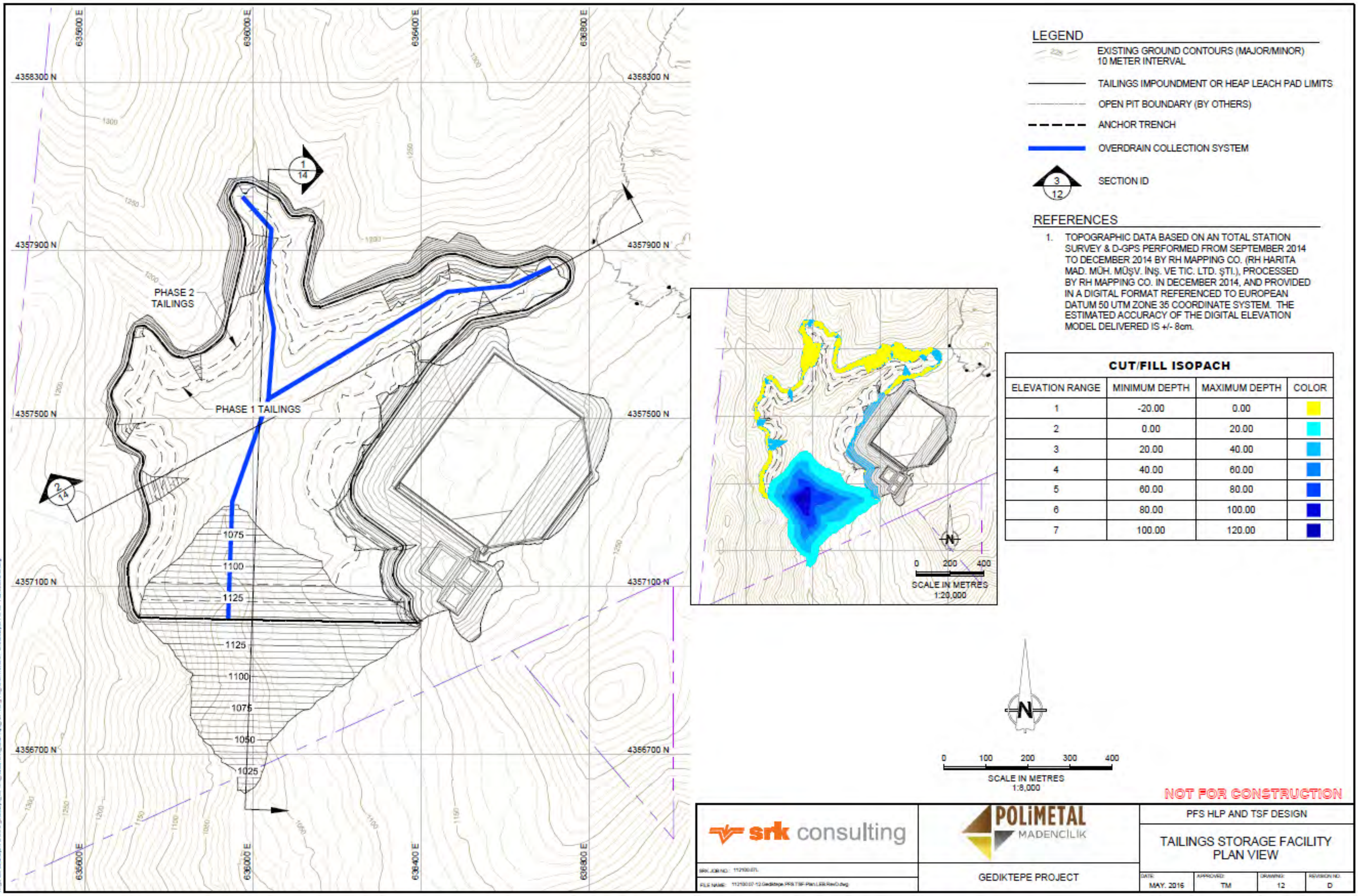
1) Prepared subgrade;
2) Geocomposite Clay Liner (GCL); and
3) 2.0 mm HDPE geomembrane synthetic liner.

An over drain collection system will be installed in select areas of the TSF to provide a preferential flow path for entrained moisture in the tailings that can be removed from internal low-point sumps to minimize head on the TSF liner during its operational life, and accelerate head elimination during closure.

Diversion channels will be constructed around each phase of the TSF to divert non-impacted up gradient surface water around the facility. SRK delineated the up gradient watershed basin area and estimated the time of concentration for these basins. The diversion channels were sized for the maximum peak flow rate for the 100-year, 24 hour rainfall event with no onsite detention or retention structures.

Based on the set of assumed parameters, the calculated FOS values for static and pseudo-static stability analysis met the minimum FOS values as identified in the design criteria.

Figure 18-2 depicts the configuration of the ultimate PFS tailing storage facility.



CUT/FILL ISOPACH

ELEVATION RANGE	MINIMUM DEPTH	MAXIMUM DEPTH	COLOR
1	-20.00	0.00	
2	0.00	20.00	
3	20.00	40.00	
4	40.00	60.00	
5	60.00	80.00	
6	80.00	100.00	
7	100.00	120.00	

Figure 18-2
Ultimate Configuration of Tailings Storage Facility

18.3 Waste Storage Design

The waste storage area is located immediately east of the pit in the same valley. Waste will be placed in lifts from the bottom up to provide some compaction and improve the waste storage facility stability. The storage facility will be constructed at a slope of 2.5:1 but the overall slope angle will be shallower because of access roads in the storage facility design. The waste storage area will hold waste that will not be used for the tailings storage facility dam construction or placed back into the pit as backfill.

18.4 Water Supply

A site specific water supply system has not been designed at this time. A site wide water management study is in progress for input to subsequent engineering work. IMC anticipates that the process makeup water will be sourced from wells, surface run off and the fresh water impoundment. Polimetal provided costs and conceptual designs of a waste water treatment plant and a freshwater pond. The fresh water pond is designed to be immediately south of the tailing storage facility.

18.5 Power Supply

Polimetal applied to General Management of TEDAS (Turkish Electricity Distribution Comp.) for a 10 MW power allocation permit for the mining operation in Gediktepe on 13 March 2014. TEDAS stated and approved that power would be supplied from the Dursunbey Substation and directed Polimetal to get in touch with Uludag Elektrik Dagitim A.S. (UEDAS, Uludag Power Distribution Company). UEDAS found Polimetal's application suitable to supply required power from Dursunbey Substation.

With the progress on metallurgical studies, defining processing equipment and considering possible future capacity increases, Polimetal increased the power requirement to 20 MW and studied possible power transmission line (PTL) routes. Based on a PTL route selection study, it was decided to use the current village PTL and upgrade it to supply power to the Gediktepe project and the villages on the PTL route.

This approach is considered as a social project for the stake holders of the project, and will be environmentally sound. It will eliminate the current power outages experienced in the villages by looping the power line. For that reason, a new application was submitted to UEDAS on 24 October 2014 to increase the power demand to 20 MW and to use & upgrade current village PTL route (19 km) with construction of additional new 14 km PTL. The PTL total length will be around 33 km.

Polimetal has completed the topographic survey and engineering of the planned PTL. The PTL project was approved by UEDAS and the government power authority. The expropriation file has been prepared and submitted to UEDAS to take a decision on expropriation. The forest permitting process has begun. Figure 18-3 shows the planned route for the power line from Dursunbey to Gediktepe.

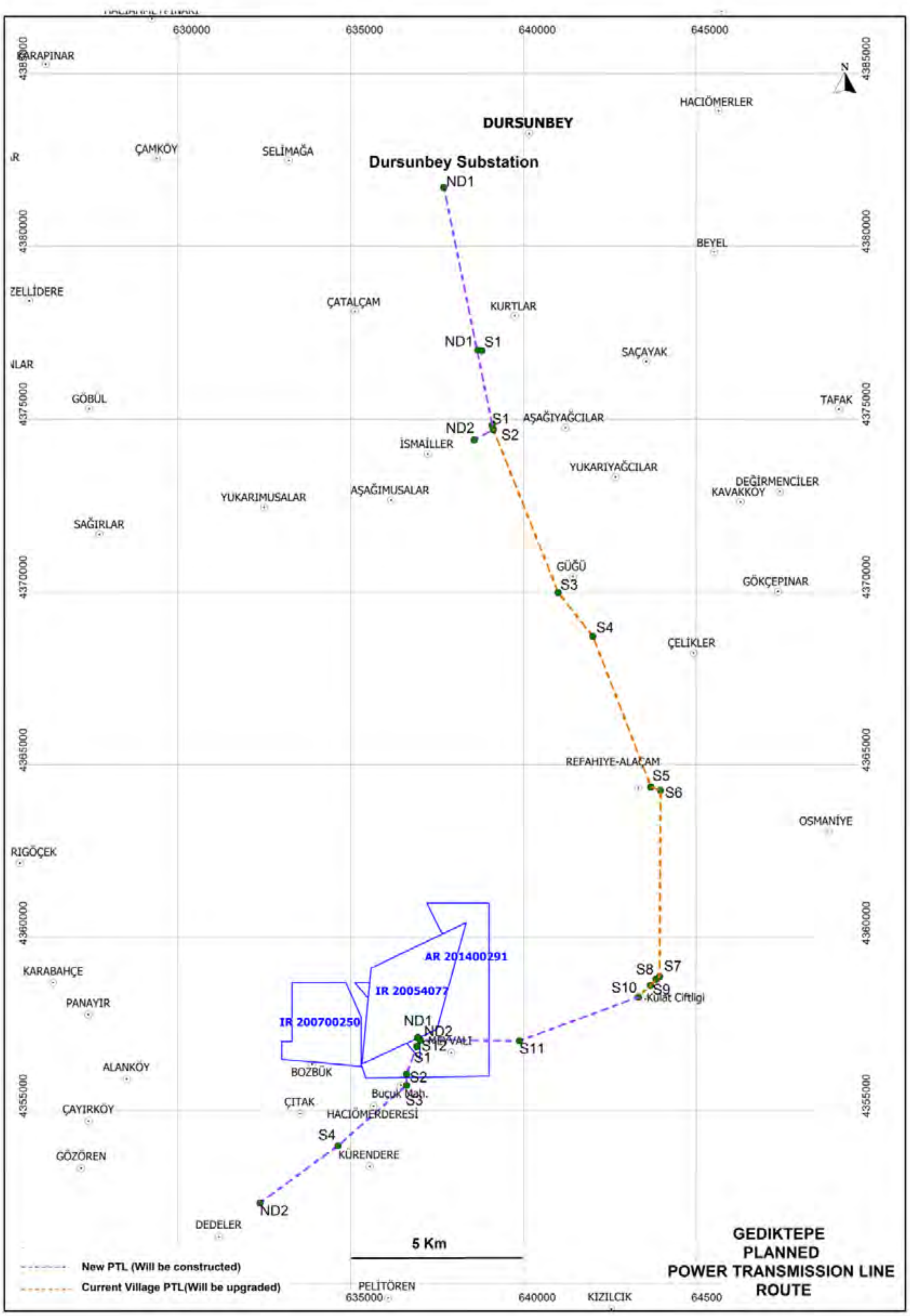


Figure 18-3
Proposed Power Transmission Line Route (PTL)
Truncated UTM Coordinates

18.6 Access Road Upgrade

The access road to the Gediktepe site that bypasses the town of Hacıömerderesi has been designed by a road engineering firm. Concentrate shipment and supply delivery will constitute a substantial demand on the road. Polimetal has developed an estimate of constructing the road using the General Directorate of Highways unit prices. The road construction and maintenance costs are incorporated into the economic analysis

18.7 Mine Site Buildings

Technical and Administrative Buildings:

Polimetal has planned the layout of the project site buildings which include: technical and administrative offices, a laboratory, warehouses, storage, first aid, security, and public relations facilities.

Camp Buildings:

Polimetal is intending to build a construction style camp for the mine workers. Those workers will live on site during the shift rotation and will be provided bus service to the nearest city (Bigadic/Balıkesir) on their days off. The camp area (around 41,000 m2 land), which is 6 km south of the project area, has already been purchased and designing of lodging buildings/facilities and temporary office buildings is in progress.

Polimetal received quoted estimates for the cost of erected pre-fabricated buildings that were included in the cost analysis for the project. Additional estimates for foundation work and furnishings were also included in the mine site buildings costs. Costs to operate the camp are included in the estimated project owner's costs.

18.8 Water Diversions

Water diversion channels have been designed by SRK Turkey and their construction costs have been included in the financial analysis. The designs include channels to catch contact water and send it to water treatment and also to divert non-impacted (non-contaminated) water around the project site to the fresh water pond that will be constructed immediately south of tailing storage facility. The location of the water diversion channels is shown in Figure 18-4.

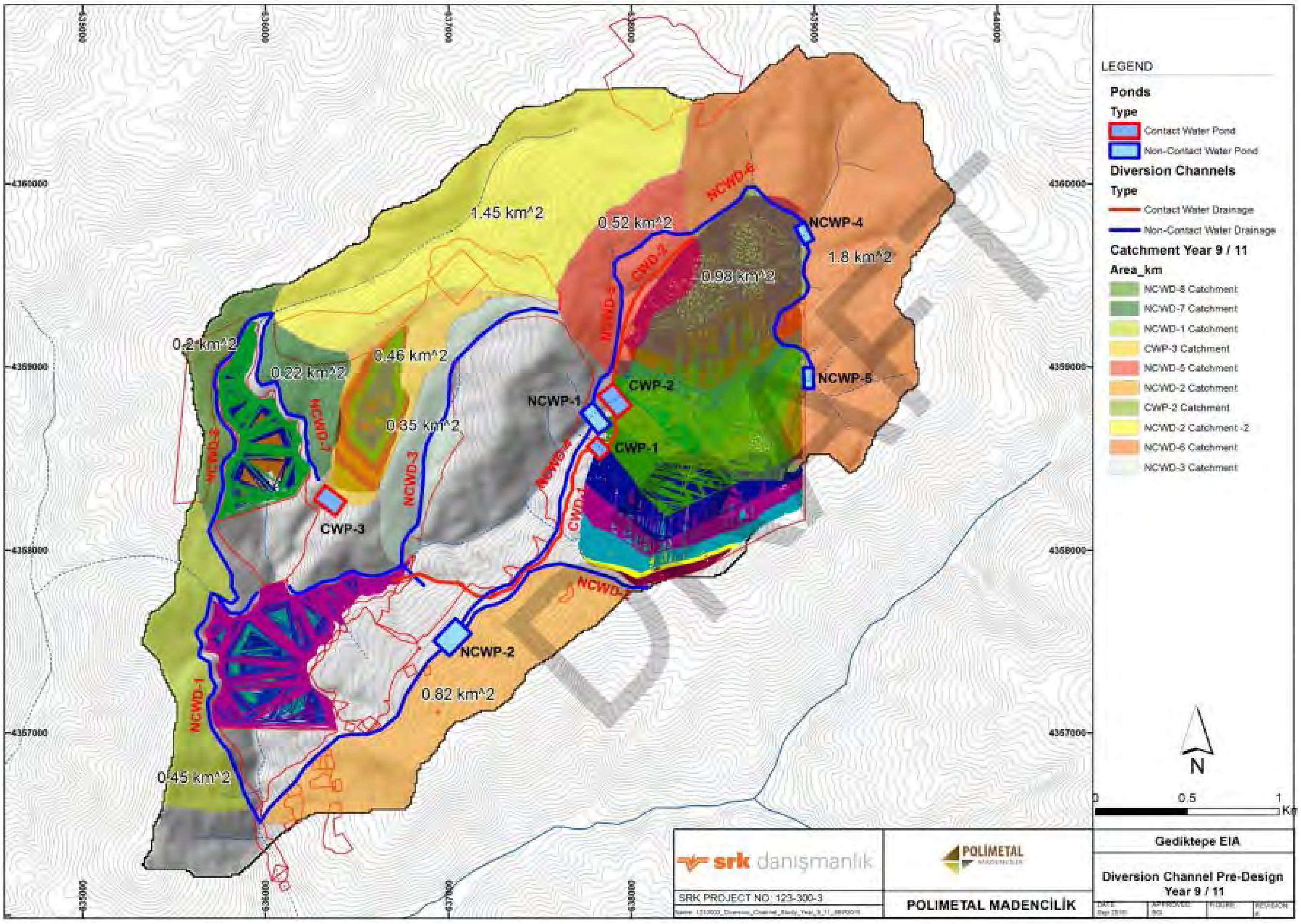


Figure 18-4
Water Diversion Channels

19.0 MARKET STUDIES AND CONTRACTS

The Gediktepe project is currently planned to produce the following three products:

1) A gold and silver dore from the heap leach process and Merrill Crowe
2) A copper concentrate
3) A zinc concentrate

The metallurgical testing to date indicates that the gold-silver dore and both concentrates will be of marketable quality. No specific contracts have been entered for delivery of dore or concentrate at this time. However, Polimetal has contacted metals trading organizations and have obtained estimated product shipment and treatment charges that have been used in the financial analysis for this project.

Concentrate Quality

The metallurgical testing has indicated that concentrate grades for the copper and zinc concentrates will typically be: 30.95% Cu and 54.3% Zn. The lock cycle tests that were completed do not indicate any deleterious elements will be present to hinder or encumber concentrate marketing.

Block modeling of mercury and arsenic result in low values in the ore that confirm the metallurgical test work.

Smelting Refining and Freight Terms and Source

Polimetal obtained estimated smelter terms from more than one source. IMC reviewed that information and selected the information shown on Table 15-6 in Section 15 as reasonable estimates of the smelting, refining, and freight terms that will be encountered by Polimetal.

IMC holds the opinion that the selected values are typical of those incurred by other base metal producers.

Government Royalty

The current structure for calculating royalties payable to the Turkish government were provided by Polimental financial personnel. The Turkish royalty rate is based on and changes with metal prices. For the metal prices used in the Pre-Feasibility Study, the royalty payable to the government is 4%. The royalty applies to net smelter return less processing costs and less site G&A costs.

A 30% increase is applied to the royalty rate for mining operations on Foresty lands where Gediktepe is located.

A 50% reduction in royalty rate is provided for operations that produce doré or concentrate on site which are the planned products of the Gediktepe project.

The resulting royalty payable to the Turkish government is:

Royalty = 4% * (1.3) * (0.5) = 2.6%

Metal Prices

The estimated metal prices for the project design basis were established by the project qualified persons. The base case metal prices for the project are:

Au: $1,000/oz,   Cu: $2.50/lb,   Ag: $15.00/oz. Zn: $1.00/lb.

For comparison, the Spot Price for the metals on 30 April 2016 were:

Au $1,286/oz,   Cu: $2.29/lb,   Ag: $17.85/oz, Zn: $0.88 /lb.

The three year backward averages for the same metals as of 30 April 2016 were:

Au $1,233/oz,   Cu: $2.82/lb,   Ag: $18.03/oz, Zn: $0.89 /lb.

The design prices for the project design are generally similar to the spot prices and the 3 year backward average values. A number of major copper producers in North America are using copper prices between $2.00 and $2.50 /lb copper for mine planning and reserve determination.

No contracts have been entered for product delivery or supply sourcing at this time.

Financial analysis in Section 22.0 has applied the following base case metal prices as requested by Polimetal.

Au $1,250/oz,   Cu: $2.75/lb,   Ag: $18.25/oz, Zn: $1.00 /lb.

Sensitivity analysis on metal prices indicates that the project is robust to substantially lower metal prices.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section has been written by Oguz Atil Karamercan of Polimetal and edited by John Marek of IMC.

20.1 Environmental Baseline Studies

Baseline Water Quality

Environmental baseline studies were started at the project site on September 15, 2013 with Topçuoğlu Madencilik San. ve Tic. Ltd. Şti. (Topçuoğlu). For water quality purposes, samples from 5 developed stand pipes (fountains) and 3 creeks were taken and measured for temperature, pH, EC & TDS values. Also, samples were sent to ALS (Prag), and were also analyzed 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. In December 2014, MS construction and commissioning was completed and the station was handed over to TSMS. Since then, online meteorological data has been collected from the MS and the meteorological data was incorporated into hydrology and hydrogeology studies by SRK.

- Water quality sampling and evaluation has been done. Five water reservoirs, 22 fountains, 11 water springs, and 10 surface water locations have been identified. Five water reservoirs and 7 surface water locations were studied and sampled. These were observed for three seasons.

Water samples were analyzed for temperature, pH, electrical conductivity, salinity, total soluble solids, oxidation-reduction potential, soluble oxygen, and flow amounts.

Acid Rock Drainage

A geochemical characterization program 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 program 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 10.0% to 66.0%) consists of quartz, which is considered environmentally inert.
- 8 out of 12 samples were found to contain a carbonate (calcite) concentration of 0.5% to 29.8%.
- 1 sample was found to contain a dolomite concentration of 5.9%.
- The main sulfur mineral is pyrite. 9 samples were found to contain a pyrite concentration of 0.1% to 0.8%, 1 sample was found to contain a marcasite concentration of 2.3%. The massive pyrite sample contains 85.5% pyrite.
- 1 sample was found to contain a hematite concentration of 37.7%, and 1 sample was found to contain an ankerite concentration of 5.2%.
- 5 samples were found to contain a magnetite concentration of 1.4% to 4.7%.

The results of the geochemical characterization are summarized on Table 20-1 from Golder's "Limited Environmental Baseline Study Report".

<u>Flora and Fauna</u>

Additional information regarding the site environmental conditions included:

- Land usage, protection zones and archeological status were assessed.
- Fauna & 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 desk top 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 Characterization 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

*WR: Waste Rock, PAG: Potentially Acid Generating, NNP: Net Neutralization Potential, NPR: Neutralization Potential Ratio, NAG: Non Acid Generating

20.2 Environmental Impact Assessment Studies and Reporting

On November 1, 2014 an agreement was signed with SRK to continue the baseline studies and to prepare the EIA report. The following studies were performed by SRK;

- Meteorological data of Dursunbey MS was compiled,
- Fauna & 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 mine plan and facilities layout, and all the EIA studies focused within the red line boundary on Figure 20-1

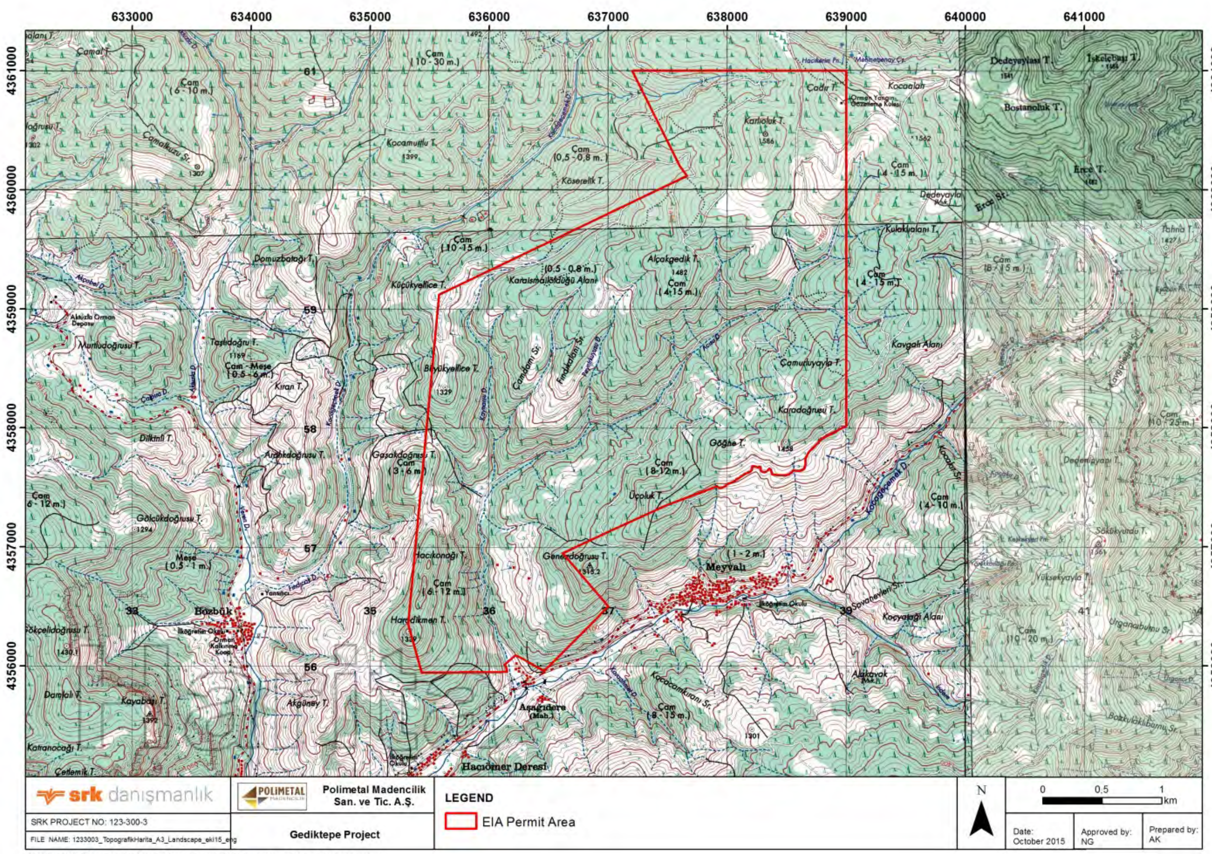


Figure 20-1
EIA Boundary

20.2.1 Fauna, Flora & Hydrobiology Studies

Nineteen endemic flora species (1 local, 6 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 November 11, 2015.

During fauna studies, phototraps 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.

20.2.3 Public Participation Meeting

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. Around 120 individual 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 demands were addressed in the EIA report. Alternative water sources have been identified and designs of the pipelines for these alternative water sources are in progress. Engineering of the by-pass road has been completed.

20.2.4 Hydrology and Hydrogeology Studies

SRK studied regional and project area hydrology and characterized catchment basins. Figure 20-2 illustrates the drainage catchment basins that were defined.

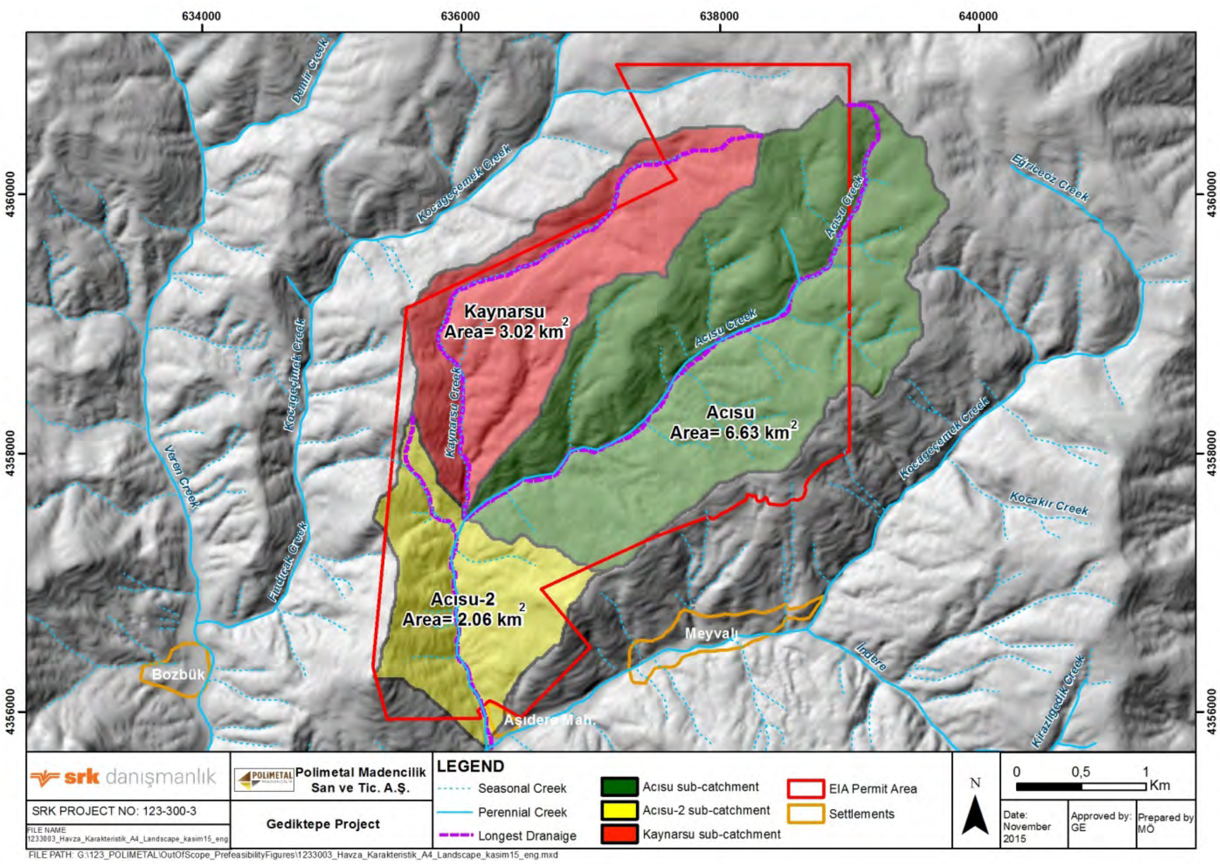


Figure 20-2
Project Area Drainage Catchment Basins

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 illustrates the locations of the weirs.

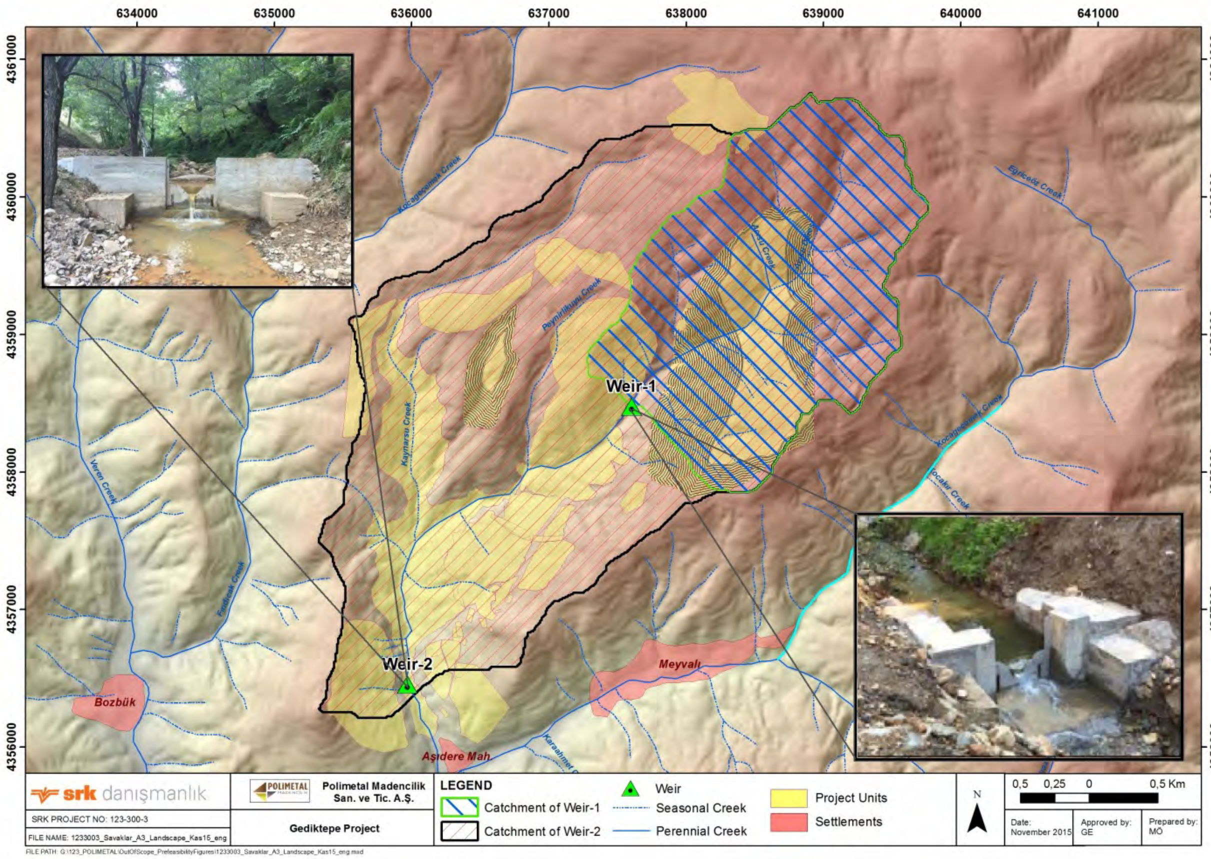


Figure 20-3
Location of Weirs for Flow Rate Measurements

Thirteen water observation wells were drilled and tested to define hydrogeology properties of the basin. Table 20-2 illustrates the locations and the summarized test results from the 13 wells. Figure 20-4 illustrates the static water levels within the wells over time.

A 3 dimensional calibrated underground water flow model was established by SRK Based on the aquifer test results and static water levels. A stylized illustration of the flow model is presented on Figure 20-5.

Table 20-2
Water Observation Wells Locations and Summary Test Results

Hole #	X	Y	Z	Drilled Depth (m)	Casing Depth (m)	Drilling Diameter (in.)
GTMW-01A	636698	4357838	1127	112	109	14 (0-32m), 10 (32-112m)
GTMW-01B	636709	4357842	1128	32	30	7
GTMW-02A	637592	4358509	1239	182	170	10
GTMW-03	636943	4359026	1396	330	107	10 (0-104m), 7 (104-330m)
GTMW-03B	636940	4359020	1399	325	270	10 (0-104m), 7,5 (104-325m)
GTMW-04A	637526	4357895	1292	56	39	10
GTMW-04B	637515	4357909	1290	206,5	126,5	10 (0-59m), 4,7 (59-206,5m)
GTMW-06	636399	4358312	1263	122	121	10
GTMW-08A	636809	4357213	1233	70	61,5	10
GTMW-09	636099	4356108	985	92	92	10
GTMW-11	635960	4359176	1278	116	96	10
GTMW-14	638980	4359748	1443	86	84	10
GTMW-15	638508	4358198	1446	88	88	10

Hole #	Flow Amount (l/s)	Drawdown (m)	Pumping Time (hr)	Test Type	Hydraulic Conductivity (m/s)
GTMW-01A	0,23	28,57	72	Pump Test	$4,2E^{-08}$
GTMW-02A	0,22	66,32	72		$8,2E^{-09}$
GTMW-03B	2,04	43,04	72		$1,60E^{-07}$
GTMW-04A	1,51	16,92	20,5		$1,30E^{-06}$
GTMW-08A	0,62	26,35	38		$1,4E^{-07}$
GTMW-09	0,45	24,99	26		$7,8E^{-08}$
GTMW-11	12,2	26	39		$1,7E^{-06}$
GTMW-14	4,33	63,53	72		$6,7E^{-07}$
GTMW-15	0,26	39,96	15		$1,3E^{-07}$
W1	25	41,7	7,5		$3,4E^{-06}$
GTMW-04B	0,1	117	4	Airlift Test	$5,3E^{-08}$
GTMW-06	0,4	48	2	Build Up Test	$5,4E^{-08}$

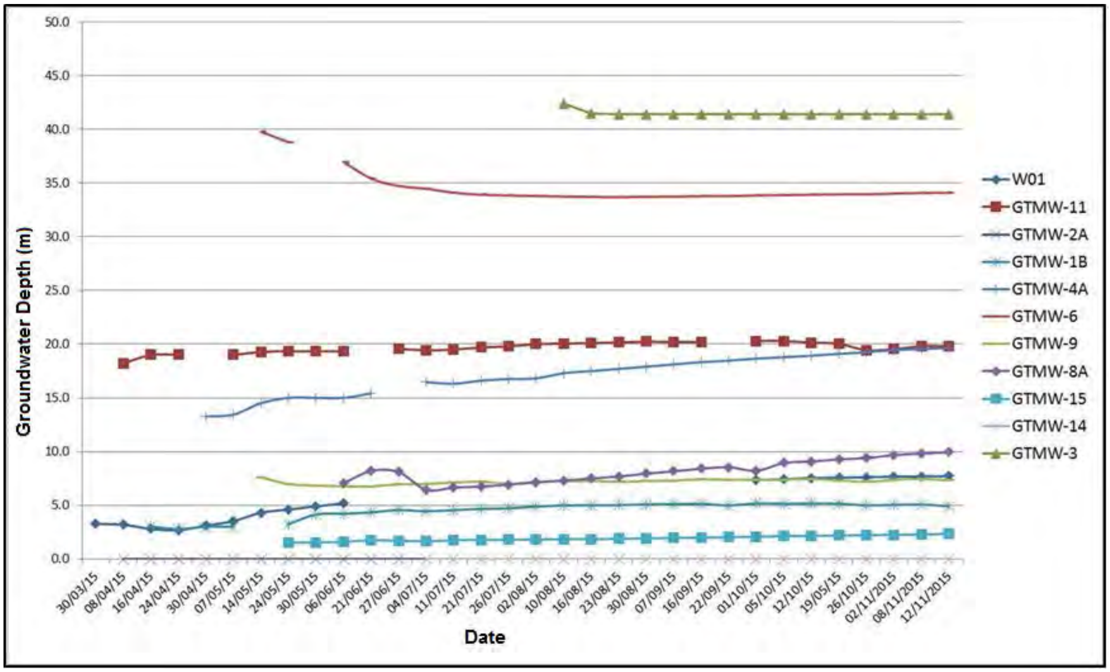


Figure 20-4
Water Levels in Wells March – December 2015

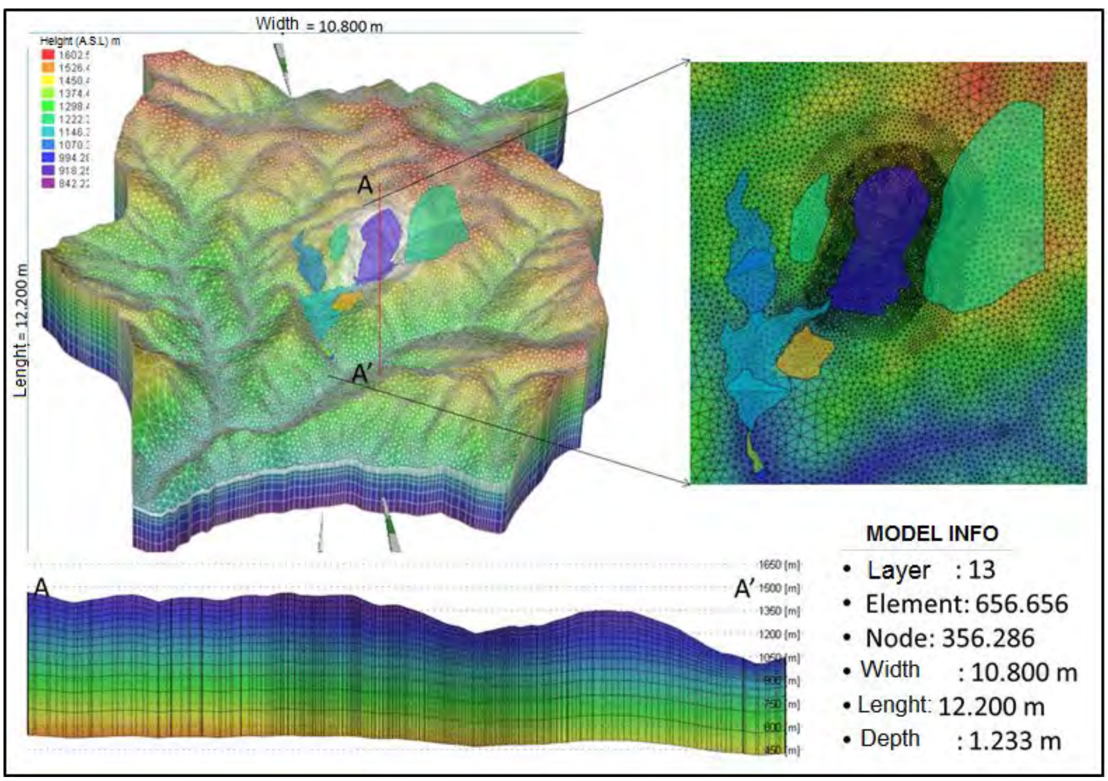


Figure 20-5
Illustration of the Ground Water Flow Model

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 is planned at the south side of the TSF. The final capacity of the water dam will be calculated within the site wide water management study which will be prepared by SRK Turkey, Vancouver & Denver.

Incorporating the meteorology and weir flow information, a site wide water balance was calculated for the project. The quality of all surface water sources has been analyzed and the flow rates have all been measured.

20.2.5 Alternative Surface Water Sources for Villages Near the Project

SRK modelled the annual drawdown cone of the water for the current village water sources based on the mine plans and the mine disturbance of the village water sources. Based on this study, 5.6 ltr/sec of water in the wet season and 3.1 ltr/sec of water in the dry season will be affected by mining activities as shown on Table 20-3.

Table 20-3
Water Amount Affected During Mining

	Water Usage of Villages, (ltr/sec)		Water Amount That Will be Affected During Mining (ltr/sec)	
	Wet Season	Dry Season	Wet Season	Dry Season
Hacıömerderesi Village	3.0	1.8	3.0	1.8
Aşıderesi Village	0.5	0.4	0.5	0.4
Meyvalı Village	13.1	5.0	1.1	0.5
Gardens	1.0	0.4	1.0	0.4
Total	17.6	7.6	5.6	3.1

Figure 20-6 illustrates the draw down cone and the water sources that are affected by mining.

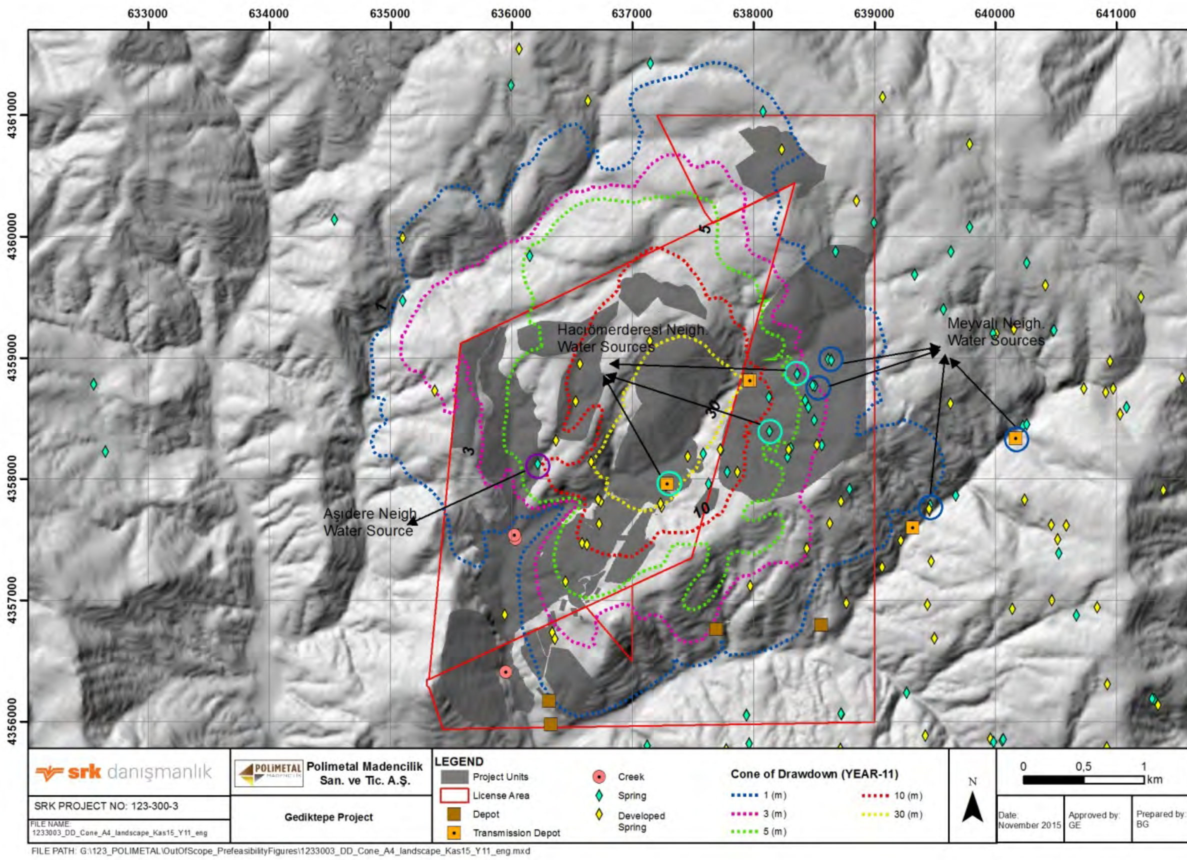


Figure 20-6
Drawdown Cone and Water Sources Affected by Mining Activity

To identify alternative village water sources, SRK performed an extended hydro-census study at the north and south sides of the project area. Fifteen alternative water sources were identified, flow rates were measured, and qualities were analyzed. Figure 20-7 illustrates the alternative water sources that were identified by SRK.

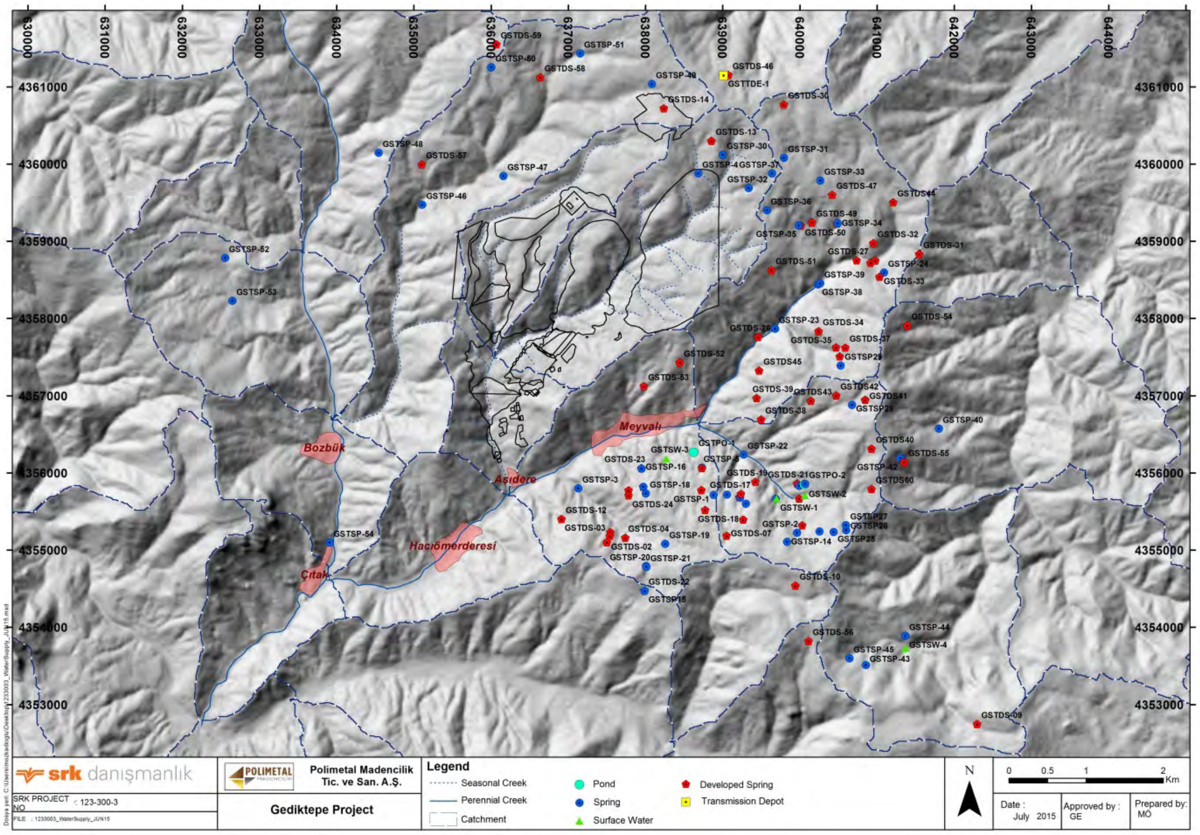


Figure 20-7
Identified Alternative Surface Water Sources

SRK has recommended water supplies for the villages from 5 alternative sources that will provide 11 ltr/sec of water. The alternative water sources, water storage tanks at the villages, and planned pipeline routes were surveyed. Engineering of the pipeline is in progress at the time of this report preparation.

20.2.6 ARD & Metal Leaching

SRK performed a gap analyses for ARD and metal leaching resulting in the selection of new samples for testing. All samples were selected from diamond drill core. 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.

SRK focused on selecting samples to represent all ore and waste lithologies in the mine. Table 20-4 is a list of the 55 static test samples which are comprised of: 12 gossan samples, 25 chlorite sericite schist samples, 5 quartz feldspar schist samples, 7 quartz schist samples, 3 fault zone samples, and 3 samples of massive pyrite.

As scheduled in the project execution plan, Polimetal will develop a waste rock management plan to schedule waste rock production and dumping sequences. For that reason, kinetic tests on 5 samples (4 waste rock and one ore samples) are ongoing.

Approximately 60% of the waste rock is quartz feldspar schist and chlorite sericite schist. Total sulfide amount of these rocks is lower than 0.1%, which is accepted as inert waste rock according to Turkish regulation.

Marble & dacite rocks that were tested for sulfide content are available in the EIA boundary and will be used if needed to encapsulate the PAG waste rock. Details of this design will be defined in the waste rock management plan, which will be completed before waste rock mining begins.

Table 20-4
ARD / ML Analysis Sample List

Static Test Sample No	Kinetic Test Sample No	Sample Type	Lithology	Zone	ABA	Total Rock	Static NAG	NAG Solution Analysis	Leachate Analysis	Kinetic Test	XRD	Drill No	Start (m)	Finish (m)
GT-Ore-1	HC 1	Ore	Gossan	Oxide	X	X	X	X	X	X		DRD-033	3,0	5,0
GT-Ore-2		Ore	Gossan	Oxide	X	X			X			DRD-020	42,0	43,6
DGS-9		Waste+Ore	Gossan	Oxide	X	X	X		X			DRD-019	9,1	15
DGS-12		Ore	Gossan	Oxide	X	X	X		X		X	DRD-001	31,9	37,5
DGS-13		Waste	Gossan	Oxide	X	X	X	X	X			DRD-012	52	56
DGS-14		Ore	Gossan	Oxide	X	X	X	X	X			DRD-062	1,8	5
DGS-15		Ore	Gossan	Oxide	X	X	X		X			DRD-013	12,4	17,4
DGS-16		Ore	Gossan	Oxide	X	X	X		X		X	DRD-015	23,8	30
DGS-17		Waste	Gossan	Oxide	X	X	X		X		X	DRD-008	5,5	9,5
GT-WR-11		Waste	Gossan	Oxide	X	X	X		X			DRD-023	4,0	8,0
GT-WR-21		Waste	Gossan	Oxide	X	X	X		X			DRD-005	2,0	6,0
GT-WR-22		Waste	Gossan	Oxide	X	X	X	X	X		X	DRD-015	0,0	3,8
GT-WR-2	HC 3	Waste	Chlorite Sericite Schist	Ox/Sul	X	X	X	X	X	X	X	DRD-116	5,7	7,8
GT-WR-3		Waste	Chlorite Sericite Schist	Oxide	X	X	X		X			DRD-006	17,5	21,0
DGS-1		Waste	Chlorite Sericite Schist	Oxide	X	X	X		X		X	DRD-048	8,3	13,7
GT-WR-7	HC 4	Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X	X		DRD-022	111,0	115,0
DGS-6	DGS-6	Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X	X	X	DRD-069	29,2	35
GT-WR-10	HC 6	Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X	X		DRD-122	24,0	28,0
GT-Ore-3	HC 2	Ore	Chlorite Sericite Schist	Sulfide	X	X	X	X	X	X	X	DRD-012	95,0	98,5
DGS-22	DGS-22	Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X	X	X	DRD-012	109,2	117,0
DGS-2		Waste	Chlorite Sericite Schist	Sulfide	X	X	X		X			DRD-003	35	43
DGS-3		Waste	Chlorite Sericite Schist	Sulfide	X	X	X		X			DRD-024	103,5	109,5
DGS-4		Waste	Chlorite Sericite Schist	Sulfide	X	X	X		X			DRD-015	46,5	54,5
DGS-5		Waste	Chlorite Sericite Schist	Sulfide	X	X	X		X			DRD-048	50	55
DGS-7		Waste	Chlorite Sericite Schist	Sulfide	X	X	X		X			DRD-020	128,5	136,5
DGS-8		Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X		X	DRD-043	19	27

*ABA: Acid Base Accounting, NAG: Non Acid Generating, XRD: X-Ray Diffraction

Table 20-4, Continued
ARD / ML Analysis Sample List

GT-WR-1		Waste	Chlorite Sericite Schist	Sulfide	X	X	X		X			DRD-013	27,0	29,5
GT-WR-4		Waste	Chlorite Sericite Schist	Sulfide	X	X						DRD-031	21,0	23,2
GT-WR-5		Waste	Chlorite Sericite Schist	Sulfide	X	X						DRD-038	52,5	56,5
GT-WR-6		Waste	Chlorite Sericite Schist	Sulfide	X	X						DRD-039	28,8	31,7
GT-WR-8		Waste	Chlorite Sericite Schist	Sulfide	X	X						DRD-073	175,6	179,5
DGS-26		Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X		X	DRD-066	43,5	50,8
GT-WR-18		Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X			DRD-009	71,0	75,4
GT-WR-19		Waste	Chlorite Sericite Schist	Sulfide	X	X						DRD-071	53,0	56,0
GT-WR-20		Waste	Chlorite Sericite Schist	Sulfide	X	X						DRD-096	88,0	91,0
DGS-21		Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X			DRD-070	37,9	42
DGS-23		Waste	Chlorite Sericite Schist	Sulfide	X	X	X	X	X			DRD-020	90,5	98,5
GT-WR-12	HC 5	Waste	Quartz Feldspar Schist	Oxide	X	X	X	X	X	X	X	DRD-016	26,5	29,5
DGS-24		Waste	Quartz Feldspar Schist	Oxide	X	X	X		X			DRD-012	2	10
GT-WR-13		Waste	Quartz Feldspar Schist	-	X	X	X		X			DRD-027	22,0	27,5
GT-WR-9		Waste	Quartz Feldspar Schist	Sulfide	X	X	X		X			DRD-073	50,0	54,0
GT-WR-14		Waste	Quartz Feldspar Schist	Sulfide	X	X	X	X	X			DRD-157	18,3	21,0
GT-WR-15	HC 7	Waste	Quartz Schist	Sulfide	X	X	X	X	X	X		DRD-002	78,5	84,5
DGS-29	DGS-29	Waste	Quartz Schist	Sulfide	X	X	X	X	X	X	X	DRD-002	68,5	76,5
GT-WR-16		Waste	Quartz Schist	Sulfide	X	X	X					DRD-116	65,0	67,0
GT-WR-17		Waste	Quartz Schist	Sulfide	X	X	X	X	X			DRD-040	49,0	55,0
DGS-27		Waste	Quartz Schist	Sulfide	X	X	X		X		X	DRD-004	148,5	153,5
DGS-28		Waste	Quartz Schist	Sulfide	X	X	X	X	X			DRD-014	30,4	38,4
DGS-30		Waste	Quartz Schist	Sulfide	X	X	X	X	X			DRD-002A	34	40
DGS-10		Waste	Fault Zone	Sulfide	X	X	X		X			DRD-017	50	55,5
DGS-11		Waste	Fault Zone	Sulfide	X	X	X	X	X		X	DRD-022	195	203
DGS-25		Ore	Fault Zone	Oxide	X	X			X			DRD-004	31	34
DGS-18		Ore	Massive Pyrite Zone	Sulfide	X	X	X	X	X		X	DRD-043	43	49
DGS-19		Ore	Massive Pyrite Zone	Sulfide	X	X	X	X	X			DRD-062	6,8	11,6
DGS-20		Ore	Massive Pyrite Zone	Sulfide	X	X	X	X	X			DRD-012	83	88

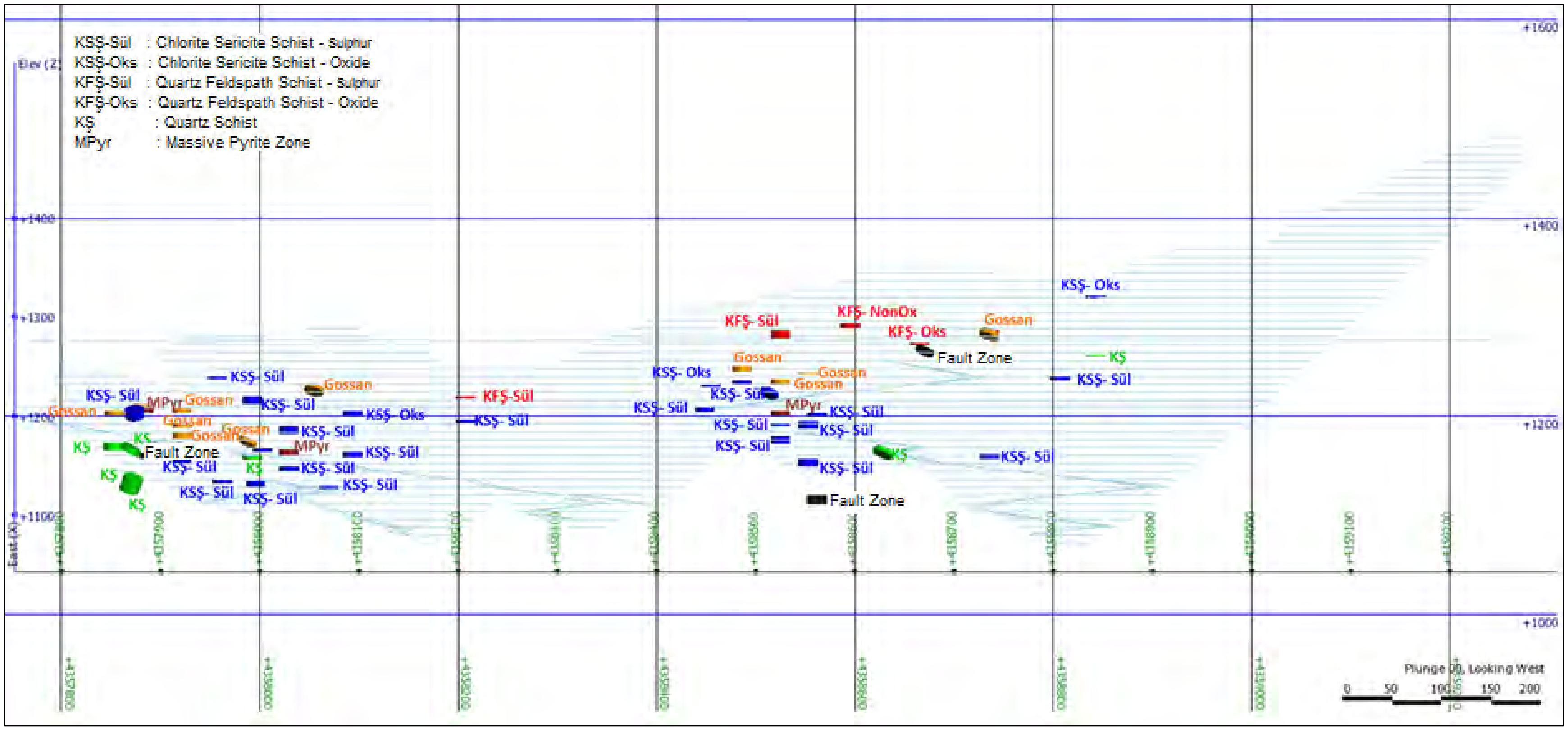


Figure 20-8
ARD Sample Locations in Longitudinal Profile, Looking West
Pit Outline Shown in Light Blue

The following Table 20-5 shows the kinetic test sample list. For the EIA report, the 38 week results of kinetic tests were utilized. However, SRK has advised Polimetal to continue kinetic testing on samples HC2, HC4, HC7, DGS22 & DGS29 to see the net acid potential and soluble metal in these samples after longer periods of time.

Table 20-5
List of Kinetic Sample Tests

Kinetic Test Sample No	Static Test Sample No	Sample Type	Lithology	Zones	Drill No	Start (m)	Finish (m)	Test Duration (week)
HC 1	GT-Ore-1	Ore	Gossan	Oxide	DRD-033	3	5	38
HC 2	GT-Ore-3	Ore	Chlorite Sericite Schist	Sulfide	DRD-012	95	98,5	> 40 & still continue
HC 3	GT-WR-2	Waste	Chlorite Sericite Schist	Ox/Sul	DRD-116	5,7	7,8	38
HC 4	GT-WR-7	Waste	Chlorite Sericite Schist	Sulfide	DRD-022	111	115	> 40 & still continue
HC 5	GT-WR-12	Waste	Quartz Feldispat Schist	Oxide	DRD-016	26,5	29,5	38
HC 6	GT-WR-10	Waste	Chlorite Sericite Schist	Sulfide	DRD-122	24	28	38
HC 7	GT-WR-15	Waste	Quartz Schist	Sulfide	DRD-002	78,5	84,5	> 40 & still continue
DGS-6	DGS-6	Waste	Chlorite Sericite Schist	Sulfide	DRD-069	29,2	35	38
DGS-22	DGS-22	Waste	Chlorite Sericite Schist	Sulfide	DRD-012	109,2	117	> 40 & still continue
DGS-29	DGS-29	Waste	Quartz Schist	Sulfide	DRD-002	68,5	76,5	> 40 & still continue

The results of kinetic tests at week 38 are summarized in the following graphs (Figures 20-9 through 20-13), which show pH, EC, SO_4 concentration, and total Ficklin metals concentration.

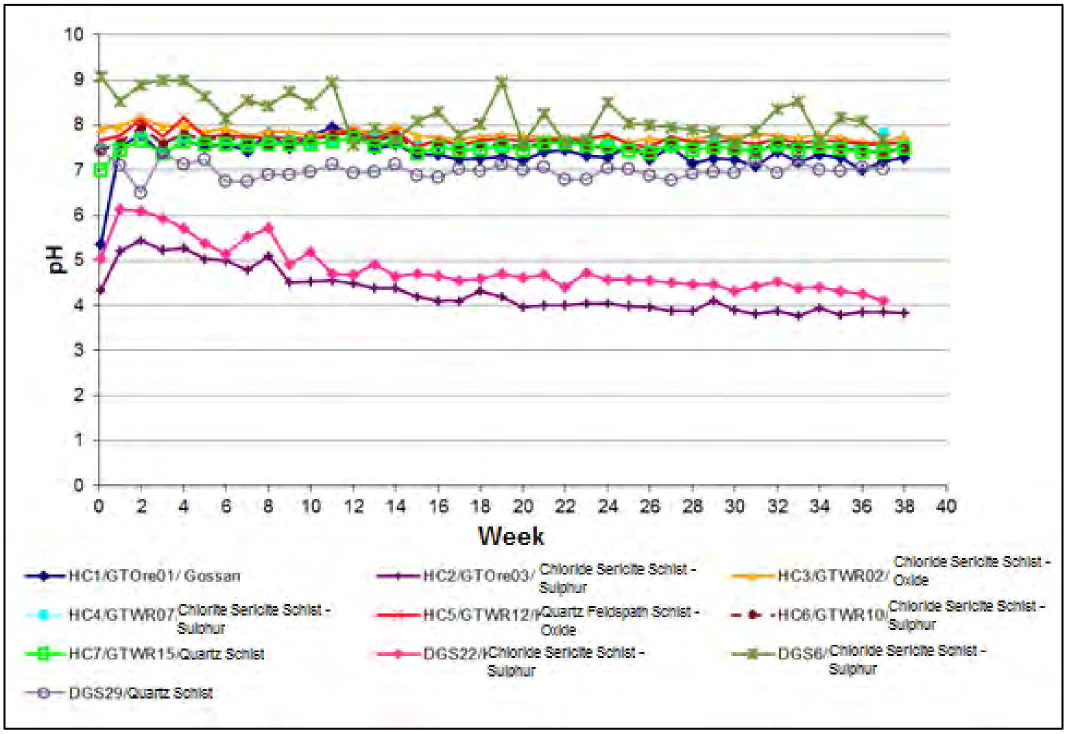


Figure 20-9
Leachate Weekly pH Change

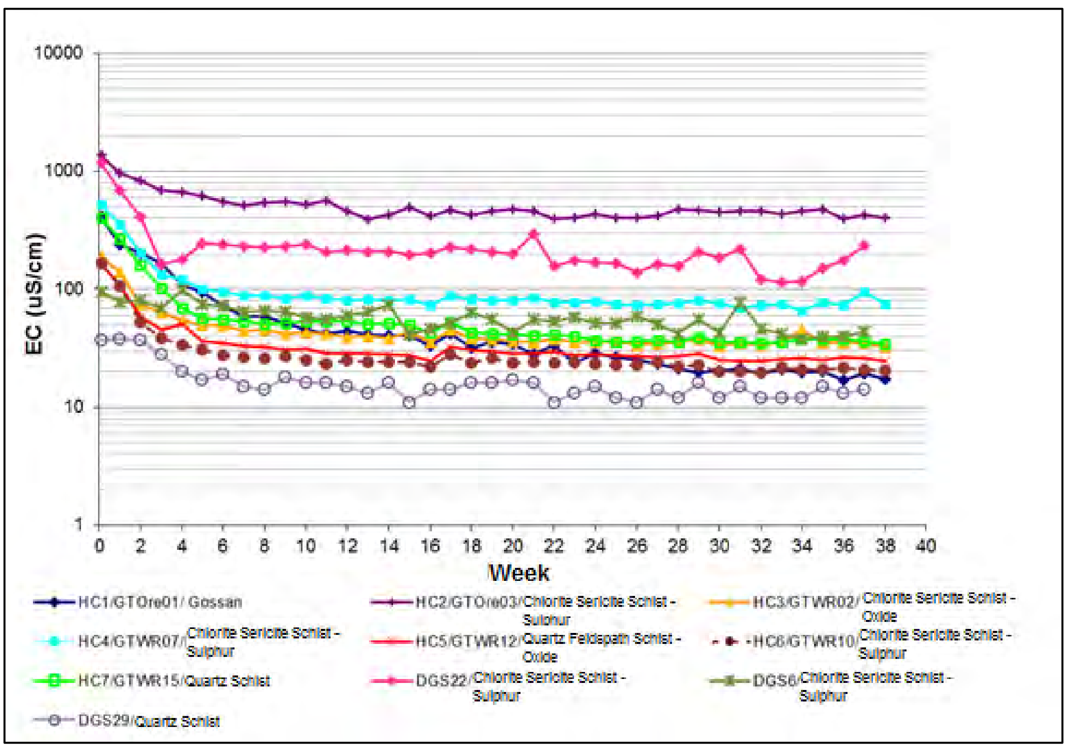


Figure 20-10
Leachate Weekly EC Change

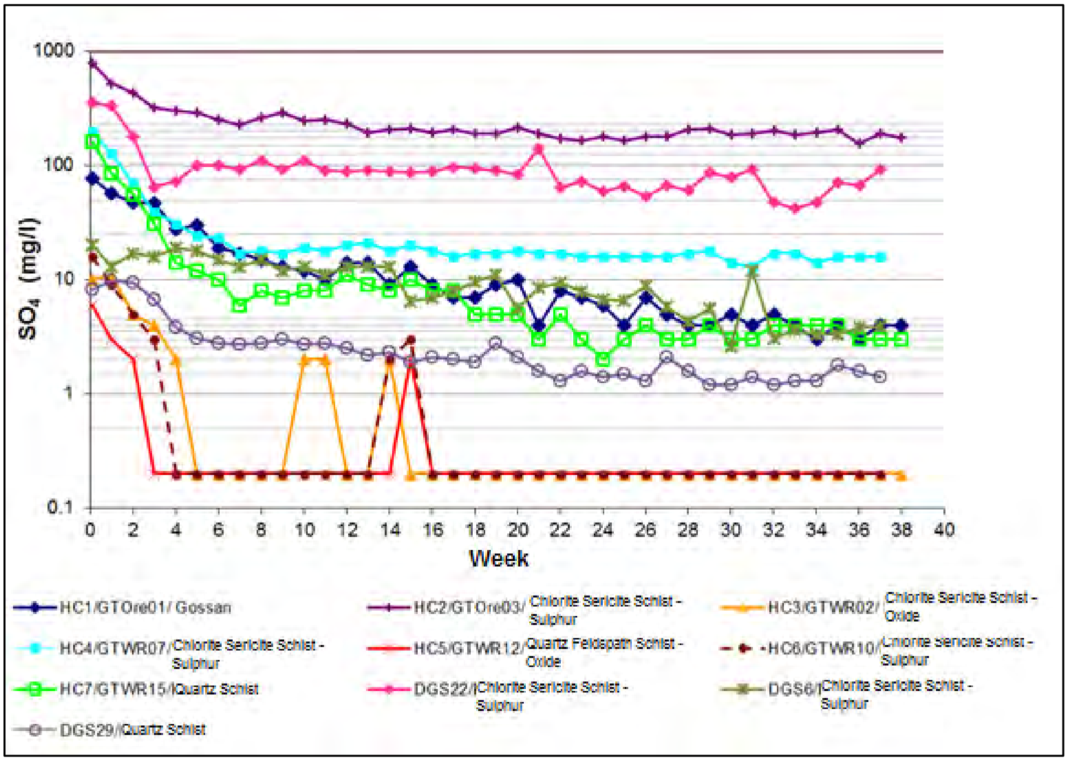


Figure 20-11
Leachate Weekly SO_4 Concentration Change

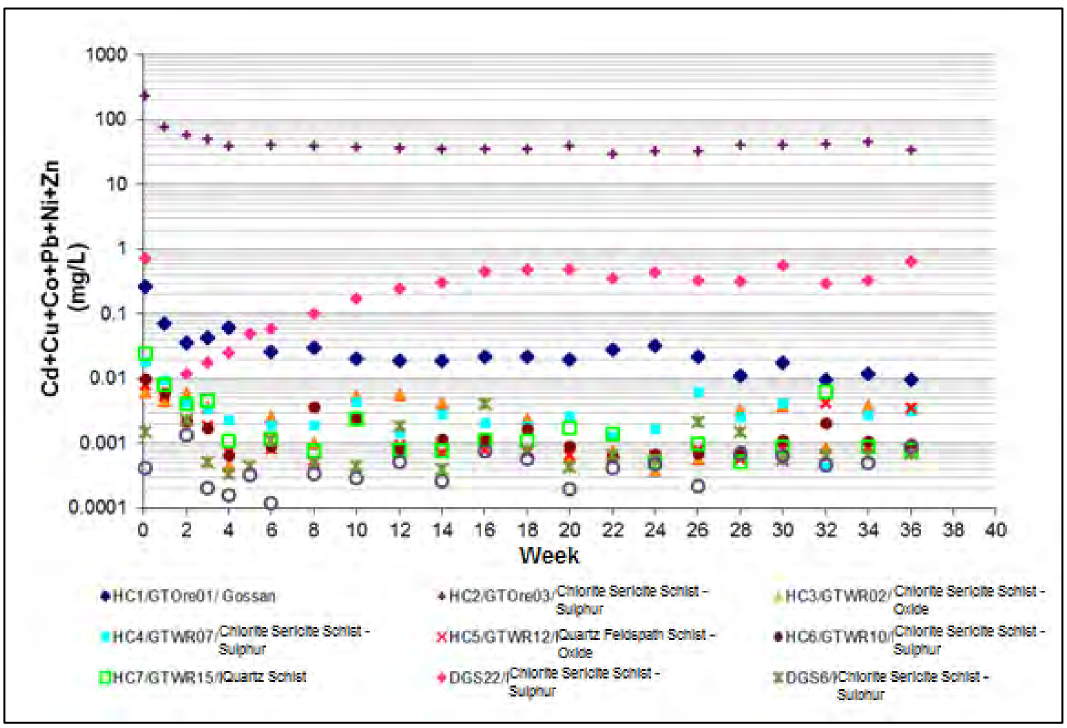


Figure 20-12
Leachate Weekly Total Ficklin Metals Concentration Change

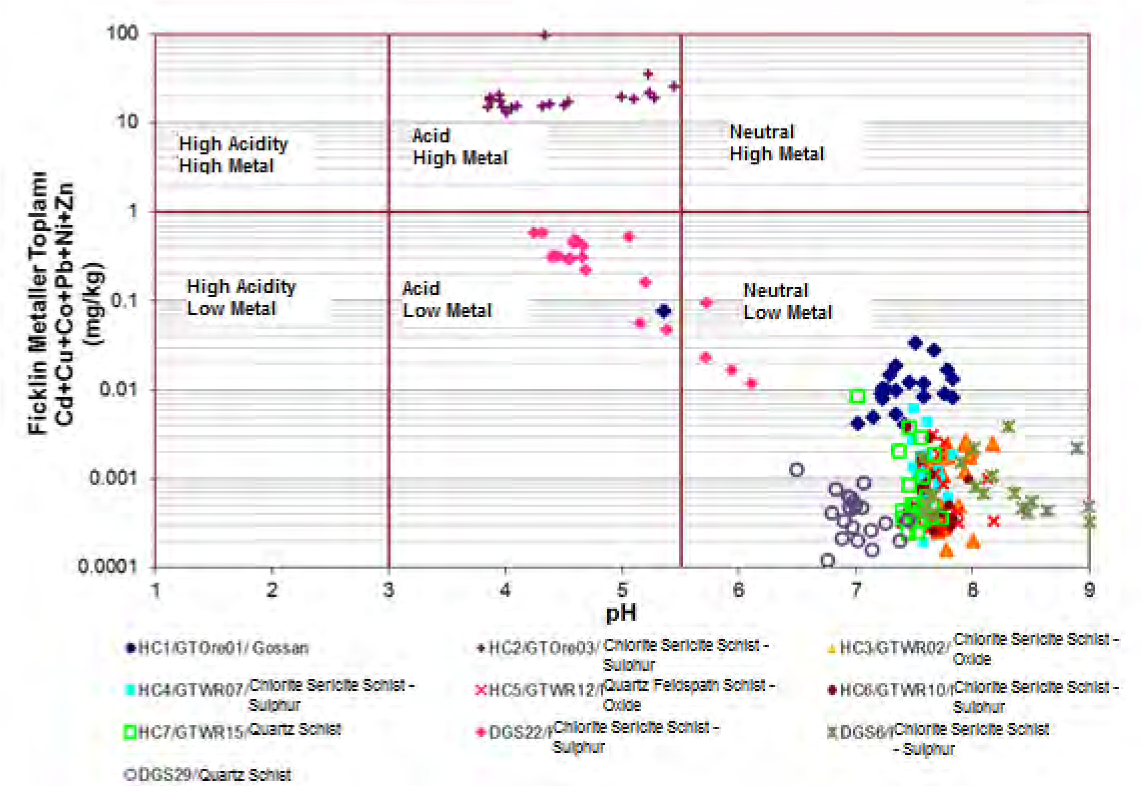


Figure 20-13
Leachate Weekly Total Ficklin Metals Graph

Based on the static & kinetic test results, SRK concluded that;

- HC4 sample (sulfide 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 feldspat schist), HC6 (sulfide zone chlorite sericite schist) & DGS6 (sulfide zone chlorite sericite schist) samples were identified as not acid generating.

20.2.7 Emissions

SRK measured PM_{10} parameters at the site and created an air quality model for PM_{10} (Figures 20-14 and 20-15). 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.

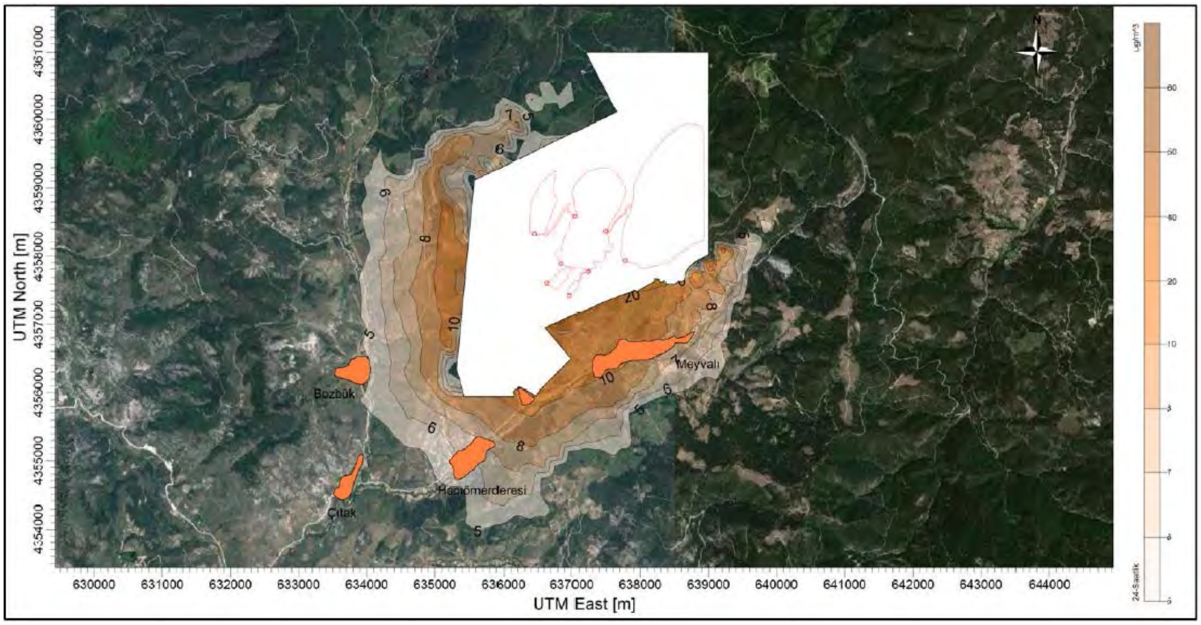


Figure 20-14
24 Hours PM_{10} Concentration Dispersion

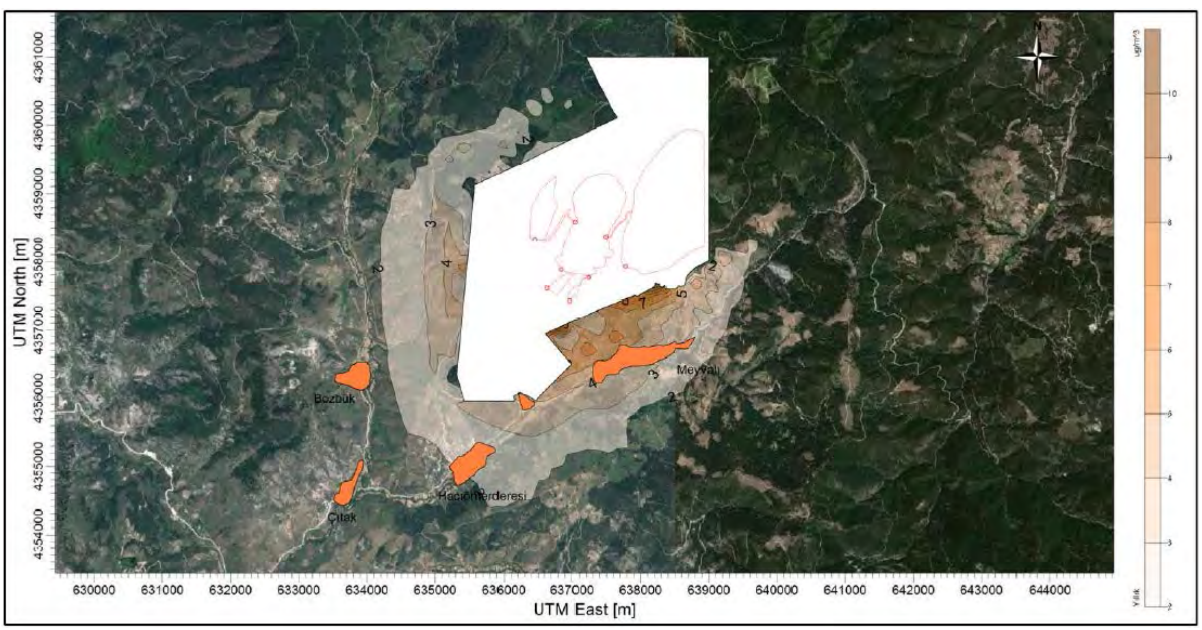


Figure 20-15
Annual PM_{10} Concentration Dispersion

Table 20-6
Air Quality of Modeling of PM_{10} at Nearby Villages

		24 Hrs	Total	Annual	
	Baseline	Increase		Increase	Total
	(µg/m^3)	(µg/m^3)	(µg/m^3)	(µg/m^3)	(µg/m^3)
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 & Meyvalı Villages.

20.2.8 EIA Status

Based on the studies that are summarized in this section, SRK compiled the EIA report and submitted it to Ministry of Environment and Urbanization on December 15, 2015. The first evaluation commission meeting was held with the participation of 18 government institutions on January 13, 2016.

Additional information was requested by the Water and Sewage Administration of Balıkesir Municipality. That information has been prepared. A revised EIA report was re-submitted to Ministry of Environment and Urbanization in late February 2016.

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 as shown in the PFS will require a total 379.2 hectares of forest permit area over the life of the mining operation.

At this time, Polimetal expects that the following additional permits will be required:

- EIA Permit (in progress)
- Forest Permits,
    - The pre-forest permit application was done on January 19, 2016. Permanent forest permit application and the application of following permits will be started after receiving the EIA positive certificate for the project
- Workplace Opening & Working permit
- Explosive usage and storage permits
- Environmental permit (including emission & 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 others that are not foreseen at this time and some of the above may become unnecessary as more planning and detail is completed at Gediktepe.

20.4 Land Ownership

90.78% of the project area belongs to the Forest Department and the rest are private lands owned by Municipality, Treasury, and individuals (1,068,313.4 m^2). To date, 403,000 m^2 of private lands have been purchased by Polimetal, and the purchasing process still continues.

Figure 20-16 shows land ownership within the EIA boundary. Areas owned by Forestry Department are shown in Green.

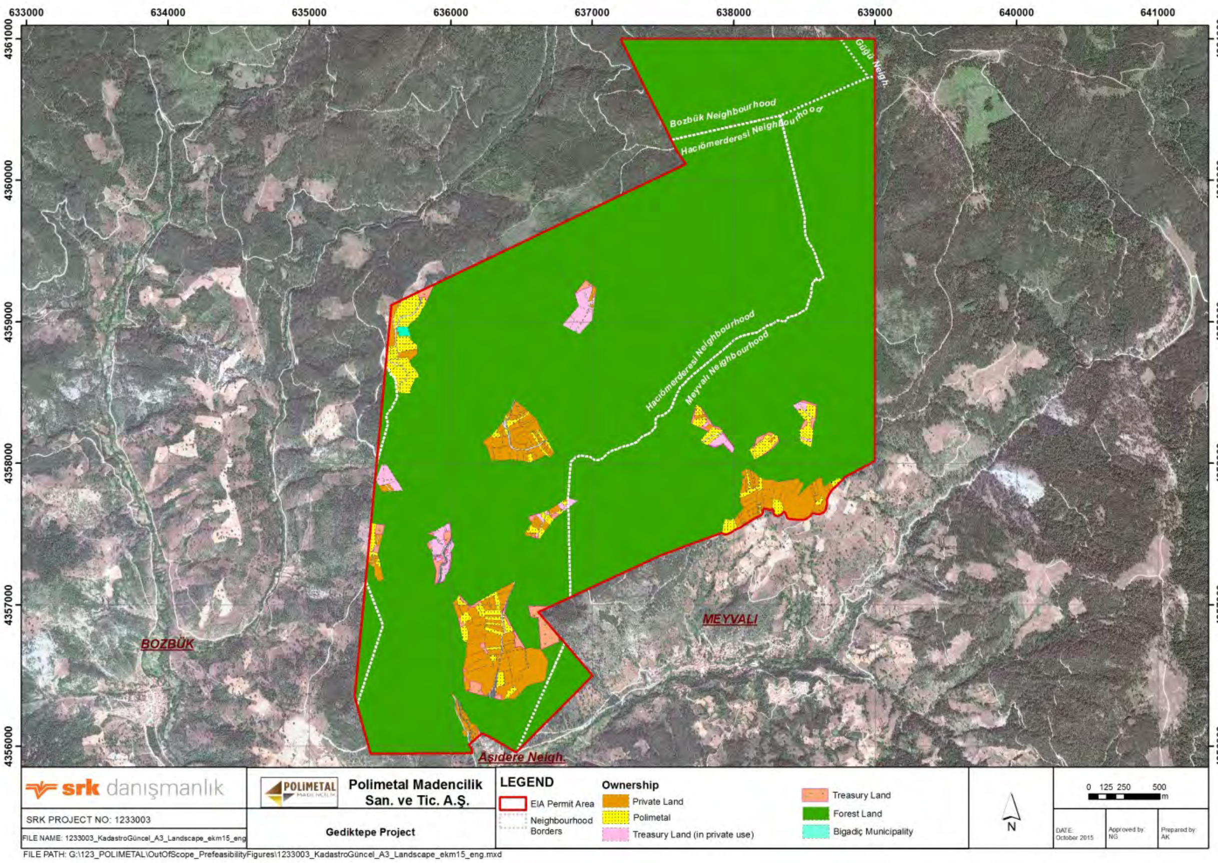


Figure 20-16
Land Ownership with the EIA Boundary,
Green Areas are Owned by the Forestry Department

20.5 Social or Community Impact

Polimetal has been drilling on the site since 2012 and has the support of the local community. Currently, the exploration camp is established and is 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.

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. About 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 programs and, transportation.

The Turkish State Hydraulics Department (DSI) has designed and planned to construct a potential water storage pond that would be located within the footprint of the TSF, for local irrigation purpose. Because of this conflict, Polimetal has applied to the Mining Bureau to take a public welfare decision in favor of Gediktepe project. The Mining Bureau visited the site, took ore samples and all the project details and the public welfare decision was taken by the Mining Bureau and the DSI's water storage pond has been cancelled with the approval of three ministers (Minister of Forest and Water Works, Minister of Energy and Natural Resources & Minister of Development) in October 2015.

20.6 Closure

The details of project closure are still being finalized 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 Heap Leach Pad and the Tailing Storage Facility have been provided by SRK. Those costs are included in the project financial analysis as late stage capital expenditures in years 5 and 13.

A plan has been developed and costs have been included for back filling the mine pit with waste following mining to an elevation that will prevent pit lakes from forming.

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 decommissioning of the plant facilities.

A summary of the estimated closure and reclamation cost are summarized in Section 21.

21.0 CAPITAL AND OPERATING COSTS

21.1 Operating Costs

The expected operating costs over the Gediktepe mine life are estimated to total $695.1 million USD. These costs include the costs of mining, processing, general and administrative (G&A) costs, and government permitting fees. The average operating costs over the life of mine by category are provided on Table 21-1.

All costs are presented in 4th quarter 2015 U.S. Dollars. Costs in Turkish Lira were converted to U.S. Dollars at the exchange rate of: 3.00 Lira / U.S.Dollar.

Table 21-1
Gediktepe Mine Life Operating Cost by Category

OPCOST Category	Unit Cost	Units	Total Cost ($000's)
Mining	1.45	$/tonne material	221,126.5
Oxide Ore Processing	9.51	$/tonne ore	30,640.8
Sulfide Ore processing	11.88	$/tonne ore	257,678.7
Site Wide G&A	7.45	$/tonne ore	185,661.4
		Total:	695,107.5

21.1.1 Mining Operating Costs

A contract mining operating cost has been provided by a local contract mining company based on an internal mine plan developed by Polimetal. The estimated contract mining costs for moving mine rock are: $1.26/tonne of waste and $0.84 /tonne of ore. In addition to the direct mine cost, there are contractor charges for: stockpile rehandle at the crusher, land clearing, and road construction. The total mine cost also includes the Polimetal supervisory personnel.

This total mine operating cost estimate includes:

1. Contractor provided:
   Drilling, blasting, loading, and hauling of material from the mine to the crusher, low grade stockpile or waste storage facility. Maintenance of the waste storage areas and stockpiles is included in the mining costs. Maintenance of mine mobile equipment is included in the operating costs.

2. Road construction costs have been included based on the contractor rates for construction and the amount of road built annually.

3. An additional allowance for mine supervision, mine engineering, geology and ore control is included as a separate Polimetal staff category and is in addition to the cost of contract mining.

The additional mining staff requirements and costs are presented in Table 21-2.

Table 21-2
Polimetal Mining Staff Requirements

Personnel Title	#persons	Annual Cost
Mine Manager	1	$ 179,933
Mine Production Superintendent	1	$ 98,423
Shift Engineers	4	$ 61,958
Mine Planning Superintendent	1	$ 98,423
Mine Planning Engineer	3	$ 61,958
Blasting Engineer	1	$ 61,958
Rock Mechanics Engineer	1	$ 78,046
Map Engineer	1	$ 61,958
Surveyor	1	$ 45,871
Surveyor Helper	1	$ 34,073
Mine Geology Manager	1	$ 179,933
Geologist	2	$ 61,958
Chief of Resource Geology	1	$ 78,046
Resource Geologist	1	$ 72,683
Geotech Engineer	1	$ 78,046
Sampler	3	$ 25,941
Grade Control Helper	2	$ 29,588
SHE - Helper	3	$ 29,588
Total Staff Personnel	29	$ 1,850,780

Costs for other activities that will be required for mining at Gediktepe have also been applied to the overall mining cost. These activities include: topsoil removal, haul road construction and also blast hole assaying for ore control. The cost for these ancillary activities is provided in Table 21-3.

Table 21-3
Life of Mine Additional Gediktepe Mining Costs

	Cost/Unit		#Units	Cost
Assaying Cost	0.49	$/ore ton	24,915,000	$12,208,350
Topsoil Removal	3.22	$/m^3	363,800	$1,170,223
Haul Road Surface Construction	46.34	$/m	17,502	$811,098
Haul Road Cut/Fill	3.65	$/m^3	204,521	$745,573
			Total	$14,935,245

Note: $901,081 of these additional costs is in pre-production and is capitalized.
The remaining $14,034,164 is included in operating costs.

Not including pre-production mining which is capitalized, the total mining cost for the entire mine life inclusive of supervision, roads, clearing, and mining is $221,126,510. Dividing by the total material moved over the mine life of 152,565 ktons results in a total mining cost per tonne of total material of $1.45/tonne.

21.1.2 Processing Operating Costs

Operating costs for ore processing at Gediktepe were developed by RDI. Two sets of ore processing costs were provided: 1) oxide ore processing, and 2) sulfide ore processing. These costs were determined based on reagent, energy and consumable costs obtained in Turkey and the personnel costs are based on salaries in the country. The total energy requirement for the plant was estimated from the demand KW determined by GR Engineering Services in the basic engineering for the project. The costs are provided in Tables 21-4 to 21-6.

Table 21-4
Ore Processing Variable Costs

Parameter for Oxide Ore	Cost $/tonne
Reagents	2.91
Repair & Maintenance Supplies (5% of Purchased Equipment Cost)	0.52
Wear Items (0.01 kg/t)	0.04
Electric Power (7.2 c/kw for 2887 kw)	1.66
Heavy Mobile Equipment Operation	0.04
Total Variable cost for Oxide ore processing	**5.16**
Parameter for Sulfide Ore	Cost $/tonne
Reagents	4.72
Repair & Maintenance Supplies (5% of Purchased Equipment Cost)	0.43
Electric Power (7.2 c/kw for 11,727 kw)	3.12
Heavy Mobile Equipment Operation	0.04
Sundry Costs (laboratory, safety etc.)	0.50
Tailings Operating Costs, from SRK	0.10
Total Variable cost for Sulfide ore processing	**8.90**

The sum of the costs in tables 21-4 through 21-6 will not be equal to the processing costs presented in table 21-1. This is because during years 3 and 4 when sulfide and oxide ores are being processed simultaneously, the sulfide personnel positions that have comparable oxide personnel positions are expected to cover both sulfide and oxide processes.

Table 21-5
Oxide Ore Processing, Labor Costs

Category	No.	Yearly Salary including Benefits	Total Cost/Tonne
Salaried			
Process Manager	1	179,933	179,933
Plant Superintendent	1	98,423	98,423
Senior Engineer	4	67,032	268,126
Shift Supervisor	6	48,263	289,575
Foreman	2	48,263	96,525
Senior Metallurgist	1	72,683	72,683
Met Lab Technician	1	45,871	45,871
Maint. Superintendent	1	98,423	98,423
Chief Electrical Engineer	1	67,031	67,031
Electrical&Automation Engineer	1	61,958	61,958
Instrumentation&Automation Technician	2	56,157	112,314
Senior Maint. Engineer	1	67,031	67,031
Senior Maint. Planner	1	64,350	64,350
Maint. Planner	1	42,900	42,900
Clerk	3	34,856	104,569
Mobile Eq. Maint. Engineer	1	61,958	61,958
Mobile Eq. Maint. Planner	1	42,900	42,900
Maintenance Foreman	1	48,263	48,263
Maintenance Shift Supervisor	4	48,263	193,050
Mobile Eq. Maintenance Shift Supervisor	1	48,263	48,263
Sub-total Cost	35		2,064,146
Subtotal Salary Cost/Tonne at 3,000 tpd processing throughput			1.97
Hourly Personnel			
Front End Loader Operation	4	49,249	196,997
Crusher Operator	4	29,588	118,353
Plant Operator	12	29,588	355,060
Crusher Helper	4	25,941	103,765
Plant Helper	4	25,941	103,765
Refinery Operator	2	29,888	59,776
Laborer	1	25,941	25,941
Sub-Total	31		963,658
Assay Lab			
Chemist	1	61,471	61,471
Assayer	4	34,856	139,424
Sample Preparation	8	25,941	207,531
Sub-Total	13		408,426
Plant Maintenance			
Mechanic / Welder	8	40,216	321,728
Mechanic Helper	4	29,588	118,353
Electrician	8	40,216	321,728
Maintenance Planner	0	61,958	0
Instrumentation Technician	0	56,157	0
Laborer	4	25,941	103,765
Sub-Total	24		865,575
Total Hourly Personnel	68		2,237,659
Subtotal Hourly Cost/Tonne at 3,000 tpd processing throughput			2.13

*Note: During years 3 and 4 when both oxide and sulfide processing are concurrent, redundant positions in oxide and sulfide are not double counted.

Table 21-6
Sulfide Ore Process Personnel Costs

Category	No.	Yearly Salary including Benefits	Total Cost,$
Salaried			
Process Manager	1	179,933	179,933
Plant Superintendent	2	98,423	196,847
Chief Engineer	2	67,032	134,063
Process Engineer	8	72,683	581,464
Shift Supervisor	4	48,263	193,050
General Foreman	1	58,988	58,988
Senior Metallurgist	1	72,683	72,683
Metallurgist	1	67,321	67,321
Met Lab Technician	1	45,871	45,871
Maint. Superintendent	1	98,423	98,423
Chief Electrical Engineer	1	67,031	67,031
Chief Mechanical Engineer	1	67,031	67,031
Electrical&Automation Engineer	2	61,958	123,916
Instrumentation&Automation Technician	4	56,157	224,628
Senior Maint. Engineer	2	67,031	134,063
Senior Maint. Planner	1	64,350	64,350
Maint. Planner	1	42,900	42,900
Maintenance Foreman	1	48,263	48,263
Maintenance Shift Supervisor	4	48,263	193,050
Clerk	3	34,856	104,569
Mobile Eq. Maint. Chief Engineer	1	67,031	67,031
Mobile Eq. Maint. Planner	1	61,958	61,958
Mobile Eq. Maintenance Shift Supervisor	2	48,263	96,525
Sub-total	46		2,923,958
Subtotal Salary Cost/Tonne at 6,500 tpd processing throughput			1.29
Hourly Personnel			
Front End Loader Operation	4	49,249	196,997
Control Room Operators	4	40,216	160,864
Crusher Operator	8	29,588	236,707
Grinding Operator	8	29,588	236,707
Flotation Operator	8	29,588	236,707
Filter Operator	8	29,588	236,707
Dryer Operator	4	29,588	118,353
Plant Helpers	8	25,941	207,531
Sub-Total	48		1,630,572
Assay Lab			
Assayer	8	34,856	278,848
Samplers	8	25,941	207,531
Chemist	2	61,471	122,942
Laborers	2	25,941	51,883
Sub-Total	20		661,203
Plant Maintenance			
Mechanics	16	40,216	643,456
Electricians	12	40,216	482,592
Laborers	4	25,941	103,765
Mobile Eq. Maintenance Mech.	2	56,157	112,314
Mobile Eq. Maintenance Elec.	1	56,157	56,157
Sub-Total	35		1,398,284
Total Hourly Personnel	103		3,690,059
Subtotal Hourly Cost/Tonne at 6,500 tpd processing throughput			1.62

21.1.3 Site Wide General and Administrative Costs

The costs attributed to project site G&A are the remaining project costs that could not be directly applied to mining or processing. These costs include an administrative staff and costs associated with their operations, forestry permits and fees, camp operating costs, employee transport, water treatment costs, and access road maintenance. The administrative staff is comprised of 48 salary positions and 17 hourly positions. The following categories are included in the administration costs: management, human resources, finance, information technology, procurement, community relations, environmental, health and safety, and engineering. An additional $1.75 million per year has been included in the G&A costs to cover any unforeseen costs during the mining operation. The total life of mine G&A costs are presented in Table 21-7.

Table 21-7
Life of Mine General and Administrative Costs

Cost Category	Cost ($000's)
Administrative Labor Costs	57,559
Administrative Costs (Not Labor)	15,890
Water Treatment	650
Employee Transportation	14,640
Access Road Maintenance	330
Camp Operating Costs	21,280
Unforeseen Mine Op. Costs	22,750
Project Permitting and Fees	52,562
Total	185,661

21.2 Capital Costs

The expected capital costs over the Gediktepe mine life are estimated to total $266.7 million. These costs include: processing plant equipment, leach pad and tailings storage facilities, project infrastructure, the cost of pre-production mining, and reclamation. The estimated capital costs over the life of mine by category are provided in the following Table 21-8.

Table 21-8
Gediktepe Capital Costs, U.S. Dollars x 1000

Cost or Income Item		Totals Project Life Costs x1000	Capital Costs in Years Shown, USD x 1000										
			Preprod -2	Preprod -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 13
Capital Costs													
Initial Capital Costs													
Plant		$ 46,381.2	$ -	$ 46,381.2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure		$ 41,972.5	$ 6,089.30	$ 35,883.16	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Site Investigation and Proj. Eng.		$ 6,900.0	$ 6,100.0	$ 800.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Private Land purchase		$ 1,600.0	$ -	$ 1,600.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-Production Mining		$ 3,153.9	$ -	$ 3,153.9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Contingency	Avg. 20%	$ 19,711.2	$ 2,210.2	$ 17,501.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Subtotal		$ 119,718.7	$ 14,399.5	$ 105,319.2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital Costs													
Plant		$ 81,052.7	$ -	$ -	$ 27,058.3	$ 49,994.4	$ -	$ -	$ 2,000.0	$ -	$ 2,000.0	$ -	$ -
Infrastructure		$ 23,336.5	$ -	$ -	$ 3,587.7	$ 6,233.3	$ 2,180.4	$ 1,434.8	$ 2,282.3	$ 3,686.4	$ 659.9	$ 3,271.7	$ -
Site Investigation and Proj. Eng.		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation		$ 17,661.7	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,685.5	$ -	$ -	$ -	$ 14,976.2
Contingency	Avg. 22%	$ 26,779.8	$ -	$ -	$ 5,751.2	$ 11,076.4	$ 683.8	$ 401.9	$ 1,969.3	$ 1,047.9	$ 557.3	$ 947.6	$ 4,344.3
Subtotal		$ 148,830.7	$ -	$ -	$ 36,397.3	$ 67,304.1	$ 2,864.1	$ 1,836.7	$ 8,937.2	$ 4,734.3	$ 3,217.1	$ 4,219.3	$ 19,320.5
Total		$ 268,549.4	$ 14,399.5	$ 105,319.2	$ 36,397.3	$ 67,304.1	$ 2,864.1	$ 1,836.7	$ 8,937.2	$ 4,734.3	$ 3,217.1	$ 4,219.3	$ 19,320.5

21.2.1 Processing Plant Capital Costs

The capital costs were developed by GRES for two processing plants: (1) 3,000 mtpd heap leaching of the oxide ore, and (2) 6,500 mtpd flotation of the sulfide ore to produce two concentrates, namely copper concentrate and zinc concentrate.

The following assumptions were made to develop the costs for the project:

1. The three-stage crushing circuit was sized to crush 8,000 mtpd of ore and would be capable of treating both oxide and sulfide ores simultaneously.

2. The costs for heap leach pads for oxide ore and tailings storage for sulfide ore tailings were not included in the capex for the processing plants. They are included in the infrastructure capital.

3. Corporate costs including health, safety and human resource costs were not included in this section. These costs are provided later in the owner's G&A costs.

4. The capex was estimated using equipment quotations in Turkey for over 80% of the plant equipment. The bid packages from two to three bidders for each section (mills, crushers, flotation, thickeners, etc.) were reviewed and the most suitable bidder based on technical and cost basis was selected.

5. The capex includes withholding tax and equipment insurance and transportation.

The capital cost for the oxide processing operation (not including the heap leach pad foundation) was prepared using basic engineering to a Class 3 estimate level and is discussed in Section 17.1. The cost was estimated to be $51,245,821. The breakdown of that cost is provided on Table 21-9. The initial capital cost for the sulfide plant was prepared using basic engineering to a Class 4 estimate level and is discussed in Section 17.2. It was estimated to be $ 91,880,244. The breakdown of that cost is summarized on Table 21-10.

A sustaining capital cost of $4 million is expected to maintain the sulfide processing plant for 10 years of operation. The costs are estimated to be $2 million in year 5 and $2 million in year 7. A contingency of $698,326 has been added to the $4 million dollar sustaining capital split evenly between years 5 and 7 which increases the final estimate of sulfide plant sustaining capital to $4,698,326.

Table 21-9
Capital Cost for 3,000 Tonnes/Day Heap Leach Process
(Crush/Screen/Agglomerate/Heapleach)

Type	Class	Supply Cost $	Install Cost $	Install Manhours	Freight Cost $	Subtotal Cost $	Contingency Cost $	Project Total Cost $
DIRECT COSTS	**Buildings**	353,737	80,134	1,646	177,336	611,207	61,121	672,328
	Civil works	191,179	1,903,630	38,565	-	2,094,809	209,481	2,304,290
	Construction equipment	324,333	447,211	9,937	164,169	935,713	93,571	1,029,285
	Earthworks	3,632,363	162,138	24,273	-	3,794,501	1,171,568	4,966,070
	Electrical installations	3,066,947	700,165	13,892	118,377	3,885,489	388,549	4,274,038
	Mechanical equipment	10,844,389	925,490	16,652	265,958	12,035,838	941,241	12,977,078
	Owners Equipment	887,218	-	-	-	887,218	88,722	975,940
	Piping	1,658,876	596,186	12,227	38,839	2,293,901	229,390	2,523,291
	Platework	1,681,656	427,166	8,776	75,292	2,184,114	218,411	2,402,525
	Structural steel	2,560,360	1,043,168	21,658	196,064	3,799,592	379,959	4,179,551
DIRECT COSTS Total		25,201,059	6,285,288	147,627	1,036,035	32,522,382	3,782,013	36,304,396
INDIRECT COSTS	**Commissioning**	29,060	305,066	2,975		334,126	33,413	367,538
	Engineering design	-	2,588,176	27,083		2,588,176	258,818	2,846,993
	Initial fills	351,980	-	-	75,330	427,310	42,731	470,041
	Insurance Spares	1,342,309	-	-	80,539	1,422,848	93,240	1,516,087
	Project and procurement management	14,000	1,278,143	11,660		1,292,143	129,214	1,421,357
	Supervision and Construction Management	147,225	1,898,261	28,476		2,045,486	204,549	2,250,035
	Temporary construction facilities	165,900	19,474	400	4,633	190,006	19,001	209,007
	Vendor Commissioning	32,350	60,252	517		92,602	9,260	101,862
INDIRECT COSTS Total		2,082,824	6,149,371	71,111	160,501	8,392,696	790,224	9,182,921
	Customs Duty (20% of mech. Equip)	2,168,878	-	-	-	2,168,878	188,248	2,357,126
	Withholding Tax(20% of Install Cost)	-	2,486,932	-	-	2,486,932	914,448	3,401,379
Grand Total		**29,452,761**	**14,921,590**	**218,738**	**1,196,536**	**45,570,888**	**5,674,934**	**51,245,821**

Table 21-10
Capital Cost for 6,500 mtpd Sulfide Floatation Process

Type	Class	Supply Cost $	Install Cost $	Install Manhours	Freight Cost $	Subtotal Cost $	Contingency Cost $	Project Total Cost $
DIRECT COSTS	**Buildings**	50,000	4,966	102	166,967	221,933	22,193	244,127
	Civil works	495,639	4,507,511	93,479	2,333	5,005,482	539,004	5,544,487
	Construction equipment	254,234	837,606	18,847	162,000	1,253,840	167,007	1,420,847
	Earthworks	2,655,938	45,000	-		2,700,938	895,331	3,596,269
	Electrical installations	5,779,412	1,277,787	25,353	226,570	7,283,768	1,028,676	8,312,444
	Mechanical equipment	19,408,484	2,624,016	40,500	593,755	22,626,255	2,998,017	25,624,272
	Owners Equipment	296,992	-	-	-	296,992	59,398	356,390
	Piping	2,730,884	1,490,905	30,578	91,656	4,313,445	1,086,365	5,399,810
	Platework	1,481,533	609,743	12,527	88,244	2,179,521	296,441	2,475,962
	Structural steel	4,049,124	1,702,259	35,342	346,481	6,097,864	945,167	7,043,030
DIRECT COSTS Total		37,202,240	13,099,792	256,727	1,678,006	51,980,038	8,037,599	60,017,638
INDIRECT COSTS COSTS	**Commissioning**	38,400	584,399	5,220		622,799	179,160	801,959
	Engineering design	-	4,258,579	45,855		4,258,579	851,716	5,110,295
	Initial fills	906,460	-	-	92,245	998,705	199,741	1,198,446
	Insurance Spares	2,792,316	-	-	167,539	2,959,855	295,985	3,255,840
	Project and procurement management	-	2,591,250	24,100		2,591,250	518,250	3,109,500
	Supervision and Construction Management	727,680	4,111,438	81,375		4,839,118	967,824	5,806,941
	Temporary construction facilities	263,000	189,868	3,900	4,633	457,500	72,513	530,014
	Vendor Commissioning	120,220	155,155	1,001		275,375	58,569	333,944
INDIRECT COSTS Total		4,848,076	11,890,688	161,451	264,417	17,003,181	3,143,757	20,146,939
	Customs Duty (20% of mech. Equip)	3,881,697	-	-	-	3,881,697	599,603	4,481,300
	Withholding Tax(20% of Install Cost)	-	4,998,096	-	-	4,998,096	2,236,271	7,234,368
Grand Total		**45,932,012**	**29,988,577**	**418,177**	**1,942,423**	**77,863,013**	**14,017,231**	**91,880,244**

21.2.2 Mining Capital Costs

It is expected that Gediktepe will be mined by a contract miner and therefore no capital cost is allocated for the purchase of mine mobile equipment. The mining cost that will be incurred in pre-production for waste stripping is applied as a capital cost.

The preproduction stripping cost includes moving 349 ktonnes of material at the contractor mining rate. The costs of road construction, topsoil removal, blast hole assays, and one year of the Polimetal mining staff are included which results in a total mine capital estimate of $3,153,860 million USD before a 25% contingency is added.

21.2.3 Heap Leach Pad and Tailings Storage Facility Capital Costs

SRK developed a capital cost estimate for the HLP and TSF that includes site preparation; earthworks; geosynthetic liners; overliner/overdrain and piping; mechanical equipment; engineering, and contingency. An allowance for Engineering, Procurement, and Construction Management (EPCM) costs has been added to the SRK estimate at 10% of the total costs. Also, $555,000 has been added to the SRK heap leach pad cost estimate to cover 12 months of construction quality assurance.

Based on the designs and cost assumptions, SRK developed capital costs for the HLP and the TSF that are summarized in Table 21-11. The capital cost estimates also include a contingency which was recommended by SRK for PFS level design. SRK operating cost estimates for pumping sulfide tails to the TSF were included in Table 21-4.

Table 21-11
Heap Leach Pad and Tailings Storage Facility
Capital Expenditure Estimate

Description	Heap Leach Pad and Solution Ponds (US$000's)	TSF (US$000's)
Site Preparation	$ 956.0	$ 2,259.8
Earthworks	$ 5,780.5	$ 12,080.5
Underdrain	$ 234.4	$ 1,059.1
Geosynthetics	$ 880.8	$ 5,564.5
Overliner	$ 2,218.0	$ 1,113.3
Diversion Channel	$ 352.0	$ 1,751.4
Equipment	$ 1,077.5	$ 2,003.6
EPCM, CQA	$ 1,704.9	$ 2,583.2
Subtotal of all Capital Items	**$ 13,204.2**	**$ 28,415.6**
Contingency on Capital Cost	$ 4,084.2	$ 8,528.7
Total Capital Cost	**$ 17,288.4**	**$ 36,944.3**

21.2.4 Site Infrastructure Cost

The infrastructure capital costs required for Gediktepe also include the following items:

1) Power transmission line
2) Access road
3) Water treatment plant
4) Freshwater pond
5) Runoff water diversion channels
6) Technical and administrative buildings
7) Construction of a camp for technical staff and some administration staff
8) Administrative Supplies

A breakout of the project infrastructure capital costs is provided on Table 21-12.

Table 21-12
Gediktepe Project Infrastructure Capital Costs

Category	Description	Cost $USD
Power Transmission Line	33 km + Substation	$3,310,718
Access Bypass Road	3.32 km	$1,544,832
Water Treatment Plant		$4,345,687
Freshwater Pond		$2,600,000
Water Diversion Channels	11.5 km	$1,245,763
Buildings		
Camp Cost	Dorms, Cafeteria	$2,999,998
Admin Building Cost	offices, lab, storage, core shed, first aid	$4,339,377
Administrative Supplies	Environmental Supplies, Weighbridge, Vehicles	$4,402,482
Total		**$24,788,857**
Contingency @ 25%		**$6,197,214**

*It is expected that $1,099,686 of the power transmission line cost will be reimbursed by the government in year two of mining

21.2.5 Reclamation

Reclamation costs for the Heap Leach Pad and the Tailing Storage Facility have been provided by SRK. Those costs are included in the project financial analysis as late stage capital expenditures in years 5 and 13.

Costs have been included for back filling the mine pit with waste following mining to an elevation that will prevent pit lakes from forming.

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 decommissioning of the plant facilities.

No costs have been included for continuing treatment of runoff water.

A summary of the reclamation costs is provided in Table 21-13.

Table 21-13
Estimated Gediktepe Reclamation Costs

Reclamation Action	Total $USD	Year 5 $USD	Year 13 $USD
Leach Pad Closure	2,685,492	2,685,492	
Final Pit Backfill	2,879,015		2,879,015
TSF Closure	9,070,412		9,070,412
Site Rehabilitation of 379 ha.	3,026,733		3,026,733
Total:	17,661,652	2,685,492	14,976,160
Contingency :	5,251,910	907,573	4,344,337

21.2.6 EPCM Costs

EPCM costs were included in the capital costs presented above, but are summarized again in Table 21-14. Except for the processing plant which had EPCM costs included in the estimate, EPCM costs of site infrastructure components were estimated at 10% of installed equipment cost. A 25% contingency was applied to the EPCM costs estimated at 10% to reflect the accuracy of the project's capital cost estimates.

Table 21-14
Estimated EPCM Costs for Gediktepe Infrastructure

Infrastructure Item	Cost $000's
Oxide Processing Plant	6,116
Heap Leach Pad	1,705
Sulfide Mill	12,146
Tailings Storage Facility	2,583
Water Treatment Plant	358
Water Diversions	113
Access Bypass Road	140
HLP closure	224
TSF Closure	756
Total	24,141
Contingency	4,492

21.2.7 Site Investigation and Engineering Work

An additional $6.9 million is estimated for site investigations, studies, and engineering to move the project into detailed design stage. The cost of site investigation and engineering work is provided in Table 21-15.

Table 21-15
Site Investigation and Engineering Work

Area of Focus	pre-prod. Yr -2 $000's	pre-prod. Yr -1 $000's
Site Wide water management	670	0
General minesite engineering	480	0
Tailing storage/freshwater pond	450	0
Waste rock management	400	0
EIA Revision	0	300
Definitive Feasibility Study	600	0
Detailed Slope Stability Study w/oriented core drilling	0	500
Phase 4 Drilling	3,500	0
Total	6,100	800
Contingency	650	200

21.2.8 Land Purchase

Polimetal estimates that the cost to buy the remaining private lands within the project footprint will be $1,600,000. A 25% contingency has been added to this cost.

21.2.9 Accuracy of Estimate

Pre-Feasibility level Engineering has been completed and supported by budgetary equipment quotes and contractor rates to produce an estimate of project capital costs.

The accuracy of the cost estimates for pre-production mining, general site capital cost items, and the oxide processing circuit is expected to be in the +20% to -15% range of actual project costs. These cost estimates fall into the class 3 cost estimate category of the 5 class estimate guideline put forth by the American Association of Cost Engineers(AACE).

The accuracy for the TSF and HLPF costs is expected to be lower; in the +25% to -18% range of actual project costs. The accuracy for these two items is lower than the other capital cost items because additional geotechnical drilling is required to add detail to the designs. These cost estimates fall in between the class 3 and class 4 cost estimate categories of the AACE cost estimate guidelines.

The accuracy for the oxide processing circuit is expected to be in the +20% to -15% range and the accuracy for the sulfide processing circuit is expected to be in the +30% to -20% range. The AACE estimate classes of the oxide processing capital costs and the sulfide processing capital costs are class 3 and class 4 respectively.

The contingencies applied to each cost area are provided in Table 21-16. The amount of contingency applied to the cost centers corresponds with the accuracy of the estimates. Contingencies between 12% and 34% have been applied.

Table 21-16
Capital Cost Center Applied Contingency

Cost Area	Conting.
Pre-Production Mining	25%
Oxide Plant	12%
Sulfide Plant	18%
Private Land Purchase	25%
Site Investigation and Engineering	25%
Non TSF/HLP Infrastructure	25%
Non TSF/HLP Reclamation	25%
HLP Construction	32%
TSF Construction	30%
HLP Reclamation	34%
TSF Reclamation	32%

22.0 ECONOMIC ANALYSIS

The Gediktepe project economic analysis is a conventional discounted cash flow model that is based on the mine plan and estimated project costs that are presented in previous sections. The analysis calculates annual cash flow projections over the life of mine as it is currently understood and incorporates Turkish taxes and permit fees. The analysis is based on 2015 fourth quarter U.S. dollars.

The financial model is summarized with three metrics:

1) Discounted and Non-discounted net present value
2) Internal rate of return or Return on Investment
3) Non-discounted payback period (years of production required to pay back the initial investment).

The base case metal prices for the financial analysis are provided in Table 22-1 and the base case project results are summarized on Table 22-2.

Table 22-1
Base Case Metal Prices

Metal	Metal Price
Gold	$1,250/oz.
Silver	$18.25/oz.
Copper	$2.75/lb
Zinc	$1.00/lb

Table 22-2
Base Case Financial Analysis Results

Metric	Results
After Tax Undiscounted Cash Flow	$745.2 Million
After Tax NPV@5%	$475.2 Million
After Tax NPV@10%	$308.7 Million
After Tax IRR	46.5%
Payback Period, From Process Plant Start	2.5 Years

The start date for the economic analysis is assumed to be the two years prior to oxide process plant startup. The process plant startup represents the start of Year 1. All discounted metrics are discounted to the beginning of Year -2.

All values are expressed in U.S. dollars, unless otherwise noted. For cost estimates received in Turkish Lira, an exchange rate of 3.0 TL / $USD was applied. A flat exchange rate over the project life was applied. Costs are estimated in 2015 Q4 dollars.

22.1 Revenue

Revenue is calculated as the value of the payable metal less the smelting and refining and freight charges. The estimated treatment, refining, and freight terms are on Table 22-3.

Table 22-3
TCRC Terms

Oxide Ore	
Gold Pay: 99%	Silver Pay: 98%
Gold Transport: $5.00/oz.	Silver Transport: $0.50/oz.
Sulfide Ore	
Copper Conc Terms Cu Grade of 30.95%	Zinc Conc Terms Zn Grade of 54.3%
Copper: pay lesser of 96.5% or Cu content less 1%	Zinc: Pay lesser of 85% of Zn content or Zn content less 8%
Gold: pay lesser of 90% or Au content less 1 g/t	Gold: Pay 70% after (1 g/t deduct from Au content)
Silver: pay lesser of 90% or Ag content less 30 g/t	Silver: Pay 70% after (93.31 g/t deduct from Ag content)
Treatment Charge: $85.00/dry tonne	Treatment Charge: $259.80/dry tonne
Refining Charges:	escalator of $0.10/dollar zinc price above $1850/t
cu: $0.085/lb	(if Zn>$1850/t, treatment = $259.80 + (Zn pr-1850)*.1)
Au: $5.00/oz	de-escalator of $0.14/dollar zinc price below $1850/t
Ag: $0.40/oz	(if Zn<$1850/t, treatment = $259.80 - (1850-Zn pr)*.14)
Assume conc. contains 9% water	Assume conc. contains 9% water
Ocean Freight: $40.00/wet tonne	Ocean Freight: $45.00/wet tonne
Port Charge: $10.00/wet tonne	Port Charge: $10.00/wet tonne
Land Freight: $14.10/wet tonne	Land Freight: $14.10/wet tonne
Insurance: 0.088% of CIF	Insurance: 0.11% of CIF

Figure 22-1 illustrates the cumulative recoverable metal produced over the mine life. The smelting and refining loses have not been incorporated into the figure.

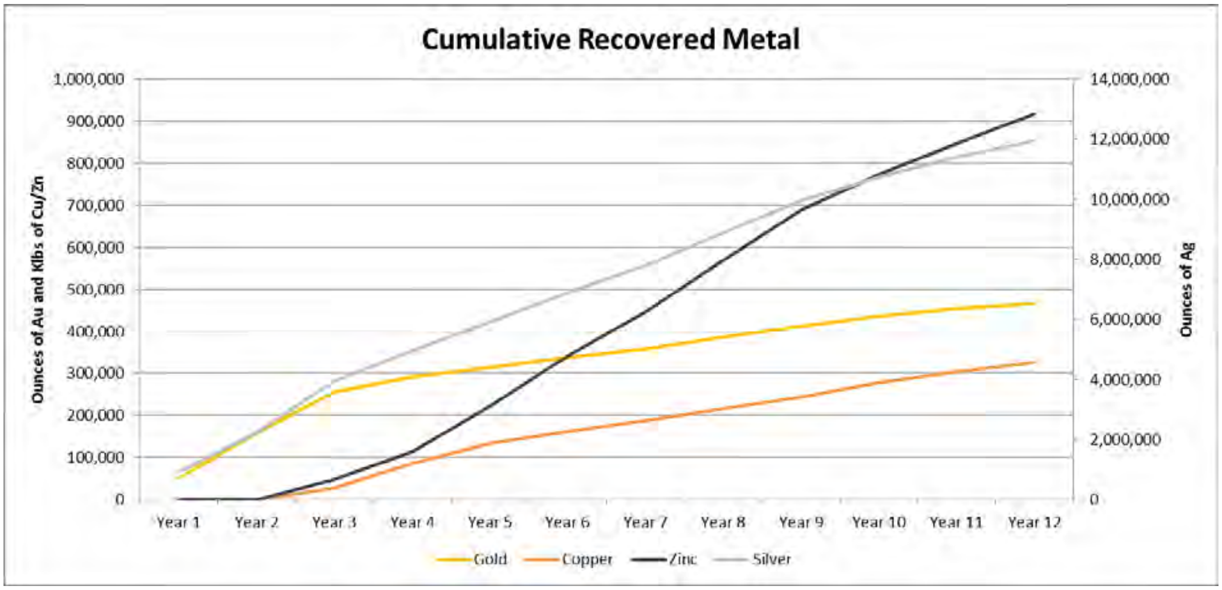


Figure 22-1
Cumulative Recovered Metal

The combined value of the project's payable metal at the base case prices given in Table 22-1 is $2,293.73 million USD. Total smelting, and refining, and freight fees are expected to be $391.42 million USD. The resulting net smelter return is treated as gross revenue in the cash flow analysis and amounts to $1,902.31 million USD over the mine life.

22.2 Capital Cost

The details of the capital cost estimate were presented in Section 21. A summary of the initial and sustaining capital costs are provided in Table 22-4.

Table 22-4
Gediktepe Capital Costs, U.S. Dollars x 1000

Cost or Income Item	Totals Project Life Costs x1000	Capital Costs in Years Shown, USD x 1000										
		Preprod -2	Preprod -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 13
Capital Costs												
Initial Capital Costs												
Plant	$ 46,381.2	$ -	$ 46,381.2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure	$ 41,972.5	$ 6,089.30	$ 35,883.16	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Site Investigation and Proj. Eng.	$ 6,900.0	$ 6,100.0	$ 800.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Private Land purchase	$ 1,600.0	$ -	$ 1,600.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-Production Mining	$ 3,153.9	$ -	$ 3,153.9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Contingency Avg. 20%	$ 19,711.2	$ 2,210.2	$ 17,501.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Subtotal	$ 119,718.7	$ 14,399.5	$ 105,319.2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital Costs												
Plant	$ 81,052.7	$ -	$ -	$ 27,058.3	$ 49,994.4	$ -	$ -	$ 2,000.0	$ -	$ 2,000.0	$ -	$ -
Infrastructure	$ 23,336.5	$ -	$ -	$ 3,587.7	$ 6,233.3	$ 2,180.4	$ 1,434.8	$ 2,282.3	$ 3,686.4	$ 659.9	$ 3,271.7	$ -
Site Investigation and Proj. Eng.	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation	$ 17,661.7	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,685.5	$ -	$ -	$ -	$ 14,976.2
Contingency Avg. 22%	$ 26,779.8	$ -	$ -	$ 5,751.2	$ 11,076.4	$ 683.8	$ 401.9	$ 1,969.3	$ 1,047.9	$ 557.3	$ 947.6	$ 4,344.3
Subtotal	$ 148,830.7	$ -	$ -	$ 36,397.3	$ 67,304.1	$ 2,864.1	$ 1,836.7	$ 8,937.2	$ 4,734.3	$ 3,217.1	$ 4,219.3	$ 19,320.5
Total	$ 268,549.4	$ 14,399.5	$ 105,319.2	$ 36,397.3	$ 67,304.1	$ 2,864.1	$ 1,836.7	$ 8,937.2	$ 4,734.3	$ 3,217.1	$ 4,219.3	$ 19,320.5

22.3 Operating Cost

The average mining cost per tonne ore processed is: $8.87 per tonne. This equates to an average operating cost of $25.83/tonne of oxide ore and $28.20/tonne of sulfide ore. The total cash operating cost includes: mining, processing, supporting facilities, and site wide G&A. The details of the operating cost estimate were presented in Section 21 and are summarized below in Table 22-5.

Table 22-5
Gediktepe Operating Cost by Category

OPCOST Category	Unit Cost	Units	Total Cost ($000's)
Mining	1.45	$/tonne material	221,126.5
Oxide Ore Processing	9.51	$/tonne ore	30,640.8
Sulfide Ore processing	11.88	$/tonne ore	257,678.7
Site Wide G&A	7.45	$/tonne ore	185,661.4
		Total:	695,107.5

22.4 Royalties, Depreciation, and Depletion

The only royalties applicable to the project are to the Turkish government. There are no royalties payable to previous property owners or private parties.

There is a 2.6% royalty payable on: the revenue net of smelting from the project less processing costs less site G&A. The basis for this royalty rate is described in Chapter 19. This royalty is estimated to be $37.6 million USD over the mine life at base case metal prices.

Depreciation is calculated using the declining balance method starting with the first year of ore production. The initial capital and sustaining capital used a 10 year life and 20% rate except for structures which used a 50 year life and a 4% rate. Any remaining asset value which was not depreciated by the last year of production is fully depreciated in the last year of production.

Depletion of the land and concession costs is applied at the rate at which the resource is mined.

22.5 Taxation

Taxable income for corporate tax purposes is defined as metal revenues minus operating expenses, royalties, depreciation, and depletion.

The Turkish corporate tax rate is 20% of taxable income. Turkish investment incentives are expected to reduce the payable tax by 70% during the first 3 years of production. The investment incentive lowers the effective corporate tax to 16% over the mine life. A total of $155.9 million USD is expected to be paid in corporate tax over the life of mine.

22.6 Results

The financial model results are presented in terms of NPV, IRR, and payback period (in years of ore production required to make the cumulative cash flow positive). These economic indicators are presented on both a pre-tax and after-tax basis. The NPV is presented both undiscounted and at 5%, 10% and 15% discount rates, as shown in Table 22-6. On an after-tax basis, the project has an $NPV_{5\%}$ of $475.2 million, an IRR of 46.5%, and a payback period of 2.5 years.

Table 22-6
Financial Model Results, Pre-Tax and Post-Tax

Parameter	Unit	Pre-Tax Value	Post Tax Value
Undisc. Cash Flow	$M	901.03	745.17
NPV5%	$M	576.5	475.2
NPV10%	$M	377.1	308.7
NPV15%	$M	250.1	202.4
IRR	%	51.0%	46.5%
Payback Period	Yrs	2.4	2.5

Note: Base Case Metal Prices Used

The undiscounted Cash Flows generated by the project financial model are provided graphically in Figure 22-2. A summary of the financial model is presented in Table 22-8.

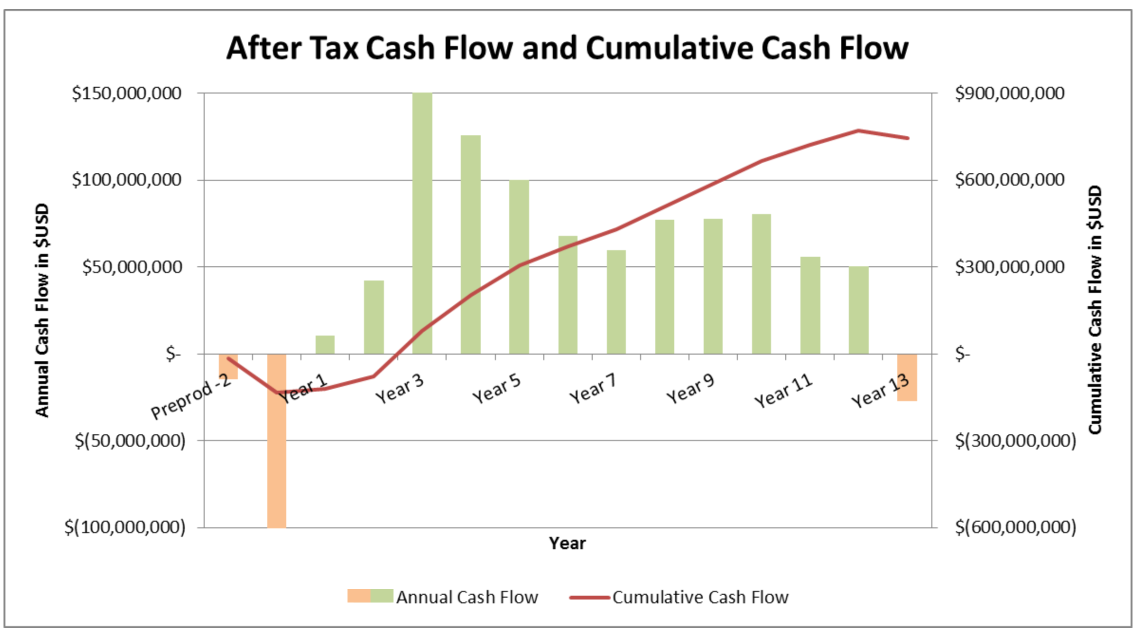


Figure 22-2
Undiscounted After Tax Cash Flow

Figure 22-3 presents the gross revenue and operating costs experienced by the project on an annual basis.

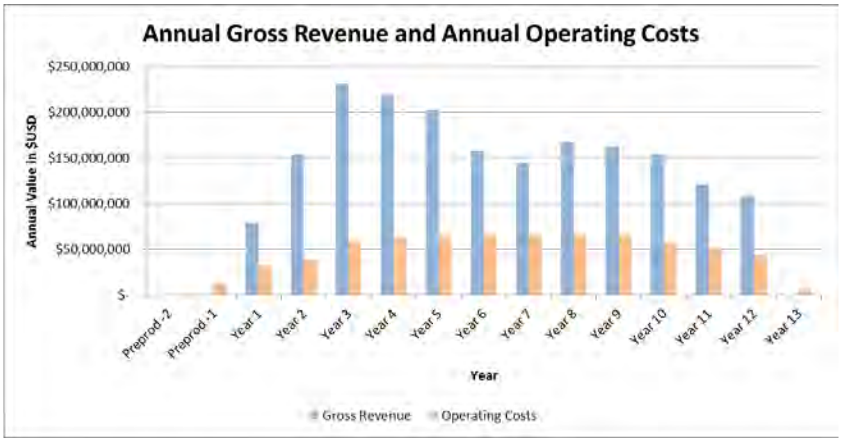


Figure 22-3
Annual Gross Revenue and Annual Operating Costs

22.7 Sensitivity

The economic sensitivity of the project was evaluated with respect to OPEX, CAPEX, and metal prices between -30% and +30% of the base case values. Change in metal prices is indicative of changes in metal recovery and/or processed head grades.

Financial results appear to be most sensitive to metal prices and least sensitive to changes in operating cost. The project remains economically feasible over the entire range of the sensitivity analysis. A spider graph depicting the results on project IRR by varying the OPEX, CAPEX and metal price inputs (one category at a time) is provided in Figure 22-4. A spider graph depicting the effect on project NPV at a 5% discount rate by varying the OPEX, CAPEX and metal price inputs (one category at a time) is provided in Figure 22-5.

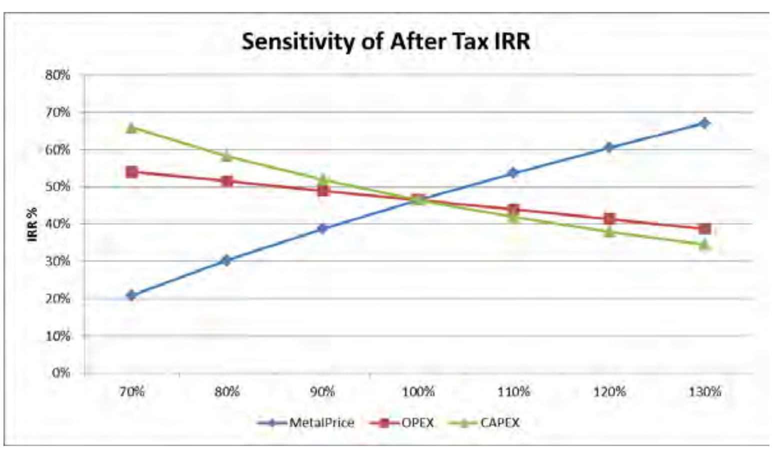


Figure 22-4
Sensitivity of After Tax IRR

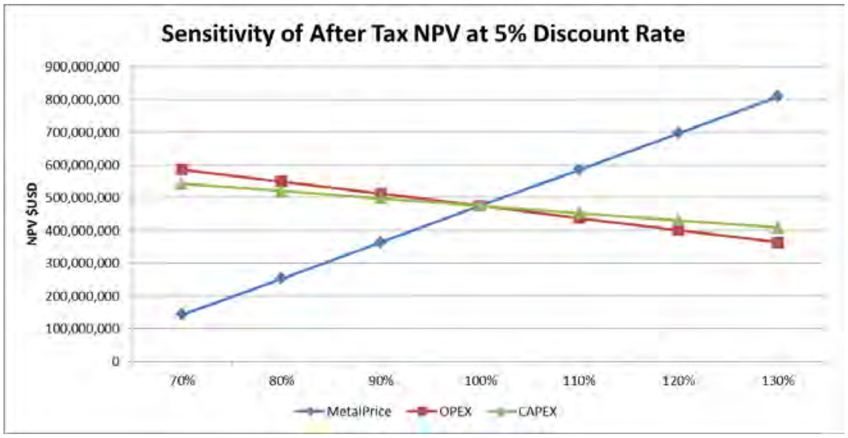


Figure 22-5
Sensitivity of After Tax $NPV_{5\%}$

In response to current volatility of the metal markets Polimetal desired to present a sensitivity of the project economics at metal prices more conservative than the base case prices used. The metal prices used in this conservative evaluation are: $950/oz. Au, $13.50/oz. Ag, $2.25/lb Cu, and $0.80/lb Zn. The economic indicators for the project at these metal prices are presented in table 22-7

Table 22-7
Financial Model Results, Pre-Tax and Post-Tax
Metal Prices Used: $950/oz. Au, $13.50/oz. Ag, $2.25/lb Cu, $0.80/lb Zn

Parameter	Unit	Pre-Tax Value	Post Tax Value
Undisc. Cash Flow	$M	477.10	406.15
NPV5%	$M	288.7	243.8
NPV10%	$M	173.7	144.2
NPV15%	$M	101.5	81.4
IRR	%	31.3%	28.9%
Payback Period	Yrs	3.2	3.3

22.8 Financial Model Summary

A summary of the financial model is presented in Table 22-8.

Table 22-8
Gediktepe Financial Model Summary

Cost or Income Item	Unit Cost or Avg	Units	Sens-itivity Factor	Totals	Time Period														
					Preprod -2	Preprod -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Mine Production																			
Heap Leach Ore		Ktonnes		3,223		92	886	1,048	1,048	149	0	0	0	0	0	0			
Heap Leach Grade Au		gm/t		2.955		1.246	2.146	3.735	2.995	3.051	0.000	0.000	0.000	0.000	0.000	0.000			
Recoverable Leach Grade Au		gm/t		2.453		0.884	1.708	3.157	2.499	2.569	0.000	0.000	0.000	0.000	0.000	0.000			
Heap Leach Grade Ag		gm/t		77.7		32.3	68.4	85.9	76.7	111.1	0.0	0.0	0.0	0.0	0.0	0.0			
Sulfide Mill Ore		Ktonnes		21,692		0	0	379	1,193	2,275	2,275	2,275	2,275	2,275	2,275	2,275	2,275	1,920	
Sulfide Mill Grade Cu		%		0.986		0.000	0.000	0.789	1.246	1.671	1.420	0.767	0.750	0.807	0.809	0.994	0.727	0.793	
Sulfide Mill Grade Zn		%		2.35		0.00	0.00	1.72	1.66	1.64	2.62	2.95	2.55	3.00	2.93	2.05	1.84	2.00	
Sulfide Mill Grade Au		gm/t		0.93		0.00	0.00	0.77	0.90	0.95	0.97	0.95	0.87	1.18	1.06	1.00	0.72	0.65	
Sulfide Mill Grade Ag		gm/t		35.3		0.0	0.0	28.2	34.5	30.8	38.3	40.0	36.1	44.8	42.9	31.6	27.9	25.6	
Waste		Ktonnes		127,650		257	4,878	9,068	9,317	10,576	16,225	16,225	16,225	16,225	15,090	8,328	3,756	1,480	
Total Material		Ktonnes		152,565		349	5,764	10,495	11,558	13,000	18,500	18,500	18,500	18,500	17,365	10,603	6,031	3,400	
Process Plant Production																			
Ore Placed onHeap Leach		Ktonnes		3,223			978	1,048	1,048	149									
Heap Leach Grade Au		gm/t		2.955			2.061	3.735	2.995	3.051									
Recoverable Leach Grade Au		gm/t		2.453			1.631	3.157	2.499	2.569									
Heap Leach Grade Ag		gm/t		77.7			65.0	85.9	76.7	111.1									
Sulfide Mill Ore		Ktonnes		21,692			0	0	1,572	2,275	2,275	2,275	2,275	2,275	2,275	2,275	2,275	1,920	0
Sulfide Mill Grade Cu		%		0.986			0.000	0.000	1.136	1.671	1.420	0.767	0.750	0.807	0.809	0.994	0.727	0.793	0.000
Sulfide Mill Grade Zn		%		2.35			0.00	0.00	1.68	1.64	2.62	2.95	2.55	3.00	2.93	2.05	1.84	2.00	0.000
Sulfide Mill Grade Au		gm/t		0.93			0.00	0.00	0.87	0.95	0.97	0.95	0.87	1.18	1.06	1.00	0.72	0.65	0.000
Sulfide Mill Grade Ag		tm/t		35.3			0.0	0.0	33.0	30.8	38.3	40.0	36.1	44.8	42.9	31.6	27.9	25.6	0.000
Ore Contained Metal																			
Copper		Klbs		471,416			0	0	39,367	83,809	71,220	38,469	37,616	40,475	40,575	49,854	36,463	33,567	0
Zinc		Klbs		1,124,806			0	0	58,112	82,254	131,406	147,958	127,896	150,465	146,954	102,818	92,285	84,657	0
Gold		oz		951,295			64,803	125,839	144,777	83,955	70,583	69,486	63,634	85,943	77,312	72,777	52,370	39,815	0
Silver		oz		32,694,980			2,045,126	2,893,612	4,252,443	2,787,019	2,803,787	2,922,424	2,638,045	3,273,730	3,141,414	2,311,974	2,044,271	1,581,137	0
Metal Recovered to Dore																			
Gold Recovery:	Variable	ounces		254,166			51,277	106,383	84,201	12,305	0	0	0	0	0	0	0	0	0
Silver Recovery:	45%	ounces		3,624,792			920,307	1,302,125	1,162,818	239,542	0	0	0	0	0	0	0	0	0
Sulfide Concentrate Recovered Metal																			
kTonnes Copper Conc.				478				0	40	85	72	39	38	41	41	51	37	34	0
Copper Recovery:	69%	Klbs		326,220				0	27,242	57,996	49,284	26,621	26,030	28,009	28,078	34,499	25,232	23,228	0
Gold Recovery:	17%	oz		110,963				0	7,546	11,926	12,140	11,952	10,945	14,782	13,298	12,518	9,008	6,848	0
Silver Recovery:	12%	oz		3,030,706				0	205,214	277,328	344,866	359,458	324,480	402,669	386,394	284,373	251,445	194,480	0
kTonnes Zinc Conc.				766				0	40	56	89	101	87	102	100	70	63	58	0
Zinc Recovery:	82%	Klbs		916,717				0	47,361	67,037	107,096	120,585	104,235	122,629	119,768	83,797	75,213	68,996	0
Gold Recovery:	16%	oz		101,286				0	6,888	10,886	11,082	10,909	9,991	13,493	12,138	11,426	8,222	6,251	0
Silver Recovery:	22%	oz		5,297,576				0	358,707	484,761	602,814	628,321	567,180	703,852	675,404	497,074	439,518	339,944	0
Operating Costs																			
Mine																			
Owner Staff	$0.16	$/t total		$ 24,060,144	$ -	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ 1,850,780	$ -
Mining Cost	$1.31	$/t total		$ 200,220,225	$ -	$ 1,303,079	$ 7,975,357	$ 13,589,156	$ 15,631,338	$ 17,010,517	$ 24,248,407	$ 23,946,239	$ 23,800,846	$ 23,800,846	$ 22,367,545	$ 13,828,353	$ 8,054,737	$ 4,663,805	$ -
Total[1]				$ 221,126,510	$ -	$ -	$ 9,826,137	$ 15,439,936	$ 17,482,118	$ 18,861,298	$ 26,099,187	$ 25,797,020	$ 25,651,626	$ 25,651,626	$ 24,218,325	$ 15,679,133	$ 9,905,518	$ 6,514,585	$ -
Process																			
Processing Staff	$3.16	$/t ore		$ 78,642,105	$ -	$ 449,463	$ 4,301,805	$ 4,301,805	$ 8,020,388	$ 8,020,388	$ 6,614,017	$ 6,614,017	$ 6,614,017	$ 6,614,017	$ 6,614,017	$ 6,614,017	$ 6,614,017	$ 6,614,017	$ 636,120
Heap Leach Direct Cost	$5.16	$/t ore		$ 16,645,059	$ -	$ -	$ 5,050,843	$ 5,412,356	$ 5,412,356	$ 769,505	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Suflide Mill Direct Cost	$8.90	$/t ore		$ 193,032,290	$ -	$ -	$ -	$ -	$ 13,988,879	$ 20,244,720	$ 20,244,720	$ 20,244,720	$ 20,244,720	$ 20,244,720	$ 20,244,720	$ 20,244,720	$ 20,244,720	$ 17,085,654	$ -
Total				$ 288,319,455	$ -	$ 449,463	$ 9,352,648	$ 9,714,161	$ 27,421,622	$ 29,034,612	$ 26,858,737	$ 26,858,737	$ 26,858,737	$ 26,858,737	$ 26,858,737	$ 26,858,737	$ 26,858,737	$ 23,699,671	$ 636,120
Owners Costs																			
Sitewide G&A				$ 133,099,544	$ -	$ 7,536,042	$ 9,816,042	$ 9,816,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 10,056,042	$ 5,370,996
Land Usage/Forestry Fee				$ 52,266,116	$ 110,358	$ 4,545,340	$ 2,731,757	$ 2,731,757	$ 4,494,181	$ 3,803,343	$ 3,803,343	$ 4,457,301	$ 4,214,813	$ 4,214,813	$ 4,343,931	$ 4,339,733	$ 4,339,733	$ 4,135,711	$ -
License and Compliance Fees				$ 295,750	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ 21,125	$ -
Total				$ 185,661,410	$ 131,483	$ 12,102,507	$ 12,568,924	$ 12,568,924	$ 14,571,348	$ 13,880,510	$ 13,880,510	$ 14,534,468	$ 14,291,980	$ 14,291,980	$ 14,421,098	$ 14,416,900	$ 14,416,900	$ 14,212,878	$ 5,370,996
Total Operating Cost	Price Fctr.		1	$ 695,107,375	$ 131,483	$ 12,551,970	$ 31,747,710	$ 37,723,021	$ 59,475,089	$ 61,776,420	$ 66,838,434	$ 67,190,225	$ 66,802,344	$ 66,802,344	$ 65,498,160	$ 56,954,770	$ 51,181,155	$ 44,427,134	$ 6,007,116
Gross Income - Sales																			
Heap Leach																			
Au Payability:	99%			$ 314,530,399	$ -	$ -	$ 63,455,560	$ 131,648,609	$ 104,198,563	$ 15,227,667	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Ag Payability:	98%			$ 64,829,410	$ -	$ -	$ 16,459,689	$ 23,288,511	$ 20,797,001	$ 4,284,209	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Subtotal				$ 379,359,809	$ -	$ -	$ 79,915,248	$ 154,937,120	$ 124,995,564	$ 19,511,877	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sulfides																			
Cu Payability:	97%			$ 865,705,317	$ -	$ -	$ -	$ -	$ 72,292,746	$ 153,906,760	$ 130,788,509	$ 70,644,216	$ 69,078,438	$ 74,328,399	$ 74,512,608	$ 91,551,957	$ 66,960,033	$ 61,641,651	$ -
Zn Payability:	85%			$ 779,209,622	$ -	$ -	$ -	$ -	$ 40,257,102	$ 56,981,716	$ 91,031,766	$ 102,497,600	$ 88,599,620	$ 104,234,847	$ 101,802,701	$ 71,227,145	$ 63,930,706	$ 58,646,419	$ -
Au in Cu Conc Payability:	85%			$ 117,494,534	$ -	$ -	$ -	$ -	$ 7,990,253	$ 12,628,421	$ 12,854,880	$ 12,655,063	$ 11,589,373	$ 15,652,315	$ 14,080,423	$ 13,254,513	$ 9,537,921	$ 7,251,373	$ -
Ag in Cu Conc Payability:	84%			$ 46,448,924	$ -	$ -	$ -	$ -	$ 3,145,125	$ 4,250,365	$ 5,285,449	$ 5,509,095	$ 4,973,008	$ 6,171,344	$ 5,921,915	$ 4,358,327	$ 3,853,678	$ 2,980,618	$ -
Au in Zn Conc Payability:	53%			$ 67,108,605	$ -	$ -	$ -	$ -	$ 4,563,742	$ 7,212,895	$ 7,342,240	$ 7,228,112	$ 6,619,429	$ 8,940,033	$ 8,042,225	$ 7,570,496	$ 5,447,714	$ 4,141,721	$ -
Ag in Zn Conc Payability:	40%			$ 38,402,533	$ -	$ -	$ -	$ -	$ 2,600,292	$ 3,514,071	$ 4,369,846	$ 4,554,749	$ 4,111,529	$ 5,102,276	$ 4,896,056	$ 3,603,330	$ 3,186,101	$ 2,464,283	$ -
Subtotal				$ 1,914,369,535	$ -	$ -	$ -	$ -	$ 130,849,259	$ 238,494,228	$ 251,672,690	$ 203,088,834	$ 184,971,396	$ 214,429,214	$ 209,255,928	$ 191,565,769	$ 152,916,152	$ 137,126,065	$ -
Sales Cost																			
Dore Au	$5.00	$/oz. Au		$ 1,258,122	$ -	$ -	$ 253,822	$ 526,594	$ 416,794	$ 60,911	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Dore Ag	$0.50	$/oz. Ag		$ 1,776,148	$ -	$ -	$ 450,950	$ 638,041	$ 569,781	$ 117,376	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Cu Concentrate Transport Cost	$70	$/t con.		$ 33,676,916	$ -	$ -	$ -	$ -	$ 2,812,270	$ 5,987,147	$ 5,087,821	$ 2,748,140	$ 2,687,229	$ 2,891,459	$ 2,898,625	$ 3,561,475	$ 2,604,821	$ 2,397,930	$ -
Zn Concentrate Transport Cost	$76	$/t con.		$ 58,148,493	$ -	$ -	$ -	$ -	$ 3,004,185	$ 4,252,259	$ 6,793,243	$ 7,648,880	$ 6,611,744	$ 7,778,522	$ 7,597,023	$ 5,315,323	$ 4,770,827	$ 4,376,487	$ -
Copper Conc. Treatment	$85	$/t Cucon.		$ 40,638,213	$ -	$ -	$ -	$ -	$ 3,393,589	$ 7,224,740	$ 6,139,516	$ 3,316,203	$ 3,242,702	$ 3,489,147	$ 3,497,794	$ 4,297,661	$ 3,143,259	$ 2,893,602	$ -
Zinc Conc. Treatment	$295	$/t Zncon.		$ 226,104,583	$ -	$ -	$ -	$ -	$ 11,681,472	$ 16,534,482	$ 26,414,843	$ 29,741,903	$ 25,709,103	$ 30,246,003	$ 29,540,263	$ 20,668,102	$ 18,550,882	$ 17,017,531	$ -
Copper Conc. Cu Refining Charge	$0.09	$/lb Cu		$ 26,758,164	$ -	$ -	$ -	$ -	$ 2,234,503	$ 4,757,118	$ 4,042,554	$ 2,183,548	$ 2,135,152	$ 2,297,423	$ 2,303,117	$ 2,829,788	$ 2,069,674	$ 1,905,287	$ -
Copper Conc. Au Refining Charge	$5.00	$/oz. Au		$ 469,978	$ -	$ -	$ -	$ -	$ 31,961	$ 50,514	$ 51,420	$ 50,620	$ 46,357	$ 62,609	$ 56,322	$ 53,018	$ 38,152	$ 29,005	$ -
Copper Conc. Ag Refining Charge	$0.40	$/oz. Ag		$ 1,018,059	$ -	$ -	$ -	$ -	$ 68,934	$ 93,159	$ 115,845	$ 120,747	$ 108,997	$ 135,262	$ 129,795	$ 95,525	$ 84,464	$ 65,329	$ -
Copper Conc. Insurance	0.088%	of CIF		$ 845,473	$ -	$ -	$ -	$ -	$ 68,375	$ 139,621	$ 121,950	$ 73,161	$ 70,495	$ 79,348	$ 77,905	$ 89,662	$ 66,014	$ 58,943	$ -
Zinc Conc. Insurance	0.110%	of CIF		$ 724,478	$ -	$ -	$ -	$ -	$ 39,314	$ 56,292	$ 83,962	$ 92,992	$ 80,984	$ 96,834	$ 93,721	$ 67,906	$ 59,415	$ 53,058	$ -
Subtotal				$ 391,418,627	$ -	$ -	$ 704,773	$ 1,164,636	$ 24,321,178	$ 39,273,617	$ 48,851,153	$ 45,976,195	$ 40,692,763	$ 47,076,608	$ 46,194,565	$ 36,978,460	$ 31,387,508	$ 28,797,173	$ -
Total				$ 1,902,310,716	$ -	$ -	$ 79,210,476	$ 153,772,484	$ 231,523,646	$ 218,732,488	$ 202,821,538	$ 157,112,638	$ 144,278,634	$ 167,352,607	$ 163,061,363	$ 154,587,309	$ 121,528,645	$ 108,328,891	$ -
Royalties																			
Government Royalty on Ore	2.60%	ofnetproc.		$ 37,622,531	$ -	$ -	$ -	$ 1,489,511	$ 3,418,724	$ 4,927,798	$ 4,571,251	$ 4,214,140	$ 3,008,705	$ 2,681,326	$ 3,281,249	$ 3,166,320	$ 2,946,103	$ 2,086,578	$ 1,830,825
Salvage Value																			
Percentage of Buildings+Equipment	0%			$ -														$ -	
Net Operating Income Pre-Tax				$ 1,169,580,810	$ (131,483)	$ (12,551,970)	$ 47,462,766	$ 114,559,952	$ 168,629,832	$ 152,028,270	$ 131,411,852	$ 85,708,274	$ 74,467,584	$ 97,868,937	$ 94,281,954	$ 94,466,219	$ 67,401,386	$ 61,815,179	$ (7,837,941)
Depreciation																			
Amortization pre-production mining				$ 3,942,324			$ 154,750	$ 225,796	$ 354,596	$ 383,552	$ 359,975	$ 359,975	$ 359,975	$ 359,975	$ 359,975	$ 359,975	$ 359,975	$ 303,803	
Depreciation of Fixed Assets				$ 211,155,101	$ -	$ -	$ 25,432,770	$ 32,482,033	$ 26,846,325	$ 21,574,645	$ 18,827,957	$ 16,934,208	$ 14,101,556	$ 13,095,561	$ 9,725,012	$ 21,942,304	$ 7,798,638	$ 1,752,932	$ 641,160
Depletion																			
of expl/develop. Expenses +Land Value				$ 17,000,000	$ -	$ 62,773	$ 604,535	$ 973,670	$ 1,529,079	$ 1,653,943	$ 1,552,278	$ 1,552,278	$ 1,552,278	$ 1,552,278	$ 1,552,278	$ 1,552,278	$ 1,552,278	$ 1,310,054	$ -
Taxable Income				$ 937,483,385	$ (131,483)	$ (12,614,744)	$ 21,270,711	$ 80,878,452	$ 139,899,833	$ 128,416,129	$ 110,671,642	$ 66,861,812	$ 58,453,775	$ 82,861,123	$ 82,644,688	$ 70,611,661	$ 57,690,495	$ 58,448,389	$ (8,479,101)
Turkish Taxes																			
Loss Carry Forward						$ (131,483)	$ (12,746,226)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Taxable income				$ 945,962,485	$ -	$ -	$ 8,524,485	$ 80,878,452	$ 139,899,833	$ 128,416,129	$ 110,671,642	$ 66,861,812	$ 58,453,775	$ 82,861,123	$ 82,644,688	$ 70,611,661	$ 57,690,495	$ 58,448,389	$ -
Corporate Tax	20%				$ -	$ -	$ 1,704,897	$ 16,175,690	$ 27,979,967	$ 25,683,226	$ 22,134,328	$ 13,372,362	$ 11,690,755	$ 16,572,225	$ 16,528,938	$ 14,122,332	$ 11,538,099	$ 11,689,678	$ -
Investment Incentive on capital	40%			$ 47,618,359	$ 393,776.18	$ 20,465,438	$ 8,840,924	$ 16,772,574	$ 1,145,647	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Investment Incentive Carry Forward					$ 393,776	$ 20,859,214	$ 27,995,241	$ 28,592,125	$ 1,757,805	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Corporate Tax Reduction	70%					$ -	$ 1,193,428	$ 11,322,983	$ 19,585,977	$ 1,230,464	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Payable Corporate Tax	16%			$ 155,859,646	$ -	$ -	$ 511,469.10	$ 4,852,707	$ 8,393,990	$ 24,452,762	$ 22,134,328	$ 13,372,362	$ 11,690,755	$ 16,572,225	$ 16,528,938	$ 14,122,332.27	$ 11,538,098.96	$ 11,689,677.88	$ -
Net Operating Income after Taxes				$ 781,623,739	$ (131,483)	$ (12,614,744)	$ 20,759,242	$ 76,025,745	$ 131,505,843	$ 103,963,367	$ 88,537,313	$ 53,489,450	$ 46,763,020	$ 66,288,898	$ 66,115,751	$ 56,489,329	$ 46,152,396	$ 46,758,712	$ (8,479,101)
Add Back Depreciation and Depletion																			
Operating Cashflow After Taxes				$ 1,013,721,165	$ (131,483)	$ (12,551,970)	$ 46,951,297	$ 109,707,244	$ 160,235,842	$ 127,575,508	$ 109,277,523	$ 72,335,912	$ 62,776,829	$ 81,296,712	$ 77,753,016	$ 80,343,886	$ 55,863,287	$ 50,125,501	$ (7,837,941)
Capital Costs																			
Initial Capital Costs																			
Plant				$ 46,381,169	$ -	$ 46,381,169	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure				$ 41,972,454	$ 6,089,296	$ 35,883,158	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Site Investigation and Project Eng. Work				$ 6,900,000	$ 6,100,000	$ 800,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Private Land purchase				$ 1,600,000	$ -	$ 1,600,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-Production Mining				$ 3,153,860	$ -	$ 3,153,860	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Contingency Avg.	20%			$ 19,711,229	$ 2,210,187	$ 17,501,042	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Subtotal				$ 119,718,713	$ 14,399,483	$ 105,319,229	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital Costs																			
Plant				$ 81,052,731	$ -	$ -	$ 27,058,347	$ 49,994,384	$ -	$ -	$ 2,000,000	$ -	$ 2,000,000	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure				$ 23,336,468	$ -	$ -	$ 3,587,688	$ 6,233,260	$ 2,180,354	$ 1,434,827	$ 2,282,345	$ 3,686,397	$ 659,857	$ 3,271,741	$ -	$ -	$ -	$ -	$ -
Site Investigation and Eng. Work				$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation				$ 17,661,652	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,685,492	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 14,976,161
Contingency Avg.	22%			$ 26,779,802	$ -	$ -	$ 5,751,249	$ 11,076,446	$ 683,763	$ 401,864	$ 1,969,344	$ 1,047,916	$ 557,284	$ 947,599	$ -	$ -	$ -	$ -	$ 4,344,337
Subtotal				$ 148,830,654	$ -	$ -	$ 36,397,283	$ 67,304,091	$ 2,864,117	$ 1,836,691	$ 8,937,180	$ 4,734,313	$ 3,217,141	$ 4,219,340	$ -	$ -	$ -	$ -	$ 19,320,497
Total	Price Fctr.		1	$ 268,549,367	$ 14,399,483	$ 105,319,229	$ 36,397,283	$ 67,304,091	$ 2,864,117	$ 1,836,691	$ 8,937,180	$ 4,734,313	$ 3,217,141	$ 4,219,340	$ -	$ -	$ -	$ -	$ 19,320,497
Cash Flow																			
Before Tax Cash Flow				$ 901,031,444	$ (14,530,966)	$ (117,871,199)	$ 11,065,483	$ 47,255,861	$ 165,765,715	$ 150,191,579	$ 122,474,671	$ 80,973,961	$ 71,250,443	$ 93,649,597	$ 94,281,954	$ 94,466,219	$ 67,401,386	$ 61,815,179	$ (27,158,439)
Before Tax Cumulative Cash Flow					$ (14,530,966)	$ (132,402,166)	$ (121,336,683)	$ (74,080,822)	$ 91,684,893	$ 241,876,472	$ 364,351,143	$ 445,325,105	$ 516,575,548	$ 610,225,145	$ 704,507,098	$ 798,973,317	$ 866,374,703	$ 928,189,882	$ 901,031,444
After Tax				$ 745,171,798	$ (14,530,966)	$ (117,871,199)	$ 10,554,014	$ 42,403,153	$ 157,371,725	$ 125,738,817	$ 100,340,343	$ 67,601,599	$ 59,559,688	$ 77,077,372	$ 77,753,016	$ 80,343,886	$ 55,863,287	$ 50,125,501	$ (27,158,439)
After Tax Cumulative Cash Flow					$ (14,530,966)	$ (132,402,166)	$ (121,848,152)	$ (79,444,998)	$ 77,926,727	$ 203,665,544	$ 304,005,887	$ 371,607,485	$ 431,167,174	$ 508,244,546	$ 585,997,562	$ 666,341,448	$ 722,204,736	$ 772,330,237	$ 745,171,798
Financial Items	Before Tax NPV,		5%	$576,538,514															
			10%	$377,064,737															
			15%	$250,068,854															
	Before Tax IRR		51%																
Non-discounted payback from initial ore processing:			2.4	yrs															
	After Tax NPV,		5%	$475,238,194															
			10%	$308,693,460															
			15%	$202,381,200															
	After Tax IRR		47%																
Non-discounted payback from initial ore processing:			2.5	yrs															

23.0 ADJACENT PROPERTIES

There are currently no operating mineral properties in the immediate area of Gediktepe. Other companies hold licenses that are north, east, northwest, and southwest of the current licenses held by Polimetal.

24.0 OTHER RELAVENT DATA AND INFORMATION

Two additional items were prepared as part of this prefeasibility study which are summarized in this section:

1) Project Execution Plan and Schedule
2) Project Risk Register

In order to produce gold from an oxide heap leach, a number of significant steps need to be accomplished. A project execution plan has been developed by Polimetal to provide a clear understanding of all tasks and responsibilities required to implement the project.

A summary of the project schedule as well as the major tasks and protocols within the execution plan are presented in this section. The complete execution plan is too extensive for presentation in a Technical Report.

The project risk register was assembled with input from the engineering team and Polimetal. The risk register is intended to provide an understanding to Polimetal management of the project tasks and execution items that will need to be addressed in order to provide high confidence that the project performs as evaluated in the financial analysis of Section 22.

24.1 Project Execution Plan and Schedule

The project execution plan for Gediktepe addresses both the heap leach facility and the sulfide operation that follows. The first and immediate tasks relate to the execution of the oxide portion of the project. Sulfide data collection, design and construction is planned to continue in parallel with the construction of the oxide heap leach facility.

There are 7 major work areas that must be addressed prior to oxide production:

1) EIA Submittal and Permit Approval
2) Power Transmission Line permitting, land acquisition, and construction
3) Heap Leach Facility data collection and site investigation, design, site preparation, earthwork and liner placement.
4) Oxide Plant final testing, detailed design, contractor selection, construction, construction management, testing and commissioning
5) Updated mine plans followed by contractor selection, mine pre-production striping and establish ore control procedures.
6) Infrastructure including access roads, construction camp water diversion, water treatment, maintenance shops, fuel tankage, and worker social support.
7) Engineering testing, additional drilling, kinetic testing and a finalized waste rock management plan.

In addition, the following two major work areas are required for the sulfide plant which will be executed in parallel with construction and operation of the heap leach facility.

8) Stage 1 Tailing Facility, including selection of the engineering firm, detailed design, contractor selection, site preparation and construction.
9) Sulfide Processing plant, including selection of the engineering firm, additional testing, detailed design, selection of the EPCM contractor, equipment purchase, construction, and commissioning of the sulfide plant.

The project execution plan describes how the design presented in the PFS will be implemented. This plan provides a description of the organization of the project development from engineering to ore processing with an emphasis on expediting oxide ore mining and processing.

Figure 24-1 summarizes the tasks and estimated time frames that are planned for the 9 major work areas that incorporate 120 summarized line items.

The objectives of the project execution plan are

- Provide a plan for managing project execution that promotes safety in project development and operations along with environmental stewardship and compliance.
- Identify engineering work required for the design of a successful project.
- Provide the shortest timeline to oxide ore production.
- Outline commissioning of both the oxide and sulfide processing streams
- Provide a schedule that identifies: when tasks need to be completed to keep the project on schedule, when capital expenditures are expected to occur, and when income from metal production can be anticipated

The major items in the execution plan are summarized in the following sub-sections.

24.1.1 Project Management

Managing of the project from engineering through commissioning will be managed through a hired Engineering, Procurement, Construction and Management (EPCM) firm working with the Polimetal management team.

Polimetal will retain responsibility for purchase orders, contracting, project and purchase award and warehousing. Requests for quotes and commercial terms will be jointly set by Polimetal and the EPCM contractor.

Polimetal has created an environmental management group that is responsible for ensuring that permit applications and project designs are compliant with environmental regulations. Having an environmental management group is critical for timely construction of the oxide portion of the project as there is no buffer time built into the oxide construction schedule for noncompliance in receiving permits.

Polimetal has established an internal health & safety management group. This group has begun the necessary planning to ensure that the company is compliant in health and safety laws in construction and operations. The group will promote a culture of safety at the project site; training of locals in best practices has already been in place for the site work that has been completed to date.

All process implementation work on the Gediktepe Project will be performed under an EPCM style of contract with the EPCM Contractor. The tasks will include: project management, engineering, design, drafting, procurement, contract management, construction management and commissioning of the oxide process plant facilities and associated infrastructure. The sulfide process work will be addressed in parallel.

The EPCM Project management team members are summarized as follows:

Project Director
Project Manager
Design Manager
Senior Lead Engineers
Procurement and Contracts Officer
Quality Engineer
Scheduler and Cost Controller
Construction Manager (Site Based)
Site Project Engineer (Site Based)
Site Discipline Supervisors (Site Based)
Safety Advisor (Site Based
Material Control Officer (Site Based)
Commissioning Manger
Workshop QA Manger
Workshop Expediter

EPCM Contractor will be responsible for the following functions necessary to provide a complete, safe, quality and technically compliant project including:

1) Project Management
2) Engineering, Drafting and Document Control
3) Construction Management
4) Cost and Schedule Control
5) Commissioning Management

Quality assurance of the site infrastructure construction will be overseen by the project manager and Polimetal quality engineers. Contractors will be briefed on Polimetal policies and the contractor's quality assurance plan will be reviewed before the contractor arrives at site. The project manager and quality engineers will prepare and implement a plan to assure compliance with Polimetal policies and engineered designs by correcting non-compliances and implementing preventive measures.

24.1.2 Engineering

Basic engineering through detailed engineering for project construction will be completed by engineering firms selected by Polimetal.

Detail engineering of the heap leach facility will be completed by Golder Associates.

Basic engineering of oxide processing and sulfide processing was completed by GR Engineering Services in Perth.

A site wide water management study (including basic engineering) will be prepared by SRK.

Detail engineering and preparation of drawings of ex-pit haul roads, sitewide layout, water diversions, etc. will be prepared by Norma Engineering and has been started.

Pit optimization and mine planning (long-medium-short term), will be done in house by Polimetal.

Detailed engineering is still required for: the tailings storage facility, the oxide processing circuit, and the sulfide mill. On the mining side of the project, detail engineering is required for ore blending if results of geometallurgical testing warrant it.

Detailed engineering designs for all infrastructure will be prepared by already selected engineering firms and all tender documents and drawings will be prepared by these engineering firms. The Supply Chain Dept. of Polimetal will tender out for the construction of these infrastructures. Polimetal will preferably focus on utilizing regional construction companies for these small scale infrastructure works.

24.1.3 Procurement and Supply Chain Management

In current markets, there are not expected to be any items with long lead times for procurement at the Gediktepe project. Polimetal will solicit bids from multiple equipment vendors to award contracts for supplying equipment to Gediktepe.

The scope and policy of the procurement and supply chain management has specified the authorities, liabilities, and applications to be undertaken in order to supply the requested goods and services in accordance with the specific needs in the correct time, amount, and quality.

The Polimetal supply chain team will consist of:

- Warehouse Chief
- Procurement Chief
- Foreign Trade Specialists
- Contract Specialist
- Buyer

Expediting of purchased goods will be carried out to ensure their timely delivery, including equipment, materials, and services, according to the Project Schedule.

Generally, vendors where possible will be responsible for the delivery of major equipment to the construction site. Construction contractors will be required to deliver the materials and equipment items for which they are responsible and also unload and unpack other equipment items and materials which are delivered to the site.

EPCM contractor would undertake regular inspection and expediting visits to all major equipment suppliers and contractors providing site materials to ensure quality, schedule and cost compliance with contracts and orders.

Transportation logistics will be managed by a dedicated third party freight forwarder.

24.1.4 Security

General site security is to be supplied by an external security organization. The service provider will provide skilled resources that will monitor and report security related matters site wide & social facility area.

The security service provider is to report directly to the Human Resources and Administration Manager. The scope of services to be provided includes:

- Managing the ingress and egress of employees, contractors, visitors, deliveries and to act as a barrier for un-notified access,
- To issue and retrieve visitor PPE/ID tags, and maintain records of visitors and deliveries
- To provide site security and assistance in the event of unruly behavior, criminal acts or incidents requiring, discipline and control
- Assist and support in carrying out random drug and alcohol testing as required.
- Act under the Law (5188 numbered) of Private Security Services.

Each Contractor shall be responsible for the management and control of their specific site areas within the overall project site.

24.1.5 Construction

The progression of construction activities for key project infrastructure has been developed as input to the project execution plan. The construction activities for the HLPF, the TSF, and mineral processing facilities are summarized below; the timing of the construction activities can be seen in detail in the execution schedule in Figure 24-1.

Construction of the heap leach pad foundation will include the following activities in order:

-Foundation preparation

Topsoil removal, excavation of unsuitable material

-Construction of under drain system

-Compacted Fill

Consists of material from cuts within HLP footprint and excavations of other project infrastructure

-Install Liner System

Leak detection, composite liner, soil liner

-Install Solution Collection System

Piping placed directly on the geomembrane

-Placement of overliner

Free draining gravel placed to protect collection system

Construction of the tailings storage facility will include the following activities in order:

- Foundation preparation

Topsoil removal, excavation of unsuitable material

-Construction of under drain system

-Compacted fill of the embankment

Consists of imported mine waste

- Install Liner System

Leak detection, geo-composite liner, 2mm HDPE geomembrane

-Install overdrain

Construction of the mineral processing facilities will include the following activities in order:

-Bulk earthworks by site earthworks contractor

-Concrete installations

-Structural steel installations

-Equipment installations

-Piping and electrical installations

24.1.6 Commissioning

Once constructed, commissioning of the oxide processing circuit is expected to take approximately two months. Throughput rates of the crushing circuit during the first quarter of oxide processing are estimated to be about 2,200 tpd. After the first quarter, oxide production is planned at 3,000 tpd. A month delay is expected between the beginning of oxide ore crushing and the first doré pore to account for loading of the heap and percolation time of the cyanide solution through the ore.

During a commission period of one year (during year 3 of mining), the mill is planned to process an average of 4,500 tpd. The crushing circuit will process both oxide ore and sulfide ore on a campaign basis during the year that both ores are available. Oxide production will be phased out in year 4 and sulfide ore production will be sustained at 6,500 tpd through the end of the mine life.

The plant would be pre, dry, and wet commissioned by the designated commissioning team and then after practical completion, transferred to Polimetal by a combined team consisting of EPCM Contractor, Polimetal operations personnel and selected vendor representatives.

Upon completion of successful commissioning and production ramp up, Polimetal with the EPCM contractor will perform a plant performance test to verify that the plant complies with the design criteria and obligations under the Contract.

EPCM contractor will be responsible for all pre, dry, and wet commissioning activities which will be requested by Polimetal. The EPCM contractor commissioning team will assist Polimetal employees to operate the equipment during the ore commissioning and ramp-up phases and for performance testing of the plant.

Technical support will be provided during the commissioning phase from the project management team, design engineers, and the construction team. Construction personnel will be on hand during the commissioning period to implement modifications to any equipment, plate work, piping, and electrical items that are deemed necessary during the commissioning process.

Any changes to drawings will be recorded for the purposes of finalizing "As-built" drawings.

24.1.7 Health and Safety Management

Polimetal will provide a safe and healthy environment for all personnel required to work in and about any area of the project both on and off site. The adoption of an applicable Occupational Health and Safety Management System aligned with local legislative standards and industry best practices has provided the framework to map, develop and implement the Project Health and Safety Management System.

The system consists of policies, plans, procedures and forms and has been developed as a dynamic system to ensure due consideration is given to all project health and safety risks (real and potential), and to ensure that the Project manages such risks with appropriate control measures to achieve the required standard for the Project's duty of care and due diligence obligations. All contractors will comply with the Project Health and Safety Management System.

The Project Health and Safety Management plan addresses:

-Commitment and Leadership in providing a safe work environment
-Communication and Involvement in identifying, assessing, and mitigating risks
-Maintaining a site wide Emergency Response unit
-Implementing Health and Safety Standards

24.1.8 Execution Schedule

A project execution schedule has been developed to define the tasks and responsibilities to move the project from its current PFS status to oxide ore production and then onto sulfide ore production. Figure 24-1 summarizes the schedule. This schedule has been developed with consideration for construction limitations due to seasonal weather and the assumed start of oxide ore production is during late summer. The following bullets summarize the major items that are to be completed within each major work area over the specific critical time frames.

<u>Tasks during Pre-Production Year -2</u>

- EIS Submittals and Permits
  - Apply for and receive all Permits
- Project Transmission Line
  - Forest Permitting
  - Purchase Land and Expropriation of Land
  - Construction
- Heap Leach Facility
  - Geotech Drilling Permits
  - Geotech Drilling
  - Detail Design
  - Contractor Mobilization for Clearing and Grubbing
- Oxide Process Plant
  - Basic Engineering (Complete)
  - Selection of EPCM Contractor and Establish Contract
  - Start Detailed Design
- Mine Preproduction
  - Software Training
  - Mine Planning
  - Mining Contractor Selection
- Infrastructure
  - Access Road Design and Access Road Permitting
  - Start Access Road Construction
- Engineering and Site Investigation
  - Kinetic Tests
  - Hydrogeologic Modeling
  - Phase 4 Permits and Drilling
  - Site Wide Water Management Plan
  - Waste Rock Management Plan
- Sulfide Tailing Storage
  - Contractor Selection
  - Detail Design
- Sulfide Processing
  - Sulfide Ore Test Work including Variability Testing
  - Basic Engineering

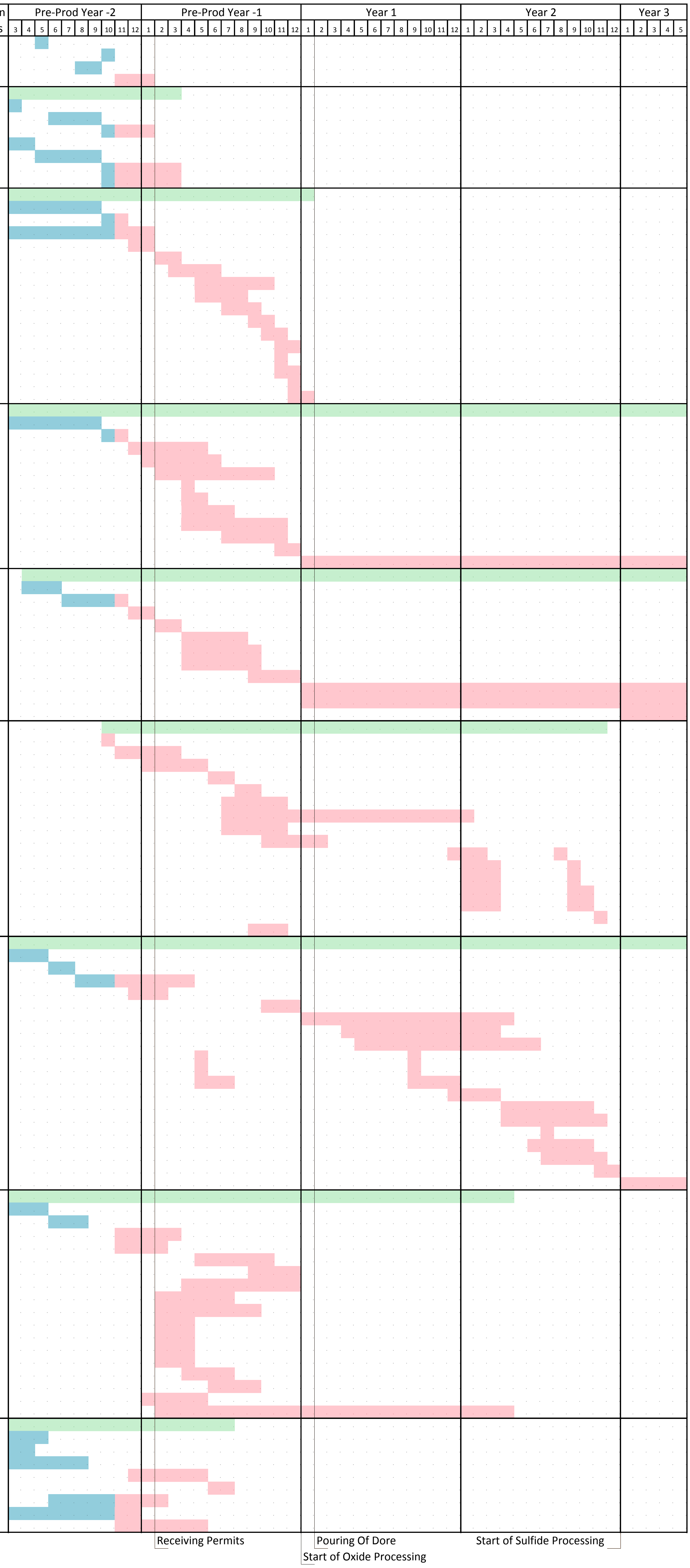
Tasks during Pre-Production Year -1 Culminating in Oxide Ore Production

- Power Transmission Line
  - Complete Construction
- Heap Leach Facility
  - Clear, Grub and Remove Unsuitable Material
  - Cut and Fill, Blasting as Required
  - Waste Rock Fill and Compaction
  - Subgrade Preparation
  - Soil Liner
  - Anchor Trench Excavation
  - Geomembrane Liner Placement
  - Double HDPE Liner and Drain Net
  - Over liner and Piping Net
- Oxide Process Plant
  - Complete Detailed Design
  - Obtain Bids and Select Equipment
  - Manufacturing and Procurement of Equipment
  - Clear, Grub, and Topsoil Removal
  - Cut and Fill for Plant
  - Construction and Installation
  - Testing
  - Begin Loading the Pad with Oxide Ore
  - Begin Irrigation of Oxide Ore
- Mine Preproduction
  - Contractor Mobilization
  - Road Pioneering
  - Clear, Grub, and Topsoil Removal
  - Preproduction Stripping
- Infrastructure
  - Water Diversion Channels
  - Water Treatment Plant
  - Clean Water Pond Construction
  - Clay and Over liner Excavation
  - Establish Security
  - Install Fuel Storage Tanks
  - Install the Concrete Plant
  - Construct and Equip the Assay Lab
  - Build All Workshops and Warehouse
  - Set Up Social Facilities for the Contractors
- Sulfide Tailing Storage
  - Clear and Grub
  - Remove Unsuitable material and Complete Required Cut and Fill
  - Start Waste Rock Fill
  - Start the Under drain
- Sulfide Processing
  - Select the EPCM Contractor for the Sulfide Plant

Tasks during Production Years 1 and 2 Culminating in Production of Sulfide Concentrates

- Mine Production
    - Oxide Ore Production
    - Pre-Stripping for Continued Sulfide Production
- Sulfide Tailing Storage
    - Continue Waste Rock Fill
    - Geomembrane and Subgrade Preparation
    - Excavate Anchor Trenches
    - Construct Leak Detection System
    - HDPE Geomembrane and GCL Liner Placement
    - Install Over drain and Piping
- Sulfide Processing
    - Process Plant Detail Design
    - Select, Bid, and Procure Equipment
    - Clear, Grub, and Topsoil Removal
    - Cut and Fill
    - Building and Equipment Foundations
    - Install Ball Mills
    - Install Float Cells
    - Build the Plant Building
    - Thickener/Filtration
    - Reagent Mixing Facilities
    - Testing
    - Commissioning and Commercial Production of Concentrates Aug 2019
- Infrastructure
    - Continue Social Facilities for the Contractors

	Task	Duration Months
1	Submit EIA	1
2	Receiving EIA Positive Certificate	1
3	Prepare Application for Forest Permit	2
4	Applying For and Receiving Permit	3
	Power Transmission Line	13
5	Project Approval	1
6	Forest Permitting	4
7	Purchase Private Land	4
8	Connection Agreement	2
9	Taking Expropriation Decision	5
10	Expropriation	6
11	Construction	6
	HLPF Ph-1	23
12	Phase 2 Geotech Holes Permit	7
13	Phase 2 Geotech Drilling	2
14	HLP Detail Design	11
15	Selecting Contractor/Contractor mob	2
16	Clear and Grub	2
17	Unsuitable Material Removal	4
18	Cut/Fill	6
19	Waste Rock Fill	4
20	Blasting	3
21	Geomembrane Subgrade Preparation	2
22	Soil Liner	2
23	Single Anchor Trench	2
24	Double Anchor Trench	1
25	HDPE Geomembrane Placement	2
26	Double HDPE Line + Drain Net	1
27	Overliner + Piping Network	2
	Oxide Processing	62
38	Processing Plant Basic Designs	7
39	Selection of EPCM Contractor&Agreement	2
40	Detail Design/Engineering	6
41	Bid and Selection of Equipment	6
42	Equipment Procurement/Manufacturing	9
43	Clear and Grub	1
44	Topsoil Removal	2
45	Cut/Fill for Crusher, Aggl., Conveyor	4
46	Construction	8
47	Installation	5
48	Pre-Commissioning&Testing	2
49	Commercial Oxide Ore Production	40
	Mining Pre-Production	195
50	MineSight Training	3
51	Mine Planning	5
52	Contractor Selection	2
53	Contractor Mobilization	2
54	Road Pioneering	5
55	Clear and Grub	6
56	Topsoil Removal	6
57	Pre-Production Mining	4
58	Production Mining	150
59	Oxide Feed to Crusher	39
60	Sulfide Feed to Crusher	150
	1st Stage Tailings Storage Facility	38
61	Bid and Selection of Eng. Firm	1
62	TSF Detail Design	5
63	Selecting Contractor/Mob	5
64	Clear And Grub	2
65	Unsuitable Material Removal	2
66	Cut/Fill	5
67	Waste Rock Fill	19
68	Blasting	5
69	Underdrain	5
70	Geomembrane Subgrade Preparation	9
71	Single Anchor Trench	9
72	Leak Detection	9
73	HDPE Geomembrane Placement	10
74	GCL	10
75	Overdrain and Piping	1
76	Diversion Channel	3
	Sulfide Processing Plant	196
77	Sulfide Ore Met Tests	3
78	Basic Engineering	2
79	Sulfide Ore Detailed Testwork & Engineering	9
80	Variability Testwork	3
81	Selection of EPCM Contractor&Agreement	3
82	Processing Plant Detail Designs	16
83	Bid and Selection of Equipment	12
84	Equipment Procurement	14
85	Clear and Grub	1
86	Topsoil Removal	1
87	Cut and Fill	4
88	Building and Equipment Foundation	4
89	Ball Mills	7
90	Flotation Cells	8
91	Plant Building	1
92	Thickener/Filtration	5
93	Reagent Mixing/Other Buildings	5
94	Pre-Commissioning&Testing	2
95	Commercial Cu & Zn Concentrate Production	150
	Other Infrastructure	38
96	Access Road Design	3
97	Access Road Permitting	3
98	Access Road Construction	5
99	Construction of Camp	4
100	Water Diversion Channels	6
101	Water Treatment Plant	4
102	Clean Water Pond Construction	9
103	Clay And Overliner Excavation	6
104	Construction of Offices	8
105	Security	3
106	Fuel Tank 1	3
107	Fuel Tank 2	3
108	Concrete Plant	3
109	Laboratory Construction/Testing	4
110	Maintenance Workshops/warehouses	4
111	Contractor Maintenance Workshops	5
112	Social Facilities of Contractors	27
	Studies/Engineering/Geology	17
113	Kinetic Tests	3
114	Hydrogeological Modeling	2
115	Phase 4 Drilling Permits	6
116	Phase 4 Drilling	6
117	Updating Geological Resource Estimate	2
118	Site Wide Water Management Plan	9
119	General Minesite Engineering	10
120	Waste Rock Management Plan	7



Completed Activities

Scheduled Activities

Figure 24-1
Project Execution Schedule

24.2 Risk Register

The risk register is a list of items that might potentially become problems for successful completion and execution of the project. The list was generated by the project engineering team and by Polimetal with each team member providing more guidance in their individual areas of project responsibility.

The risk register listed the following items:

1) Risk Area or Item
2) Impact of the Risk
3) Risk Mitigation, items planned or required to limit the risk of exposure
4) Likelihood Ranking 1 to 5
5) Consequences Ranking 1 to 5
6) Severity = product of likelihood and consequence, 1 to 25

Table 24-1 provides a simplified explanation of the risk metric used to identify which mitigation tasks are critical to assuring a successful project.

Table 24-1
Risk Severity (Likelihood X Consequence)

Likelihood		Consequence				
		Insignificant	Minor	Moderate	Major	Catastrophic
		1	2	3	4	5
Very Likely	5	5	10	15	20	25
Quite Likely	4	4	8	12	16	20
Somewhat likely	3	3	6	9	12	15
Unlikely	2	2	4	6	8	10
Very Unlikely	1	1	2	3	4	5

For this study, risks with a severity of greater than 9 are identified as action items needing additional attention to mitigate the risk.

The risk register is divided into categories that generally parallel the project execution plan, with the addition of resource estimation, geology, and mine planning to the list. There are 66 line items on the risk register.

The risk register is presented in Table 24-2. The severity column is highlighted to match the severities presented in Table 24-1. Risk metrics are assigned to reflect the likelihood and consequence of the risk at the time of the writing of this PFS. Consequently, if a risk has already been mitigated, it will have a low severity. If the risk needs to be mitigated between the writing of the PFS and beginning of operations, it will have a high severity.

Table 24-2
Risk Register

Risk Area	Impact of Risk	Risk Mitigation	Risk Metrics at Time of PFS		
			Likelihood	Conseq.	Severity
QA-QC Database					
Drill & assay data security & correctness	Loss of revenue	1) Use access based softwares 2) Implement comprehensive Qa/QC management system	1	4	4
Lack of QA-QC data for Base metals	Uncertainty in base metal data leading to uncertainty in resource.	1) Select assay pulps to submit to an independent lab for check assays of silver, copper and zinc. 2) Adjust sample submitting procedures so that a minimum of 1 base metal standard is included in each drill hole.	3	4	12
Assay data	Loss of revenue	1) Use automated laboratory file capture 2) Implement standardised sampling and QC reporting 3) Automating import of assay files 4) Assay QC monitoring and analysis	1	4	4
Geology and Resource					
Over estimate the resource	Shorter mine life	Get independent audit	1	4	4
Over estimate the grade	Loss of revenue	Get independent audit	1	4	4
Structural complexity	Change in production sequance	Decrease drill spacing	3	3	9
Differance of estimated grade and actual grades	Loss of revenue	Implement grade control sampling & update the resource model	2	3	6
Mining					
Slope angles too steep	Highwall failure, Greater cost of waste stripping	Investigate rock response to mining in initial phases	2	3	6
Bad weather during pre-production	Delay of ore production	Plan to complete pre-production in dry season	3	2	6
Slow contractor mobilization	Delay of ore production	Write contract to encourage timely mobilization	2	4	8
High degree of dilution	Increased processing costs, reduced metal production	Implement stringent protocols for blast hole sampling and ore control.	3	4	12
Contractor mining cost underestimate	Increased mining costs	Apply contingency to contractor cost	3	2	6
Processing					
Inadequate Crusher Design	Inefficient operation resulting in improper coarse/fine crush, Increased maintenance costs	1) Design by proven technologies and simulations. 2) Complete additional testwork. 3) Overdesign crusher capacity	2	3	6
Potential hazards in the plant design have not been identified and mitigated	Accidents, damage to equipment, injury to persons	1) Prepare the documentation needed to perform a Hazard Analysis (HazAn), i.e., flowsheets, P&IDs, GAs, etc. 2) Establish Health & Safety Management Unit	2	3	6
Oxide Processing					
Gold recovery from oxide ore on heap falls short of expectations	Loss of revenue	1) Confirm that recoveries allow for field conditions 2) Perform larger scale column testing 3) Demonstrate robustness with sensitivity analysis 4) Evaluate Project using conservative recoveries	1	4	4
Excessive consumption of cement and lime	Increased operating costs	1) Perform larger scale column testing 2) Use conservative estimates	1	3	3
Copper consumes excessive cyanide / disrupts Merrill Crowe process	Increased operating costs	1) Perform larger scale column testing 2) Use conservative estimates	2	3	6
Merrill Crowe Plant is unsuitable for solution coming off of heap.	Increased operating costs/Loss of Revenue	1) Select experienced design/manufacturing/installation company 2) Add an acid leach circuit ahead of Merrill Crowe plant	4	3	12
Sulfide Processing					
Ore is harder than anticipated	Reduced Throughput/Revenue	Perform additional Grindibility Studies 2) Overdesign grinding/crusher circuits	1	4	4
Improper equipment selection	Inefficient operation resulting in poor recovery/conentrate grade	1) Perform additional testwork for settling and thickening, etc. 2) Perform additional geometallurgical testwork.	4	4	16
Process water has unsuitable Chemistry	Poor concentrate quality and recovery	Test recycling of water in lock cycle tests.	4	3	12
Contaminants in Cu or Zn concentrate result in penalty charges	Loss of revenue or increased smelter charges	1) Multielement analysis of concentrate products from the lab 2) Perform additional geometallurgical testwork.	4	3	12
Sulfide tailings do not consolidate to estimated ultimate density as a result of the fine grind size	Increased cost for tailing impoundment	1) Use conservative design factors 2) Perform Tailings Characterization 3) Perform additional geometallurgical testwork.	3	2	6
Unanticipated dilution in the sulfide plant feed reduce the grade	Increased process operating cost	1) Close attention to ore control for mining selectivity	3	3	9
Unanticipated dilution in the sulfide plant feed introduces deleterious materials harmful to the Cu/Zn separation	Loss of revenue	Additional variability testing	3	4	12
Heap Leach Pad					
Design Cannot be permitted	Delayed Startup, regulatory and public perception issues	Review from EIA Consultant	2	3	6
Not meeting regulations in operation	Delayed Startup, regulatory and public perception issues	Review from EIA Consultant	1	4	4
Cost estimate too low	Reduced Cash Flow	1) Develop and perform Phase II geotechnical program 2) Review by local contractors 3) Include a contingency in costs	2	4	12
Solution Release due to pipe break	Metal Loss, Fines, Production Hiatus	Double Containment of Pipes	1	4	4
Pond Overflow	Fines, Production Hiatus	1) Perform detailed water balance 3) Develop sitewide water management plan	3	4	12
Foundation slope failure	Loss of containment, regulatory and public perception issues, loss in revenue and additional cost.	Develop and perform Phase II geotechnical program	4	5	20
Ore Slope Failure	Loss of containment, regulatory and public perception issues, loss in revenue and additional cost.	1) Perform laboratory testing on site specific ore material 2) Perform ore perculation tests at site conditions	3	4	6
Blockage of pregnant solution	Slope instability, Loss of revenue	Perform ore percolation tests representing site conditions	3	4	4
Liner Leakage	Loss of containment, regulatory and public perception issues, loss in revenue and additional cost.	1) Develop and perform Phase II geotechnical program 2) Develop CQC & CQA monitoring 3) Testing and documentation 3) Install robust liner system	4	3	12
Tailing Storage Area					
Design Cannot be permitted	Delayed Startup, regulatory and public perception issues,	Review from EIA Consultant	2	3	6
Not meeting regulations in operation	Delays in processing, regulatory and public perception issues,	Review from EIA Consultant	1	4	4
Cost estimate too low	Reduced Cash Flow	Develop and perform Phase II geotechnical program	3	4	12
Embankment opertopping	Loss of containment, regulatory and public perception issues, Fines, Production Hiatus	1) Perform detailed water balance 3) Develop sitewide water management plan	3	4	12
Insufficient Storage Capacity	premature end to mining, additional construction capital	1) Perform Tailings Characterization 2) Develop Tailings and supernatant management plan 3) long term spillway maintenance	4	3	12
Embankment Slope Failure	Loss of containment, regulatory and public perception issues,Loss of revenue and additional cost.	1) Develop and perform Phase II geotechnical program 2) Conduct embankment monitoring	4	5	20
Tailings consolidation takes longer than expected / does not consolidate	Longer closure time and higher closure costs, long term maintenance	Perform Tailings Characterization	3	4	12
Liner Leakage	Loss of revenue,additional costing and legal fines.	1) Install robust liner system 2) Develop and perform Phase II geotechnical program	4	3	12
Infrastructure					
Need for repair shops, warehouse, and other support facilities have not been identified	Increased capital cost	Identify the support facilities required and prepare cost estimates	4	2	8
No engineered road to project site	Slow down the construction	Design bypass road	1	4	4
Not Enough Water for the Project	Slow down & stop the production	1) Perform detailed water balance. 2) Construct clean water pond for stoarage of non-contaminated water.	3	4	12
Delay Power Transmision Line	Delay the project start up	Plan and get approval from authorities to be supplied the required power amount.	1	4	4
Permitting					
Failure to obtain EIA Positive Certificate	Delay the project start up	1) Work with int. & locally experienced EIA consultant company. 2) Prepare the EIA report to meet all legal & env. requirements and according to the views of local people and official institutions. 3) Additional studies/data might be requested officially	1	4	4
Failure to obtain the Temporary Operating Certificate in a timely manner	Delay the project start up	Prepare all the required documents in a timely manner.	3	4	12
Failure to obtain Forest Permit	Delay the project start up	Apply for permit in a timely manner.	4	4	16
Failure to obtain agricultural charactersitic changes permits	Delay the project start up	Apply for permit in a timely manner.	4	4	16
Environmental					
Non-compliant with EIA commitments	Implementation of administrative and criminal penalties.	Establish Environmental Management Unit & comply with all environmental legislation.	2	3	6
Non-complaint with environmental legislation	Implementation of administrative and criminal penalties. Also, possible production hiatus.	Establish Environmental Management Unit & comply with all environmental legislation.	2	3	6
Waste water treatment plant is not adequate	Delay in receiving EIA permits, temporary relocation of camp	Conduct additional waste water treatment design work.	3	3	9
Health & Safety					
Non-complaint with health & safety legislation	Implementation of administrative and criminal penalties.	Establish Health & Safety Management Unit & comply with all related legislation.	1	3	3
Land Usage					
Not able to purchase private lands	Delay the project start up	Ask for expropiration	4	3	12
Marketing					
Market Volatility	Unpredictable income	1) Hedgeing, futures and options alternatives 2)Off take agreements	3	3	9
Decreasing Metal Prices	Decreasing income/ profits	1) Keep cash costs under control 2) Consider sales contracts	4	3	12
Currency devaluation	Decreased purchasing power in foreign currencies	Keep expenses in TL as mush as possible.	3	3	9
Contract Terms	TC/RC costs, Penalties, Smelter Limitations, Payment timing, Sampling, assaying and umpire procedures	1) Negotiation and in-period adjustment 2)Having strong contract or negotiable terms	3	3	9
Concentrate Quality	Reduced demand for Concentrate	1) Sell the product with blending option via trader 2) Additional variability testing with analysis of concentrates produced	3	4	12
Governemnt					
Missing Government Incentives.	Loss of Revenue and delay of project start up.	Follow new legislations on time.	3	3	9
Additional Unanticipated Tax Obligations	Loss of revenue, additional cost and delay project start up.	Predict additional tax responsibilities like Custom Tax etc.	4	2	8
Project Funding					
Difficulty qualifying for a loan/receiving a poor interest rate on loans	Loss of Revenue and delay of project start up.	Using Bank accounts and payments for insreasing credit rating of the company	2	3	6
Delay in receiving capital advance payment from shareholders	Loss of Revenue and delay of project start up.		2	3	6
Social					
Not getting social licanse to operate	Delay the project start up	Inform all stakeholders about the project and communicate project benefits.	2	4	8
NGO involvement	Delay the project start up	Inform all stakeholders about the project and communicate project benefits.	2	4	8

A scan down the 66 items indicates that there are 24 risks with a severity greater than 9. The tasks required to mitigate these risks are provided below followed by the risks that will be mitigated by completing the tasks.

1) Complete Phase II of the geotechnical program. Phase II drilling is now in progress with forestry permits in hand.
   Risks Mitigated:
   - Underestimate of HLP capital costs.
   - HLP foundation slope failure.
   - HLP liner leakage.
   - Underestimate of TSF capital costs.
   - TSF embankment slope failure.
   - TSF liner leakage.
2) Generate a site wide water balance. (requisite work and observations for water balance are currently underway.)
   Risks Mitigated:
   - Solution ponds overflow.
   - TSF embankment overtopping
   - Not enough water for the project
3) Geometallurgical testing / variability testing on Sulfide ores.
   Risks Mitigated:
   - Improper plant equipment selection
   - Contaminants in Cu or Zn concentrate result in penalty charges
   - Unanticipated dilution in the sulfide plant feed introduces deleterious materials harmful to the Cu/Zn separation
   - Concentrate quality
4) Perform additional test work for settling and thickening of slurry during processing and as tails.
   Risks Mitigated:
   - Improper thickening equipment selection
   - Additional costs of increasing tailings storage capacity
   - Tailings consolidation takes longer than expected
5) Perform oxide ore percolation tests at site conditions (Already Underway)
   Risks Mitigated:
   - Ore slope failure
   - Blockage of pregnant solution
6) Preparation of permit applications in a timely manner
   Risks Mitigated:
   - Failure to obtain the Temporary Operating Certificate in a timely manner
   - Failure to obtain forest permit
   - Failure to obtain agricultural characteristic changes permits
7) Submit pulps to third party lab to check base metal assays, adjust QA/QC procedures to include more base metal standards in sample submissions.
   Risk Mitigated:
   - Lack of QA-QC data for base metals

8) Keep cash costs under control, consider sales contracts if possible

   Risk Mitigated:

   - Decreasing metal prices

9) Implement stringent protocols for blast hole sampling and ore control

   Risk Mitigated:

   - High degree of dilution

10) Test recycling of water in lock cycle tests

   Risk Mitigated:

   - Process water has unsuitable chemistry

25.0 INTERPRETATION AND CONCLUSIONS

This prefeasibility study indicates that the Gediktepe project is an economically robust project over a wide range of metal price assumptions and project cost estimates. Processing testing that was completed during the last year has developed a flow sheet and approach that produces marketable concentrates for both copper and zinc at reasonable process recoveries.

The heap leach component of the project can be quickly moved toward production with financial commitment to geotechnical data collection and additional metallurgical testing followed by more detailed engineering of the heap leach facility and oxide process plant design.

The development of sulfide mining and processing can be established during the oxide production period and consequently has several years available to complete: preproduction stripping, detailed testing, detailed engineering, and construction.

As a result of the prefeasibility study, a mineral reserve can be declared for the project. The mineral reserve is summarized below.

Table 25-1
Gediktepe Mineral Reserve, 1 June 2016

Classification	Cutoff NSR $/Tonne	**Oxide Mineral Reserves**					Payable Metal			
		Oxide Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	15.16	1,456	2.98	74.7	0.12	0.17	118.0	1,541.4		
Probable	15.16	1,767	2.93	80.3	0.18	0.35	133.6	2,010.9		
Proven+Probable	15.16	3,223	2.95	77.7	0.15	0.27	251.6	3,552.3		

Classification	Cutoff NSR $/Tonne	**Sulfide Mineral Reserves**					Payable Metal			
		Sulfide Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	14.55	10,425	0.84	31.0	1.04	2.05	64.3	1,924.6	160.2	326.6
Probable	14.55	11,267	1.00	39.3	0.93	2.63	83.4	2,724.8	154.6	452.6
Proven+Probable	14.55	21,692	0.93	35.3	0.99	2.35	147.7	4,649.4	314.8	779.2

Classification	Cutoff NSR $/Tonne	**TOTAL MINERAL RESERVES**					Payable Metal			
		Total Ktonnes	Gold gm/t	Silver gm/t	Copper %	Zinc %	Gold Kozs	Silver Kozs	Copper Mlbs	Zinc Mlbs
Proven	15.16/14.55	11,881	1.11	36.3	0.93	1.82	182.3	3,466.0	160.2	326.6
Probable	15.16/14.55	13,034	1.26	44.9	0.83	2.32	217.0	4,735.6	154.6	452.6
Proven+Probable	15.16/14.55	24,915	1.19	40.8	0.88	2.08	399.3	8,201.7	314.8	779.2

Notes:

Mineral Reserve Based on Metal Prices of:
$1,000/oz Gold, $15.00/oz Silver, $2.50/lb Copper, $1.00/lb Zinc
Payable Metal is not shown for copper and zinc in the oxide zone because there is no plan to recover copper or zinc from the oxide zone. Their grades are shown because copper and zinc have an impact on the design of the oxide process and oxide process costs.
Pit slope angles are 48 degrees in fresh rock and 42 degrees in weathered rock
Ktonnes are 1000 metric tonnes
Mlbs are millions of pounds of copper and zinc metal
Kozs are 1000 troy ounces of gold and silver.

The qualified person for the mineral reserve is John Marek of Independent Mining Consultants, Inc. The mineral reserve could change as more drilling and engineering is completed. Metal prices could materially change the mineral reserve in a positive or negative way.

The PFS plan is contained within the project license area and is a compact mine and process facility. Infrastructural design and evaluation is sufficiently developed that an execution plan can be established. There will however, be changes to the execution plan as more detailed information becomes available for advanced project design.

The project financial analysis reports the following results in summarized form:

ROI After Tax:	46.5%
Non-discounted Payback	2.5 Years
Initial Capital	$119.7 Million USD
Sustaining Capital	$148.8 Million USD

Sustaining capital includes the construction of the sulfide processing plant.
These parameters have been established at metal prices of:

Au: $1,250/oz, Cu: $2.75/lb, Ag: $18.25/oz. Zn: $1.00/lb.

As a sensitivity, Polimetal desired to also present the project financial results at more conservative metal prices. The conservative metal prices used in the evaluation are: $950/oz. Au, $13.50/oz. Ag, $2.25/lb Cu, and $0.80/lb Zn. The after tax economic results at the lower prices are summarized:

ROI After Tax:	28.9%
Non-discounted Payback	3.3 Years

26.0 RECOMMENDATIONS

This prefeasibility study indicates that the Gediktepe project is an economically robust project over a wide range of metal price assumptions and project cost estimates. This section lists many of the items that should be completed should Polimetal decide to continue development of the project. Estimated costs for continued development of the project are accounted for in Section 21.2.7 of this report.

1) Complete many more check assays on base metals within the existing data base. Issue pulps to a 3rd party lab on a 1 in 20 to 25 basis.

2) Insert base metal standards in the sample stream during Phase 4 drilling amounting to 1 out of 20 or 25 of the samples submitted.

3) Complete the Phase 2 geotechnical investigation program.

4) Complete detailed site wide water balance and design project water requirements.

5) Complete detailed engineering of the heap leach facility with the results of Phase 2 geotechnical drilling.

6) Complete detailed engineering of oxide processing circuits.

7) Apply for environmental permits as time and engineering warrants.

8) Perform large scale column leach tests.

9) Perform characterization of agglomerated oxide ore.

10) Complete additional lock cycle testing on sulfides, including multi-element analysis of concentrates.

11) Complete additional lock cycle test to determine the effect of recycling water

12) Complete geometallurgical testing of sulfide ores.

13) Consider pilot plant scale testing of the flotation plant.

14) Perform tailings characterization.

15) Update the project execution plan on a regular basis as the above information becomes available.

16) Perform more detailed engineering regarding the mine camp including requirements for water, sewage, buildings, etc.

17) Consider lowering the elevation of the crusher location to be nearer in elevation to the pit exit.

27.0 REFERENCES

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28.0 SIGNATURES AND CERTIFICATES

DATE AND SIGNATURE PAGE

The effective date of the report titled "Technical Report, Prefeasibility Study Gediktepe Project, Balikesir Province, Turkey" is June 1, 2016.

Signed this 13 of September, 2016

"signed and sealed"
Deepak Malhotra, RM SME **Resource Development Inc.**
"signed and sealed"
Terry Mandziak, P.E. Colorado **SRK Consulting (U.S.), Inc.**
"signed and sealed"
John Marek, RM SME **Independent Mining Consultants, Inc.**

Resource Development Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Dr Deepak Malhotra, do hereby certify that:

1. I am currently employed as President of Resource Development Inc. (RDi) with the business address:

   Resource Development Inc. (RDi)
   11475 W. I-70 Frontage Road North
   Wheat Ridge, CO 80033

2. This certificate applies to the Technical Report titled "Prefeasibility Study Gediktepe Project, Balikesir Province, Turkey" with an effective date of June 1, 2016.

3. I graduated with a Master of Science in Metallurgical Engineering in 1973 and a PhD in Mineral Economics in 1977 from Colorado School of Mines.

4. I am a Registered Member of the Society of Mining, Metallurgy and Exploration Inc. (SME) and Canadian Institute of Mining, Metallurgy and Petroleum (CIM) in good standing.

5. I have worked as a metallurgical engineer/mineral economist for over 43 years.

6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

7. I am the Qualified Person (QP) for Sections 13, 17 and parts of 21 (Metallurgy) of the Technical Report.

8. I visited the project site for two days in August 2014 and one day in September 2015.

9. I have worked on Gediktepe previously, acting as a qualified person for "Mineral Resource and Preliminary Economic Assessment, Gediktepe Project, Balikesir Province, Turkey" dated December 22, 2014.

10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

11. I am independent of the issuer applying all the tests in Section 1.5 of the National Instrument 43-101.

12. I have read National Instrument 43-101 and Form 43-101 F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

14. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 13 September, 2016

"Signed"

Dr Deepak Malhotra, Ph.D.
SME Registered Member, Membership No. 2006420
Principal Metallurgist/Mineral Economist
President, Resource Development Inc.

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F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Terry Mandziak, PE do hereby certify that:

1. I am Associate Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "Technical Report, Prefeasibility Study Gediktepe Project, Balikesir Province, Turkey" with an Effective Date of June 1, 2016 (the "Technical Report").
3. I graduated with a degree in Bachelor of Science in Civil Engineering from the University of Saskatchewan in 1989 and a Master of Science from the University of Saskatchewan in 1991. I am a Registered Professional Engineer in the state of Colorado (#31709). I have worked as an Engineer for a total of 24 years since my graduation from university. My relevant experience includes the design of heap leach and tailings facilities.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Gediktepe property on August 28-29, 2014 for 2 days.
6. I am responsible for the preparation of Tailings Storage Facility and Heap Leach portions of Sections 1, 18 and 21 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my involvement was as a qualified person for the technical report "Mineral Resource and Preliminary Economic Assessment, Gediktepe Project, Balikesir Province, Turkey," dated December 22, 2014
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th Day of September, 2016.

"Signed"

Terry Mandziak, PE [CO 31709]
Principal Consultant (Geotechnical Engineer)

CERTIFICATE OF QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

1. I am currently employed as the President and a Senior Mining Engineer by:

   Independent Mining Consultants, Inc.
   3560 E. Gas Road
   Tucson, Arizona, USA 85714

2. I graduated with the following degrees from the Colorado School of Mines
   Bachelors of Science, Mineral Engineering – Physics 1974
   Masters of Science, Mining Engineering 1976

3. I am a Registered Professional Mining Engineer in the State of Arizona USA
   Registration # 12772
   I am a Registered Professional Engineer in the State of Colorado USA
   Registration # 16191

   I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers, Registration # 2021600

4. I have worked as a mining engineer, geoscientist, and reserve estimation specialist for more than 40 years. I have managed drill programs, overseen sampling programs, and interpreted geologic occurrences in both precious metals and base metals for numerous projects over that time frame. My advanced training at the university included geostatistics and I have built upon that initial training as a resource modeler and reserve estimation specialist in base and precious metals for my entire career. I have acted as the Qualified Person on these topics for numerous Technical Reports.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for sections 1 through 12, 14 through 16, and 18 through 27 of the Technical Report titled "Technical Report, Prefeasibility Study Gediktepe Project, Balikesir Province, Turkey", with an effective date of June 1, 2016.

7. I visited the Gediktepe property on July 8, 2014 during which time I reviewed the drill core, core handling procedures, sample preparation, core logging and site conditions.

8. Independent Mining Consultants, Inc., and this author have worked on Gediktepe previously, acting as a qualified person for “Mineral Resource and Preliminary Economic Assessment, Gediktepe Project, Balikesir Province, Turkey”, dated December 22, 2014

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

10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

11. I am independent of the issuer applying the tests in Section 1.5 of NI 43-101.

12. I have read national Instrument 43-101 and Form 43-101F1, and the sections of the Technical Report that I am responsible for been prepared in compliance with that instrument and form.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: 13 September, 2016.

“Signed”

John M. Marek
Registered Member of the SME