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Technical Report for the Didipio Project

Located in Luzon,
PHILIPPINES

Prepared for
OceanaGold Corporation

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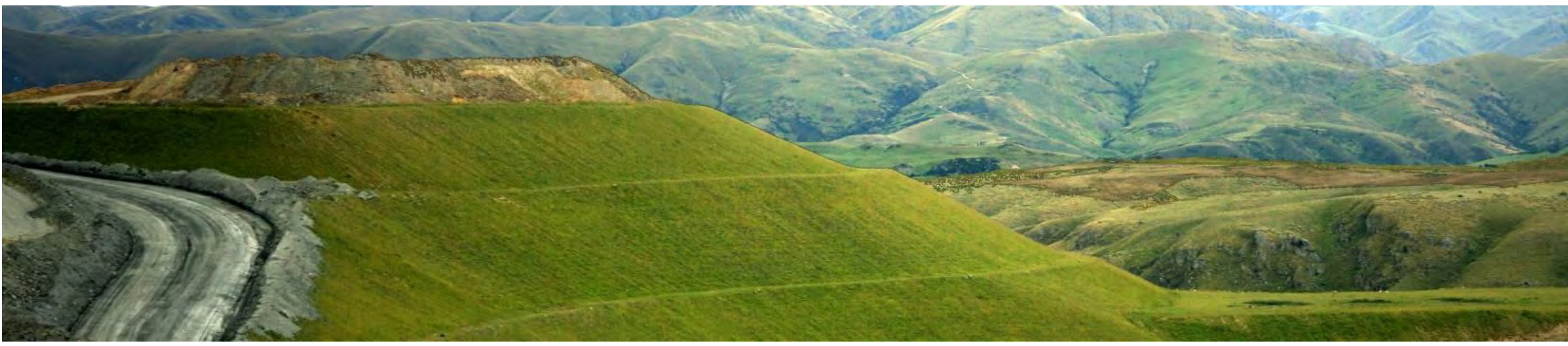
Melbourne, Victoria
AUSTRALIA

July 29, 2011



R Redden (BE (Min) Hon, MBA AusIMM, CP), Development and Technical Services Manager, OceanaGold (New Zealand) Limited.

J Moore (BSc(Hons), Grad Dip (Physics), CP AusIMM), Group Mine Geology Manager, OceanaGold (New Zealand) Limited.

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TECHNICAL REPORT CERTIFICATION AND SIGN OFF

The effective date of this Technical Report and sign off is July 29, 2011.

29/07/2011

Rodney Thomas REDDEN

Date of Signature: July 29, 2011

29/7/2011

Jonathan Godfrey MOORE

Date of Signature: July 29, 2011

CERTIFICATE OF AUTHOR

As a qualified person responsible for the report "Technical Report for the Didipio Project" dated July 29, 2011, (the "Technical Report") to which this certificate applies, I, Rodney Thomas Redden do hereby certify that:

1. I, Rodney Thomas Redden, am the Exploration and Development Manager of Oceana Gold (New Zealand) Limited. My business address is OceanaGold, Taunton Mews, 22 Maclaggan Street, Dunedin, New Zealand.

2. I graduated with a B.E (Mining), Hons degree from the University of Wollongong, Australia in 1993. Subsequently I received a postgraduate Master of Business Administration also from the University of Wollongong in 2005. I am the holder of a 1st Class Mine Managers Certificate for New Zealand and Western Australia

4. I am a member in good standing of the Australasian Institute of Mining and Metallurgy.

5. I have worked as an open pit and underground engineer in various countries and metals in the mining industry and more recently as a manager of various aspects of mining operations continuously for a total of 18 years since my graduation.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and confirm 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.

7. Since my employment with OceanaGold in 2004-2005 and 2007 I have been to site a number of times, most recently in November 2010.

8. I am responsible for the sections detailed in section 0 of the Technical Report.

9. I am not independent of OceanaGold Corporation applying all the tests in section 1.4 of NI 43-101 because I am an employee of Oceana Gold (New Zealand) Limited.

10. Prior to my employment with OceanaGold in November 2004 I had no involvement with the Didipio Project.

11. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.

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

29/07/2011

Rodney Thomas REDDEN

Date of Signature: July 29, 2011

CERTIFICATE OF AUTHOR

Jonathan Godfrey Moore
Group Mine Geology Manager
22 Maclaggan Street, Dunedin
NEW ZEALAND

As a qualified person responsible for the report titled "Technical Report for the Didipio Project" dated 29 July 2011, (the "Technical Report") to which this certificate applies, I, Jonathan Godfrey Moore do hereby certify that:

1. I, Jonathan Godfrey Moore, am the Group Mine Geology Manager for OceanaGold Corporation.
2. I graduated with a BSc (Hons) Mining degree in geology from the University of Otago in 1985 and a Graduate Diploma (Physics) in 1993 also from the University of Otago.
3. I am a member and chartered professional in good standing with the Australasian Institute of Mining and Metallurgy.
4. I have worked as a geologist in the mining industry for over 20 years since my graduation.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and confirm 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.
6. My most recent personal inspection of the Didipio Project was in August 2008, although I spent 2 days at the OceanaGold core facility nearby at Cordon in June 2011.
7. I am responsible for the sections set out in section 2.3.1of the Technical Report.
8. I am not independent of OceanaGold Corporation applying all the tests in section 1.5 of NI 43-101 because I am an employee of Oceana Gold (New Zealand) Limited.
9. Prior to my employment with OceanaGold in May 1996 I had no involvement with the Didipio Project.
10. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portions 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.

29/07/2011

Jonathan Godfrey MOORE

Date of Signature: 29 July, 2011

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1 SUMMARY

1.1 Description of Property

1.1.1 Location

The Didipio Project is located in the north of Luzon Island approximately 270km NNE of Manila, in the Philippines.

The Financial or Technical Assistance Agreement (FTAA) now covers about 158km^2 (compared with the original 370km^2) located in the Provinces of Nueva Vizcaya and Quirino. The proposed mining area comprises 9.75 km^2 within the FTAA.

1.1.2 Ownership

A FTAA was made and entered into by and between the Republic of the Philippines and Climax Arimco Mining Corporation (CAMC) on 20 June 1994. The FTAA was subsequently assigned by CAMC to Australasian Philippines Mining Inc (APMI) (renamed OceanaGold Philippines Inc. (OGPI)), now a wholly owned subsidiary of OceanaGold Corporation (OGC). The FTAA covers an original area of 370km^2 located in the Provinces of Nueva Vizcaya and Quirino. Over the years of exploration and subsequent relinquishments, the remaining area covers only about 158km^2.

All the primary requirements to be fulfilled under the FTAA have been met. Surface rights have been acquired for the vast majority of the land required for the current expanded project. Purchase agreements are being negotiated with remaining landowners. The Company expects to have acquired all the land required by the current project footprint in the near future.

A third party has a contractual right to be granted an 8% free carried interest in the operating vehicle that will be formed to undertake the management, development, mining and processing of ore on, and the marketing of products from, the Didipio Project, subject to satisfaction of certain conditions.

1.2 Geology and Mineralisation

1.2.1 Didipio Project Geology

The geology of the Didipio region is typical of an island arc setting, consisting of volcanic, volcaniclastic and sedimentary rocks, intruded by porphyries of intermediate to felsic composition.

The Didipio Gold-Copper Deposit is hosted within a multiphase stock, which is in turn part of a larger alkalic intrusive body, the Didipio Igneous Complex.

1.2.2 Didipio Project Deposit Mineralisation

Chalcopyrite and gold are the main economic minerals in the deposit. Chalcopyrite occurs as fine-grained disseminations, aggregates, fracture fillings and stock work veins, particularly within the vein zone of alteration.

Chalcopyrite can replace magnetite and is, in turn, replaced by bornite. Bornite occurs as alteration rims around and along fractures within chalcopyrite grains.

1.2.3 Surface Oxidation

The deposit is oxidised from the surface to a depth of between 15m and 60m, averaging 30m. The oxide zone forms a blanket over the top of the deposit and largely comprises silicification, clay and carbonate minerals, accompanied by secondary copper minerals including malachite and chrysocolla.

1.2.4 Primary Ore

Primary ore is the dominant ore in this deposit and is considered relatively clay free. High clay or high moisture contents in the ore are not expected apart from the influence of high rainfall events on operations.

1.2.5 Resource Estimate

The drill hole database for the Didipio Gold-Copper Deposit, used for resource estimation, comprises 98 diamond core holes totalling 39,421.2m. 10,047 three-metre composited samples were used for modelling. Holes are drilled on nominal 50m EW sections with a vertical separation of 120m to 180m between holes, except in the higher-grade core area, where separation was reduced to approximately 50m.

Two or three metre cut core samples (half or quarter) were assayed for gold by fire assay with AAS (atomic absorption spectroscopy) finish and for copper by acid digest and AAS. Quality control measures confirm these results are reasonably accurate and precise. Drill collars were surveyed by a registered surveyor and hole deviation was measured using a down hole camera. Approximately 2300 rock density measurements were made at 5m-10m intervals. Of these, paper records were found for only 1173. These records were used to assign bulk densities for the OGC resource model by rock type.

The resource was classified in accordance with CIM standards. This summary section should be read in conjunction with the total report in order to understand all the necessary technical and commercial information relevant to estimating the resources. Table 1.1 reports total mineral resources at a 0.4 g/t eqAu cut-off grade above 2,390mRL, and at 1.5 g/t eqAu cut-off grade below 2,390mRL and above the 2,180mRL where the gold equivalence is eqAu = g/t Au + 2.06 x % Cu. This contained gold equivalence is based on metal prices of US$950 per ounce for gold and US$2.85 per pound of copper.

Table 1.1: Didipio Project Mineral Resources at 0.4 g/t eqAu and 1.5 g/t eqAu Cut-Off

Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (kt)
Measured	15.96	1.67	0.56	0.86	90.0
Indicated	54.21	0.73	0.37	1.27	201.1
Measured & Indicated	**70.17**	**0.95**	**0.41**	**2.13**	**291.0**
Inferred	30.73	0.44	0.23	0.44	72.1

Note: Resources are reported down to 2,180mRL, which represents the base of the proposed sublevel open stope mine. A cut-off of 0.4g/t eqAu has been used above the 2,390mRL and 1.5g/t eqAu cut-off below the 2,390mRL.

All mineral reserves reported are included within the mineral resources reported for the same deposit.

1.3 Exploration Concept

The Didipio Gold-Copper Deposit is an alkalic porphyry deposit that lies at the margin of the Surong stock near the juncture of the Biak Shear Zone and Tatts Fault. While the lateral bounds of the Didipio Gold-Copper Deposit have been well defined through drilling, the depth extent is less well defined and there remains limited potential to increase the resource beyond the current depth.

Some porphyry deposits form „camps‟ and it is not uncommon for the discovery of additional porphyry mineralisation in geologically favourable locations around the margin of a large parent stock such as the Surong stock. This concept is a valid exploration strategy. More than 30 exploration targets have been identified within the FTAA, through a combination of stream sediment sampling, soil sampling, rock-chip sampling, and for a small number of prospects, limited drilling. The predominant mineralisation style of these targets is interpreted to comprise alkalic-associated porphyry-style mineralisation and higher-level associated epithermal gold mineralisation. Only a few of these targets have received follow-up exploration.

It is beyond the intended scope of this document to address the exploration potential of the FTAA outside the immediate confines of the Didipio Project.

1.4 Operations

1.4.1 Mining

OceanaGold has prepared a revised mine plan and estimates for the Didipio Project. The revised mine plan is based on a 300m deep open cut down to an elevation of 2380mRL and underground mining by sublevel open stoping beyond the open cut.

1.4.2 Open Cut and Underground Mines

Open cut mining will be by conventional drill and blast with loading of haul trucks by hydraulic excavators. The open cut design was guided by the results of Whittle Four-X pit optimisation.

The underground mine plan is based on extraction of the deposit by sublevel open stoping with paste backfilling, with haulage of the ore and waste up a decline in diesel trucks that are loaded by diesel loaders.

1.4.3 Waste Dump

During the pre-production year, open cut waste rock will be used to form the ROM stockpile base and the start of the tailings dam wall. From the start of Year 1, part of the waste will be used to heighten the tailings dam wall. The remainder will be placed in a dump to be formed off the downstream (eastern) face of the tailings dam wall between the wall and the pit, providing a buttress for the tailings dam wall.

Underground mining will be concurrent with open pit in the last seven years of operations. Underground waste will be stacked together with open cut waste.

1.4.4 Ore Processing

A mineral processing facility will be constructed to the north of the open cut mine. Ore processing will be by conventional SAG/ball mill grinding circuit followed by froth flotation for recovery of gold/copper concentrate. A gravity circuit is incorporated within the grinding circuit to produce gold bullion on site. Concentrate will be transported by road to existing port facilities for export.

1.4.5 Tailings Disposal

Flotation tailings will be thickened at the processing plant prior to being pumped to tailings storage facility constructed approximately three kilometres distance from the plant. Process water from the tailings dam will be re-cycled to the processing plant.

A significant amount of tailings will be stored underground as a paste tailings backfill once underground mining commences.

1.4.6 Mineral Reserves

The mineral reserves are summarised in Table 1.2.

Table 1.2: Mineral Reserves

Source	Reserve Class	Tonnes	Au (g/t)	Cu (%)	Gold (Moz)	Copper (kt)
Open Pit	Proven	13,790,000	1.60	0.59	0.71	81
	Probable	30,950,000	0.55	0.39	0.55	121
Underground	Probable	5,910,000	2.25	0.45	0.43	27
Total Proven		13,790,000	1.60	0.59	0.71	81
Total Probable		36,860,000	0.82	0.40	0.97	148
Total Proven and Probable		50,650,000	1.03	0.45	1.68	229

*notes: Reserves are based on the following metal price assumptions. $950/Ounce Au and $2.85/lb Cu. Using a copper to gold equivalance factor of Au (g/t) eq = **2.08** X Cu (%), the Cut-off grade for the open pit reserve is 0.5g/t **AuEq** and for the underground 1.9g/t **AuEq.***

Circa 400kt @ 0.67g/t Au and 0.24% Cu of underground low grade "mineralised waste" is part of the ore inventory as it is necessary underground development material that must come to surface, at which point it is economically attractive to treat it rather than send it to waste based on processing, overheads and concentrate costs.

1.4.7 Schedule

The overall mine life will be about 17 years with the open pit operating on its own for the first 4 years. Underground development commences with minimal ore in years 5 to 7. Open pit and underground activities are parallel until the pit is completed in year 12. Final stock-piles will be consumed in years 16 and 17.

Key points to note in the development sequence include:

- Waste mining commences in the open cut nine months in advance of the start of ore processing to provide fill for construction of the tailings dam wall and to establish a stockpile of ore.
- The processing plant ramps up production using open cut ore from 2.5 Mtpa in 2013 to 3.5 Mtpa by year 3 (2015).
- Portal site preparation at RL2690 is completed as part of stage 4 open cut pre-strip in year 4 (2016).
- Decline and underground infrastructure development continues through year 7 (2019) when the first underground stopes come into production. Development ore is first available in year 6 (2018).
- First underground production ore is mined from the stopes in year 7 (2019). The open cut is still in full production at this time.
- The process plant continues to process 3.5 Mtpa of combined underground and open cut feed. The underground component increases to planned full production rate of 1.2 Mtpa by the end of year 9 (2021).
- When the open pit is depleted in year 12 (2024), open cut feed is replaced by lower grade ore stockpiled in earlier production years. Processing rate continues at 3.5 Mtpa using 1.2 Mtpa underground ore supplemented by low grade stockpile.
- Underground mine production ramps down in years 15 and 16 (2027 and 2028) when the last stopes are completed
- Underground ore feed is replaced by remaining low grade stockpiles, which are finally depleted in year 17 (2029).

1.5 Status of Project Development

Project development became active again in late 2010 with a project construction team assembled in-house to undertake the implementation designs and to prepare for construction. As of the writing of this report, the general status of the project is:

- The OGC in-house construction team is assembled and currently based in Manila and on-site. The team is lead by Martyn Creaney.
- General Manager of Operations, Brennan Lang has been appointed and started in early June.
- Ausenco has been contracted and is delivering on an engineering and procurement contract for the process plant engineering and design.
- Both grinding mills have been purchased and are in storage.
- Crusher and concentrate filters have been ordered.
- The process plant site earthworks are more than 80% complete.
- GHD have conceptually designed and completed specifications for the tailings storage facility (TSF) and the flow-through waste dump.
- Repairs to the main access road are underway, though the road is accessible for heavy traffic at present.
- The diesel-powered site power station contract has been awarded to VeryPower from China.
- A six year mining contract is out for tender to three pre-selected Philippines experienced contractors.
- Surface rights acquisition is underway and is close to completion.
- A document to modify the Environmental Compliance Certificate has been lodged with the relevant Philippines authorities.

1.6 Costs

1.6.1 Project Capital Costs

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented in this report.

Table 1.3: Capital Cost Estimate Summary, June 2011

Item	US$M
Mining	19.13
Process plant	32.98
Power plant	10.51
Tailings Storage Facility (TSF)	7.12
Infrastructure & Services	28.09
Indirect costs	87.34
Total	**185.17**
Money spent/invoiced up to Jun/11	(19.16)
Remaining Capex	**166.01**

- The estimate does not include allowance for escalation during the construction period. However, the construction period is relatively short; and
- The total estimate does not include additional working capital associated with the start-up of the operation.

1.6.2 Working Capital

There is an allowance for working capital representing approximately 9% of revenue over two years of production in the financial model.

1.6.3 Deferred Capital and Sustaining Capital

Ongoing deferred and sustaining capital expenditure is estimated at US$161.5M, relating principally to ongoing capital expenditure for the underground mine, process plant, the TSF and an allowance for closure costs at the end of life of the mine.

1.6.4 Operating Costs

The operating cost estimates developed by OGC are summarised in
Table **1.4**.

Table 1.4: Projected Operating Costs

Sections		Life of Mine	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
Volumes																			
Total Material Mined	Mt	207.8	8.8	20.1	23.1	22.4	23.6	23.2	20.1	19.9	22.4	11.3	3.7	3.5	2.5	1.2	1.0	0.6	0.2
Total Ore Mined	Mt	52.9	1.6	8.6	5.8	4.1	5.8	2.4	1.8	4.4	4.4	2.3	2.8	3.3	2.5	1.2	1.0	0.6	0.2
Open Cut - Total Ore Mined	Mt	44.7	1.6	8.6	5.8	4.1	5.8	2.4	1.7	4.2	3.9	1.4	1.7	2.1	1.3	0.0	0.0	0.0	0.0
Open Cut - Total Waste Mined	Mt	154.9	7.2	11.5	17.3	18.3	17.8	20.9	18.3	15.5	17.9	9.0	0.9	0.2	0.0	0.0	0.0	0.0	0.0
Open Cut - Total Material Mined	Mt	199.6	8.8	20.1	23.1	22.4	23.6	23.2	20.0	19.7	21.8	10.5	2.6	2.3	1.3	0.0	0.0	0.0	0.0
Open Cut - Total Material Moved (incl. rehandling)	Mt	219.6	8.8	20.1	23.5	23.5	23.8	25.0	22.2	20.7	21.8	12.1	3.6	2.8	2.4	2.3	2.5	2.9	1.4
Underground production	Mt	8.2	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.2	0.5	0.9	1.1	1.2	1.2	1.2	1.0	0.6	0.2
Total Ore Milled	Mt	52.9	0.3	2.5	3.1	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	1.6
Product sold																			
Gold in dore	Koz	436.4	0.8	21.5	26.3	28.7	27.5	29.1	29.3	14.5	29.9	24.0	34.5	50.1	49.4	23.6	22.2	16.4	8.5
Gold in concentrate	Koz	1134.2	3.4	54.5	66.4	74.6	68.5	70.5	71.1	33.2	79.8	60.4	95.4	131.0	128.9	67.9	61.3	45.4	21.8
Copper in concentrate	Mlb	482.7	2.3	38.1	41.3	40.8	39.1	37.3	37.0	29.3	34.0	27.5	24.9	30.2	32.6	22.9	21.7	17.4	6.3
Concentrate (dry) sold	Kt	888.7	4.4	72.0	78.0	77.1	73.9	70.5	70.8	55.8	66.9	51.3	45.5	56.8	61.7	37.1	29.6	27.4	9.9
Concentrate (wet) at mine gate	Kt	987.5	4.9	80.0	86.6	85.6	82.1	78.3	78.6	62.0	74.3	57.0	50.6	63.1	68.6	41.3	32.9	30.4	11.0
Mining costs																			
Open Cut	US$/t moved	2.2	0.6	2.1	2.1	2.1	2.1	1.7	2.0	2.3	2.1	2.9	4.1	4.4	4.6	3.1	2.6	2.4	3.6
Underground	US$/t mined	33.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	50.6	37.4	32.1	31.1	29.2	29.4	33.6	40.7	72.5
Processing costs																			
Power cost	US$/t milled	5.4	1.1	6.9	6.9	5.7	5.5	5.4	5.3	5.2	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.2	5.5
Reagents costs	US$/t milled	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
Spares costs	US$/t milled	1.8	1.7	2.0	1.8	1.9	1.9	1.8	1.9	1.8	1.9	1.9	1.9	1.8	1.9	1.8	1.8	1.8	1.8
Maintenance costs (ex owners team)	US$/t milled	0.2	0.0	0.3	0.2	0.2	0.2	0.1	0.2	0.1	0.2	0.1	0.2	0.1	0.2	0.1	0.2	0.1	0.1
Plant Labour	US$/t milled	0.6	1.2	0.8	0.6	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.8
Maintenance Labour	US$/t milled	0.4	0.8	0.5	0.4	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.6
Others	US$/t milled	0.3	0.7	0.4	0.3	0.3	0.3	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4
Unit Processing	US$/t milled	11.2	8.2	13.4	12.8	11.4	11.3	11.0	11.1	10.8	10.9	10.8	10.9	10.7	11.0	10.7	10.9	10.8	11.9
Other site costs																			
Overheads & site costs	US$/t milled	4.5	8.5	6.6	5.3	4.5	4.6	4.3	4.4	4.4	4.5	4.5	4.2	4.2	4.2	4.1	4.1	4.0	5.4
Logistics																			
Dore transport costs	US$ 000/month	10	Flat																
Land Transport & Ship loading	JS$/t concentrate	58.7	74.4	67.5	67.1	55.3	55.8	52.3	52.3	55.2	52.9	56.4	58.3	54.9	53.9	62.1	67.5	69.4	119.1
Sea Freight	JS$/t concentrate	30.0	Flat																
Concentrate agent fees & insurance	% of revenue	1%	Flat																
Terms of sales																			
Payable Copper in concentrate	%	96.7%	Flat																
Payable Gold in concentrate	%	97.5%	Flat																
Gold Dore refining charge	US$/oz	6.0	Flat																
Gold in concentrate refining charge	US$/oz	6.0	Flat																
Copper concentrate treatment charge (TC)	US$/dmt Con.	80.0	Flat																
Copper concentrate refining charge (RC)	Usc/lb cu	8.0	Flat																

[*] Commissioning phase starts in November 2012.

1.6.5 Cash Costs

The calculation of unit cash costs is detailed below and presented using two different price assumptions: one at spot prices (US$1530/oz Au and US$4.05/lb Cu) and another at long term estimate of US$1050/oz Au and US$3.0/lb Cu:

Table 1.5: Operating Costs

	Spot Price	Long Term Price
(+) Total Operating costs *US$M*	1,724	1,711
(+) Total Excise Duty & Royalties *US$M*	174	125
(+) Total Sales deduction	226	195
= Total Cash costs *US$M*	2,123	2,032
Total Gold sold *Moz*	1.57	1.57
Total Copper in concentrate sold *Mlb*	483	483
Total Equivalent Gold sales *Moz*	2.85	2.95
Cash costs per *EqAu* sold *US$/EqAu*	746	689
Copper Gross Revenue *US$M (price x volumes)*	1955	1448
Net of By product Cash costs per oz sold US$/oz	**107**	**372**

[*] total operating costs change is due to variable costs associated to revenue such as insurance

1.7 Economic Analysis

1.7.1 Assumptions

The financial analysis methodology, discount rates, exchange rates, commodity prices and financial parameters applied in the financial model were sourced from OGC. The inputs are consistent to the capital costs, operating costs and taxes sections of this report. The annual cash flow is based on spot prices (US$1530/oz Au and US$4.05/lb Cu).

1.7.2 Cash Flow

The financial analysis indicates that the project had a positive net cash flow and an acceptable internal rate of return and supports the declaration of mineral reserves, which were estimated with the following prices: US$950/oz Au and US$2.85/lb Cu.

The annual cash flow below is unleveraged pre tax, using spot prices (US$1530/oz Au and US$4.05/lb Cu) and covers the operating years of full production which is forecast to start in 2013.

Table 1.6: Annual Unleveraged Pre Tax Cash Flow and Annual Production

	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029
Pre-tax cash flow (US$m)	106	168	167	114	137	123	34	129	81	151	235	243	110	97	60	18	3
Production																	
Gold in Dore (Koz)	21.5	26.3	28.7	27.5	29.1	29.3	14.5	29.9	24.0	34.5	50.1	49.4	23.6	22.2	16.4	8.5	0.0
Gold in Concentrate (Koz)	55.0	67.0	75.4	69.2	71.2	71.8	33.5	80.6	61.0	96.4	132.4	130.2	68.6	61.9	45.8	22.0	0.0
Total Gold Produced (Koz)	76.5	93.3	104.1	96.7	100.3	101.1	48.1	110.5	85.0	130.9	182.5	179.6	92.3	84.1	62.2	30.5	0.0
Copper in concentrate (Mlb)	38.5	41.7	41.2	39.5	37.7	37.3	29.6	34.4	27.8	25.2	30.5	32.9	23.2	22.0	17.6	6.4	0.0

**table uses Mineral Resources of which >95% are Mineral Reserves*
NOTE: Variances to other Tables in this report are due to rounding

1.7.3 Net Present Value (NPV), Internal Rate of Return (IRR) and Payback

The results below are based on the unleveraged net cash flow post tax.

Table 1.7 to Table 1.9 indicate the NPV, IRR and Payback sensitivity of the Didipio Project to gold prices and copper prices.

Table 1.7: NPV Post Tax @ 10% – Sensitivities to Metal Prices (US$M)

		Copper price - US$/lb (flat)														
		2.20	2.40	2.60	2.80	3.00	3.20	3.40	3.60	3.80	4.00	4.20	4.40	4.60	4.80	5.00
Gold Price - US$/oz (flat)	950	-8	23	51	78	104	129	151	173	194	215	236	256	276	296	316
	1000	14	42	69	96	122	145	167	188	209	230	250	270	291	310	330
	1050	34	61	89	114	139	160	182	203	224	244	265	285	305	324	344
	1100	53	80	106	132	154	176	197	218	239	259	279	299	319	339	358
	1150	72	99	125	148	170	191	212	233	253	273	294	313	333	353	373
	1200	91	116	141	163	185	206	227	247	268	288	308	328	347	367	387
	1250	108	134	157	179	200	221	241	262	282	302	322	342	361	381	401
	1300	127	151	172	194	215	236	256	276	297	317	336	356	376	395	415
	1350	144	166	188	209	230	250	271	291	311	331	350	370	390	410	429
	1400	160	181	203	224	244	265	285	305	325	345	365	384	404	424	443
	1450	175	197	218	238	259	279	300	320	339	359	379	399	418	438	458
	1500	190	212	233	253	274	294	314	334	354	373	393	413	432	452	472
	1550	205	227	247	268	288	309	328	348	368	387	407	427	447	466	486
	1600	221	241	262	282	303	323	342	362	382	402	421	441	461	480	500
	1650	235	256	276	297	317	337	357	376	396	416	436	455	475	495	514
	1700	250	271	291	311	331	351	371	391	410	430	450	469	489	509	528
	1750	265	285	306	326	346	365	385	405	424	444	464	484	503	523	542
	1800	279	300	320	340	360	379	399	419	439	458	478	498	517	537	556
	1850	294	314	334	354	374	394	413	433	453	472	492	512	532	551	570
	1900	308	329	349	368	388	408	428	447	467	487	506	526	546	565	584
	1950	323	343	363	383	402	422	442	462	481	501	521	540	560	579	598

Table 1.8: IRR – Sensitivities to Metal Prices (%)

	Copper price - US$/lb (flat)														
Gold Price - US$/oz (flat)	2.20	2.40	2.60	2.80	3.00	3.20	3.40	3.60	3.80	4.00	4.20	4.40	4.60	4.80	5.00
950	9%	12%	15%	18%	21%	24%	26%	28%	31%	33%	35%	38%	40%	42%	44%
1000	11%	14%	17%	20%	23%	25%	28%	30%	32%	35%	37%	39%	41%	43%	45%
1050	13%	16%	19%	21%	24%	27%	29%	31%	34%	36%	38%	40%	40%	44%	44%
1100	15%	18%	20%	23%	26%	28%	31%	33%	35%	37%	39%	42%	44%	46%	48%
1150	17%	19%	22%	25%	27%	30%	32%	34%	36%	39%	41%	43%	45%	47%	49%
1200	18%	21%	24%	26%	29%	31%	33%	35%	38%	40%	42%	44%	46%	48%	50%
1250	20%	23%	25%	28%	30%	32%	35%	37%	39%	41%	43%	45%	47%	49%	51%
1300	22%	25%	27%	29%	32%	34%	36%	38%	40%	42%	44%	46%	48%	50%	52%
1350	24%	26%	28%	31%	33%	35%	37%	39%	42%	44%	46%	48%	50%	52%	54%
1400	25%	27%	30%	32%	34%	36%	39%	41%	43%	45%	47%	49%	51%	53%	55%
1450	27%	29%	31%	33%	35%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%
1500	28%	30%	32%	35%	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%
1550	29%	32%	34%	36%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%	58%
1600	31%	33%	35%	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%	59%
1650	32%	34%	36%	38%	41%	43%	45%	47%	49%	51%	53%	54%	56%	58%	60%
1700	33%	35%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%	58%	60%	62%
1750	35%	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%	59%	61%	63%
1800	36%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%	58%	60%	62%	64%
1850	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%	59%	61%	63%	65%
1900	38%	41%	43%	44%	46%	48%	50%	52%	54%	56%	58%	60%	62%	64%	66%
1950	40%	42%	44%	46%	48%	50%	52%	53%	55%	57%	59%	61%	63%	65%	67%

Table 1.9: Payback in Months (from Commissioning)

	Copper price - US$/lb (flat)														
Gold Price - US$/oz (flat)	2.20	2.40	2.60	2.80	3.00	3.20	3.40	3.60	3.80	4.00	4.20	4.40	4.60	4.80	5.00
950	124	74	58	49	40	36	34	32	30	28	27	25	24	23	22
1000	116	61	53	43	37	35	33	31	29	27	26	25	24	23	22
1050	68	56	47	39	36	33	31	30	28	27	25	24	23	22	21
1100	59	51	42	37	34	32	30	29	27	26	25	24	23	22	21
1150	54	46	38	36	33	31	29	28	27	25	24	23	22	21	21
1200	50	41	37	34	32	30	29	27	26	25	24	23	22	21	20
1250	44	38	35	33	31	29	28	26	25	24	23	22	21	21	20
1300	40	36	34	32	30	28	27	26	25	23	23	22	21	20	20
1350	37	35	33	31	29	28	26	25	24	23	22	21	21	20	19
1400	36	34	32	30	28	27	26	24	23	22	22	21	20	20	19
1450	35	32	31	29	28	26	25	24	23	22	21	21	20	19	19
1500	33	31	30	28	27	26	24	23	22	22	21	20	20	19	18
1550	32	30	29	27	26	25	24	23	22	21	20	20	19	19	18
1600	31	30	28	27	25	24	23	22	22	21	20	19	19	18	18
1650	30	29	27	26	25	24	23	22	21	20	20	19	19	18	18
1700	29	28	27	25	24	23	22	21	21	20	19	19	18	18	17
1750	29	27	26	25	24	23	22	21	20	20	19	19	18	18	17
1800	28	27	25	24	23	22	21	21	20	19	19	18	18	17	17
1850	27	26	25	24	23	22	21	20	20	19	19	18	18	17	17
1900	27	25	24	23	22	21	21	20	19	19	18	18	17	17	17
1950	26	25	24	23	22	21	20	20	19	19	18	18	17	17	17

1.8 Status of Exploration

More than 30 exploration targets have been identified within the FTAA through a combination of stream sediment sampling, soil sampling, rock-chip sampling and, for a small number of prospects, limited drilling. The predominant mineralisation style of these targets is interpreted to comprise epithermal gold and alkalic-associated porphyry-style mineralisation. Only a few of these targets have received follow-up exploration.

Two more advanced alkalic porphyry deposits („True Blue" and „D'Fox" deposits) have been partially drill tested. These are located approximately 0.6km and 3.2km respectively from the Didipio Gold-Copper Deposit. Limited drilling has demonstrated these are low-grade alkalic porphyry deposits that may have future potential to provide supplementary feed to the Didipio operation.

1.9 Conclusions and Recommendations

1.9.1 Exploration and Resources

The Didipio Project contains significant mineral resources defined by existing data in the Didipio Ridge deposit. There is some potential for expanding existing resources as well as converting some Inferred resources to Indicated classification.

There is significant potential to discover and define additional resources within the Didipio project area at a number of other nearby prospects.

The existing database at the time of the resource estimate for Didipio Ridge is considered satisfactory for resource estimation, although some minor issues with data completeness and quality remained to be resolved.

1.9.2 Other

The open pit mining operation is relatively straight forward and the ultimate pit design is smaller than the most economic case, meaning that it is extremely robust and insulated to metal price reductions or increased operating costs. The pit walls slopes are fairly conservative, so there exists some potential upside that should cancel out any localised small scale wall problems.

The underground operation is a higher level of complexity again using paste backfill, it is more exposed to metal price decreases or operating cost increases, however the selective nature of the mining method will allow mining to adjust to a degree.

The Company controls the vast majority of land in the mining footprint to support the Didipio Project and is confident of acquiring a small number of outstanding parcels in the near term, as all parties are in negotiations. The impact on the project will be either increased capital cost which should be relatively minor or a delay to some areas in initial construction while negotiations are complete.

OGC was granted an Environmental Compliance Certificate in 2004, the current project requires modifications to this certificate. Consideration by the authorities is currently in progress. If these modifications are not approved the project may need to be altered.

The Philippines central government continues to be supportive of mining both in general and with OceanaGold. The timely development of the project will depend on local community support. To this end, OGC has been actively engaged with the community.

The capital cost estimates, including the working capital allowances in the first two years, have been recently prepared and construction is underway. Potential cost and time overruns should not materially affect the robust economics of the project, but refer to Table 22.6 for these sensitivities.

It would be prudent to follow up on recommendations in section 26 to ensure total project capital is accurately considered.

2 INTRODUCTION

2.1 For Whom the Report has been Prepared

This report has been prepared at the request of OceanaGold Corporation (OGC). The report is for the general investing community use. It provides an update on the status of the Didipio Project and will be lodged with SEDAR as per TSX requirements.

References in this report to „OceanaGold‟ include Oceana Gold Limited, OceanaGold Corporation, OceanaGold (Philippines) Inc. and its subsidiaries and associates, as the context requires.

2.2 Purpose of Report

This report has been prepared to satisfy OGC‟s obligations as a reporting issuer in Canada.

2.3 Sources of Information

Jonathan Moore, Group Mine Geology Manager for OceanaGold, and Rodney Redden, Development and Technical Services Manager for OceanaGold, have been involved in the preparation of this report. Each is listed below with their respective items of responsibility and sources of information.

2.3.1 Jonathan Moore

Mr Moore holds a BSc (Hons) in Geology, a GradDip in Physics and has over 20 years of experience in exploration, open pit and underground mining and resource geology. He has worked in epithermal gold, porphyry copper and gold, mesothermal gold and lead-zinc deposits within Australia, New Zealand and the Philippines.

Mr Moore has been employed with OceanaGold since 1996 in a variety of project, mine geology, exploration and resource geology roles. He is the Group Mine Geology Manager.

Mr Moore is the qualified person for resources and geology including the following sections of this report: 1.2, 1.3, 1.8, 1.9.1, 6 to 12, 14, 23, 25, 26.1, 26.3.

- No sources

2.3.2 Rodney Redden

Rodney Redden has a BE, mining (hons) and an MBA. He has worked as a mining engineer and project Manager for various underground and open pit projects in Australia, Tanzania, New Zealand, Kazakstan, Russia and Philippines.

He is the qualified person for reserves, mine design, scheduling, costs, financial evaluation, processing and tailings dam construction, environmental aspects of the report. This includes sections 1.1, 1.4 to 1.7, 1.9.2, 4, 5, 13, 15 to 22, 25, 26.2, 26.3.

Sources were:

- Jonathan Moore for the mineral resource model;
- Ausenco for processing aspects;
- RDCL for open pit geotechnical aspects;
- Australian Mining Consultants for underground mining geotechnical aspects;
- Meyer Water & Environmental Solutions for surface water and groundwater aspects;
- GHD for tailings dam and waste dump designs;
- Martyn Creaney (Didipio Construction Project Director), Construction cost estimates ;
- OGC corporate for shipping;
- Waste characterisation, Department of Mineral Resources, New South Wales Mineral Resources Development Laboratory;
- Brad Norman for Surface Rights Acquisition; and
- Romanchito Gozar and David Cook for Environmental and Permitting.

Other reports relied upon include:

- Dinkidi Deposit, Preliminary Geotechnical Assessment – K Rosengren and Associates, January 1994.
- Geochemical Assessment of Process Tailings Didipio Gold-Copper Project – EGI, April 1995.
- Didipio Gold-Copper Project Dinkidi Deposit, Geotechnical Report for Open Pit Mining – Barrett Fuller and Partners, September 1995.
- Dinkidi Deposit, Geotechnical Review, Report Number 95016 – K Rosengren and Assocs, September 1995.
- Environmental Impact Statement and Scoping Study for CAMC‟s Didipio Gold-Copper Project – Maunsell Philippines Inc., December and May 1997.
- Dinkidi Proposed Open Pit Geotechnical Review – K Rosengren and Associates, July 2003.
- Environmental Impact Statement Amendments for CAMC‟s Didipio Gold-Copper Project – Gaia South Inc., July 1999 and April 2004.
- Environmental Compliance Certificate 9801-001-301, Aug 1999, amended Jan 2000, revised Aug 2004.
- Environmental Protection and Enhancement Programme (EPEP) for the Didipio Gold-Copper Project – approved by DENR, January 2005.
- Didipio Gold-Copper Project Feasibility Study Report – Ausenco Limited, February/March 2005.
- Didipio Gold-Copper Project Definitive Feasibility Study Report – Ausenco Limited, July 2005.
- Process Design Criteria – Ausenco Ltd, March 2006.
- Flotation and Comminution Testing of Didipio Gold-Copper Ore for CMS – AMMTEC Ltd, June 2006.

2.4 Scope of Inspection

2.4.1 Rodney Redden

A number of visits have been made to the Didipio site, most recently in November 2010. These have been with various mining contractors and have included reviews of:

- dump locations;
- haul roads;
- the Processing plant site;
- the tailings dam site; and
- Infrastructure such as Accommodation village, explosives magazines, river crossings.

2.4.2 Jonathan Moore

A number of weeks were spent inspecting core at Cordon core facility during 2008. During this time two brief visits were made to the Didipio site. The last visit to the Didipio site was in August 2008, although 2 days were spent at the OceanaGold core facility nearby at Cordon in June 2011.

3 RELIANCE ON OTHER EXPERTS

Jonathan Moore, Group Mine Geology Manager for OceanaGold, and Rodney Redden, Development and Technical Services Manager for OceanaGold have been involved in the preparation of this report. Their reliance on experts is listed below.

3.1 Rodney Redden

Mr Redden has relied, and believes he has a reasonable basis to rely, on information provided by the following third parties for the following areas of the report.

Table 3.1: Reliance on third Parties

Source	Context
Jonathan Moore of OceanaGold Corporation	The resource model used for mine planning and reserves reporting
Groundwater Flow Model Mine dewatering and site water supply Didipio Project June 2011, MWES Consulting, 1 June 2011	groundwater management for open cut and underground mining
Surface water flow Model – Prior to Construction Didipio Project May 2011, MWES Consulting, 24th May 2011	Surface water flow modelling, site wide water balance, the modelling of contaminant concentrations
Report for Didipio Project ECC – Concept Design Report April 2011, GHD	Design, construction and operation of the tailings dam and waste dump design incorporating the flow thru concept
Marcelo Ramos, internal communications (14 July – Royalties)	Royalties (section 4), taxation (section 22) shipping costs
Resource Development Consultants Limited „Geotechnical Assessment of the Proposed Didipio Open Pit", dated November 2008	Open Pit geotechnical, in particular for wall slope angles and safety berm widths in different types of rockmass
Australian Mining Consultants Pty Ltd	Underground geotechnical, in particular for stable stope sizes before backfilling is required
"Evaluation of Potential of Waste and Mineralised Waste to Produce Saline and Acid Mine Drainage", R Mountford & C Wall, 14 Nov 1994, Report No: R941411A, Department of Mineral Resources, New South Wales Mineral Resources Development Laboratory	Waste rock charateristation, that the waste rock does not produce an acidic leachate after exposure to air and water
Didipio Gold Copper Project Plant site stability Assessment, June 3 2011, BGC Engineering	Geotechnical stability of the process plant site with particular reference to the final open pit mine limits
Martyn Creaney, Project Director of Didipio Construction (internal communications)	Capital cost estimates relating to the processing plant and all surface infrastructure (excluding the mining works and TSF construction)
Bradley Norman, Senior Vice President - Philippines (internal communications)	Surface rights acquisition (section 4)
Mr Ramoncito P. Gozar, VP Communication and External Affairs – OGPI (internal communications)	Tenure, Nature and Extent of the Issuers Title, Environmental Liabilities and Work Permits (section 4 and section 20)
David Cook, Consultant (internal communications)	Environmental issues (section 20)

3.2 Jonathan Moore

None.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Didipio Project is located in the north of Luzon Island approximately 270km NNE of Manila, in the Philippines.

The Didipio Project is at 121.45º E 16.33º N (Longitude/Latitude – World Geodetic System 1984). The locations of the Financial or Technical Assistance Agreement (FTAA) and proposed mining areas are shown in Figure 4, along with the coordinates of the lease corners.

The FTAA and proposed mining areas straddle a provincial boundary, with part of these properties within Barangay Didipio, Municipality of Kasibu, Province of Nueva Vizcaya and part within Barangay Dingasan, Municipality of Cabarroguis, and Province of Quirino.

4.2 Area of Property

The FTAA now covers about 158km^2 (compared with the original 370km^2) located in the provinces of Nueva Vizcaya and Quirino. Parts of the original FTAA have been relinquished under the terms of the agreement that requires 10% relinquishment per annum (although some exceptions apply).

The proposed mining area comprises 12 blocks (each 0.5‟ latitude by 0.5‟ longitude, or approximately 81 hectares) or 9.75 km^2 within the FTAA. A direct impact zone of 3.25 km^2 is situated inside this 9.75km^2 area.

4.3 Tenure

The FTAA was granted on 20 June 1994 for a 25-year period, renewable for a further 25 years. The FTAA carries a minimum expenditure commitment of US$50 million and includes the fiscal regime for any development. The expenditure commitment will easily be met as the capital cost of building the Didipio mining project exceeds this amount.

The FTAA was originally granted to CAMC but was assigned to APMI in 2004 (which then changed its name to OGPI in 2007).

The Environmental Compliance Certificate (ECC) for the project was originally granted to CAMC in August 1999, with subsequent amendments in January 2000 (extension of area) and August 2004 (definition of direct impact zone).

The ECC allows for open pit and underground workings, tailings dam and impoundment, waste rock dumps, mill plant, explosive magazine, administration and housing facilities.

The ECC specifies the project mining methods, production rate, processing methods and other aspects of the mining operation. It also specifies the environmental management and protection requirements, including the submission of Annual Environmental Programme Enhancement Plans (AEPEPs) as well as a Social Development and Management Program.

In March 2005, APMI submitted a Partial Declaration of Mining Project Feasibility (PDMF) for approval by the Department of Environment and Natural Resources (DENR). In conjunction with the PDMF, APMI submitted (among other things) a Definitive Feasibility Study for the project as well as Development Work Program (DWP).

In October 2005, the PDMF was approved by the DENR which provided, in effect, the permit to operate and develop the project. The development period under the PDMF was subsequently extended for a further 3 years period from October 2009.

An application to amend the ECC to reflect operations as outlined in this report was lodged with EMB in June 2011. Securing the last permits and approvals required will not be possible until all design details have been finalised, allowing the various construction permits, and subsequent permits-to-operate, to be granted. Land acquisition is almost complete and applications for water rights have been made and are in process.

4.4 Nature and Extent of the Issuer's Title

OGC acquired its interest in the Didipio Project as a result of its merger with Climax Mining Limited. OGC‟s wholly owned subsidiary, Australasian Philippines Mining Inc (APMI), holds the FTAA that covers the Didipio Project area. Subsequently, APMI changed its name to OceanaGold (Philippines), Inc. (OGPI).

OGPI has an agreement (known as the "Addendum Agreement") (Jorge G Gonzales, Jerome P Delosa and David Gonzales) with a Philippine claim owner syndicate in respect to a substantial proportion of the FTAA, including the proposed mining area in its entirety (the "Addendum Property"). The claim owner syndicate has a contractual right, subject to satisfaction of certain conditions, to an 8% free carried interest in the operating vehicle that will be formed to undertake operations in respect of the Addendum Property (see Figure 4.2) denoted as the DMF/ECC area).

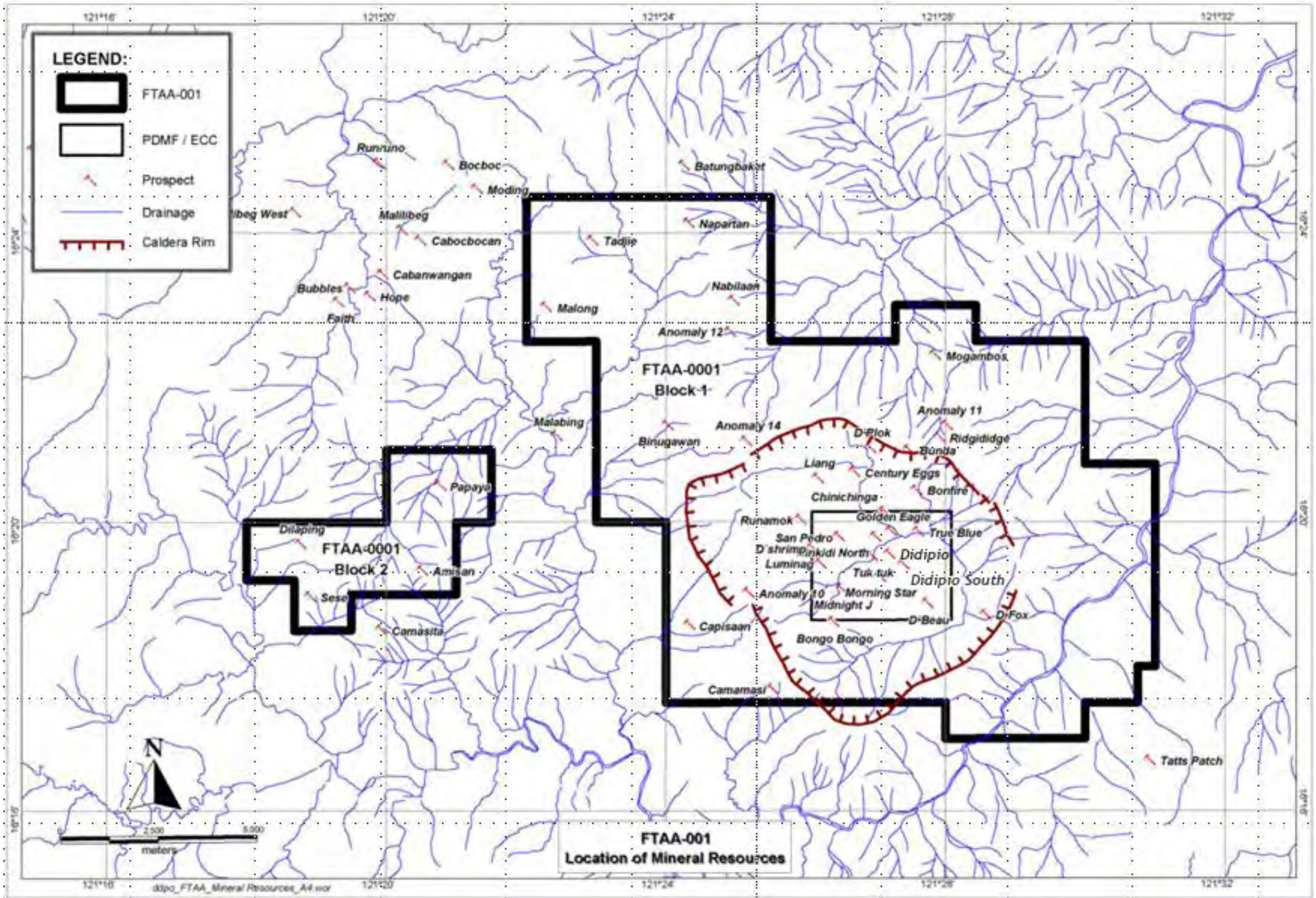
Upon the commencement of commercial production, there is a period of five years whereby the company can recover all pre-operating expenses (the Recovery Period). After this time, 60% of net profit (net of all taxes, local payments, government payments, etc) is payable to the government as a government share.

4.5 Property Boundaries

The boundary corners of the FTAA and ECC are defined in Figure 4.2. Two of the ECC boundary markers were checked in the field and consist of concrete pillars or markers annotated with their coordinates in latitude and longitude (see Figure 4.1).

Figure 4.1: NE Corner Marker for ECC

Figure 4.2: FTAA & ECC Boundaries



4.6 Surface Rights

The Company has acquired, through voluntary agreements, the surface rights to the vast majority of the land required for the Project. Purchase agreements are being negotiated with remaining private landowners. The company expects to have acquired all the land required by the current Project footprint in the near future.

4.7 Royalties

The claim owner syndicate (Gonzales) is entitled to a 2% net smelter return (NSR) royalty on production from the Addendum Property under the terms of the Addendum Agreement.

A 0.6% of 92% NSR royalty (capped at a total of A$13.5 million) is payable to Malaysian Mining Corporation.

The author is unaware of any further royalties, back-in rights, payments or other agreements and encumbrances that apply to the Didipio Project.

4.8 Environmental Liabilities

An amended Environmental Impact Statement (EIS) was completed by Gaia South Inc, environmental consultants, on behalf of CAMC in April 2004, which led to the issuance of the revised ECC on 8 August 2004.

The revised ECC sets out the work requirements relating to environmental management and protection requirements, which includes an Environmental Trust Fund, Environmental Risk Assessment and Mine Decommissioning Plan.

A submission to vary the existing ECC was lodged on 1 June 2011 with the EMB in Manila to address project changes made since the granting of the 2004 ECC. Currently this submission is under review

4.9 Work Permits

All the primary requirements to be fulfilled under the FTAA have been met and acquisition of the necessary environmental approvals and permits from the relevant government agencies is almost complete. Securing the last permits and approvals required will not be possible until all design details have been finalised, allowing the various construction permits, and subsequent permits-to-operate, to be granted. Land acquisition is almost complete and applications for water rights have been made and are in process.

4.10 Other Significant Factors

None.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, Elevation and Vegetation

The Didipio Project is located approximately 270km NNE of Manila in the southern part of the Mamparang mountain range adjacent to the border of Nueva Vizcaya and Quirino Provinces.

The project area is located within the southern part of the Cagayan Valley basin in north-eastern Luzon, the Philippines. The area is bounded on the east by the Sierra Madre Range, on the west by the Luzon Central Cordillera range and on the south by the Caraballo Mountains. The regional geology comprises late Miocene volcanics, volcaniclastics, intrusives and sedimentary rocks overlying a basement complex of pre-Tertiary tonalites and schists. This geology is indicative of an island arc depositional and tectonic setting.

The geomorphology of the project area is diverse. The project can be generally subdivided into at least six geomorphic units: ridges-and-spurs, escarpment zones, hills-and-slopes, valley-and-gully sides, infilled valley bottom and mass movement zones. Infilled valley bottoms occur as narrow strips of low and flat-lying areas within the project area. These areas occupy the main Didipio Valley. Morphological associations include the floodplain and terraces along the Didipio River.

The valley floor near the project centre is at 690-700 metres above sea level with the surrounding ridge-lines rising another 150-200m above this.

In the project area, three segments of vegetative cover were identified and consist of:

- Grassland, which covers both primary and secondary impact areas;
- Brushland (riparian), which is located within the primary impact site; and
- Low-density forest, which is located within the secondary impact area.

The project entails the development of a 9.75km^2 mine area located mainly at Barangay Didipio, Kasibu, and Nueva Vizcaya, although part of the project area is located in Cabarroguis, Quirino.

5.2 Access to Property

The main route access to Didipio is from the north, commencing from the national Maharlika Highway at Cordon, with a concrete paved road to Cabarroguis, and thereafter a gravel all-weather road to site. OGPI has an office, core storage and sample preparation facilities in the town of Cordon.

5.2.1 Road Access

Road access to the site is as follows:

From Manila:

- Leave Manila and travel north along the Maharlika Highway via San Jose, Bayombong and Cordon.
- Travel south from Cordon to Cabarroguis.
- Travel to the mine site from Cabarroguis.

The roads from Manila and Port Poro to Cabarroguis are paved. The road from Debibi (south of Cabarroguis) to site is an existing dirt road and will be upgraded to satisfy the planned operation. This upgrade work will include some approach works at the bridge crossing at Debibi, widening and realignment of the existing road from Tucod to Logpond and grading and widening of the road from Debibi to site. The costs of the road upgrade have been included in the capital estimate

An alternative access to site, suitable for vehicle sizes up to small truck, will also be available. The route for this access is from the Didipio site to Wangal, then to intersect the Maharlika Highway at Run Runo. The road to Wangal portion needs to be realigned through the TSF including two waterway crossings. The rest is utilised by local traffic and does not require any significant capital works provided that use of this access route is limited to the existing light vehicles.

5.2.2 Air Access

A helipad has been constructed at site within the secured area of the mine footprint. Helicopter transport will be used to transport doré to Manila for export to the refiner‟s works. In addition, helicopter transport may be called upon from time to time for medical evacuations or visits to site.

5.3 Proximity to Population Centres

The Didipio Project lies approximately 35km to the ESE of the provincial capital of Bayombong within the Province of Nueva Vizcaya near the heart of Northern Luzon (see Figure 5.1).

Figure 5.1: Location Map

The province consists of one congressional district, with subdivision into two sectors, the North and South sectors. The North Sector comprises seven municipalities and the South Sector has eight municipalities.

North Sector:	Municipalities of: Ambaguio, Bagabag, Bayombong, Diadi, Quezon, Solano, and Villaverde.
South Sector:	Municipalities of: Alfonso Castañeda, Aritao, Bambang, Dupax del Norte, Dupax del Sur, Kasibu, Kayapa and Santa Fe.

Nueva Vizcaya Province is composed of 15 municipalities with the city of Bayombong as the provincial capital (population approximately 375,000). The towns of Solano and Kayapa are the commercial centre and summer capital, respectively.

The nearest significant town to the Didipio Project is Cabarroguis, located approximately 20km to the north and connected by paved road to Bayombong to the west.

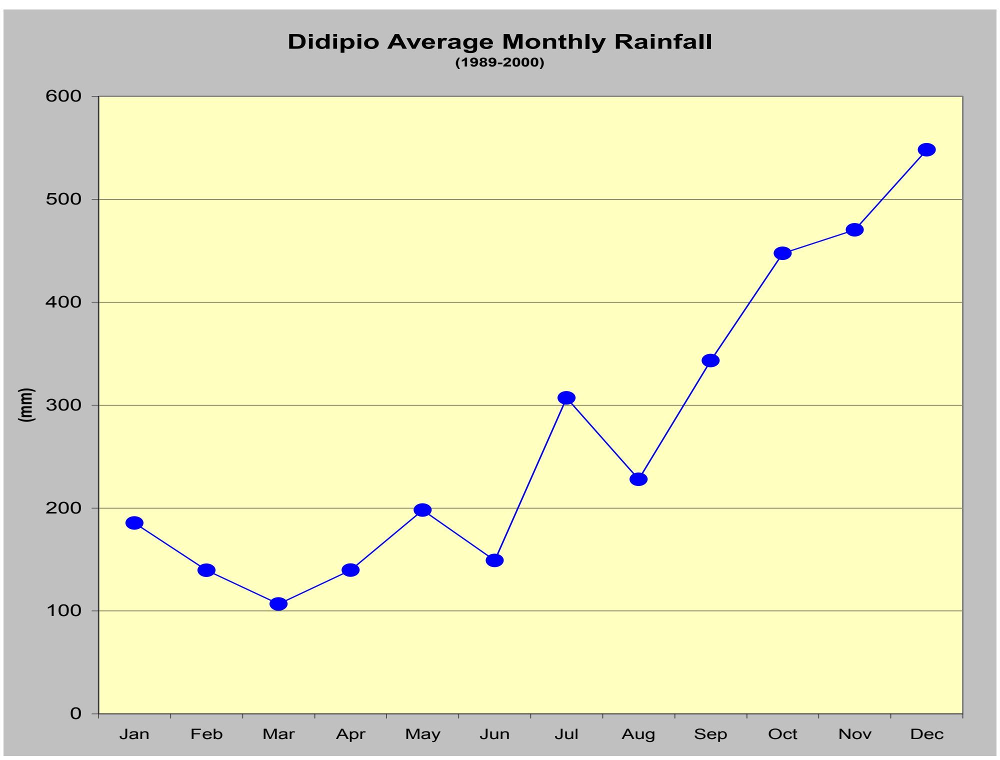
5.4 Climate and Operating Season

Didipio is located on the eastern side of Luzon, which experiences a tropical climate consisting of three main seasons:

- the south-west monsoonal season between June and September;
- the north-west monsoon season between October and January; and
- a transition period between February and May.

Didipio receives most of its annual rainfall during these monsoon seasons (see Figure 5.2), although the mine will operate year round..

Figure 5.2: Average Monthly Rainfall for Didipio



At the project site, rainfall has been monitored daily since May 1989. The mean annual rainfall calculated on site was 2929mm. September and November appear to be the most consistently wet months. The driest month is normally March. The mean annual number of rainfall days at the project site is 181. November and December have been observed to have the highest number of wet days. The least number of rainfall days are generally in March. The mean annual temperature at the project site is 22.8ºC. The hottest months are May and July and the coldest month is January.

Luzon Island's setting combined with its high rainfall, results in high humidity levels. The average annual humidity is 82% and nearly all regional weather stations report a relative humidity in excess of 70% on a monthly basis. A large majority of these stations report a relative humidity of greater than 80% for more than eight months of the year. The prevailing winds tend to conform to the dominant seasonal air streams. Consequently, north-east winds are associated with the north-east monsoon season. Local topography and diurnal effects do, however, influence this general trend to some extent.

The average annual wind speed is 3m/s, although the region is subject to the effects of an average of two tropical cyclones a year, which, together with topographical effects, can greatly influence wind speeds. In such instances, wind speeds can exceed 50m/s and may reach as much as 75m/s. The average wind speed over such surge periods normally exceeds 15m/s.

5.5 Rainfall Modelling

Rainfall events have been modelled by Knight Piesold based upon analysis of all historic records. The short duration storm event results are tabulated in Figure 5.3 and form the basis for design parameters for the project.

Figure 5.3: Rainfall Modelling

Knight Piésold CONSULTING

DIDIPIO GOLD PROJECT
Rainfall Events

Project: PE701-00029
Date: 25/06/2007

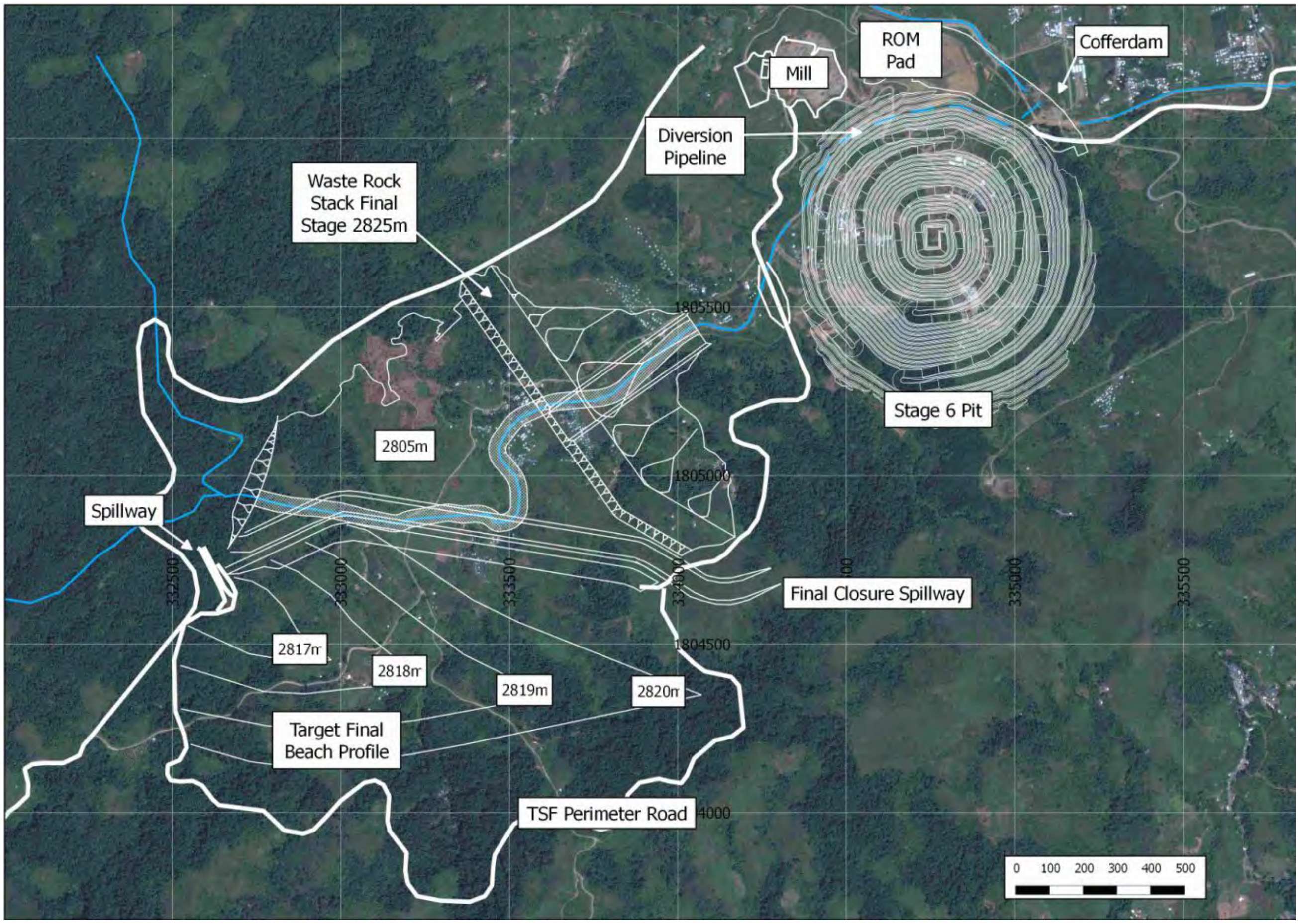
P:\PE701-00029 Didipio\Site_Data\Climate\Rainfall Data\2007 Rainfall Analysis\[701-29am_ShortDurationStorms_070625.xls]Sheet1

PRECIPITATION IN DURATION (mm) - 95% Confidence Limit			Minutes								
			10	15	30						
			Hours								
			0.17	0.25	0.50	1	2	3	6	12	24
	RR	1 in x years	0.033	0.048	0.087	0.150	0.243	0.315	0.474	0.694	1.000
AEP %	99	1 in 1	3.7	5.3	9.6	16.5	26.8	34.7	52.3	76.5	110.2
	95	-	4.0	5.8	10.5	18.0	29.2	37.8	56.9	83.3	120.0
	90	-	4.3	6.3	11.4	19.5	31.7	41.0	61.8	90.4	130.2
	80	-	4.9	7.1	12.9	22.2	36.1	46.7	70.3	102.9	148.2
	50	1 in 2	6.7	9.7	17.5	30.1	48.9	63.3	95.3	139.4	200.8
	20	1 in 5	9.5	13.7	24.8	42.7	69.3	89.7	135.0	197.7	284.6
	10	1 in 10	11.8	17.1	31.0	53.3	86.4	111.9	168.4	246.5	355.0
	5	1 in 20	14.6	21.1	38.3	65.9	106.9	138.4	208.3	304.9	439.1
	2	1 in 50	19.2	27.8	50.4	86.7	140.6	182.0	273.9	400.9	577.4
	1	1 in 100	23.5	34.0	61.7	106.1	172.1	222.9	335.5	491.0	707.1
	0.5	1 in 200	28.7	41.5	75.3	129.5	210.1	272.0	409.4	599.2	863.0
	0.2	1 in 500	37.2	53.7	97.6	167.8	272.1	352.3	530.4	776.3	1117.9
	PMP	PMP	37.3	53.9	97.9	168.4	273.1	353.5	532.2	779.0	1121.8

5.6 Infrastructure

A general site plan is shown below.

Figure 5.4: General Site Plan



5.6.1 Water

Water will be sourced from drawdown bores sunk around the perimeter of the open pit. A potable water treatment plant will be located in the process plant with a second potable plant for the camp. The accommodation areas as well as other potable needs.

5.6.2 Power Supply

Most of the power will be self-generated on site by an OGPI owned power station. Costing is based on using a high-speed, diesel-fired power station comprising package individual sets in acoustic enclosures with on-board service and protective provisions and controls, all connected to a separate central control and electrical room.

A small supply overhead line does run through the valley, but the project will not be relying on using any power from this source.

5.6.3 Sewage

Sewage from the project site will be piped to a site based sewage treatment plant, sewage from small isolated locations such as the guard house will be held in holding tanks and then transferred to the sewage treatment plant.

5.6.4 Refuse Disposal

Refuse disposal facilities will comply with the commitments of the ECC. It is anticipated that scrap metal and other refuse waste will be either recovered or disposed of at a local waste recovery facility or will be buried in a suitable location on site. Waste oils and lubricants will be recovered and disposed of at a registered waste treatment or disposal facility in accordance with Philippines Government requirements.

5.6.5 Accommodation

Single-status accommodation will be made available in a central camp for all personnel recruited from outside the region. The accommodation will consist of varying standards of sleeping quarters, with their allocation based upon the role of the person.

The styles of permanent operational accommodation and the numbers of buildings are as follows:

- Senior management/VIP accommodation – 1x self contained one-bedroom apartment.
- Management accommodation – 48 single bedrooms with ensuites.
- Senior staff accommodation – 132 bedrooms with shared ensuite.
- Staff accommodation – barracks-style accommodation with shared ablutions block for 306 people.
-

Other buildings/facilities within the accommodation camp are as follows:

- Kitchen and dry mess suitable for 180 persons.
- Accommodation camp gatehouse.
- Accommodation camp laundry and line storage.
- Recreation room.
- Camp office.
- Sewage treatment plant.
- Emergency generators.

The camp will be operated by a contractor, whose role will include providing meals, cleaning duties for the camp buildings, cleaning duties for the mine site buildings, laundry services, provision of linen, cutlery etc. The site operating costs include the accommodation camp operating costs.

5.6.6 Bulk Waste

The tailings storage facility will be constructed about 1km to the west of the open pit on a tributary creek to the Dinuyan River. Due to the relatively large vertical relief around the valley, tailings dam sites were limited however options were assessed. The site chosen has a relatively small rainfall recharge and allows the main river to flow by the facility.

Bulk waste shall be placed in the valley between the TSF and the mine, and in effect buttressing the downstream embankment of the TSF

5.6.7 Port Facilities

The existing copper concentrate storage and shipment facility at Poro Point is anticipated to be available for concentrate shipments from the project. This entails a 365 kilometre truck haul over an existing well maintained sealed pavement national highway. Port Irene on Casambalangan Bay, is an alternative at the north-eastern tip of Luzon Island, approximately 320km from Didipio. .

5.6.8 Personnel

Manning profiles for the project have been derived from the following sources:

- Assessment of labour requirements from first principles.
- Contractor"s assessment of labour requirements.
- Benchmarking from similar operations.
- Previous feasibility study information.

It is anticipated that there will be approximately 15 to 16 expatriates employed on the site once steady-state operations have been achieved. Therefore, the site satisfies the requirements for Filipinisation under the FTAA.

Where possible, recruitment, particularly of mining and processing plant personnel, will be from the local area. Contractors servicing the project will be obliged to follow a similar employment policy

The FTAA sets out targets for Filipinisation, which requires up to 100% Filipinos in unskilled, skilled and clerical position and up to 60% Filipinos in professional and management positions

5.6.9 Sufficiency of Surface Rights

The company has acquired, through voluntary agreements, the surface rights to the vast majority of the land required for the project. Purchase agreements are being negotiated with remaining private landowners. The company hopes to have acquired all the land required by the current project footprint in the near future.

The Company controls the vast majority of land in the mining footprint to support the Didipio Project and is confident of acquiring a small number of outstanding parcels in the near term, as all parties are in negotiations. The impact on the project will be either increased capital cost which should be relatively minor or a delay to some areas in initial construction while negotiations are complete.

6 HISTORY

6.1 Prior Ownership

Since the discovery of alluvial gold in the 1970s, the Didipio area has been held by a succession of claim holders.

In May 1975, Victoria Consolidated Resources Corporation (VCRC) and Fil-Am Resources Inc entered into an exploration agreement with a syndicate of claim owners who had title to an area covering the Didipio Valley and undertook exploration activities between 1975 and 1977.

Marcopper Mining Corporation investigated the region in 1984 and Benguet Corporation examined the Didipio area in September 1985, but neither of these companies held title to the project.

The area was investigated in April 1985 by a consultant geologist (E P Deloso) engaged by local claim owner Jorge Gonzales.

Geophilippines Inc investigated the Didipio area in September 1987 and made mining lease applications in November 1987.

In 1989, Cyprus Philippines Corporation (Cyprus) and subsequently Arimco NL (as Arimco Mining Corporation (AMC) in the Philippines) entered into an agreement with Geophilippines Inc and the local claim owner, Jorge Gonzales, to explore the Didipio area. Subsequently, it was decided to allow Climax to take over control of AMC (Climax-Arimco Mining Corporation (CAMC)) and the entire Cyprus-Arimco NL interest in the project.

In 1996, an application was made to transfer ownership of the Didipio project from CAMC to APMI, which was finally approved in 2004.

In 2006, Oceana Gold Limited (OGL) merged with Climax Mining Limited. OGC now operates and manages the Didipio project through its wholly owned subsidiary OGPI.

6.2 Previous Work

The Didipio area was first recognised as a gold province in the 1970s, when indigenous miners from Ifugao Province discovered alluvial gold deposits in the region. Gold was mined either by the excavation of tunnels following high-grade quartz-sulphide veins associated with altered dioritic intrusive rocks, or by hydraulicing in softer, clay-altered zones. Gold was also recovered by panning and sluicing gravel deposits in nearby rivers, and small-scale alluvial mining still takes place. No indications of the amount of gold recovered have been recorded.

Since 1975, exploration work carried out in the area has been managed by the following:

a) From 1975 to 1977, Victoria Consolidated Resources Corporation (VCRC) and Fil-Am Resources Inc undertook a stream geochemistry programme, collecting 1204 panned concentrates samples that were assayed for gold, copper, lead and zinc. A large area of hydrothermal alteration was mapped, but, although nine drill holes were planned to test it, no drilling eventuated. Despite recognition of an altered diorite intrusive (the Didipio Gold-Copper Deposit), no further work was undertaken.

b) Marcopper Mining Corporation investigated the region in 1984, followed in April 1985 by a consultant geologist (E P Deloso) engaged by local claim owner Jorge Gonzales. Work by Deloso included geological mapping, panning of stream-bed sediments and ridge and spur soil sampling. Deloso described the Didipio Gold-Copper Deposit as a protruding ridge of diorite with mineralised quartz veinlets within a vertically dipping breccia pipe containing a potential resource. The resource is not compliant with CIM guidelines and is therefore not quoted.

c) Benguet Corporation examined the Didipio area in September 1985 and evaluated the bulk gold potential of the diorite intrusion. Work included grab and channel sampling of mineralised outcrops, with sample gold grades ranging up to 12 g/t Au and copper averaging 0.14% Cu.

It was concluded that the economic potential of the diorite intrusion depended on the intensity of quartz veining and the presence of a clay-quartz-pyrite stockwork at depth.

Geophilippines Inc investigated the Didipio area in September 1987 and carried out mapping, gridding, rockchip and channel sampling over the diorite ridge. In November 1987, Geophilippines Inc commissioned the DENR, Region I, to undertake a geological investigation of the region in conjunction with mining lease applications.

d) Between April 1989 and December 1991 Cyprus and then AMC carried out an exploration programme that included the drilling of 16 diamond core holes into the Didipio Ridge deposit. Although this work outlined potential for a significant deposit, both companies assessed as low the probability of obtaining secure title to the area. Consequently, it was decided to allow Climax to take over control of AMC (now Climax-Arimco Mining Corporation (CAMC)) and the entire Cyprus-Arimco NL interest in the project.

From 1992, Climax exploration work concentrated on the Didipio Gold-Copper Deposit, although concurrent regional reconnaissance, geological, geophysical and geochemical programmes delineated other gold and copper anomalies in favourable geological settings within the Didipio area.

Diamond drilling and other detailed geological investigations continued on the Didipio Project and elsewhere in the Didipio area through 1993, and were coupled with a preliminary Environmental Impact Study (EIS) and geotechnical and water management investigations.

Up to the decision to commence the Project Development Study (PDS) in January 1994, 21 diamond drill holes had been drilled by Climax for a total of 7480m, forming the basis for a preliminary resource estimate (not quoted as it is not compliant with CIM guidelines).

Additional diamond drilling was completed at Didipio Project as part of the GRD 1995 PDS, providing a database of 59 drill holes within the deposit. A model of the deposit was developed and a resource estimate made (not quoted as it is not compliant with CIM guidelines). The work identified the key parameters for potential project development, which included the likelihood of underground block caving for ore extraction. The economics of this scenario were dependent in part on the delineation of a central core of higher-grade gold and copper mineralisation, but drill intersections in this area were too widely spaced to confirm geological and grade continuity at the measured resource category.

A programme of 17 additional diamond drill holes was designed to provide closer spaced sampling data primarily within an area lying above the 2400mRL. This programme was completed in June 1997, with all drill core assays received by early August 1997. These data have been utilised for the GRD 1998 Definitive Feasibility Study (DFS).

6.3 Historical Estimates

Several resource estimates have been made since 1985. The chronology of these is presented below. None of the resource estimates are quoted as they do not adhere to the CIM guidelines.

- Work by Deloso in April 1985 suggested a potential resource.
- In September 1985, Benguet Corporation estimated the total resource potential.
- In December 1993, Climax produced an estimate based on available data including the first 21 diamond drill holes; interpolation method was inverse distance squared into 25 x 25 x 25m blocks.
- Snowden Associates (Snowden) produced a resource estimate in 1995 using additional drill holes (up to DDDH65). This model effectively used a 3g/t eqAu interpretation and wire-framing of the high grade core of mineralisation. Interpolation was by indicator kriging into 15x15x15m blocks and classification was based on search radii and number of samples.
- The Minproc DFS estimate used all 79 holes (up to hole DDDH83) plus the data for nine surface trenches. The stockwork and high grade core were modelled separately and grades were interpolated using ordinary or indicator kriging (with grade top cutting) into 15 x 15 x 15m blocks.

6.4 Previous Production

There has been no large-scale mining at Didipio to date and there are no records of the production by artisan miners.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The project area is situated within the southern part of the meridional Cagayan Valley basin in north-eastern Luzon and is bounded on the east by the Sierra Madre Range, on the west by the Luzon Central Cordillera range and to the south by the Caraballo Mountains (see Figure 7.1).

The regional geology comprises late Miocene volcanic, volcaniclastic, intrusive and sedimentary rocks overlying a basement complex of pre-Tertiary age tonalite and schist (Figure 7.2)), which have been interpreted to represent an island arc depositional and tectonic setting.

The basal sequence of the Caraballo Group is of Cretaceous to Eocene age and comprises andesitic pyroclastics, andesitic lavas and basaltic tuffs with inter-layered beds of sandstone, shale and tuff. The Caraballo Group includes the Alimit Volcanics and is intruded by tonalites, diorites, quartz diorites and gabbros of the Coastal Batholith (27 to 49 Ma) and the Dupax Batholith (26 to 33 Ma).

The Caraballo Group is unconformably overlain by the Mamparang Formation of the Oligocene age, comprising andesitic and basaltic lavas and volcaniclastic rocks („Dark Diorite‟). This was intruded by various alkalic plutonic rocks including syenite, monzonite and a variety of K-feldspar-rich igneous rocks that comprise the Palali Batholith (17 to 25 Ma). This batholith includes intrusive rocks found in the Didipio area (Didipio Igneous Complex).

Unconformably overlying the Caraballo Group and Mamparang Formation, the Palali Formation comprises basaltic and andesitic lavas, mudstones, sandstones and dacitic pyroclastics of early to middle Miocene age.

Regionally, the volcanics and sediments are folded about meridional anticlinal and synclinal axes and are cut by prominent, steeply dipping, north-west and north-trending faults sub-parallel to the major Philippine Fault zone (Figure 7.1). A set of later, steeply north dipping, east-north-east-trending faults are associated with the batholitic intrusions.

Recent geological mapping in the Didipio region has been interpreted to indicate the Didipio Gold-Copper Deposit is hosted within the multiphase Didipio Stock, which is in turn part of a larger alkalic intrusive body, the Didipio Igneous Complex. The Didipio Igneous Complex consists of:

1. An early composite clinopyroxene-gabbro-diorite-monzodiorite pluton that comprises medium-grained, clinopyroxene-biotite rich microdiorites and monzodiorites of the dark diorite (pre-mineralisation);
2. The Surong clinopyroxene to biotite monzonite pluton. Breccia textures on the margins of the Surong pluton are interpreted to indicate that the Surong monzonite intruded into the Dark Diorite. The Didipio area lies within a circular physiographic feature, approximately six to eight kilometres in diameter. The Pimadek Porphyry (latite porphyry) occupies the topographic highs of the Didipio circular feature and is characterised by coarse K-feldspar phenocrysts (<20mm to 30mm) in a pale grey-green feldspathic groundmass. Pyroclastic deposits (ignimbrites, autobreccias) recognised in the area suggest that the Pimadek Porphyry could represent both the feeder dyke and extrusive product of an intra-caldera ignimbrite;
3. The Cu-Au mineralised Didipio Stock; and
4. Post-mineralisation andesite dykes.

Figure 7.1: Northern Luzon – Major Geological Subdivisions and Structural Elements

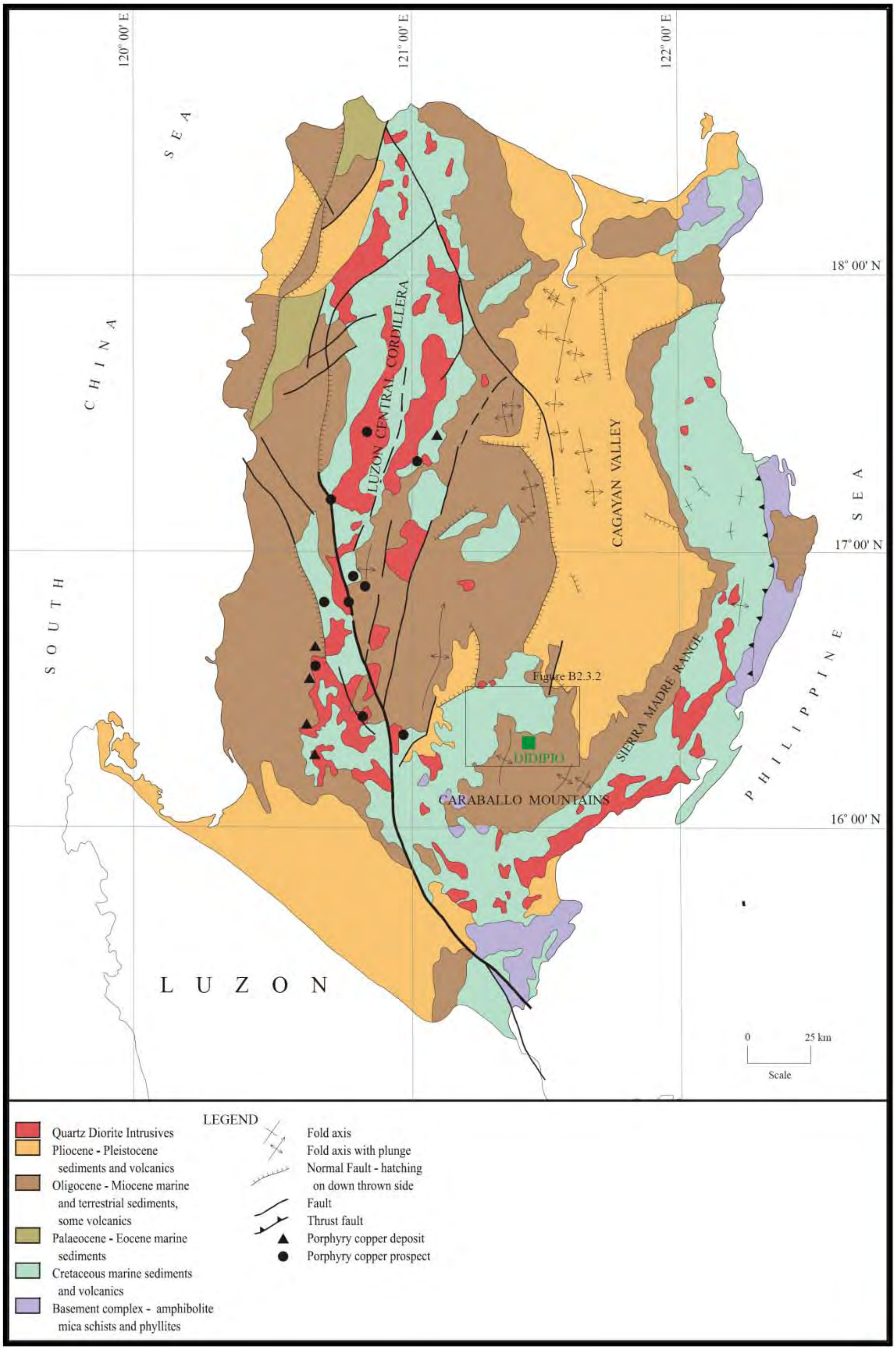
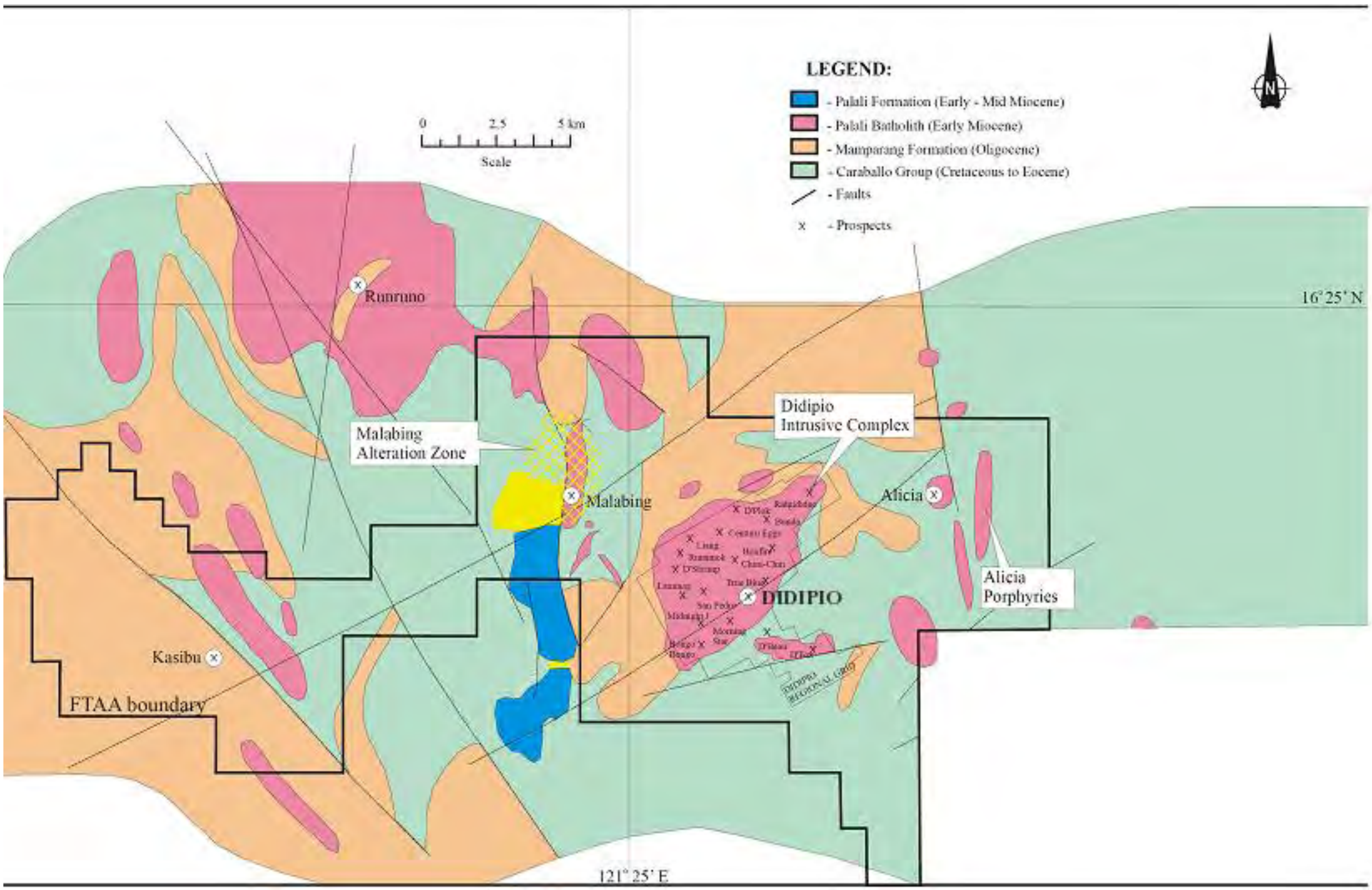


Figure 7.2: Regional Geology



7.2 Local Geology

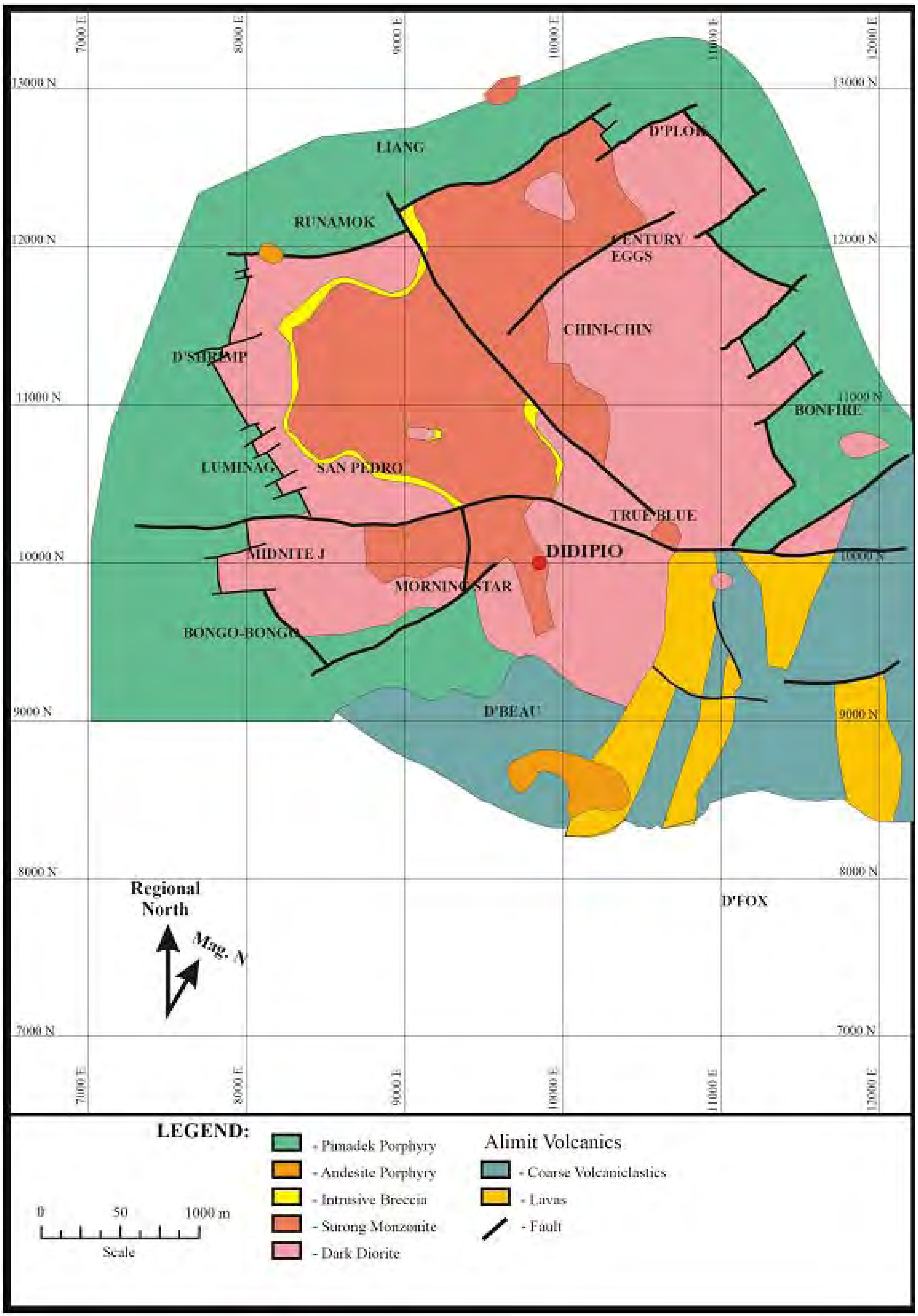
The Didipio Project has been identified an alkalic gold-copper porphyry system, roughly elliptical in shape at surface (450m long by 150m wide) and with a vertical pipe-like geometry that extends to at least 800m to 1000m below the surface.

The local geology comprises north-north-west-trending, steeply (80° to 85°) east-dipping composite microdiorite intrusive, in contact with volcaniclastics of the Mamparang Formation (see Figure 7.3). The microdiorite lies in a circular topographic depression that is coincident with a circular IP anomaly.

The area is cross-cut by a north-north-west-trending regional magnetic lineament, which is possibly a geophysical expression of major strike-slip faulting. North to north-west trending strike-slip faults in the Luzon Cordillera area have been recognised as major controls on the emplacement and elongation of porphyry deposits (Sillitoe and Gappe, 1984) and a similar structural control may have been important in the Didipio area.

Porphyry-style mineralisation is closely associated with a zone of K-feldspar alteration within a small composite porphyritic monzonite stock intruded into the main body of diorite (Dark Diorite). The extent of alteration is marked by a prominent topographic feature – the Didipio Ridge – some 400m long and rising steeply to about 100m above an area of river flats and undulating ground.

Figure 7.3: Local Geology

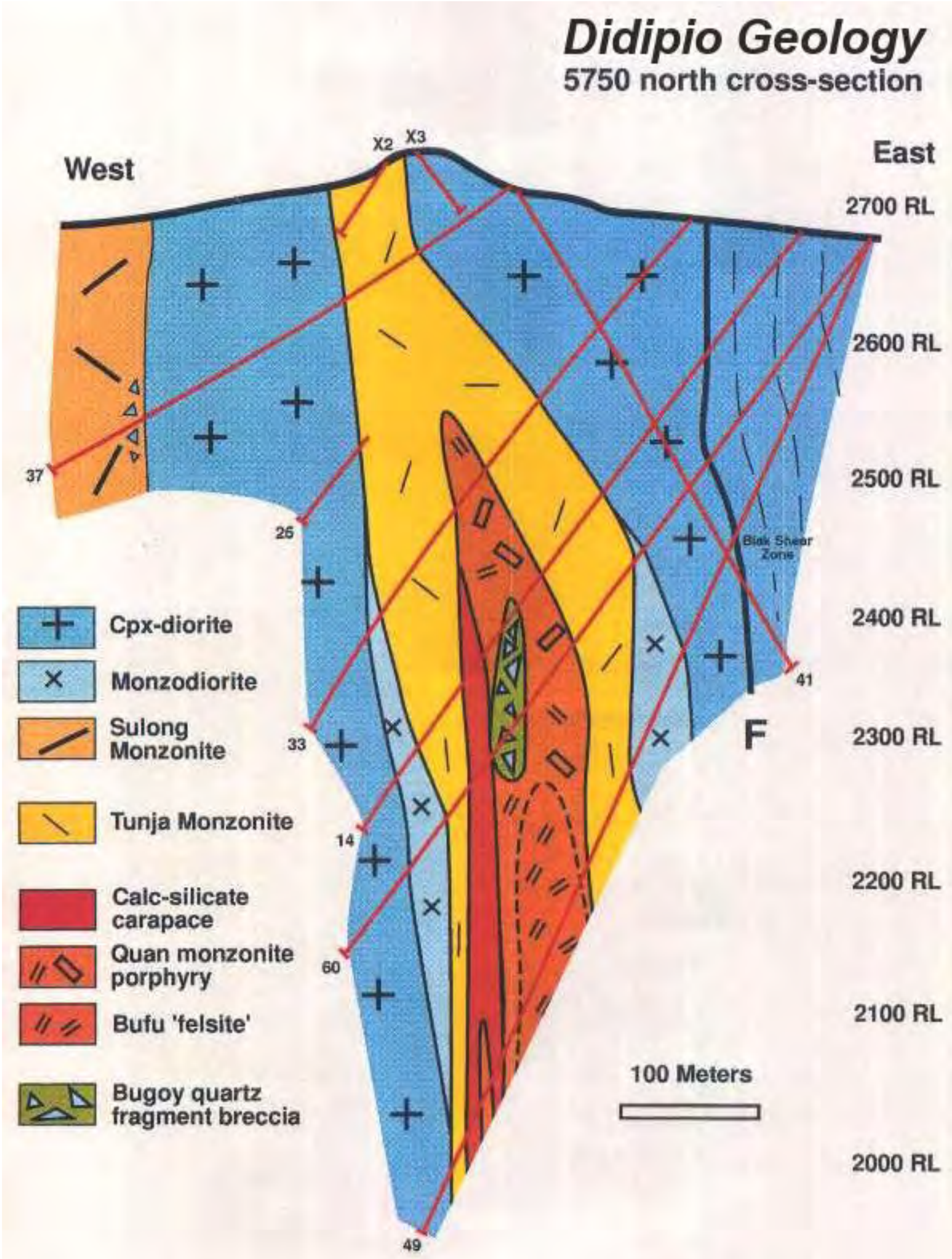


7.3 Deposit Geology

7.3.1 Lithology

The Didipio Gold-Copper Deposit is hosted by a series of hydrothermally altered and structurally controlled Miocene intrusives, which were emplaced along the regional Tatts Fault structure. Mineralisation is predominantly hosted by the Tunja monzonite, which intrudes the Dark Diorite. Minor mineralisation occurs in the surrounding Dark Diorite units, particularly in the upper part of the deposit where it is overlies the Tunja. The core of the Tunja is intruded by the Quan monzonite porphyry, which is spatially related to the higher-grade mineralised zones. The relationship of the Quan and a deeper intrusive, termed the Bufu, is uncertain, as Quan/Bufu contacts are both graduated and faulted in places. However, the two intrusives are probably related. The Bufu is a very distinctive vuggy equigranular to crystal-crowded felsite. The Bugoy breccia, a high-grade quartz-sulphide breccia, is developed immediately above the Bufu. The northern end of the deposit is truncated by a post-mineralisation fault zone, the Biak Shear (see Figure 7.4).

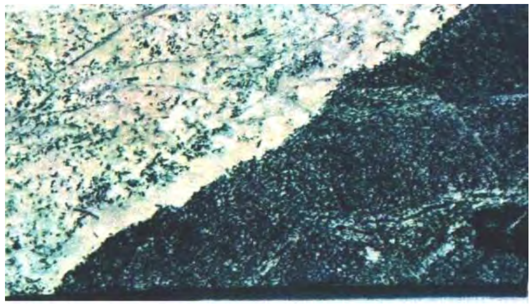
Figure 7.4: Didipio Project Geology



7.3.1.1 Dark Diorite

The Dark Diorite is a grey-black medium-grain equigranular to weakly plagioclase and clinopyroxene-phyric clinopyroxene-diorite (see Figure 7.5).

Figure 7.5: Sharp Intrusive Contact Tunja (left) and Dark Diorite



7.3.1.2 Tunja Monzonite

The Tunja stock intrudes the Dark Diorite. Monzonite dykes penetrate into the surrounding Dark Diorite for over 100 metres. The Tunja stock comprises a white to pale pink-grey medium to coarse-grained holocrystalline plagioclase-phyric biotite-monzonite. Euhedral plagioclase prisms are set in a very coarse granular mosaic of primary anhedral K-feldspar. Interstitial sites that once held primary biotite account for approximately 10% of the volume (Wolfe, 1996).

7.3.1.3 Quan

The Quan is a porphyritic monzosyenite that intrudes the Tunja. In the upper parts of the deposit sharp intrusive contacts are evident, but at depth the contacts are masked by intense alteration. Rare Tunja xenoliths also occur within the Quan.

7.3.1.4 Bufu

At depth the Quan grades into a distinctive bleached-white vuggy equigranular to crystal-crowded leucocratic quartz-syenite.

7.3.1.5 Bugoy Breccia

The Tunja and Quan immediately above the Bufu have been pervasively silica-sericite altered and brecciated, forming the distinctive Bugoy quartz-fragment breccia. This breccia comprises angular to sub-rounded (milled) quartz fragments in a very fine grain sericite-clay matrix.

7.3.1.6 Biak Shear Zone

The Biak Shear Zone truncates the northern end of the deposit. Intrusives within the shear zone are extensively carbonate veined and sheared. The Biak Shear is a major aquifer.

7.3.2 Hydrothermal Events and Alteration

Within the Didipio Project deposit, alteration defines the broad limits of mineralisation. Alteration textures, intensity and mineralogy vary and it is suggested that the different textures observed in the igneous rocks – Bufu, Quan and Tunja diorites – reflect a continuum of intrusive and alteration events. Alteration appears to have been focused along lithological contacts, particularly between the Quan, Bufu and Tunja porphyries, where it has overprinted the intrusive contacts and caused destructive modification of original rock textures in many parts of the deposit.

The outer limits between unaltered and altered rocks are relatively abrupt and characterised by the introduction of carbonate and alteration of magnetite. Eight alteration zones are recognised, comprising two fundamental, overlapping alteration types, namely pervasive and vein alteration. The pervasive alteration types are listed in Table 7.1 and exhibit a generally concentric distribution (see Figure 7.7) from the inner or core zone to the outer limits of alteration. The dominant trend appears to be for a decrease outward in K-feldspar alteration relative to sericite-carbonate-clay, but the transitions are gradual and subjective and there can be repetitions of either type at several points down a single drill hole. Vein alteration types are listed in Table 7.2. These display several periods of emplacement and also overlap as seen in Figure 7.6.

Table 7.1: Pervasive Alteration Types

Zone	Alteration Mineralogy	Occurs Within Unit
Leached	Carbonate-K-feldspar-muscovite±sericite-silica	Bufu diorite
K-feldspar-SCC	K-feldspar±sericite-carbonate-clay	Quan diorite
SCC-K-feldspar	sericite-carbonate-clay-K-feldspar	Tunja diorite
SCC-K-feldspar-biotite	sericite-carbonate-clay-K-feldspar-biotite	Tunja diorite
Mixed	sericite-carbonate±silica-K-feldspar	Quan/Tunja diorite
Skarn	calc-silicate(diopside-hedenbergite)-magnetite-K-feldspar	Tunja diorite

Table 7.2: Vein Alteration Types

Zone	Alteration Mineralogy
QFS	Quartz-feldspar-carbonate-chalcopyrite-pyrite±magnetite veins
CSS	Calc-silicate (actinolite-tremolite?)-feldspar-sulphide veins

Figure 7.6: Stylised Section Didipio Project-Style Porphyry Gold-Copper Alteration-Mineralisation

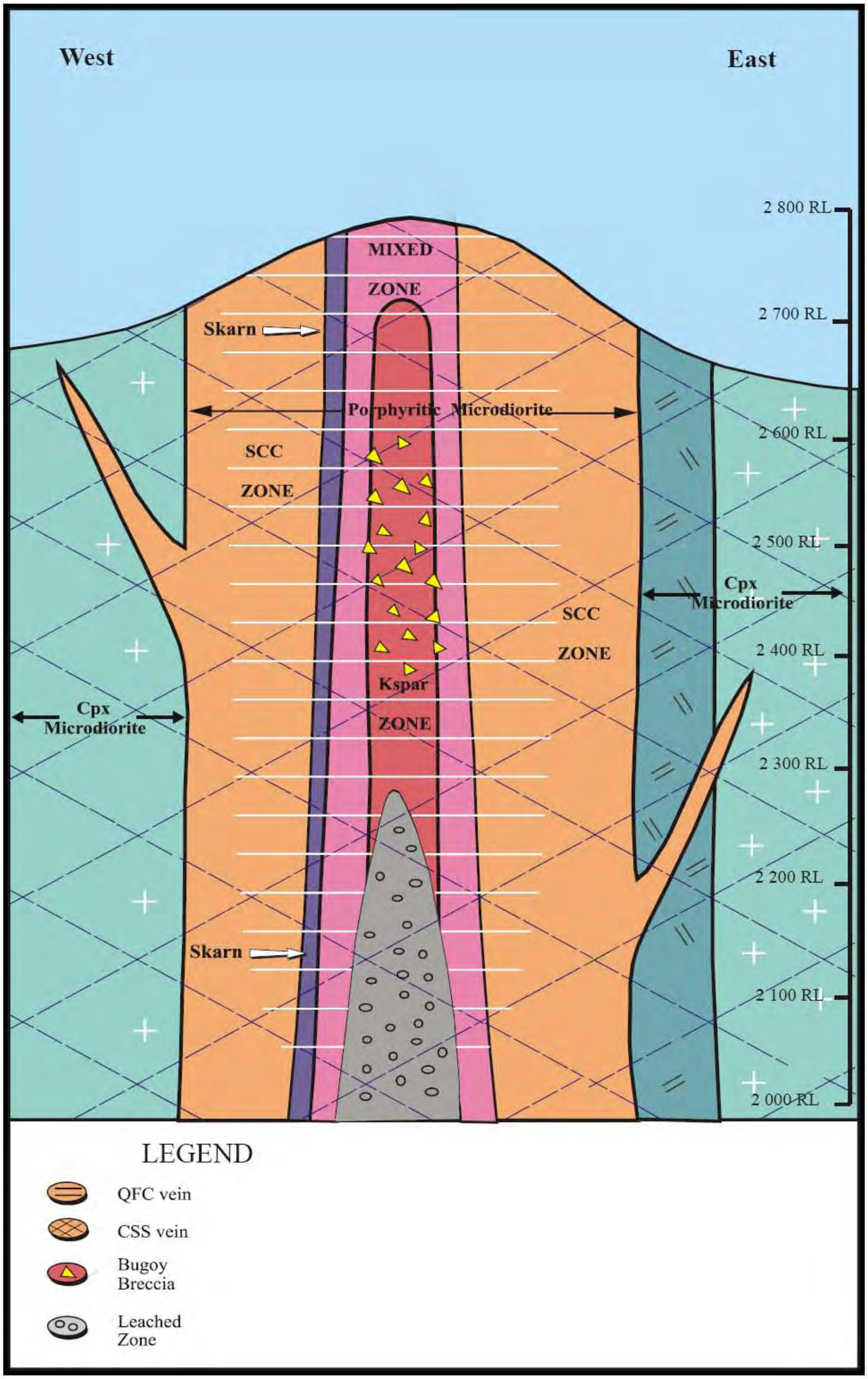
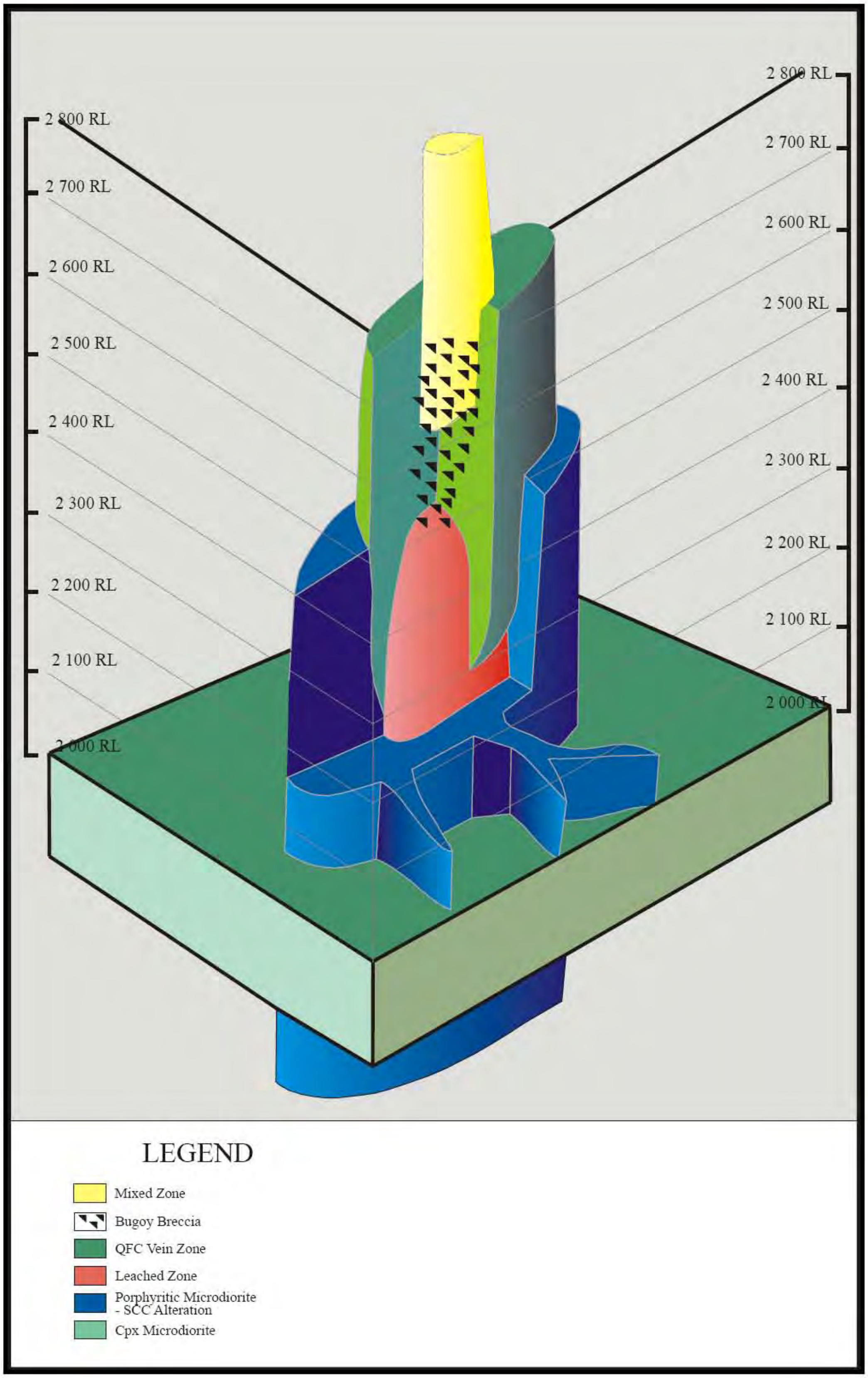


Figure 7.7: Illustration of 3D Relationships between Rock Types and Alteration – Didipio Project Deposit



7.3.3 Faulting

The main fault structures within the Didipio Project and the surrounding area have been grouped into three types based on orientation and style, with all directions referring to the drilling grid:

- Three steeply dipping faults and fault sets, trending northerly, north-westerly and east-north-easterly;
- A late, shallow west-dipping fault set with an undefined strike; and
- Two steeply dipping quartz-sulphide vein sets, trending north-south and east-west.

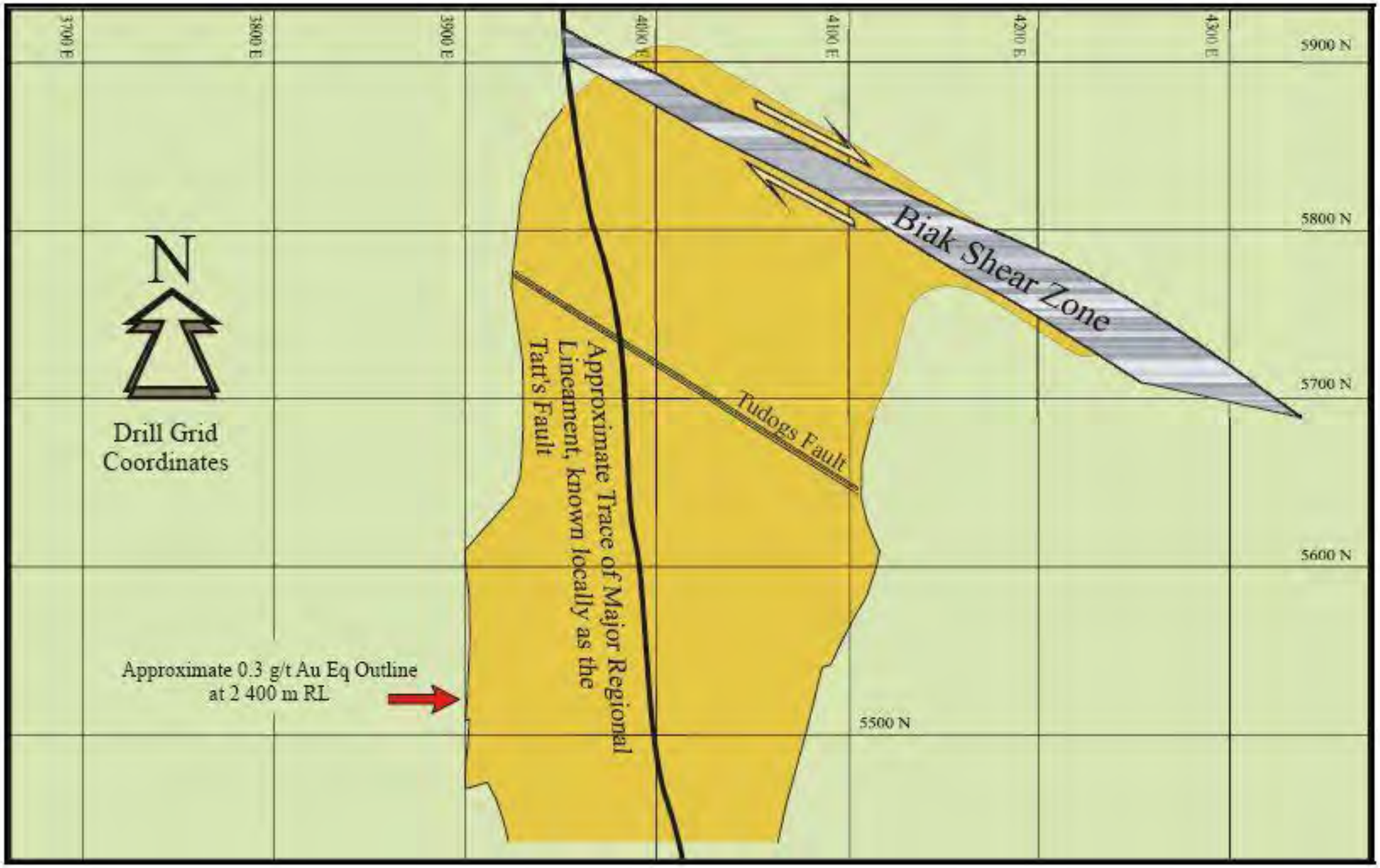
Three major faults have been named and their relative locations are shown in Figure 7.8.

Tatts Fault – a major grid north-south trending, steeply (80° to 85°) east-dipping fault passing through the centre of the deposit, which is regarded as a major structure controlling emplacement of the Quan diorite and Bugoy breccia, as well as being a possible major conduit for mineralisation and alteration. No movement on this fault has yet been recorded and although the fault is regarded as part of a regional lineament, it has not been demonstrated to be a major plane of weakness.

Biak Shear Zone – a major north-west-trending, steeply north-east-dipping fault, against which mineralisation is displaced at the northern end of the deposit. This fault comprises a 30m to 35m wide zone of anastomosing chlorite-haematite shear planes and contains carbonate-sulphide veins, with remobilised gold-copper mineralisation. The shear zone is hosted principally in Dark Diorite, which can be strongly carbonated. Movement on the fault is dextral with the horizontal component of movement being between 50m and 100m, north block to the east. High-grade mineralisation associated with QFC veining and brecciation appears to terminate against the Biak Shear, although low-grade mineralisation has been intersected in a Tunja diorite equivalent further to the north.

Tudogs Fault – less clearly defined than the Biak Shear Zone, particularly in shallower parts of the deposit. This fault is broadly parallel to the Biak Shear, striking north-west and dipping steeply. In places, it appears to form the southern limit of the high-grade mineralisation and is marked by brecciated or heavily fractured host rocks. No direction of movement has been established for this fault.

Figure 7.8: Approximate Location of Major Structures (2400mRL)



7.3.4 Brecciation

A number of different breccia types are evident, of which the most important in terms of mineralisation is the Bugoy breccia. This unit was originally identified between Sections 5750 N and 5800 N on the drilling grid as a breccia pipe, and was thought to have formed by reactivation of the intrusive contacts (Garrett, 1995). It appears to be rooted in the Leached Zone (Bufu) and extends upwards as a possible hydrothermal breccia containing rounded to sub-rounded pebbles of quartz and occasionally skarn material, up to 50mm in diameter, in a sandy-chlorite-sulphide gouge matrix.

The Bugoy breccia can be more than 15m wide close to the Bufu and is often host to high-grade gold mineralisation. The upper part of the Bugoy breccia is a more typical hydrothermal breccia, with angular monzodiorite porphyry clasts in a chlorite-sulphide matrix.

Contact breccias are common on the margins of the deposit where monzodiorite (Tunja) intrudes the Dark Diorite.

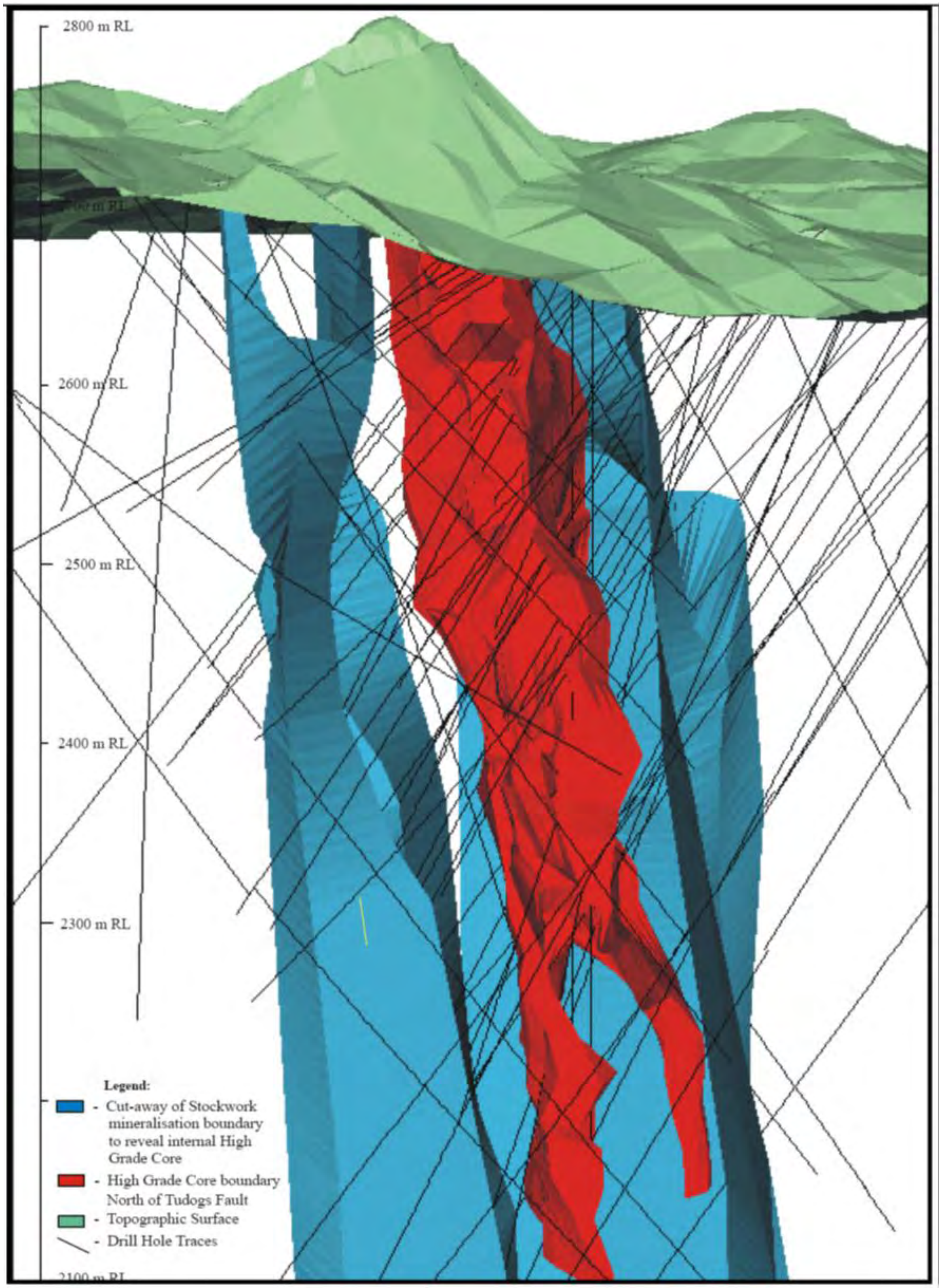
7.4 Geometry of Mineralisation

Porphyry-style gold-copper mineralisation has been recorded over a strike length of approximately 450m (grid references 5400 N to 5850 N on the drilling grid at the surface), a width of up to 150m (grid references 3900 E to 4050 E at the surface) and to a vertical depth of greater than 800m). The tabular composite intrusive and associated alteration and mineralisation strike grid north-south and dip steeply (80° to 85°) east. Higher-grade gold and copper mineralisation is closely associated with the Quan diorite and Bugoy breccia, both of which are elongated in plan view along the north-south trending, steeply east-dipping Tatts Fault Zone.

This mineralisation may have been remobilised, or formed during a slightly later phase of hydrothermal mineralisation with a strong structural control, and is surrounded by stockwork mineralisation that extends as a steeply east-dipping ellipsoidal shaped body, 110m to 140m wide, from the surface to a depth of 500m (grid references 5550 N to 5800 N). Below 500m depth (2350mRL to 2300mRL), the mineralisation is more tightly constrained forming a carapace around the Bufu syenite, with extensions of higher-grade mineralisation continuing southwards along discrete structures as shown in Figure 7.9.

Higher gold-copper grades are also localised within the footwall (west) skarn, which is 5m to 15m wide, sub-vertical, open at depth and contains vein-type mineralisation over a strike length of 150m (grid references 5675mN to 5825mN).

Figure 7.9: Three-Dimensional View of the Didipio Project Deposit Mineralisation Outlines (North Facing)



7.5 Weathering and Oxidation

The deposit is oxidised from the surface to a depth of between 15m and 60m, averaging 35m. The oxide zone forms a blanket over the top of the deposit and consists largely of secondary silicification, clay and carbonate minerals, accompanied by secondary copper minerals including malachite and chrysocolla.

There is evidence that oxidation of the gold-copper sulphide mineralisation can extend along fractures down to 2400mRL within the core of the deposit, but such instances appear rare.

A transition zone, 5m to 15m thick, is present between the oxide and sulphide zones over most of the deposit. This unit is imprecisely defined, with sulphides being observed near the surface. The location of these recently interpreted surfaces, however, appears to agree with independently determined copper float recovery sample results.

Supergene enrichment is not a characteristic feature of gold-copper porphyry deposits in the Philippines and does not occur to any significant degree at the Didipio Project.

7.6 Mineralisation and Mineralogy

Chalcopyrite and gold (electrum), along with pyrite and magnetite, are the main metallic minerals in the deposit. Chalcopyrite occurs as fine-grained disseminations, aggregates, fracture fillings and stockwork veins, particularly within the QFC zone of alteration. It is present in a variety of fracture fillings and vein types, including quartz, quartz-carbonate, quartz-feldspar, carbonate-sericite, quartz-chlorite and calc-silicate (actinolite)-K-feldspar pegmatitic veins. Chalcopyrite has locally replaced magnetite and may, in turn, have been replaced by bornite. Bornite occurs as alteration rims around and along fractures within chalcopyrite grains. Chalcopyrite and bornite often occupy a central position in veins and appear to be relatively late-stage minerals.

Visible gold is not common but has been detected in drill cores, as for example in DDDH47 at 777m down hole and DDDH34 at 394m down hole. Polished section and scanning electron microscope studies have resulted in identification of gold both as isolated grains (up to 80 microns in diameter) and as two micron to 15 micron-sized grains either on the margins of, or as inclusions in, chalcopyrite and galena. Gold grades are commonly higher where bornite is present.

Pyrite is the other main sulphide mineral, occurring principally as disseminations and fracture fillings. Minor sulphides include pyrrhotite, hypogene chalcocite and covellite, and sphalerite. In addition, very minor amounts of molybdenite, galena, hessite (Ag_2Te) and tetrahedrite have been observed from polished section and scanning electron microscope work carried out (Mitsui, 1993).

The occurrence of telluride minerals is unusual in Philippine calc-alkaline porphyry deposits (Sillitoe and Gappe, 1984) and such minerals may be indicative of a late-stage epithermal mineralisation event at the Didipio Project. Open-space filling textures have locally been observed in quartz veins and may support the existence of a late-stage epithermal event.

Magnetite is both primary, crystallising with ilmenite from the diorite to monzonite melts, and also as a secondary mineral in veins, accompanying the earlier stages of hydrothermal alteration. However, the marked decrease in magnetic susceptibility levels associated with more intense alteration and mineralisation towards the core of the deposit is indicative of magnetite destruction as a predominant feature of the main mineralising event.

Highest gold and copper grades (up to 50 g/t Au and 5% Cu) occur in the QFC Zone and the Bugoy breccia, in the area immediately surrounding the Leached Zone; within skarn mineralisation; Mixed Zones; and, less commonly, at contacts between altered and unaltered rocks.

Brecciation of the QFC at the top of the Leached Zone (Bugoy breccia) is characterised by very high gold-copper grades. Here the gold and copper may well have been remobilised and concentrated within the breccia matrix.

Garrett (1995) noted that, within the QFC Zone, highest-grade mineralisation is generally coincident with an overlap of Mixed Zone alteration. Where the Mixed Zone does not coincide with the QFC Zone (that is at depth), grades are typically low. Garrett suggested that the Mixed Zone is both contemporaneous with, and post-dates, the QFC Zone. The Mixed Zone is also notable in that it includes significant disseminated chalcopyrite-bornite-pyrite mineralisation, a feature not common in other alteration zones.

Very high grade gold-copper mineralisation is also a feature of the skarn zone, where it occurs typically as coarse (2mm to 4mm) disseminations of chalcopyrite-bornite-magnetite overprinting the calc-silicate matrix.

Outside the QFC Zone, chalcopyrite and gold mineralisation are generally lower grade, occurring in CSS veins and as fine disseminations (in selvage alteration to the veining) accompanying SCC-K-feldspar alteration. Minor disseminated chalcopyrite may also occur with magnetite and chlorite as retrograde alteration of mafic grains. Locally, there is strong development of disseminated mineralisation.

Figure 7.10: Didipio Project Deposit – Plan at 2300mRL showing Geology and Mineralisation Outlines

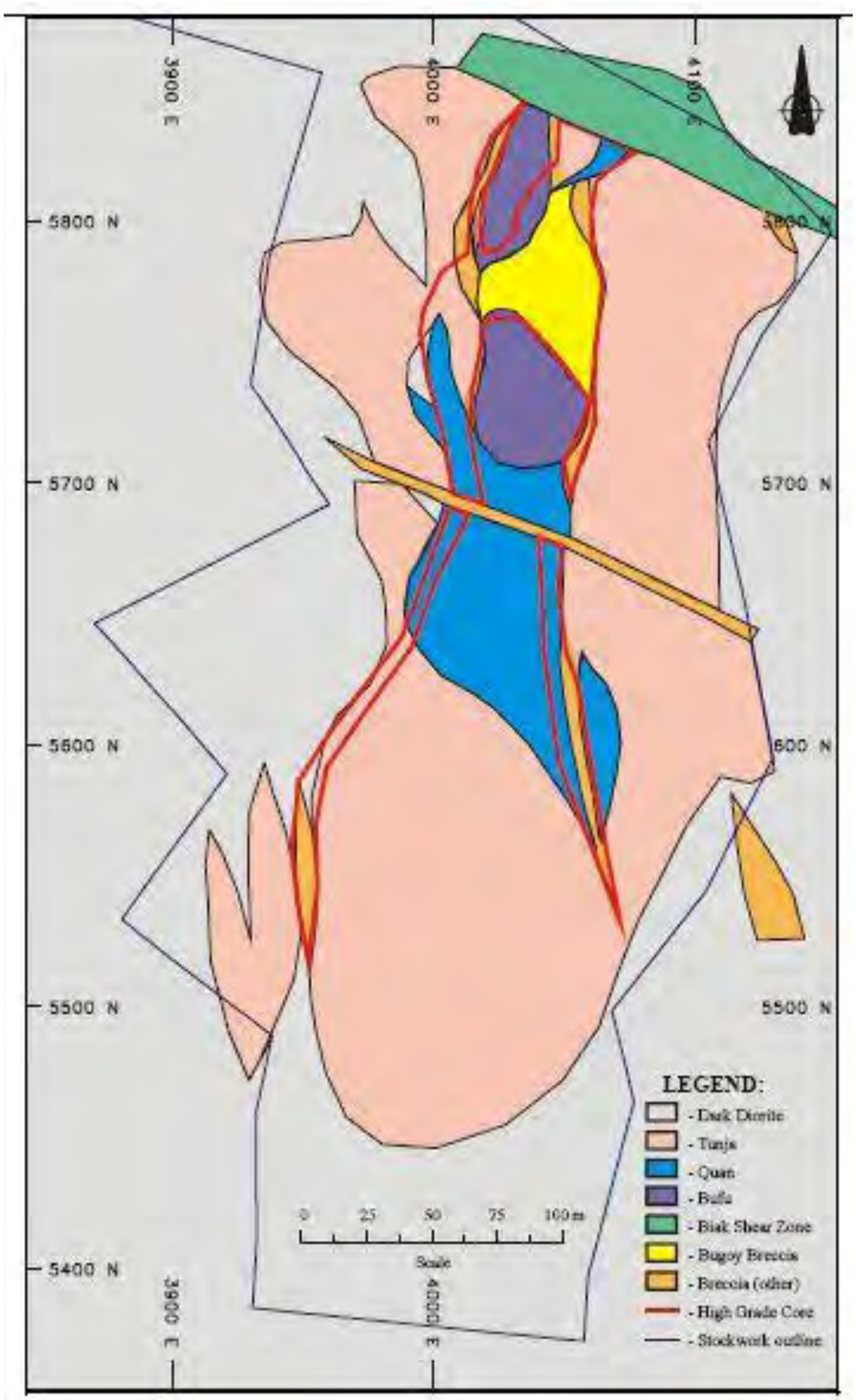


Figure 7.11: Didipio Project Deposit – Geological Cross-Section 5800 N

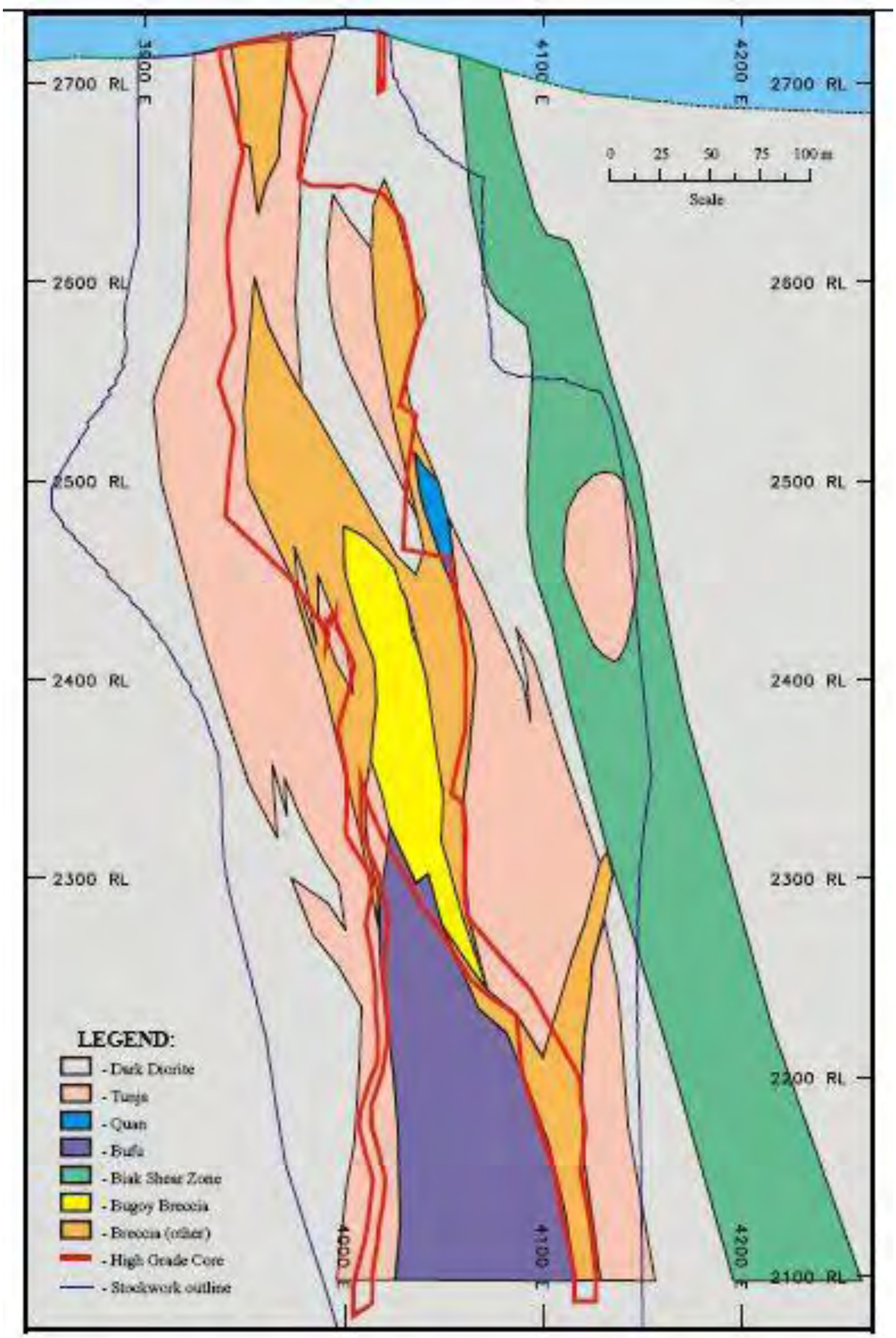
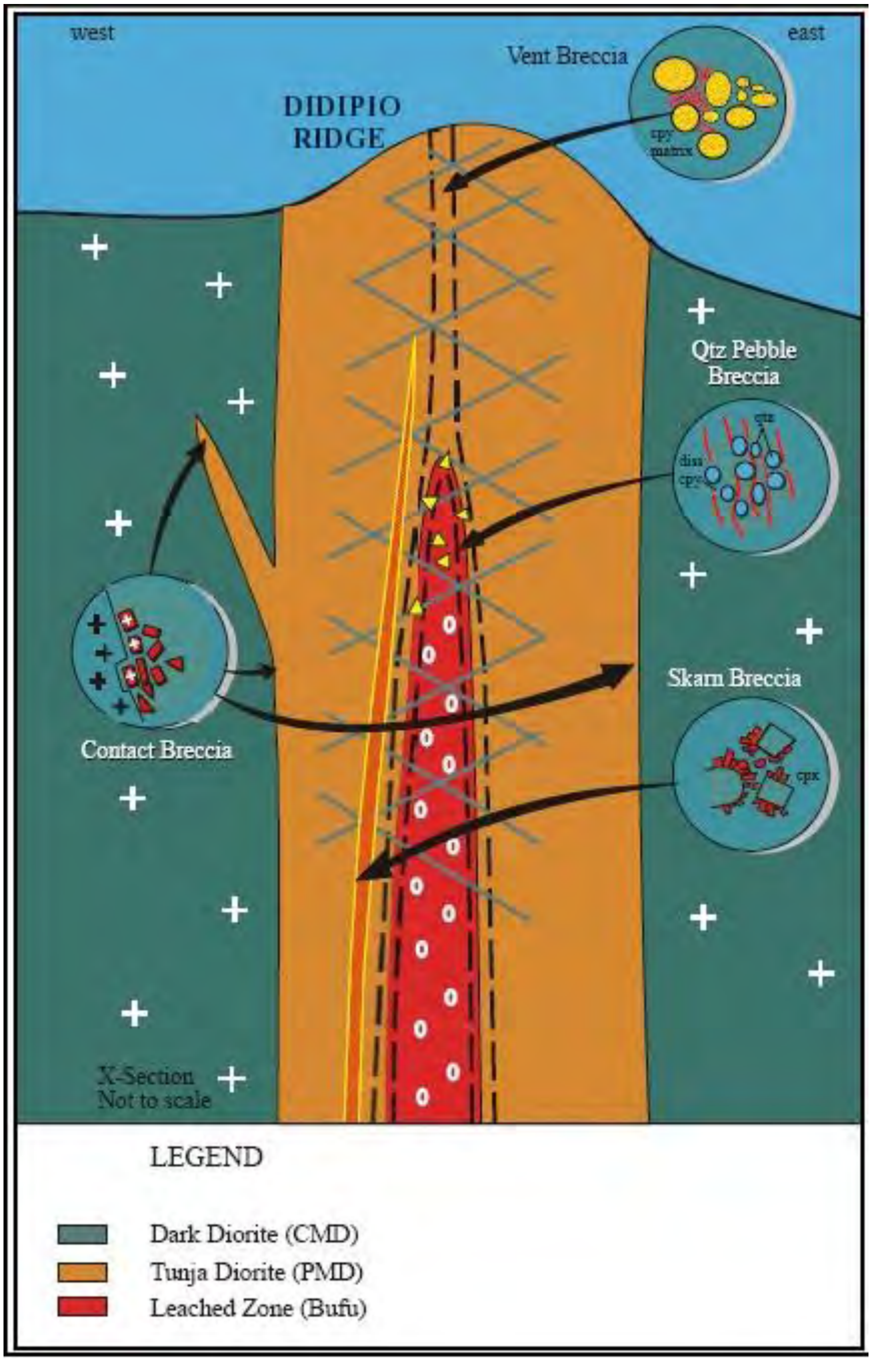


Figure 7.12: Stylised Cross-Section showing Breccia Fabrics associated with Alteration and Mineralisation



8 DEPOSIT TYPES

The Philippines Archipelago constitutes one of the world's premier porphyry copper provinces and is a typical area for the study of island arc porphyry systems.

8.1 Description of Deposits

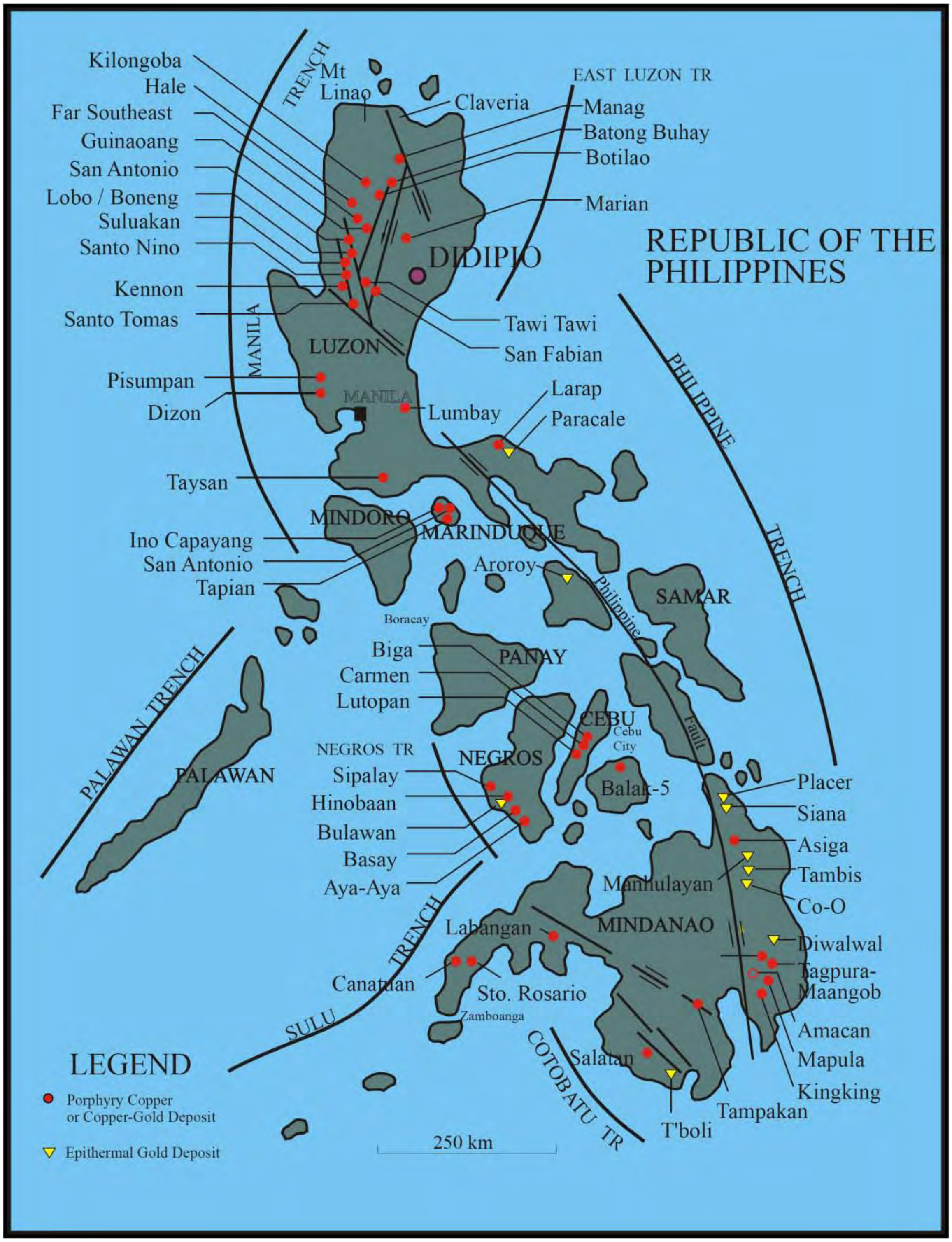
In a comprehensive review, Sillitoe and Gappe (1984) reported the characteristics of 48 mineralised predominantly calc-alkaline porphyry deposits in the Philippines, many of which have been in production (see Figure 8.1). The size of the deposits varies from 50 Mt to more than 300 Mt and copper grades are characteristically 0.40% Cu to 0.55% Cu, with gold content varying from 0.1 g/t Au to 0.4 g/t Au.

The list following does not cover all known characteristics, but it provides a framework into which it is possible to fit many of the geological features of calc-alkaline porphyry deposits and construct a generalised genetic model of a typical Philippines copper-gold porphyry deposit.

Sillitoe and Gappe (1984) found that the majority of calc-alkaline porphyry deposits studied:

- occurred in subduction settings;
- were emplaced into volcanic, volcano-sedimentary or subordinate fine-grained sedimentary sequences of late Mesozoic (95 Mya) to Neogene (5.3 Mya) age;
- are centred on small (mainly <0.5 km^2 in plan), roughly cylindrical composite stocks of diorite to quartz-diorite porphyry;
- show the development of syn-mineral and post-mineral intrusive phases. These may occur as low-grade deep cores to the deposits or as larger, phaneritic plutons that truncate the deposits at depth;
- were emplaced in strike-slip fault zones of regional extent;
- show development of widespread K-silicate, sericite-clay-chlorite and propylitic alteration, combined with more restricted sericitic, advanced argillic and calc-silicate development;
- are characterised by pyrite-chalcopyrite-bornite-magnetite mineralisation introduced as part of the K-silicate alteration phase;
- are characterised by widespread overprinting of K-silicate alteration by the sericite-clay-chlorite assemblage, with attendant partial alteration of magnetite to haematite;
- contain ore zones having steep cylindrical forms preferentially developed in intrusive rocks;
- show a positive correlation between gold and hydrothermal magnetite;
- show evidence that a major part of the gold was introduced with K-silicate-related copper mineralisation, and that more than 50% of the gold is closely associated with chalcopyrite and bornite;
- contain hydrothermal breccias of syn- and post-mineral age, as pipes, dykes and irregular bodies; and
- exhibit thin (generally <50m) supergene profiles developed since the Pliocene. In situ oxidation of pyrite mineralisation resulted in goethitic cappings containing oxide copper minerals. Supergene enrichment is not common, probably due to the low pyrite content and neutralising capacity of the K-silicate alteration style.

Figure 8.1: Distribution of Porphyry Cu-Au Deposits and Epithermal Deposits, Philippines Archipelago



While the Didipio Gold-Copper Deposit has many broad similarities to the geological features documented by Sillitoe and Gappe (1984), it is not a classic, large porphyry-style deposit. Rather, it is a smaller alkaline mineralised stock containing disseminated and stockwork gold-copper mineralisation that has been overprinted by late stage, structurally controlled, higher-grade, gold-copper mineralisation. There is no commonly agreed, detailed model for the formation of the Didipio Gold-Copper Deposit, although there is general agreement about the style of the mineralisation and many of the key elements. The framework appears to be as follows:

- Intrusion of Dark Diorite as a composite intrusive of clinopyroxene microdiorite (CMD) followed by porphyritic monzonite porphyry, with intrusive breccia developed along the contacts. The later intrusive (and all subsequent intrusives) appears likely to have been controlled by the grid north-trending Tatts Fault.
- Intrusion of biotite clinopyroxene monzodiorite (Tunja diorite), probably accompanied by some potassic metasomatism and biotite-magnetite alteration along the contacts and for up to 200m into the Dark Diorite. Some pervasive K-feldspar alteration and veining may have accompanied this event.
- Intrusion of Quan monzonite porphyry into the Tunja intrusive, with accompanying magmato-hydrothermal alteration leading to formation of mineralised skarn and calc-silicate pegmatite at the Tunja/Quan contact and calc-silicate-K-feldspar veining (CSS veins of Garrett, 1995) extending into adjacent Tunja rocks. K-feldspar flooding also extended along the contact into Tunja diorite.
- Bufu "microgranite" emplaced as a separate but related intrusive, or possibly representing a deeper crystallising phase of the Quan. Development of a silica-rich cap to the Bufu and build up of hot SiO_2-CO_2 rich fluids beneath this cap.
- Multiple pressure release events related to continuing movement on the Tatts Fault, or due to overpressuring. Initially, weak development of quartz+K-feldspar-sulphide stockwork and irregular veining (QFC of Garrett, 1995) concentrated in the Quan above the Bufu intrusive, and in adjacent Tunja rocks.
- Formation of Bugoy breccia due to a combination of physical disruption and hydrothermal brecciation of the silica cap, quartz-sulphide stockwork veins and local adjacent skarn rocks. The timing of this event is unclear, but the matrix is often strongly mineralised and thus the event accompanies a significant period of hydrothermal alteration and mineralisation.
- Cooling and mixing of magmato-hydrothermal and meteoric waters leading to pervasive sericite-chlorite-carbonate-sulphide alteration (SCC and mixed alteration styles of Garrett, 1995) occurring along contacts and other fractures and cavities within Quan and Tunja lithologies.
- Late-stage mixing, cooling and collapse of the hydrothermal system, with clay-carbonate-zeolite alteration along open fractures in Quan, Tunja and Bugoy breccia.

Garrett (1995) recognised post-mineralisation shearing and brecciation as exemplified by the Biak Shear, with associated remobilisation of gold-copper mineralisation into these shear zones. Wolfe (1996) suggested that there was a more extensive post-mineralisation carbonate alteration event, with carbonate+sulphide and late silica veining within the body of the deposit as well as within the Biak Shear.

An age date of 23.2 ± 0.6 My has been reported by Wolfe (1996) for a rock specimen tested for Newmont from a K-feldspar vein within the nearby True Blue prospect biotite monzodiorite. It is likely that this date is broadly synchronous with the intrusion of the Didipio Project monzonite suite and its associated mineralisation.

9 EXPLORATION

9.1 Exploration Work and Results

The exploration history of the Didipio Project is detailed in sections 6 and 10 of this report.

An infill drilling programme, targeting mineralisation both within the open pit and underground designs, was completed in mid 2008. Twenty-one infill drill holes for 7390.6m were drilled and incorporated into the existing drill hole database for October 2008 resource update.

9.2 Interpretation

Exploration has defined a substantial gold-copper resource at Didipio. The resource estimate is detailed in Section 14. The present programme aims to improve our understanding of the high-grade gold/copper core of the deposit as well as improve confidence within the open pit design. Results to date confirm the geological and grade models established previously. High grades were encountered within the Bugoy breccia and wide intervals of low grade were encountered through the monzonite ore body.

9.3 Details of Operators

All drilling at Didipio has been performed by contractors, as detailed in section 10, while most of the sample preparation was performed by Climax personnel at Cordon and assaying by Analabs (see section 11). Samples taken during the 2008 infill drilling programme were prepared and analysed by McPhar Laboratories of Manila.

Historically, three grids have been used in the collection of survey data within the Didipio Project area. These are discussed in the previous, 2010 technical report.

10 DRILLING

10.1 Drilling and Drill Hole Surveying

As at December 31, 2009, the complete drill hole database for the Didipio project contained 341 holes for a total of 81,992.9m drilled. The drill hole database for the Didipio Ridge deposit comprises 183 holes totalling 46,177.9m, although only 98 holes totalling 39,421.2m are diamond core holes considered suitable for resource estimation.

An infill drilling program at the Didipio Project was completed in mid-2008. This program, which aimed to improve our understanding of the high grade gold/copper core of the deposit as well improve confidence within the open pit design, comprised 21 infill drill holes for 7,390.6m. These drill holes were incorporated into the October 2008 resource update.

An in-fill programme was designed and undertaken in the first half of 1997 to reduce drill hole spacing to approximately 50m down dip on sections 25m to 50m apart, concentrating on the high-grade mineralisation in the north-western part of the deposit.

Up to 31 July 1995, a total of 74 diamond drill holes had been drilled on the Didipio project (see Figure 10.1). Fifty-nine of these holes were drilled at Didipio Ridge, including oxide definition holes, largely on 50m sections, with a vertical separation of 120m to 180m. Diamond drilling on site has been carried out by several different contractors, but from January 1994 (from drill hole DDDH29 onwards) all holes were drilled by one of two contractors, Core Drill Asia or Diamond Drilling Company of the Philippines. Both contractors used Longyear drilling rigs and wireline drilling methods.

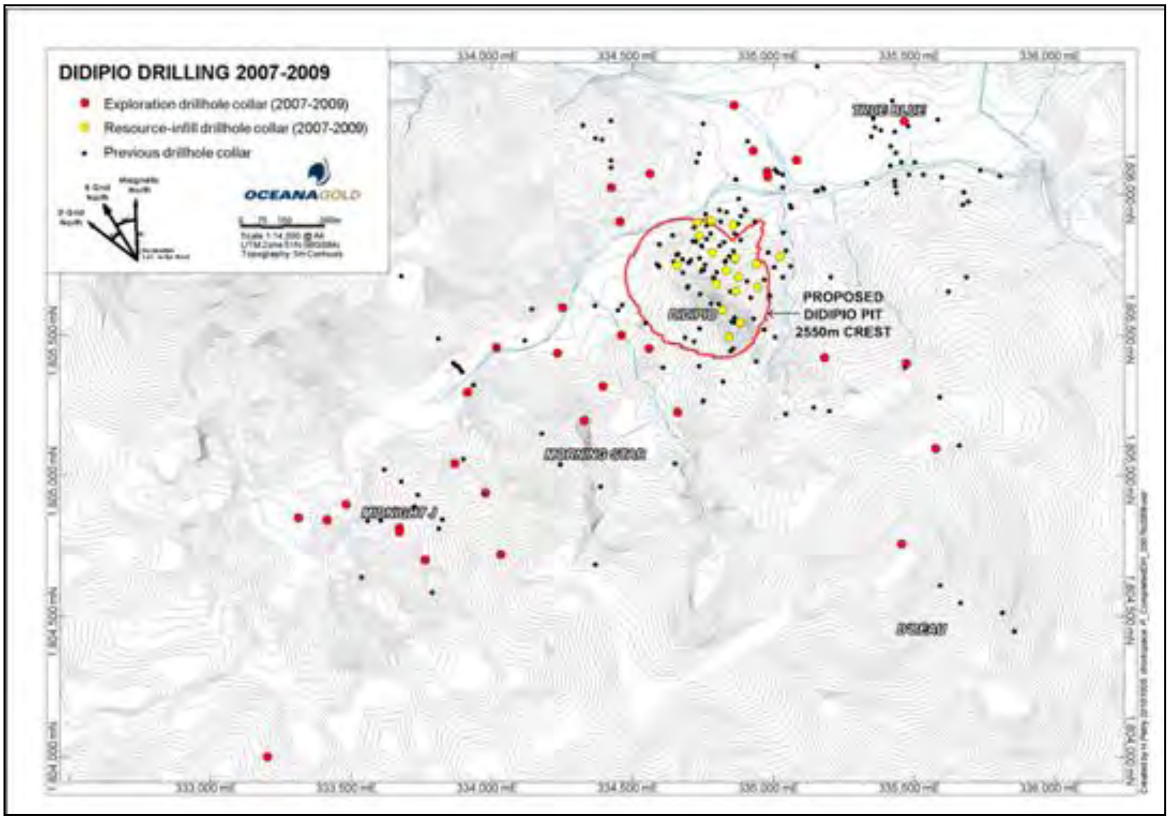
Earlier holes were collared using 5¼" roller bits to refusal (generally less than 10m depth), cased off and then drilled HQ (63.5 mm core diameter) as far as possible, reducing to NQ (47.6 mm core diameter) as required. Depth limitations with HQ equipment were generally around 600m.

Where possible, all drill holes have been surveyed down hole, generally at 50m to 100m intervals, using an Eastman survey camera. Overall, down hole directional changes are generally minor: holes tend to steepen by 3° in the first 100m and 1° per 100m or less thereafter. Little change in azimuth was noted where holes were drilled perpendicular to strike, whereas drill hole DDDH47, which was drilled sub-parallel to strike, deviated by 15° over 1005m.

Down hole survey readings were examined for anomalous values related to local high concentrations of magnetite. Within the mineralised zones, low magnetic susceptibility readings on the drill core indicated little potential for magnetic interference on down hole azimuth measurements, whereas a few spurious azimuths from more highly magnetic, generally universalised units were noted and rejected from the database.

The mineralisation at Didipio Ridge has a steep easterly dip and the majority of holes were drilled at around 60° to the west, which is considered appropriate. At a 1.0 g/t equivalent gold grade, the mineralisation averages around 80m in true thickness; the nominal sample length of 2m or 3m is considered more than adequate to define the grade distribution within this zone.

Figure 10.1: Didipio Project Geology Plan, showing Drill Hole Locations



10.2 Tunnel Sampling and Trenching

Tunnel sampling and trench sampling data were excluded from the resource estimation database.

During early exploration at the Didipio Project by Climax, a total of eight trenches were cut down to bedrock across part of the ridge at irregular intervals, for a total length of 237m. Depths from surface varied from less than 1m to 2m. These trenches were channel chip sampled in 10cm wide by 5cm deep channels, at intervals ranging from 2m to 5m (averaging 3m), providing a total of 155 samples in the database.

In addition, 21 near-horizontal tunnels were developed by local miners to investigate high-grade gold mineralisation in shears, veins and breccias in the upper part of the Didipio Ridge. Tunnel location and orientation depended on topography. Channel sampling along the walls was carried out by Climax over 2m sample intervals to provide a total of 178 samples to the database.

Both trenches and tunnels only investigated the oxide zone. They were surveyed by tape and compass only and geologically mapped at 1:100 scale.

In 2008 five trenches for 88m on the spine of the Didipio hill top were excavated and channel / chip sampled at 2m intervals. The results confirmed strong copper mineralisation within the oxide zone.

10.3 Infrastructure Sterilisation Drilling

A total of 56 diamond drill holes have been drilled for sterilisation and infrastructure. Drill hole collar locations are shown in Figure 10.2.

At the time of issue of the GRD 1998 DFS, no significant mineralisation had been intersected during sterilisation drilling.

Additionally, the following exploration has been conducted over the intended plant site, underground infrastructure, waste dump and tailings dam sites, and the accommodation village:

- induced polarisation surveys;
- aerial geophysical surveys (including magnetics);
- geochemical surveys; and
- geological mapping.

10.4 Logging Procedures

Immediately after retrieval from a drill hole, a drill core is colour photographed in wet and dry state. Some cores, particularly from early drill holes, were also re-photographed after splitting with a diamond saw. Later in the programme, a digital camera was used to photograph each core.

On site, core logging and marking up is carried out in several stages.

Initial geological logging is carried out by the site geologist using logging sheets and/or notes to construct a brief geological log that includes:

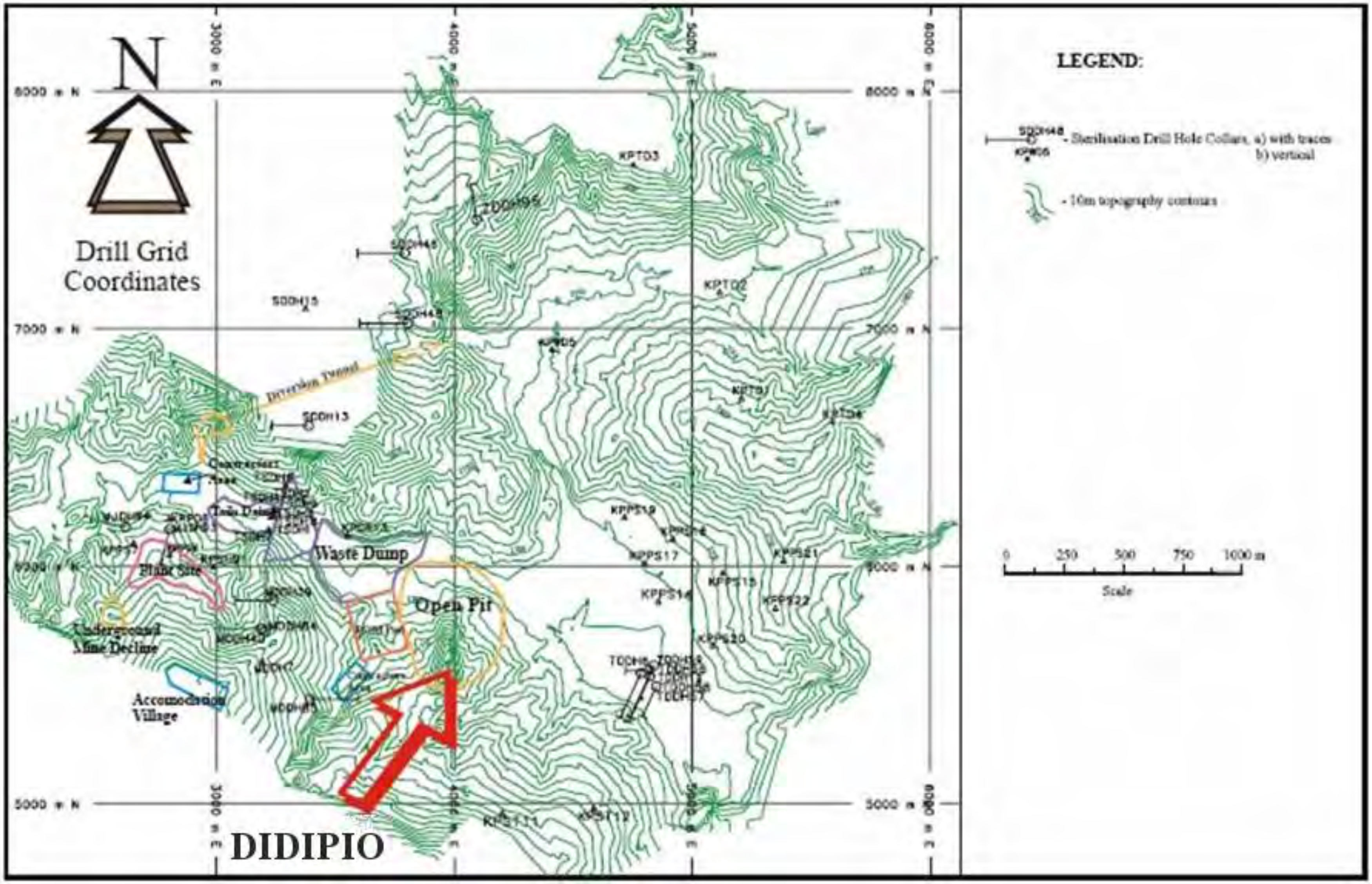
- lithology;
- alteration; and
- mineralisation.

Geotechnical logging uses standard logging forms:

- recoveries;
- orientations;
- rock quality – RQD; and
- physical property measurements:
  - point load testing (after DDDH31);
  - magnetic susceptibility measurements are taken at approximately four readings per metre; and
  - specific gravity determinations.

Detailed geological logging is generally carried out after the core is split and sampled. For consistency in geological interpretation, Sam Garret of Climax (1995-97) has logged all Didipio Project drill cores. All physical property data is included in the database.

Figure 10.2: Position of Sterilisation and Infrastructure Holes around the Didipio Project Deposit



10.5 Sampling Methods

Ninety-eight holes totalling 39,421.2m define the Didipio Gold-Copper Deposit. These drill holes are generally spaced on sections with 25m to 50m along strike separations and with vertical separations of 50m in the north-west of the deposit. To the south-east, vertical separations up to 150m are more usual. This covers an approximate area of 300m across strike by 550m along strike. Down hole core sample intervals are generally 2m or 3m. A plan view of the drill hole collars is provided in Figure 10.1.

From this drilling, 11,635 samples were used for resource estimation.

Sample intervals were defined during the initial logging of cores on site. Core was cut in half using a diamond saw either on site (up to hole DDDH16) or at Cordon (holes DDDH17 onwards). Core has typically been sampled in intervals 2m or 3m under supervision of the site geologist or sample preparation manager, generally crossing rock type boundaries. After sampling, the remaining half core was stored for further technical and/or metallurgical purposes. In 1992, all drill cores on site were moved and stored at Climax"s facilities at Cordon.

10.6 Core Recovery and Sample Quality

Core recoveries were generally better than 95%, although in local areas of severe structural deformation recovery was as low as 50%.

A review of core recoveries indicated that there was no strong relationship between core recovery and grade, so there appears to be no systematic bias in grade due to poor sample recovery. Therefore, sampling is considered representative.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

Since 1989, sample preparation of Didipio drill cores has been conducted in four phases, with each phase using slightly different sample preparation procedures. Almost all of the samples (89%) were prepared by Climax employees. Details of each method are described in detail below and are summarised in Table 11.1.

Table 11.1: Didipio Project Sample Preparation

Phase	Period	Company	Sample preparation	Drill holes	Number of samples	% of total database
1	1989	CYPRUS	ANALABS (MANILA)	DDDH1-5	352	3.1%
2	1990-1	ARIMCO	ANALABS (MANILA)	DDDH8-11	350	3.1%
3		ARIMCO	AMC	DDDH14-16	252	2.2%
4	1992-1998	CLIMAX	CLIMAX	>DDDH-18	8051	70.39%
2	2008	OGC	McPHAR (MANILA)	>DDH-200	2442	21.3%

At the town of Cordon, Climax maintained a sample preparation facility comprehensively stocked with diamond saws, crushers, pulverisers, mills and riffle splitters. A large working area was kept relatively clean and dust free by means of an efficient extraction system. The sample preparation and core storage areas were under the supervision of experienced local staff.

The following sample preparation sequence was used by Climax:

- Oven-dry quarter core samples.
- Jaw crush to minus 6mm.
- Disc pulverise to minus 2mm.
- Hammer mill to minus 1mm.
- Riffle split into two by 2kg samples and fine pulverised with one split to minus 200 mesh (second split stored in freezer for future test work or analysis).
- Screen >95% minus 200 mesh.
- Riffle split 150g to 200g for assay.
- All sample rejects stored.
- Prepared samples air freighted to Analabs Proprietary Limited (Analabs) in Perth, Western Australia for assay.

For the 2008 OGC drilling (DDH0201 to DDH0221), the diamond core was cut at Didipio. Half core was transported to the McPhar facility in Manila for crushing and pulverising to 90% passing 200 mesh. Gold was fire assayed with an AAS/GTA finish, while an acid digest was for the copper.

11.2 Analytical Methods

11.2.1 Gold Assay Procedures

The standard gold assay procedure used by Analabs in Perth (NATA certified)4 was as follows:

- Laboratory Method Code 313:
  - A 50g sample pulp was fired with litharge and flux and the lead-silver button cupelled. This was followed by acid dissolution of the silver-gold prill, and gold content was measured by Atomic Absorption Spectrometry (AAS) to a 0.005ppm Au lower detection limit.
  - Assay for gold in samples from DDDH1 to DDDH6 were performed by Analabs in Manila, but this practice was discontinued in November 1989. The same procedures were used by the Manila and Perth laboratories.

11.2.2 Copper and Silver Assay Procedures

The standard procedures used by Analabs, Perth, for copper and silver assays were as follows:

- Laboratory Method Code 101:
  - Perchloric acid digest then AAS finish to a 4ppm lower detection limit for copper and a 2ppm lower detection limit for silver.
  - For samples containing >1% Cu:
- Laboratory Method Code 104:
  - Mixed acid digest followed by volumetric dilution and AAS finish to a 25ppm Cu lower detection limit.
  - Results for silver have generally been close to or less than the detection limit of 2ppm Ag and have therefore not been included in any resource estimates.

11.2.3 Analysis of Other Elements

Sulphur analyses were carried out by Analabs, using the Leco method, on 833 composites made up of assay sample pulps. These composites were selected by Climax to coincide approximately with the boundaries of the 15m square mining blocks proposed as part of the GRD 1995 PDS, and thus do not coincide with geological or grade boundaries. No check sulphur analyses have been undertaken.

Metallurgical testing has shown evidence of minor quantities of other metalliferous elements including molybdenum, lead and zinc, but none are present in significant amounts.

11.2.4 In Situ Density Determinations

In situ density determinations have been carried out at regular intervals on a large number of drill core samples (DDDH1-DDDH28 every 5m; DDDH29-DDDH61 every 10m). The method involved drying and sealing the selected sample with a waterproofing compound, then weighing the sample both in air and in water. Each sample comprised approximately 10cm of half drill core.

Data from a total of 2302 samples were statistically analysed. Paper records for 1173 SG measurements were located at the Cordon core facility in August 2008. These were scanned, entered into Excel and finally loaded into Minesight for 3D Geological Coding. The SG values are tabulated in Table 11.2 and are similar to that used by Hellman and Schofield in its 2007 resource estimate.

Table 11.2 and are similar to that used by Hellman and Schofield in its 2007 resource estimate.

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[4] The National Association of Testing Authorities (NATA) is Australia's national laboratory accreditation authority and the largest such system in the world. NATA accreditation recognises and promotes facilities competent in specific types of testing, measurement, inspection and calibration.

Table 11.2: Statistics for Specific Gravity Data by Rock Type

	Oxide	Trans	Tunja	Bufu	Biak	D diorite	Breccia
No. samples	31	NA	474	17	86	558	7
Mean	2.42	NA	2.51	2.39	2.66	2.73	2.56
Median	2.35	NA	2.52	2.37	2.72	2.75	2.57
Mean minus extremes*	2.51	NA	2.51	2.40	2.67	2.73	NA
Minimum	2.09	NA	2.09	2.01	2.08	2.00	2.54
Maximum	3.03	NA	3.18	2.66	3.11	3.50	2.58
Value used	2.20	2.40	2.50	2.35	2.67	2.72	2.45

* Mean excluding values outside 2.5% and 97.5% quantiles

Additional density measurements have been made for Drill Holes DDDH66 onwards, but results were not available for inclusion in the GRD 1998 DFS.

Australian Geostandards conducted check measurements on 22 adjoining and 11 identical drill core samples. Check measurements were on average 1.3% higher than original measurements.

11.2.5 Magnetic Susceptibility

Four magnetic susceptibility readings were taken every metre along the core axis, although data from DDDH1 to DDDH5 were excluded from the database after inconsistent readings were noted. Magnetic susceptibility values are recognised as significant indicators of alteration (and mineralisation) at Didipio.

In the central part of the deposit, above the 2300mRL, lower magnetic susceptibility readings can generally be related to higher-grade gold-copper mineralisation. However, below the 2300mRL, some areas of magnetite-rich, high-grade gold-copper mineralisation occur with correspondingly high magnetic susceptibility. These zones have been interpreted to be associated with skarn alteration.

11.2.6 Sample Security

Industry-standard sample security measures appear to have been in use at Didipio. There is no specific documentation of these procedures and the author of this report did not take independent samples for checking. However, data verification measures by the author of this report suggest that copper assays are consistent with mineralisation observed in core and gold assays are generally consistent with mineralised features. Extensive metallurgical testwork and independent verification work by other companies also confirms database results.

11.3 Statement of Sample and Assaying Adequacy

The author considers that the sample preparation, security and analytical procedures used for the Didipio project are appropriate and adequate for the style of mineralisation being assessed.

12 DATA VERIFICATION

12.1 Verification Procedures

An extensive external review was completed by Hellman and Schofield in February 2007. This included a one-week visit to the Didipio site and OGPI‟s Manila office (see section 12.2) and formed the basis of the previous two Didipio Project NI 43-101 technical reports.

Twenty-one drill holes have been drilled since this review. The QAQC for this campaign of drilling is discussed in section 12.3.

12.2 Hellman and Schofield Review

12.2.1 Field Verification

Fifteen drill hole collar locations were picked up in the field using a hand-held GPS. These data were downloaded and converted to the local drill grid for comparison with surveyed locations. The GPS collar locations were on average within 7.2m (between 1.8 and 15.0m) of the surveyed locations in 2D (northing and easting) and on average within 8.6m (between 2.2 and 22.9m) in 3D (including elevation). This is well within the accuracy of the GPS unit used and confirms the drill hole collar locations.

Two lease boundary corner markers were also located in the field and checked by GPS. The GPS coordinates for these markers were within five seconds of one degree or about 150m of the nominal coordinates. This difference appears to be due to different coordinate systems being used and suggests that the lease boundary corners are accurately located in the field.

Examination of outcrops, tunnels and trenches visually confirmed the geology and (copper) mineralisation in the field when compared with available maps. Similarly, examination of the core for a number of holes at Cordon also generally confirmed the geology and mineralisation of the deposit when compared with drill hole logs and assays.

Core and sample storage facilities at Cordon were inspected and found to be in reasonable condition. The Climax sample preparation facilities there were also visited; although no longer in use.

Discussions were held with the former project geologist and other personnel to verbally verify various details of the geology and drilling.

12.2.2 Database Validation

Validation of the Didipio database consisted of checking the digital data against original data sources such as assay certificates, logging sheets, collar and down hole survey records, etc.

At least two sets of data were located; one from the 1998 DFS and one from the 2001 Conceptual Study. The data from the later study was assumed to be most up-to-date and was used where available.

Some original data records could not be located for a number of reasons. Few former Climax personnel remain with OGC, so the continuity of knowledge in the project has largely been broken. APMI has relocated offices since the merger and not all reports and data had been organised or located at the time of this review; some information was still in storage and may not have been catalogued correctly. In addition, there was a fire at the Didipio camp in 2005, which may have destroyed some paper records before they were relocated to Manila.

H&S selected six holes from the resource database for detailed checking, covering different years and drilling programs, and representing around 7.5% of the entire database. Holes checked in detail were DDDH4, 25, 42, 53, 83 and DOX3.

The remainder of the database was subjected to a range of checks for completeness and internal

Some down hole surveys were checked where this information was recorded on the drill hole logs and all those checked proved accurate. No original down hole camera discs could be located, so the reading and transcription of this information could not be checked. Magnetic declination was assumed to be negligible, although investigation showed variations of around 1⅓° at the time of drilling and slightly higher variation now (see Table 12.1).

Table 12.1: Magnetic Declination at Didipio

Year	Magnetic declination	Change (per annum)
1989	1° 17'W	0° 1' W
1997	1° 25'W	0° 0' W
2007	1° 44'W	0° 3' W

(source http://www.ngdc.noaa.gov/seg/geomag/jsp/struts/calcDeclination)

Geological logs could only be located for some holes (DDDH1-47, WDDH1-5) so only limited checking could be completed. As the geological logs were the only source of information for collar and down hole surveys, checking of this data was also limited. Available logs confirmed that database geology is reasonably accurate.

12.3 QAQC

A more detailed description of the QAQC analysis can be found in the 2010 Report.

12.3.1 Hellman and Schofield QAQC Analysis for Pre-2008 Drilling

Hellman and Schofield reviewed the QAQC measures for all sampling data available as at January 2007. Their full analysis can be found in the June 2008 technical report. In summary:

- They considered that the sample preparation, security and analytical procedures used for the Didipio Project were appropriate and adequate for the style of mineralisation concerned.
- They noted that the lack of copper standards was a concern.
- In lieu of copper standards, 890 inter-laboratory analyses confirmed that the copper analyses were reproducible within acceptable limits.

12.3.2 Hellman and Schofield QAQC Analysis for 2008 Drilling

QAQC measures employed at Didipio included Cu-Au standards sample resplits, replicate analyses and inter-laboratory check assays.

This analysis was undertaken by Arnold van der Heyden of Hellman and Schofield on behalf of OGC and can be found in the previous 2010 technical report. A brief summary is below.

The QAQC for the recent (2008) drilling program comprised standards, blanks and duplicates (both field and laboratory duplicates). There were no twinned holes or inter-laboratory check analyses for the recent drilling program.

Examination of the data for the standards and blanks revealed a number of obvious errors where the standard or blank was mislabelled. There were 43 of these errors, which were rectified prior to further data analysis.

12.3.3 Assay QAQC Summary

The available resource drilling has been assessed and OceanaGold considers the data to be of a suitable quality for resource estimation purposes.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 General

Test work programmes on the copper-gold deposit at Didipio have been conducted in three major stages. The first programme was conducted from 1990-1993 and basically incorporated a number of bench-scale flotation tests to determine the characteristics of the materials. The second programme was conducted by a number of laboratories from 1994-1995 with more detailed test programmes, including locked cycle flotation tests and two mini-pilot plant studies. The third phase was conducted in 1997, testing primarily underground material and including confirmatory tests based on the flow sheet developed in the previous test work. Recent test work managed by Ausenco and conducted by AMMTEC has generally confirmed previous results.

13.2 Ore Mineralogy

The Didipio mineralogy work has been based on the principal diorites (Tunja monzonite, Dark Diorite and Quan monzosyenite) together with the higher-grade breccia and skarn zones (Bugoy breccia) and the quartz-feldspar-carbonate altered zones. Volumetrically, OGC estimates that the Tunja monzonite will comprise more than 75% of the projected mill feed.

Mineralogical studies were carried out from 1994-1995 by Wally Fander of Central Mineralogical Services and by Ian Pontifex of Pontifex and Associates. In addition, Amdel conducted some optical and X-ray diffraction studies. All three groups are well respected in the industry.

The principal mineralogical characteristics of the primary ore are as follows:

- The principal sulphide minerals are pyrite, chalcopyrite and bornite, with traces of chalcocite and digenite; chalcopyrite is the principal copper mineral; bornite generally contributes less than 20% of the contained copper.
- Magnetite comprises approximately 5-7% of ore, but there are few composite grains with the sulphides.
- The sulphides are generally well liberated; liberation is generally >92% in the float concentrates.
- Minor or trace talc/sericite is present in the higher-grade samples.
- There is little or no evidence of oxidation in the sulphide samples tested except for some tarnishing.

13.3 Metallurgical Samples

The Minproc Feasibility Study reported on the following testwork:

- The Phase 1 testwork was based on samples obtained from early stages of drilling of the deposit, and as such is considered less than wholly representative.
- The Phase 2 testwork studied five separate composites of primary material both low grade and high grade from three vertical sections of the deposit.
- Within the second phase testwork, a programme was conducted on sample composites made up of a large number of mineralisation intercepts.
- Nine variability samples tested in Phase 2 were selected to represent ore feed for the first five years of production and to test each of the four main rock types.
- Two pilot plant studies were carried out. The first was based on approximately 2t of sample comprising 140m of intersections from a single PQ drill hole. The second pilot plant test programme was based on 1.25t of quarter core samples selected from throughout the deposit representing approximately 600m of core.
- Comminution testwork was conducted on a number of composites from HQ core; media competency testwork was conducted on portions of the pilot plant PQ sample.

- In 2006 confirmatory testwork was conducted at AMMTEC‟s laboratory in Perth; three drillholes were sampled and composited into three samples, used for flotation tests and for comminution tests.

13.3.1 Comminution Testwork

A number of studies were conducted to investigate the physical and comminution characteristics of the various mineralised samples. Three laboratories conducted testwork as follows:

- AMMTEC conducted standard comminution tests, including Bond Work Indices tests, on HQ samples from different rock types at different deposit depths and JK Pendulum tests on PQ core from the pilot plant testwork sample.
- Amdel conducted Media Competency tests on the PQ core intersections.
- Lakefield Research in Canada conducted Aerofall grinding evaluation tests on PQ core.

Minproc evaluated the data to determine the appropriate circuit design and correct mill sizing. Table 13.1 summarises the various comminution results.

Table 13.1: Measured Grinding Results

Material type		Bond indices			JK Tech Parameters					
		Ball – BW*i* kWh/t	Rod – RW*i* kWh/t	Abrasion - A*i*	A	b	A*b	DW*i*	t_a	SG
Tunja diorite	Range	12.3 – 14.8	13.2 – 15.2	0.204 – 0.315						
	Average	13.8	14.3	0.277						
Dark diorite	Range	13.8 – 15.1	15.0 – 17.5	0.185 – 0.371						
	Average	14.1	16.2	0.255						
Quan diorite	Range	13.2 – 14.8	13.9 – 15.5	0.211 – 0.337						
	Average	14.1	14.9	0.295						
PQ samples	Range	12.7 – 12.9	12.5 – 16.3	-	71.2	0.54	38.5	-	0.39	2.67
	Average	12.8	14.4	-						
2006 testwork	Average	14.1	14.1	0.1456	74.6	0.90	67.2	3.9		

Note: these indices are standard measures for estimating the power required in grinding, the abrasiveness of the ore, and the suitability of the ore for SAG mill grinding.

These results suggest that the Didipio rock types can be classified as having a low to moderate level of competency, which suggests a relatively low power consumption to reduce the material to the required particle size distribution for processing. The Abrasion Indices also suggest relatively low levels of abrasive wear on grinding media, liners, plant chutes and pipes. Ausenco has adopted 14.6 kilowatt-hours per tonne (“kWh/t”) for the Ball Mill Work Index and 14.5kWh/t for the Rod Mill Work Index with an Abrasion Index of 0.26.

The 2006 testwork programmes were carried out by JKTech Proprietary Limited (“JK”) and Dr Steve Morrell of SMCC Proprietary Limited. JK comments that the DWi, or drop weight index, at 3.9 is relatively low, indicating that the Didipio material is fairly soft with relatively low power requirements to grind to a specified size, with a minimum of critical size development. The parameters A, b and the product A*b also indicate a relatively soft material.

Other comminution tests conducted on the PQ samples are shown in Table 13.2.

Table 13.2: Other Measured Grinding Results

Tested	Autogenous	Unconfined compressive strength			Impact Crushing Work Indices - kWh/t				
	WI	Range	Peak	Failure	102-76mm	76-51mm	51-38mm	38-25mm	25-19mm
	kWh/t	MPa	MPa						
PQ – Avg	13.2				38.9	23.2	9.4	8.7	6.7
PQ – Max					57.8	45.4	13.7	15.4	11.3
PQ – Min					28.3	16.2	6.5	3.8	3.9
Tunja monzonite		45 – 130	130	shear					
Dark diorite		45 – 175	175	shear					
Quan monzosyenite		50 – 110	110	cataclisis					

The Impact Indices indicate that there could be a need for a recycle pebble crusher after the SAG mill since the rock competency increases significantly from the 51mm fraction to the 76mm fraction. However, this is not supported by other data that suggests there will be a minor amount of critical size build-up. It would be appropriate in the plant design to allow for the possible later insertion of a recycle pebble crusher if required.

13.3.2 Gravity Gold Recovery Testwork

Consistent gold recoveries were difficult to attain based on flotation testwork alone. This is not unusual with copper-gold deposits that contain high levels of gold with a significant amount of free gold. Hence it was decided to investigate the use of gravity recovery techniques prior to flotation. Optimet carried out testwork on the nine variability samples based on tabling and hand panning the table concentrates. The overall recovery to a gravity product was approximately 20% or more, indicating that gravity recovery to bullion was likely to be economically viable.

Subsequently, tests were undertaken using a laboratory Knelson high G-force concentrator followed by amalgamation of the Knelson concentrates.

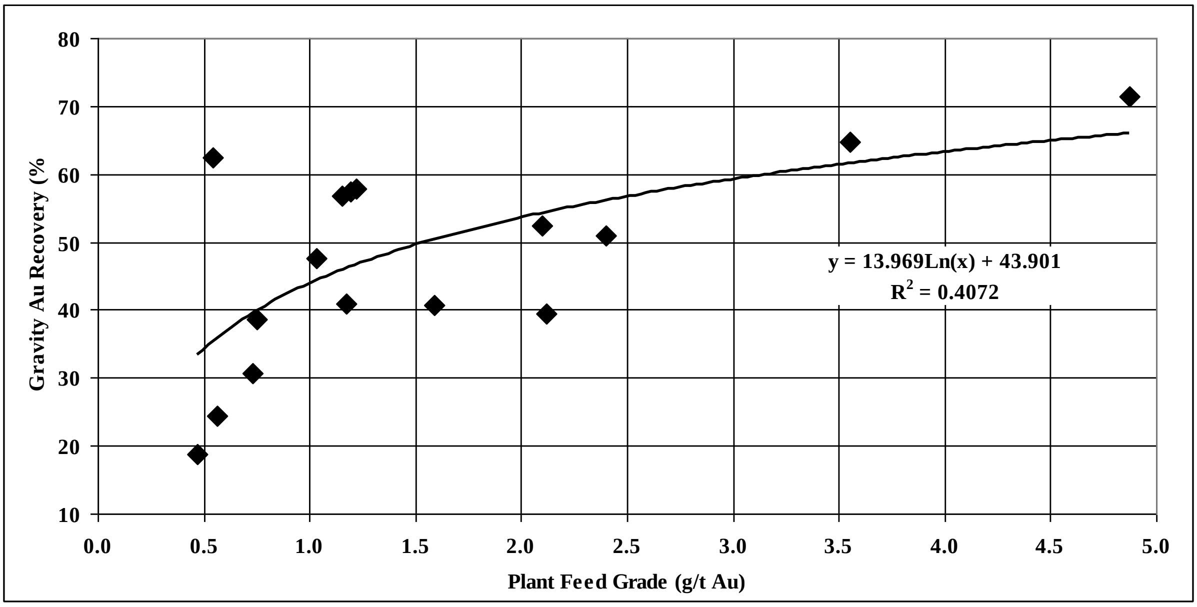
Table 13.3 summarises the results for the nine variability samples, the four composites and the most recent 2006 testwork programme.

Table 13.3: Gravity Recovery Results

Sample	Knelson Concentrate		Head Grade	
	% Wt	Amalgam Au Rec %	Assayed – g/t Au	Calculated – g/t Au
V01 (150µm)	2.0	18.8	0.45	0.47
V01 (106µm)	2.0	24.5	0.52	0.56
V02	2.0	30.6	0.64	0.73
V03	3.1	38.7	0.53	0.75
V04	2.9	47.6	0.83	1.03
V05	2.9	57.9	1.00	1.22
V07	3.1	62.5	0.57	0.54
V08	1.8	57.5	0.96	1.19
V09	2.6	56.9	0.94	1.15
Comp 1		71.4	3.56	4.87
Comp 2		64.8	3.24	3.55
Comp 3		52.4	2.61	2.10
Comp 4		40.8	1.33	1.59
DDH71 - LS0001		39.5	2.12	
DDH71 - LS0002		51.0	2.40	
DDH77 - LS0003		41.0	1.17	
Average		**47.2**		
Median		**49.3**		

The average of the 16 tests indicates a gravity gold recovery of approximately 47%. The gravity recovery is dependent on head grade and ranges from around 35% for 0.5 g/t Au material to >50% for material of 1.5 g/t Au and above. Figure 13.1 provides a graphical representation of the data.

Figure 13.1: Gravity Recovery Results



13.3.3 Flotation Recovery Testwork

Four flotation testwork programmes were conducted from 1990-1995 including optimisation testwork for grind size and reagents, mini-pilot plant studies conducted by AMMTEC, and confirmatory work based on samples from different RL levels. Further confirmatory work was conducted in March 2006.

13.3.4 Preliminary Flotation Testwork

This work tested samples as they became available from early drilling. A standardised flotation procedure was developed. General conclusions were that:

- copper flotation kinetics were rapid;
- copper recoveries were generally high with acceptable concentrate grades;
- over-grinding was detrimental to good metallurgical performance; and
- gold recovery to copper concentrate generally ranged from 80-90%.

The tests incorporated three stages of rougher flotation followed by two stages of cleaning and used standard reagents.

Table 13.4 summarises the results of these preliminary tests.

Table 13.4: Preliminary Flotation Testwork Results

Sample	Location/Drill hole	Head grade		Grind	Concentrate grade		Recovery %	
		% Cu	g/t Au	% - 75µm	% Cu	g/t Au	Cu	Au
Comp 1	DDH4	2.95	6.65	80.0	33	66	98.0	87.0
Comp 2	DDH3	1.30	1.51	80.0	25	24	97.0	79.0
Comp 3	DDH1	0.80	1.35	80.0	18	28	97.0	87.0
	9/60/150	0.54	0.89	76.0	26	42	87.0	85.0
	9/150/219	0.71	2.73	67.0	24	92	92.0	92.0
	10/151/205	0.32	0.36	60.0	20	20	84.0	79.0
	D11	0.59	1.09	87.5	25	57	72.0	90.0
	D14	0.50	1.97	99.4	25	188	44.0	86.0
P1	Low Grade + 2600RL	0.37	0.47	88.6	30	36	89.0	83.9
P2	High Grade + 2600RL	1.00	1.10	83.6	28	29	95.6	89.5
P3	Low Grade 2,600-2400RL	0.29	0.69	80.0	25	60	84.8	86.4
P4	High Grade 2600-2400RL	0.77	2.13	84.8	25	70	95.7	94.6
P5	High Grade -2400RL	0.45	1.96	80.0	35	144	92.8	88.4

Note: All testwork conducted by Fox Anamet/Metcon

13.3.5 Optimisation Flotation Testwork

A number of tests were conducted to optimise grind characteristics, flotation reagents and other parameters. Two new sample composites were made up, representing high-grade and low-grade samples.

The reagent regime chosen comprised sodium ethyl xanthate (SEX) as the collector, with the pH adjusted to 10.5 using lime. BDA notes that collector S701 has been included in the most recent process flow diagrams.

A number of grind optimisation tests reviewed the effect of particle size distribution on flotation performance. Grind size distributions from a P80 of 53µm to a P80 of 212µm were tested. All results indicated the optimum grind size was P80 = 75µm, with performance generally deteriorating from 75µm to 212µm. An initial coarse grind and rougher float followed by a regrind was tested, but poorer gold recovery resulted compared with a P80 = 75µm primary grind. This result was confirmed by locked cycle tests that compared primary grind only with a coarser primary grind followed by concentrate regrind. The tests indicated that copper metallurgical performance is not significantly different but the gold recovery suffers significantly in the regrind case.

13.3.6 Ore Variability Tests

The test laboratory Optimet conducted standard flotation tests on a number of individual sample intersections. Five composites were chosen from individual 70m vertical sections, roughly equating to the first five years of plant operation. In addition, four individual rock type samples were tested. The results (Table 13.5) confirmed that a good flotation response could be expected, with copper recoveries in the mid 90s and a concentrate grade of 25-27% Cu. Gold recoveries ranged between 75-85%. The use of S701 as a replacement for the collector SEX was tested but was not successful as both copper and gold recoveries dropped.

Table 13.5: Variability Flotation Testwork Results

Test	Sample/Rock Type	Collector	Head Grade		Cu:S	Recovery at 27% Cu	
			Cu	Au	Ratio	Cu %	Au %
59	V01	SEX	0.74	0.38	1.14	84.0	76.0
60	V02	SEX	0.69	0.66	1.08	96.0	86.0
61	V03	SEX	0.42	0.61	0.86	94.2 (23.6% Cu)	84.8
62	V04	SEX	0.51	0.89	0.96	95.0	85.5
63	V05	SEX	0.46	1.17	0.94	90.5	80.0
64	V06 (Bugoy breccia)	SEX	0.72	3.61	0.96	96.0	81.0
65	V07 (Dark diorite)	SEX	0.17	0.66	1.00	87.0	84.0
66	V08 (Quan monzosyenite)	SEX	0.25	0.97	0.78	90.0 (21.7% Cu)	85.2
67	V09 (Tunja monzonite)	SEX	0.39	1.06	0.93	94.8 (22.9% Cu)	79.8
68	V02	SEX	0.68	0.66	1.05	96.0	89.0
69	V03	S701	0.42	0.50	0.84	86.0	85.5
70	V04	S701	0.48	0.75	0.92	82.5	68.5
71	V05	S701	0.42	1.19	0.89	90.0	68.0
72	V08 (Quan monzosyenite)	S701	0.25	1.20	0.81	76.0	56.0
73	V09 (Tunja monzonite)	S701	0.38	1.01	0.95	35.0	28.5

13.3.7 Confirmatory Tests

Four composites representing different depths within the proposed underground resource were tested with gravity separation prior to flotation. The results of these tests generally confirmed the conclusions from previous work:

- gravity gold recovery ranged from a low of 40% to a high of 70%;
- gravity recovery was directly related to gold head grade;
- copper recoveries were high, with reasonably good concentrate grades; and
- recleaning the concentrates generally reduces copper recovery and reduces gold recovery significantly.

13.3.8 Pilot Plant Tests

Two mini-pilot plant tests were undertaken. One sample was based on a large number of HQ quarter core intersections from a range of drill holes throughout the resource. The second sample used intersections from a single PQ drill hole (DDH 55).

The tests incorporated some bench-scale tests to determine the suitability of each composite to the Didipio flow sheet. Comminution characteristics were also determined. The overall metallurgical response was similar to previous tests and the samples were deemed suitable for pilot testing.

The pilot comminution and classification circuit was set to target a flotation feed particle size distribution of P80 = 106µm. Power consumptions indicated indices of 12.4-13.9kWh/t, similar to the expected index range.

The pilot tests were generally based on a feed rate of about 150kg/hour and each test ran for 6-7 hours. The flow sheet incorporated a flash flotation step after the ball mill prior to classification. After classification, a set of rougher-scavenger flotation cells was used with two stages of cleaning. Several difficulties were experienced with the pilot trials leading to problems in calculating the recoveries. The test results were reported using three means of calculating recoveries. The first method utilised the standard two-product method based on copper assays and, as such, assumes that the circuit was operating in steady-state. The second method used gold assays and the two-product method for gold recovery only. The third method utilised calculations based on the actual test product weights and assays. Table 3.6 summarises these results.

Table 13.6: Pilot Plant Testwork Results

Trial	Head Grade		Concentrate			Copper Recovery %		Gold Recovery %		
			Type	Grade		On Wt	On Cu Assay	On Wt	On Cu Assay	On Au Assay
	% Cu	g/t Au		% Cu	g/t Au					
1	0.592	0.848	Recleaner	24.2	17.0	66.4	96.4	45.1	65.6	81.2
			Batch	24.6	31.2		96.5		85.4	86.0
2	0.592	0.848	Flash	22.3	27.5	80.9	82.6	82.7	84.4	66.9
			Recleaner	12.2	15.8	6.1	14.6	6.5	15.6	20.1
			Total	21.0	26.0	87.0	97.2	89.2	100.0	87.0
3	0.424	0.957	Flash	22.3	42.8	75.0	80.6	75.6	81.2	68.6
			Recleaner	14.5	23.4	18.4	15.1	15.6	12.8	18.7
			Total	20.1	37.5	93.4	95.7	91.2	94.0	87.3
3	0.424	0.957	Recleaner	25.6	24.3	83.9	94.6	78.2	88.1	75.3
			Batch	27.9	57.2		95.1		86.4	90.2

While the results are varied, overall the pilot plant performance reported good copper and gold recoveries and the concentrate grades were adequate and could be considered saleable. Analyses of the concentrates indicated no deleterious elements that would incur any smelter penalties.

The inclusion of flash flotation during the pilot plant tests resulted from a recommendation from flotation cell supplier Outokumpu (OKU). Minproc suggests that a flash flotation cell on the cyclone underflow could be expected to recover about 50% of the copper and 40% of the gold to a coarse concentrate.

Tailings samples from the pilot plant products were tested by OKU to determine pulp settling characteristics. Underflow densities of 56-62% solids by weight were indicated. Samples from the thickener tests were examined by an Australian consultant, Slurry Systems Proprietary Limited, to determine the pulp viscosity and rheology characteristics. The conclusion was that the tailings should not present any major pumping problems.

Concentrate samples were tested by OKU to assist in thickener design. OKU advised that underflow densities of about 50% solids could be achieved with a feed density of 20% solids. A filtration vendor, Larox, tested some of the pilot plant concentrates to determine filtration characteristics with a design transportable moisture limit of 9%.

13.3.9 2006 Confirmatory Tests

A confirmatory testwork programme was carried out on new samples. Batch tests were conducted as well as locked cycle tests. Gravity gold was removed prior to flotation testwork. Table 13.7 summarises the results, which are generally consistent with the results from the early programmes.

Table 13.7: 2006 Confirmatory Testwork Results

Sample	Head Grade		Concentrate			Recovery %			
	% Cu	g/t Au	Type	Grade		Copper	Gold		
				% Cu	g/t Au		Total	Gravity	Flotation
LS0001	1.00	2.12	Locked Cycle	23.6	22.8	95.6	90.0	43.5	46.5
			Batch	28.4	16.7	94.3	88.1	39.5	48.6
LS0002	1.09	2.40	Locked Cycle	26.5	23.1	94.8	91.2	49.4	41.8
			Batch	28.5	24.2	93.6	91.8	51.0	40.8
LS0003	0.81	1.17	Locked Cycle	29.2	17.6	95.9	92.9	46.6	46.3
			Batch	26.5	23.2	95.7	90.5	41.0	49.5

13.4 Summary – Metallurgical Performance

In the 1998 Feasibility Study report, Minproc summarised the metallurgical performance from testwork and projected the likely operating plant performance. The parameters and key process criteria are shown in Table 13.8.

Table 13.8: Minproc Proposed Metallurgical Criteria

Parameter	Units	Value
Plant throughput	Mtpa	2.5
Plant operating time	hours	8,000 – 8,300
Plant overall utilisation	%	91.3 – 94.8
Work index	kWh/t	14.5
Gravity gold recovery	%	40
Overall gold recovery	%	75 – 95
Copper concentrate grade	% Cu	26 – 30
Copper recovery to concentrate	%	94.8

Ausenco has considered the testwork results and has created algorithms to determine the gravity and flotation gold recoveries as well as copper recoveries for use in plant design.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The Didipio Gold-Copper Deposit resource estimate was updated by OGC in February 2011 in order to reduce the block size from 15mE x 15mN x 20mRL to 15mE x 15mN x 10mRL, to be more in line with the intended five to ten metre flitch height. The model was also translated from the UTM National Grid to a truncated version of it, called the Project grid.

Prior to this, in October 2008, the Didipio Gold-Copper Deposit resource estimate was revised by OGC to accommodate 21 additional infill drill holes.

In March 2007, an estimate was completed by Hellman and Schofield as part of the initial OceanaGold NI 43-101 lodging.

Despite the changes between 2010 and 2011, the estimates provide no significant change in resources (see Table 14.7).

14.2 Geological Interpretation

As discussed in section 0, the following geological entities have been interpreted: the Dark Diorite, Tunja monzonite, Bugoy breccia, Bufu syenite, Biak Shear, Lanut Dyke and Two Dogs Fault. Of these, the Biak Shear was considered to represent the only “hard” grade boundary. All other geological contacts, while contributing to the distribution of mineralisation, are not considered to represent hard grade boundaries. The Dark Diorite, Tunja, Bufu and Bugoy have, however, been modelled as distinct SG domains. Most of the economic mineralisation is centred on the Tunja monzodiorite, with some peripheral mineralisation within the Dark Diorite.

14.3 Data Used for Estimation

Drilling is generally directed at dips between -45 and -75 towards 215°. Drill holes are centred on approximate 50m sections, but in some areas drilling has been filled in to 25m. Vertical spacing is typically around 50m in the higher-grade area above the Bufu syenite, but further to the south-east vertical spacing of 100-150m is more usual.

Ninety-eight diamond drill holes at Didipio were used for grade estimation, while all trench and tunnel samples were omitted. The trenches and tunnels were not used because their location in space could not be reconciled with current topography (trenches commonly 10-20m below topography, but reported to be only 1-2m in depth) and the tunnels commonly follow high-grade structures (and may therefore not be representative).

All samples were composited to nominal 3.0m intervals.

14.4 Data Analysis

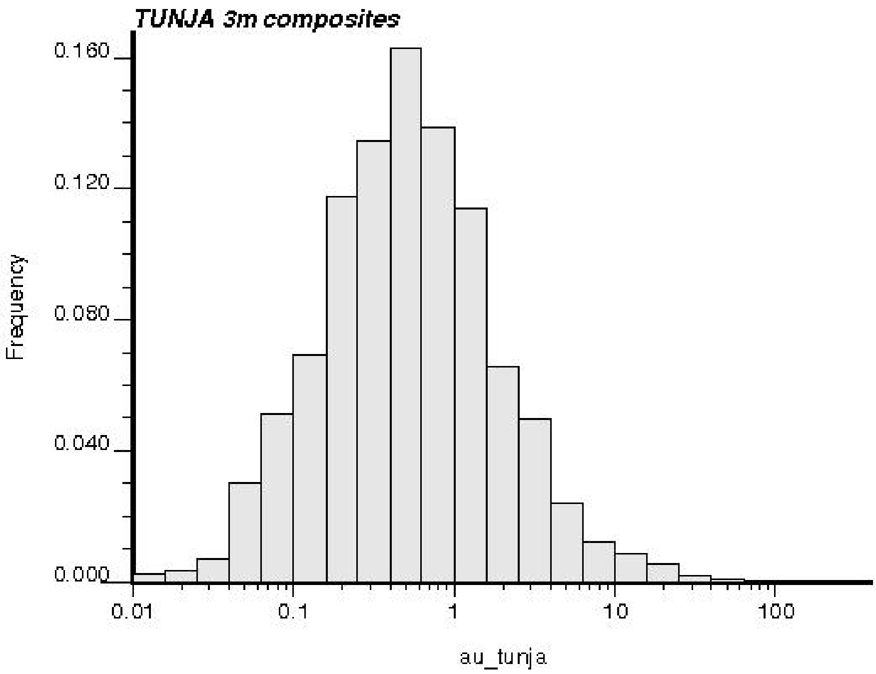
Table 14.1 shows both gold and copper grade statistics by geological domain.

Table 14.1: 3m Composited Drill Hole Sample Gold and Copper Summary Statistics by Lithology

	Tunja	Bufu	Biak	Dark Diorite
No. Samples	5,377	221	983	3,076
GOLD				
Mean	1.24	0.73	0.23	0.30
Median	0.50	0.37	0.05	0.12
Maximum	57.5	16.4	16.70	11.90
CV	2.33	2.06	3.10	2.20
COPPER				
Mean	0.45	0.13	0.08	0.18
Median	0.30	0.08	0.03	0.08
Maximum	6.94	2.88	1.01	3.28
CV	1.08	1.76	1.61	1.47

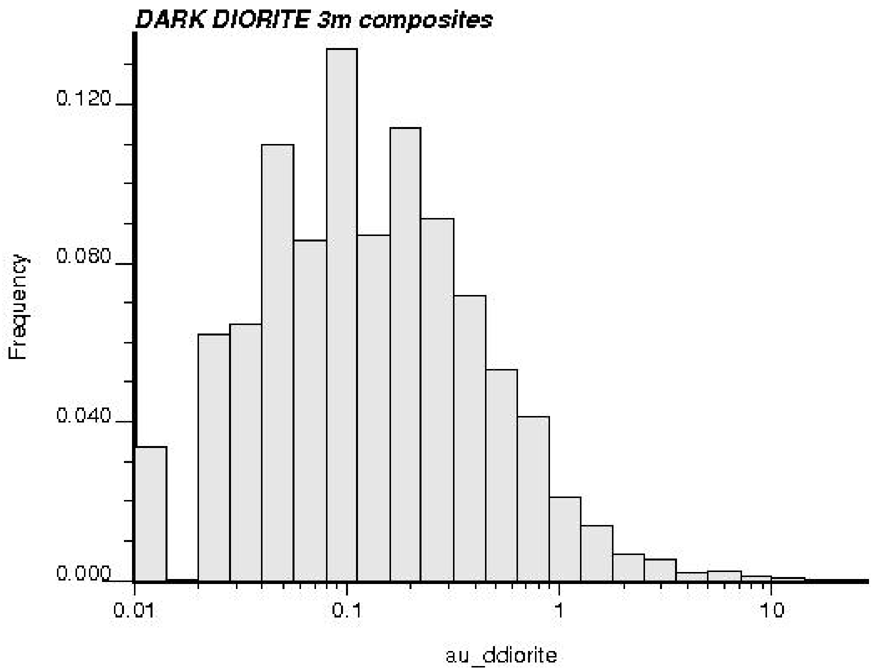
The histogram for gold grade within the Tunja monzodiorite (Figure 14.1) shows a single, approximately log-normal distribution, with perhaps a very small high-grade population. Fifty per cent of the total gold is contained within the highest 8% of the gold grades.

Figure 14.1: Histogram of 3m Composited Gold Grades for Tunja



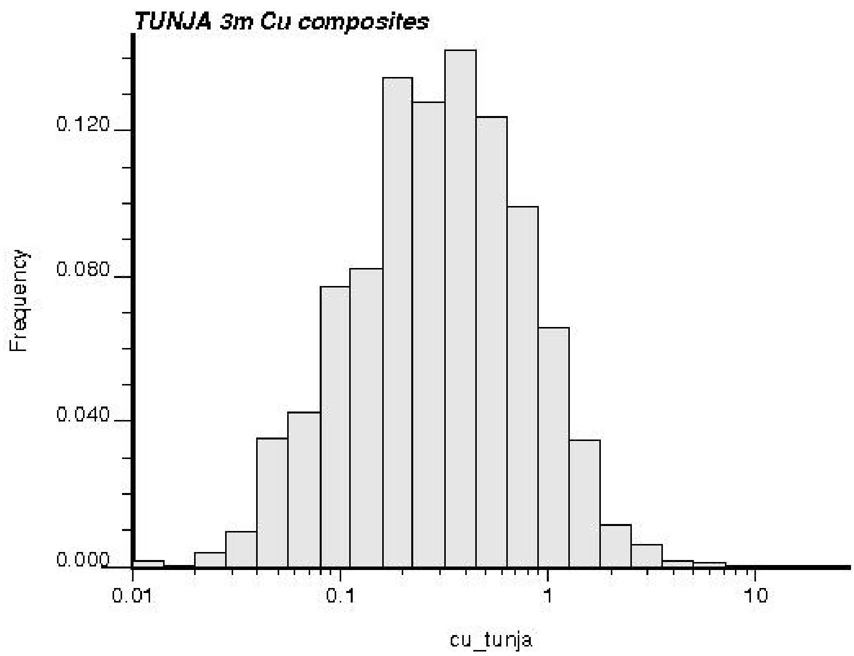
The histogram for gold grades within the Dark Diorite (Figure 14.2) shows a single, approximately log-normal distribution. Fifty per cent of the total gold is contained within the highest 9% of the gold grades.

Figure 14.2: Histogram of 3m Composited Gold Grades for Dark Diorite



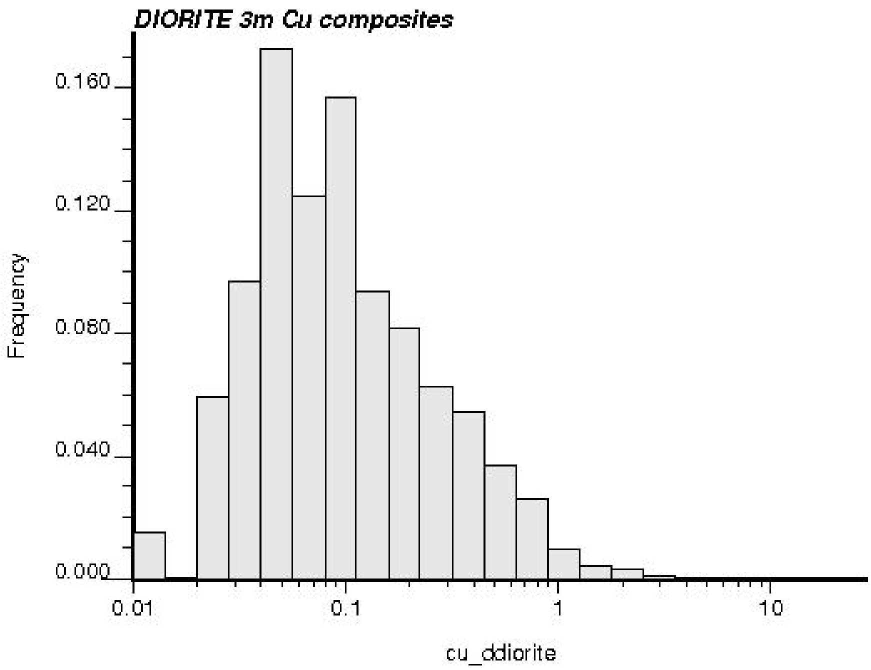
The histogram for copper within the Tunja monzodiorite shows a single approximately log-normal distribution (Figure 14.3). The copper distribution is less skewed than the gold distribution, with 50% of the total metal in 19% of the samples.

Figure 14.3: Histogram of 3m Composited Copper Grades for Tunja Monzodiorite



The histogram for copper within the Dark Diorite shows a single approximately log-normal distribution (Figure 14.4). The copper distribution is less skewed than the gold distribution, with 50% of the total metal in 13% of the samples.

Figure 14.4: Histogram of 3m Composited Copper Grades for Tunja Dark Diorite



The available specific gravity data was found as 10m intervals, although the actual measurements were performed on 10cm pieces of core approximately every 10m. These are summarised in Table 14.2.

Table 14.2: Statistics for Specific Gravity Data by Rock Type

	Oxide	Trans	Tunja	Bufu	Biak	D diorite	Breccia
No. samples	31	NA	474	17	86	558	7
Mean	2.42	NA	2.51	2.39	2.66	2.73	2.56
Median	2.35	NA	2.52	2.37	2.72	2.75	2.57
Mean minus extremes*	2.51	NA	2.51	2.40	2.67	2.73	NA
Minimum	2.09	NA	2.09	2.01	2.08	2.00	2.54
Maximum	3.03	NA	3.18	2.66	3.11	3.50	2.58
Value used	2.20	2.40	2.50	2.35	2.67	2.72	2.45

* Mean excluding values outside 2.5% and 97.5% quantiles

The specific gravity value was assigned to all model blocks for each respective geological domain.

A brecciated zone has been logged/interpreted above the Bufu syenite. This broadly equates to a unit previously termed the Bugoy breccia – typically associated with high-grade gold mineralisation. Although this was not modelled as a discrete grade domain, it was assigned a slightly lower SG than the surrounding Tunja host rock.

There were insufficient density data available for analysis of the oxide and transition zones, so historical average values of 2.2 t/m^3 and 2.4 t/m^3, respectively, were used.

14.5 Variography

Spatial analysis of grades (variography) commenced with variogram maps to determine the principal directions of continuity. Both gold and copper show a strike slightly west of north and a steep easterly dip, consistent with the observed geology (Figure 14.5 and Figure 14.6). Variogram maps in the plane of mineralisation (approximately N-S) are fairly isotropic, suggesting no significant plunge component to the mineralisation.

Figure 14.5: Variogram Maps for Gold (LHS=Plan, RHS=E-W Section)

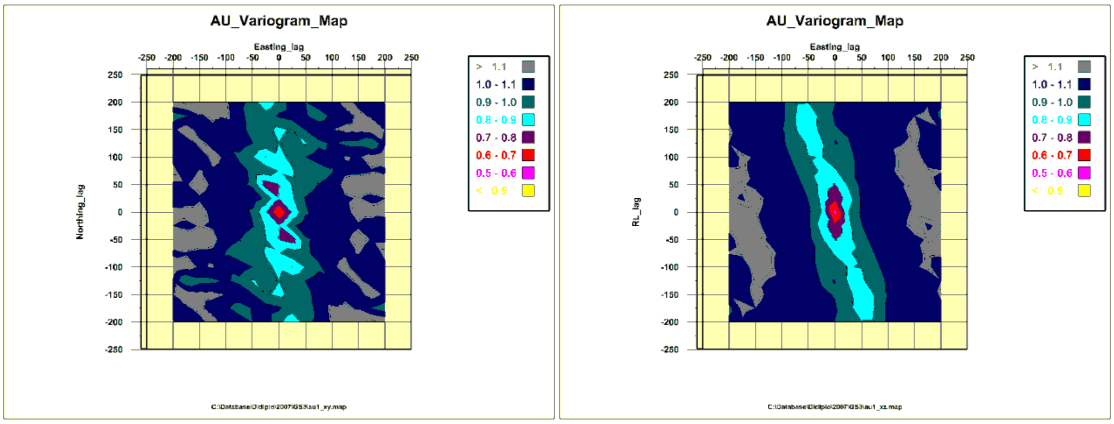
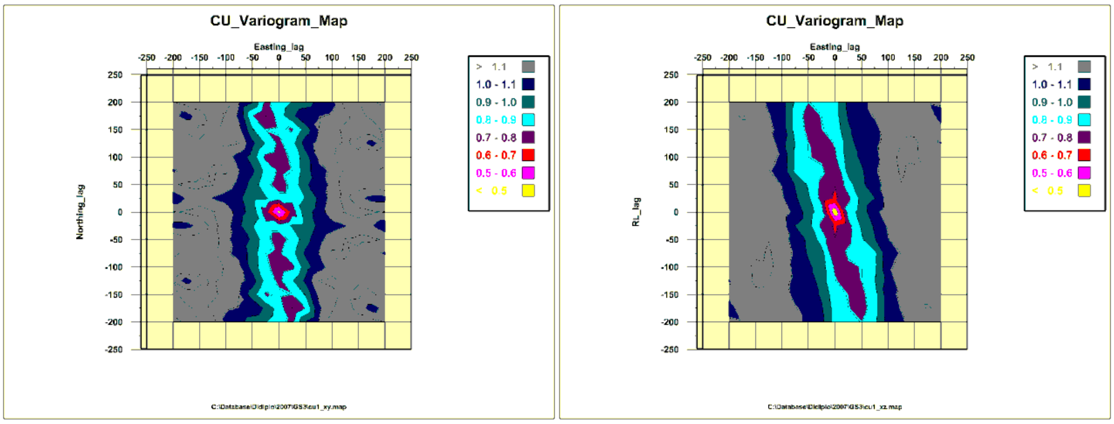


Figure 14.6: Variogram Maps for Copper (LHS=Plan, RHS=E-W Section)



14.6 Resource Estimation

Ordinary kriging was considered the appropriate estimation method for gold and copper because these elements have moderate coefficients of variation and their grade distributions are reasonably smooth and gradational, i.e. there is generally a smooth gradation from high to low grades.

There was insufficient data in the oxide and transition zones to determine whether these zones are enriched or depleted in gold or copper. Therefore, no boundaries were used between primary, transition and oxide mineralisation during grade estimation (note that all oxide mineralisation has been classified as Inferred).

Grade top cutting was not used, although the influence of DDDH83 (the most intensely mineralised drill hole in the estimate) was reduced. Furthermore, as DDDH83 sits in close proximity to the interpreted Biak Shear plane, mineralisation further than approximately 20m to the north of DDDH83 was demoted to Inferred classification.

14.7 Resource Classification

The resource model for Didipio Deposit has been classified to JORC and CIM standards. The resource classification is based on the estimation searches. These define the base classification, to which the following modifications were made:

- All resource within the Biak Shear and within 10m of the interpreted southern plane was classified as Inferred. This primarily reflects uncertainty in the geological interpretation.
- All oxide resource was classified as Inferred. Current metallurgical test work suggests that little copper will be recovered from oxide material. Furthermore, due to topographic/land access restrictions, much of the near-surface resource is sparsely drilled. Inferred classification was felt to reflect the limited drilling and sole dependence on gold mineralisation for the oxide zone. It is very likely that prudent grade control strategies will define considerable gold resource within the oxide zone.
- The classification of measured resource was based both on search criteria and three-dimensional geometry. These criteria ensure that all measured resource has data falling within both hemispheres of the search.

14.8 Mineral Resources

The mineral resources quoted here include the mineral reserves described in this report. These mineral resources were prepared by Jonathan Moore, Group Mine Geology Manager for OGC.

The resource estimate is divided into two zones for reporting purposes: an open-cut resource that includes all material above an elevation of 2,390mRL (base of the updated open-pit design); and an underground resource between 2,390 and 2,180mRL (vertical extent of the underground designs). The open-cut resource uses a 0.4g/t eqAu cut-off grade, while the underground resource uses a 1.5g/t eqAu cut-off grade. The equation for contained gold equivalent is g/t eqAu = g/t Au + 2.06 x % Cu, based on metal prices of US$950 per ounce for gold and US$2.85 per pound for copper. This contained gold equivalence does not account for metallurgical recoveries. The open-cut, underground and combined resource estimates are presented in table 14.3, Table 14.4 and Table 14.5 classified using CIM guidelines. All mineral reserves reported are included within the mineral resources reported for the same deposit.

The estimate of Measured and Indicated Mineral Resources has increased by 0.13 Moz of gold and 0.02 Mt of copper compared to the Company"s most recent resource/reserve update as at December 31, 2010, while the estimate of Inferred Mineral Resources has increased by 0.13 Moz of gold and 0.02 Mt of copper compared to the December 31, 2010 update. These increases are due to the lowering of the open pit / underground resource reporting boundary (from 2,540mRL to 2,390mRL) to the base of the expanded open pit. This has resulted in a greater proportion of the total resource being reported at the 0.4 g/t eqAu open pit cut-off.

Table 14.3: Open cut Resource Estimate

Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (kt)
Measured	15.10	1.59	0.56	0.77	84.7
Indicated	46.38	0.53	0.36	0.79	164.7
Measured & Indicated	**61.48**	**0.79**	**0.41**	**1.56**	**249.3**
Inferred	28.18	0.35	0.22	0.32	62.8

(above 2,390mRL at 0.4 g/t eqAu cut-off grade)

Table 14.4: Underground Resource Estimate

Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (kt)
Measured	0.87	3.22	0.61	0.09	5.3
Indicated	7.83	1.91	0.47	0.48	36.4
Measured & Indicated	**8.69**	**2.04**	**0.48**	**0.57**	**41.7**
Inferred	2.55	1.46	0.37	0.12	9.3

(between 2,390mRL and 2,180mRL at 1.5 g/t eqAu cut-off grade)

Table 14.5: Combined Resource Estimate

Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (kt)
Measured	15.96	1.67	0.56	0.86	90.0
Indicated	54.21	0.73	0.37	1.27	201.1
Measured & Indicated	**70.17**	**0.95**	**0.41**	**2.13**	**291.0**
Inferred	30.73	0.44	0.23	0.44	72.1

(at 0.4 g/t eqAu cut-off grade above 2,390mRL and at 1.5 g/t eqAu cut-off grade below 2,390mRL)

All mineral reserves reported are included within the mineral resources reported for the same deposit.

The resource is tabulated below in Table 14.6 according to material type.

Table 14.6: Resource estimate by material type

(at 0.4 g/t eqAu cut-off grade above 2,390mRL and at 1.0 g/t eqAu cut-off grade below 2,390mRL)

OXIDE	Mt	Au g/t	Cu %	Au Moz	Cu Kt
MEASURED	0.01	0.69	0.93	0.00	0.1
INDICATED	0.12	0.40	0.51	0.00	0.6
MEASURED & INDICATED	**0.13**	**0.41**	**0.54**	**0.00**	**0.7**
INFERRED	4.05	0.38	0.47	0.05	19.1

TRANSITIONAL	Mt	Au g/t	Cu %	Au Moz	Cu Kt
MEASURED	0.38	0.71	0.80	0.01	3.0
INDICATED	0.97	0.35	0.58	0.01	5.6
MEASURED & INDICATED	**1.35**	**0.45**	**0.64**	**0.02**	**8.7**
INFERRED	0.56	0.37	0.34	0.01	1.9

SULPHIDE	Mt	Au g/t	Cu %	Au Moz	Cu Kt
MEASURED	15.58	1.70	0.56	0.85	86.9
INDICATED	53.12	0.74	0.37	1.26	194.8
MEASURED & INDICATED	**68.70**	**0.96**	**0.41**	**2.11**	**281.7**
INFERRED	26.13	0.45	0.20	0.38	51.2

TOTAL	Mt	Au g/t	Cu %	Au Moz	Cu Kt
MEASURED	15.96	1.67	0.56	0.86	90.0
INDICATED	54.21	0.73	0.37	1.27	201.1
MEASURED & INDICATED	**70.17**	**0.95**	**0.41**	**2.13**	**291.0**
INFERRED	30.73	0.44	0.23	0.44	72.1

14.9 Model Validation

Two sensitivity models were constructed as cross-checks of the ordinary kriged model. Both models provided similar estimates to the ordinary kriged model.

The block model grades were compared with nearby drill hole grades visually on screen in Minesight. Bench by bench comparisons of modelled grade versus composite grades were also made. These are presented in Figure 14.7 and Figure 14.8 for both copper and gold.

Figure 14.7 shows reasonable overall agreement between the modelled and composited grades, although some differences occur between the 2300mRL and 2450mRL where high-grade mineralisation is developed above the Bufu syenite – the volume in which the influence of DDDH83 has been mitigated. In this volume, the modelled grades are significantly lower than the mean composited grade. Further infill drilling in proximity to DDDH83 would resolve this disparity.

Figure 14.7: Resource Model (Measured and Indicated) versus Sample Gold Grades by Bench

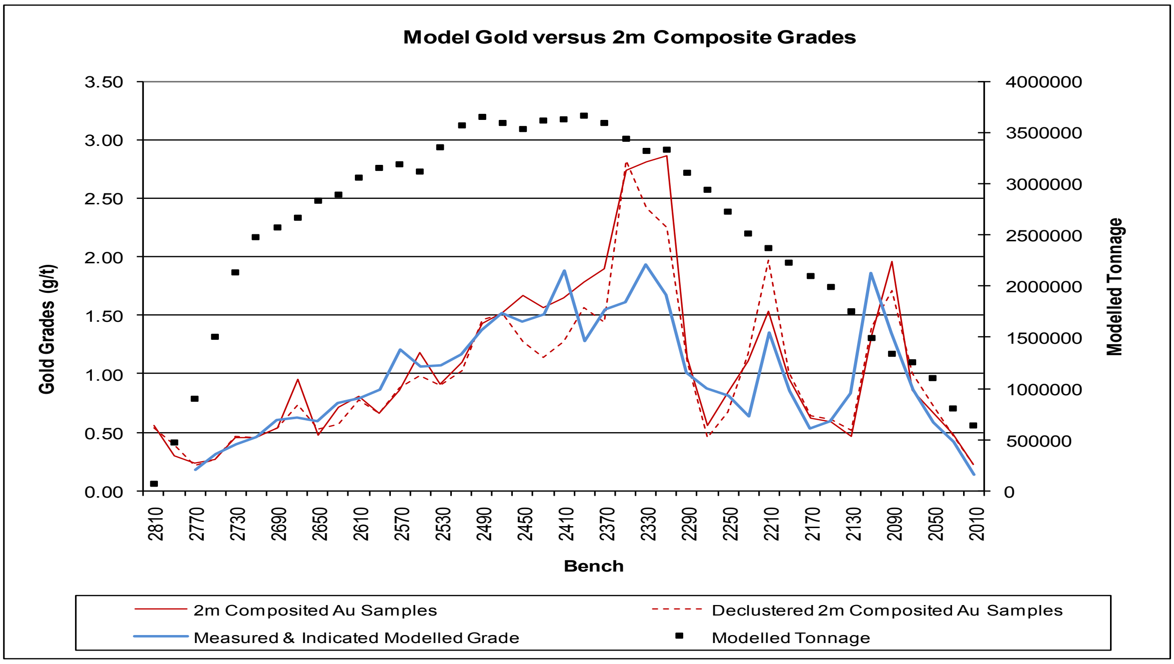
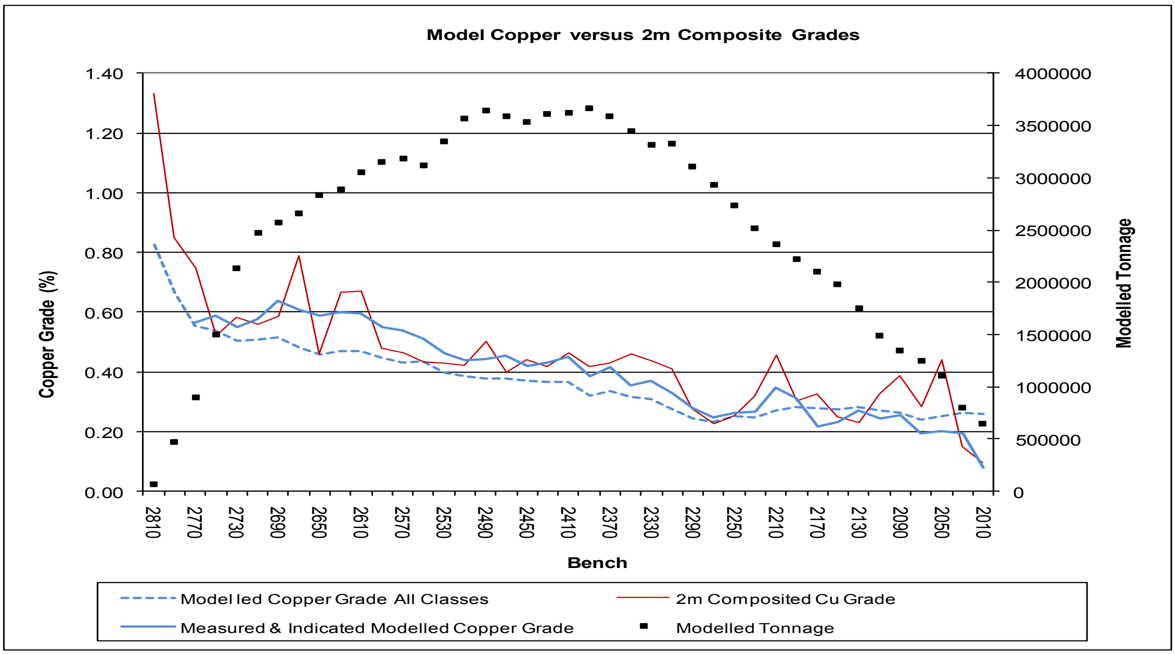


Figure 14.8 shows reasonable overall agreement between the modelled and composited grades and is not compounded by the grade spike above the Bufu syenite, seen in the gold comparison.

Figure 14.8: Model versus Sample Copper Sample Grades by Bench



The current resource estimate is compared with the previous, estimate in Table 14.7. The two estimates are reasonably close for all classes.

For the purposes of this comparison, both estimates have been tabulated using the gold equivalence and underground/open pit transition used in the 2010 report. The previous economic parameters of US$800/ounce gold and US$2.60/pound copper led to the gold equivalence of eqAu = Au (g/t) + 2.23 x Cu (%). The underground/open pit transition was then 2,540mRL.

Table 14.7: Comparison of 2010 and Current Resource Estimates, using 2010 Economic Parameters

Model	Class	Tonnes (Mt)	Au (g/t)	Cu (%)
OGC 2011	Measured	15.64	1.70	0.57
	Indicated	44.56	0.78	0.41
	Measured & Indicated	**60.20**	**1.02**	**0.45**
	Inferred	22.0	0.42	0.26
OGC 2010	Measured	15.58	1.72	0.57
	Indicated	44.49	0.80	0.41
	Measured & Indicated	**60.07**	**1.04**	**0.45**
	Inferred	21.15	0.45	0.26

(at 0.4 g/t eqAu cut-off grade above 2,540mRL and 1.0 g/t eqAu below)

15 MINERAL RESERVE ESTIMATE

15.1 Reporting Standard

The reserves were compiled with reference to the Canadian National Instrument 43-101 (NI 43-101). This section summarises the main reserves considerations and provides references to the sections of the study where more detailed discussions of particular aspects are covered.

The equation for contained gold equivalent is g/t eqAu = g/t Au + 2.06 x % Cu, based on metal prices of US$950 per ounce for gold and US$2.85 per pound for copper.

15.2 Open Cut Reserves

15.2.1 Ore Loss and Dilution

A dilution procedure was applied to the open cut resource block model which smears adjacent blocks together to simulate the effects of movement across defined ore boundaries during blasting and mining. However, because the ore zones are so broad on each bench the overall dilution edge effects were minimal and there was little difference between the overall in situ and diluted tonnes and grades as a result no dilution was applied to the block model during optimisation.

No mining losses were applied. It was considered that the resource estimation technique applied to the broad ore zones provides an adequate estimate of the run of mine (ROM) tonnes and grades.

15.2.2 Cut-Off Grade

Both gold and copper contribute significantly to the value of each block, particularly in the open cut where the copper grades are higher. In order to express a cut-off grade which accounts for the value of both metals, a net metal value (NMV) was calculated for each resource block. This calculation applies process recoveries and smelter and refinery returns to each metal then multiplies by the price per unit for each metal to determine the payable value of metal in each tonne for the block. It then deducts the realisation costs (concentrate transport and smelter and refinery charges) expressed as $ per tonne of ore to determine the NMV per tonne for the block.

For the open cut, if the NMV is greater than or equal to the combined processing and site general and administration (G&A) cost per tonne of ore, then the block is classed as ore. If the NMV is less than this combined cost the block is classed as waste.

This marginal cut-off grade methodology determines which blocks should be targeted in the mine design. Once the pit design is prepared all blocks above the open cut NMV cut-off that lie within the pit shell are reported as ore.

Metal prices used in the NMV calculation were US$950/oz for gold and US$2.85/lb for copper.

The estimates of process and site fixed costs used for this analysis are:

Process	US$9.58/t
Site G&A	US$3.46/t
Total	US$13.04/t

Analysis of the open cut NMV grade distribution shows several million tonnes of ore at just above the marginal cut off NMV of US$13.04 per tonne, which has a small average profit margin. If all this material is included in the reserve then one of two scenarios arises. Either:

- If ore is processed at the average grade mined each month the project remains profitable but at a low net monthly cash flow over a longer period; or
- If an elevated cut-off grade is applied to generate higher early cash flows, large low-grade stockpiles are accumulated to be processed during prestripping of subsequent pit stages.

Experience from earlier studies shows that the contribution of higher early cash flows to present value of the project outweighs the effect of costs brought forward by mining faster to maintain 2.5 Mtpa ramping up to 3.5Mtpa in year 3 of high-grade feed. In order to achieve this, an elevated cut-off strategy was utilised in the first 4 years of the project.

15.2.3 Open Cut Reserves

Using a cut-off NMV of US$13.04 per tonne and a pit base at RL2380, the Didipio Project open cut reserves are 44.70 Mt at 0.88 g/t Au and 0.46% Cu.

Table 15.1: Open Cut Ore Reserves by Grade Range

Ore Type	Mt	Au g/t	Cu %
Ore Grade	20.25	1.45	0.57%
Low Grade	24.49	0.40	0.37%
Total	**44.74**	**0.88**	**0.46%**

The average ratio of the open cut is 3.41 bcm of waste for each bcm of ore.

15.2.4 Open Cut Reserves Categories

The open cut reserves are derived from the Measured and Indicated Mineral Resource blocks in the resource model. Proven Mineral Reserves are taken from Measured Resources and Probable Mineral Reserves are taken from Indicated Resources. No extraordinary risk factors were identified to warrant downgrading of the open cut reserve categories in the resource to reserve conversion.

15.3 Underground Reserves

15.3.1 Ore Recovery and Dilution

The underground mine plan is based on sublevel open stoping (SLOS). The stopes are relatively small at 20 x 20 metres in plan area and have heights of either 30 or 60 metres depending on the ground conditions. They are arranged in a “chequerboard” pattern to extract the broad ore zones. Cemented fill is placed as each stope is completed to allow extraction of the adjacent stopes without creating unstable spans. The stoping sequence occurs in three passes (primary, secondary and tertiary) so that the current production stope is always surrounding on four sides by either unmined or filled ground. Loss and dilution factors were applied as follows in Table 15.2.

Table 15.2: Stope Loss and Dilution

	Dilution		Recovery	
	30m stopes	**60m stopes**	**30m stopes**	**60m stopes**
In ore development headings	0.0%	0.0%	100%	100%
Unmined ground on all sides	5.0%	2.5%	96%	98%
Stope fill on one side	5.0%	5.0%	98%	98%
Stope fill on two sides	7.0%	7.0%	98%	98%
Stope fill on three sides	10.0%	10.0%	95%	95%
Stope fill on four sides	12.0%	12.0%	90%	90%
Small isolated bench stopes	10.0%		85%	

Dilution grades were set at 0.65 g/t Au and 0.25% Cu based on the estimated average of surrounding in situ material and rock and tailings material in the fill.

The underground loss/dilution model resulted in a reduction of 2.1% of the tonnes, 6.4% of the contained gold and 6.2% of the contained copper compared to the in situ stopes.

15.3.2 Cut-Off Grade

The underground cut-off grade is calculated by dividing the projects total life of mine cost (inclusive of re-handling to ROM by the open pit fleet, processing and site over-heads) by the gold price per gram of US$30.54/g = US$950/oz.

In ore development and production	US$36.14 per tonne
Re-handling to ROM by open pit fleet	US$ 0.52 per tonne
ROM Loader	US$ 0.23 per tonne
Processing	US$10.50 per tonne
Site fixed costs	US$ 3.00 per tonne
Total	US$50.39 per tonne

At US$950/oz the cut-off grade is 1.65 g/t AuEq.

The underground cut off grade was established on a common 20 m by 20 m by 30 m SLOS geometry mine design that targeted effective extraction of material within a better than 1.8 g/t AuEq grade shell. The grade shells are irregular so to achieve mineable stope geometries for a regular plausible layout it is necessary that the stopes transition the grade shell boundary. In effect the mineable stope geometry excavates a significant portion of mineralisation above 1.8 g/t AuEq and the remainder will be of mineralisation below 1.8 g/t AuEq.

The use of a 1.8 g/t AuEq grade shell as opposed to a 1.65 g/t AuEq grade shell is to allow a contingency against the dilution affect by the mineralization of less than 1.8 g/t AuEq. The stope grades as delivered to the surface portal stockpile allow for ore loss and dilution factors as a result of position in extraction sequence. The inclusion of a stope and thus its specific development within the overall design is based on the delivered factored stope geometry having an AuEq value of better than 1.65 g/t.

15.3.3 Underground Reserve Categories

The underground reserves are derived from the Measured and Indicated Mineral Resource blocks in the resource model. Proven Mineral Reserves are taken from Measured Mineral Resources and Probable Reserves are taken from Indicated Resources.

A small amount of underground low grade "mineralised waste" is included in ore inventory as it is necessary underground development material that must come to surface, at which point it is economically attractive to treat it rather than send it to waste based on processing, overheads and concentrate costs.

15.4 Total Reserves

15.4.1 Reserves as at June 2011

Table 15.3: Ore Reserves

Source	Reserve Class	Tonnes	Au (g/t)	Cu (%)	Gold (Moz)	Copper (kt)
Open Pit	Proven	13,790,000	1.60	0.59	0.71	81
	Probable	30,950,000	0.55	0.39	0.55	121
Underground	Probable	5,910,000	2.25	0.45	0.43	27
Total Proven		13,790,000	1.60	0.59	0.71	81
Total Probable		36,860,000	0.82	0.40	0.97	148
Total Proven and Probable		50,650,000	1.03	0.45	1.68	229

notes: Reserves are based on the following metal price assumptions. $950/Ounce Au and $2.85/lb Cu. Using a copper to gold equivalance factor of Au (g/t) eq = ***2.08*** *X Cu (%), the Cut-off grade for the open pit reserve is 0.5g/t* ***AuEq*** *and for the underground 1.9g/t* ***AuEq.***

Circa 400kt @ 0.67g/t Au and 0.24% Cu of underground low grade "mineralised waste" is part of the ore inventory as it is necessary underground development material that must come to surface, at which point it is economically attractive to treat it rather than send it to waste based on processing, overheads and concentrate costs.

15.5 Sensitivity of Reserves

The Reserves in this section are most sensitive to the following:

- Operating costs. In particular the Diesel price has a significant impact on the project due to on site power generation for all needs.
- Final permits for the large open pit have as yet not been received, though the relevant applications are being processed.

16 MINING METHODS

16.1 Mining Operations

16.1.1 Project Description

The Didipio Valley is located in the Philippines on Luzon in mountainous terrain 270km north-east of Manila.

The Didipio orebody is the highest grade and best defined of several zones of gold-copper mineralisation at Didipio. The deposit is an elliptical shape in plan and has a near-vertical plunge. At its widest it is 400m north-south and 200m east-west. The surface expression of the deposit is a steep-sided hill that rises 100m above the valley floor and merges into a much higher ridge to the south. The deposit extends to at least 700m below the valley floor.

16.2 Deposit

Gold and copper grades are zoned from a high-grade core outwards to a lower-grade halo. The high-grade core reaches widths of 250m north-south by 100m east-west, but breaks into a number of narrower zones below 400m in depth. Gold grades within the core tend to increase with depth, whereas copper grades tend to decrease with depth.

16.3 Resource Model

16.3.1 Resource Block Model

The resource block model was prepared by Jonathan Moore of OGC. It uses exploration data gathered since 1992 and replaces the earlier models prepared for Climax Mining Limited and, more recently, Hellman & Schofield (H&S). The resource is an ordinary kriged block model for gold and copper with domaining of oxide and primary material. AMDAD imported the OGC Datamine model to Surpac and ran a series of check reports to verify that the Surpac model matches the original Datamine version. Mr. Moore has since updated the resource model in 2011 to convert to a UTM grid and a 10m block size for current work.

16.4 Topography Model

Topography over the mine and tailings dam areas uses a ground survey completed in early 2008. The topographic coverage was extended by digitising 20-metre contours from the 1:50,000 regional map. Although the extended topography is less accurate than the ground survey, it fitted well along the join of the two datasets and its only design use is to determine waste dump volumes over part of the waste dump area.

16.5 Net Metal Value Calculation

Extensive variability testing on metallurgical recovery resulted in empirically derived formulae for recovery of gold to the gravity circuit and to concentrate and copper to concentrate. These formulae are functions of copper and gold head grades and vary from oxide to primary ore. When the concentrate weight recovery and transport and smelting costs are also considered it becomes impossible to define a simple metal equivalent relationship between gold and copper. Since both metals contribute significantly to overall value, especially in the upper open cut zone, it was decided that ore should be defined by the estimated net metal value (NMV) of each tonne of material rather than by a simple gold or copper equivalence.

NMV for Didipio is determined by:

- Applying the empirical formulae for gravity gold recovery and gold and copper recovery to flotation concentrate in the oxide and primary zones to estimate the recoverable metal per tonne. The empirical formulae also include estimation of tonnes of concentrate per tonne of ore.
- Multiplying the recoverable gold and copper by the nominated prices to estimate gross revenue per tonne.
- Subtracting ex-mine costs relating to transport of dóre and concentrate, smelting of concentrate and refining of gold and copper.

The resulting NMV is compared against the combined mine site cost per tonne of ore processing and site general and administration cost. The cut-off grade is the NMV, which is equal to the combined site processing and general and administration cost. Mined tonnes with NMV values greater than this combined cost will generate a profit and are classed as ore. Lower NMV values would create a loss if processed and so are classed as waste.

For open-cut mining the site costs only include processing and general and administration because at the time ore/waste definition is done the material is already exposed on the mining bench and the assumption is made that it would cost the same to mine the material to the ROM pad or to the waste dump.

For underground mining, ore production costs (development in ore and production drilling, blasting, mining and haulage) are added also because these costs will only be incurred on tonnes within stopes defined by the NMV cut-off grade.

16.6 Optimisation Inputs

16.6.1 Mining Costs

Based on a detailed tender submitted by contractors, with explosives quoted by Orica, and diesel supplied by OGC, average mining cost of the optimised shell was US$2.42/tonne ore and waste.

16.6.2 Mill Feed Rate

In November 2010, optimisations were run at 2.5, 3.5 and 5.0 Mtpa. The shell to guide the design was taken from the 2.5 Mtpa ramping up to 3.5 Mtpa by year 3 run.

Current mine plan is for 3.5 Mtpa during the open cut phase.

16.6.3 Processing Costs

Based on 2007 estimate for 2.0 Mtpa factored up by 20% to US$9.58/tonne of ore.

Current financial model uses US$11.52/tonne of ore at 2.5 Mtpa ramping up to 3.5 Mtpa by year 3.

16.6.4 Fixed Costs

Based on 2007 estimate of US$7,200,000 per year at 2.0 Mtpa factored up by 20% to US$8,640,000 per year, or US$3.46 per tonne of ore at 2.5 Mtpa.

Current financial model uses US$4.50 per tonne of ore at 2.5 Mtpa ramping up to 3.5 Mtpa by year 3.

16.6.5 Pit Slopes

Overall wall slopes are based on the recommendations by RDCL in its report „Geotechnical Assessment of the Proposed Didipio Open Pit", dated November 2008 (see Table 16.1).

Table 16.1: Overall Pit Slopes

Parameter	Value
Pit slopes	
above base of total oxidation:	
Bench height	15 m
Bench slope	55^0
Berm width	10 m
Inter-ramp slope	36^0
Below base of total oxidation:	
Bench height	15 m
Bench slope	60^0
Berm width	5 m
Inter-ramp slope	47^0
Blasting bench height	15 m
Ramp widths	
Sized for 90 tonne rigid body trucks Plus safety berms and drainage	25m
Ramp grades	10%
Default waste density	2.56t/bcm
Final stacked waste density	2.13t/bcm

Simple pits were designed using these guidelines but also including ramps. The overall slopes for the optimisations were scaled off these designs and allowance was made for flatter slopes through the Biak Shear Zone. The adjusted "practical" overall slopes are shown in Table 16.2.

Table 16.2: Adjusted Slopes for Pit Optimisation

Wall	North	East	West	South
Overall Slope	40°	42°	42°	38°

16.6.6 Ore Loss and Dilution

Ore losses and dilution have been considered. Ore body grade distribution is diffuse. There is no internal waste in the open cut ore blocks at the economic cut-off grade ranges considered in this open cut mining proposal. Ore losses and dilution are therefore not consequential in the low since ore/waste boundaries are "fuzzy".

16.6.7 Process Recoveries

In 2004, Ausenco reviewed the process recovery estimates developed by Minproc for the 1998 Interim Feasibility Study. Ausenco developed a set of empirical formulae for gravity gold recovery, gold and copper recovery to concentrate and concentrate weight recovery based on the head grades of gold and copper.

Ausenco carried out further reviews during 2005 and 2006. The optimisation uses the 2006 Ausenco recovery formulae. Note that it is assumed that there is zero concentrate recovery in the oxide zone, so only oxide ore containing sufficiently high gold grades to pay for treatment from the gravity gold recovered is classed as ore.

When the Ausenco formulae are applied to the ore captured within optimisation, the average recoveries are:

Gravity gold	38.6%
Gold to concentrate	53.4% for total gold recovery of 92%
Copper to concentrate	93.5%
Concentrate weight recovery	1.9%
Concentrate grades	40.7 g/t Au and 27.5% Cu

16.6.8 Metal Prices and Selling Costs

Concentrate transport costs and smelter terms were taken from estimates provided by OGC in February 2007. These are unchanged in the current NMV calculations.

OGC provided long-term metal prices of US$950/oz for gold and US$2.85 per pound for copper.

A discount factor of 10% was used to determine the relative present values of the optimised shell sequences.

16.7 Open Cut Mining

16.7.1 Basis of Open Cut Mine

The Didipio deposit will be mined by both open pit and underground methods. Open pit pre-strip operations will start in 2012 followed by production scheduled to start at the end of 2012. Material mined during the pre-strip phase will be used for the run of mine (ROM) stockpile base and the first TSF embankment construction.

Open pit and underground mines will run concurrently in the last 7 years of operations. The open pit operation has been scheduled to deliver to the mill ore of sufficient grade to achieve 200,000 gold equivalent (AuEq) ounces (oz) per year until the underground mine starts. Low grade ore stockpiled during open pit operations will be milled at the end of both open pit and underground operations. Waste rock mined post pre-strip phase will be used for ongoing TSF embankment and waste rock stack construction. **Figure 5-4** shows the revised project site development plan

16.7.2 Open Cut Mining Objectives

The objectives of the open pit mine are to:

- provide ore to the process plant to produce at least 200,000 oz AuEq annually until the underground mine comes into production;
- establish sufficient high grade ore stockpiles for initial ore feed as quickly as possible with as little pre-production mining as practicable;
- provide waste rock for construction of the stockpile area and the TSF wall; and
- achieve the aforementioned objectives at the minimum cost and maximum Net Present Value (NPV), consistent with best practice and environment stewardship.

16.7.3 Factors Affecting Open Cut Mining

The main factors affecting open pit mining are:

- pit wall stability;
- ground water inflows; and
- surface water inflows from direct rainfall and from the Dinauyan River, which is on the northern side of the orebody.

16.7.4 Pit Optimisation

Whittle Four-X Optimisation software was used to perform pit optimisations using the Lersch-Grossman algorithm. The optimisation was performed in November 2010 on a resource block model updated in June 2008.

A set of nested Whittle pit shells was produced by varying gold price. The shells were used to select the ultimate pit shell and interim shells to guide the open pit design and cutback selection process.

Optimisations were based the following parameters that were updated during 2010:

- gold price;
- mining costs;
- processing costs;
- discount rates and royalties;
- general and administration costs;
- mining recovery and dilution;
- mining and process throughput rates; and
- processing recovery.

Previous optimisation studies limited the open pit base to 2540RL. The objectives during the current optimisation and final pit shell selection were to:

- remove pit size constraints;
- maximise NPV;
- reduce risk by minimising exposure to metal price, recovery and other input parameter changes; and
- keep the processing plant site in place.

A design shell with a pit base at RL2380 was selected to achieve the above objectives while avoiding the associated risk of otherwise "profitable" larger whittle shells with high incremental strip ratio and correspondingly lower marginal net value.

16.7.5 Pit Design

Pit design parameters used are shown in Table 16.3. Mining was scheduled in 6 pit stages or push backs to smooth resultant cashflow. The stages are presented below.

Table 16.3: Mine Design Parameters

Parameter	Value
Pit slopes	
above base of total oxidation:	
Bench height	15 m
Bench slope	55^0
Berm width	10 m
Inter-ramp slope	36^0
Below base of total oxidation:	
Bench height	15 m
Bench slope	60^0
Berm width	5 m
Inter-ramp slope	47^0
Blasting bench height	15 m
Ramp widths	
Sized for 90 tonne rigid body trucks Plus safety berms and drainage	25m
Ramp grades	10%
Default waste density	2.56t/bcm
Final stacked waste density	2.13t/bcm

Figure 16.1: Stage 1 Open Pit

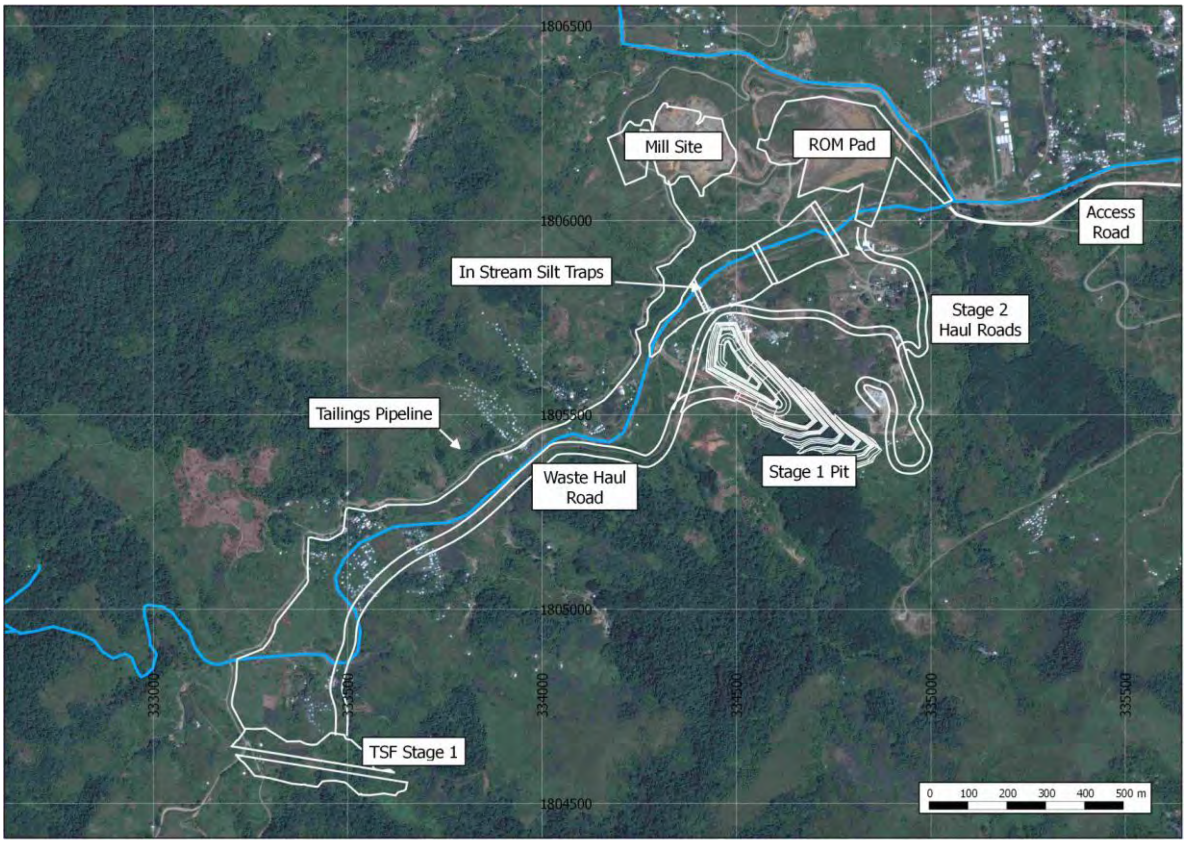


Figure 16.1 shows the Stage 1 pit.

Stage 1

- Interim pit floor is RL2660.
- One ramp from surface to RL2660.
- Preproduction year to provide waste fill for the initial lift of the tailings dam.
- 40 tonne articulated trucks haul waste to TSF stage 1 embankment.
- Material handling entirely in waste.

Stage 2

- Mining “Didipio Hill”.
- 90 tonne rigid body trucks haul ore and waste to ROM pad and waste rock stack or TSF.
- Mining predominately ore.
- Substantial amount of low grade ore to stockpile.

Stages 3 and 4 are expansions of stage 2 pit in all directions.

- Two ramps exit the pit; waste to the south and ROM ore to the north.
- Stage 3 requires a flood protection bund close to the Dinauyan River.
- Stage 4 Dinauyan River flow diverted as the pit rim crosses the river

Figure 16.2 shows the Stage 4 Open Pit.

Figure 16.2: Stage 4 Open Pit

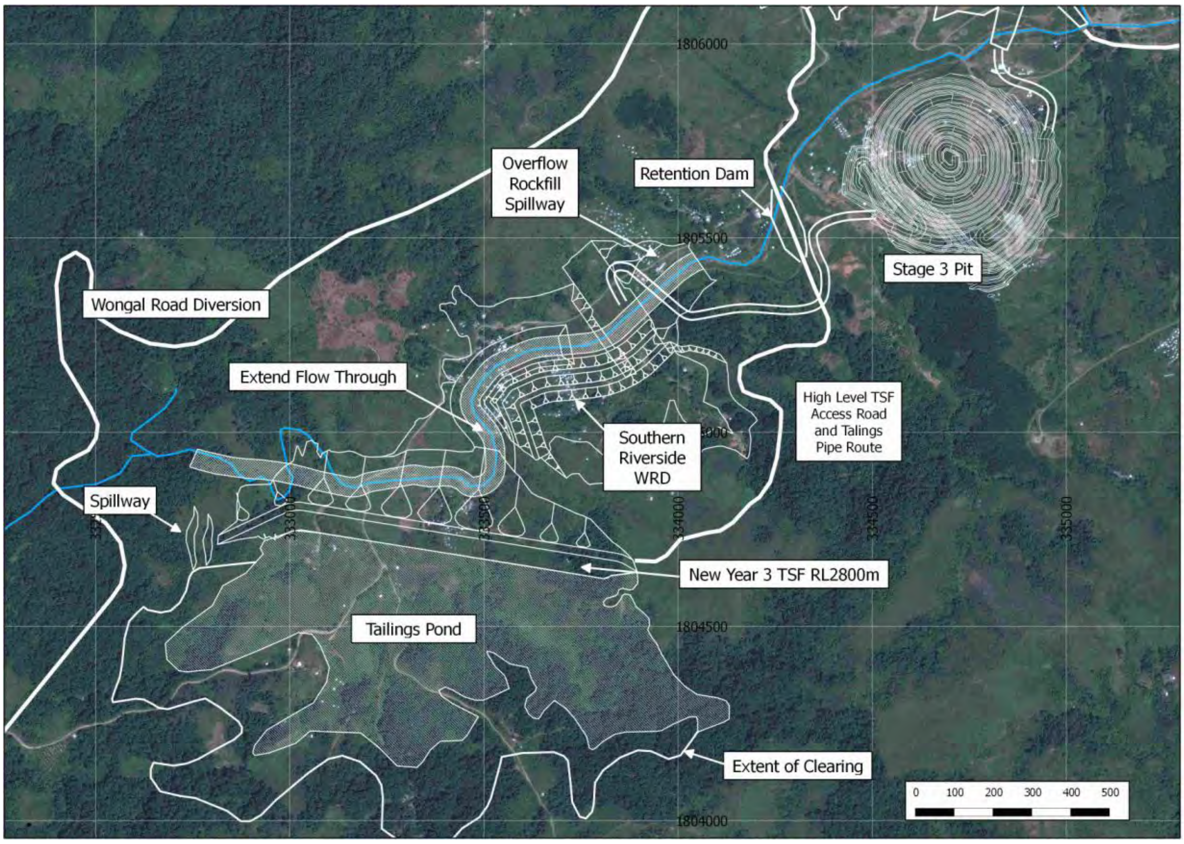


Figure 16.3 shows the Final Open Pit, with Stage 6 completed.

Stages 5 and 6

- Final pit floor is RL2380.
- Two ramps exit the pit; waste to the south and ROM ore to the north to RL2525.
- Reduced ramps at pit bottom, handling primarily ore.

Figure 16.3: Stage 6 Open Pit (Final)

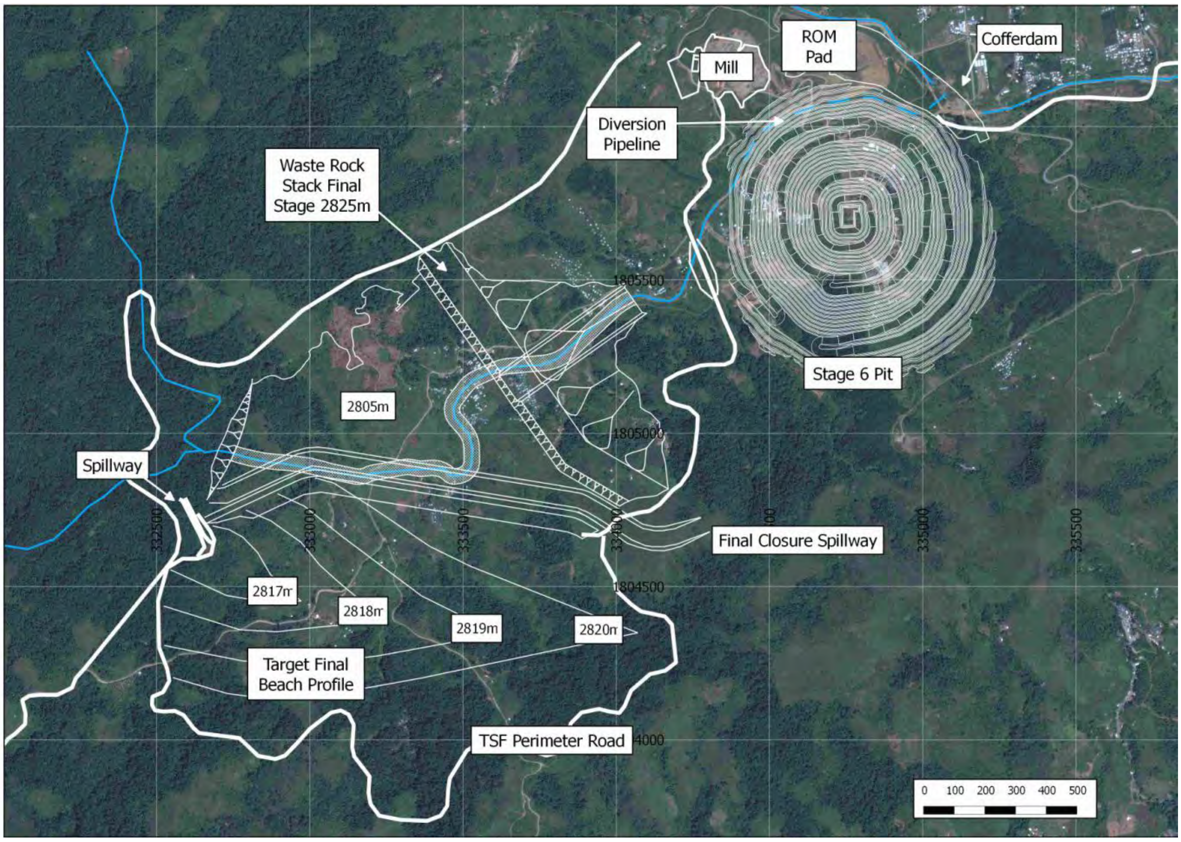
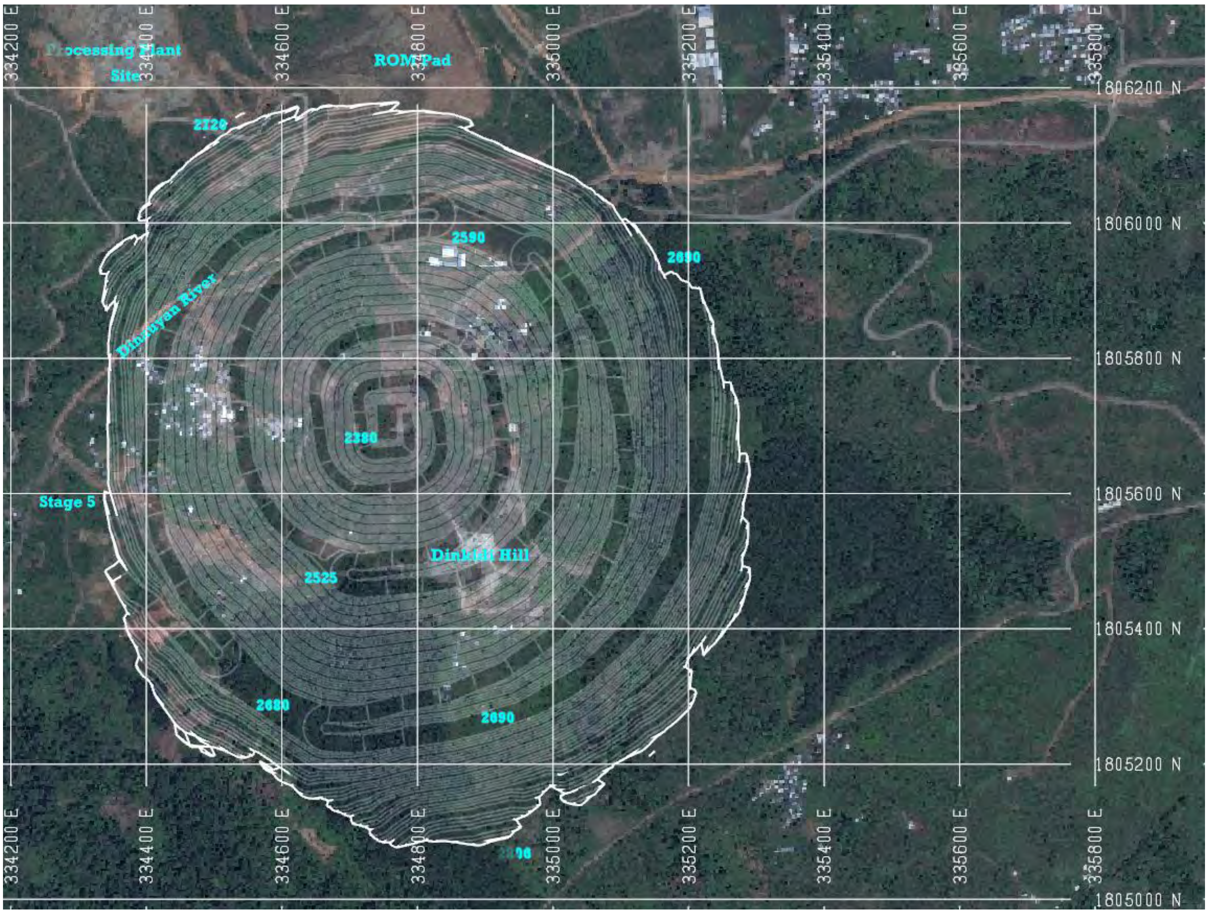


Figure 16.4: Stage 6 Open Pit (Final)



16.7.6 Grade Control

Grade control will use a dedicated reverse circulation (RC) drill rig to drill all mining benches and keep the sampling and ore definition process ahead of mining.

16.7.7 Drill and Blast

Waste and ore will be drilled and blasted to fragment rock to a size that is suitable for load and haul operations and construction of the different stages of the TSF and waste stack. A dedicated blasthole drill rig will be utilised.

Blasting will utilise 100% emulsion delivered from an onsite emulsion manufacturing plant by mobile mixing units.

Detonators, boosters and packaged explosives will be stored in magazines to be constructed at site. Blast design is suitable for the site conditions and results in a powder factor of 0.67 kg/bcm.

16.7.8 Loading and Hauling

Loading and hauling operations will be carried out using conventional truck and excavator method. The operation will utilise 40 ton ADTs and a mix of 110 ton and 180 ton excavators at the beginning of the project and switch to 90 ton rigid body dump trucks and 180 ton excavators.

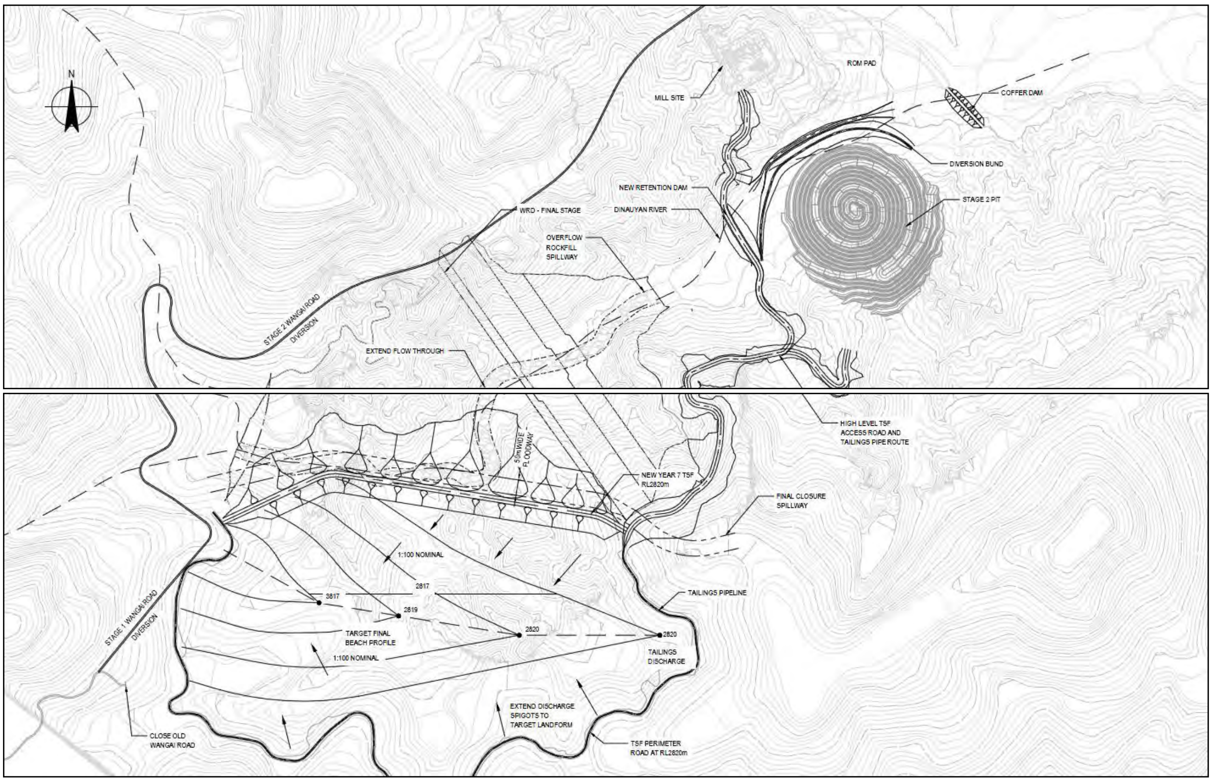
Waste will be hauled to the waste stack and TSF embankments and ore will be hauled to the ROM or to low-grade stockpile.

16.7.9 Waste Rock Disposal

Waste rock will be used to build the TSF wall and the remainder of the waste will be placed on the adjoined Waste Rock Dump (WRD). Waste characterisation studies have shown negligible potential for acid drainage from the waste stacks. The waste stack will be formed in a series of 10m lifts in the Dinauyan valley.

The overall mine waste storage concept provides an integrated solution to store waste rock materials and tailings while initially diverting and eventually throttling Dinauyan River flows, to mitigate the risk of flooding the downstream mine pit. A conceptual layout is provided in Figure 16.5.

Figure 16.5: Conceptual Layout of Mine Waste Storage and Surface Water Management



The WRD is designed as a “Flow Through” Dump, constructed by stacking fresh coarse non sulphide waste rock (Type 1) from a minimum tip head for the dump foundation of 20m. Segregation of the waste rock will result in large boulders concentrating to the base of the drain forming a thick permeable zone for flow to pass through before the general waste rock is stacked above the drain. This is further discussed under 16.7.11 Surface Water Management

16.7.10 Groundwater Management

Ground water inflows to the pit are expected to come from the Biak Shear Zone. A system of advance dewatering bores will be installed to maintain the water table below the pit floor.

16.7.11 Surface Water Management

Didipio is subject to a high annual rainfall that includes intense, sometimes cyclonic, rain events. Surface water management for the open cut consists of:

- Preventing surface flows from entering the pit, and
- Disposing of water from rain that falls directly on to the pit area.

Most of the surface water that could enter the pit comes from upstream in the Dinuyan Valley. This water will be captured as part of the tailings emplacement water management system. Other minor flows from the southern side of the pit will be intercepted in surface drains and directed around the pit crest to the Surong River.

Rainfall directly on to the pit area will be removed by in-pit sump pumps.

The Dinuyan River flow averages 2.6m^3/s during the wettest month and is subject to high flow events. A Concept Design for surface water management of the Dinuyan River was prepared in May 2011 by GHD. Progressive diversion of the Dinauyan River is required in order to excavate the planned mine pit. The initial diversion will be limited and include a series of in-stream silt traps. As the pit develops the diversion will become more significant with a diversion bund to protect the pit. When the pit eventually extends across the valley a piped diversion will be constructed. By this time the waste rock stack WRS will have been developed to provide a “flow-through” under drain and a retention pond that will allow storage of major floods and release at a manageable flow rate.

The WRS will be constructed in a way to allow for detention of flood flows once the flow through drain has reached its full flow capacity. When the drain reaches capacity, excess water will back up within the retention basin provided between the WRS and TSF crests.

The WRS detention basin is sized to cater for storage of a PMF event, with the inclusion of an Overtopping Rockfill Spillway to cater for a 1:100 year flood event.

The WRS will be managed to maintain dumping access during flood events.

16.7.12 Open Cut Geotechnical

The northern pit wall, approximately 315m in height, is affected by the Biak Shear Zone and broken ground to the north. It is designed at a conservative overall angle of 37^0, flatter than that recommended after geotechnical review by RDCL in 2008. Costing has been allowed for pre-splitting and horizontal drain hole installation on all intermediate and final pit walls.

The highest wall is 428m on the southeast corner and 38.5^0 from pit floor to crest. This wall will be formed in the more competent Dark Diorite rock. The top 45m are above the base of oxidation and are laid back at an overall angle of less than 36^0.

16.8 Underground Mining – Sub Level Open Stope with Paste Backfill

A decline access from RL2690 to the underground ore body between RL2390 and RL2180 will be developed in parallel with the open pit mining stage. An infrastructure level at RL2420 will accommodate primary routes for intake and exhaust ventilation, dewatering reticulation, backfill reticulation, electrical reticulation and miscellaneous infrastructure such as an explosive magazine, emergency refuge, satellite refuelling station, etc.

Once RL2420 infrastructure development is well advanced then decline access with supporting ventilation, dewatering and power reticulation to RL2180 becomes the priority to establish the bottom up 1-1.2Mtpa Sub-Level Open Stope (SLOS) mining sequence as soon as practical.

Due to the underground operating in parallel with the open cut, a mining method involving sequenced excavation of geotechnically stable voids that are immediately backfilled with cemented mill tailings is necessary to avoid a caving situation affecting the open pit.

The underground mining sequence requires a partially recoverable crown pillar of mineralisation between RL2420 and RL2390 to manage water ingress from the open pit sump. The open pit will excavate the crown pillar region at the end of the underground operation at which time the underground workings will be flooded.

16.8.1 Factors Affecting Underground Mine Design

The main factors affecting underground mine design were:

- Concurrent 1-1.2Mtpa underground production and open pit mining.
- Geotechnical stability effects of concurrent open pit and underground mining.
- Large groundwater inflows expected from the Biak Shear Zone.
- Rainfall seepage expected from the open pit excavation.

16.8.2 Selection of Mining Method

Block caving, panel caving and sublevel caving mining methods were considered. Geotechnical risks to the overlying open pit and adjacent processing plant were assessed and SLOS with cemented tailings backfill in the mined out stopes was selected as the best means of safely and economically exploiting the massive geometry of the underground ore zone.

Figure 16.6: Underground Workings (Plan View)

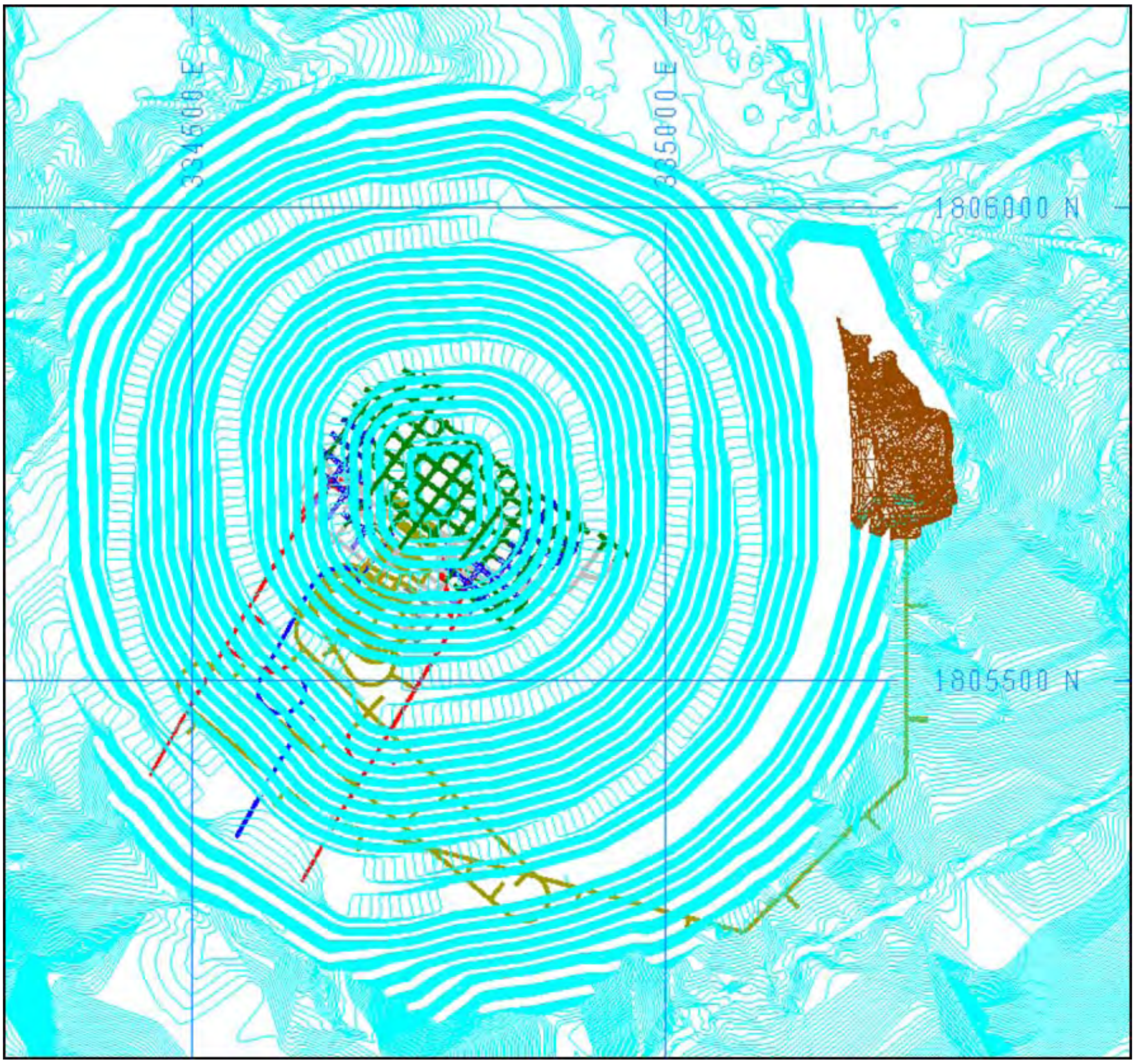


Figure 16.6 and Figure 16.7 for underground layout and typical section and plan view details.

The use of cemented fill is designed to enhance resource recovery while maintaining the integrity of the surrounding rock mass outside the stopes; this in turn will help reduce water inflow risks associated with alternative caving options.

Development headings will be mined using conventional two boom jumbo drill and blast equipment. All waste and ore will be hauled out of the mine using a fleet of 10 tonne capacity load-haul-stack (LHD) underground applicable front end loaders and 50 tonne underground applicable haul trucks. Development rates and ore production were scheduled based on current practices in the mechanised mining industry. Owner/operator costing was done from first principles using current 2011 costs for supplies and labour. All development cross sectional profiles are planned with appropriate ground support, including rock bolting, meshing and shotcrete where required.

A maximum production rate of 1.2 Mtpa has been scheduled which will be achieved using conventional underground equipment including a fleet of five to six underground trucks. The scheduling and costing is based on owner/operator estimates.

The decline is designed at a 1 in 8 gradient, as a conservative contingency against a 1 in 7 limit typical in similar underground mining applications. The decline profile is 5.0 m wide by 5.5 m high to accommodate 50 tonne underground applicable haul trucks travelling beneath 1.2mØ ventilation duct. General level development and ore access drifts are 5.0 m by 5.0 m and the draw point drifts are 4.5 m by 4.5 m.

Figure 16.7: Underground Workings (Section Looking North – 307°)

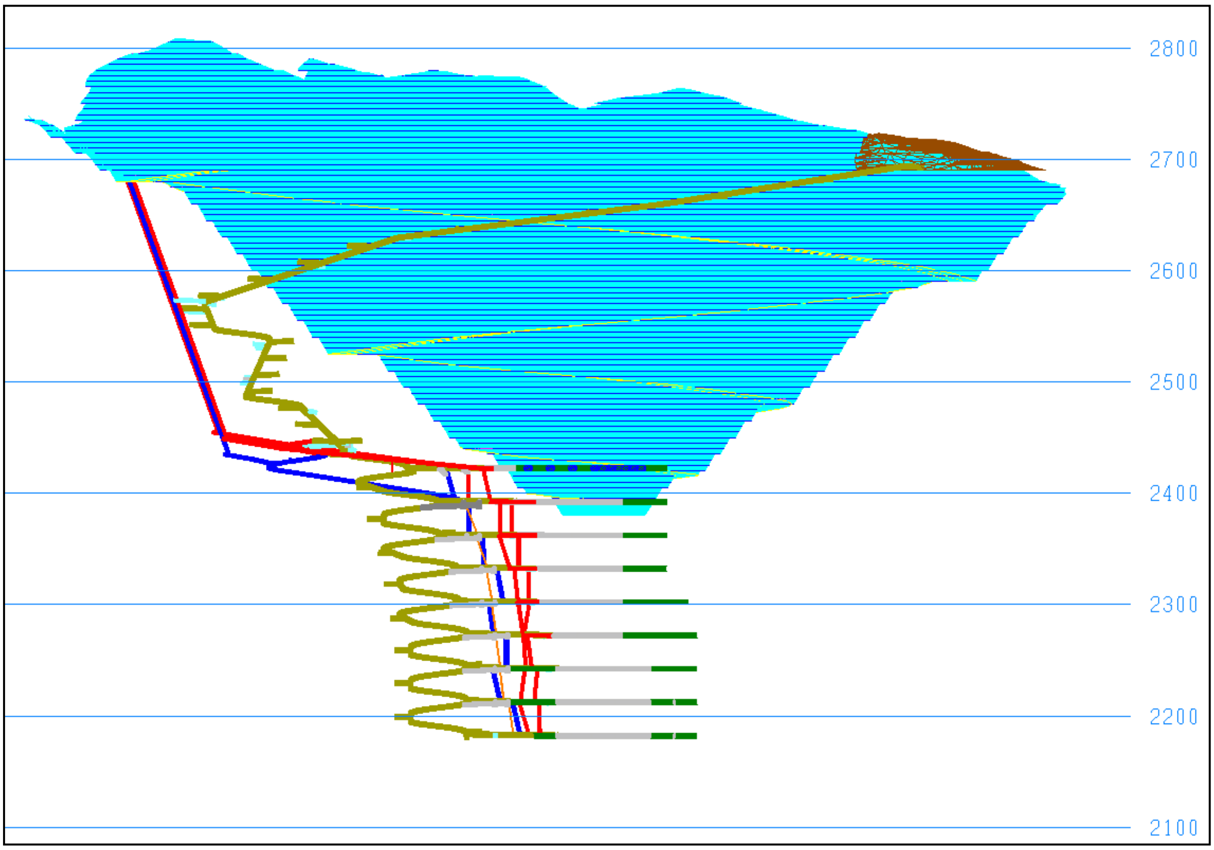


Figure 16.8: Underground Workings (Plan View)

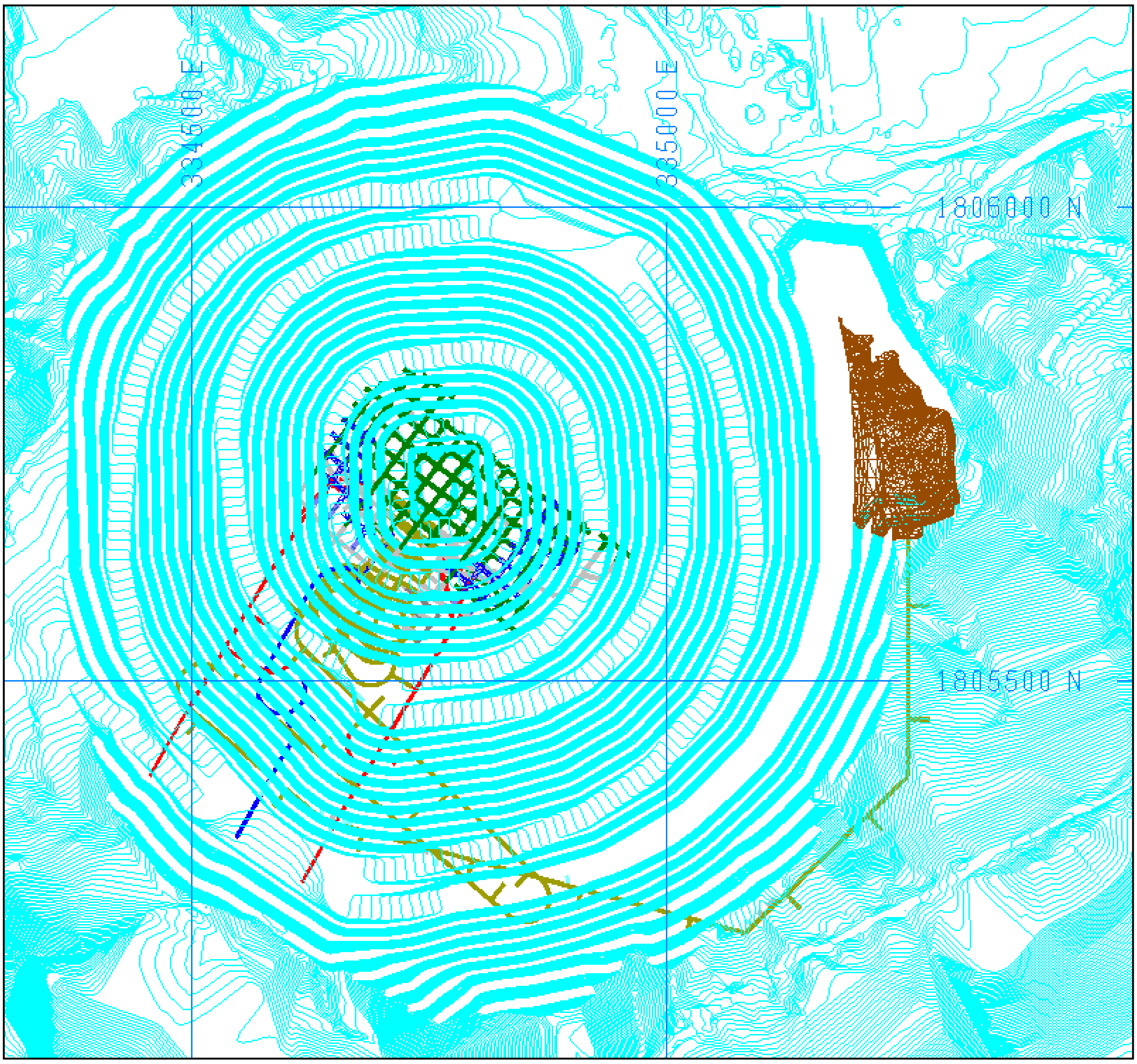


Figure 16.9: Typical Section during Primary Sequence Production

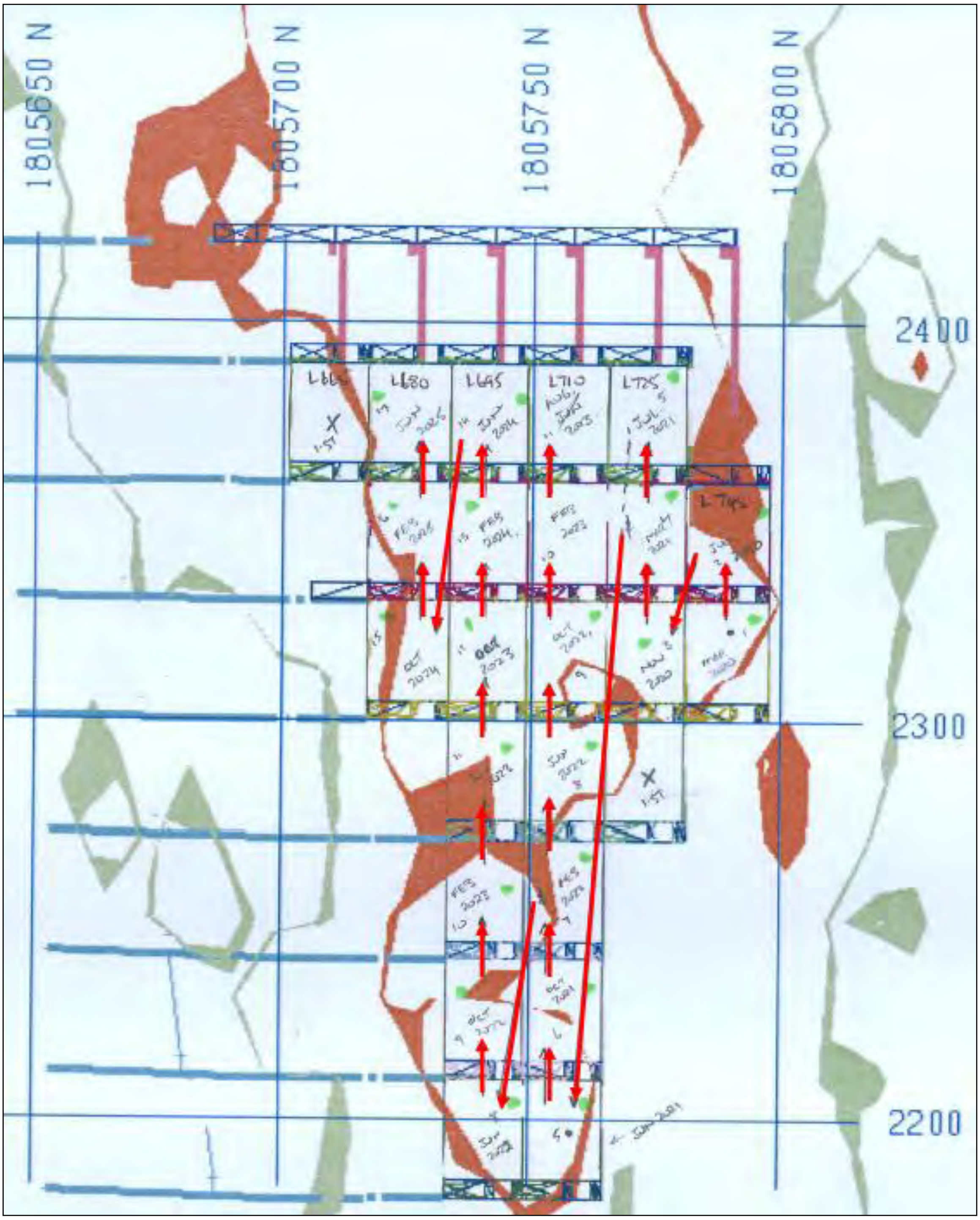
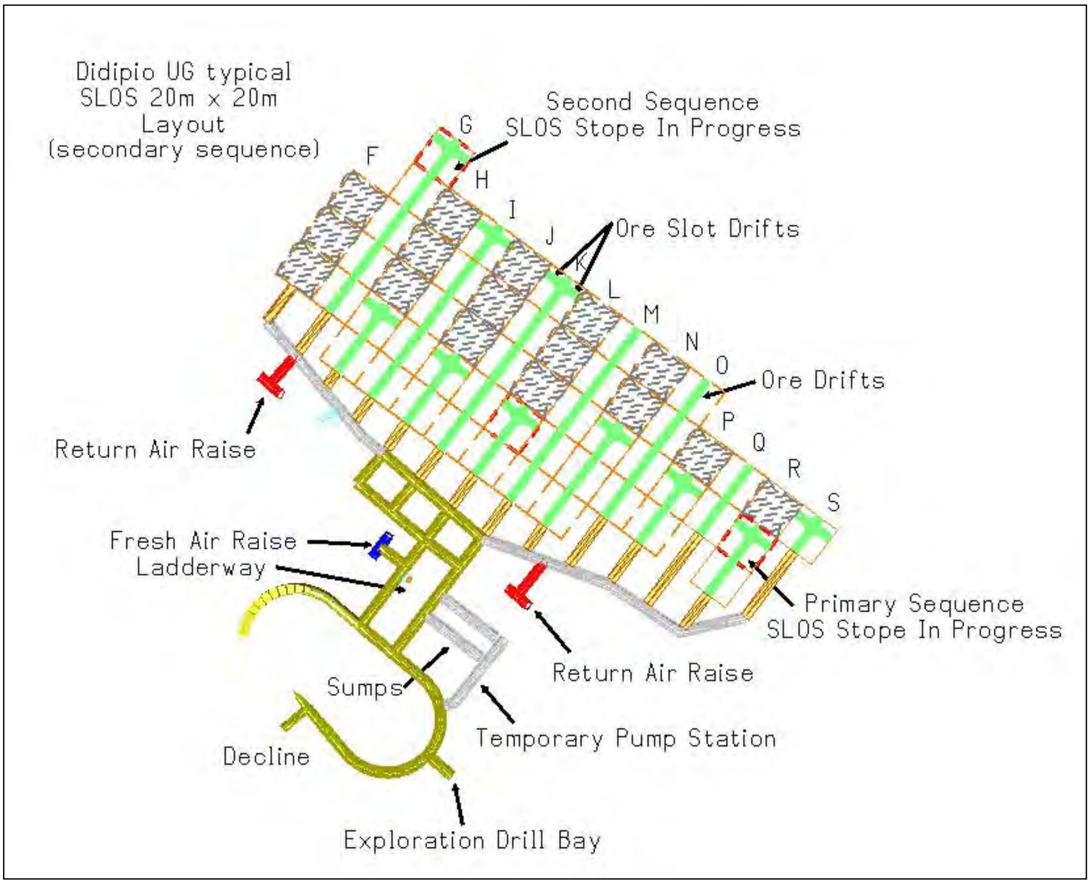


Figure 16.10: Typical Level Development Layout during Overall Sequence



16.8.3 Underground Geotechnical and Groundwater

A substantial amount of geotechnical work has been carried out on the Didipio Project, most recently by Australian Mining Consultants in 2008. The concurrent excavation of both open pit and underground stages requires the mine design to manage regional geotechnical stability. This is achieved by the selected Sub Level Open Stoping (SLOS) mining method utilising cemented tailings backfill in the mined out stopes.

Geotechnical studies led to the adoption of a recommended stope height of 60m, except in areas of „poorer" ground conditions, i.e. within the Bugoy breccia rock unit, where the height is limited to 30m. Stope size was limited to 20m x 20m in plan area.

Potential fracture propagation and subsequent ingress of water related to the Biak Shear Zone will be managed by the stope and fill sequence and appropriate stand-off distances to avoid exposing the shear zone into underground excavations.

Potential seepage of rain water from the open pit into underground workings will be managed by scheduling open pit mining to stand approximately 50 meters away from underground workings until the underground workings are completed. Allowance is made for contingency pumping of seepage into the upper level underground sumps. This may include provision for relief drainage via boreholes directly into underground sumps.

Ground support for decline and lateral development has been detailed and costed appropriate for expected conditions using typical industry average pattern bolting regime for fair to good ground. Addition of shotcrete to the above profile dictated regimes caters for poor ground.

- Fair to Good: Blasts well, support with bolts and roof mesh and cable bolt allocation for intersections and large wedges of rock.
- Poor and Very Poor: More fractured ground to highly fracture ground requiring more difficult drilling. As for Fair to Good classification and 75mm thickness of shotcrete.

- Proportions of each type of ground conditions were applied to each development profile to determine advance rates, explosive, drill steel and ground support usage and overall costs.

16.8.4 Mine Dewatering Strategy

Water ingress above RL2390 is expected to be via seepage through intact rock, minor permeable fractures and un-grouted diamond drill holes. It is expected that the majority of water volume will report to the upper drifts of RL2420 and RL2390, drain along the drifts to one of two 700m^3 capacity dirty water sumps. Once suspended solids settle in the dirty water sump, the water will overflow through cleanable filter screens into a common 250m^3 clean water dam. The settled solids in each alternating dirty water sump will be periodically excavated via LHD and deposited in a stope void being actively filled with cemented paste.

Each level below RL2390 will have a similar dirty water sump and clean water dam arrangement, which will supply the relocatable pump station until the next deeper levels pump station excavation is complete and the system can progressively stage down to RL2210. The relocatable pump station will be a modular version of the permanent RL2390 pump station of same type and duty of pumps. The relocatable pump station will discharge into the RL2390 clean water sump. Water introduced by mining and filling activities, and ground water ingress below RL2390 will report to each levels dirty water sumps and transfer via drain holes to the level where the relocatable pump station resides at the time.

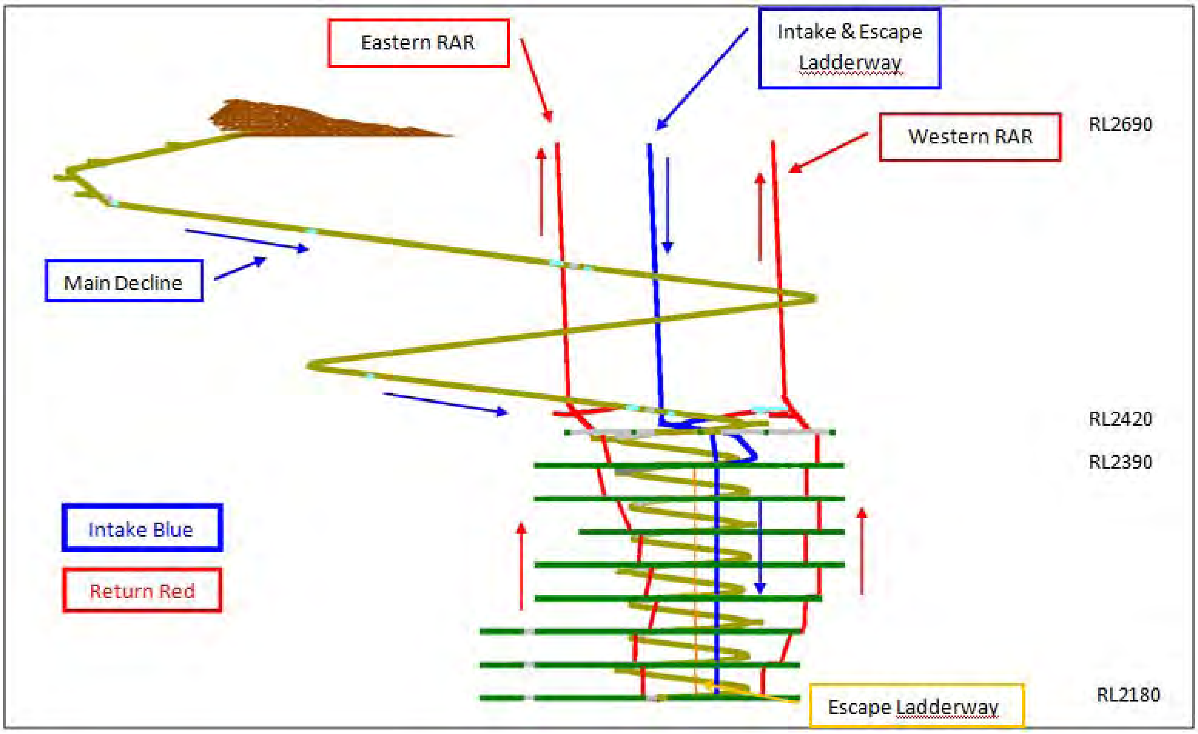
The excavations between RL2210 and RL2180 will act as an emergency sump in times of major rainfall events or the unlikely event when the dewatering system is overwhelmed and short term water storage is required.

16.8.5 Ventilation Network

The proposed ventilation system for the underground mine has fresh air entering the mine via the main decline and Fresh Air Raise (FAR) fitted with an escape ladder-way and return air will exit the mine via the western and eastern Return Air Raises (RAR‟s). All ventilation raises have been designed at 3.5 m by 3.5 m and are assumed to be mined using an Alimak miner. **Error! Reference source not found.**

Intake air will be distributed to each level from auxiliary fans hung in the decline or from auxiliary fans mounted into constructed brick or shotcrete walls adjoining the FAR. The numerous work locations required to achieve 1-1.2Mtpa cannot be achieved via 2 or 3 auxiliary fans hung in the decline. The clean intake air will be directed to the working areas with auxiliary fans and 1.2mØ flexible duct, and then contaminated air will be exhausted to surface via underground installed primary fans that exhaust into the eastern and western RAR‟s. Two 550 kW primary fans have been allowed for in the cost model based on a ventilation review conducted by Mine Ventilation Australia Pty Ltd.. Refer again to Figure 16.10 and Figure 16.11 for the proposed underground development. The primary fans will be located UG since the raise collars are within the active open pit, hence possible damage. Installing the primary fans underground also simplifies electrical reticulation within an active open pit and avoids open pit blast induced vibration trip-outs.

Figure 16.11: Ventilation Network



16.8.6 Sub-Level Open Stope Design

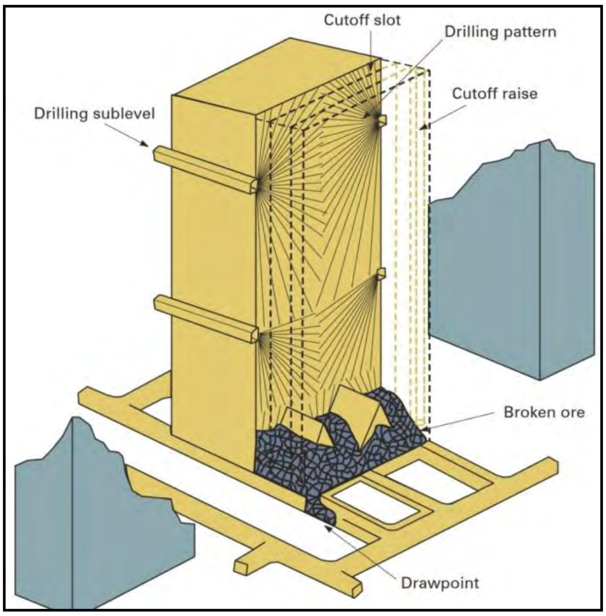
The general concept of sublevel open stoping is shown schematically in Figure 16.12 below.

Open stopes were scheduled in a bottom up retreat column extraction and cemented fill sequence, as shown in the typical cross section illustrated in Figure 16.9 above.

Retreat sequencing of primary and secondary stopes is further illustrated in the typical level development layout presented in Figure 16.10 above.

This sequence avoids time consuming excavation of drifts through cement backfilled stopes which would require substantial additional geotechnical monitoring.

Figure 16.12: Sub-Level Open Stoping Method (Schematic Section)



The main elements of the SLOS operations as applied to Didipio are described.

- The first step in extraction of a stope would be the progressive development of a vertical expansion slot. This creates a void over the full height of the stope that will provide expansion volume for larger production blasts. Once the expansion slot is in place, the stope would be opened up by drilling and blasting of main rings.
- To provide access for drilling and blasting, a single central access would be developed on each level. Then an expansion slot drive would be developed along the edge of the stope, usually on the hanging wall side.
- A vertical rise is then developed between levels in the expansion slot drive, as shown in Figure 16.13 and Figure 16.14 below.

Figure 16.13: Schematic Showing Development for Stope

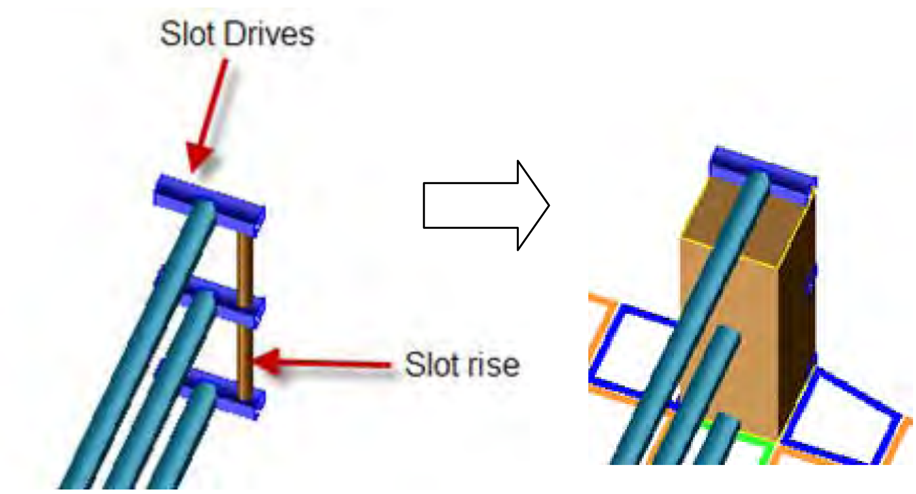
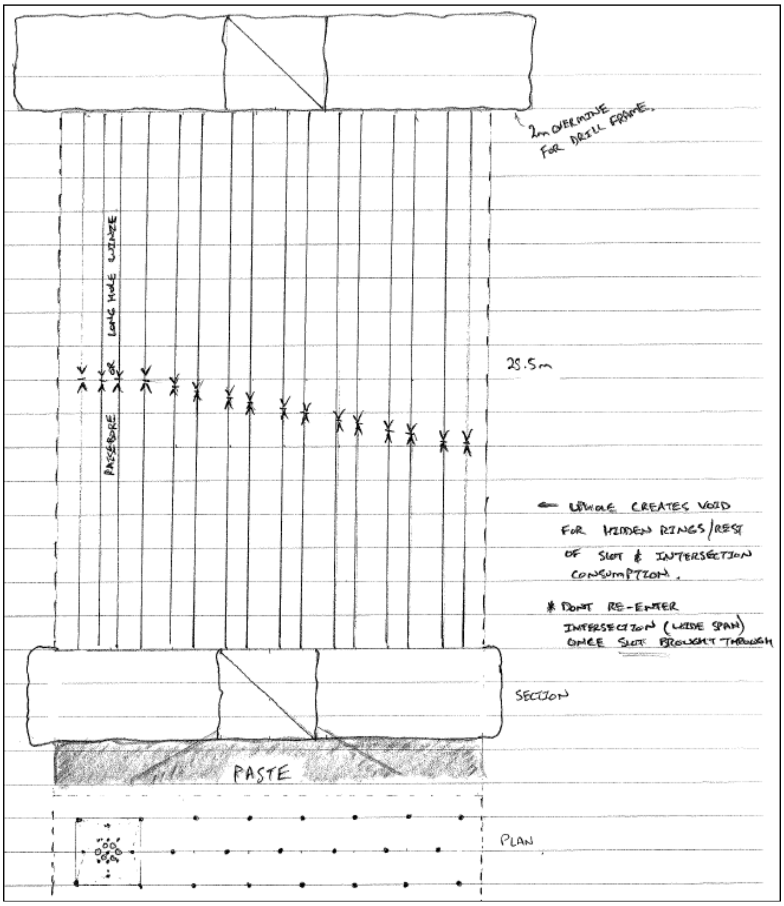


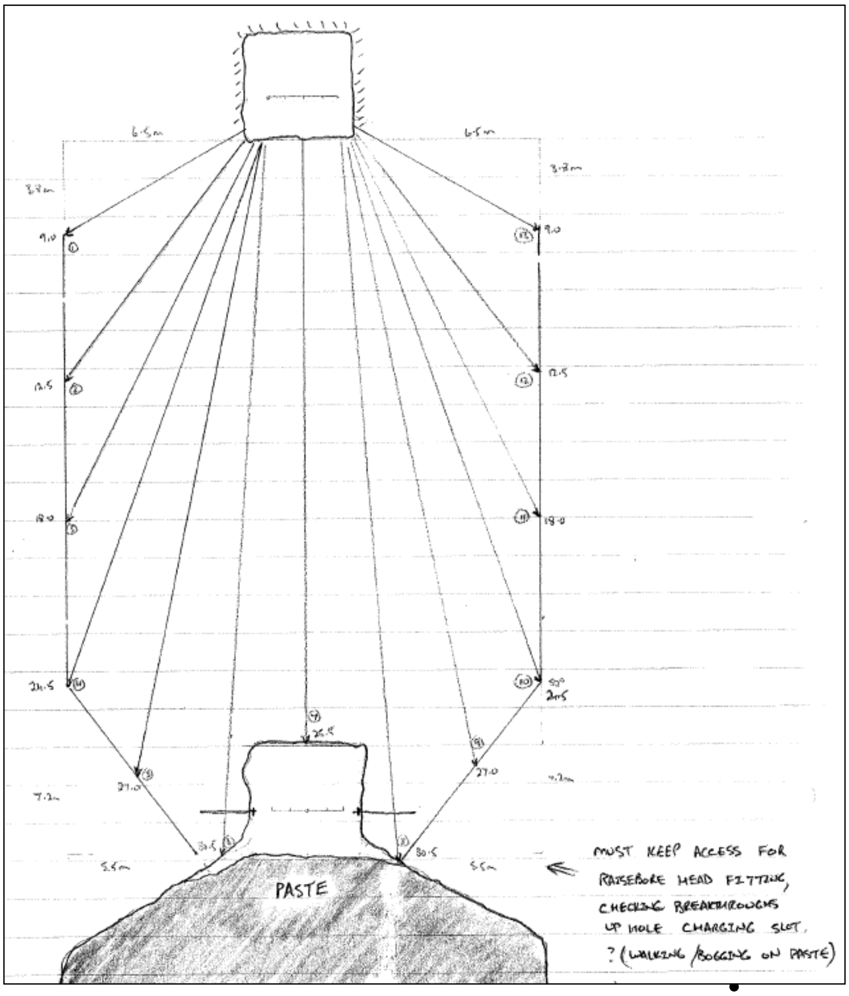
Figure 16.14: SLOS Slot Production Drilling Section



- The rise is then progressively expanded by drilling and blasting small rings of longholes to form a slot the total width of the stope.

- Main rings of fanned blastholes will be drilled from the central access. These rings will be drilled in planes parallel to the expansion slot drive back towards the footwall, as shown in Figure 16.15.

Figure 16.15: SLOS Ring Drilling Section



- After the expansion slot is opened up over the height of the stope, the main rings would be fired progressively back from the slot to the opposite side of the stope.

- After blasting, the fragmented rock would be loaded out or „mucked‟ from drawpoints at the base level of the stope. Rock can be mucked conventionally, with the operator sitting in the loader, until the drawpoint brow is „cracked‟ open. After this point, remote controlled mucking will be required due to the risk of rock landing on the loader from high up in the stope.

- Oversize ore would be moved to nominated bays for secondary breakage during normal firing times. Hang-ups in the stope draw points would be cleared quickly to maintain production.

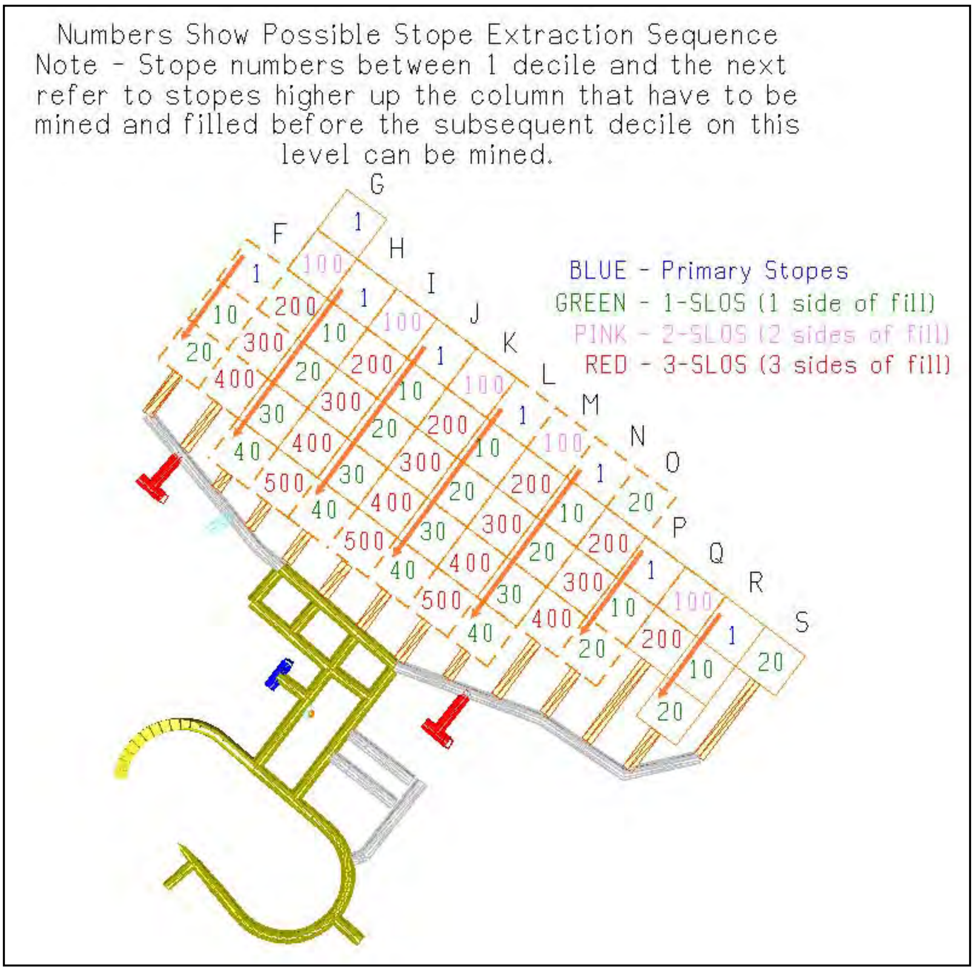
- Once the stope has been mucked „clean‟, it would be prepared for backfilling, including:

  - Construction of barricades that would contain the fill within the stope at each level opening.
  - Installation of services to deliver the backfill to the stope.

- The type of fill to be placed is determined by the sequence-type of the stope. Primary and secondary stopes, against which tertiary stopes will be extracted, will have cement-added backfill. Tertiary stopes, extracted against cement-filled primary and secondary stopes, will be filled with uncemented fill.

- The sequencing of stope extraction and backfilling will be important to maximise recovery and achieve the required production rate. An indicative stope sequence is shown in Figure 16.16 below.

Figure 16.16: Schematic Showing Stope Sequencing



16.8.7 Production Drilling and Blasting

All draw points and slot drives were designed at 4.5 m wide by 4.5 m high to accommodate both production drilling and mucking activities. The production drill required for the slot and main rings will drill blast up-holes and blast down-holes of 89mmØ and 200mmØ slot relief holes. Rates for production drilling and blasting were based on a Tamrock Solo rig. The study assumes that all ore, including the slot, will be production long-hole drilled and blasted.

16.8.8 Production Mucking and Ore Haulage

Production mucking rates were based on 10 t capacity Caterpillar 2900G LHD"s. Productivity for these machines assumes two 12 hour shifts per day with 7.2 work hours per shift. On this basis two dedicated production LHD"s with remote operation capability will be required to mine 1.2 Mtpa. Atlas Copco MT5010 50t trucks were assumed for the production haulage estimates. Haul cycle estimates were prepared for each level and the truck fleet was increased with depth as the haul length up the decline increased.

Studies from operating SLOS mines have shown that one of the biggest impediments to high production rates is coupling of the production loaders and trucks. The production loaders running from the draw points to the decline can take 10 to 12 minutes per bucket. The truck cycles from the loading point in the footwall drive to the ROM pad and back start at 35 minutes in the top lift of RL2360 and increase to 50 minutes at the bottom lift of RL2180. If either the trucks or the loaders are kept waiting at the loading point the efficiency of the system will fall rapidly.

The SLOS mine plan addresses this problem by using every second access drift between the footwall drive and the mineralised region as a stockpile location when the level"s dedicated stockpiles adjacent to the truck loading loop become full.

16.8.9 Underground Mine Design Specifications

Table 16.4: Underground Mine Design Dimensions

Design Parameter	Value
Height between extraction levels	30 m
Decline Profile	5.0 m x 5.5 m
Lateral Access Development Profile	5.0 m x 5.0 m
Lateral Ore Development Profile	4.5 m x 4.5 m
Ventilation Raise Profile (Alimak)	3.5 m x 3.5 m
Decline Gradient	1 in 8
Extraction Drive Spacing	20 m

16.8.10 Layout of Sublevels

The main decline and other permanent development openings are located outside a 50m buffer zone from all the production levels. After completion of the access decline, the bottom two sublevels are developed in each panel. A third sublevel would also be developed if the stopes have multiple lifts.

The north-south draw-points are spaced on 20 m centers as per advice from AMC Consultants. The draw points are set out to traverse from north to south which is easier to ventilate, results in shorter tramming distances between source and stockpile and allows for more draw points and therefore more production flexibility.

16.8.11 Mine Drainage

Application of a SLOS with cement fill mining method maintains the integrity of the surrounding rock mass outside the stope region, thus avoiding the significant water inflow risks associated with a caving option. Water inflow through intact rock, mining activities and un-grouted diamond drill holes will be managed through a sump, dam and pump station arrangement capable of an active 220l/s with an additional 110l/s on standby for short durations.

16.8.12 Mine Backfill

The backfill medium proposed is paste fill with various percentage of binder or cement added depending on how the fill mass will be exposed later on in the sequence. For costing purposes, 95% of the SLOS stopes will be cement paste filled with a 6% binder addition, which is a common percentage for Australian mines that expose an entire wall during extraction of an adjacent subsequent stope.

SLOS stopes that are last in a sequence will still require a minimum 1% binder addition to ensure a competent cured fill mass that is not prone to liquefaction.

Use of paste fill rather than sand fill is necessary since the processing plant will grind the rock fragments too fine to obtain a useful quantity of coarse sand sized fragments required for sand filling methods.

The backfill plant is designed on the tailings line access road southwest of the open pit. The cement is mixed at the plant. It is costed to be pumped using a positive displacement pump via schedule 80 steel lines to boreholes collared on the RL2690 haul road close to the fresh and return underground ventilation raises.

Once underground, backfill is reticulated from the RL2420 drives to the crown of stope void via production drillholes pre-drilled for the purpose.

16.8.13 Production Rate

The target production rate is set at 1.2 Mtpa. To achieve this, the SLOS operation should have:

- Two stopes available for full-scale production;
- One stope coming into production; and
- One stope finishing production.

To account for inevitable delays to production, the schedule has been prepared assuming four stopes available at any time.

16.9 Equipment Fleet and Manning

Open cut mining will be carried out by a contractor. The final configuration of the mining equipment fleet and workforce will be decided in negotiation with the contractor. It is expected that the fleet will comprise a fleet similar to that presented in the table below.

Table 16.5: Major Mining Mobile Fleet

Equipment	Description	Number
Komatsu HM400	40 tonne articulated dump trucks	12
Cat 777	90 tonne rigid dump trucks	13
Terex (O&K) RH40	110 tonne excavator	1
Terex (O&K) RH90	180 tonne excavator	3
Cat 16M	Grader	2
Cat D9	Track mounted bulldozer	3
Cat 834	Rubber tyred bulldozer	1
Cat 773	Water truck – 50 tonne	1
Atlas Copco L8	Drill rig for Grade control	1
Ingersoll Rand DM45	Drill rig for blast holes	1
Mobile mixing unit	Explosives delivery truck	2
988 size loader	Crusher feed and portal stockpile re-handling	2
Cat IT38	Multi-purpose service vehicle	1
Cat 336D	Ancillary Excavator	1
Cat 385Cl	Ancillary Excavator	1
Underground		
Jumbo	Twin boom development jumbo	2
Solo	Long hole drill rig	2
50D haulage	Articulated 50 tonne truck	6
Cat 2900G	Articulated loader	4
Spraymec	Shotcrete spraying equipment	1
Normet cement carrier	Cement carrier	2
Volvo 120	Multi-purpose service vehicle	4
Normet Charmec	Production charging machine	1
Cat 12G grader	Road grader	1
Stores truck with Hiab	Underground stores – delivery	2
Cat IT38	Multi-purpose service vehicle	1

Table 16.6: Breakdown of Workforce

Department	Outside Recruits	Local Recruits	Total
General Manager plus department Managers	4	0	4
Mining technical	26	5	31
Processing	68	42	110
Finance and Administration	77	12	89
Camp/village operations	4	112	116
Security	125	0	125
Mining Contractors	135	40	175
TOTAL	**445**	**205**	**650**

The underground has been designed and estimated on the basis of owner operator mining. Fleet and manning numbers take into account operating conditions at other Filipino underground operations. Some greater efficiency has been applied to avoid the extremely large numbers of personnel commonly employed in Filipino underground mines.

16.10 Production Schedule

The production schedule is summarised in Table 16.7.

Table 16.7: Annual Production Schedule

Mining		Year Total	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Tonnes																					
OC Ore	kt	**20,250**	210	1,270	2,270	1,720	3,120	1,570	1,000	890	2,870	910	1,370	1,800	1,250	0	0	0	0	0	0
OC low grade ore	kt	**24,450**	1,430	7,320	3,570	2,400	2,630	780	720	3,320	1,020	530	360	300	70	0	0	0	0	0	0
Underground ore	kt	**8,160**	0	0	0	0	0	0	100	210	520	880	1,110	1,200	1,190	1,160	990	600	200	0	0
Total	kt	**52,860**	**1,640**	**8,590**	**5,840**	**4,120**	**5,750**	**2,350**	**1,820**	**4,420**	**4,410**	**2,320**	**2,840**	**3,300**	**2,510**	**1,160**	**990**	**600**	**200**	**0**	**0**
Grades																					
Equivalent gold grade g/t																					
OC Ore	g/t	**2.62**	2.35	3.43	2.67	2.94	2.20	2.83	3.88	1.74	2.10	1.77	2.19	2.77	3.85	0.00	0.00	0.00	0.00	0.00	0.00
OC low grade ore	g/t	**1.17**	1.41	1.63	0.99	1.12	0.79	0.74	0.85	1.01	0.79	0.71	0.68	0.71	0.89	0.00	0.00	0.00	0.00	0.00	0.00
Underground ore	g/t	**2.94**	0.00	0.00	0.00	0.00	0.00	0.00	1.69	1.97	2.40	2.60	2.78	3.25	3.16	2.96	3.19	3.05	3.74	0.00	0.00
Total	g/t	**2.00**	1.53	1.90	1.64	1.88	1.55	2.14	2.56	1.20	1.84	1.84	2.23	2.76	3.44	2.96	3.19	3.05	3.74	0.00	0.00
Gold grade g/t																					
OC Ore	g/t	**1.45**	0.68	1.48	1.24	1.57	1.04	1.72	2.59	0.71	1.07	1.08	1.53	1.86	2.57	0.00	0.00	0.00	0.00	0.00	0.00
OC low grade ore	g/t	**0.40**	0.32	0.54	0.33	0.32	0.35	0.28	0.30	0.39	0.37	0.36	0.37	0.38	0.48	0.00	0.00	0.00	0.00	0.00	0.00
Underground ore	g/t	**2.00**	0.00	0.00	0.00	0.00	0.00	0.00	1.03	1.15	1.56	1.70	1.93	2.29	2.13	2.04	2.05	2.09	2.81	0.00	0.00
		1.05	0.36	0.67	0.68	0.84	0.72	1.24	1.60	0.49	0.97	1.15	1.54	1.88	2.31	2.04	2.05	2.09	2.81	0.00	0.00
Copper grade %																					
OC Ore	%	**0.57**	0.81	0.95	0.69	0.67	0.56	0.54	0.62	0.50	0.50	0.34	0.32	0.44	0.62	0.00	0.00	0.00	0.00	0.00	0.00
OC low grade ore	%	**0.37**	0.53	0.53	0.32	0.39	0.21	0.22	0.27	0.30	0.21	0.17	0.15	0.16	0.20	0.00	0.00	0.00	0.00	0.00	0.00
Underground ore	%	**0.43**	0.00	0.00	0.00	0.00	0.00	0.00	0.30	0.37	0.38	0.40	0.38	0.43	0.46	0.41	0.51	0.43	0.42	0.00	0.00
		0.46	0.57	0.59	0.47	0.51	0.40	0.44	0.47	0.34	0.42	0.32	0.32	0.41	0.53	0.41	0.51	0.43	0.42	0.00	0.00
Open cut Mining																					
Total Ore mined	kbcm	**17,510**	690	3,360	2,270	1,620	2,220	930	680	1,630	1,510	560	680	830	530						
Waste mined	kbcm	**59,630**	3,120	4,630	6,660	7,120	6,870	8,240	6,770	5,810	6,680	3,320	320	90	0	0	0	0	0	0	0
Waste:Ore volume ratio		**3.41**	**4.52**	**1.38**	**2.93**	**4.40**	**3.09**	**8.86**	**9.96**	**3.56**	**4.42**	**5.93**	**0.47**	**0.11**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
Processing																					
Tonnes																					
OC Ore	kt	**20,250**	210	1,270	2,270	1,720	3,120	1,570	1,000	890	2,870	910	1,370	1,800	1,250	0	0	0	0	0	0
OC low grade ore	kt	**24,490**	40	1,190	830	1,750	380	1,930	2,410	2,390	110	1,710	1,020	500	1,060	2,340	2,510	2,900	1,420	0	0
Underground ore	kt	**8,160**	0	0	0	0	0	0	100	210	520	880	1,110	1,200	1,190	1,160	990	600	200	0	0
Total	kt	**52,900**	**250**	**2,460**	**3,100**	**3,470**	**3,500**	**3,500**	**3,510**	**3,490**	**3,500**	**3,500**	**3,500**	**3,500**	**3,500**	**3,500**	**3,500**	**3,500**	**1,620**	**0**	**0**
Grades																					
Equivalent gold grade g/t																					
OC Ore	g/t	**2.62**	2.35	3.43	2.67	2.94	2.20	2.83	3.88	1.74	2.10	1.77	2.19	2.77	3.85	0.00	0.00	0.00	0.00	0.00	0.00
OC low grade ore	g/t	**1.17**	2.08	2.05	1.69	1.56	1.52	1.54	1.41	1.19	1.50	1.15	1.10	1.05	0.96	0.91	0.81	0.75	0.70	0.00	0.00
Underground ore	g/t	**2.94**	0.00	0.00	0.00	0.00	0.00	0.00	1.69	1.97	2.40	2.60	2.78	3.25	3.16	2.96	3.19	3.05	3.74	0.00	0.00
Total	g/t	**2.00**	2.31	2.76	2.41	2.25	2.13	2.12	2.12	1.38	2.13	1.68	2.06	2.69	2.74	1.59	1.48	1.15	1.07	0.00	0.00
Gold grade g/t																					
OC Ore	g/t	**1.45**	0.68	1.48	1.24	1.57	1.04	1.72	2.59	0.71	1.07	1.08	1.53	1.86	2.57	0.00	0.00	0.00	0.00	0.00	0.00
OC low grade ore	g/t	**0.40**	0.53	0.71	0.59	0.54	0.59	0.48	0.40	0.39	0.45	0.30	0.32	0.30	0.38	0.33	0.34	0.32	0.36	0.00	0.00
Underground ore	g/t	**2.00**	0.00	0.00	0.00	0.00	0.00	0.00	1.03	1.15	1.56	1.70	1.93	2.29	2.13	2.04	2.05	2.09	2.81	0.00	0.00
Total	g/t	**1.05**	0.66	1.11	1.06	1.05	0.99	1.03	1.04	0.52	1.12	0.85	1.31	1.78	1.76	0.89	0.82	0.62	0.66	0.00	0.00
Copper grade %																					
OC Ore	%	**0.57**	0.81	0.95	0.69	0.67	0.56	0.54	0.62	0.50	0.50	0.34	0.32	0.44	0.62	0.00	0.00	0.00	0.00	0.00	0.00
OC low grade ore	%	**0.37**	0.75	0.65	0.53	0.49	0.46	0.52	0.49	0.39	0.51	0.41	0.38	0.36	0.28	0.28	0.23	0.21	0.16	0.00	0.00
Underground ore	%	**0.43**	0.00	0.00	0.00	0.00	0.00	0.00	0.30	0.37	0.38	0.40	0.38	0.43	0.46	0.41	0.51	0.43	0.42	0.00	0.00
Total	%	**0.46**	0.80	0.80	0.65	0.58	0.55	0.53	0.52	0.42	0.48	0.39	0.35	0.43	0.46	0.33	0.31	0.25	0.20	0.00	0.00

16.10.1 Mine Development Sequence

Key points to note in the development sequence include:

- The Dinauyan River will have a primary sediment control structure built to ensure any final sediment stirred up from construction and mining operations are contained. The first phase of the advance dewatering bores will also be installed in year 0 to start depressing the water table before the pit goes below ground level. Apart from this structure, the Dinuyan River will continue to flow in its natural course during early open pit mining operations.

- Waste mining commences in the open cut nine months in advance of the start of ore processing to provide fill for construction of the tailings dam wall and to establish a stockpile of ore.

- The mill ramps up production using open cut ore to 3.5 Mtpa by year 3 (2014).

- The mill processes 3.5 Mtpa open cut feed only until the year 6 (2018).

- Portal site preparation at RL2690 is completed as part of stage 4 open cut pre-strip in year 4 (2016).

- Decline and underground infrastructure development continues through year 7 (2019) when the first underground stopes come into production. Development ore is first available in year 6 (2018).

- First underground production ore is mined from the stopes in year 7 (2019). The open cut is still in full production at this time.

- The mill continues to process 3.5 Mtpa of combined underground and open cut feed. The underground component increases to planned full production rate of 1.2 Mtpa by the end of year 9 (2021).

- When the open pit is depleted in year 12 (2024), open cut feed is replaced by lower grade ore stockpiled in earlier production years. Milling rate continues at 3.5 Mtpa using 1.2 Mtpa underground ore supplemented by low grade stockpile.

- Underground mine production ramps down in years 15 and 16 (2027 and 2028) when the last stopes are completed.

- Milling continues at 3.5 Mtpa as underground feed is replaced by remaining low grade stockpiles, which are finally depleted in year 17 (2029).

16.11 ROM Ore Stockpile

The stockpile strategy is designed to:

- Provide 2.5 Mtpa of ore ramping to 3.5 Mtpa for mill feed over the first three years;

- Use the highest-grade ore available at any time;

- Defer lower-grade ore until high grade from the open cut and underground mines is depleted; and

- Keep the working higher-grade stockpiles at manageable levels that can be stored on the ROM pad while maintaining sufficient stocks to cover the lower open cut ore mining rates that will occur between the successive pit wall cutbacks.

16.12 Mine Closure

When open cut mining is complete in year 12 (2024), the open cut mine will be maintained in a safe condition to enable continued use of the open cut ramp for dewatering of the pit above the underground workings at RL2390, and to act as a sump to collect rainwater for removal by pumping it before it enters the underground mine below.

Upon completion of underground mining in year 16 the entry to the decline will be sealed and the underground workings will be allowed to flood. It is expected that the pit will then flood up to the level of water flowing in the Dinuayan River. All remaining mine facilities on surface will be removed and the areas disturbed by mining will be rehabilitated in accordance with commitments made in the EPEP.

The major mine closure phase will occur at the completion of open pit mining and ore processing in year 15. At this time, the tailings beach will have reached full capacity at RL2820. The following mine closure actions will be taken in year 15:

- The flow through waste rock stack will remain in operation.
- The waste rock stack will be contoured in accordance with the final closure plan, topsoiled and planted with suitable vegetation.
- The slight beach angle of the tailings will mean that the final tailings area will be partly covered by a shallow lake. Since there are no toxic chemicals in the tailings the rest of the surface area is available for revegetation or cultivation. A channel will be formed to direct the flow to the Diduyon River across tailings to upstream of the flow through waste dump and this water will be available in the long term land use of the tailings beach surface.
- The surface and groundwater flow into the pit and underground workings will eventually flood the pit to the level of the water table, which should be at, or close to, the lowest point on the pit crest at RL2695. The pit will become a permanent lake and sediment trap for water flowing over the tailings dam area. Overflows from the pit will be directed to the Surong River.
- The process facility, workshops and offices will be removed and the areas will be cleaned and ripped in preparation for either revegetation or cultivation.

17 RECOVERY METHODS

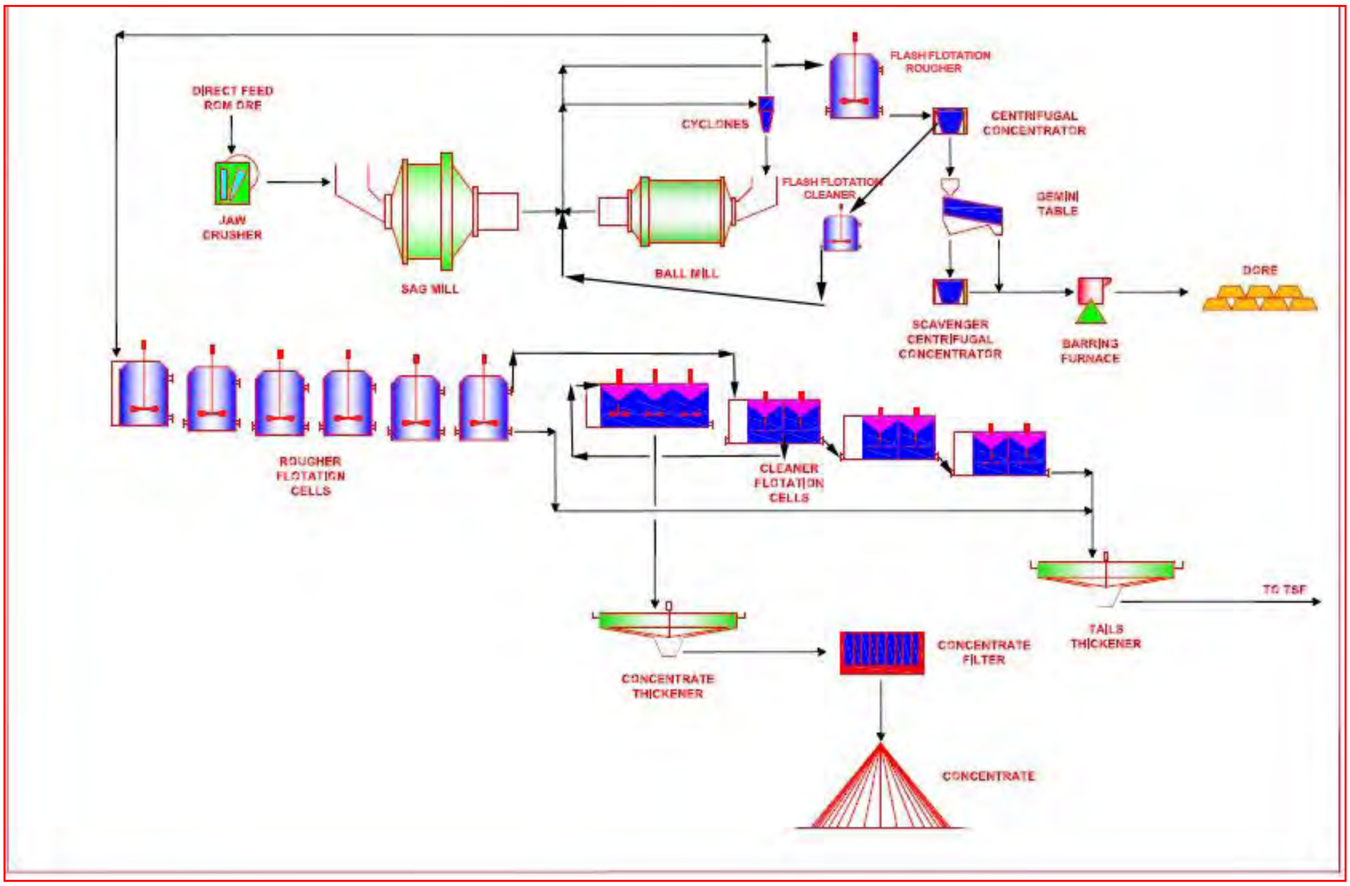
17.1 Recoverability

A full suite of metallurgical test work can be found in Section 13.

17.1.1 Metallurgical Process Plant Design

Ausenco completed a DFS on the project in July 2005, based on a 2Mtpa operation. This was updated to 2.5Mtpa throughput in the PDP and Ausenco has worked with LCPI to undertake sufficient detailed engineering to allow submission of an EPC proposal. The EPCM was awarded to Ausenco in 2007 and detailed design followed a process review, conducted mid 2007. In 2008, the project was put in care and maintenance. In 2009, OGC engaged Arccon to revisit the processing plan taking into consideration the equipment already purchased. Figure 17.1 presents the latest process flow designed by Arccon.

Figure 17.1: Plant Process Flow 2009



The planned process plant for the Didipio Project is a conventional plant for treating gold-copper ores. It comprises a primary open circuit crushing plant that will reduce the maximum 800mm run of mine (ROM) material to approx P80/175mm. The coarsely crushed material will be conveyed to the SAG mill via a transfer bin. The combined SAG and ball mill discharge will be pumped to hydrocyclones for classification, with the fines overflow sent to flotation; the coarse underflow will be split with a approximately 60 % returning to a closed circuit ball mill, with the ball mill discharge similarly pumped to the hydrocyclones for classification. The other 40% will report to the Flash Flotation Rougher. There will also be the option to take a portion of the cyclone underflow back to the SAG mill feed if required.

The cyclone underflow that reports to the Flash Flotation rougher cell will concentrate the coarse copper and gold particles, this will then be directed to a Falcon gravity concentrator, with the higher density tailings to return to the ball mill and the lighter density tailings returning to the mills discharge hopper so that residence time and density in the Ball Mill is optimised. The gravity concentrate will gravitate to the gold room, where it will be further concentrated using a shaking table. The tailings from the Falcon concentrator will report to the Flash Flotation Cleaner with the concentrate from this reporting to the Concentrate thickener and the tailings reporting back into the grinding circuit by way of the mills discharge hopper. Concentrates from the shaking table will be further treated and ultimately smelted into doré bars high in gold and silver. The tailings from the shaking table will be returned to the grinding circuit.

The cyclone overflow, at a nominal particle size distribution of P_{80} = 75µm, will be fed to the rougher flotation cells. Concentrate from the rougher cells will be retreated in a cleaner, scavenger-cleaner and recleaner set of cells to produce final concentrate. Tailings from the rougher cells and the scavenger-cleaner cells are discharged to a tailings thickener to allow a better solids/liquid separation and reagent recycle prior to the pulp being discharged to the Tailings Storage Facility.

Flotation concentrates will be thickened in a concentrate thickener with the thickened pulp then filtered to produce a concentrate filter cake of about 10% moisture, which will be suitable for transport. Trucks will be loaded using a front end loader (FEL) and will transport the concentrates to the port for shipment to smelters.

Process water will come from the dewatering bores and from tailings dam reclaim. Reagents will be received in bulk and mixed on site as necessary.

The following is a more detailed description of the process flowsheet.

17.1.2 Primary Crushing

The crushing circuit will be situated next to the ROM pad. Mining trucks will haul ore from the open pit to the ROM pad. ROM ore will be fed by a FEL through an 800mm square aperture static grizzly into a 100-tonne live capacity ROM bin. The FEL will be required to remove oversize material retained by the static grizzly.

The ROM ore will be reclaimed from the ROM bin by an apron feeder and will be discharged on to a static grizzly into a single toggle crusher. Fines will bypass the crusher. Static grizzly bars will be set at nominally 100mm clearance.

The single toggle crusher, selected to handle 900mm maximum lump size, will crush the ROM ore to a typical P_{80} product size of 175mm. An overhead travelling crane will be provided for changing out crusher jaw plates and for maintenance on other adjacent equipment. Dust suppression water sprays will be provided at the ROM bin and at the head of the transfer bin feed conveyor, emergency stockpile feed conveyor and SAG mill feed conveyor. The sprays will be manually turned on/off from the plant control system.

The ore from the crusher is transported via conveyor 1 (CV-001) and CV-006 to a transfer bin. The transfer bin has a live capacity of approx 5mins of mill feed. A apron feeder located beneath the bin transfers the crushed ore onto the mill feed conveyor CV-003, if CV-03 (or the SAG mill) is offline a diverter gate at the top of the bin will direct the ore coming onto CV-002 the Extra Finer Ore (EFO) conveyor, CV-002 discharges ore onto a 5,000 tonnes EFO pile. If the crusher is offline then the ore from this pile is feed onto CV-003 via the EFO feeder which is a low profile belt feeder. The ROM FEL is utilised to feed this EFO feeder as required.

17.1.3 Grinding

The 7.3m diameter by 4.57m effective grinding length (EGL) grate discharge SAG mill will be fitted with steel liners and pulp discharges and will initially process 2.5 Mtpa. The SAG mill will be equipped with a 4300 kW wound rotor induction motor and Liquid Resistance Starter (LRS) including heat exchanger and capability to provide speed variation through a Slip Energy recovery (SER) unit. The feed spout/chute, mounted on a retractable trolley, will be supplied with bolt-in Ni-hard liners.

Discharge from the SAG mill will flow through a rubber-lined trommel and into a common mill discharge hopper. Oversize from the trommel screen (scats) will be directed to the scats recycle conveyor for return on to the SAG mill feed conveyor. The scats recycle conveyor will be elevated sufficiently at the discharge end to accommodate a recycle crusher if required in future. Supporting structure for the recycle crusher has not been allowed for.

The 5.5m diameter by 8.38m EGL ball mill will be supplied with rubber liners, 4300 kW wound rotor induction motor, LRS, trommel screen and retractable feed spout/chute. Discharge from the ball mill will flow through a rubber-lined trommel into the common mill discharge hopper. The combined SAG and ball mill discharge will be pumped to a hydrocyclone. The hydrocyclone underflow will be split, with approximately 60% reporting to ball mill feed. The other 40% will be directed to a Flash Flotation Rougher (FF Rghr) cell for concentration of the coarse copper and gold particles. The concentrate from the FF Rghr will report to a gravity circuit and the hydrocyclone overflow will gravitate on to a trash screen. The FF Rghr has two tailings outlet the „top valve‟ which is the lower solid material from the upper part of the float cell and the „bottom valve‟ which is the higher/coarse material at the bottom cone of the float cell. This is split so that the volume reporting to the Ball mill is minimised to maximise residence time along with maintaining a higher feed density to the Ball Mill so that the grinding efficiency in the mill is optimise and wear rates reduced.

A tower crane located beside the floatation rougher cells will be utilised to facilitate hydrocyclone replacement and maintenance on the cyclone cluster.

17.1.4 Gravity Circuit

The purpose of the gravity circuit is to recover free gold. The gravity circuit utilises a Falcon model SB2500 concentrator. A by-pass option will be installed that will allow the FF Rghr concentrate to by-pass the concentrator and report directly to the FF Clnr for when the concentrator is in a rinse cycle or is offline for maintenance. The other gravity circuit components consist of a surge bin for the concentrate, a Gemini table treating all the concentrate and a further Falcon model SB250 concentrator on the table tails, all of this will be located in the secured area gold room.

The concentrate from this unit will gravitate to the gold room for further processing. The tails from the concentrator will report to the FF Clnr where the coarse copper and gold particles will be concentrated with the concentrate then reporting to a hopper to be combined with the Re-cleaner concentrate and pumped to the concentrate thickener. The tails from the FF Clnr will gravitate into a hopper and then be pumped back to the combined mills discharge hopper to be pumped back to the cyclones.

17.1.5 Flotation Circuit

Cyclone overflow will be conditioned with reagents in the rougher feed box. The overflow from the conditioner feeds the first of six rougher flotation cells. Tank cells of $40m^3$ will be used for the roughers. Rougher concentrates are pumped to the cleaner cells, with the ability to pump the rougher concentrate directly to final concentrate if the grade is sufficiently high. The rougher tailings report to the tailings thickener.

Concentrate from the cleaner cells feeds the bank of recleaner cells. Tailings from the recleaner cells rejoin the rougher concentrate as feed to the cleaner cells. Concentrate from the recleaner cells will be directed to the concentrate thickener with the FF Cleaner concentrate. The tails from the cleaner cells feed into the cleaner/cleaner-scavenger cells with the tailing from these cells reporting to the cleaner-scavenger cells. The tails from the cleaner-scavenger cells joins the tails from the rougher cells and are pumped to the tails thickener. The concentrate from the cleaner/cleaner-scavenger cleaner cells can be fed to either the feed of the re-cleaner cells or the cleaner cells dependent on concentrate grade. The concentrate from the cleaner-scavenger cells reports back to the feed of the cleaner cells.

17.1.6 Concentrate Handling

Final copper concentrate will be thickened in a 12m diameter conventional thickener. The underflow will be pumped at about 58-60% solids to a storage tank. A vertical Larox plate and frame pressure filter press will produce a concentrate filter cake at about 10% moisture, which will be suitable for transport and sea freight to smelter customers. The filter cake will discharge to a concentrate stockpile of about 15 days‟ capacity located below the filter. The concentrate will be loaded into trucks using a FEL. A weighbridge will be used to control the loading of the trucks. The trucks will carry the concentrate to the port in preparation for ship loading.

Thickener overflow and filtrate from the filter will be returned to the process water tank.

17.1.7 Tailings Handling

Combined flotation tailings will gravitate to a combined final tailings hopper and be pumped to a 20m diameter conventional thickener. Flocculent will be dosed to the thickener feed box by variable speed helical rotor pumps to aid in the settling of concentrate and to provide necessary clarity in thickener overflow.

Two stage variable speed thickener underflow pumps will pump thickened tails to the Tailing Storage Facility (TSF). A portion TSF decant will be reclaimed and pumped to the process water tank with the remaining being returned to the water catchment.

17.1.8 Gravity Gold Concentrate Treatment

The concentrates from the Falcon concentrator will be treated using a Gemini shaking table. Concentrates from the table will be filtered and dried prior to smelting in a standard diesel-fired barring furnace. The tailings and middlings product from the table will be retreated in a small Falcon concentrator, with the concentrates joining the table concentrates for smelting. The tailings from the secondary concentrator will be returned to the grinding circuit. The gold room will be fully enclosed and screened inside.

The dried gravity concentrates will be mixed in batches with fluxes designed to allow the best separation of the gold and silver into doré. These batches will be smelted and poured into moulds to produce the gold/silver doré bars, which are expected to be about 80% gold, 15% silver and about 5% base metals.

17.1.9 Reagents

A number of reagents will be imported to the site, generally in bulk form. Hydrated lime will be imported in 1t bulka-bags and stored in a purpose built reagent shed. The hydrated lime will be mixed with water to a solids density of about 20% solids and distributed to the plant using a ring main system, whereby the slurry can be fed to various distribution points as required.

It is planned to use two collectors, SIBX and S701. The SIBX will be delivered in pellet form in 850kg bags sealed inside wooden crates; this will be mixed on site and stored in a tank with a two-day capacity. A ring main system will distribute the reagent to the addition points. The S701 will be delivered in 200L drums transferred into a storage tank. The reagent will be distributed to the various points in the flotation circuit using dedicated dosing pumps.

The frother comes in 1000L IBC containers and will be pumped to a storage tank to be distributed to the selected flotation points with solenoid-operated valves and timers.

Flocculant will be delivered in 25kg bags. This powder will be mixed in a proprietary mixing plant to a 0.25% solution and then stored in a storage tank. Flocculant distribution will be by a variable speed pump.

17.1.10 Water

Raw water will be supplied from the mine dewatering bores and pumped to a mine water transfer tank. Excess water will be directed to the drainage channel and the balance pumped to the raw water storage tank located in the process plant area. About 300m^3 will be reserved for firefighting needs. Raw water will be used primarily for feed to the potable water treatment plant, heat exchangers and reagent mixing. Potable water will be produced using conventional water filtration followed by chlorination and ultra-violet sterilisation.

Process water will be stored in a tank, with the overflow directed to the process plant environmental pond. Process water will comprise recycled plant water from the concentrate thickener overflow, the tailings thickener overflow, the tailings pond decant, any surface run-off that is collected in the process plant environmental pond and make-up from the raw water supply if necessary. Centrifugal pumps will reticulate the water to the plant distribution points.

17.1.11 Air

Low-pressure air at about 50kPa will be supplied to all the flotation cells. Dry, high-pressure air at about 700kPa for plant and filter requirements will be produced by using duty and standby rotary screw compressors.

17.1.12 Instrumentation and Process Control Philosophy

The proposed plant process control system will be a PLC-based Supervisory Control and Data Acquisition System (SCADA) with PC-based operator control stations. Process control cubicles will be located in the main plant substation and will contain the PLCs, power supplies and I/O cards for instrument monitoring and loop control.

The majority of the plant will be controlled from the main control room, which will be located to give good visibility and access to plant areas, a second control room will be located at the crusher area for control of this section of the plant. An operator control station will be installed in the main plant control room. The compact plant will the need for any other stations as radio communication with the control room will allow desired control functions to be advised to the central control from field operators when necessary. The control station will provide the following functions:

- Graphic displays of all plant areas, showing equipment status and analogue values for critical process variables.
  - Alarm display and logs, showing the tag number and title of the alarm, date and time; and
  - Trend displays with variable time and process variable axes for each analogue process variable.
- Control loop displays, showing controller settings and trending of process variable, set point and output.
  - Printouts of trend pages and alarm logs generated by the control system.

A standby control station will provide redundancy. Power will be supplied from an uninterruptible power supply unit (UPS) with 20 minutes of standby capability.

17.1.13 Plant Buildings and Layout

Ancillary buildings include administration buildings, warehouse, reagent stores, security gate house, ablutions, training and first aid.

17.1.14 Throughput Ramp-up Strategy

The process plant is budgeted to commence operations at 2.5Mtpa, then ramping up to 3.5Mtpa by the end of the second year. Various adjustments to the process plant have been designed in order to maximize throughputs and ease this transition. Some of these changes include:

- Increasing the ball mill charge to draw full mill power
- Coarsening the grind from 75 micron s to 106 microns
- Utilise the capacity if the flash flotation circuit to recover coarse liberated copper ahead of the main flotation circuit
- Implementation of expert control systems to maximise throughput and flotation recovery
- Addition of a third stage of tailings pumps to cover the higher throughput to the full dam height
- Addition of pebble crushing in the grinding circuit
- Judicious review of the initial design of key pumps and equipment to accommodate 3.5mtpa without major downtime

The overall design philosophy of the design by Ausenco has been not to preclude future expansion. In addition to the items outlined above additional allowances have been made in the plant layout to accommodate additional process equipment in the future should a further increase in tonnage be warranted.

18 PROJECT INFRASTRUCTURE

Much of the project infrastructure is discussed in Section 5 of this report. The Tailings Storage Facility is discussed in Section 20.

19 MARKET STUDIES AND CONTRACTS

19.1 Product Off-Take

No detailed off-takers investigation has been done. OGC believe that after three separate independent investigations into likely concentrate quality there will be a ready market for off-take into the nearby South East Asian Markets.

The concentrate product from the Didipio process plant will have the benefits should the market and requirements change for copper concentrate in that it will be low in impurity metals that are undesirable for smelters and could incur penalties or be rejected because of them. These include elements and metals such as manganese oxide, cadmium, arsenic, lead and zinc, all which show very low levels in test work that have been completed (see Table 19.1 below).

The circuit design with the inclusion of a flash floatation cleaner in conjunction with a multi-stage cleaning circuit for the rougher concentrate will allow the copper concentrate grade to be increased should a change in copper percentage increase as a requirement to meet market conditions.

The ability to increase the pH level in the circuit using lime will give the circuit the ability to reduce the amount of pyrite being collected in the concentrate should this become a requirement for the final concentrate product.

Table 19.1: Didipio Concentrate, Metals Composition Analysis

	Report 93053 Metcon Pty Ltd 1993					10.02.1253 Amtec 1995		10.02.1255 Metcon 1997		
Element	P1	P2	P3	P4	P5	Trial 1 Recleaner Con	Trial 3 Recleaner Con			
Test number	1	2	7	4	9	Survey 1	Survey 2	D11/1	D14/1	D16/1
%Cu	30.0	28.2	24.9	25.5	34.6	24.2	25.6	24.6	24.9	39
g/t Au	36.1	28.9	60.0	69.6	144.0	17.0	24.3	57.3	188.0	80.2
g/t Ag	93.0	114.0	76.0	97.0	124.0	69.0	73.0	80	125	132
% S	27.0	25.1	27.5	23.9	26.1	29.6	28.2	29.8	28.2	26.3
%Fe	22.2	24.4	22.1	23.2	24.5	26.7	23.9	29.2	25.5	25.3
%MgO	0.92	0.92	0.64	1.40	0.77	1.28	1.42			
ppm As	91	110	<5	<5	56	230	45	70	160	20
ppm Cd	3.50	1.9	8.2	1.4	3.9	<2	<2			
ppm Pb	681	370	1440	98	2280	599	221	155	1070	375
ppm Zn	696	754	1330	310.00	547	483	260	716	445	636

19.1.1 Market Studies

No Specific Market studies have been undertaken

19.1.2 Commodity Price Assumptions

Commodity price assumptions were US$950 per oz for gold and USD$2.85 per lb for copper based on OGC‟s long term price analysis.

19.2 Contracting Strategy

19.2.1 Construction

OceanaGold have put together an in-house construction team to build the project. The key major design consultants are Ausenco for the Processing Plant, GHD (Tasmania/Manila) for the tailings dam and GHD (Manila) for other site Infrastructure.

The in-house construction team contains:

- Project Director;
- Project Manager;
- Construction Managers;
- Area Managers for earthworks, piping, electrical;
- Procurement Manager; and
- Support staff for document controls, HR, OH&S.

The Major contracts currently in place for the project are:

- Ausenco Pty Ltd for Engineering and Procurement on the Process Plant;
- GHD for tailings dam design, geotechnical investigations and construction supervision for the TSF; and
- Shenzhen VeryPower for Power station fabrication.

The terms and conditions of all contracts are consistent with industry standard.

Other Major Contracts still outstanding are:

- Main Accommodation Village;
- Site Earthworks;
- Concrete, steelwork, piping, electrical and instrumentation for Process Plant; and
- Fuel storage and Dispensing.

19.2.2 Operations

The operations phase will utilise the following major contractors. All of which are still to be awarded:

- A six year open pit mining contract. (Three tenderers);
- Explosives Supply;
- Major Consumables: (Diesel, metallurgical reagents and media);
- Access Road Maintenance;
- An on-site assay lab for metallurgical and grade control samples;
- Concentrate road haulage;
- Port warehousing and ship loading;
- Security; and
- Camp administration and services.

OGC intend to self operate the processing plant, the underground development and operation and all administration support services not listed elsewhere as contracted.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The author has not undertaken any legal due diligence on ownership, tenement or licensing issues. The following is based on information provided by Mr Ramoncito P. Gozar and Mr David Cook in June 2011.

20.1 Environment and Statutory Approvals

20.1.1 Regulatory Issues, Licences and Programmes

All the primary requirements to be fulfilled under the FTAA have been met. Acquisition of the necessary environmental approvals and permits from the relevant government agencies is almost complete. Securing the last permits and approvals required will not be possible until all design details have been finalised, allowing the various construction permits, and subsequent permits-to-operate, to be granted. Table 20.1 lists the permits which are required to enable commencement of mine operations at the Didipio Project.

Table 20.1: Regulatory Requirements and Permits

Permit/Programme	Regulatory Agency	Comment
Environmental Impact Statement (EIS)	DENR/EMB	The EIS was submitted in 1998, with an amendment lodged a few months later. A further amendment to the EIS and issued ECCs was approved by DENR Secretary 6 August 2004. OGPI has applied for a variation of the ECC which is being considered. An Environmental Performance Report and Management Plan was submitted in 2011 as a requisite for the application of a revised ECC
Environmental Compliance Certificate (ECC)	DENR/EMB	Revised ECC was obtained 6 August 2004. Extension of validity of the ECC was approved 9 Dec 2004. An application to amend the ECC to reflect operations under the current mine plan was lodged with EMB in June 2011. A revised ECC (covering the project design changes) will be drafted following deliberations and endorsement by the review committee prior to approval by the DENR Secretary.
Permit to Construct and Operate TSF, Waste Treatment Facility (WTF), Pollution Source Equipment (PSE) and Pollution Control Equipment (PCE)		Permits to construct and operate any new installations are required. Securing these permits will not be possible until all design details have been finalised, allowing the various construction permits, and subsequent permits-to-operate, to be granted. OGPI has a valid permit for Pollution Source Equipment in camp.
Environmental Protection and Enhancement Programme (EPEP); Final Mine Closure Plan	DENR/EMB/MGB	The EPEP is a strategic environmental management plan which is required to cover the life of mine An EPEP was approved by the MGB in January 2005. The financial requirements for environmental-related expenses, which form a component of the EPEP, have not been documented at this stage or included in the operating cost projections A Mine Closure Plan is required to be submitted to the DENR five years before planned mine closure that covers a 10-year post-decommissioning period. A new EPEP is being drafted and will be required as soon as the revised ECC will be issued.
Annual EPEP (AEPEP)	DENR	OGPI has submitted the Annual EPEP (AEPEP) for 2007, 2008, 2009, 2010 and 2011. The AEPEP is a yearly environmental management work plan based upon the EPEP.
Environmental Management Plans and Programs (7)	EMB	Various plans and programs submitted on 21 Feb 2001 to EMB and other agencies and have been approved. Environmental plans for the site are ongoing and will expand with construction.
Mine Rehabilitation Fund and Mine Waste and Tailings Fees Reserve Fund	MGB Region 2	Agreements signed 15 Oct 2004. Contingent Liability and Rehabilitation Fund (CLRF) This is a financial requirement in the form of an environmental guarantee fund to provide for rehabilitation and compensation costs arising from any adverse environmental impacts. A component of the CLRF is the Mine Rehabilitation Fund (MRF), which is a deposit to ensure availability of funds for compliance with the commitments

Permit/Programme	Regulatory Agency	Comment
		and performance of activities stipulated in the EPEP/AEPEP. OGPI will be required to contribute to the CLRF on a periodic basis as stipulated in the environmental provisions of the Mining Act 1995. The company is already maintaining bank deposits under the MRF to service the Monitoring Trust Fund (MTF), Environment Trust Fund (ETF) and the Rehabilitation Cash Fund (RCF), which collectively form the MRF.
Social Development and Management Programme (SDMP)	MGB Region 2	A five-year SDMP was approved by DENR on 8 Feb 2005. The permittee/lessee is required to allocate a minimum of 1.125% (based on a recent administrative order dated 5 May 2010) of the direct mining and milling costs annually to a SMDP once operational. Annuam updates of the SDMP have been prepared and submitted since 2008 following consultation with the community. OGC will submit a new SDMP in 2012.
MOAs with Affected Communities and Local Government	EMB	The DENR reviewed the Memorandums of Agreement (MOAs) that had been executed with affected communities and considered that all required local approvals had been obtained, and approved the grant of the ECC at that time. The Social Development Management Programme (SDMP), which formed part of the ECC approvals, covers the required interaction with the local communities. The company is maintaining the MOAs with the local communities to cover the period before the SDMP becomes effective at commencement of commercial production. The current MOA with the Didipio community was executed in 2006.
Zoning and Location Clearances	HLUR (Region 2)	Approved March 2007
Land Surface Rights Acquisition		The company has acquired, through voluntary agreements, the surface rights to the vast majority of the land required for the current expanded project. Purchase agreements are being negotiated with remaining private landowners. The company expects to have acquired all the land required by the current project footprint in the near future.
Water Permits	NIA, NWRB	All clearances for the project from the National Irrigation Authority have already been obtained during the ECC permitting process. Application for Water Use Permits was remade on 2011; and is being processed. National Water Resource Board (NWRB) legislative provisions require that only national companies or individuals hold water rights. This provision has required OGPI to establish a services agreement covering the provision of the required water permits with a national entity, which has in turn applied for the water permits necessary for the development and operation of the project.
Final Mine Rehabilitation Decommissioning Plan		There is a requirement for such a plan to be submitted to the DENR at least five years before mine closure. The current EPEP document submitted to the DENR contains details which can form the basis of a detailed Mine Closure Plan. A conceptual Mine Closure Plan has been costed in the project financial model. This closure plan will be refined and finalised throughout the life of the mine in consultation with stakeholders. Current costings are expected to cover possible closure scenarios. Development of such a plan and financial provisioning for final closure/decommissioning costs at an early stage is considered to be best practice.

Abbreviations: DENR – Department of Environment and Natural Resources; EMB – Environmental Management Bureau; MGB – Mines and Geosciences Bureau; NIA – National Irrigation Administration; WTF – Waste Treatment Facility; PSE – Pollution Source Equipment; PCE – Pollution Control Equipment; NWRB – National Water Resource Bureau, HLUR – Housing, Land Use Regulatory Board; BG – Bank Guarantee.

20.1.2 Conclusions

All the primary requirements to be fulfilled under the FTAA have been met and acquisition of the necessary environmental approvals and permits from the relevant government agencies is almost complete. Securing the last permits and approvals required will not be possible until all design details have been finalised, allowing the various construction permits, and subsequent permits-to-operate, to be granted. Land acquisition is almost complete and applications for water rights have been made and are in process.

20.2 Environmental Issues

The issues discussed below cover the main environmental risk areas identified to date.

20.2.1 Biophysical Setting

The Didipio Project is located approximately 270km north of Manila in the southern part of the rugged, forested, Mamparang mountain range, straddling the borders of Nueva Vizcaya and Quirino provinces on Luzon Island. The site is located 30km south of the Quirino provincial capital of Cabarroguis, at an elevation of between 500 and 1100m above sea level. The project is located in a generally cleared area below the forest line in a relatively isolated and sparsely populated valley that currently has all-weather road access. The project is sited largely within Barangay Didipio, Kasibu Municipality, Nueva Vizcaya.

There is some dispute about the boundary line between the two provinces within the valley. The MGB advises that resolution of this issue is a legal matter that must be resolved in court. In the meantime, all provincial entitlements will be paid into an escrow account and should affect neither OGPI nor the Didipio Project. The Barangay is the primary beneficiary of the mine development royalties, with 40% going to the local Barangay and 20% to the relevant province. Thereafter, additional royalties go to the National Government.

The project lies to the south-west of the more densely populated Cagayan Valley. The major economic activity here is agriculture with rice, corn, vegetables and citrus being the main products. Commercial activity centres on trading, some manufacturing and food processing. Most families in the area earn below the poverty line. Cabarroguis is the local municipal centre; although commercial activity is strong in areas such as retailing, agriculture is the municipality"s main economic activity.

The project site is located along the Dinauyan River, which flows into the Didipio River, which eventually discharges into the Diduyon River. The Diduyon River is used as a source of irrigation water.

The mine site area experiences a tropical climate consisting of three main seasons: the south-west monsoon season in June-September; the north-west monsoon in October-January; and a transition period in February-May. Didipio receives most of its rainfall during the monsoon seasons, experiencing a mean annual rainfall of 3047mm. The wettest months are September and November and the driest month is normally March. The maritime setting of the Philippines results in relatively small temperature ranges being experienced. The mean annual temperature at the project site is 22.8°C, the hottest months being May and July and the coldest month January. The average annual humidity is high at 82%, with a relative humidity in excess of 80% for more than eight months of the year. The region is subject to an average of two cyclones annually.

20.2.2 Water Management

20.2.2.1 Water Catchments

The Didipio Project is sited along the Dinauyan River, which has a catchment area generating some 27Mm^3 maximum annual water flow. The Dinauyan flows into the Didipio River; the Camgat River flows into the Surong River (36Mm^3 maximum annual water flow), which also flows into the Didipio; the Didipio becomes the Diduyon River, downstream of the confluence with the Alimit River. The existing river water quality is generally good, except for elevated suspended solids and copper levels resulting from illegal mining at Didipio. Elevated mercury levels have been recorded in sediments of in the Dinauyan and Didipio Rivers resulting from illegal mining in the catchment.

The project incorporates river water diversion facilities (the open pit diversion) that carry the waters from the Dinauyan to the Didipio River.

20.2.3 Water Supply

The daily water demand for the project is estimated at 189283m^3, of which 16,428m3 will be recycled water for the process plant, sourced from the process and TSF. It is expected that the TSF return water, which will include direct precipitation, will generally exceed the processing plant"s requirements. Make-up provision in dry periods and during construction will be sourced from boreholes, sited around the open pit area and used subsequently to dewater the pit. At the start-up of processing plant operations, borehole water will be used for commissioning, in conjunction with the build-up of direct precipitation water accumulating in the TSF. Sampling has indicated that borehole water from the mine area is suitable for potable use, with appropriate treatment.

Water from the mine dewatering bores will be collected in a raw water transfer tank. Raw water required for make-up to the processing plant will be pumped to a raw water storage tank. The excess water from the dewatering bores will be directed into the sediment pond system. An off-take from the dewatering bores will direct water to a water treatment plant to allow water to be treated to provide the potable water required for the site clean water and potable water supply. In consultation with the local community, alternative water supply will be made available if required.

The proposed overall approach to water management at the Didipio Project is to minimise discharge from the operating site, direct all dirty water flows, including any waste rock seepage and plant area runoff, to the processing plant. Runoff from disturbed areas will be discharged to the Dinauyan River via a settling pond, and diversion of clean surface water flows around disturbed areas of the site.

Water will be recycled from the TSF initially via a gravity fed pipe and after year 5 from floating pontoon-mounted pumps, with water pumped to the plant for reuse in the process cycle. A project design water balance has been completed by Knight Piésold and this has been updated by MWES Consulting, covering the range of possible rainfall events. Discharge is necessary in most years and this will be via the decant system. Water discharged via the decant system will need to meet acceptable Philippines discharge standards. All current indications are this can be achieved without the need for further treatment. A water discharge permit will need to be granted prior to discharge commencing.

The DENR acknowledges the need to discharge and has indicated verbally that a water discharge permit containing discharge water quality standards is likely to be granted. Should further treatment be required prior to discharge, a water treatment plant will be utilised to meet the discharge standards. In the event of a storm in excess of the design capacity of the decant system, TSF storage water discharged via the spillway will need to meet acceptable discharge standards.

20.2.4 Acid-Generating Material

Testwork undertaken on waste material samples indicates that leachate from the weathered material will be alkaline, thereby having an acid-neutralising capacity. Similarly, tailings liquor samples have also been found to be slightly alkaline. It is proposed that, should potentially acid-generating material be identified in the waste (e.g. from low-grade stockpile reject material), it will be placed in engineered cells and encapsulated in non-acid forming waste. Final designs for the TSF, waste dump and the low-grade stockpile are being finalised.

Mine and TSF decant discharge water will be subject to regular monitoring prior to discharge. However, it should be noted that the dilution factor is very high in both cases.

20.2.5 Tailings Disposal

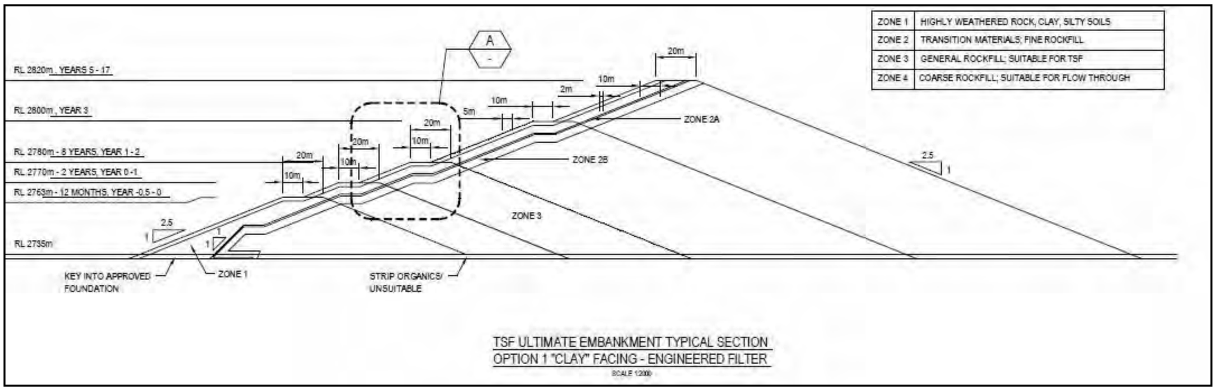
The design details for tailings disposal have recently been reviewed by CMS. The tailings dam design has been developed by GHD, a respected specialist consulting firm in tailings dam design. GHD has also conducted geotechnical site investigations.

The overall Tailings Storage Facility concept provides an integrated solution to store waste rock materials and tailings as previously discussed. It will be built in increments over the life of the project as a staged downstream construction design.

The TSF will be located in a tributary area above the Dinauyan Valley. The TSF is designed as a cross-valley impoundment, located approximately 2 km south west of the proposed plant site. The facility is being designed to store a total of 50Mt of tailings, more than sufficient to accommodate the current Didipio Gold-Copper Deposit reserves. If further deposits are processed through the Didipio plant in the future, the TSF wall could be raised.

OGC plans to pump all the tailings from the tailings thickener (sited near the process plant), at 60% solids, into the TSF for storage. The tailings will be deposited into the TSF using a spigotting method.

Figure 20.1: TSF Concept Cross-Sections



Tailings will be directed against the TSF wall to enhance wall impermeability and will also be moved to other sites. Tailings liquor samples from testwork indicate an alkaline liquor, with low levels of Pb, Cu, Zn, and Hg. Decant water is to be discharged from the TSF using a gravity decant system. Return water from the TSF to the process water will be initially returned by gravity pipeline to the process plant, after approximately 5 years the waste rock stack height requires the return water to be pumped over the TSF wall and gravity piped to the plant for reuse in the process cycle.

Tailings waste characterisation studies have been undertaken and indicate that the tailings are low in both total and soluble metals. Monitoring throughout the life of mine will be undertaken to ensure that the tailings characterisation is understood and potential changes managed throughout the life of the project.

The Hydrologic Design Storm Event for the TSF storage volume (below the spillway) is one in 100 years average return interval for a 24-hour event, plus maximum operating volume of tailings and water. The Hydrologic Storm Event for Spillway Design (which will be available to pass major storm events greater than the 1:100 average return intervals) is to contain and pass a probable maximum precipitation rainfall event. Ongoing monitoring and risk reviews will be required by DENR to ensure compliance and TSF containment integrity.

The TSF has a contained catchment and all precipitation within the catchment will be collected in the TSF. Water collected in the TSF will be used, as required, in the process plant. Water in excess to this requirement will flow into a decant system, which will be discharged to the Dinauyan River at a standard suitable for discharge and in accordance with a discharge permit. It is not expected that this water will require treatment prior to discharge. However, monitoring will be required to ensure that it complies with discharge standards and DENR approval will need to be obtained prior to release.

Table 20.2: Didipio Project Tailings Storage Facility

TSF component	Storage	Method	Construction	Closure
TSF	Single cell; water to be reclaimed via floating pontoon pumping system; approx 50Mt tailings capacity at 9150tpd rate (dry tails); designed to store one in 100 year /24hr storm event. Decant and spillway systems designed to manage maximum precipitation event.	Tailings pumped from the tailings thickener in pipeline approx. 2km from plant; deposited via spigots along embankment (to increase impermeability) and at other sites within the TSF. Return water system directing water to the plant, via pipeline. Excess water discharged to Didipio River, via gravity decant system.	Valley dam: upstream deposition changing to cross-valley prior to closure; single embankment with crest width of 10m for each raise. Constructed as a multi-zoned earth and rock fill dam.	Decommissioned as a mainly dry facility with revegetation and/ or cultivation; the spillway to replace the plugged decant systems for Alimat Creek flow.
TSF decant	A level adjustable gravity decant will allow flexibility in managing water level.	Gravity fed pipe discharging to the Dinauyan River.	Through the wall keyway grouted pipe.	Decant closed and sealed by pumping concrete into the pipe and tower from the discharge end.

The TSF wall will be constructed from waste rock material from the open pit. The waste rock dump downstream of the wall will eventually provide a downstream buttress for the TSF wall, to reduce the risk of dam wall failure. The downstream face will be brought to full height during the first seven years of mine life.

A spillway draining into the Dinauyan River will be constructed on the western side of the TSF wall and adjacent waste rock dump as a “last line of defence” for managing surplus decant/rainfall waters. After mine decommissioning, this spillway is planned to carry water to the Dinauyan River, once the decant system is removed.

The TSF is designed to be decommissioned as a mainly dry facility, with final tailings generated from the processing of oxide material to provide a suitable capping for re-establishment of vegetation. Upon closure, it will be allowed to flow to the downstream river system. After decommissioning, the surface of the TSF will be revegetated or cultivated. A post decommissioning monitoring program will monitor water quality to ensure that water quality criteria are met.

Engineering concept designs have been produced by GHD in May 2011. These were incorporated in the TSF and Diversion Construction Permit application.

20.2.5.1 Seismic Design Criteria

A seismic hazard assessment of the site has been undertaken by Knight Piésold, which shows that the site is located in a seismically sensitive zone. Three major sources of seismic activity are present in the vicinity of the site: the Philippine Fault (40km to the west); the Manila Trench (125km to the west); and the East Luzon Trench (70km to the east). Of these three sources, the East Luzon Trench is reported as having the greatest influence on seismic activity at the mine site. A major earthquake occurred along the Philippine Fault in 1990, with an intensity level of 6.4.

The results of the seismic hazard evaluation have been used to determine a design ground acceleration value for the TSF and for a waste rock dump stability analysis.

The Operating Basis Earthquake (OBE) for the TSF has been selected as the one in 200 year return period event. This corresponds to a maximum bedrock acceleration of 0.26g. A conservative design earthquake magnitude of 8.0 has been assigned to the OBE. The TSF would be expected to function in a normal manner after the OBE.

The Maximum Design Earthquake (MDE) for the TSF has been determined based on a hazard classification of the facility, with consideration of the consequences of failure. A HIGH hazard classification has been assigned to the TSF. Based on the HIGH hazard classification and the results of the seismic hazard analysis, Maximum Design Earthquake scenarios equal to the deterministically derived Maximum Credible Earthquake (MCE) events have been selected.

The following Maximum Design Earthquake scenarios have been considered for seismic design of the tailings facility:

1. Magnitude 8.0 Intraslab Subduction event with a maximum acceleration of 0.55g; and

2. Magnitude 8.5 Interface Subduction event with a maximum acceleration of 0.33g.

The design allows limited deformation of the tailings dam under seismic loading from the MDE, provided that the overall stability and integrity of the facility is maintained and there is no release of stored tailings or water.

20.2.6 Waste Rock Dumps

Waste material will be used in construction of the TSF and other infrastructure. A large waste rock stack will be established across the Dinauyan River Valley and will be operational throughout open pit mining. The waste rock stack will be built in progressive lifts of approximately 10m in height. As each lift is completed the faces of the lift will be rehabilitated. Progressively from year 5 the waste rock stack buttresses the wall of the TSF.

A flow through drain has been designed into the waste rock stack to allow more than the average annual flow in the Dinauyan River to pass through the waste rock stack. This flow through drain will have the effect of attenuating flood flows in the Dinauyan River during the peak of the flood and increase the duration of slightly higher than average flows after the flood even has passed. No additional waste rock dumps are planned. Waste generated from underground mining will be crushed and be available for road maintenance.

The open pit will be developed to a level 320m below the floor of the Dinauyan Valley. Dewatering of the pit and its environs will be by perimeter boreholes and by pumping from a sump located in the pit.

Following completion of the open-cut operation, access to the pit will be restricted by fencing, but cut-off drains will be maintained to minimise surface water flow through the base of the pit and into the underground zone. The roadways on each underground extraction level will be graded to direct any water filtering to a drainage raise, where it will be directed to development voids below the production levels for containment before pumping to a surface settlement dam for removal of sediment and hydrocarbons before release into the Surong/Didipio River system.

The decommissioning phase will make provision for the surface and groundwater flows to enter and be retained in the pit and the remaining open underground workings, eventually flooding the pit to the level of the lowest point on the pit crest. The pit is intended to become a permanent lake and sediment trap for water flowing over the tailings dam and waste rock areas. Overflows from the pit are planned to be directed to a reinstated river channel that flows into the Didipio River.

Given the potential for some minor wall rock acid drainage to develop during and after mining, and in view of the high rainfall in this area, it is proposed that the final pit will be flooded, which will submerge any potential acid-generating pit wall rock. Surface flow from the completed pit will have to ensure the water discharged meets the water quality discharge criteria. This will be assisted both by the extent of dilution that is likely to occur and the basic nature of the geology. The pit wall will be cleared of any recoverable ore any remaining ore will be submerged below the lake water level. Environmental monitoring of water quality in the vicinity of the closed open pit will be undertaken by a long-term, multi-partite committee for which funding has already been agreed to under an MOA. This provision should ensure that waters discharged from the mined areas do not cause long-term effects on downstream water quality and riverine users.

20.2.7 Rehabilitation/Revegetation

The EPEP document contains a progressive rehabilitation strategy and objective. A schedule for ongoing rehabilitation works has been prepared and costed.

20.2.8 Fuel and Chemicals

Diesel storage tanks will be located adjacent to the mine power station in an appropriately bunded area. Plant chemicals will also be stored in an appropriately bunded area, as required by the regulations. Waste oils and lubricants will be recovered and transported to a registered facility for treatment, recovery or disposed.

20.3 Environmental Management

20.3.1 Environmental Protection and Enhancement Programme

This programme is a regulatory requirement and involves a conceptual environmental management plan for the life of mine, including an estimated total cost. APMI submitted an EPEP to the DENR, which was approved with the issuance of a revised ECC in August 2004. OGPI will further revise this EPEP to incorporate the final design for the TSF and associated works and any operating conditions imposed by DENR. An annual environmental protection and enhancement programme work plan (AEPEP), based on the approved EPEP strategy, is a statutory requirement.

The AEPEP makes provision for monitoring of meteorological data, noise levels, and water quality data from designated measurement stations within the river and TSF systems, water quality and flow velocity data from the stream gauging stations, and groundwater data. Air quality monitoring will be carried out to ensure compliance with Philippine ambient air quality objectives during both construction and operation of the mine, and similarly the noise and vibration monitoring will check for compliance with noise and vibration requirements.

Recent work on site in conjunction with the Nuerva Viszcaya State University has established baseline conditions for ambient air and water quality.

20.3.2 Rehabilitation and Mine Closure

Five years before mine closure, OGPI is required to submit to the DENR its final Mine Closure Plan, which needs to include cost estimates. A conceptual Mine Closure Plan is included in the EPEP, which was approved by the DENR in January 2005. This will be updated when the revised EPEP is submitted to DENR.

The main rehabilitation and closure issues facing the project will be the closure of the waste rock dump, the open pit and TSF. Closure planning will ensure that these structures are geotechnically and geochemically stable landforms. Rehabilitation will be undertaken progressively during the operating phase and this is considered an operating expense. In addition an allocation of US$2.5 million and the proceeds of final asset sales have been made to cover final closure cost.

20.3.3 Noise and Impacts on Villages

A noise assessment has been conducted and noise mitigation measures proposed. Noise levels from construction and operation of the open-cut and processing plant are not perceived to be issues of concern, particularly as the nearest village is approximately 1km from the noise-generating areas. Noise effects of the power station have been assessed as part of the tendering process. Noise from the power station will comply with the statutory requirements.

20.3.4 Health and Safety Issues Associated with Road Transport

The use of existing roads in the project area by mine vehicles and the construction of access, service and haul roads raises positive and negative potential health, safety and environmental issues. The haulage of concentrate from the plant site to the port has the potential for significant effects on villages located along the route. At this stage, the final transport route (dependent on which port is finally selected) has not been determined. Therefore, the extent of the impact on affected settlements cannot be assessed. However, it is planned to ship up to 85,000tpa in 20t truck loads, which amounts to approximately 10-15 truckloads per day.

The possibility of accidents along the project"s access, supply and haul routes, especially where they involve people who do not directly benefit from the project, is a risk and plans are being developed to minimise this risk.

20.3.5 Biodiversity Impacts

The proposed management measure to ensure protection of important biodiversity focuses on the establishment of a DENR requirement for an Avian Protection Zone. Baseline environmental studies have identified a depleted wildlife environment in the vicinity of the project, apart from the possible presence of some endangered bird species.

20.4 Conclusions

OceanaGold is committed to establishing and maintaining environmental management and monitoring programmes that are well planned and effective. OceanaGold believes that the risk associated with the potential for off-site water contamination via site run-off, potential leachate seepage, TSF excess water decant or waste rock dump seepage is low.

The location and management of the TSF associated waste rock dump, the disposal of mine and open pit drainage, the direction of waste rock dump run-off and seepage, as well as plant area run-off to the concentrator, significantly reduce the risk of unforeseen effects on the downstream water quality and aquatic environment. The proposed haulage of concentrate from the plant site to the port has the potential for health, environmental and safety effects on villages located along the proposed route, but the access and haulage routes have not yet been finalised.

While all environmental approvals have been acquired for the project, apart from some water permits and some land acquisition, environmental approval for the project (revised ECC dated 11 August 2004) is for a 2Mtpa sized project. Based on the results of further optimisation studies, OGPI intends to increase the throughput of the Didipio Project to 3.5 Mtpa. An application has been made to the EMB to amend the ECC to reflect the current project. The amended ECC is expected to be issued by the end of 2011.

A further uncertainty for the project is the the provincial boundary situation in the Dinauyan/Didipio Valley between the Nueva Vizcaya and Quirino Provinces. While the MGB assures OGPI that this will not affect the development, it may potentially distort local tax/revenue collection (albeit that all such charges fall within the Government share under the FTAA in any event).

21 CAPITAL AND OPERATING COSTS

21.1 Capital and Operating Costs

21.1.1 Capital Estimate of July 2011

The capital cost estimate totals US$185.2M, with a cost base of second quarter 2011. This estimate is summarised in Table 21.1.

Table 21.1: Capital cost estimate summary, June 2011

Item	US$M
Mining	19.13
Process plant	32.98
Power plant	10.51
Tailings Storage Facility (TSF)	7.12
Infrastructure & Services	28.09
Indirect costs	87.34
Total	**185.17**
Money spent/invoiced up to Jun/11	**(19.16)**
Remaining Capex	**166.01**

The capital cost estimate was derived from the following sources and assumptions:

- Open pit mining costs based on first principles calculation benchmarked against 2009 contract rates tendered by Filipino contractors and current OGC operations in New Zealand;
- Process plant estimates built from material take off supplied by Ausenco incorporating local contractors rates and vendors equipment;
- Power plant costs based on vendor documents;
- Bulk quantities for the TSF were provided by GHD and costs estimated by OGC;
- Infrastructure services include construction camp, operations village, roads, IT, communication, batch plant and community projects. Most of the estimates were based on vendors quote and OGC analysis;
- Indirect costs include surface rights acquisition, engineering, project management, catering, insurance, power/fuel supply and freight costs;
- The estimate does not include allowance for escalation during the construction period. However, the construction period is relatively short;
- The capital expenditure profile is estimated to have 35% spent over the remainder of 2011 and 65% spent during 2012; and
- The total estimate does not include additional working capital associated with the start-up of the operation.

21.1.2 Mine Development Costs

Open cut capital and operating costs are based on first principles calculation and benchmarked with quotes obtained from Filipino mining contractors in late 2009 and current OGC operations in New Zealand.

Underground mine capital and operating costs are estimated from first principles by OGC using vendor quotes for equipment and explosives and OGC diesel price assumptions.

21.1.3 Working Capital

There is an allowance for working capital representing approximately 9% of revenue over two years of production in the financial model.

21.1.4 Deferred Capital and Sustaining Capital

Ongoing deferred and sustaining capital expenditure is estimated at US$161.5M, relating principally to ongoing capital expenditure for the underground mine, process plant, the TSF and an allowance for closure costs at the end of life of the mine.

The deferred and sustaining capital cost figure includes:

- US$29.0M spread over 2016-2020 for underground establishment (including site preparation, light vehicles, dewatering, electrical, ventilation fans and back fill plant);
- US$66.0M spread over 2016-2020 for underground development (decline and underground works); and
- US$57.0M including an allowance of US$1.2M per year for sustaining capital and further incremental costs for power plant and process plant upgrades as well as completion of the Tailings Storage Facility;

21.1.5 Accuracy

The accuracy of the capital cost estimate (allowing for working capital) for the project is within ±20%. The capital cost was based on a complete review of the Project at feasibility level. Capital estimating used detail quantity take-offs and vendors quotations for equipment and services.

21.1.6 Operating Costs

The operating cost estimates developed by OGC are summarised in Table 21.2.

Table 21.2: Projected Operating Costs

Sections		Life of Mine	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
Volumes																			
Total Material Mined	Mt	207.8	8.8	20.1	23.1	22.4	23.6	23.2	20.1	19.9	22.4	11.3	3.7	3.5	2.5	1.2	1.0	0.6	0.2
Total Ore Mined	Mt	52.9	1.6	8.6	5.8	4.1	5.8	2.4	1.8	4.4	4.4	2.3	2.8	3.3	2.5	1.2	1.0	0.6	0.2
Open Cut - Total Ore Mined	Mt	44.7	1.6	8.6	5.8	4.1	5.8	2.4	1.7	4.2	3.9	1.4	1.7	2.1	1.3	0.0	0.0	0.0	0.0
Open Cut - Total Waste Mined	Mt	154.9	7.2	11.5	17.3	18.3	17.8	20.9	18.3	15.5	17.9	9.0	0.9	0.2	0.0	0.0	0.0	0.0	0.0
Open Cut - Total Material Mined	Mt	199.6	8.8	20.1	23.1	22.4	23.6	23.2	20.0	19.7	21.8	10.5	2.6	2.3	1.3	0.0	0.0	0.0	0.0
Open Cut - Total Material Moved (incl. rehandling)	Mt	219.6	8.8	20.1	23.5	23.5	23.8	25.0	22.2	20.7	21.8	12.1	3.6	2.8	2.4	2.3	2.5	2.9	1.4
Underground production	Mt	8.2	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.2	0.5	0.9	1.1	1.2	1.2	1.2	1.0	0.6	0.2
Total Ore Milled	Mt	52.9	0.3	2.5	3.1	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	1.6
Product sold																			
Gold in dore	Koz	436.4	0.8	21.5	26.3	28.7	27.5	29.1	29.3	14.5	29.9	24.0	34.5	50.1	49.4	23.6	22.2	16.4	8.5
Gold in concentrate	Koz	1134.2	3.4	54.5	66.4	74.6	68.5	70.5	71.1	33.2	79.8	60.4	95.4	131.0	128.9	67.9	61.3	45.4	21.8
Copper in concentrate	Mlb	482.7	2.3	38.1	41.3	40.8	39.1	37.3	37.0	29.3	34.0	27.5	24.9	30.2	32.6	22.9	21.7	17.4	6.3
Concentrate (dry) sold	Kt	888.7	4.4	72.0	78.0	77.1	73.9	70.5	70.8	55.8	66.9	51.3	45.5	56.8	61.7	37.1	29.6	27.4	9.9
Concentrate (wet) at mine gate	Kt	987.5	4.9	80.0	86.6	85.6	82.1	78.3	78.6	62.0	74.3	57.0	50.6	63.1	68.6	41.3	32.9	30.4	11.0
Mining costs																			
Open Cut	US$/t moved	2.2	0.6	2.1	2.1	2.1	2.1	1.7	2.0	2.3	2.1	2.9	4.1	4.4	4.6	3.1	2.6	2.4	3.6
Underground	US$/t mined	33.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	50.6	37.4	32.1	31.1	29.2	29.4	33.6	40.7	72.5
Processing costs																			
Power cost	US$/t milled	5.4	1.1	6.9	6.9	5.7	5.5	5.4	5.3	5.2	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.2	5.5
Reagents costs	US$/t milled	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
Spares costs	US$/t milled	1.8	1.7	2.0	1.8	1.9	1.9	1.8	1.9	1.8	1.9	1.9	1.9	1.8	1.9	1.8	1.8	1.8	1.8
Maintenance costs (ex owners team)	US$/t milled	0.2	0.0	0.3	0.2	0.2	0.2	0.1	0.2	0.1	0.2	0.1	0.2	0.1	0.2	0.1	0.2	0.1	0.1
Plant Labour	US$/t milled	0.6	1.2	0.8	0.6	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.8
Maintenance Labour	US$/t milled	0.4	0.8	0.5	0.4	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.6
Others	US$/t milled	0.3	0.7	0.4	0.3	0.3	0.3	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.4
Unit Processing	US$/t milled	11.2	8.2	13.4	12.8	11.4	11.3	11.0	11.1	10.8	10.9	10.8	10.9	10.7	11.0	10.7	10.9	10.8	11.9
Other site costs																			
Overheads & site costs	US$/t milled	4.5	8.5	6.6	5.3	4.5	4.6	4.3	4.4	4.4	4.5	4.5	4.2	4.2	4.2	4.1	4.1	4.0	5.4
Logistics																			
Dore transport costs	US$ 000/month	10	Flat																
Land Transport & Ship loading	JS$/t concentrate	58.7	74.4	67.5	67.1	55.3	55.8	52.3	52.3	55.2	52.9	56.4	58.3	54.9	53.9	62.1	67.5	69.4	119.1
Sea Freight	JS$/t concentrate	30.0	Flat																
Concentrate agent fees & insurance	% of revenue	1%	Flat																
Terms of sales																			
Payable Copper in concentrate	%	96.7%	Flat																
Payable Gold in concentrate	%	97.5%	Flat																
Gold Dore refining charge	US$/oz	6.0	Flat																
Gold in concentrate refining charge	US$/oz	6.0	Flat																
Copper concentrate treatment charge (TC)	US$/dmt Con.	80.0	Flat																
Copper concentrate refining charge (RC)	Usc/lb cu	8.0	Flat																

[*] Commissioning phase starts in November 2012.

The operating cost estimates were derived from the following sources and assumptions:

- Open pit mining costs from first principles and benchmarked against contract rates tendered by Filipino contractors and OGC New Zealand operation;
- Underground mining costs estimates based on first principles and benchmarked with OGC New Zealand operations;
- Labour costs estimated based on OGC operating experience, bench-marked against Philippines mining industry standards;
- Power costs based on operating metrics as provided by vendor, including diesel consumption;
- Reagent costs were based on the consumptions as determined from the testwork and prices obtained from OGC operations in New Zealand;
- Spare and maintenance costs were based on OGC operations in New Zealand;

- Overheads and site costs include administrative cost associated with maintaining the site at Didipio and an office in Manila. These costs include meals, accommodation, transport, community relations and general staff;
- Land transport includes cost from site to Port Poro;
- Sea freight is based on cost from Philippines to China; and
- Terms of sale are based on internal benchmarking analysis. Sales contracts have not been negotiated.

Apart from the above onsite and logistics costs and revenue deduction factors the following costs apply for calculation of total cash costs: excise duty and royalties. These are explained in the financial section.

21.1.7 Cash Costs

The calculation of unit cash costs is detailed below and presented using two different price assumptions: one at spot prices (US$1530/ oz Au and US$4.05/lb Cu) and another at long term estimate of US$1050/oz Au and US$3.0/lb Cu:

Table 21.3: Cash Costs

	Spot Price	**Long Term Price**
(+) Total Operating costs *US$M*	1,724	1,711
(+) Total Excise Duty & Royalties *US$M*	174	125
(+) Total Sales deduction	226	195
(treatment charge, refining charge, metal losses) *US$M*		
= Total Cash costs *US$M*	2,123	2,032
Total Gold sold *Moz*	1.57	1.57
Total Copper in concentrate sold *Mlb*	483	483
Total Equivalent Gold sales *Moz*	2.85	2.95
Cash costs per *EqAu* sold *US$/EqAu*	746	689
Copper Gross Revenue *US$M (price x volumes)*	1,955	1,448
Net of By product Cash costs per oz sold US$/oz	**107**	**372**

[*] total operating costs change is due to variable costs associated to revenue such as insurance

22 ECONOMIC ANALYSIS

22.1 Economic Analysis

22.1.1 Assumptions

The financial analysis methodology, discount rates, exchange rates, commodity prices and financial parameters applied in the financial model were sourced from OGC. The inputs are consistent to the capital costs, operating costs and taxes section of this report. The annual cash flow and sensitivity are based on spot prices (US$1530/oz Au and US$4.05/lb Cu).

22.1.2 Cash Flow

The financial analysis indicated that the project had a positive net cash flow and an acceptable internal rate of return and supports the declaration of mineral reserves, which were estimated with the following prices: US$950/oz for gold and US$2.85/lb for copper.

The annual cash flow below is unleveraged pre tax, using spot prices (US$1530/oz Au and US$4.05/lb Cu) and covers the operating years of full production which is forecast to start in 2013.

Table 22.1: Annual Unleveraged Pre Tax Cash Flow and Annual Production

	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029
Pre-tax cash flow (US$m)	106	168	167	114	137	123	34	129	81	151	235	243	110	97	60	18	3
Production																	
Gold in Dore (Koz)	21.5	26.3	28.7	27.5	29.1	29.3	14.5	29.9	24.0	34.5	50.1	49.4	23.6	22.2	16.4	8.5	0.0
Gold in Concentrate (Koz)	55.0	67.0	75.4	69.2	71.2	71.8	33.5	80.6	61.0	96.4	132.4	130.2	68.6	61.9	45.8	22.0	0.0
Total Gold Produced (Koz)	76.5	93.3	104.1	96.7	100.3	101.1	48.1	110.5	85.0	130.9	182.5	179.6	92.3	84.1	62.2	30.5	0.0
Copper in concentrate (Mlb)	38.5	41.7	41.2	39.5	37.7	37.3	29.6	34.4	27.8	25.2	30.5	32.9	23.2	22.0	17.6	6.4	0.0

**table uses Mineral Resources of which >95% are Mineral Reserves*
NOTE: variance to other Tables in this report is due to rounding

The price of gold and copper are key drivers in determining the pre tax cash flow above. Historically, the prices of gold and copper have been volatile. For example in the last three years the price of gold and copper have ranged from approximately US$730 - 1580 per oz Au and US$1.4 - 4.5 per lb Cu respectively.

22.1.3 Net Present Value (NPV), Internal Rate of Return (IRR) and Payback

The results below are based on the unleveraged net cash flow post tax.

Table 22.2 to Table 22.4 indicate the NPV, IRR and Payback sensitivity of the Didipio Project to gold prices and copper prices.

Table 22.2: NPV Post Tax @ 10% – Sensitivities to Metal Prices (US$M)

		Copper price - US$/lb (flat)														
Gold Price - US$/oz (flat)		2.20	2.40	2.60	2.80	3.00	3.20	3.40	3.60	3.80	4.00	4.20	4.40	4.60	4.80	5.00
	950	-8	23	51	78	104	129	151	173	194	215	236	256	276	296	316
	1000	14	42	69	96	122	145	167	188	209	230	250	270	291	310	330
	1050	34	61	89	114	139	160	182	203	224	244	265	285	305	324	344
	1100	53	80	106	132	154	176	197	218	239	259	279	299	319	339	358
	1150	72	99	125	148	170	191	212	233	253	273	294	313	333	353	373
	1200	91	116	141	163	185	206	227	247	268	288	308	328	347	367	387
	1250	108	134	157	179	200	221	241	262	282	302	322	342	361	381	401
	1300	127	151	172	194	215	236	256	276	297	317	336	356	376	395	415
	1350	144	166	188	209	230	250	271	291	311	331	350	370	390	410	429
	1400	160	181	203	224	244	265	285	305	325	345	365	384	404	424	443
	1450	175	197	218	238	259	279	300	320	339	359	379	399	418	438	458
	1500	190	212	233	253	274	294	314	334	354	373	393	413	432	452	472
	1550	205	227	247	268	288	309	328	348	368	387	407	427	447	466	486
	1600	221	241	262	282	303	323	342	362	382	402	421	441	461	480	500
	1650	235	256	276	297	317	337	357	376	396	416	436	455	475	495	514
	1700	250	271	291	311	331	351	371	391	410	430	450	469	489	509	528
	1750	265	285	306	326	346	365	385	405	424	444	464	484	503	523	542
	1800	279	300	320	340	360	379	399	419	439	458	478	498	517	537	556
	1850	294	314	334	354	374	394	413	433	453	472	492	512	532	551	570
	1900	308	329	349	368	388	408	428	447	467	487	506	526	546	565	584
	1950	323	343	363	383	402	422	442	462	481	501	521	540	560	579	598

Table 22.3: IRR – Sensitivities to Metal Prices (%)

		Copper price - US$/lb (flat)														
Gold Price - US$/oz (flat)		2.20	2.40	2.60	2.80	3.00	3.20	3.40	3.60	3.80	4.00	4.20	4.40	4.60	4.80	5.00
	950	9%	12%	15%	18%	21%	24%	26%	28%	31%	33%	35%	38%	40%	42%	44%
	1000	11%	14%	17%	20%	23%	25%	28%	30%	32%	35%	37%	39%	41%	43%	45%
	1050	13%	16%	19%	21%	24%	27%	29%	31%	34%	36%	38%	40%	40%	44%	44%
	1100	15%	18%	20%	23%	26%	28%	31%	33%	35%	37%	39%	42%	44%	46%	48%
	1150	17%	19%	22%	25%	27%	30%	32%	34%	36%	39%	41%	43%	45%	47%	49%
	1200	18%	21%	24%	26%	29%	31%	33%	35%	38%	40%	42%	44%	46%	48%	50%
	1250	20%	23%	25%	28%	30%	32%	35%	37%	39%	41%	43%	45%	47%	49%	51%
	1300	22%	25%	27%	29%	32%	34%	36%	38%	40%	42%	44%	46%	48%	50%	52%
	1350	24%	26%	28%	31%	33%	35%	37%	39%	42%	44%	46%	48%	50%	52%	54%
	1400	25%	27%	30%	32%	34%	36%	39%	41%	43%	45%	47%	49%	51%	53%	55%
	1450	27%	29%	31%	33%	35%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%
	1500	28%	30%	32%	35%	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%
	1550	29%	32%	34%	36%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%	58%
	1600	31%	33%	35%	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%	59%
	1650	32%	34%	36%	38%	41%	43%	45%	47%	49%	51%	53%	54%	56%	58%	60%
	1700	33%	35%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%	58%	60%	62%
	1750	35%	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%	59%	61%	63%
	1800	36%	38%	40%	42%	44%	46%	48%	50%	52%	54%	56%	58%	60%	62%	64%
	1850	37%	39%	41%	43%	45%	47%	49%	51%	53%	55%	57%	59%	61%	63%	65%
	1900	38%	41%	43%	44%	46%	48%	50%	52%	54%	56%	58%	60%	62%	64%	66%
	1950	40%	42%	44%	46%	48%	50%	52%	53%	55%	57%	59%	61%	63%	65%	67%

Table 22.4: Payback in Months (from Commissioning)

Gold Price - US$/oz (flat)	Copper price - US$/lb (flat)														
	2.20	2.40	2.60	2.80	3.00	3.20	3.40	3.60	3.80	4.00	4.20	4.40	4.60	4.80	5.00
950	124	74	58	49	40	36	34	32	30	28	27	25	24	23	22
1000	116	61	53	43	37	35	33	31	29	27	26	25	24	23	22
1050	68	56	47	39	36	33	31	30	28	27	25	24	23	22	21
1100	59	51	42	37	34	32	30	29	27	26	25	24	23	22	21
1150	54	46	38	36	33	31	29	28	27	25	24	23	22	21	21
1200	50	41	37	34	32	30	29	27	26	25	24	23	22	21	20
1250	44	38	35	33	31	29	28	26	25	24	23	22	21	21	20
1300	40	36	34	32	30	28	27	26	25	23	23	22	21	20	20
1350	37	35	33	31	29	28	26	25	24	23	22	21	21	20	19
1400	36	34	32	30	28	27	26	24	23	22	22	21	20	20	19
1450	35	32	31	29	28	26	25	24	23	22	21	21	20	19	19
1500	33	31	30	28	27	26	24	23	22	22	21	20	20	19	18
1550	32	30	29	27	26	25	24	23	22	21	20	20	19	19	18
1600	31	30	28	27	25	24	23	22	22	21	20	19	19	18	18
1650	30	29	27	26	25	24	23	22	21	20	20	19	19	18	18
1700	29	28	27	25	24	23	22	21	21	20	19	19	18	18	17
1750	29	27	26	25	24	23	22	21	20	20	19	19	18	18	17
1800	28	27	25	24	23	22	21	21	20	19	19	18	18	17	17
1850	27	26	25	24	23	22	21	20	20	19	19	18	18	17	17
1900	27	25	24	23	22	21	21	20	19	19	18	18	17	17	17
1950	26	25	24	23	22	21	20	20	19	19	18	18	17	17	17

22.1.4 Taxes, Royalties and Other Government Levies

The Didipio project is subject to the following taxes and royalties.

22.1.4.1 Excise Duty

The excise rate for gold is 2% on gross sales and the excise rate for copper concentrate is 2% on copper gross sales less treatment charges, refining charges, metal losses and sea freight.

22.1.4.2 Royalties

There are two sets of royalties, one at 2% of net smelter return (NSR) and the second at 0.6% of 92% times NSR, which is capped at A$13.5M. NSR means the gross income from the sale of copper and gold less treatment charges, refining charges, metal losses, sea freight, marketing and insurance cost.

22.1.4.3 Value Added Tax

The Philippines imposes a 12% value added tax (VAT) on the sale of goods and services conducted in the ordinary course of trade or business and on the importation of goods. OGC has not included VAT on the operating costs and has excluded VAT on imported goods.

22.1.4.4 Income Tax & Withholding tax

The current corporate income tax rate in the Philippines is 30%. OGC has assumed the extension of its income tax holiday certificate for a period of 6 years from commencement of operation.

The withholding tax rate in the Philippines is around 15%. The cash flows and respective NPV, IRR and Payback exclude withholding tax.

22.1.4.5 Free Carried Interest of 8%

There is a free carried interest of 8% with a claim owner, which entitles the holder to 8% of equity in the operating vehicle and dividends to be paid once OGC recovers its initial investment. A possible dividend payment is deducted from the Government share as detailed below.

22.1.4.6 Additional Government Share

In accordance to the FTAA agreement the project “Net Revenue” shall be shared on a 60/40 basis, of which 60% of the net will be the Government‟s portion and 40% will be that of the Contractor (OGPI). The Contractor shall have a period of up to five years to recover its initial investment, only after which period shall the right of the Government to share in the “Net Revenue” accrue. Contractor‟s corporate tax, excise tax, royalties, free carried interest and other taxes shall be included in the 60% Government share.

The initial investment includes not only the construction and development of the project but also payments to claim owners and landowners, exploration programs, maintenance of exploration tenement, feasibility studies, administration of offices and the net commissioning cost up to commercial production.

The following table briefly demonstrates the calculation of the additional Government share on an unleveraged cash flow.

Table 22.5: Net Revenue Calculation

1	Revenue
2	Less Operating costs
3	Less Depreciation of post development capex (excluding Underground development)
4	Less Underground mine development
5	= Net Revenue (Up to 5 years to recover the initial investment)
6	Government Share (= 60% Net Revenue post recovery of initial capex)
7	Less 2% Royalty paid
8	Less 2% Excise Duty paid
9	Less Income Tax / Withholding Tax paid
10	Less 8% Free Carried paid
11	= Additional Government Share

Note that for a leverage cash flow the Net Revenue is calculated after deducting the interest cost of the project loan.

22.1.5 Sensitivity on Other Key Value Drivers Variables

Table 22.6 presents sensitivity on total capital costs, total operating costs and removal of the income tax holiday certificate.

For every 10% change in life-of-mine capital costs, the NPV of the Didipio Project at a 10% discount rate changes by approximately US$14M. For every 10% change in life-of-mine operating costs, the NPV of the Didipio Project at a 10% discount rate changes by approximately US$36M. These cases were run with spot prices (US$1530/oz Au and US$4.05/lb Cu).

Table 22.6: NPV @ 10% – Sensitivities on Capital and Operating Costs (US$M)

Sensitivity	NPV Variation
-10% Capex	14
-10% Opex	36
Base Case	0
+10% Capex	(14)
+10% Opex	(36)
Exclude Income Tax Holiday	(17)

23 ADJACENT PROPERTIES

There are no adjacent properties that have an impact on the potential merit of the Didipio Project. The Didipio FTAA title held fully contains all known significant gold-copper mineralisation associated with the project in the area.

24 OTHER RELEVANT DATA AND INFORMATION

None.

25 INTERPRETATIONS AND CONCLUSIONS

The Open pit mining operation is relatively straight forward and the ultimate pit design is smaller than the most economic case, meaning that it is extremely robust and insulated to metal price reductions or increased operating costs. The pit wall slopes are fairly conservative so there exists some upside potential that should cancel out any localised small scale wall problems.

The underground operation is a higher level of complexity again using paste backfill, it is more exposed to metal price decreases or operating cost increases, however the selective nature of the mining method will allow mining to adjust to a degree.

The Company controls the vast majority of land in the mining footprint to support the Didipio Project; and is confident of acquiring a small number of outstanding parcels in the near term, as all parties are in negotiations. The impact on the project will be either capital cost which should be relatively minor or a delay to some areas in initial construction while negotiations are complete.

OGC was granted an Environmental Compliance Certificate in 2004, the current project requires modifications to this certificate. Consideration by the authorities is currently in progress. If these modifications are not approved the project may need to be altered.

The Philippines central government continues to be outwardly mining supportive both in general and with OceanaGold. The timely development of the project will depend on local community support. To this end OGC has been actively engagement with the community.

The capital cost estimates including the working capital allowances in the first two years have been recently prepared and construction is underway. Potential cost and time overruns should not materially affect the robust economics of the project, but refer to Table 22.6 for these sensitivities.

However it would be prudent to follow up on recommendations in section 26 to ensure total project capital is accurately considered.

OGC considers the existing Didipio database to be satisfactory for resource estimates as reported by appropriate confidence categories using the JORC/CIM guidelines, and that the resource estimates are satisfactory for the purposes of proceeding with development and construction of the project.

26 RECOMMENDATIONS

26.1 Resources

OGC has made considerable progress in capturing geological data digitally, which is now stored in an AcQuire database. Some data (e.g. rock density and oxidation) remains in Excel format.

Some further infill drilling is required to convert all resources in and adjacent to the proposed mining development area to „Indicated" status. This is expected to allow some inferred mineralisation, particularly immediately to the south of the Biak Shear, to be included in the reserves. The current interpretation of Biak Shear requires further work.

The oxidised and transitional mineralisation on the Didipio Ridge is poorly drilled. While this mineralisation makes up a small proportion of the total open pit reserves, better definition of the oxidised and transitional mineralisation could provide significant upside in the initial stages of production. Infill drilling on the ridge is recommended to test this potential.

A retrospective check assay programme for copper with appropriate QAQC might be warranted for pre OGC drilling.

26.2 Working Capital

The cash flow modelling allows for a delay in revenue receipts for concentrate post production to model the transport delay. However it is recommended to OGC to pursue advance offtake agreement for the first year of production.

26.3 Capital Costs

The author is mindful of the current inflationary environment affecting construction projects. Whilst most of the capital estimates are recent, it would be prudent to include a global capital contingency of 10%.

26.4 Exploration Programme 2011 (Provisional)

In 2008, the project was put in care and maintenance and therefore no major exploration programme has been developed since that time. A new exploration team is being assembled.

It is beyond the intended scope of this document to address the exploration potential of the FTAA outside the immediate confines of the Didipio Project.

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28 TECHNICAL ABBREVIATIONS AND NOMENCLATURE

”	feet
#	mesh
“	inches
°	degrees
ºC	degrees celsius
$	dollars
$/t	dollars per tonne
%	per cent
<	less than
=	equals
>	greater than
±	plus or minus
2D	two dimensional
3D	three dimensional
AAS	Atomic Absorption Spectroscopy
AEPEP	Annual Environmental Programme Enhancement Plans
Ag	silver
Ag_2Te	hessite
AMC	Arimco Mining Corporation
AMDAD	Australian Mine Design and Development Proprietary Limited
AMSL	above mean sea level
APMI	Australasian Philippines Mining Incorporated
AMMTEC	AMMTEC Proprietary Limited
AMSL	Above Mean Sea Level
ANFO	ammonium nitrate fuel oil (explosive)
As	arsenic
Au	gold
AU$	Australian dollars
bcm	bank cubic metres
BDA	Behre Dolbear Australia Proprietary Limited
BFS	Bankable Feasibility Study

BG	Bank Guarantee
CAMC	Climax-Arimco Mining Corporation
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CLRF	Contingent Liability and Rehabilitation Fund
CMD	clinopyroxene microdiorite
CMS	Cullen Mining Services Proprietary Limited
$C0_2$	carbon dioxide
con	concentrate
COS	coarse ore stockpile
cost/m	cost per metre
CR	control room
Cu	Copper
CSS	calc-silicate (actinolite-tremolite?)
CV	Coefficient of variation
DDH	diamond drill hole
DENR	Department of Environment and Natural Resources
DFS	Definitive Feasibility Study
DMMC	Direct Mining & Milling Costs
DWi	drop weight index
E	East
ECC	Environmental Compliance Certificate
EGL	effective grinding length
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EMB	Environmental Management Bureau
EPC	Engineering, Procurement and Construction
EPCM	Engineering, Procurement and Construction Management
EPEP	Environmental Programme Enhancement Plans
eqAu	gold equivalence
ETF	Environment Trust Fund
FEL	front end loader
FIFO	fly in, fly out

FTAA	Financial or Technical Assistance Agreement
g	gram
GIS	geographic information system
GPS	global positioning system
g/t	grams per tonne
ha	hectare
H&S	Hellman & Schofield
HG	high grade
HGC	high grade core
HLUR	Housing, Land Use Regulatory Board
HQ	diamond core 63.5mm diameter
IBC	Intermediate Bulk Containers
JORC	Joint Ore Reserves Committee (AusIMM)
K	potassium
K-feldspar-SCC	K-feldspar±sericite-carbonate-clay
kg	kilogram
km^2	square kilometre
kPa	kilopascal
kt	thousand tonnes
kV	kilovolt
kW	kilowatt
kWhr	kilowatt hour
kWh/t	kilowatt per tonne
lb	pound
LCPI	Leighton Corporation, Philippines Incorporated
Leached	carbonate-K-feldspar-muscovite±sericite-silica
LG	low grade
LHD	load-haul-dump unit
M	million dollars
m	metres
Ma	million years ago
MAusIMM	Member of the Australasian Institute of Mining and Metallurgy

mE	metres east
MGB	Mines and Geosciences Bureau
Mixed	Sericite-carbonate±silica-K-feldspar
mm	millimetres
Mm^3	million cubic metres
mN	metres north
m/s	metres per second
m^3	cubic metres
Mo	molybdenum
Moz	million ounces
m^3/s	cubic metres per second
μm	thousandth of a millimetre
MPa	megapascal
mRL	metres relative level
MRF	Mine Rehabilitation Fund
Mt	million tonnes
MTF	Monitoring Trust Fund
mtpa	million tonnes per annum
My	million years
Mya	million years ago
N	north
NATA	National Association of Testing Authorities
NIA	National Irrigation Administration
NMV	net metal value
NPC	National Power Corporation
NPV	net present value
NQ	diamond core 47.6mm diameter
NSR	net smelter return
NWRB	National Water Resource Bureau
OGC	OceanaGold Corporation
OGL	Oceana Gold Limited
OGNZL	Oceana Gold (New Zealand) Limited

OGPI	OceanaGold (Philippines) Inc
OKU	Outokumpu
ORE	ore grade
oz	Ounce
P_{80}	80% passing
Pb	Lead
PCE	pollution control equipment
PHAD	percentage half absolute difference
PDS	Project Development Study
PHD	percentage half difference
ppm	parts per million
PQ	diamond core 85mm diameter
PSE	pollution source equipment
QAQC	Quality Assurance Quality Control
QFC	quartz-feldspar-carbonate-sulphide
QFS	quartz-feldspar-carbonate-chalcopyrite-pyrite±magnetite veins
RAR	Return Air Ways
RC	reverse circulation
RCF	Rehabilitation Cash Fund
RDCL	Resource Development Consultants
rec	percentage recovery
Rec%	Recovery
RL	relative level
ROM	run-of-mine (ore)
RQD	rock quality designation
S	Sulphur
SAG	semi-autogenous grinding
Sb	Antinomy
SCADA	Standard Supervisory Control and Data Acquisition
SCC	sericite-chlorite-carbonate-sulphide alteration
SCC-K-feldspar	sericite-carbonate-clay-K-feldspar
SCC-K-feldspar-biotite	sericite-carbonate-clay-K-feldspar-biotite

SD	standard deviation
SDMP	Social Development and Management Program
SIBX	Sodium Osobutyl Xanthate
SiO_2	silicon dioxide
Skam	calc-silicate(diopside-hedenbergite)-magnetite-K-feldspar
SLC	sublevel caving
STS	Surface-Tech Surveys
t	Tonnes
tpa	tonnes per annum
t/bcm	tonnes per bank cubic metre
tpd	tonnes per day
tph	tonnes per hour
Te	tellurium
t/m^3	tonnes per cubic metre
TSF	Tailings Storage Facility
US$	United States dollars
US$/lb	United States dollars per pound
US$/oz	United States dollars per ounce
US$/t	United States dollars per tonne
UTM	Universal Transverse Mercator
VAT	Value Added Tax
VCRC	Victoria Consolidated Resources Corporation
WBG	World Bank Group
W	west
wt	weight
WTF	Waste Treatment Facility
Zn	zinc