# TECHNICAL REPORT
# FOR THE DIDIPIO GOLD-COPPER PROJECT

## Located in Luzon,
## PHILIPPINES

## Prepared for
## OceanaGold Corporation

## Level 5, 250 Collins Street
## Melbourne, Victoria
## AUSTRALIA

## 29 October 2010

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# 1 SUMMARY

## 1.1 Description of Property

### 1.1.1 Location

The Didipio Gold-Copper 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 Financial or Technical Assistance Agreement (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$^2$.

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 Gold-Copper Project, subject to satisfaction of certain conditions.

## 1.2 Geology and Mineralisation

### 1.2.1 Didipio Gold-Copper 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 following description is from Wolfe and Cooke (2004).

The Didipio Gold-Copper Deposit is hosted within the multiphase Dinkidi 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; (2) the Surong clinopyroxene to biotite monzonite pluton; (3) the Cu-Au mineralised Dinkidi Stock, which comprises an early equigranular biotite-monzonite stock (Tunja monzonite), a thin, variably textured clinopyroxene-syenite (the Balut Dyke), and a monzosyenite porphyry (Quan porphyry) that grades, in its core, into a crystal-crowded leucocratic quartz-syenite (Bufu syenite); and (4) post-mineralisation andesite dykes.

Five main hydrothermal events are recognised in the Didipio region: (1) contact metamorphism and weak biotite-cordierite alteration is associated with emplacement of the early diorite phase; (2) K-silicate magnetite-biotite alteration and sub-economic Cu-Au mineralisation associated with the emplacement of the Surong monzonite pluton; (3) intensely developed porphyry-style alteration and ore-grade Cu-Au mineralisation that is spatially and temporally associated with emplacement of the Dinkidi Stock; (4) an advanced argillic alteration assemblage, which has overprinted the Didipio Igneous Complex and is associated with sub-economic high-sulfidation style Cu-Au mineralisation; and (5) late-stage unmineralised zeolite-carbonate veins, which are associated with post-mineral strike-slip faulting.

Emplacement of the Balut Dyke was associated with a calc-potassic-style diopside-actinolite-K-feldspar-bornite alteration assemblage and associated vein stockwork. This quartz-free mineral assemblage is associated with high gold grades (2-8 g/t Au).

Intrusion of the Bufu syenite led to the formation of a quartz-sericite-calcite-chalcopyrite stockwork vein and alteration assemblage, which has overprinted the calc-potassic assemblage. The quartz stockwork hosts the bulk of low-grade mineralisation (1-2 g/t Au) at the Didipio Gold-Copper Project. A coarse-grained assemblage of quartz-actinolite-perthite (the 'Bugoy Pegmatite') formed as an apophysis on the Bufu syenite and was subsequently brecciated by late-stage faulting.

Mineralisation is directly associated with veining, which has been subdivided into a peripheral stockwork style and a centrally located, higher-temperature series of narrow diopside-actinolite-K-feldspar-bornite veins.

### 1.2.2 Didipio Gold-Copper Project Deposit Mineralisation

Chalcopyrite and gold are the main economic minerals in the deposit. Chalcopyrite occurs as fine-grained disseminations, aggregates, fracture fillings and stockwork 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. Materials-handling problems in bins, transfer chutes, conveyors and reclaim feeders are not considered to be significant.

### 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. 14,145 two-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 2540mRL, and at 1.0 g/t eqAu cut-off grade below 2540mRL and above the 2270mRL where the gold equivalence is eqAu = g/t Au + 2.23 x % Cu. This contained gold equivalence is based on metal prices of US$800 per ounce for gold and US$2.60 per pound of copper. All mineral reserves reported are included within the mineral resources reported for the same deposit.

**Table 1-1: Didipio Gold-Copper Project mineral resources at 0.4g/t eqAu and 1.0g/t eqAu cut-off**

| Class | Tonnes (Mt) | Au (g/t) | Cu (%) | Au (Moz) | Cu (Kt) |
|---|---|---|---|---|---|
| Measured | 15.58 | 1.72 | 0.57 | 0.86 | 89.4 |
| Indicated | 44.49 | 0.80 | 0.41 | 1.14 | 183.0 |
| **Measured & Indicated** | **60.07** | **1.04** | **0.45** | **2.00** | **272.4** |
| Inferred | 21.15 | 0.45 | 0.26 | 0.31 | 54.4 |

Note: Resources are reported down to 2270mRL, which represents the base of the proposed sublevel open stope mine. A cut-off of 0.4g/t eqAu has been used above the 2540mRL and 1.0g/t eqAu cut-off below the 2540mRL.

## 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 Gold-Copper Project.

## 1.4 Operations

### 1.4.1 Mining

Australian Mine Design and Development Proprietary Limited (AMDAD) has prepared a revised mine plan and estimates for the Didipio Gold-Copper Project. The revised mine plan is based on a 150m deep open cut down to an elevation of 2540mRL 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 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.

Since the underground mine will extend the mine life well beyond the open cut phase and almost all of the fill for the tailings dam wall is sourced from open cut waste, the tailings dam wall will be built to final height before completion of the open cut mine.

### 1.4.4 Mineral Reserves

The mineral reserves are summarised in Table 1-2.

**Table 1-2: Mineral reserves**

| **Open cut** | **Tonnes (Mt)** | **Au g/t** | **Au (Moz)** | **Cu %** | **Cu (kt)** |
|---|---|---|---|---|---|
| Proven ore | 6.06 | 1.23 | 0.24 | 0.74 | 44.8 |
| Probable ore | 7.81 | 0.50 | 0.13 | 0.55 | 43.0 |
| Total ore | 13.87 | 0.82 | 0.37 | 0.63 | 87.4 |
| Ore volume | 5.43 | | | | |
| Waste volume | 9.22 | | | | |
| Waste:Ore ratio | 1.70 | | | | |
| **Underground** | **Tonnes (Mt)** | **Au g/t** | **Au (Moz)** | **Cu %** | **Cu (kt)** |
| Proven ore | 5.51 | 2.62 | 0.46 | 0.53 | 29.2 |
| Probable ore | 10.30 | 1.76 | 0.58 | 0.51 | 52.5 |
| Total ore | 15.85 | 2.06 | 1.05 | 0.52 | 82.4 |
| **Total ore** | **Tonnes (Mt)** | **Au g/t** | **Au (Moz)** | **Cu %** | **Cu (kt)** |
| Proven ore | 11.57 | 1.90 | 0.71 | 0.64 | 74.1 |
| Probable ore | 18.15 | 1.21 | 0.71 | 0.53 | 96.2 |
| Total ore | 29.72 | 1.48 | 1.41 | 0.57 | 169.4 |

Notes accompanying Table 1-2:

The open cut reserves use a Net Metal Value (NMV) cut-off of US$15.00 per tonne calculated using processing, smelting and refining recoveries and processing costs, site fixed costs and realisation costs. Metal prices of US$800/oz for gold and US$2.60/lb for copper were used.

The underground reserves are based on a sublevel open stoping mine layout. The designed stope boundaries are based on a US$41.00 NMV cut-off. Metal prices of US$800/oz for gold and US$2.60/lb for copper were used.

The tonnes and grades are stated to a number of significant digits reflecting the confidence of the estimate. Since each number and total is rounded individually the columns and rows in the above table may not show exact sums or weighted averages of the reported tonnes and grades.The Qualified Person for NI 43-101 compliance with regard to the mine planning is John Wyche (BE(Min), BComm, MAusIMM(CP), MMICA), of Australian Mine Design and Development Proprietary Limited.

### 1.4.5 Schedule

Key points to note in the mine development schedule include:

- Waste mining starts 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.
- After two months of pre-production mining a level bench will be established at 2790RL. Open cut mining then ceases for one month to allow grade control drilling of the top ore benches and the oxide gold zone.
- The open cut mine operates for 71 months. The first nine months are pre-production.
- The final wall of the open cut reaches the underground portal position at 2640RL in the fourth quarter of Year 4. This marks the start point of underground mine development.

- After six months of decline development the first orebody sublevel begins and the first development ore is hauled to the ROM pad in Quarter 2 of Year 5. The open cut is still in full production at this time.
- First underground production ore is mined from the stopes in Quarter 4 of Year 6, five months after the end of open cut mining.
- The underground mine continues to operate until the end of Quarter 2, Year 20.
- Mine production is scheduled for nearly 20 years.
- The mill processes ore at 2.5 Mtpa with predominantly ore-grade feed until the start of Quarter 3, Year 6. It continues at 2.5 Mtpa for a further eight months on a combination of low-grade ore off the stockpile and underground development and production ore. From Quarter 2, Year 7 the feed rate drops back to 1.2 Mtpa using underground ore only.

Processing will occur via a gravity circuit recovering gravity gold to doré and a flotation plant producing saleable copper concentrate with relevant gold content.

## 1.5 Economic Analysis

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 here.

The financial analysis methodology, discount rates, exchange rates, commodity prices and financial parameters applied in the financial model were sourced from OGC and BDA has verified and confirmed consistency of capital costs and operating costs inputs in the model.

### 1.5.1 Cash Costs

Cash costs average US$528/oz eqAu over the life of the mine. Cash costs after copper credits over the life of the mine are US$128/oz. Gold-equivalent ounces are calculated at US$1050/oz gold and US$3.0lb/Cu.

### 1.5.2 Cash Flow Analysis

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. Table 1-3 to Table 1-5 indicate the NPV, Net cash flow and IRR sensitivity of the Didipio Gold-Copper Project to gold prices and copper prices. NPV calculation assumes 100% equity financing of the project, therefore does not have any imputed interest component.

**Table 1-3: NPV 10% – sensitivities to metal prices (US$M)**

| | Copper price - US$/lb (flat) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Gold Price - US$/oz (flat) | 2.00 | 2.20 | 2.40 | 2.60 | 2.80 | 3.00 | 3.20 | 3.40 | 3.60 | 3.80 | 4.00 |
| 800 | 36 | 53 | 70 | 86 | 102 | 117 | 132 | 146 | 161 | 175 | 189 |
| 850 | 49 | 65 | 82 | 97 | 113 | 127 | 142 | 156 | 171 | 185 | 199 |
| 900 | 60 | 77 | 93 | 108 | 123 | 138 | 152 | 167 | 181 | 195 | 209 |
| 950 | 72 | 88 | 103 | 119 | 134 | 148 | 162 | 177 | 191 | 205 | 219 |
| 1000 | 83 | 99 | 114 | 129 | 144 | 158 | 173 | 187 | 201 | 215 | 229 |
| 1050 | 94 | 110 | 125 | 140 | 154 | 168 | 183 | 197 | 211 | 225 | 239 |
| 1100 | 105 | 120 | 135 | 150 | 164 | 178 | 193 | 207 | 221 | 235 | 249 |
| 1150 | 116 | 131 | 145 | 160 | 174 | 189 | 203 | 217 | 231 | 245 | 259 |
| 1200 | 126 | 141 | 156 | 170 | 184 | 199 | 213 | 227 | 241 | 255 | 269 |
| 1250 | 137 | 151 | 166 | 180 | 195 | 209 | 223 | 237 | 251 | 265 | 279 |
| 1300 | 147 | 162 | 176 | 190 | 205 | 219 | 233 | 247 | 261 | 275 | 289 |
| 1350 | 158 | 172 | 186 | 201 | 215 | 229 | 243 | 257 | 271 | 285 | 299 |
| 1400 | 168 | 182 | 196 | 211 | 225 | 239 | 253 | 267 | 281 | 295 | 309 |
| 1450 | 178 | 192 | 207 | 221 | 235 | 249 | 263 | 277 | 291 | 305 | 318 |
| 1500 | 188 | 202 | 217 | 231 | 245 | 259 | 273 | 287 | 301 | 314 | 328 |
| 1550 | 198 | 213 | 227 | 241 | 255 | 269 | 283 | 297 | 311 | 324 | 338 |
| 1600 | 208 | 223 | 237 | 251 | 265 | 279 | 293 | 307 | 321 | 334 | 348 |

**Table 1-4: Total Life of Mine Net Cash Flow Post Tax – sensitivities to metal prices (US$M)**

| | Copper price - US$/lb (flat) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Gold Price - US$/oz (flat) | 2.00 | 2.20 | 2.40 | 2.60 | 2.80 | 3.00 | 3.20 | 3.40 | 3.60 | 3.80 | 4.00 |
| 800 | 210 | 237 | 264 | 292 | 318 | 345 | 372 | 399 | 426 | 453 | 480 |
| 850 | 236 | 263 | 290 | 317 | 344 | 370 | 397 | 424 | 452 | 479 | 506 |
| 900 | 262 | 289 | 315 | 342 | 369 | 396 | 423 | 450 | 477 | 504 | 531 |
| 950 | 287 | 314 | 341 | 367 | 394 | 422 | 449 | 476 | 503 | 530 | 557 |
| 1000 | 313 | 339 | 366 | 393 | 420 | 447 | 474 | 501 | 529 | 556 | 583 |
| 1050 | 338 | 365 | 392 | 419 | 446 | 473 | 500 | 527 | 554 | 581 | 608 |
| 1100 | 363 | 390 | 417 | 444 | 472 | 499 | 526 | 553 | 580 | 607 | 634 |
| 1150 | 389 | 416 | 443 | 470 | 497 | 524 | 551 | 579 | 606 | 633 | 660 |
| 1200 | 414 | 442 | 469 | 496 | 523 | 550 | 577 | 604 | 631 | 658 | 686 |
| 1250 | 440 | 467 | 494 | 521 | 549 | 576 | 603 | 630 | 657 | 684 | 711 |
| 1300 | 466 | 493 | 520 | 547 | 574 | 601 | 628 | 656 | 683 | 710 | 737 |
| 1350 | 492 | 519 | 546 | 573 | 600 | 627 | 654 | 681 | 708 | 735 | 763 |
| 1400 | 517 | 544 | 571 | 599 | 626 | 653 | 680 | 707 | 734 | 761 | 788 |
| 1450 | 543 | 570 | 597 | 624 | 651 | 678 | 706 | 733 | 760 | 787 | 814 |
| 1500 | 569 | 596 | 623 | 650 | 677 | 704 | 731 | 758 | 785 | 812 | 840 |
| 1550 | 594 | 621 | 648 | 676 | 703 | 730 | 757 | 784 | 811 | 838 | 865 |
| 1600 | 620 | 647 | 674 | 701 | 728 | 755 | 783 | 810 | 837 | 864 | 891 |

**Table 1-5 : IRR – sensitivities to metal prices (%)**

| | Copper price - US$/lb (flat) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Gold Price - US$/oz (flat) | 2.00 | 2.20 | 2.40 | 2.60 | 2.80 | 3.00 | 3.20 | 3.40 | 3.60 | 3.80 | 4.00 |
| 800 | 15% | 17% | 20% | 22% | 24% | 27% | 29% | 31% | 33% | 35% | 37% |
| 850 | 16% | 19% | 21% | 23% | 25% | 28% | 30% | 32% | 34% | 36% | 38% |
| 900 | 18% | 20% | 22% | 24% | 27% | 29% | 31% | 33% | 35% | 37% | 39% |
| 950 | 19% | 21% | 23% | 26% | 28% | 30% | 32% | 34% | 36% | 38% | 40% |
| 1000 | 20% | 22% | 25% | 27% | 29% | 31% | 33% | 35% | 37% | 39% | 41% |
| 1050 | 21% | 24% | 26% | 28% | 30% | 32% | 34% | 36% | 38% | 40% | 42% |
| 1100 | 23% | 25% | 27% | 29% | 31% | 33% | 35% | 37% | 39% | 41% | 43% |
| 1150 | 24% | 26% | 28% | 30% | 32% | 34% | 36% | 38% | 39% | 41% | 43% |
| 1200 | 25% | 27% | 29% | 31% | 33% | 35% | 36% | 38% | 40% | 42% | 44% |
| 1250 | 26% | 28% | 30% | 32% | 33% | 35% | 37% | 39% | 41% | 43% | 45% |
| 1300 | 27% | 29% | 31% | 32% | 34% | 36% | 38% | 40% | 42% | 44% | 46% |
| 1350 | 28% | 30% | 32% | 33% | 35% | 37% | 39% | 41% | 43% | 45% | 47% |
| 1400 | 29% | 31% | 32% | 34% | 36% | 38% | 40% | 42% | 44% | 46% | 47% |
| 1450 | 30% | 31% | 33% | 35% | 37% | 39% | 41% | 43% | 44% | 46% | 48% |
| 1500 | 30% | 32% | 34% | 36% | 38% | 40% | 42% | 43% | 45% | 47% | 49% |
| 1550 | 31% | 33% | 35% | 37% | 39% | 41% | 42% | 44% | 46% | 48% | 50% |
| 1600 | 32% | 34% | 36% | 38% | 39% | 41% | 43% | 45% | 47% | 49% | 50% |

Table 1-6 presents a sensitivity on capital costs and operating costs across the life of mine.

For every 10% change in life-of-mine capital costs, the NPV of the Didipio Gold-Copper Project at a 10% discount rate changes by approximately US$11 million. For every 10% change in life-of-mine operating costs, the NPV of the Didipio Gold-Copper Project at a 10% discount rate changes by approximately US$20 million.

**Table 1-6 : NPV 10% – sensitivities on capital and operating costs (US$M)**

| Sensitivity | NPV Variation |
|---|---|
| -10% Capex | 11 |
| -10% Opex | 20 |
| +10% Capex | -11 |
| +10% Opex | -20 |

## 1.6 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 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.

## 1.7 Conclusions and Recommendations

### 1.7.1 Exploration and Resources

The Didipio Gold-Copper 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 via infill drilling both at the top of the Dinkidi Hill as well as at depth immediately to the south of the Biak Shear.

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 was considered satisfactory for resource estimation, although some minor issues with data completeness and quality remained to be resolved.

A number of recommendations are made in Section 20 relating to additional work at Didipio.

### 1.7.2 Behre Dolbear Australia Pty Limited

BDA considers that the capital cost estimates for the Didipio Project over the next 4-5 years are generally reasonable and consistent with planned developments and are accurate within ±15%. There is a reasonable probability that some variations to capital requirements will arise later in the project life.

The OGC initial capital cost estimate is US$140.1m including VAT and contingency (see Table 1-7).

**Table 1-7: Components of capital cost estimate**

| Item | US$M |
|---|---|
| Mining | 9.0 |
| Process plant | 24.5 |
| Tailings Storage Facility | 5.0 |
| Infrastructure | 28.7 |
| Owner's costs | 3.9 |
| Indirect costs | 38.4 |
| *Subtotal* | *109.6* |
| Contingency | 19.8 |
| VAT | 10.8 |
| **Total** | **140.1** |

BDA considers the accuracy of the above estimate is within ±15%. In BDA's experience, projects of this type may incur cost overruns above the allocated contingency. BDA recommends that both initial capital costs and ongoing requirements for deferred and sustaining capital be monitored closely and estimates revised as necessary. BDA notes, however, that the economic model for the project is not overly sensitive to capital increases within the estimated range.

# 2 INTRODUCTION

## 2.1 For Whom the Report has been Prepared

This report has been prepared at the request of OceanaGold Corporation (OGC), a reporting issuer in Canada.

References in this report to ‘Oceana’ include OceanaGold 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 Ontario, Canada.

## 2.3 Sources of Information

Jonathan Moore, Principal Resource Geologist for OceanaGold, and two mining consultants have been involved in the preparation of this report. Both of the mining consultants have acted as independent, qualified persons in their respective areas. 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 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 Principal Resource Geologist.

Mr Moore is the author of the following sections of this report: 1.1,1.2, 1.3, 1.6, 1.7.1, 4.6, 6-15, 17.1-17.9, 18, 19, 20.1 and 20.3.

- No sources

### 2.3.2 Australian Mine Design and Development Pty Limited

John Wyche of Australian Mine Design and Development Pty Limited is the independent qualified person for reserve, mine design, and scheduling aspects of the report. This includes sections 1.4, 17.10, 21.1 and mining capital and operating costs for section 21.6.

- Sources were Jonathan Moore for the mineral resource model, Ausenco for processing aspects, Kevin Rosengren and Associates for geotechnical aspects, Meyer Water & Environmental Solutions for groundwater aspects, Knight Piésold for surface water aspects, Leighton Contractors Philippines Incorporated and local Filipino contractors for open cut mining costs, Orica for explosives costing and OGC for metal prices, waste characterisation and underground dewatering methodology. OGC also updated the open cut and underground mining cost schedules to reflect changes in input costs and ramp-up after the date of cost schedules prepared by Australian Mine Design and Development.

### 2.3.3 Behre Dolbear Australia Pty Limited

John McIntyre of Behre Dolbear Australia Pty Limited is the independent qualified person for sections 1.5, 1.7.2, 2, 3, 4.1-4.5, 4.7-4.9, 5, 16, 19, 20.2 and 21.2 to 21.8.

- Sections 16 and 21.2 source: GRD Minproc, Australian Metallurgical and Mineral Testing Consultants Limited (AMMTEC), Optical Metrology Limited, JKTech Pty Ltd, the technology transfer company for the Julius Kruttschnitt Mineral Research Centre, SMCC Limited, Ausenco Limited and Arccon Pty Ltd.
- John McIntyre has sourced Mr Ian White (MSc, BSc. (Hon), DIC, MAUSIMM), a senior associate of BDA, for sections 16 and 21.2.
- John McIntyre has sourced Mr Richard Frew (BE Civil, MIE Aust), a senior associate of BDA, for sections 21.3 and 21.6.
- John McIntyre has sourced Ms Janet Epps (BSc. (Geol), MSc. (Envir.)), a Senior Associate of BDA, for section 21.4.
- John McIntyre has sourced Mr Marcelo Ramos (OGC Business Development Coordinator) for sections 21.7.

The principal reports and documents reviewed by BDA are listed below.

**Public information**

- Annual and quarterly reports 2004, 2005 and 2006 – Climax Mining Limited
- Climax Mining Limited stock exchange and press announcements – 2005 and 2006

**Didipio Project**

- Didipio Project (Stage 1) Draft Preliminary Report, Mine Geotechnical Study for Open Pit Design – Geotechnica Corporation, May 1993
- 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
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**General data**

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## 2.4 Scope of Inspection

### 2.4.1 Australian Mine Design and Development Pty Limited

- A number of visits have been made to the Didipio site, most recently in February 2008.

### 2.4.2 Behre Dolbear Australia Pty Limited

- BDA undertook a site visit to the project area in March 2010.
- BDA has reviewed resource and reserve estimates, details of mining plans and schedules, metallurgical test work, proposed flow sheets, plant design, infrastructure facilities, environmental plans, implementation plans, life-of-mine plans and capital and operating costs.
- Discussions have been held with technical staff, engineering consultants and management personnel.

BDA has undertaken a site visit to the Didipio Project in 2006 and more recently in March 2010. The review included an assessment of the access road and visits to the Didipio Gold-Copper Deposit outcrop and the sites of the proposed processing plant, Tailings Storage Facility and site accommodation. In 2006, visits were also made to the Manila office of the proposed major contractor, Leighton, and the Brisbane offices of Ausenco. Geological, mining, processing and engineering data were reviewed with Climax and its consultants, CMS, and Ausenco technical and management staff. BDA has also reviewed the project updates provided in October 2010.

# 3 RELIANCE ON OTHER EXPERTS

Jonathan Moore, Principal Resource Geologist for OceanaGold, and two mining consultants have been involved in the preparation of this report. Both of the mining consultants have acted as independent, qualified persons in their respective areas. Their reliance on experts is listed below.

## 3.1 Australian Mine Design and Development Pty Limited

Mr Wyche 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.

| | |
|---|---|
| Jonathan Moore of OceanaGold Corporation | Section 21.1 in relation to the resource model used for mine planning and reserves reporting |
| Ausenco Pty Ltd | Section 21.1 in relation to process recoveries and costs used in mine planning |
| Kevin Rosengren & Associates Pty Ltd | Section 21.1 in relation to open cut geotechnical aspects |
| Meyer Water and Environmental Solutions Pty Ltd | Section 21.1 in relation to groundwater management for open cut and underground mining |
| Knight Piésold | Section 21.1 in relation to surface water management and tailings dam design |
| OceanaGold Corporation | Section 21.1 in relation to metal prices, waste rock characterisation and underground dewatering methodology |

OceanaGold Corporation also updated the open cut and underground mining cost schedules in this report to reflect ramp-up changes after the date that the models were prepared by Australian Mine Design and Development. Mr Wyche has checked the updated models and found that they are reasonable reflections of the original estimates.

## 3.2 Behre Dolbear Australia Pty Limited

Mr McIntyre has relied, and believes he has a reasonable basis to rely, on the following individuals for information in relation to ownership, tenement and licensing issues for the following areas of this report.

| | |
|---|---|
| Mr Ramoncito P. Gozar<br>VP Communication and External Affairs - OGPI | Section 21.4 |

BDA has not undertaken any legal due diligence on ownership, tenement or licensing issues. BDA relied on information provided by OGC. BDA's scope of work excludes social issues.

# 4 PROPERTY DESCRIPTION AND LOCATION

## 4.1 Location

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

The Didipio Gold-Copper 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 4-2 and Figure 4-3, 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, 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 under the previous mine plan was lodged with EMB in December 2007, but was put on hold to accommodate further additional changes in the development plan. A revised ECC, covering the most recent project design changes, is being drafted for deliberations and endorsement by the review committee prior to approval by the DENR Secretary.

## 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 Gold-Copper Project area. Subsequently, APMI changed its name to OceanaGold (Philippines), Inc. (OGPI).

APMI has an agreement (known as the "Addendum Agreement") (Jorge G Gonzales, Jerome P Delosa and David G Gonzalesch) 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).

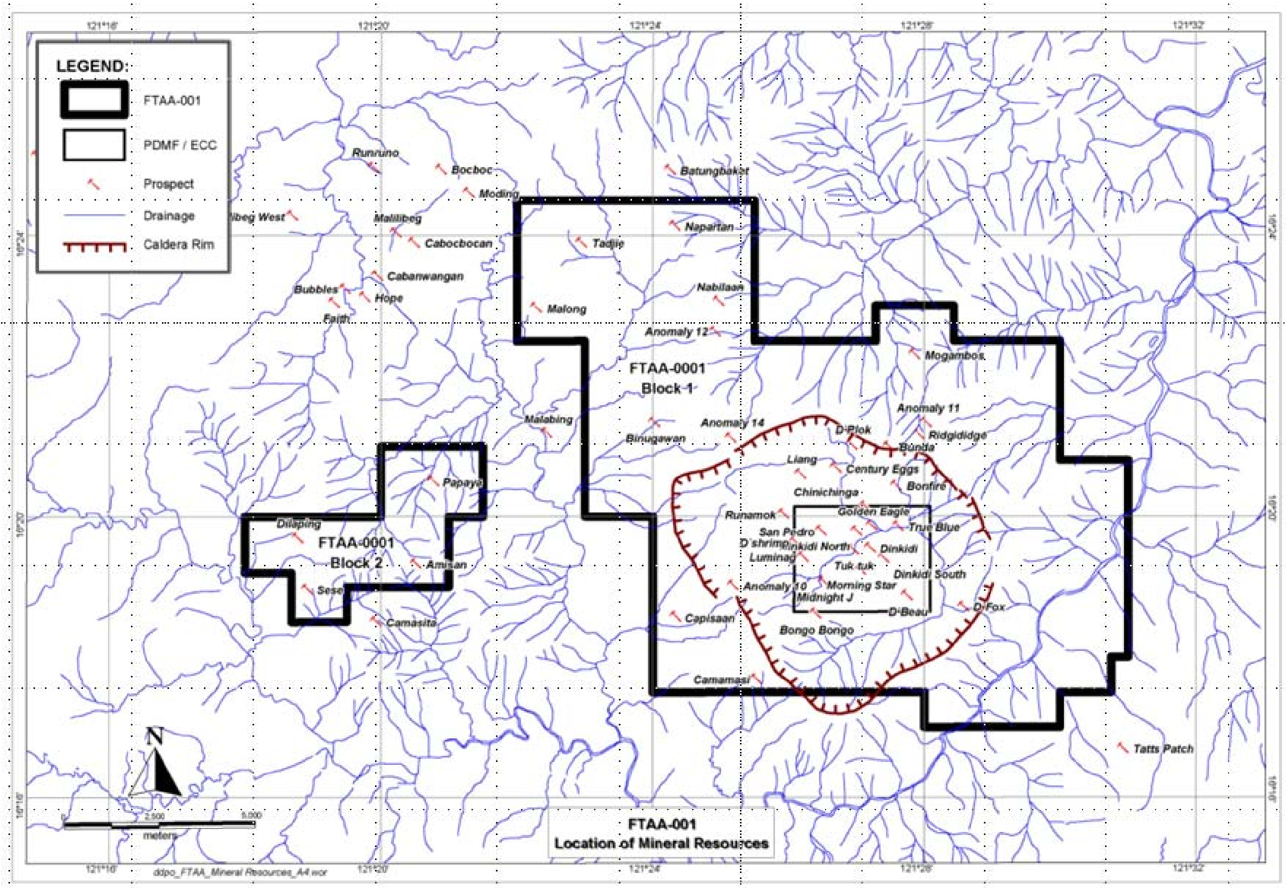
Upon the commencement of commercial production, there is a period of five years whereby the company can recover all pre-operating expenses (the Recovery Period). After this time, 60% of net revenue (net of all taxes, local payments, government payments, etc) is payable to the government as a government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

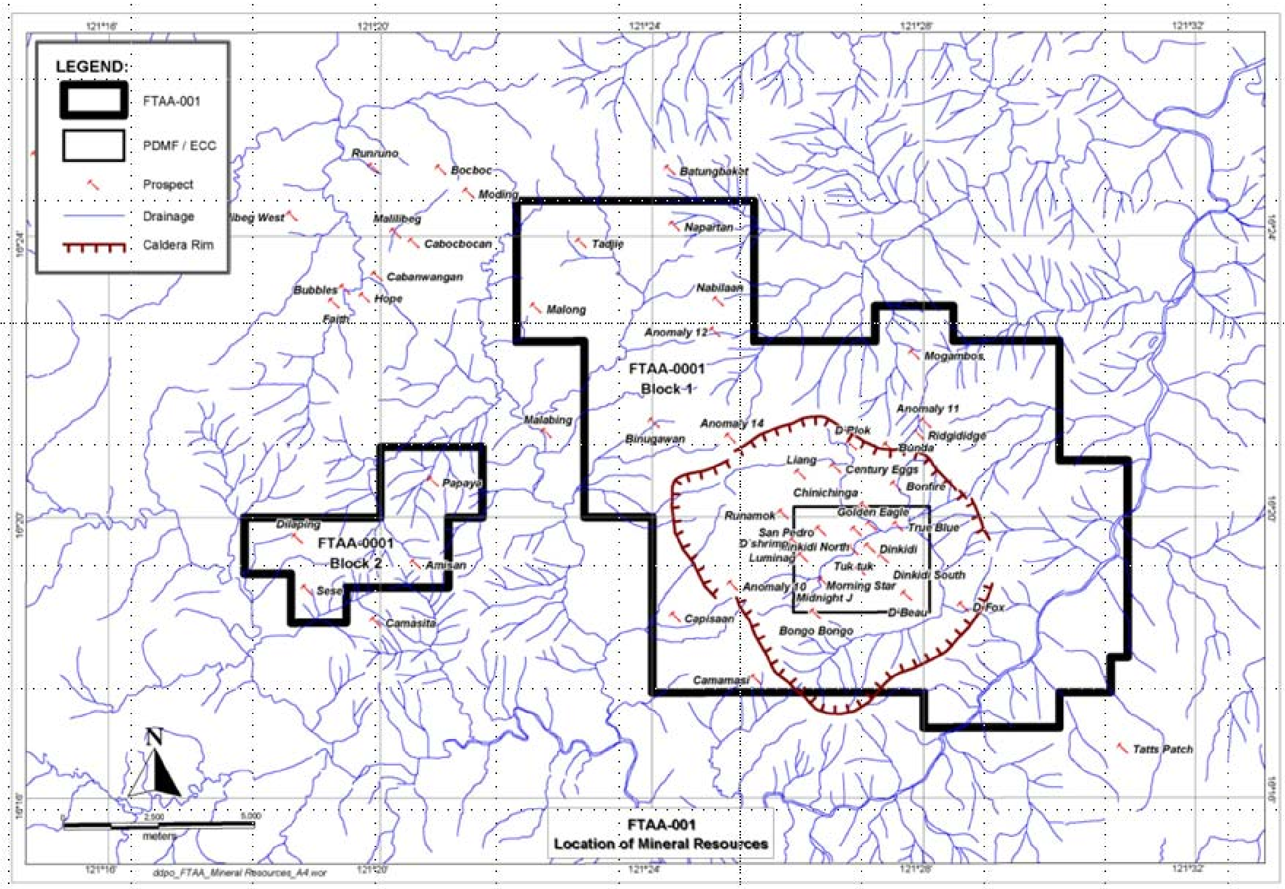
**Figure 4-1: NE corner marker for ECC**

**Figure 4-2: FTAA & ECC boundaries**



## 4.6 Location of Mineral Resources

**Figure 4-3: Location of selected prospective areas relative to FTAA & ECC boundaries**




## 4.7 Royalties, etc

The claim owner syndicate referred to in section 4.4 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% 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 Gold-Copper 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.

Section 21.4 has further details regarding environmental regulations, permitting and compliance.

## 4.9 Work Permits

The Mines and Geosciences Bureau has advised BDA that OGPI has met all the primary requirements to be fulfilled under the FTAA. Acquisition of the necessary environmental approvals and permits from the relevant government agencies has been a significant aspect of the project development. Securing the last permits and approvals required will occur when all design details have been finalised, allowing the various construction permits, and subsequent permits-to-operate, to be granted.

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

## 5.1 Topography and Vegetation

The Didipio Gold-Copper 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.

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.
- 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.

From Port Irene:

- Leave Port Irene near the Santa Ana Point, on the north-east coast, travel south along the Maharlika Highway via Tuguegarao to Cordon.
- Cordon via Cabarroguis to site.

The roads from Manila and Port Irene 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.

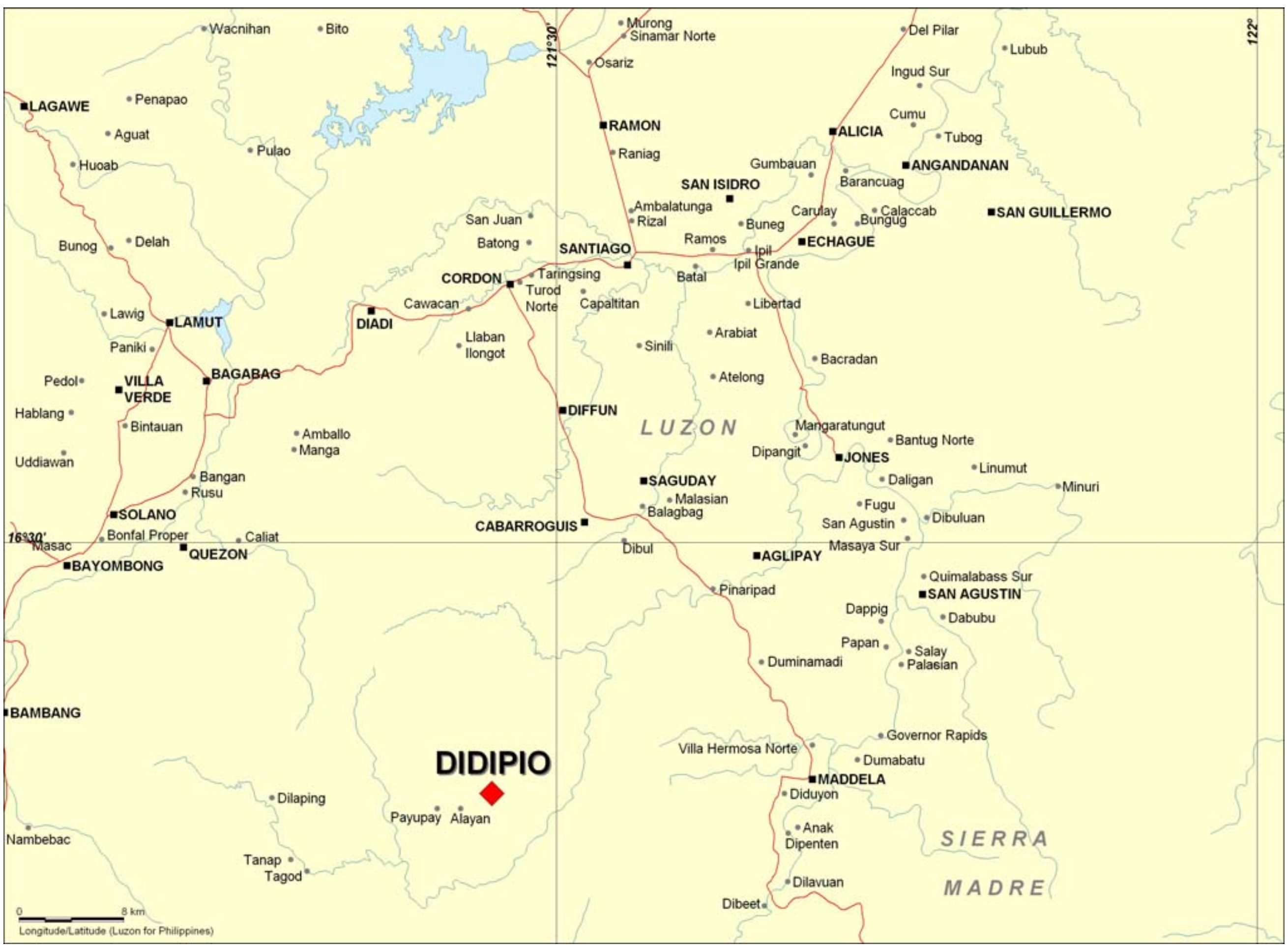
### 5.2.2 Air Access

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, 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.

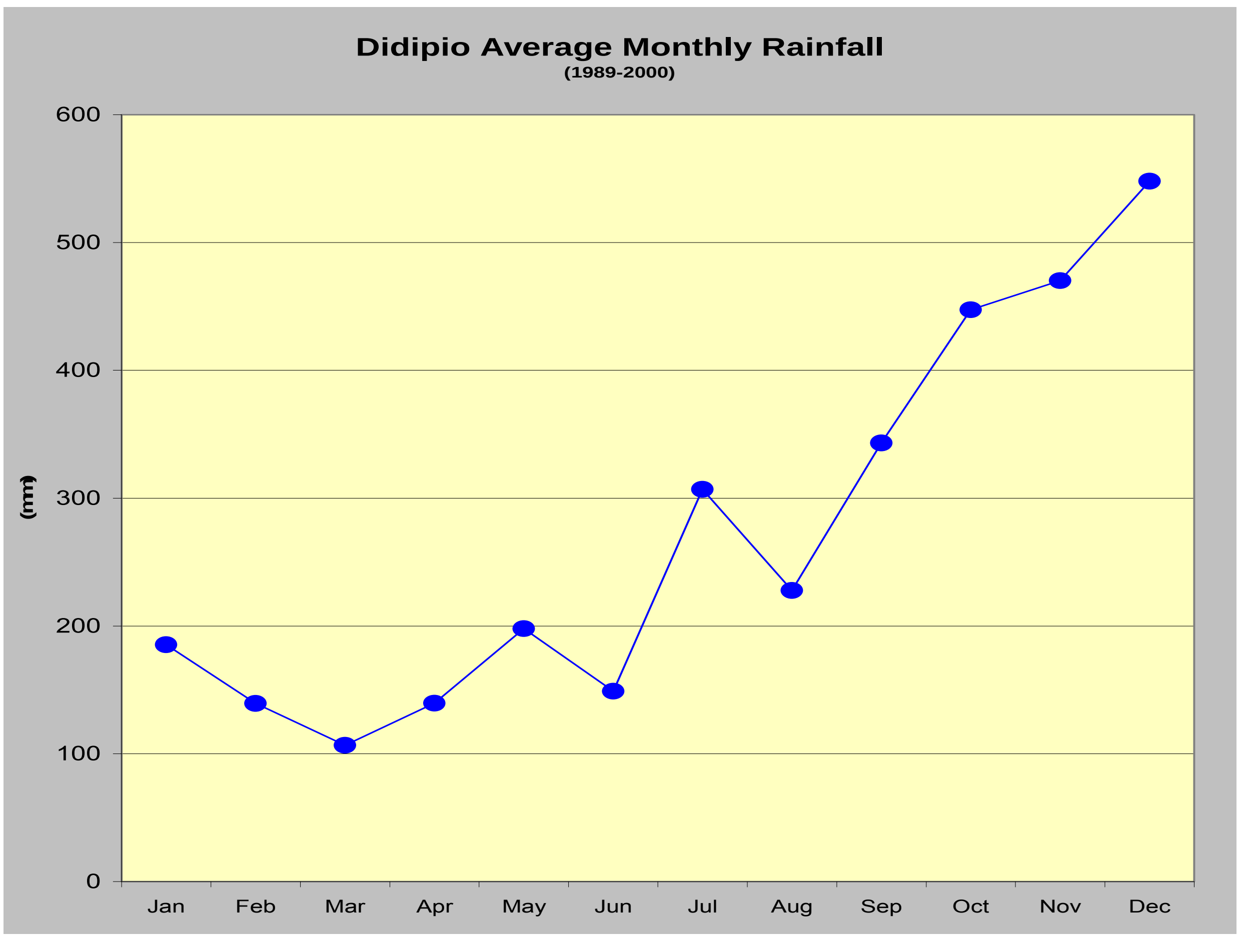
## 5.4 Climate and Operating Season

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).

**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.

<u>Rainfall modelling</u>

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 |

## 5.5 Infrastructure

### 5.5.1 Water

Water will be sourced from drawdown bores sunk around the perimeter of the open pit.

### 5.5.2 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.

### 5.5.3 Fuel Storage

OGPI has nominated an on-site fuel storage capacity of four weeks. Power generation needs would amount to about 1400m$^3$ over this period. It is assumed that the addition of the mining fleet requirements would raise this to about 3500m$^3$. Thus an allowance for this volume disposed over four tanks in a bunded, HDPE-lined tank farm with fuel handling skid and fire suppression facilities has been allowed for. This is located adjacent to the power station in the plant area, but remote from the buildings in that area. It is expected that the mining contractor would either charge his own fuel trucks directly from that facility or pipe connect to his own facility in the mining contractor's area.

### 5.5.4 Sewage

Sewage from the project site will be pumped into holding tanks for bacterial treatment.

### 5.5.5 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 disposed of either to a local waste disposal facility or will be buried in a suitable location on site. Waste oils and lubricants will be recovered and disposed of to a suitable repository, possibly in Manila.

### 5.5.6 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 – 1 x self contained one-bedroom apartment.
- Management accommodation – 2 x 8 single bedrooms with ensuites.
- Senior staff accommodation – 10 x 12 bedrooms with shared ensuite.
- Staff accommodation – 2 x 64-person, barracks-style accommodation with shared ablutions block.

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.

### 5.5.7 Port Facilities

The copper concentrate storage and shipment facilities are planed to be located at Port Irene, located on Casambalangan Bay at the north-eastern tip of Luzon Island, approximately 320km from Didipio. This site will include a 15,000 tonnes concentrate storage area.

### 5.5.8 Mine 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 seven to eight 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 plantpersonnel, will be from the local area. Contractors servicing the project will be encouraged to follow a similar employment policy. Recruitment of senior management will require time and will need to commence as soon as a decision to develop is made.

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.

# 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 hydraulicking 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 Gold-Copper 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 Gold-Copper 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 Feasibilty 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 production by artisan miners.

# 7 GEOLOGICAL SETTING

## 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 is 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 Dinkidi 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 and trachyandesite?) 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 Dinkidi Stock; and
4. Post-mineralisation andesite dykes.

**Figure 7-1: Northern Luzon – major geological subdivisions and structural elements**

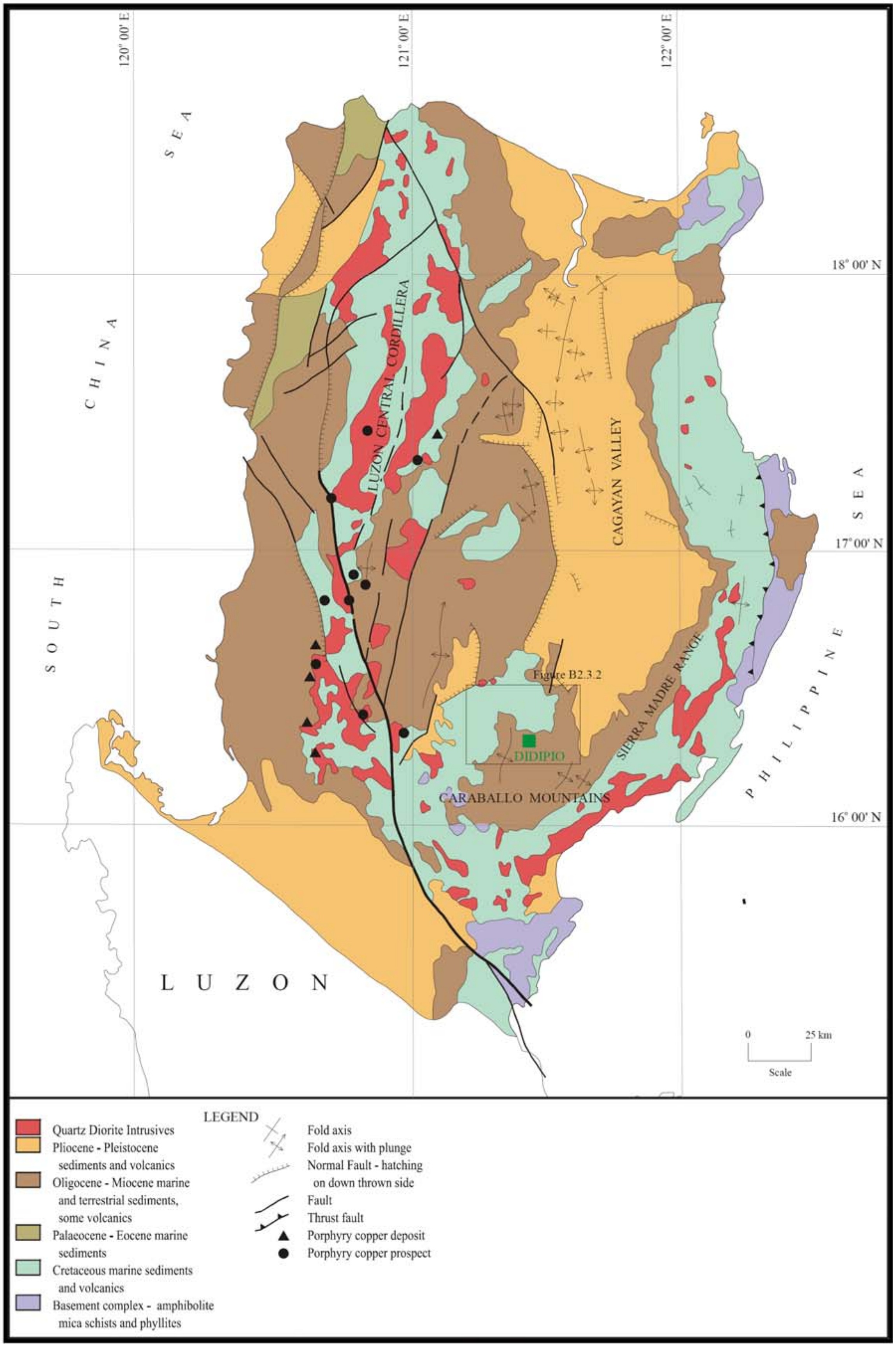
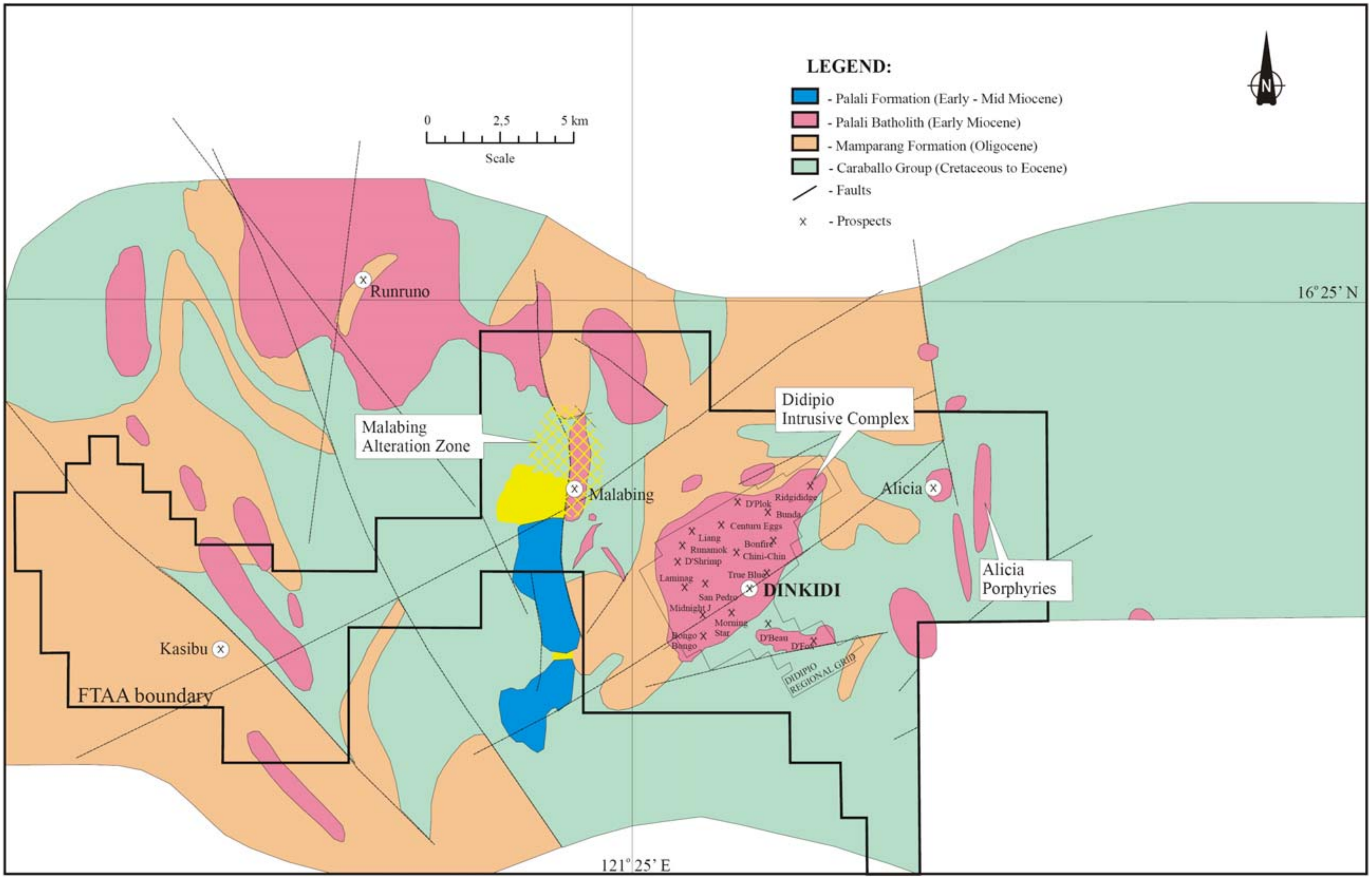


**Figure 7-2: Regional geology**



## 7.2 Local Geology

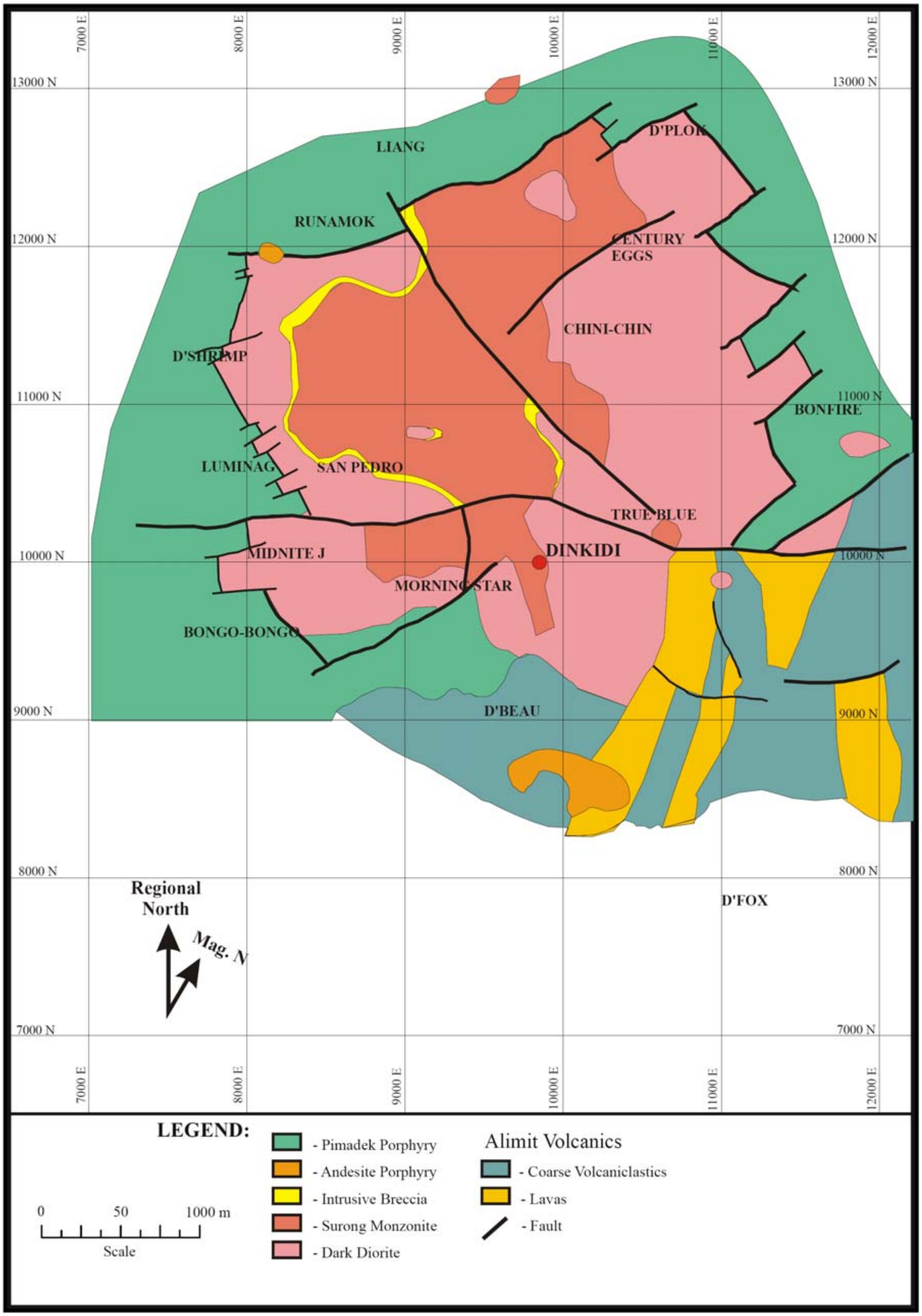
The Didipio Gold-Copper Project has been identified an alkalic gold-copper porphyry system, roughly elliptical in shape at surface (450m long by 150m wide) and with a vertical pipe-like geometry that extends to at least 800m to 1000m below the surface.

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**

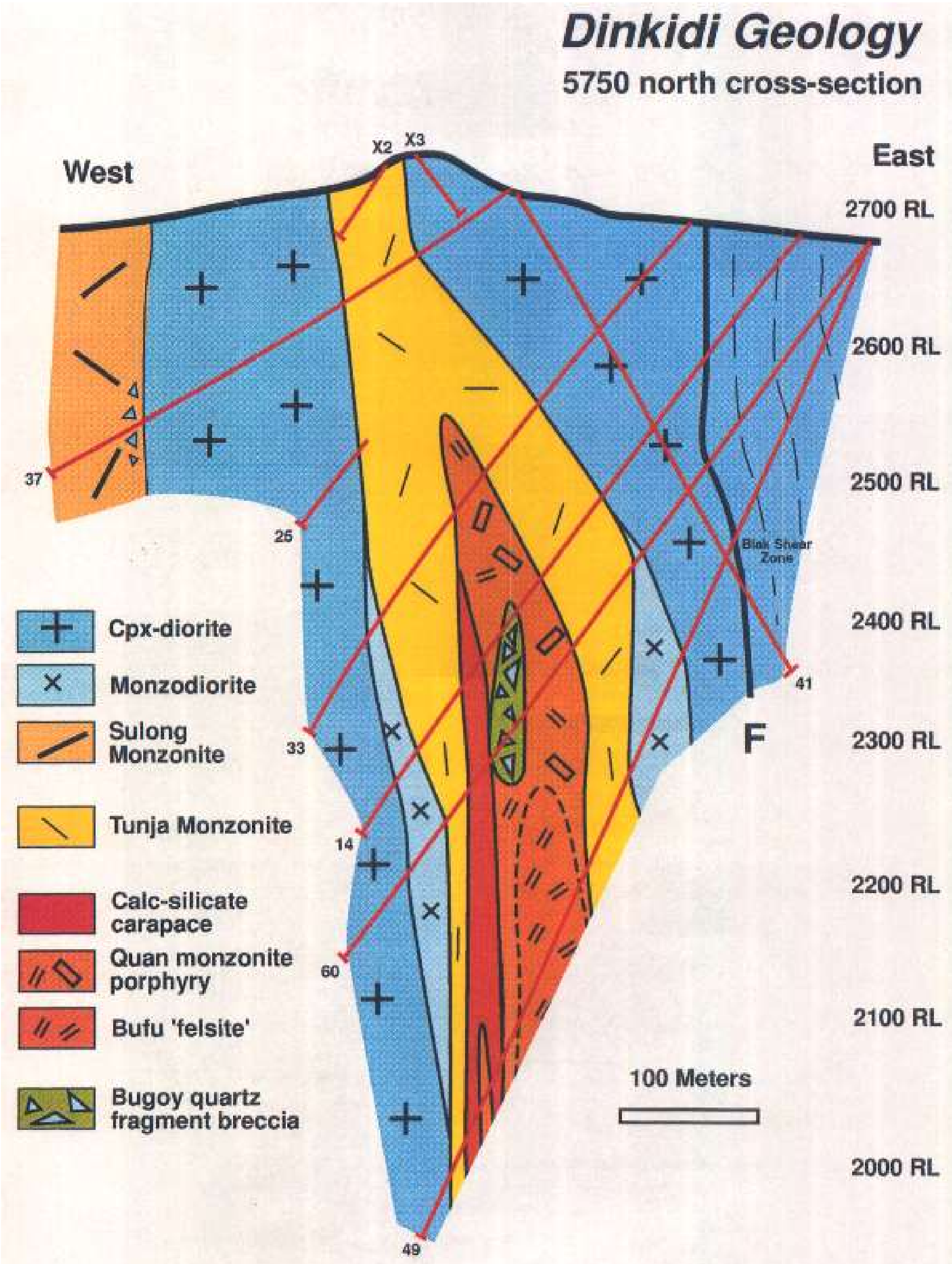


## 7.3 Deposit Geology

### 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).

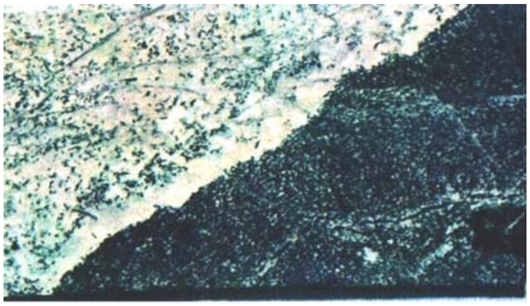
**Figure 7-4: Didipio Gold-Copper Project geology**




## 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



**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).

**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.

**Bufu**

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

**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.

**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 Gold-Copper 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 Gold-Copper Project-style porphyry gold-copper alteration-mineralisation**

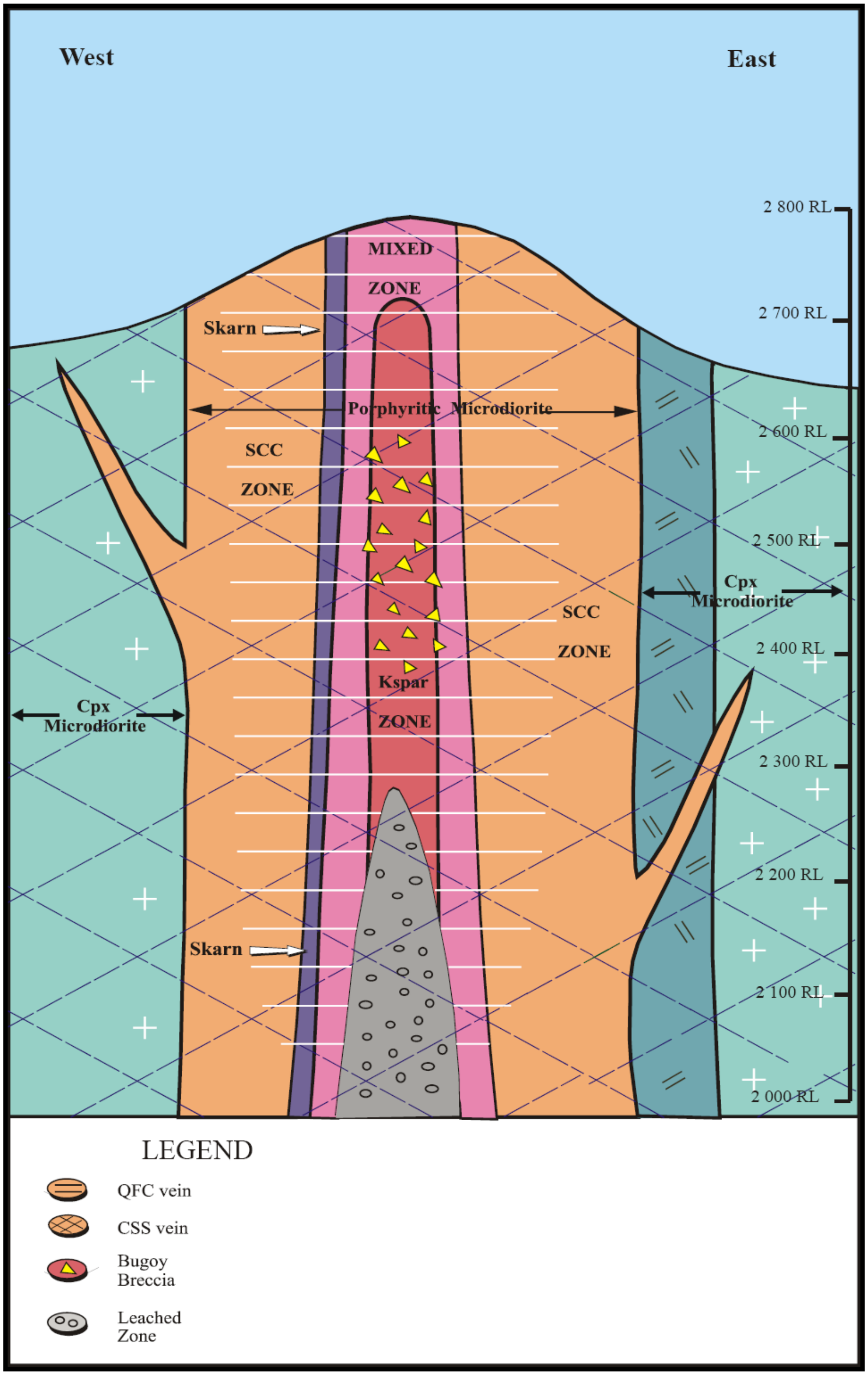
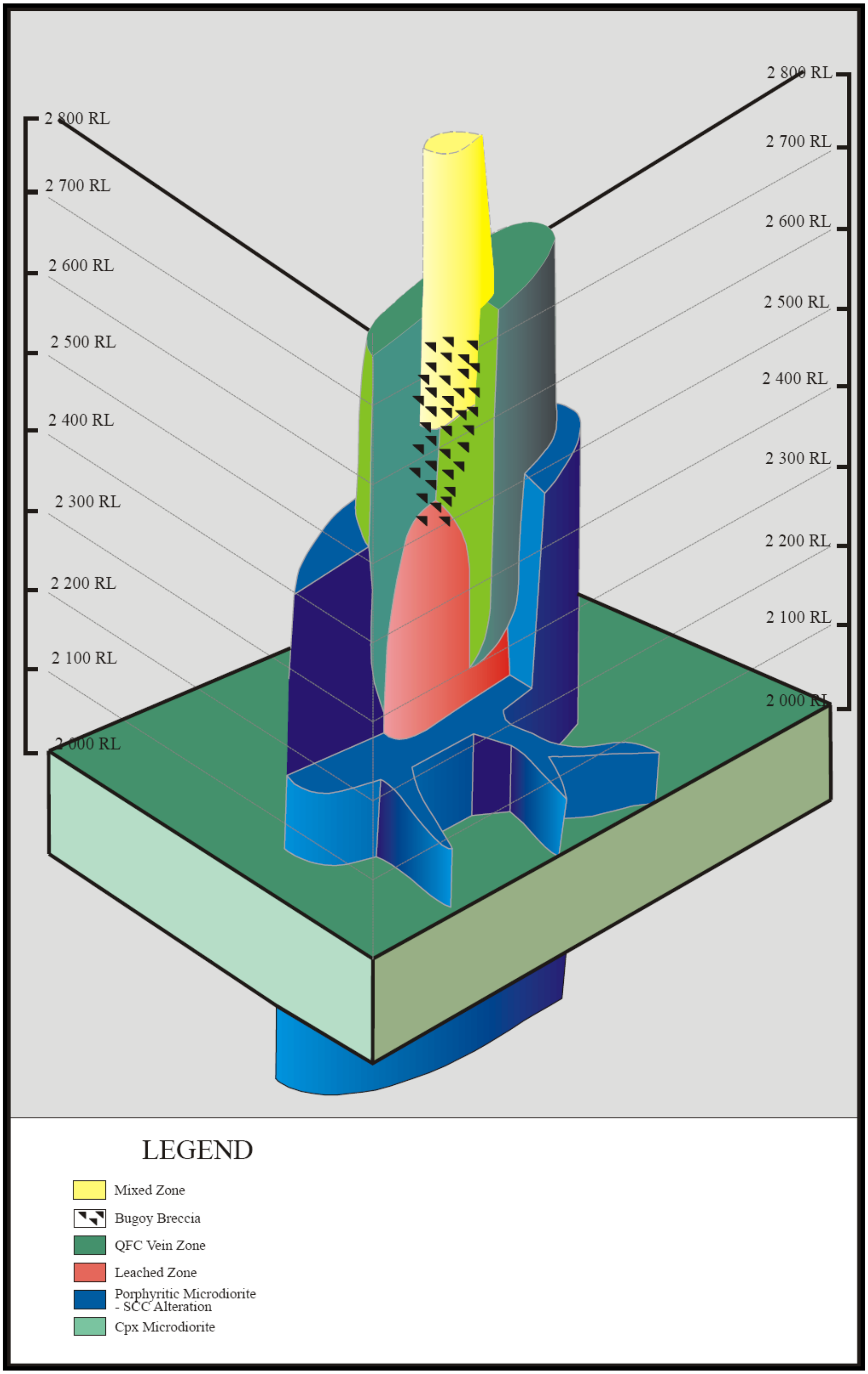


**Figure 7-7: Illustration of 3D relationships between rock types and alteration – Didipio Gold-Copper Project Deposit**



### 7.3.3 Faulting

The main fault structures within the Didipio Gold-Copper 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, northwesterly 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.

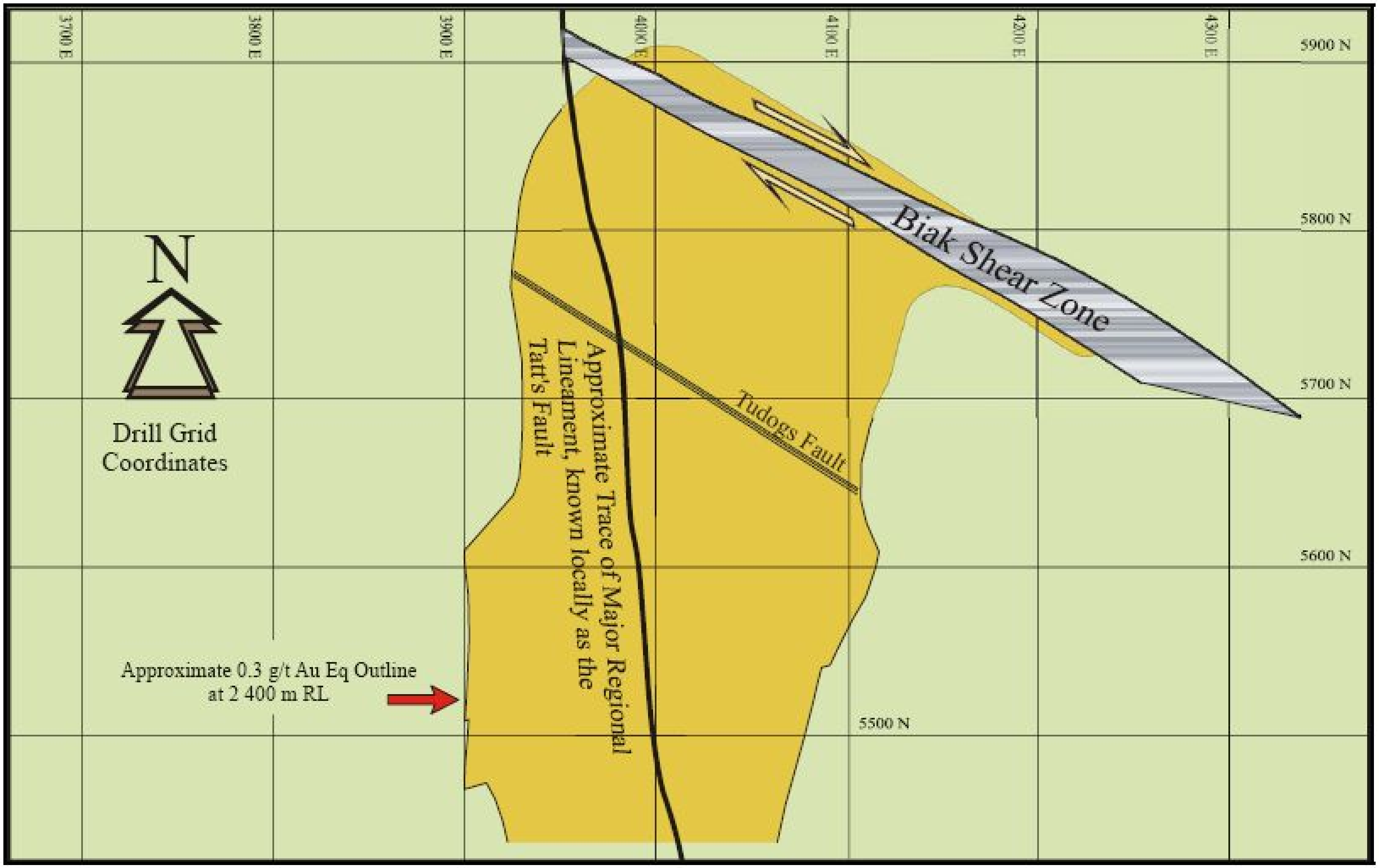
Three major faults have been named and their relative locations are shown in Figure 7-8.

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 appear 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.

# 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

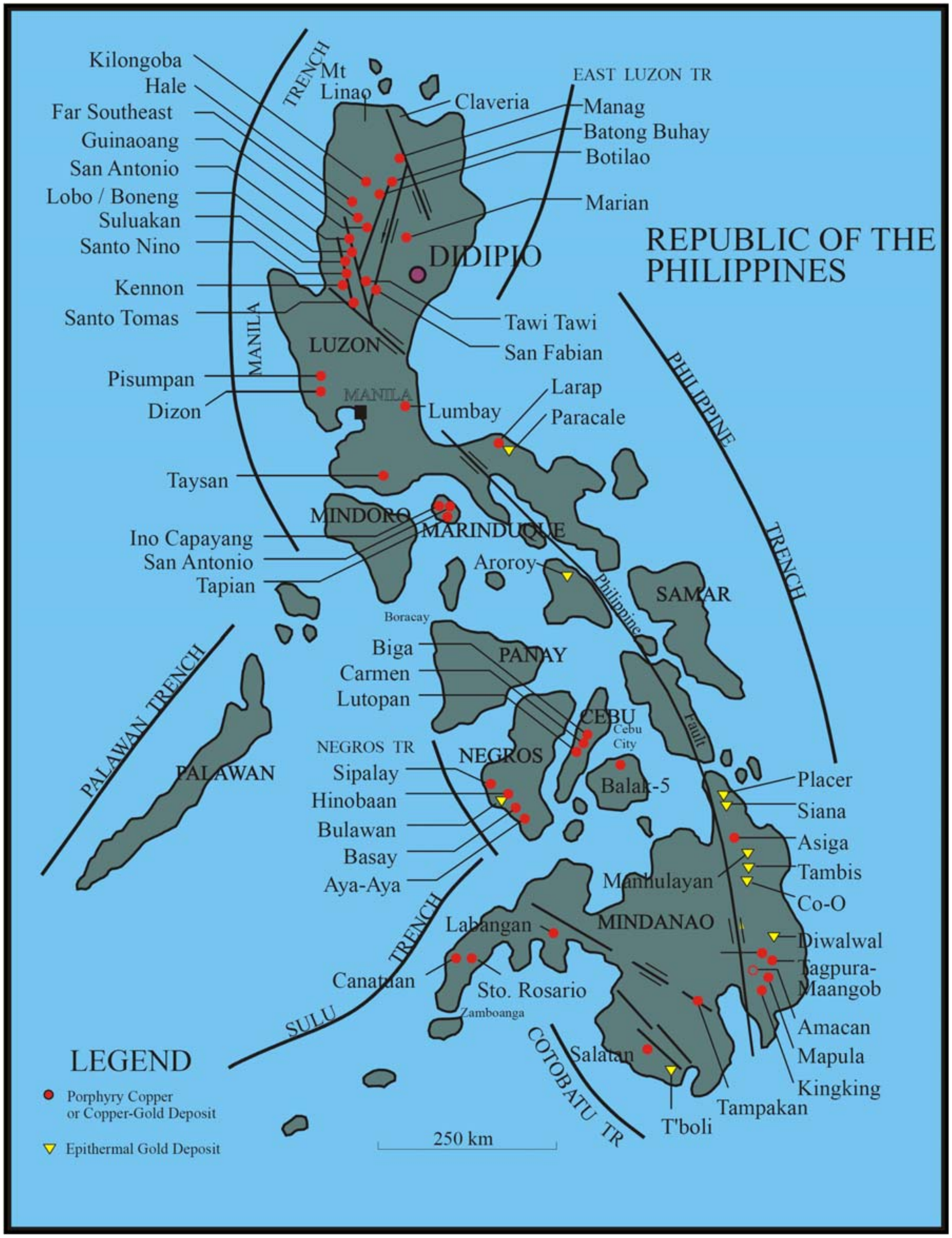
In a comprehensive review, Sillitoe and Gappe (1984) reported the characteristics of 48 mineralised predominantly calc-alkaline porphyry deposits in the Philippines, many of which have been in production (see Figure 8-1). The size of the deposits varies from 50 Mt to more than 300 Mt and copper grades 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 Gold-Copper Project monzonite suite and its associated mineralisation.

# 9 MINERALISATION

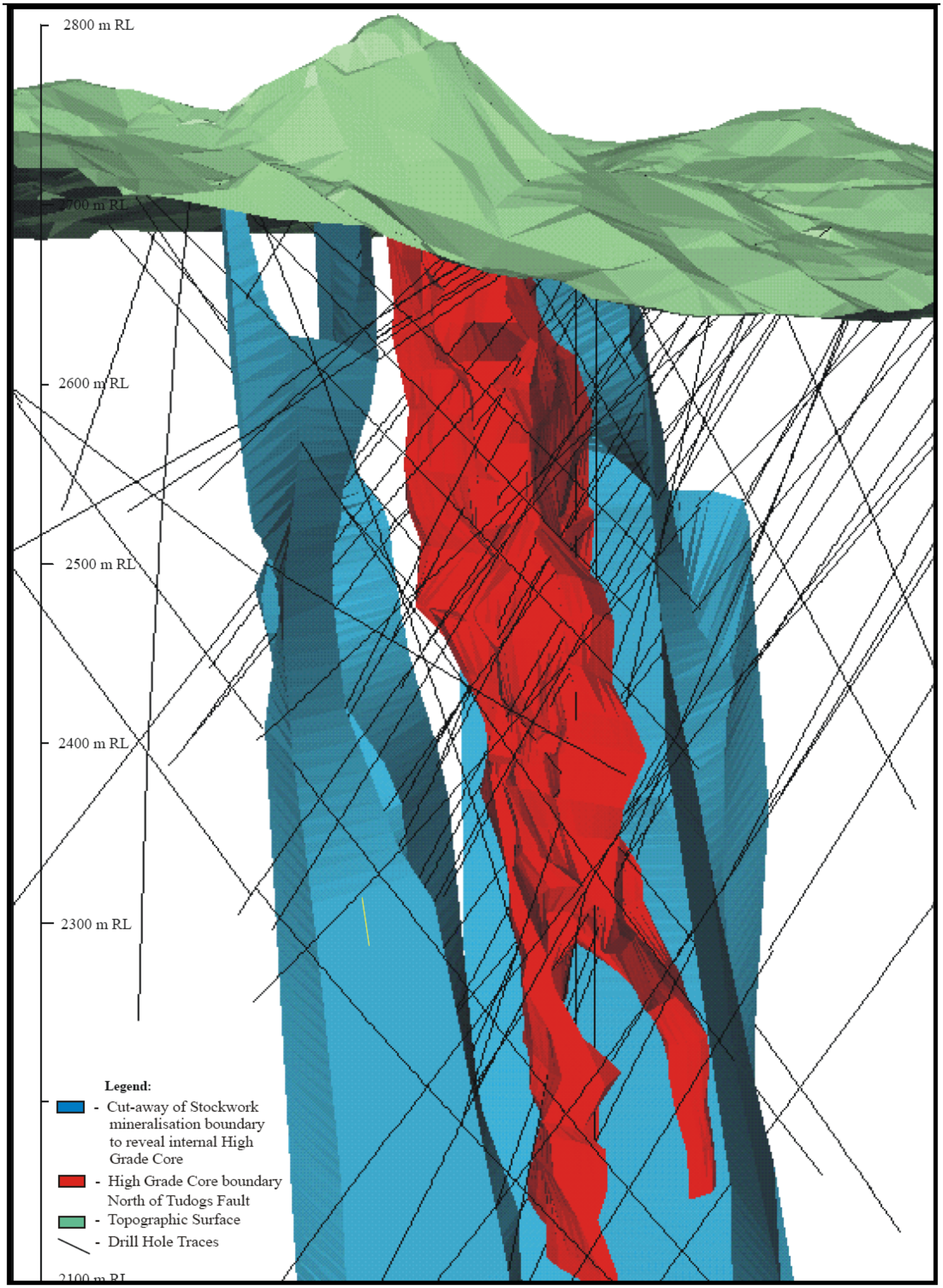
## 9.1 Geometry of Mineralisation

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 9-1.

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 9-1: Three-dimensional view of the Didipio Gold-Copper Project Deposit mineralisation outlines (north facing)**



## 9.2 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 Gold-Copper Project.

## 9.3 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 Gold-Copper 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 9-2: Didipio Gold-Copper Project Deposit – plan at 2300mRL showing geology and mineralisation outlines**

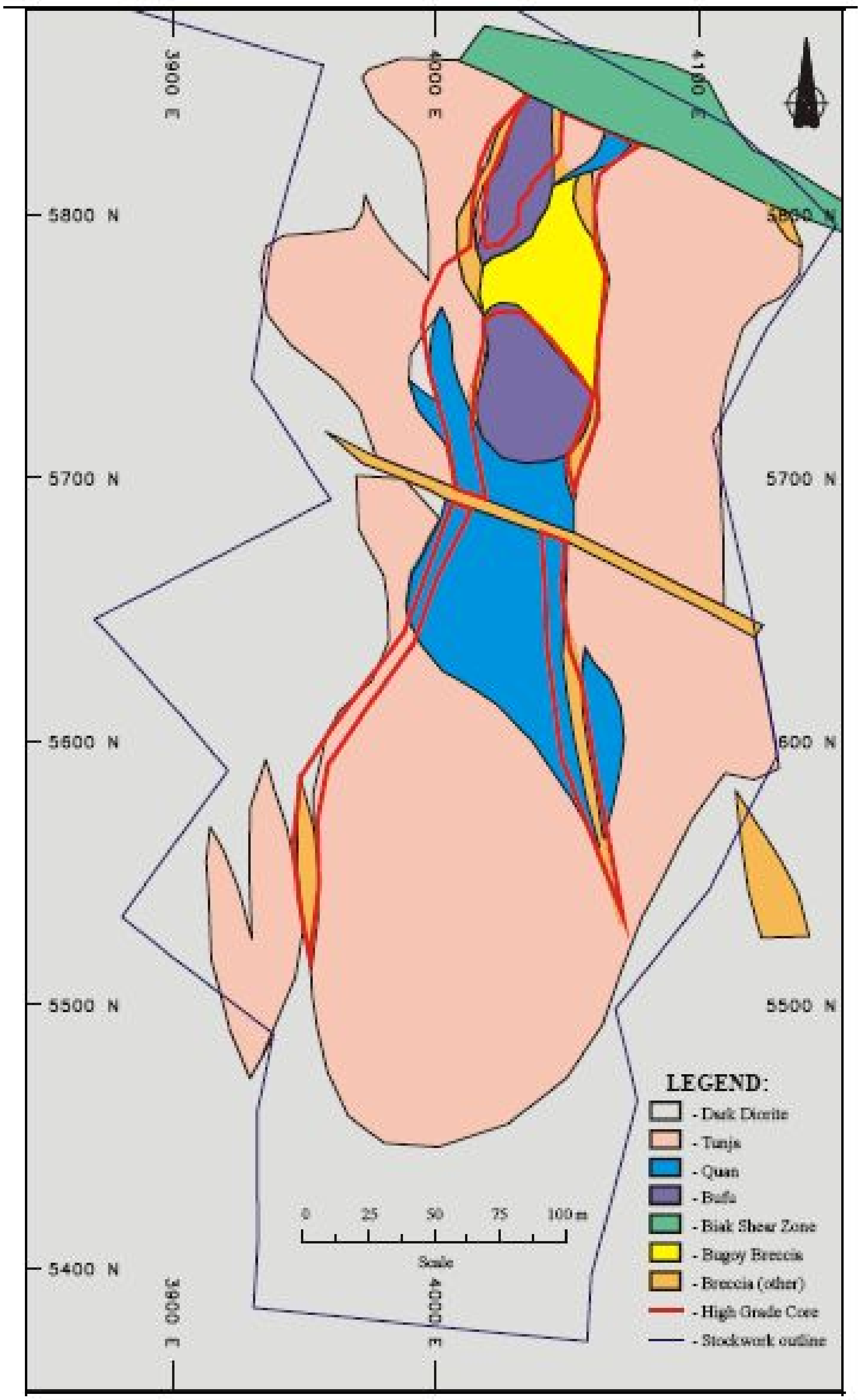


**Figure 9-3: Didipio Gold-Copper Project Deposit – geological cross-section 5800 N**

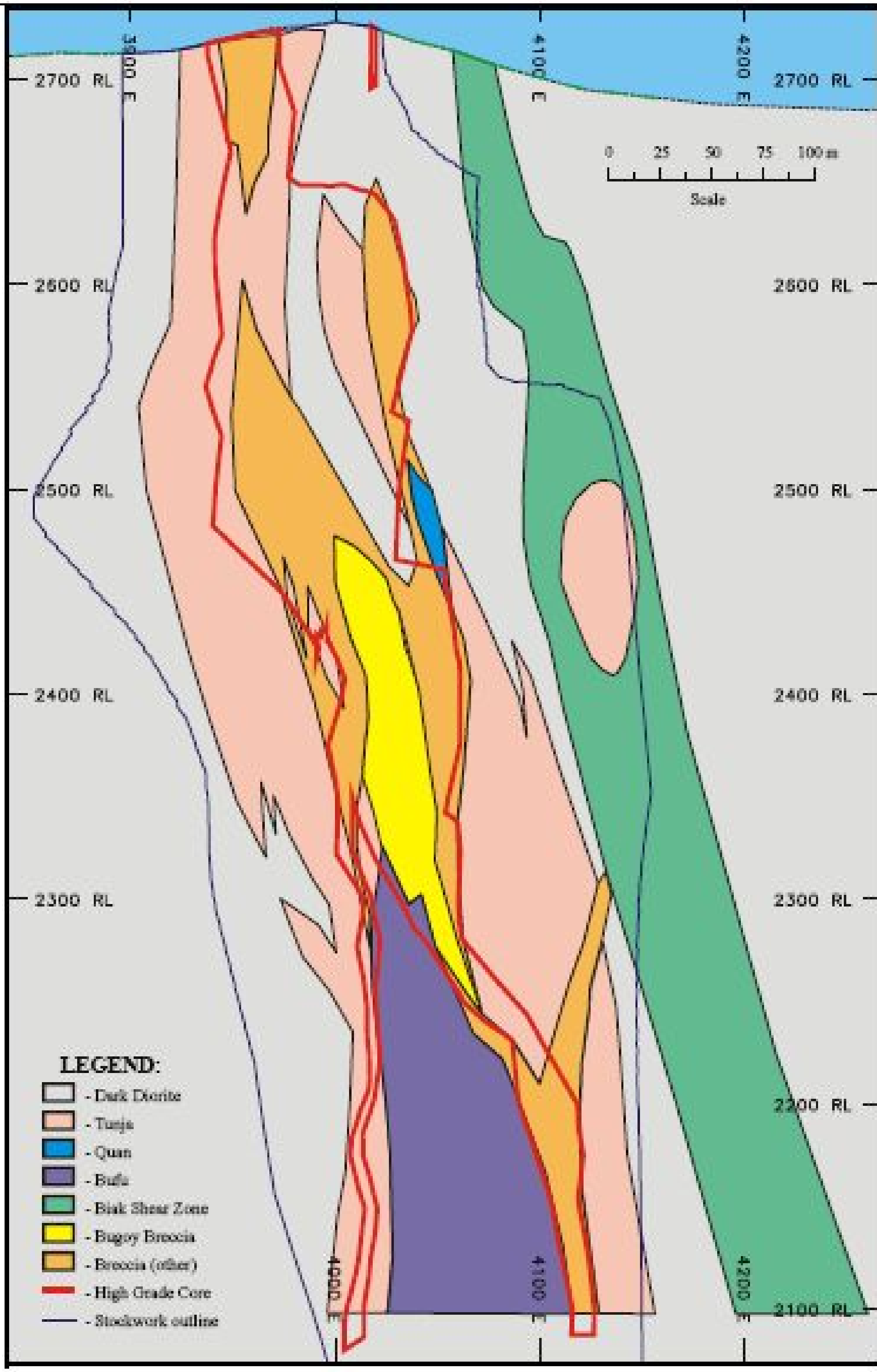
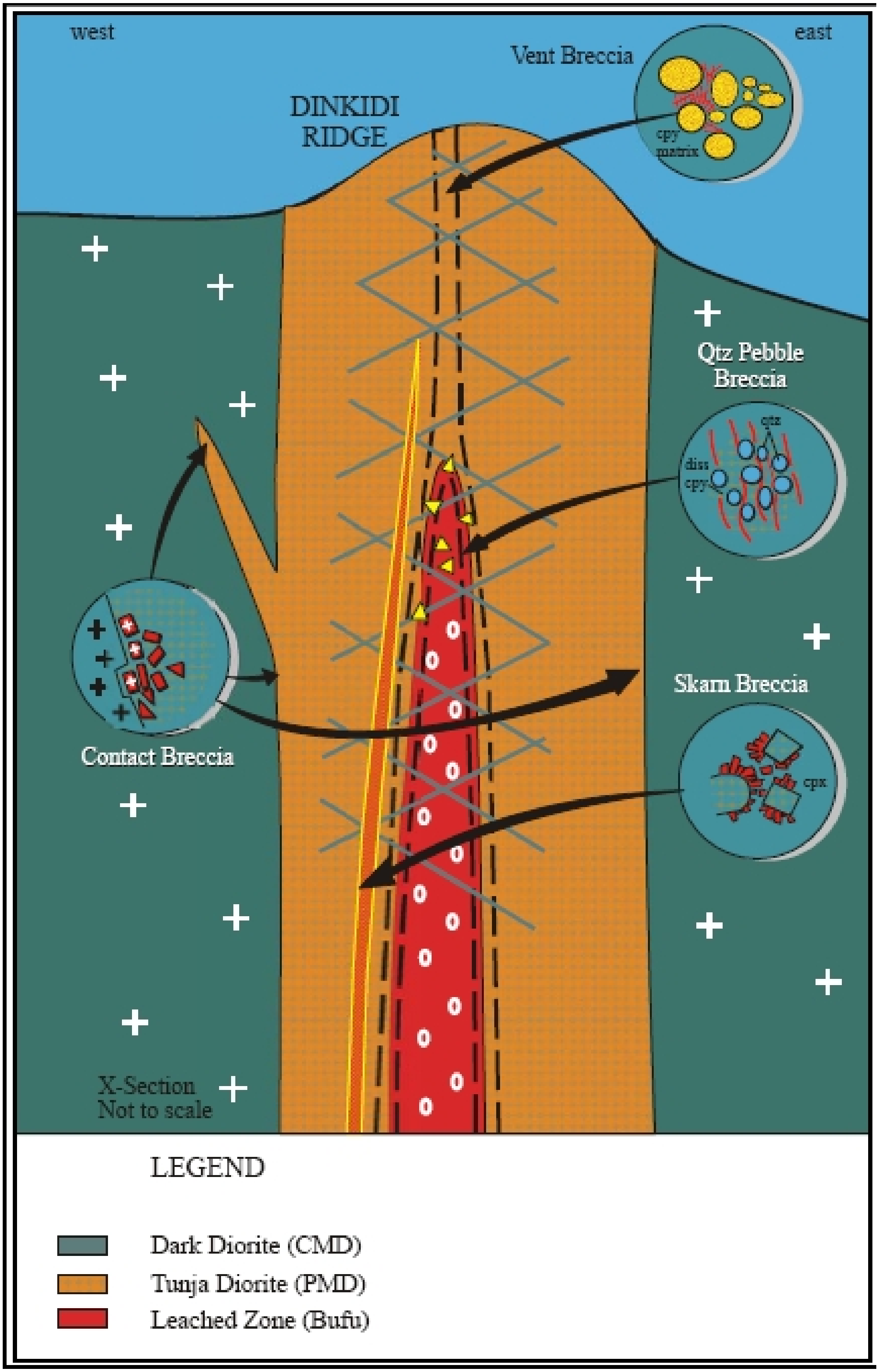


**Figure 9-4: Stylised cross-section showing breccia fabrics associated with alteration and mineralisation**



# 10 EXPLORATION

## 10.1 Exploration Work and Results

The exploration history of the Didipio Gold-Copper Project is detailed in sections 6 and 11 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.

## 10.2 Interpretation

Exploration has defined a substantial gold-copper resource at Didipio. The resource estimate is detailed in Section 17. 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.

## 10.3 Details of Operators

All drilling at Didipio has been performed by contractors, as detailed in section 11, while most of the sample preparation was performed by Climax personnel at Cordon and assaying by Analabs (see section 13). Samples taken during the 2008 infill drilling programme were prepared and analysed by McPhar Laboratories of Manila.

Relevant aspects of field surveying are discussed below to complement the exploration history of the Didipio property detailed in section 6.

Historically, three grids have been used in the collection of survey data within the Didipio Gold-Copper Project area (see Figure 10-1).

### 10.3.1 National Grid

The National Grid, known as the Philippine Transverse Mercator, is based on Universal Transverse Mercator (UTM) coordinates and is used in all national mapping.

All future activities in proximity to the Didipio Gold Copper Project will use the National Grid. This includes the October 2008 resource estimate detailed in this report. The previous, Hellman and Schofield resource estimate was calculated in the local Drill Grid, discussed in section 10.3.3 below.

### 10.3.2 Regional Grid

This grid was set up by Climax, with its northing orientation 30° west of true north (UTM), and 10 000 N, 10 000 E located in the vicinity of the Didipio Ridge. Historically, it has been assumed that magnetic declination is negligible and that true north equates closely to magnetic north.

This Regional Grid has been used to explore outlying areas. Grid lines have been cut in an east-west direction and pegs showing coordinates have been placed at convenient intervals. The vertical coordinate used by Climax approximates to the UTM elevation with the addition of 2000m. Surface elevations range between 700m and 820m AMSL.

### 10.3.3 Drill Grid

Originally, this grid was centred on the Didipio Gold-Copper Project with grid north parallel to the ridge axis, i.e. 21° to the west of the Regional Grid or 51° west of true north. A coordinate correspondence for transformation purposes is shown in Table 10-1 and has been adopted for all resource estimation prior to the current OGC estimate.

**Table 10-1: Didipio grids – coordinate comparison**

| | Drill grid | | Regional grid | |
|---|---|---|---|---|
| | Northing | Easting | Northing | Easting |
| Point 1 | 5 264.00 | 4 000.00 | 9 714.00 | 9 900.00 |
| Point 2 | 5 495.17 | 4 195.85 | 10 000.00 | 10 000.00 |

During 1992, a geodetic engineer placed nine concrete monuments on high points around the Didipio site and connected Monument 7 of the series to the Regional Grid and the National Grid using benchmarks located approximately 10km from Didipio.

Geographical and UTM values for Monument 7, adopted for the GRD 1998 DFS, are as follows in Table 10-2.

**Table 10-2: Geographical and UTM coordinates – Monument 7**

| Geographical | | UTM (Philippines Transverse Mercator) | |
|---|---|---|---|
| Latitude | Longitude | North | East |
| 16° 19' 37.50"N | 121° 27' 10.65"E | 1 805 691.87 | 548 401.14 |

Surface-Tech Surveys (STS), a company based in Perth, Australia, was retained from April 1994 to July 1995 to locate the original Monument 7 and to pick up all existing data points, such as drill holes, pits, trenches, tunnels and adits.

STS verified the historical positions of existing monuments 1, 2, 4, 6, 7 and 9 and recorded differences of up to 1m in the historical levels. Approximately 900 secondary control stations were established in the Didipio area to form what can be regarded as a secondary control network, allowing resultant accuracies to be of the order of 1:2500 horizontally and 1:5000 vertically.

Using four of the original concrete monuments close to the Didipio Ridge, a control framework was established which was extended approximately 5km along the road to Tucod and 3.5km along the Dinauyan and Surong Rivers. Photo targets adjacent to these traverses were coordinated, thus controlling the nominated areas required for a planned aerial survey. A topographic contour map based on the STS data is shown in Figure 10-2.

The traverse along the road to Tucod was extended a further 6km back to the two National Grid benchmarks used to establish the geographical and UTM coordinates for Monument 7. Data for these benchmarks received from the Bureau of Land Management (Philippines) contains horizontal values but no levels. Hence, an assumed height of 823.00m, derived from regional topographic contour plans, was assigned to Monument 7 by STS. The height difference to the National Grid is not expected to be greater than ±20m and the value does not affect the integrity of the internal survey control at Didipio. Resulting back-calculation of the latitude and longitude values for Monument 7 have agreed to within half a second of arc with the original quoted values shown above.

When using the Regional Grid or the Drill Grid, 2000m has been added to the elevation to obviate the problem of having to use negative values when referring to RL figures below mean sea level for drill intersections or underground mine openings.

All coordinate data since May 1994 have been generated on, or transferred to, the Drill Grid, which has been accepted as the standard for the deposit. With the work completed by STS, the topographic database and drill hole collar locations up to drill hole number DDDH65 had been surveyed to acceptable industry standards. Drill holes DDDH66 to DDDH83 were drilled subsequent to the STS survey. These drill hole collar locations have been surveyed by Climax using compass and tape from local secondary control stations.

**Figure 10-1: Relationship of survey grids in Didipio area**

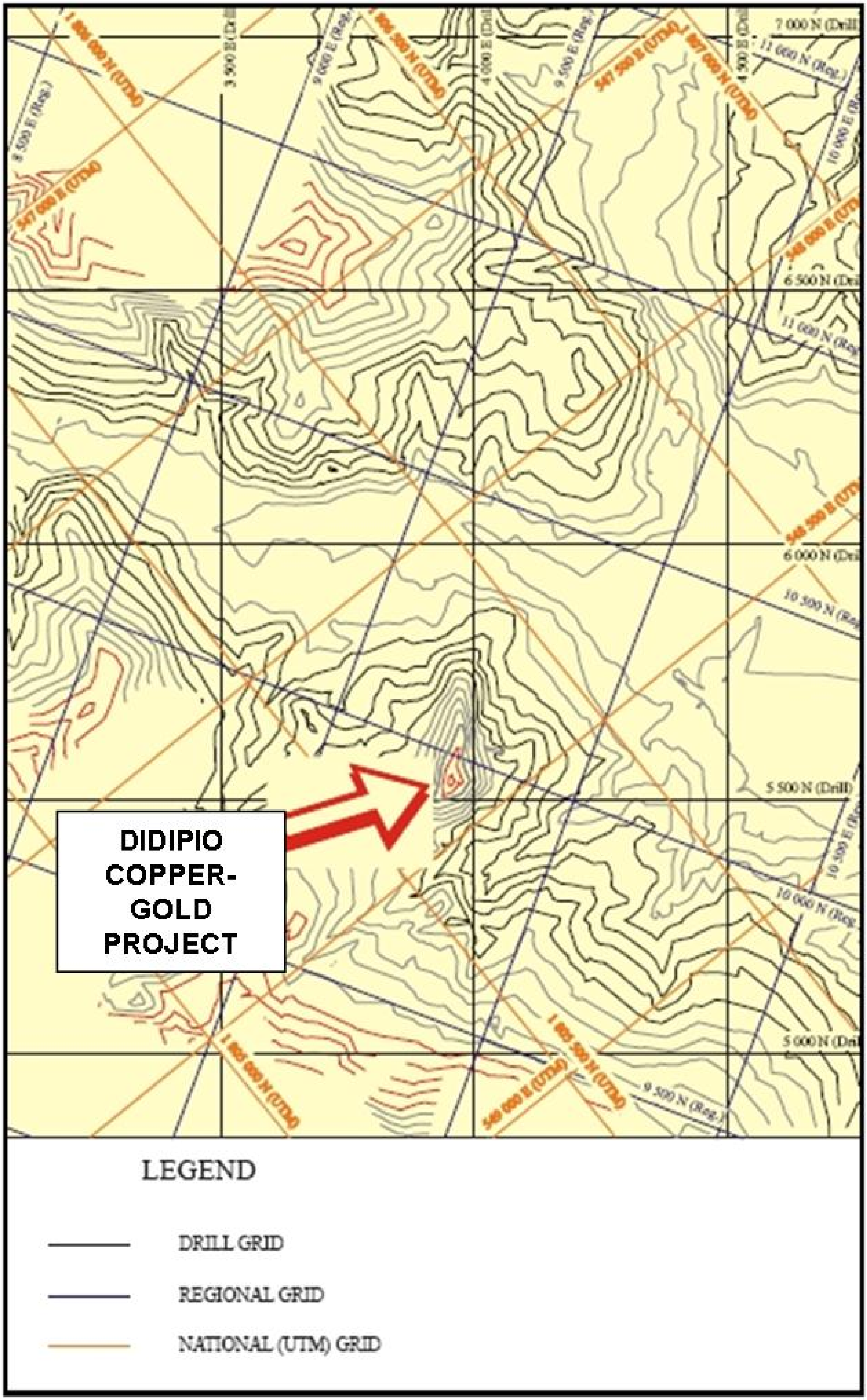
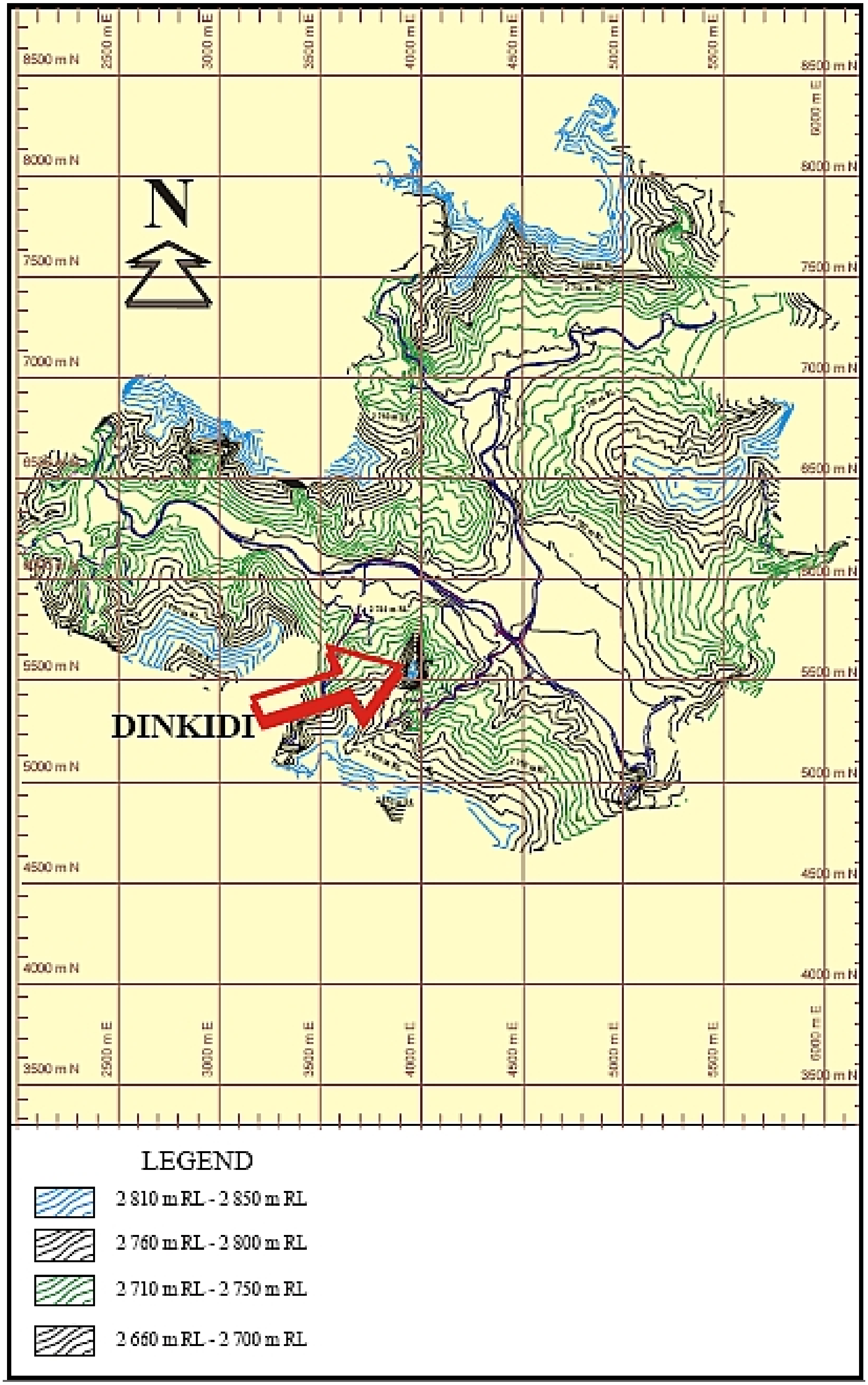


**Figure 10-2: Topographic contour plan – Didipio Gold-Copper Project Deposit**



# 11 DRILLING

## 11.1 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. The holes excluded from resource estimation are either percussion holes drilled for geotechnical purposes or small diameter (Winkie) core holes with poor sample recovery.

An infill drilling program at the Didipio Gold Copper 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.

Up to 31 July 1995, a total of 74 diamond drill holes had been drilled on the Didipio project (see Figure 11-1). 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. The GRD 1995 PDS identified the presence of a high-grade core of mineralisation potentially capable of supporting an underground mining operation, but available drill data was too widely spaced to allow estimation of measured resources of this material.

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 between sections 5700 N and 5800 N.

This programme consisted in total of 16 inclined diamond drill holes, subsequently augmented by two vertical drill holes drilled into the centre of the deposit primarily for geotechnical purposes, but sampled and assayed following geotechnical logging. Most of the drilling information used for the GRD 1998 DFS resource estimation has been completed perpendicular to the direction of the core of the mineralisation.

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.

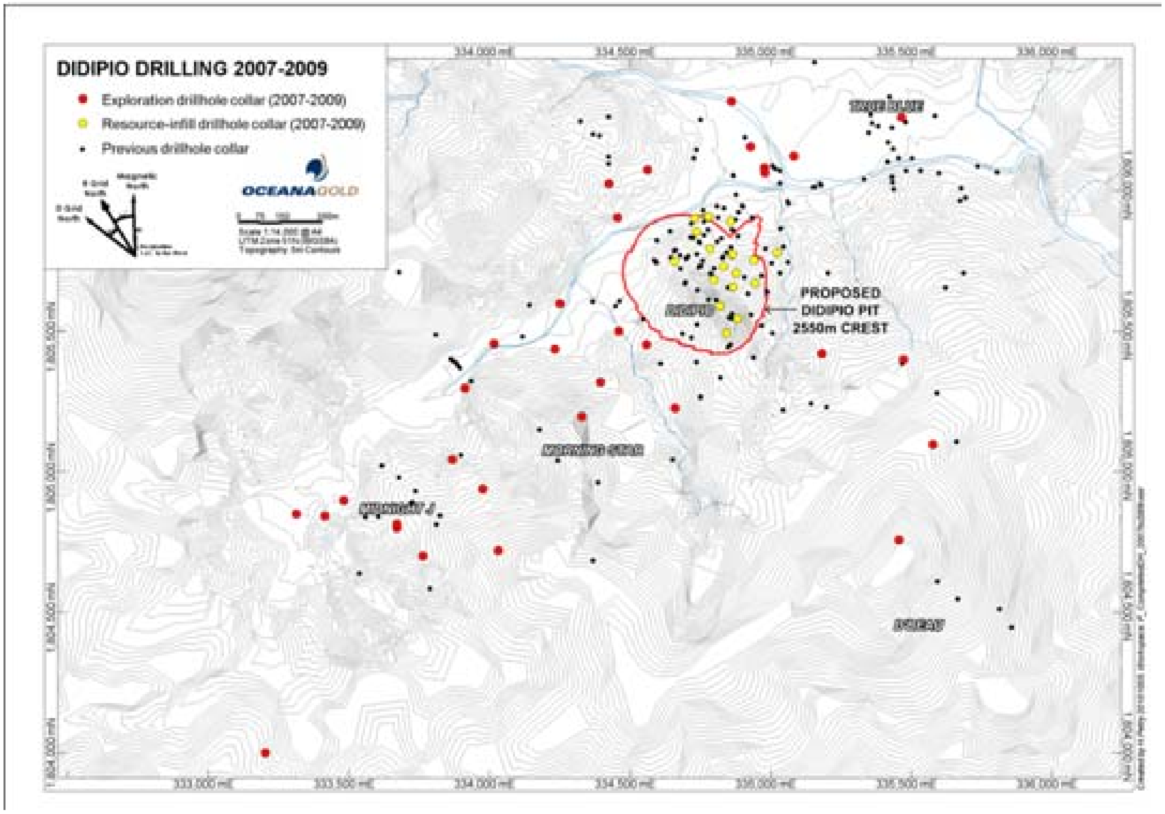
Drill hole DDDH55, drilled to provide a large metallurgical sample, was cored as PQ (85mm diameter) to 200m depth and the core from this intersection logged, but not sampled or assayed, as the entire core was required for metallurgical test work.

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 1.0g/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 11-1: Didipio Gold-Copper Project geology plan, showing drill hole locations**




## 11.2 Tunnel Sampling and Trenching

During early exploration at the Didipio Gold-Copper 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.

## 11.3 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 11-2.

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.
- Geological mapping.

## 11.4 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 rephotographed 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.
- Mineralisation.

Geotechnical logging uses standard logging forms:

- Recoveries.
- Orientations.
- Rock quality – RQD.
- Physical property measurements:

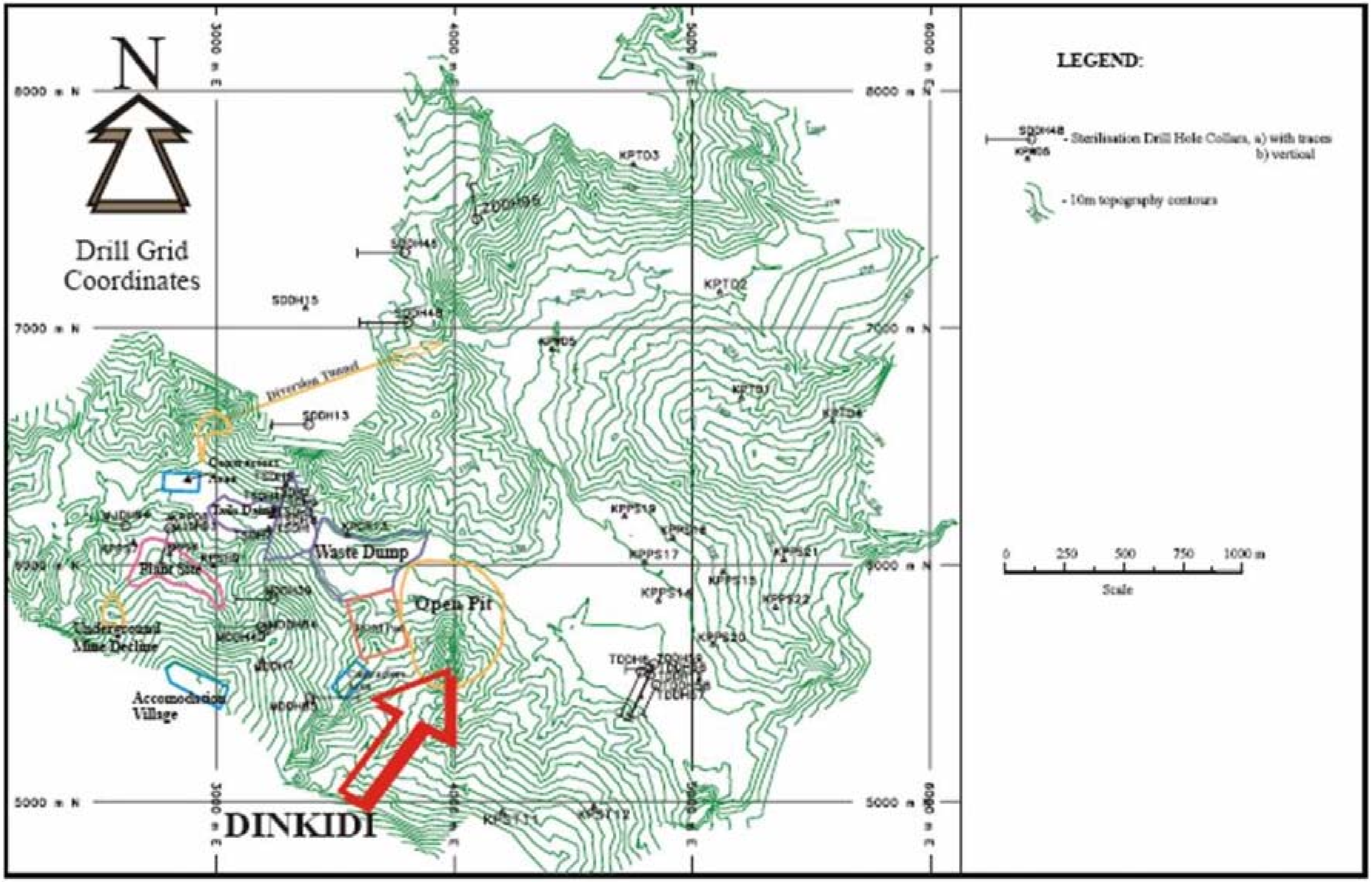
a. Point load testing (after DDDH31).

b. Magnetic susceptibility measurements are taken at approximately four readings per metre.

c. 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 Gold-Copper Project drill cores. All physical property data is included in the database.

**Figure 11-2: Position of sterilisation and infrastructure holes around the Didipio Gold-Copper Project Deposit**



# 12 SAMPLING METHOD AND APPROACH

## 12.1 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 sample intervals are generally 2m or 3m. A plan view of the drill hole collars is provided in Figure 11-1.

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

Sample intervals were defined during the initial logging of cores on site. A core was cut in half using a diamond saw either on site (up to hole DDDH16) or at Cordon (holes DDDH17 onwards). A core has typically been sampled in intervals 2m or 3m under supervision of the site geologist or sample preparation manager, generally ignoring 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.

## 12.2 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.

## 12.3 Definition of Sample Intervals

Sampling intervals are described in section 12.1 and their relationship to mineralisation is discussed in Section 11.1.

## 12.4 Summary of Mineralised Widths

Down hole intersections of mineralisation greater than 50m in length and above a 1g/t eqAu cut-off grade are shown in Table 12-1.

**Table 12-1: Summary of mineralised widths**

| Hole no. | From (m) | To (m) | Length (m) | Au (g/t) | Cu% |
|---|---|---|---|---|---|
| DDDH1 | 26 | 152 | 126 | 0.53 | 0.61 |
| DDDH3 | 15 | 102 | 87 | 0.89 | 0.81 |
| DDDH4 | 15 | 110 | 95 | 1.55 | 1.60 |
| DDDH5 | 113 | 263 | 150 | 0.66 | 0.81 |
| DDDH9 | 30 | 288 | 258 | 1.76 | 0.48 |
| DDDH11 | 117 | 292 | 175 | 1.14 | 0.60 |
| DDDH14 | 280 | 454 | 174 | 2.50 | 0.61 |
| DDDH16 | 321 | 414 | 93 | 1.05 | 0.54 |
| DDDH16 | 516 | 574 | 58 | 1.30 | 0.90 |
| DDDH18 | 171 | 231 | 60 | 0.73 | 0.30 |
| DDDH20 | 197 | 389 | 192 | 1.27 | 0.56 |
| DDDH22 | 229 | 361 | 132 | 0.89 | 0.74 |
| DDDH24 | 300 | 370 | 70 | 0.88 | 0.24 |
| DDDH25 | 58 | 214 | 156 | 2.58 | 0.81 |
| DDDH26 | 385 | 436 | 51 | 0.84 | 0.17 |
| DDDH27 | 19 | 70 | 51 | 0.39 | 0.79 |
| DDDH28 | 220 | 556 | 336 | 1.79 | 0.49 |
| DDDH31 | 280 | 520 | 240 | 0.95 | 0.46 |
| DDDH32 | 205 | 256 | 51 | 0.65 | 0.32 |
| DDDH32 | 373 | 424 | 51 | 0.89 | 0.64 |
| DDDH33 | 187 | 409 | 222 | 2.56 | 0.71 |
| DDDH34 | 178 | 430 | 252 | 2.64 | 0.45 |
| DDDH36 | 386 | 443 | 57 | 0.51 | 0.43 |
| DDDH37 | 41 | 122 | 81 | 0.64 | 0.63 |
| DDDH44 | 117 | 300 | 183 | 0.57 | 0.60 |
| DDDH47 | 324 | 414 | 90 | 1.69 | 0.63 |
| DDDH47 | 480 | 591 | 111 | 0.93 | 0.23 |
| DDDH47 | 651 | 702 | 51 | 3.54 | 0.64 |
| DDDH47 | 768 | 822 | 54 | 4.41 | 0.71 |
| DDDH49 | 204 | 255 | 51 | 0.81 | 0.23 |
| DDDH49 | 639 | 714 | 75 | 1.48 | 0.35 |
| DDDH50 | 487 | 592 | 105 | 1.91 | 0.41 |
| DDDH50 | 676 | 727 | 51 | 0.90 | 0.76 |
| DDDH51 | 225 | 588 | 363 | 2.49 | 0.44 |
| DDDH52 | 673 | 736 | 63 | 1.03 | 0.48 |
| DDDH54 | 473 | 524 | 51 | 1.88 | 0.33 |
| DDDH54 | 629 | 680 | 51 | 4.80 | 0.90 |
| DDDH60 | 414 | 507 | 93 | 6.89 | 0.67 |
| DDDH61 | 130 | 286 | 156 | 0.75 | 0.56 |
| DDDH66 | 354 | 490 | 136 | 2.40 | 0.32 |
| DDDH67 | 304 | 354 | 50 | 1.41 | 0.49 |
| DDDH67 | 404 | 520 | 116 | 4.02 | 0.45 |

| Hole no. | From (m) | To (m) | Length (m) | Au (g/t) | Cu% |
|---|---|---|---|---|---|
| DDDH68 | 385 | 475 | 90 | 1.82 | 0.23 |
| DDDH68 | 517 | 627 | 110 | 1.65 | 0.41 |
| DDDH69 | 210 | 386 | 176 | 3.88 | 0.74 |
| DDDH70 | 228 | 360 | 132 | 2.19 | 0.38 |
| DDDH71 | 56 | 284 | 228 | 1.55 | 0.79 |
| DDDH72 | 144 | 388 | 244 | 1.63 | 0.58 |
| DDDH73 | 214 | 446 | 232 | 2.01 | 0.62 |
| DDDH74 | 364 | 440 | 76 | 1.37 | 0.22 |
| DDDH75 | 170 | 376 | 206 | 2.27 | 0.45 |
| DDDH76 | 50 | 192 | 142 | 0.77 | 0.35 |
| DDDH77 | 40 | 190 | 150 | 0.78 | 0.66 |
| DDDH78 | 180 | 320 | 140 | 1.89 | 0.43 |
| DDDH79 | 120 | 320 | 200 | 3.22 | 0.58 |
| DDDH80 | 16 | 162 | 146 | 1.33 | 1.04 |
| DDDH81 | 55 | 207 | 152 | 0.97 | 0.68 |
| DDDH82 | 154 | 204 | 50 | 0.71 | 0.45 |
| DDDH82 | 252 | 444 | 192 | 3.78 | 0.51 |
| DDDH83 | 85 | 441 | 356 | 6.07 | 0.82 |
| DDH0202 | 305.00 | 465.00 | 160 | 3.59 | 0.51 |
| DDH0205 | 178.00 | 396.00 | 218 | 1.45 | 0.53 |
| DDH0206 | 278.00 | 482.00 | 204 | 3.45 | 0.83 |
| DDH0207 | 0.00 | 111.00 | 111 | 0.83 | 0.92 |
| DDH0208 | 285.00 | 613.00 | 328 | 1.03 | 0.60 |
| DDH0209 | 225.00 | 339.00 | 114 | 0.92 | 0.74 |
| DDH0210 | 63.00 | 173.00 | 110 | 0.88 | 0.74 |
| DDH0211 | 88.00 | 184.00 | 96 | 0.62 | 0.93 |
| DDH0212 | 28.70 | 89.00 | 60.3 | 0.45 | 0.75 |
| DDH0213 | 76.00 | 178.00 | 102 | 0.72 | 0.61 |
| DDH0214 | 167.00 | 311.60 | 144.6 | 1.13 | 0.35 |
| DDH0215 | 295.00 | 357.00 | 62 | 0.58 | 0.41 |
| DDH0218 | 0.00 | 169.00 | 169 | 0.33 | 0.68 |
| DDH0220 | 0.00 | 91.80 | 91.8 | 0.23 | 0.62 |
| DDH0221 | 0.00 | 56.00 | 56 | 0.21 | 0.64 |
| DOX1 | 0 | 87 | 87 | 1.74 | 1.65 |
| DOX2 | 0 | 54 | 54 | 0.48 | 0.83 |
| DOX3 | 15 | 66 | 51 | 1.38 | 0.98 |
| DOX4 | 0 | 102 | 102 | 0.55 | 1.11 |
| DOX5 | 3 | 57 | 54 | 0.43 | 1.17 |
| DOX7 | 0 | 57 | 57 | 0.29 | 0.97 |
| DOX8 | 0 | 50 | 50 | 0.52 | 0.93 |
| DOX9 | 0 | 170 | 170 | 0.60 | 0.75 |

# 13 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section is predominantly extracted from the 1998 Minproc DFS and the 2000 Climax Conceptual Study.

## 13.1 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 13-1.

**Table 13-1: Didipio Gold-Copper 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 |
| 2 | 1990-1 | ARIMCO | ANALABS (MANILA) | DDDH8-11 | 350 | 4 |
| 3 | | ARIMCO | AMC | DDDH14-16 | 252 | 4 |
| 4 | 1992-1998 | CLIMAX | CLIMAX | >DDDH-18 | 8051 | 89 |
| 2 | 2008 | OGC | McPHAR (MANILA) | >DDH0221 | 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 testwork 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 used on the copper.

## 13.2 Analytical Methods

### 13.2.1 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.

### 13.2.2 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.

### 13.2.3 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.

### 13.2.4 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,

[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.

entered into Excel and finally loaded into Minesight for 3D Geological Coding. The SG values are tabulated in Table 13-2 and are similar to that used by Hellman and Schofield in its 2007 resource estimate.

**Table 13-2: 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.

### 13.2.5 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 are generally associated with skarn alteration.

## 13.3 Quality Assurance Quality Control (QAQC) Procedures

QAQC measures employed at Didipio included standards, sample resplits, replicate analyses and inter-laboratory check assays. No copper standards or sample blanks were used in the pre-2008 drilling, although 890 inter-laboratory copper analyses were completed for this period.

### 13.3.1 Standards

A set of gold standards was prepared from Didipio Cooper-Gold Project core samples by Australian Geostandards Proprietary Limited to best represent the range of expected gold values at the Didipio Gold-Copper Project. For the 2008 drilling matrix matched, certified standards were supplied by Ore Research Exploration Pty Ltd.

Selected standard samples, of known gold content, were submitted on average every 20 samples in series with the normal drill core samples and the results were checked against the known values.

### 13.3.2 Sample Resplits

Duplicate sample splits were prepared in Cordon, under a separate sample number, and despatched to Analabs, and more recently McPhar. This was carried out on average once every 20 samples.

### 13.3.3 Replicate Assays

Replicate assays were carried out for gold and copper on pulp splits received at roughly one in 15 for gold and one in 150 for copper.

### 13.3.4 Inter-laboratory Comparisons

Check assays were performed by a number of different laboratories or companies for comparison with the original Analabs assays. AMDEL reassayed 607 samples for gold and copper; others included Becquerel (196 samples, Au only), ITS (183 Au & Cu), McPhar (58 Au & Cu) and Newmont (42 Au and Cu).

### 13.3.5 Early Samples

Some 3% of the samples were prepared without adequate quality control measures, such as sieve analysis and resampling of the coarse fraction, to determine sample variability. However, the samples were included in the Didipio Gold-Copper Project drilling database because subsequent check (gold) assays showed good correlation with original assays and composited quarter core samples for metallurgical test work showed good correlation with original assays.

### 13.3.6 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.

## 13.4 Statement of sample and assaying adequacy

The author (Jonathan Moore) 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.

# 14 DATA VERIFICATION

## 14.1 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 14.2) and formed the basis of the previous two Didipio Gold-Copper 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 14.3.

## 14.2 Hellman and Schofield Review

### 14.2.1 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, the author could confirm that the major equipment items are still present.

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

### 14.2.2 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 consistency of data.

The two datasets showed some differences in collar surveys for holes DDDH60-83, which were attributed to inaccurate collar surveys prior to the DFS. Unfortunately, this could not be confirmed because no original survey records could be located giving final collar locations. The average difference in collar location for these 24 holes is 6.3m (in northing and easting only – elevations are identical), with a maximum difference of 19m for mineralised holes. Available reports show that the 1998 surveys were accurate to 25mm in X and Y, and 100mm in Z.

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 14-1).

**Table 14-1: 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:// 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.

Inspection of the core at Cordon showed some differences between the logging and actual lithologies in the core. This was particularly evident in the Bugoy breccia, where a number of apparently "ordinary" breccias appear to have been logged as this specific unit. The Biak Shear Zone was also inspected and found to comprise zones of lightly sheared and readily identifiable rock types; lithology, structure and alteration should be kept separate in the logging.

Assay certificates were eventually located for all the holes checked in detail, except DDDH4. Only one error was detected in the holes with assay certificates (a copper value in DDDH83).

However, there were a number of differences between the assay print-out (not an original assay certificate) and database records for hole DDDH4. In particular, records were substantially different for the interval 108-121m, although these and other smaller differences may be due to reassaying of samples. The source of this disparity has not been established. This should be done in the future to ensure that the most reliable results are entered in the database.

The more accurate ore grade method assays for initial copper assays above 1% were correctly entered in the database, except for hole DDDH83. There is no evidence that ore grade assays were performed for this hole, and the situation is unclear for holes DDDH42 and 53 because no assays above 1% were recorded. Ore grade assays were confirmed for holes DDDH25 and DOX3.

No from-to errors were detected in the geology assay files, no excessive hole deviation was identified in the down hole surveys and drill hole collars were compatible with the available topography. However, the location of trenches and tunnels was not consistent with topography, with the trenches generally 10-20m below surface despite being only 1-2m deep, and some tunnels extending above topography. These excavations were only picked up by tape and compass relative to an early grid system, so their locations are not considered reliable.

Sample recovery data was checked for consistency and completeness (see Section 12.2). A substantial number of values above 100% are recorded (up to 1200%) and the database is not entirely complete. Most intervals are core recovery, although there are some recoveries for hole pre-collars as well. A small number of from-to errors and duplicate intervals were detected. Only limited checking against available records was completed and further validation of this data is required.

Only part of the reported density data was located, for holes DDDH29-61. The DFS reports additional measurements for the earlier and later holes, but this information could not be found. Nor could any original density measurement data sheets be located that recorded sample weights in air and water. A few unrealistically high values were noted (12.5 and 7.2 t/m$^3$) as well as some conspicuously low values at depth (as low as 1.78 t/m$^3$). The missing data and original records need to be located to complete and validate the density database.

Compilation of QAQC data including standards, field and analytical duplicates showed that the available data was incomplete in some areas when compared to Minproc DFS or Climax Conceptual Study.

### 14.2.3 Hellman and Schofield QAQC Analysis

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.

## 14.3 QAQC

Section 14.3 was written by Arnold van der Heyden of Hellman and Schofield on behalf of OGC.

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.

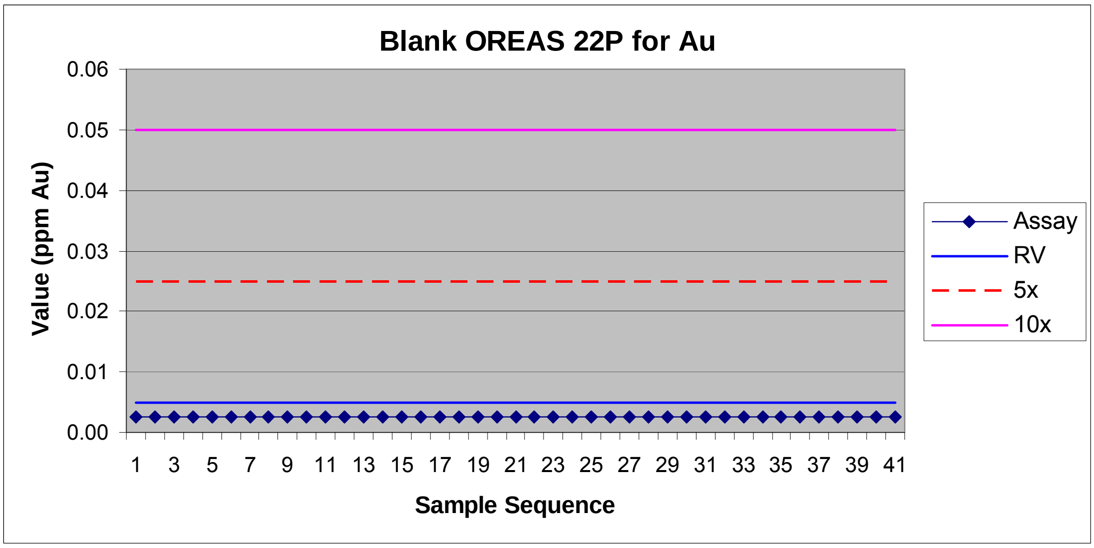
Examination of the data for the standards and blanks revealed a number of obvious errors where the standard or blank was mislabelled. There were 43 of these errors, which were rectified prior to further data analysis.

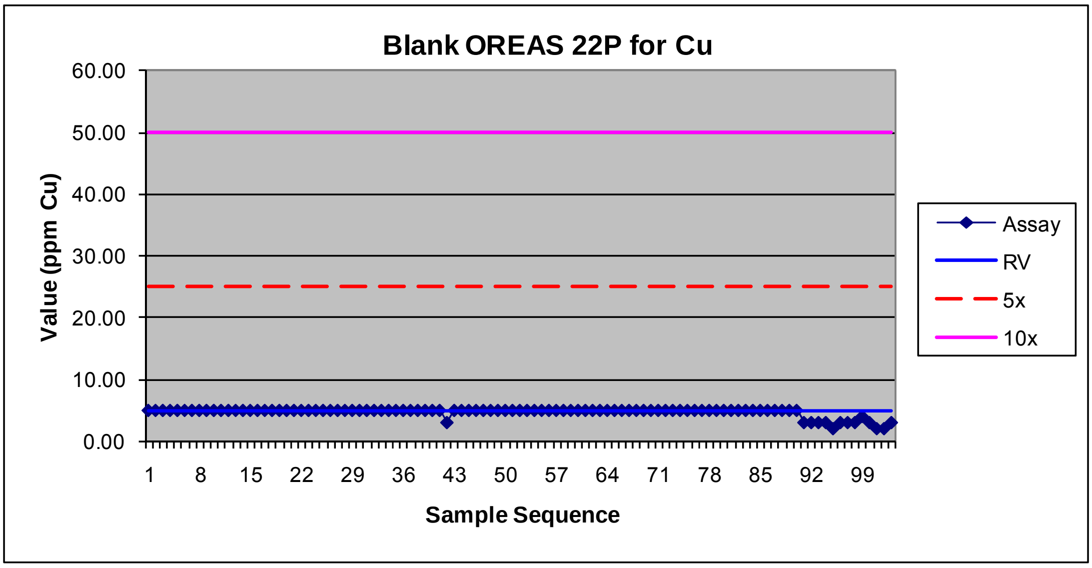
### 14.3.1 Blanks

One pulp blank was submitted as part of the QAQC program – OREAS 22P. Once all obvious errors were rectified, the blank performed well, with all assays around or below detection for both gold and copper.

Control charts (see Figure 14-1) show no variation in gold grade (all below detection), while values less than 5ppm Cu represent samples assayed by ICP, with values of 5ppm tested by AAS.

**Figure 14-1: OREAS gold and copper blanks**




This demonstrates that no contamination was introduced during the preparation of pulps for chemical analysis.

However, there are no coarse blanks to monitor potential sample contamination during the preparation of sample pulps. It is recommended that coarse blanks be included in future QAQC programs for Didipio.

### 14.3.2 Standards

A range of five copper-gold standards prepared by Ore Research were submitted with the drill hole samples. Recommended values for the five standards are shown in **Table 14-2**.

**Table 14-2: OREAS copper and gold standards**

| Standard ID | Cu | Au |
|---|---|---|
| OREAS 52Pb | 0.334 | 0.307 |
| OREAS 53P | 0.413 | 0.380 |
| OREAS 53Pb | 0.546 | 0.623 |
| OREAS 50Pb | 0.744 | 0.841 |
| OREAS 54Pa | 1.550 | 2.900 |

A summary of gold assay results (Table 14-3) for the standards show a small overall positive bias of 1.8%. The lower-grade standards have a positive bias, while the highest-grade standard has a small negative bias. Standard 53P has an anomalously high bias of 5.4%.

**Table 14-3: OREAS gold standards statistics**

| Au std (g/t Au) | Count | Rec value | Avg | Min | Max | %Diff | >+/- 5% | >+/- 10% |
|---|---|---|---|---|---|---|---|---|
| OREAS 52P | 76 | 0.307 | 0.313 | 0.297 | 0.340 | 2.0% | 13 | 1 |
| OREAS 53P | 71 | 0.380 | 0.401 | 0.364 | 0.442 | 5.4% | 34 | 14 |
| OREAS 53Pb | 52 | 0.623 | 0.632 | 0.609 | 0.670 | 1.5% | 4 | 0 |
| OREAS 50P | 66 | 0.841 | 0.855 | 0.811 | 0.920 | 1.6% | 6 | 0 |
| OREAS 54P | 36 | 2.900 | 2.859 | 2.705 | 3.047 | -1.4% | 4 | 0 |

A summary of copper assays (Table 14-4) for the standards reveals a small overall negative bias of 1.9%. The lowest grade standard has a positive bias, while the higher grade standards have an increasing negative bias.

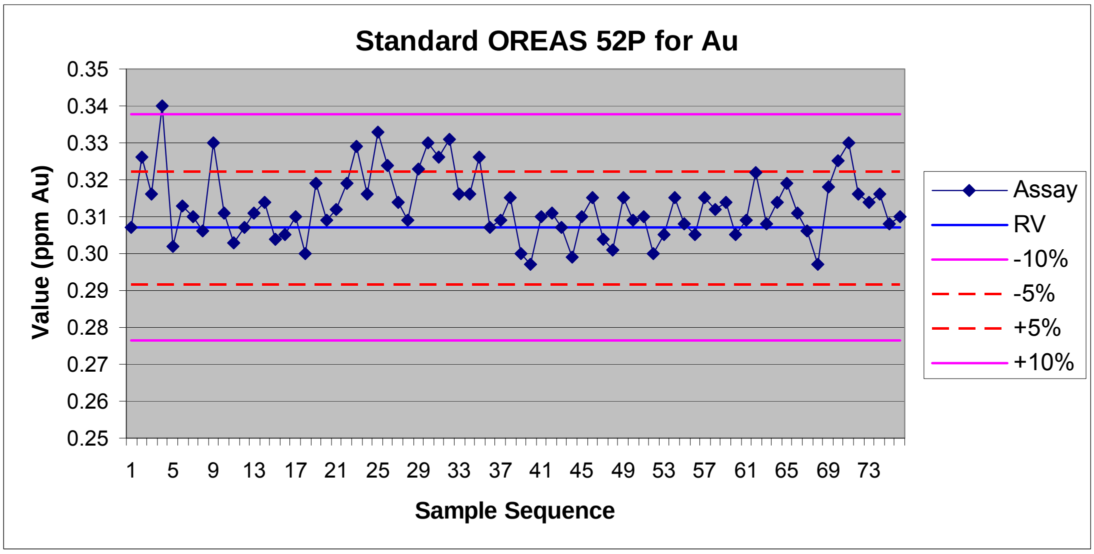
**Table 14-4: OREAS copper standards statistics**

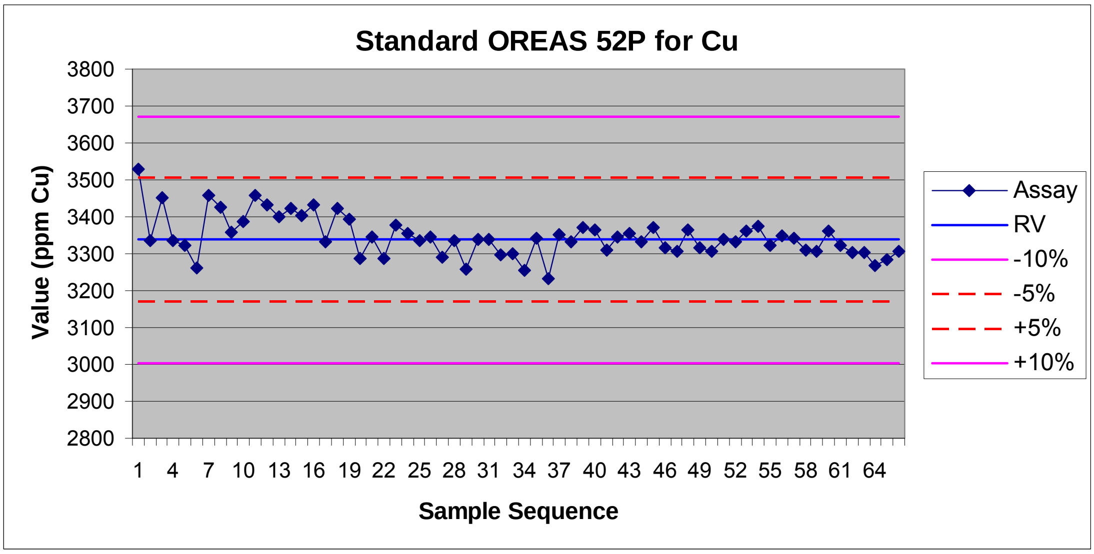
| Cu Std (ppm Cu) | Count | Rec value | Avg | Min | Max | %Diff | >+/- 5% | >+/- 10% |
|---|---|---|---|---|---|---|---|---|
| OREAS 52P | 76 | 3338 | 3346 | 3203 | 3530 | 0.2% | 1 | 0 |
| OREAS 53P | 46 | 4130 | 4109 | 3830 | 4540 | -0.5% | 4 | 0 |
| OREAS 53Pb | 52 | 5460 | 5313 | 4970 | 5708 | -2.7% | 6 | 0 |
| OREAS 50P | 66 | 7440 | 7217 | 6961 | 7447 | -3.0% | 4 | 0 |
| OREAS 54P | 33 | 15500 | 14949 | 14532 | 15310 | -3.6% | 3 | 0 |

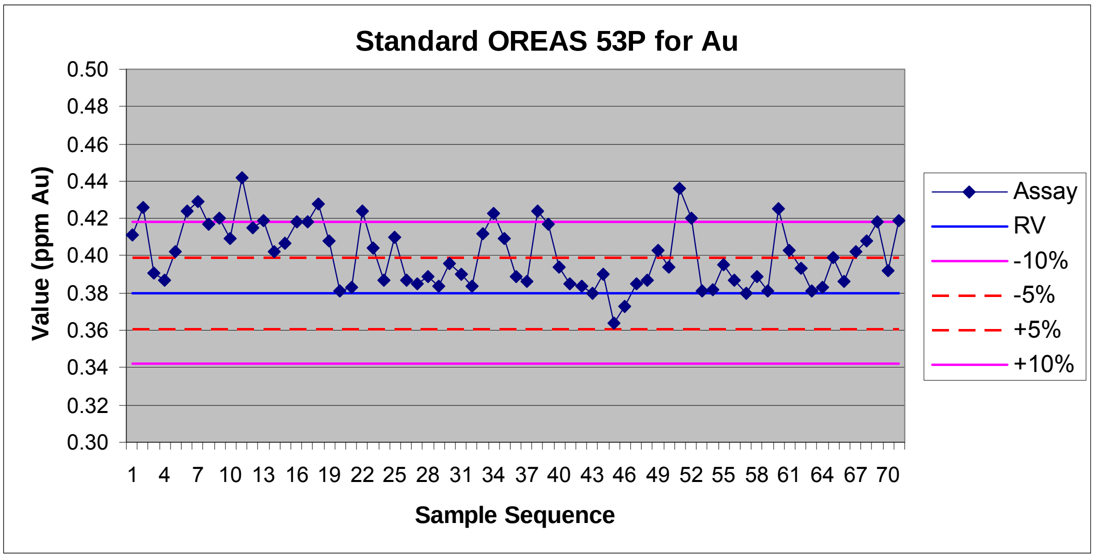
So results for both gold and copper show a trend from positive bias at low grade to a negative bias at high grade, i.e. a conditional bias. This is suggestive of a calibration problem at the laboratory and requires further investigation.

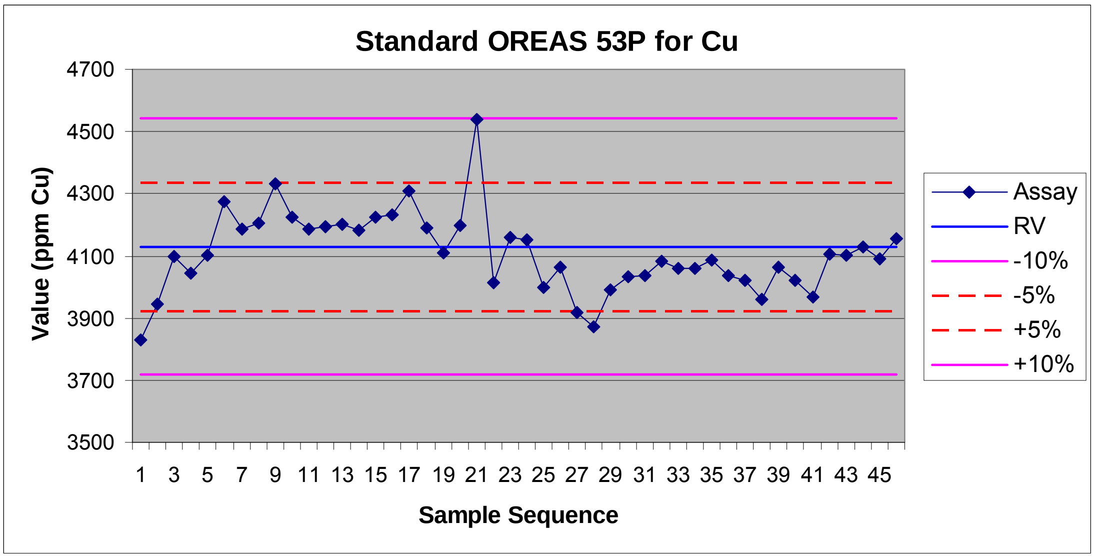
However, most commercial laboratories quote assay accuracy of +/-10%, so these results are generally within this range. The exception is standard 53P for gold, with around 20% of results above +10% of the recommended value. Control charts for the standards are presented in Figure 14-2.

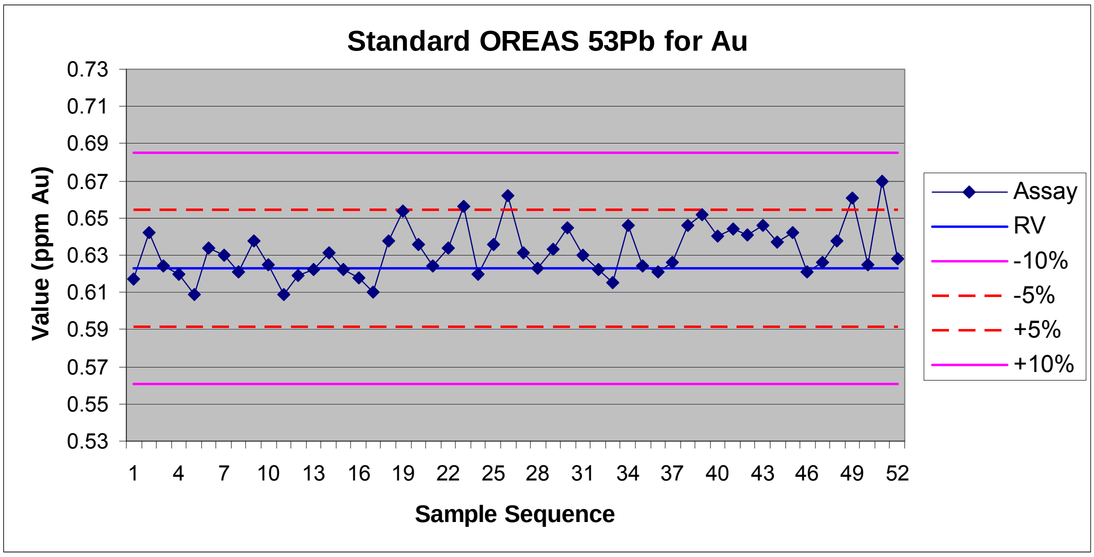
**Figure 14-2: OREAS gold and copper standards**

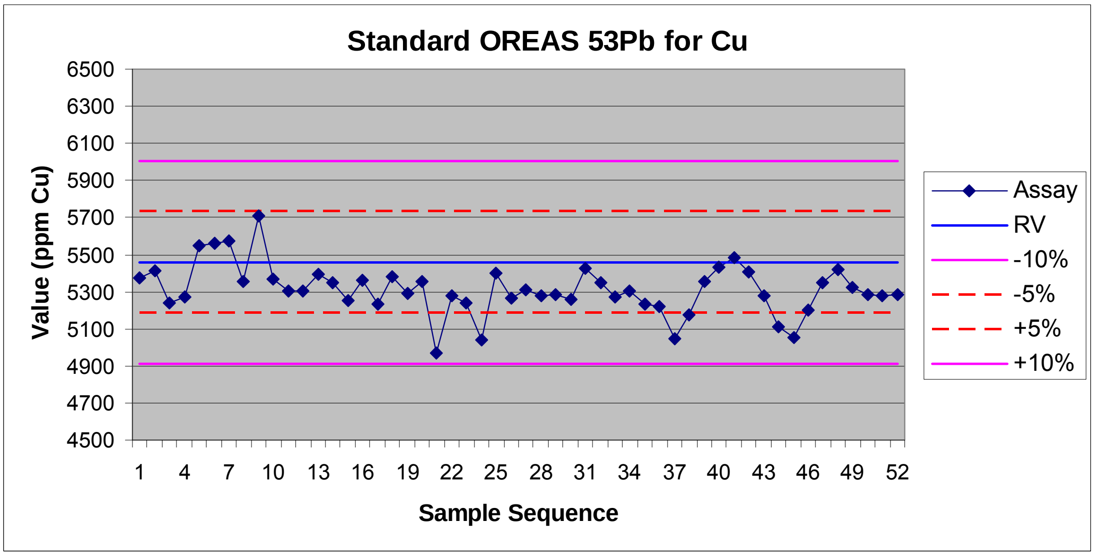


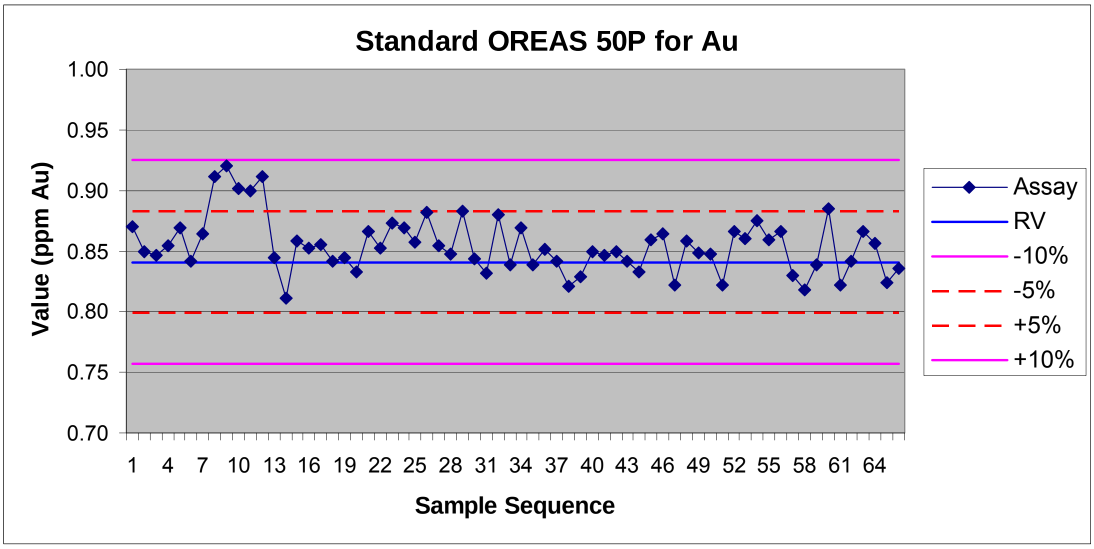



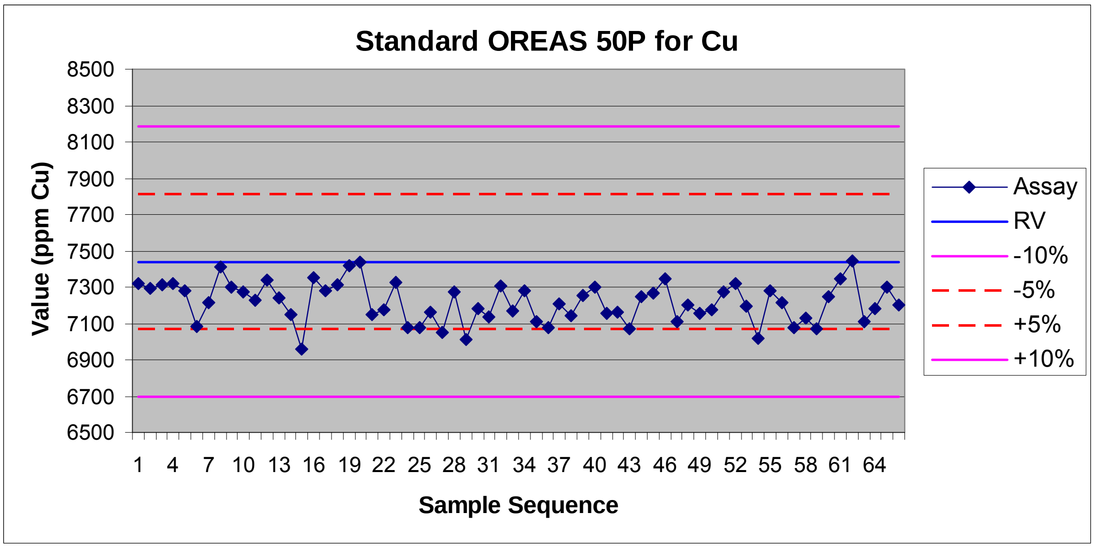

Standard OREAS 53P for Au
Value (ppm Au)
0.50
0.48
0.46
0.44
0.42
0.40
0.38
0.36
0.34
0.32
0.30
1 4 7 10 13 16 19 22 25 28 31 34 37 40 43 46 49 52 55 58 61 64 67 70
Sample Sequence
Assay
RV
-10%
-5%
+5%
+10%

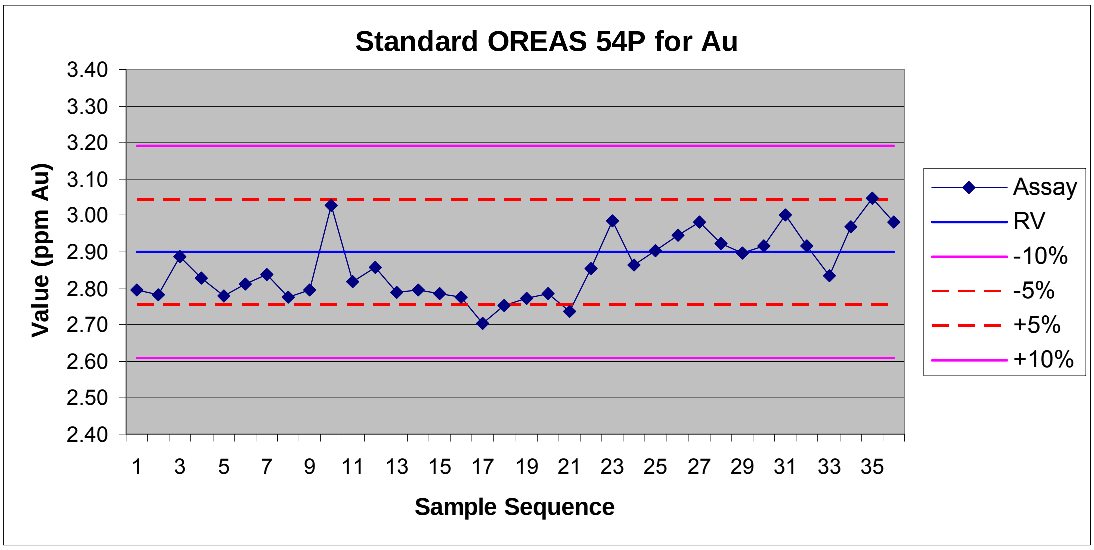

Standard OREAS 53P for Cu
Value (ppm Cu)
4700
4500
4300
4100
3900
3700
3500
1 3 5 7 9 11 13 15 17 19 21 23 25 27 29 31 33 35 37 39 41 43 45
Sample Sequence
Assay
RV
-10%
-5%
+5%
+10%

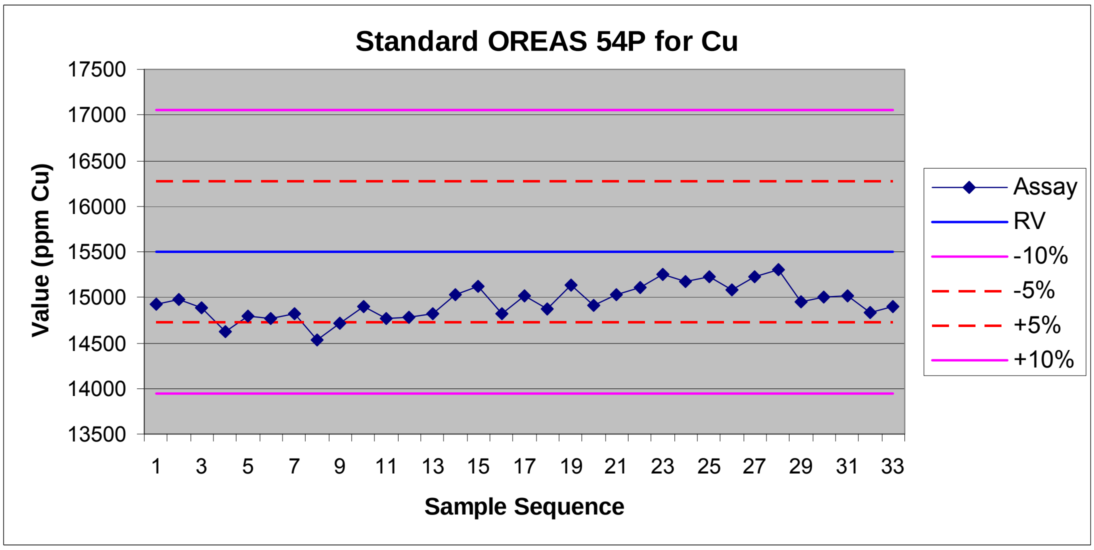

Standard OREAS 53Pb for Au
Value (ppm Au)
0.73
0.71
0.69
0.67
0.65
0.63
0.61
0.59
0.57
0.55
0.53
1 4 7 10 13 16 19 22 25 28 31 34 37 40 43 46 49 52
Sample Sequence
Assay
RV
-10%
-5%
+5%
+10%


Standard OREAS 53Pb for Cu
Value (ppm Cu)
6500
6300
6100
5900
5700
5500
5300
5100
4900
4700
4500
1 4 7 10 13 16 19 22 25 28 31 34 37 40 43 46 49 52
Sample Sequence
Assay
RV
-10%
-5%
+5%
+10%


Standard OREAS 50P for Au
Value (ppm Au)
1.00
0.95
0.90
0.85
0.80
0.75
0.70
1 4 7 10 13 16 19 22 25 28 31 34 37 40 43 46 49 52 55 58 61 64
Sample Sequence
Assay
RV
-10%
-5%
+5%
+10%


Standard OREAS 50P for Cu
Value (ppm Cu)
8500
8300
8100
7900
7700
7500
7300
7100
6900
6700
6500
1 4 7 10 13 16 19 22 25 28 31 34 37 40 43 46 49 52 55 58 61 64
Sample Sequence
Assay
RV
-10%
-5%
+5%
+10%




In summary, the results for the standards are generally within the range of accuracy quoted by most commercial laboratories and are therefore considered acceptable. The trend towards a negative bias at higher grades warrants further investigation, as do the anomalously high gold results for standard 53P.

### 14.3.3 Duplicates

Both field and laboratory duplicates were analysed as part of the QAQC for the recent drilling. The field duplicates were independently submitted by Didipio personnel, while the laboratory duplicates are internal lab checks.

The precision of the various duplicate datasets was assessed to ascertain the repeatability and integrity of sampling at various stages in the sample preparation process. Precision is defined here as the 95th percentile of the percentage half absolute difference (PHAD = |x-y|/(x+y)), where x is the original result and y is the duplicate. This is a relative measure of precision allowing comparison of results across the entire grade range; 5% of the results fall outside this bound. The mean absolute percentage difference (MAPD) is also given, which is twice the average of the PHAD.

Summaries of analyses for the various gold and copper duplicates are presented in the Table 14-5 and Table 14-6.

Table 14-5: Gold duplicates

| Type | Pairs | Q(95) PHAD | Mean PHAD | Median PHAD | Correl-ation | MAPD |
|---|---|---|---|---|---|---|
| Au Field Splits | 207 | 40.9% | 13.5% | 9.1% | 0.930 | 26.9% |
| Au Lab Splits ANA1 | 609 | 29.9% | 10.1% | 7.1% | 0.983 | 20.2% |
| Au Lab Splits ANA2 | 422 | 18.0% | 7.7% | 5.4% | 0.987 | 15.4% |
| Au Lab Splits MCP | 299 | 7.6% | 2.4% | 1.2% | 0.999 | 4.8% |
| Au Lab Splits PM6 | 24 | 6.3% | 2.2% | 1.6% | 0.999 | 4.5% |

Table 14-6: Copper duplicates

| Type | Pairs | Q(95) PHAD | Mean PHAD | Median PHAD | Correl-ation | MAPD |
|---|---|---|---|---|---|---|
| Cu Field Splits ANA1 | 206 | 2.6% | 0.8% | 0.5% | 1.000 | 1.6% |
| Cu Field Splits ANA2 | 221 | 3.8% | 1.7% | 0.9% | 0.998 | 3.4% |
| Cu Lab Splits AA1 | 17 | 2.2% | 0.9% | 0.8% | 1.000 | 1.8% |
| Cu Lab Splits ANA2 | 34 | 2.2% | 0.4% | 0.0% | 0.999 | 0.8% |
| Cu Lab Splits ICP1 | 7 | 1.4% | 0.4% | 0.2% | 1.000 | 0.8% |
| Cu Lab Splits MCP | 299 | 2.1% | 0.7% | 0.5% | 1.000 | 1.5% |

As expected, the field duplicates show poorer precision than the lab duplicates because they include the errors associated with additional stages of sample preparation.

The Au field duplicates have relatively poor precision, while the ANA1 and ANA2 lab duplicates show some improvement in precision compared with the field duplicates. The MCP and PM6 lab duplicates show much better precision than the ANA1/2 lab duplicates, suggesting that they are a different type of duplicate. It seems likely that the ANA1/2 duplicates represent different splits of the sample pulp, while the MCP/PM6 duplicates are likely from the same pulp sample.

Overall, the precision for gold is broadly similar to that of similar porphyry deposits in Australia.

The precision for copper is excellent for all types of duplicates and the field duplicates are only marginally worse than the lab duplicates.

Some datasets have a few outliers, e.g. Cu field ANA2 and Au lab MCP, but these have little impact on the statistics.

There is no obvious evidence of bias between the duplicate datasets, except the Au field duplicates – all original samples over 5g/t returned duplicates with substantially lower values.

Precision plots of selected datasets are presented below highlighting some of the features described above (see Figure 14-3 to Figure 14-7 Precision plots show the mean value of each pair on the X axis and the percentage half difference [PHD = (x-y)/(x+y)] on the Y axis. The vertical scatter is a measure of precision, while the trend line (in red) indicates any possible conditional bias.

Figure 14-3: Precision plot for gold field duplicates

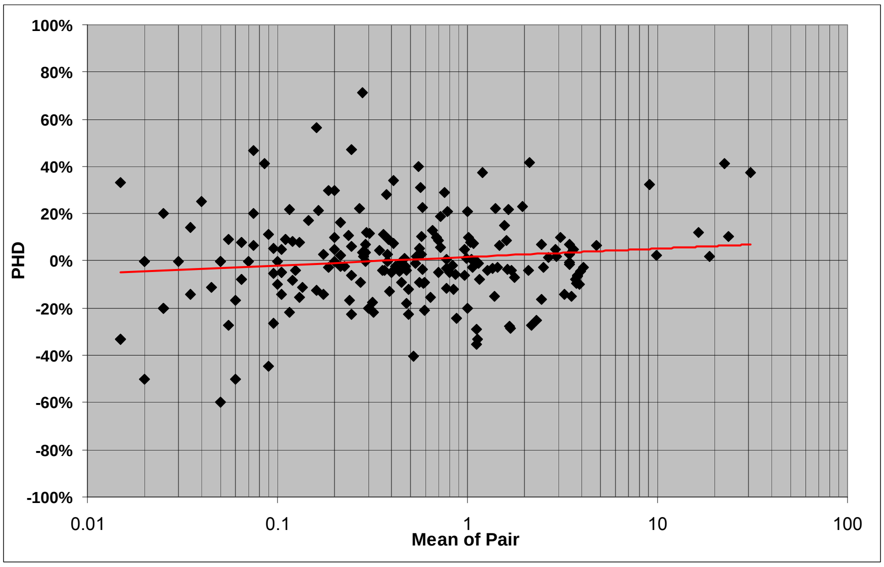



Figure 14-4: Precision plot for gold lab duplicates ANA2

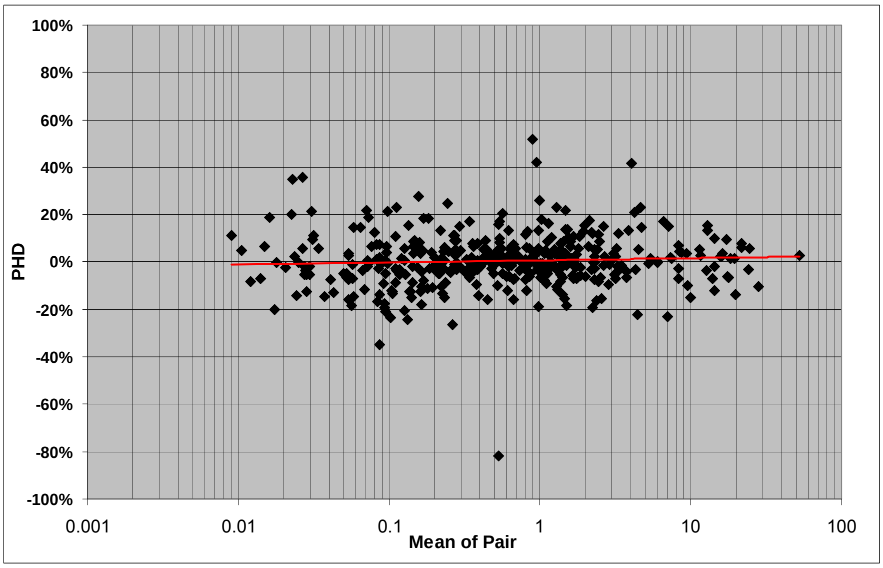


**Figure 14-5: Precision plot for gold lab duplicates MCP**

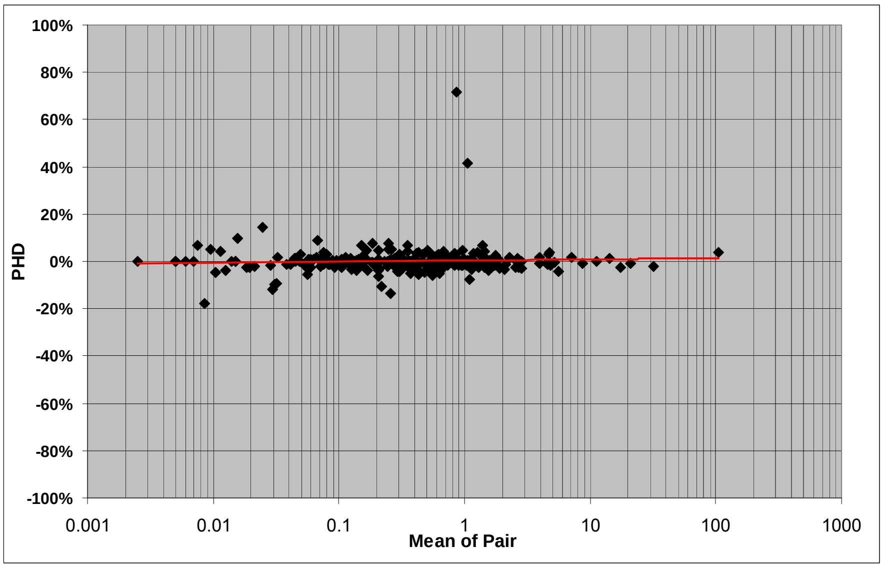



**Figure 14-6: Precision plot for copper field duplicates ANA2**

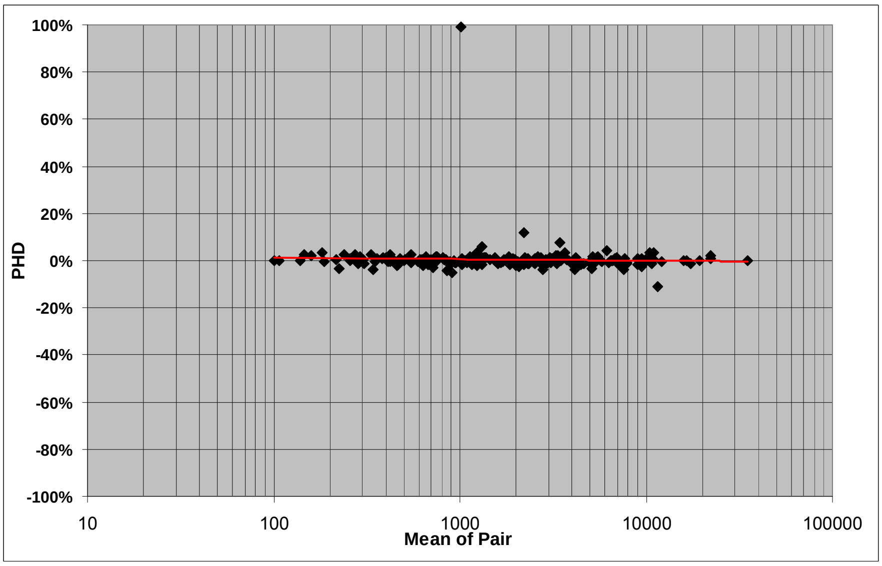
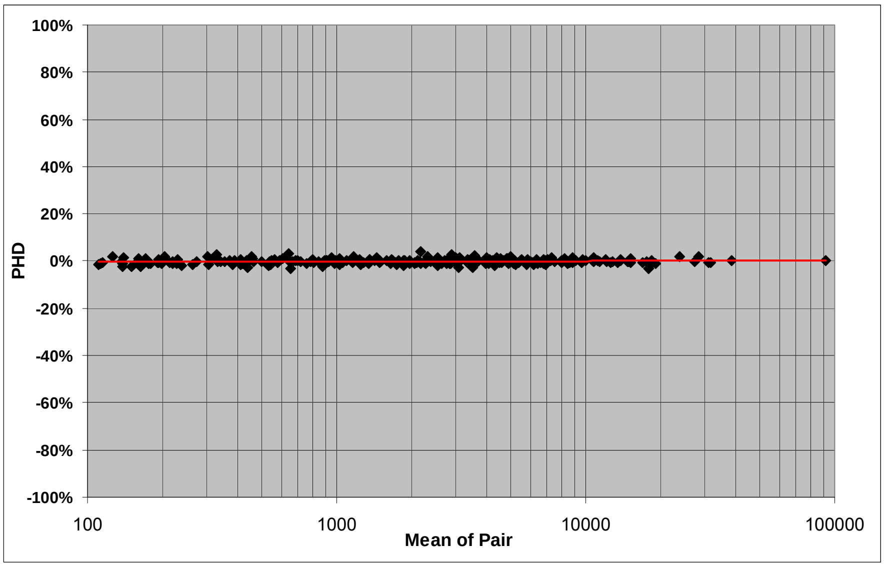


**Figure 14-7: Precision plot for copper lab duplicates MCP**




In summary, the duplicates for copper show excellent precision and the duplicates for gold show precision comparable to similar porphyry deposits. There is generally no obvious indication of bias, except the Au field duplicates where all original samples over 5g/t returned duplicates with substantially lower values.

# 15 ADJACENT PROPERTIES

There are no adjacent properties that have an impact on the potential merit of the Didipio Gold-Copper Project. The Didipio FTAA title held fully contains all known significant gold-copper mineralisation associated with the project in the area.

# 16 MINERAL PROCESSING AND METALLURGICAL TESTING

## 16.1 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.

## 16.2 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.

## 16.3 Metallurgical Testwork

### 16.3.1 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.

### 16.3.2 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 16-1 summarises the various comminution results.

**Table 16-1: 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 DW*i*, 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 16-2.

**Table 16-2: 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.

### 16.3.3 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.

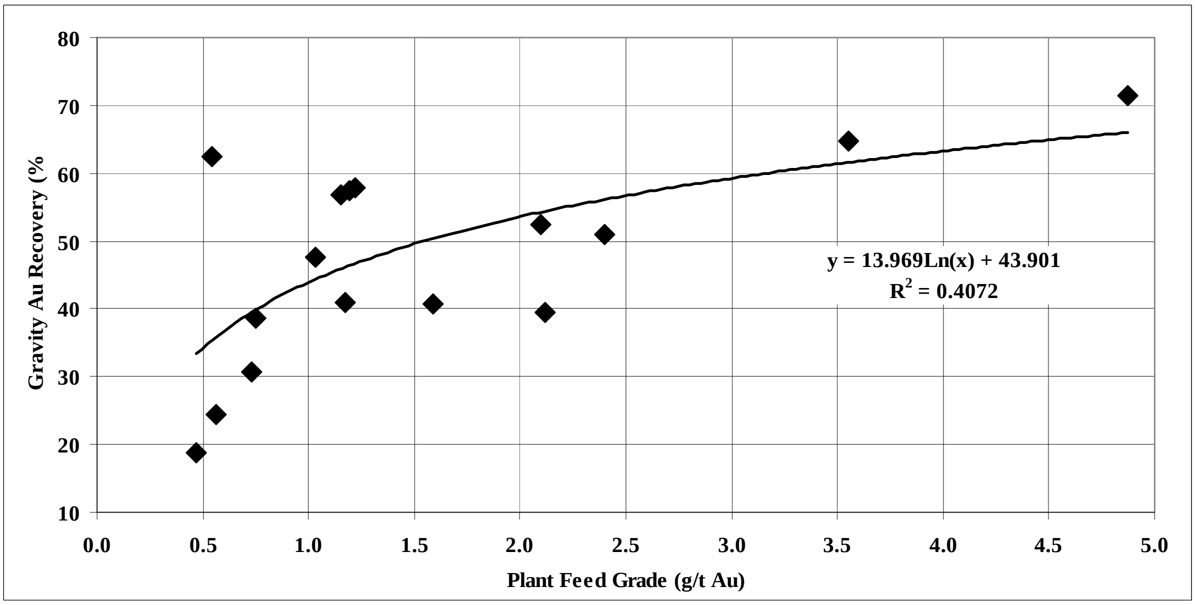
Table 16-3 summarises the results for the nine variability samples, the four composites and the most recent 2006 testwork programme.

Table 16-3: 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 0.5g/t Au material to >50% for material of 1.5g/t Au and above. Figure 16-1 provides a graphical representation of the data.

**Figure 16-1: Gravity recovery results**




## 16.3.4 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.

### 16.3.4.1 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 16-4 summarises the results of these preliminary tests.

**Table 16-4: 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 - | 0.45 | 1.96 | 80.0 | 35 | 144 | 92.8 | 88.4 |

*Note: all testwork conducted by Fox Anamet/Metcon*

#### 16.3.4.2 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 $P_{80}$ of 53µm to a $P_{80}$ of 212µm were tested. All results indicated the optimum grind size was $P_{80}$ = 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 $P_{80}$ = 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.

#### 16.3.4.3 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 16-5) 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 16-5: 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% | 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 | 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 |

### 16.3.4.4 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.

### 16.3.4.5 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 $P_{80}$ = 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 16-6 summarises these results.

**Table 16-6: 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. However, flash flotation has not been included in the metallurgical flowsheet for the project.

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%.

### 16.3.5 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 16-7 summarises the results, which are generally consistent with the results from the early programmes.

**Table 16-7: 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 |

### 16.3.6 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 16-8.

**Table 16-8: 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. These data generally agree with the Minproc parameters.

# 17 MINERAL RESOURCE AND MINERAL RESERVES ESTIMATES

## 17.1 Introduction

The Didipio Gold-Copper Deposit resource estimate was updated by OGC in October 2008 to accommodate 21 additional infill drill holes as well as to translate the estimate from the drill grid to UTM grid.

The previous resource estimate was completed in March 2007 by Hellman and Schofield.

Despite the changes, the estimates provide similar outcomes in terms of the combined Measured and Indicated resource (see Table 17-11).

## 17.2 Geological Interpretation

As discussed in section 7.3, 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.

## 17.3 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 spacings of 100-150m are more usual.

Ninety-eight diamond drill holes at Didipio (DDH0201-0221 and DOX1-9, although all hole numbers in the sequence were drilled at Didipio) were used for grade estimation, while all trenches and tunnels 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 2.0m intervals. The raw samples are commonly either 2m or 3m (94% between 2m and 3m), with a small number of samples between 0 and 3m (and not 2 or 3m) that probably arise from sampling to lithological boundaries, although most samples were taken irrespective of lithological boundaries.

## 17.4 Data Analysis

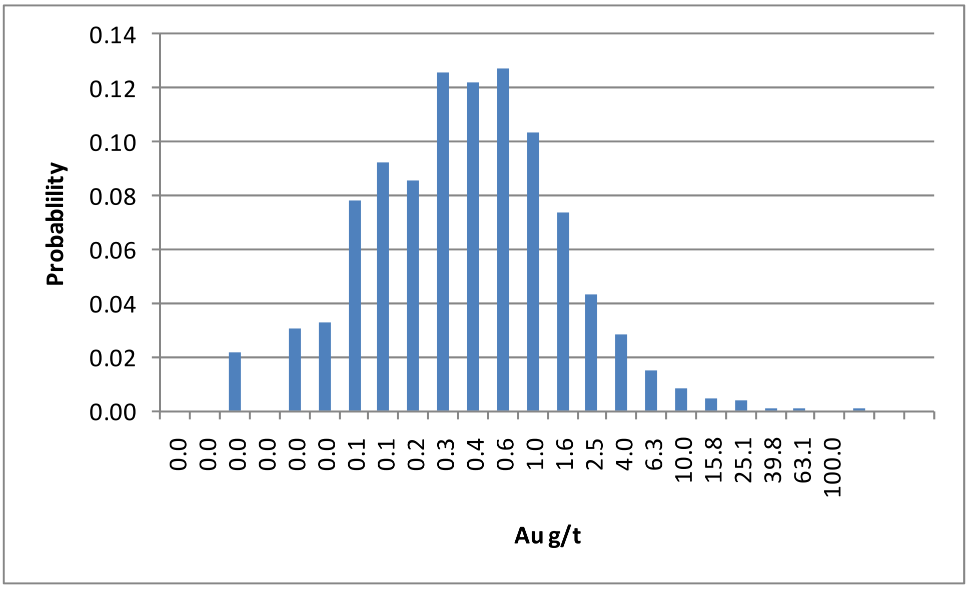
Table 17-1 shows both gold and copper grade statistics by geological domain.

**Table 17-1: Drill hole sample gold and copper summary statistics by lithology**

| | Tunja | Bufu | Biak | D diorite |
|---|---|---|---|---|
| **No. samples** | 8093 | 330 | 1288 | 4434 |
| **GOLD** | | | | |
| **Mean** | 1.24 | 0.61 | 0.25 | 0.29 |
| **Median** | 0.50 | 0.37 | 0.06 | 0.12 |
| **Min** | 0.01 | 0.02 | 0.00 | 0.01 |
| **Max** | 109.24 | 10.90 | 16.70 | 15.80 |
| **Cv** | 2.53 | 1.66 | 2.99 | 2.27 |
| **COPPER** | | | | |
| **Mean** | 0.45 | 0.11 | 0.08 | 0.17 |
| **Median** | 0.30 | 0.08 | 0.03 | 0.09 |
| **Min** | 0.000 | 0.010 | 0.000 | 0.010 |
| **Max** | 9.21 | 1.35 | 0.93 | 5.21 |
| **Cv** | 1.12 | 1.06 | 1.54 | 1.48 |

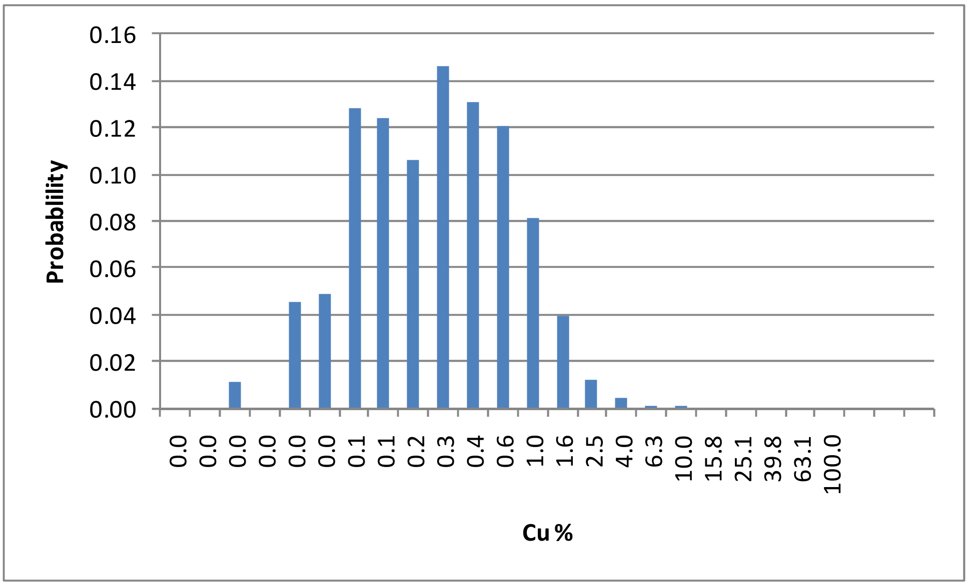
The histogram for gold grade (Figure 17-1) shows a single, approximately log-normal distribution, with perhaps a very small (<1%) high-grade population. Fifty per cent of the total gold is contained within the highest 6% of the gold grades.

**Figure 17-1: Histograms of drill hole gold grades**




The histogram for copper shows a single approximately log-normal distribution (Figure 17-2). The copper distribution is less skewed than the gold distribution, with 50% of the total metal in the 21.6% of samples.

**Figure 17-2: Histograms of drill hole copper grades**




The available specific gravity data was found as 10m intervals, although the actual measurements were performed on 10cm pieces of core approximately every 10m. 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 outcome is tabulated in Table 17-2 and is similar to that used by Hellman and Schofield in its 2007 resource estimate.

**Table 17-2: 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 SG 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 its Tunja host rock.

There was 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.

## 17.5 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 17-3 and Figure 17-4). Variogram maps in the plane of mineralisation (approximately N-S) are fairly isotropic, suggesting no significant plunge component to the mineralisation.

Figure 17-3: Variogram maps for gold (LHS=plan, RHS=E-W section)

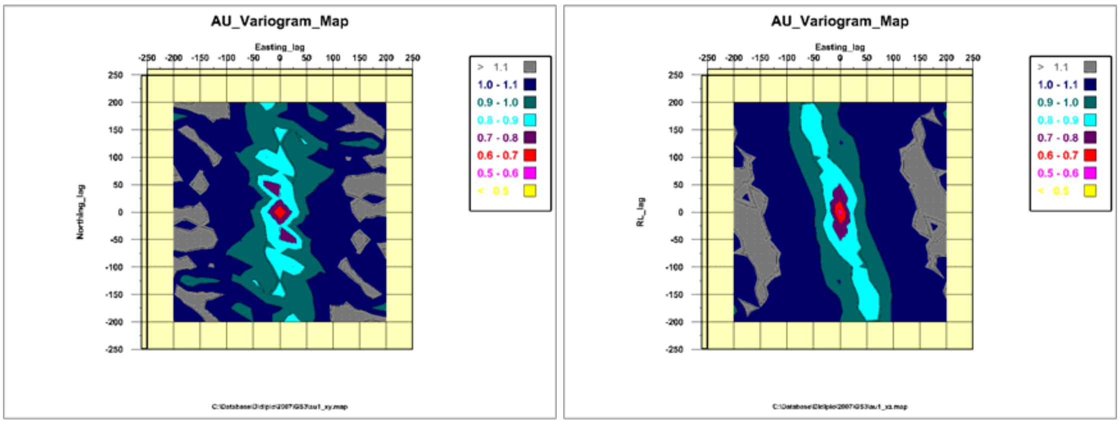
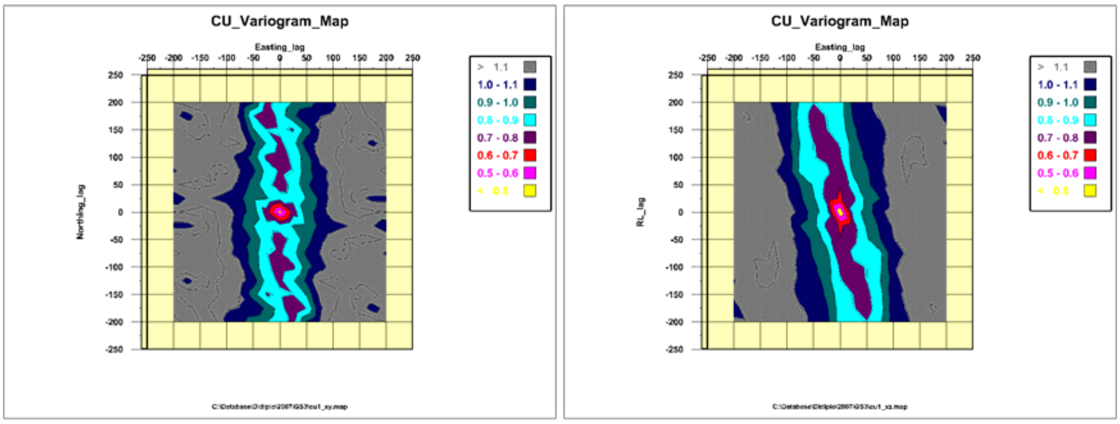



Figure 17-4: Variogram maps for copper (LHS=plan, RHS=E-W section)




The gold and copper variogram model parameters are shown in Table 17-3 and Table 17-4 respectively.

**Table 17-3: Gold variogram model parameters**

| GOLD | Tunja | Bufu | Biak | D diorite |
|---|---|---|---|---|
| Nugget | 0.10 | 0.10 | 0.10 | 0.10 |
| Sill 1, Sill 2 | 0.46 / 0.44 | 0.46 / 0.44 | 0.46 / 0.44 | 0.46 / 0.44 |
| Major Structure 1, Structure 2 | 23 / 105 | 23 / 105 | 23 / 105 | 23 / 105 |
| Minor Structure 1, Structure 2 | 41 / 58 | 41 / 58 | 41 / 58 | 41 / 58 |
| Vertical Structure 1, Structure 2 | 22 / 150 | 22 / 150 | 22 / 150 | 22 / 150 |
| Variogram Rotations | | | | |
| Major Axis | -50 | -50 | 64 | -50 |
| Plunge | 0 | 0 | 0 | 0 |
| Dip | 8 | 8 | -8 | 8 |

**Table 17-4: Copper variogram model parameters**

| COPPER | Tunja | Bufu | Biak | D diorite |
|---|---|---|---|---|
| Nugget | 0.04 | 0.04 | 0.04 | 0.04 |
| Sill 1, Sill 2 | 0.71 / 0.25 | 0.71 / 0.25 | 0.71 / 0.25 | 0.71 / 0.25 |
| Major Structure 1, Structure 2 | 16 / 280 | 16 / 280 | 16 / 280 | 16 / 280 |
| Minor Structure 1, Structure 2 | 56 / 58 | 56 / 58 | 56 / 58 | 56 / 58 |
| Vertical Structure 1, Structure 2 | 120 / 705 | 120 / 705 | 120 / 705 | 120 / 705 |
| Variogram Rotations | | | | |
| Major Axis | -50 | -50 | 64 | -50 |
| Plunge | 0 | 0 | 0 | 0 |
| Dip | 8 | 8 | -8 | 8 |

## 17.6 Resource Estimation

This model update has seen the model framework translated into the National (UTM) grid, which is the grid used for the Didipio project infrastructure. The previous Hellman and Schofield model was based in the drilling grid, which was oriented at 51° west of true north. The block size has been adjusted from 10mE x 25mN x 20mRL (Hellman and Schofield) to 15mE x 15mN x 20mRL to accommodate the long axis of the porphyry no longer being aligned parallel to the grid. The model framework is summarised in Table 17-5.

**Table 17-5: Block model parameters**

| MODPROT | X | Y | Z |
|---|---|---|---|
| **Minimum (m)** | 334,350 | 1,805,340 | 2,000 |
| **Maximum (m)** | 335,100 | 1,806,000 | 2,840 |
| **Block size (m)** | 15 | 15 | 20 |
| **Number of blocks** | 50 | 44 | 42 |
| **Length (m)** | 750 | 660 | 840 |

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.

A two-pass search strategy was used for the estimation of gold and copper grades, with parameters as shown in Table 17-6**.**

**Table 17-6: Estimation search parameters for gold and copper**

| | Tunja | Bufu | Biak | D diorite |
|---|---|---|---|---|
| **Indicated search distances** | | | | |
| **Major** | 75 | 75 | NA | 75 |
| **Minor** | 15 | 15 | NA | 15 |
| **Vertical** | 75 | 75 | NA | 75 |
| **Inferred search distances** | | | | |
| **Major** | 100 | 100 | 100 | 100 |
| **Minor** | 15 | 15 | 15 | 15 |
| **Vertical** | 100 | 100 | 100 | 100 |
| **Search rotations** | | | | |
| **Major axis** | -50 | -50 | 64 | -50 |
| **Plunge** | 0 | 0 | 0 | 0 |
| **Dip** | 8 | 8 | -8 | 8 |
| **Indicated search criteria** | | | | |
| **Min samples** | 18 | 12 | NA | 18 |
| **Max samples/hole** | 6 | 6 | NA | 6 |
| **Max samples/octant** | 6 | 6 | NA | 6 |
| **Inferred search criteria** | | | | |
| **Min samples** | 12 | 6 | 6 | 12 |
| **Max samples/hole** | 6 | 6 | 6 | 6 |
| **Max samples/octant** | 6 | 6 | 6 | 6 |
| **Rotations** | | | | |
| **Major axis** | -50 | -50 | 64 | -50 |
| **Plunge** | 0 | 0 | 0 | 0 |
| **Dip** | 8 | 8 | -8 | 8 |

Identical sample search restriction criteria were used for gold and copper:

- model blocks for Tunja were informed by all composites (i.e. Tunja, Bufu, Biak and Dark Diorite);
- model blocks for Bufu were informed by only Bufu composites;
- model blocks for Biak were informed by only Biak composites; and
- model blocks for Dark Diorite were informed by all composites (i.e. Tunja, Bufu, Biak and Dark Diorite).

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.

Grade cutting was not used, although the influence of DDDH83 (the most intensely mineralised drill hole in the estimate) was reduced. DDDH83 is high grade and near-vertical, and so had its influence reduced as a cautionary measure. This was done by completing a parallel estimate within a 50m radius of DDDH83 excluding DDDH83 grades. The average of both estimates (one including DDDH83 and one excluding DDDH83) was used as the final estimate within the 50m radius (only between the 2298 and 2413mRLs where the highest grades are encountered). 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.

Density was coded in Minesight software for Tunja monzonite, Bufu syenite, Dark Diorite and Bugoy breccia as tabulated in Table 17-2.

The oxide and transition zones were assigned nominal density values of 2.2 t/m$^3$ and 2.4 t/m$^3$ respectively.

## 17.7 Resource Classification

The resource model for Didipio Deposit has been classified to CIM standards. The resource classification is based on the estimation searches summarised in Table 17-6. 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. It is likely that detailed re-logging of the Biak Shear (using well-defined and consistent shear terminology) will increase confidence within this volume and allow some resource to be classified as Indicated. Infill drilling would also increase the confidence in the interpreted position of the Biak Shear.
- 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. It is recommended that the near-surface mineralisation (oxide and transitional zones) be infill drilled. This could lead to a significant increase in near surface Indicated resource.
- The classification of Measured resource was based both on search criteria and three-dimensional geometry. As a first pass, a kriging sweep was set up using the Tunja Indicated search dimensions/rotations as presented in Table 17-6. The only changes were to increase the minimum sample number to 30 and the minimum drill hole and octant requirement to five. The blocks meeting these criteria were then used as a guide to wireframe volume that was geometrically continuous. These criteria ensure that all Measured resource has data falling within both hemispheres of the search.

## 17.8 Mineral Resources

The mineral resources quoted here include the mineral reserves described in this report. These mineral resources were prepared by Jonathan Moore, Principal Resource Geologist 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 2540mRL (base of the open-pit design); and an underground resource between 2540 and 2270mRL (vertical extent of the sub-level cave). The open-cut resource uses a 0.4g/t eqAu cut-off grade, while the underground resource uses a 1.0g/t eqAu cut-off grade. The equation for contained gold equivalent is g/t eqAu = g/t Au + 2.23 x % Cu, based on metal prices of US$800 per ounce for gold and US$2.60 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 17-7, Table 17-8 and Table 17-9 classified using CIM guidelines. All mineral reserves reported are included within the mineral resources reported for the same deposit.

**Table 17-7: Open cut resource estimate**

(above 2540mRL at 0.4 g/t eqAu cut-off grade)

| Class | Tonnes (Mt) | Au (g/t) | Cu (%) | Au (Moz) | Cu (Kt) |
|---|---|---|---|---|---|
| Measured | 7.72 | 1.15 | 0.65 | 0.28 | 50.2 |
| Indicated | 25.41 | 0.42 | 0.40 | 0.34 | 100.4 |
| **Measured & Indicated** | **33.13** | **0.59** | **0.45** | **0.63** | **150.6** |
| Inferred | 16.38 | 0.31 | 0.24 | 0.16 | 39.8 |

**Table 17-8: Underground resource estimate**

(between 2540mRL and 2270mRL at 1.0 g/t eqAu cut-off grade)

| Class | Tonnes (Mt) | Au (g/t) | Cu (%) | Au (Moz) | Cu (Kt) |
|---|---|---|---|---|---|
| Measured | 7.86 | 2.29 | 0.50 | 0.58 | 39.2 |
| Indicated | 19.07 | 1.30 | 0.43 | 0.80 | 82.6 |
| **Measured & Indicated** | **26.94** | **1.59** | **0.45** | **1.37** | **121.8** |
| Inferred | 4.78 | 0.93 | 0.31 | 0.14 | 14.6 |

**Table 17-9: Combined resource estimate**

(at 0.4 g/t eqAu cut-off grade above 2540mRL and at 1.0 g/t eqAu cut-off grade below 2540mRL)

| Class | Tonnes (Mt) | Au (g/t) | Cu (%) | Au (Moz) | Cu (Kt) |
|---|---|---|---|---|---|
| Measured | 15.58 | 1.72 | 0.57 | 0.86 | 89.4 |
| Indicated | 44.49 | 0.80 | 0.41 | 1.14 | 183.0 |
| **Measured & Indicated** | **60.07** | **1.04** | **0.45** | **2.00** | **272.4** |
| Inferred | 21.15 | 0.45 | 0.26 | 0.31 | 54.4 |

The resource is tabulated below in Table 17-10 according to material type.

**Table 17-10: Resource estimate by material type**
(at 0.4 g/t eqAu cut-off grade above 2540mRL and at 1.0 g/t eqAu cut-off grade below 2540mRL)

| OXIDE | Mt | Au g/t | Cu % | Au Moz | Cu Kt |
|---|---|---|---|---|---|
| MEASURED | 0.03 | 0.52 | 0.63 | 0.00 | 0.2 |
| INDICATED | 0.31 | 0.43 | 0.52 | 0.00 | 1.6 |
| **MEASURED & INDICATED** | **0.35** | **0.43** | **0.53** | **0.00** | **1.8** |
| INFERRED | 2.67 | 0.39 | 0.47 | 0.03 | 12.5 |

| TRANSITIONAL | Mt | Au g/t | Cu % | Au Moz | Cu Kt |
|---|---|---|---|---|---|
| MEASURED | 0.05 | 0.68 | 0.76 | 0.00 | 0.4 |
| INDICATED | 0.90 | 0.39 | 0.58 | 0.01 | 5.2 |
| **MEASURED & INDICATED** | **0.95** | **0.40** | **0.59** | **0.01** | **5.6** |
| INFERRED | 0.66 | 0.35 | 0.37 | 0.01 | 2.4 |

| FRESH | Mt | Au g/t | Cu % | Au Moz | Cu Kt |
|---|---|---|---|---|---|
| MEASURED | 15.50 | 1.73 | 0.57 | 0.86 | 88.8 |
| INDICATED | 43.28 | 0.81 | 0.41 | 1.12 | 176.2 |
| **MEASURED & INDICATED** | **58.77** | **1.05** | **0.45** | **1.98** | **265.0** |
| INFERRED | 17.83 | 0.46 | 0.22 | 0.26 | 39.5 |

| ALL | Mt | Au g/t | Cu % | Au Moz | Cu Kt |
|---|---|---|---|---|---|
| MEASURED | 15.58 | 1.72 | 0.57 | 0.86 | 89.4 |
| INDICATED | 44.49 | 0.80 | 0.41 | 1.14 | 183.0 |
| **MEASURED & INDICATED** | **60.07** | **1.04** | **0.45** | **2.00** | **272.4** |
| INFERRED | 21.15 | 0.45 | 0.26 | 0.31 | 54.4 |

## 17.9 Model Validation

The resource model was validated in number of ways. Initially, 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 are presented in Figure 17-5, Figure 17-6 and Figure 17-7 for both copper and gold.

Figure 17-5 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.

**Figure 17-5: Resource Model (Measured and Indicated) versus 2m composited sample gold grades by bench**

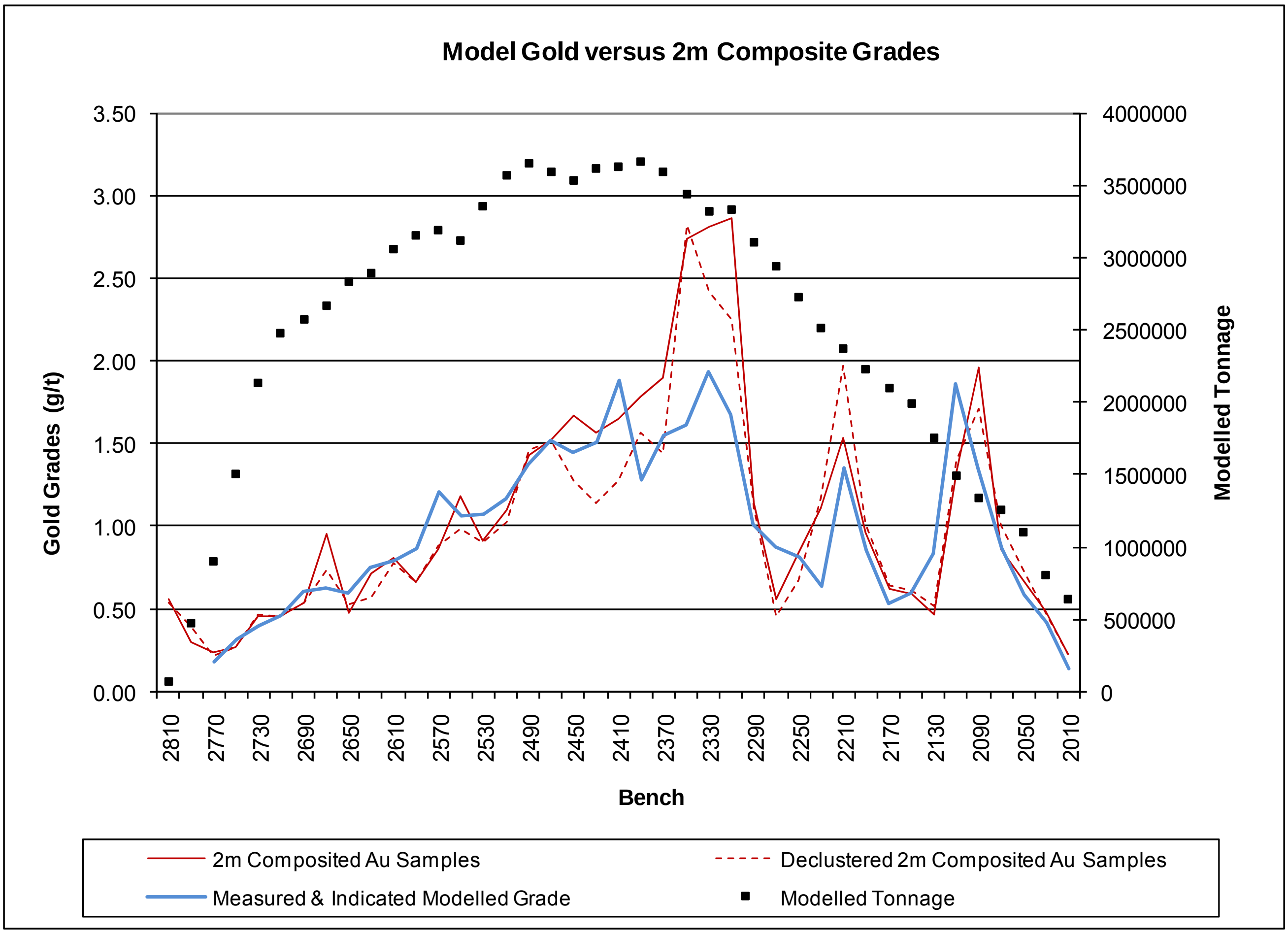



Figure 17-6 includes Inferred modelled mineralisation in the bench comparison, which is generally less well supported by sample data.

**Figure 17-6: Model (including Inferred) versus 2m composited sample gold grades by bench**

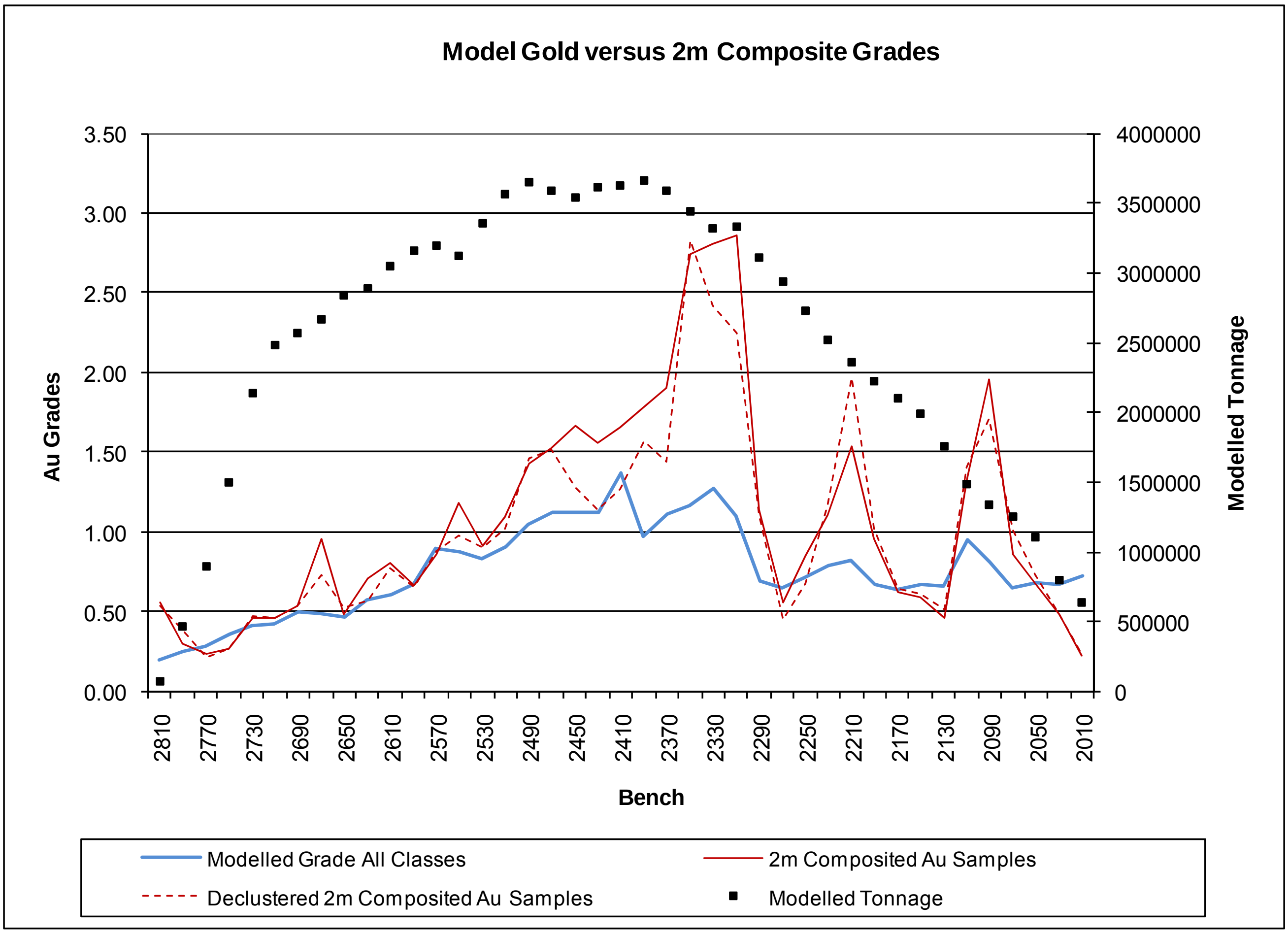
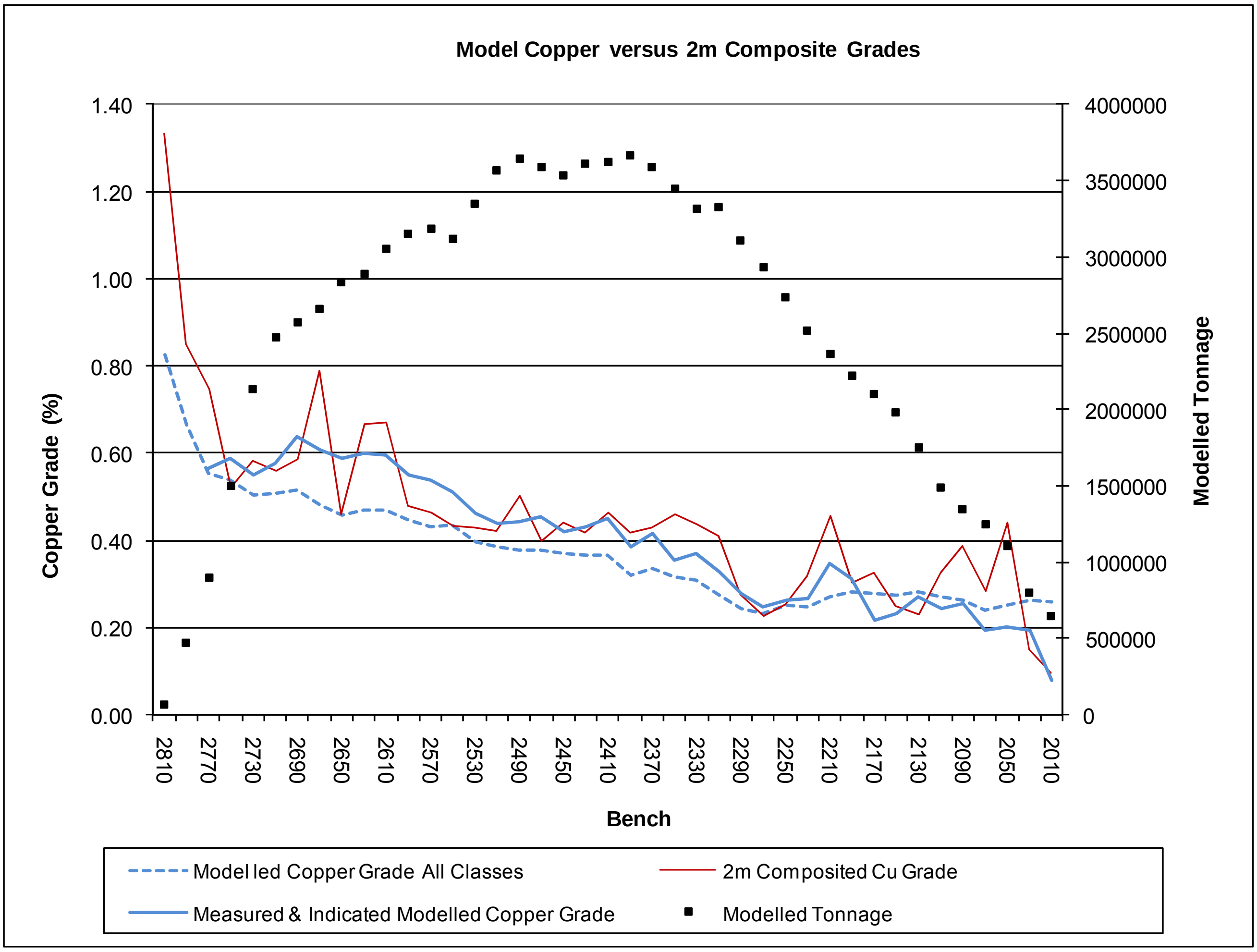



Figure 17-7 shows reasonable overall agreement between the modelled and composited grades and is not compounded by the grade spike above the Bufu syenite, seen in the gold comparison.

**Figure 17-7: Model versus 2m composited sample copper grades by bench**



The current resource estimate is compared with the previous, Hellman and Schofield estimate in Table 17-11. The two estimates are reasonably close in terms of combined Measured and Indicated resources, although the current estimate is more conservative in terms of the division between Measured and Indicated classes.

For the purposes of this comparison, both estimates have been tabulated using the gold equivalence and underground/open pit transition used in 2007. The previous economic parameters of US$500/ounce gold and US$1.9/pound copper led to the gold equivalence of eqAu = Au (g/t) + 2.61 x Cu (%). The underground/open pit transition was then 2270mRL.

**Table 17-11: Comparison of 2007 and current resource estimates, using 2007 economic parameters (at 0.4g/t eqAu cut-off grade above 2270mRL and 1.0g/t eqAu below)**

| Model | Class | Tonnes (Mt) | Au (g/t) | Cu (%) |
|---|---|---|---|---|
| H&S 2007 | Measured | 34.7 | 1.40 | 0.49 |
| | Indicated | 29.3 | 0.65 | 0.40 |
| | **Measured & Indicated** | **64.00** | **1.06** | **0.45** |
| | Inferred | 21.1 | 0.4 | 0.3 |
| OGC 2010 | Measured | 15.7 | 1.71 | 0.57 |
| | Indicated | 47.5 | 0.77 | 0.40 |
| | **Measured & Indicated** | **63.30** | **1.01** | **0.44** |
| | Inferred | 23.8 | 0.4 | 0.2 |

# 17.10 Mineral Reserve Estimates

## 17.10.1 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.

## 17.10.2 Open Cut Reserves

### 17.10.2.1 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. The parameters used smeared 10% of each block with each of its neighbours in an east-west direction. This process retains all the metal within the main body of the ore zone but shows the effect of dilution across the grade cut-off boundaries. It gives a realistic model of how overall dilution varies with mineralisation width on each bench.

In most cases the cut-off grade boundaries were unchanged and the ore blocks on the edges of the ore zones simply suffered a reduction in grade as they were diluted by dilution from sub-economic blocks outside the cut-off boundaries. In some cases, low-grade blocks were rendered uneconomic by dilution from even lower-grade blocks outside the cut-off grade boundary, while in other cases lower-grade blocks were rendered economic by metal smeared from higher-grade blocks inside the cut-off boundaries.

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.

Table 17-12 shows the smearing procedure diluting the gold grades of blocks within the pit design to produce a migration of metal into the lower-grade ranges.

**Table 17-12: Dilution effect of smear method**

| In situ grade | In situ in pit | | | Diluted In pit | | | Change | | |
|---|---|---|---|---|---|---|---|---|---|
| | tonnes | Au g/t | Au oz | tonnes | Au g/t | Au oz | tonnes | Au g/t | Au oz |
| > 5.0 g/t Au | 78,223 | 6.85 | 17,232 | 78,447 | 6.37 | 16,071 | 100% | 93% | 93% |
| 4.5 -> 5.0 g/t Au | 22,500 | 4.72 | 3,411 | 22,500 | 4.21 | 3,043 | 100% | 89% | 89% |
| 4.0 -> 4.5 g/t Au | 11,250 | 4.02 | 1,454 | 11,250 | 3.61 | 1,304 | 100% | 90% | 90% |
| 3.5 -> 4.0 g/t Au | 84,023 | 3.76 | 10,155 | 84,122 | 3.44 | 9,293 | 100% | 91% | 92% |
| 3.0 -> 3.5 g/t Au | 80,156 | 3.19 | 8,229 | 80,354 | 3.00 | 7,745 | 100% | 94% | 94% |
| 2.5 -> 3.0 g/t Au | 206,889 | 2.71 | 18,039 | 207,087 | 2.61 | 17,364 | 100% | 96% | 96% |
| 2.0 -> 2.5 g/t Au | 471,646 | 2.19 | 33,178 | 471,844 | 2.14 | 32,479 | 100% | 98% | 98% |
| 1.5 -> 2.0 g/t Au | 844,040 | 1.74 | 47,326 | 844,013 | 1.75 | 47,406 | 100% | 100% | 100% |
| 1.0 -> 1.5 g/t Au | 1,695,814 | 1.21 | 65,862 | 1,696,482 | 1.21 | 65,834 | 100% | 100% | 100% |
| 0.5 -> 1.0 g/t Au | 4,991,616 | 0.71 | 113,783 | 4,991,420 | 0.72 | 114,902 | 100% | 101% | 101% |
| 0.0 -> 0.5 g/t Au | 14,639,135 | 0.19 | 89,425 | 16,407,239 | 0.19 | 98,643 | 112% | 98% | 110% |

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

#### 17.10.2.2 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$800/oz for gold and US$2.60/lb for copper.

The estimates of process and site fixed costs provided by OGC for this analysis are:

| | | |
|---|---|---|
| First six years while open cut is being processed at 2.5 Mtpa | Process | US$9.50/t |
| | Site G&A | US$3.00/t |
| | Total | US$12.50/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$12.50 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 then very large low-grade stockpiles are accumulated to be processed once all the higher-grade material is depleted. This is undesirable at Didipio because there is limited space for stockpiles and if the sulphide ore stays on stockpile for too long it may partially oxidise and lose recovery.

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 of high-grade feed. In order to achieve this elevated cut-off strategy without generating excessive low-grade stockpiles, the open cut cut-off grade was set at US$15.00 per tonne. The potential ore between US$12.50 and US$15.00 NMV is 1.2 Mt with an average NMV of US$13.66.

It would return a margin of US$1.66 per tonne but most of this would be lost in stockpiling costs or lost recovery due to oxidation.

#### 17.10.2.3 Open Cut Reserves

Using a cut-off NMV of US$15.00 per tonne and a pit base at RL2540, the Didipio Gold-Copper Project open cut reserves are 13.87 Mt at 0.82 g/t Au and 0.63% Cu. The open cut ore was divided into two grade ranges based on NMV.

**Table 17-13: Open cut ore reserves by grade range**

| Ore type | Mt | Au g/t | Cu % | NMV range |
|---|---|---|---|---|
| Ore Grade | 12.22 | 0.89 | 0.68% | >25.00 |
| Low Grade | 1.65 | 0.27 | 0.25% | 15.00 – 25.00 |
| **Total** | **13.87** | **0.82** | **0.63%** | |

The average ratio of the open cut is 1.70 bcm of waste for each bcm of ore.

#### 17.10.2.4 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.

### 17.10.3 Underground Reserves

#### 17.10.3.1 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 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 17-14.

**Table 17-14: 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.

#### 17.10.3.2 Cut-off Grade

The cut-off grades used to define the stope boundaries use a similar methodology to the open cut, except that underground production costs (in ore development, production drilling, blasting and hauling) per tonne are added to the processing and site fixed costs. Underground production costs were estimated against preliminary mine schedules.

| | | |
|---|---|---|
| Remaining project life with underground ore at 1.2 Mtpa | Process | US$10.20/t |
| | Site G&A | US$4.60/t |
| | Underground production | US$26.21/t |
| | Total | US$41.01/t |

NMV values were composited over 30 and 60-metre intervals in the block model depending on the ground conditions. The composites were contoured to guide definition of the stope boundaries. The contours are irregular in places so the stope limits range between the US$35 and US$45 NMV contours in places. The stope boundaries fit practical working shapes to the NMV contours and the stopes are projected up 30 or 60 metres depending on the ground conditions. All the material inside the resulting stope volumes is subjected to the loss and dilution process and is classed as ore, regardless of whether individual blocks are above or below the NMV cut-off.

#### 17.10.3.3 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. No extraordinary risk factors were identified to warrant downgrading of the open cut reserve categories in the resource to reserve conversion.

### 17.10.4 Total Reserves

#### 17.10.4.1 Reserves as at September 2010

**Table 17-15: Ore reserves**

| **Open cut** | **Tonnes (Mt)** | **Au g/t** | **Au (Moz)** | **Cu %** | **Cu (kt)** |
|---|---|---|---|---|---|
| Proven ore | 6.06 | 1.23 | 0.24 | 0.74 | 44.8 |
| Probable ore | 7.81 | 0.50 | 0.13 | 0.55 | 43.0 |
| Total ore | 13.87 | 0.82 | 0.37 | 0.63 | 87.4 |
| Ore volume | 5.43 | | | | |
| Waste volume | 9.22 | | | | |
| Waste:Ore ratio | 1.70 | | | | |
| **Underground** | **Tonnes (Mt)** | **Au g/t** | **Au (Moz)** | **Cu %** | **Cu (kt)** |
| Proven ore | 5.51 | 2.62 | 0.46 | 0.53 | 29.2 |
| Probable ore | 10.30 | 1.76 | 0.58 | 0.51 | 52.5 |
| Total ore | 15.85 | 2.06 | 1.05 | 0.52 | 82.4 |
| **Total ore** | **Tonnes (Mt)** | **Au g/t** | **Au (Moz)** | **Cu %** | **Cu (kt)** |
| Proven ore | 11.57 | 1.90 | 0.71 | 0.64 | 74.1 |
| Probable ore | 18.15 | 1.21 | 0.71 | 0.53 | 96.2 |
| Total ore | 29.72 | 1.48 | 1.41 | 0.57 | 169.4 |

Notes accompanying Table 17-15:

The open cut reserves use a Net Metal Value (NMV) cut-off of US$15.00 per tonne calculated using processing, smelting and refining recoveries and processing costs, site fixed costs and realisation costs. Metal prices of US$800/oz for gold and US$2.60/lb for copper were used.

The underground reserves are based on a sublevel open stoping mine layout. The designed stope boundaries are based on a US$41.00 NMV cut-off. Metal prices of US$800/oz for gold and US$2.60/lb for copper were used.

The tonnes and grades are stated to a number of significant digits reflecting the confidence of the estimate. Since each number and total is rounded individually the columns and rows in the above table may not show exact sums or weighted averages of the reported tonnes and grades.

The Qualified Person for NI 43-101 compliance with regard to the mine planning is John Wyche (BE(Min), BComm, MAusIMM(CP), MMICA), of Australian Mine Design and Development Proprietary Limited.

# 18 OTHER RELEVANT DATA AND INFORMATION

## 18.1 Density

Density data and calculations are presented in sections 14 and 17.

## 18.2 Topography

Contours were formed by ground survey at a five-metre interval covering the three valleys (Surong, Upper Dinauyan and Didipio). The accuracy has been observed to be ~ 0 – 0.35m. This is interpolation error between observed ground heights. The contour plan was amended using new values where possible and contours calculated to 2.5m for specific areas (for design or other purposes such as soil volume calculations).

# 19 INTERPRETATIONS AND CONCLUSIONS

The Didipio Gold-Copper Project contains significant mineral resources defined by existing data in the Didipio Ridge deposit. There is limited potential for expanding existing resources, but there is significant potential to discover and define additional resources within the Didipio Project area at a number of other prospects. The existing database for Didipio Ridge is considered satisfactory for resource estimation, although some minor issues with data completeness and quality remain to be resolved.

BDA considers that the capital cost estimates for the Didipio Project over the next 4-5 years are generally reasonable and consistent with planned developments and are accurate within ±15%. There is a reasonable probability that some variations to capital requirements will arise later in the project life.

# 20 RECOMMENDATIONS

## 20.1 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 will 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.

OGC considers that the existing database for Didipio is satisfactory for resource estimates as reported by appropriate confidence categories using the JORC/CIMM guidelines, and that the resource estimates are satisfactory for the purposes of proceeding with development and construction of the project.

## 20.2 Capital Costs

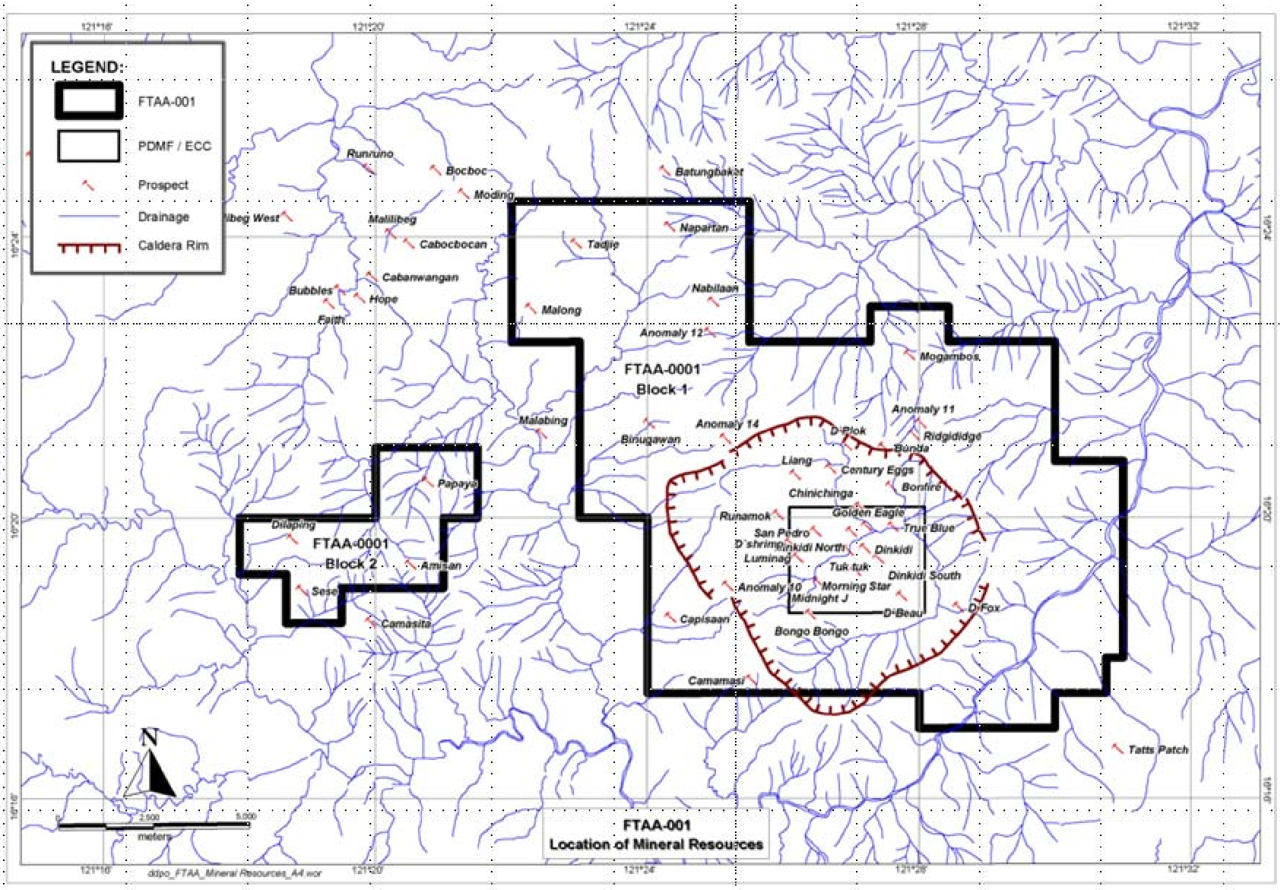
BDA recommends that both initial capital costs and ongoing requirements for deferred and sustaining capital be monitored closely and estimates revised as necessary throughout the life of the project. BDA notes, however, that the economic model for the project is not overly sensitive to capital increases within the estimated range of accuracy of ±15%.

## 20.3 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 Gold-Copper Project. Figure 20-1 shows the exploration prospects within the Didipio FTAA.

**Figure 20-1: Exploration prospects within the Didipio FTAA**




Some of the more advanced prospects nearer the Didipio Gold-Copper Project have been partially drill tested, notably the True Blue and D'Fox alkalic porphyry prospects. Limited drill results to date for both prospects are consistent with lower-grade porphyry-style gold-copper mineralisation. Future exploration activities will focus on definition of higher-grade domains at each of these prospects.

The 2011 exploration programme will focus on increasing the mineral resource coming from within FTAA-001 area (near mine areas) as an additional ore for the Didipio mine. This programme will also assess the mineralisation potential of all areas with approved exploration permits within the FTAA-001 area.

A provisional exploration budget of approximately US$1.5 million has been allocated to the Didipio FTAA for 2011. These expenditures are not included in the financial analysis.

# 21 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

## 21.1 Mining Operations

### 21.1.1 Project Description

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

The Dinkidi Gold-Copper 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.

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.

### 21.1.2 Resource Model

#### 21.1.2.1 Resource Block Model

The resource block model was prepared by Jonathan Moore of OceanaGold Corporation (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.

#### 21.1.2.2 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.

#### 21.1.2.3 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 Didpio 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.

## 21.1.3 Open Cut Mining

### 21.1.3.1 Basis of Open Cut Mine

Pit optimisation studies conducted from 2003 to 2009 indicated an optimal pit shell extending down to 2400RL, or 300 metres below the valley floor. However, the known high-grade mineralisation extends to at least 2000RL. If the maximum potential of the resource is to be realised, at least part of the reserve must be recovered by underground mining. This makes it necessary to select an open cut/underground cut-off point. The factors affecting this decision for Didipio are:

- The underground mining method selected for Dinkidi is sublevel open stoping with cemented paste fill. This method requires a long, costly development period before full production is reached.
- The tailings dam wall requires up to 6.9 Mbcm of bulk fill. If this is sourced from open cut waste it carries nil extra cost to the project. If there is no open cut the fill must be won from somewhere else during the period that considerable costs are being incurred on underground development with no compensating revenue. Based on the average open cut mining costs, this would add almost $40M to the initial capital cost of the project.
- The mill throughput is designed at 2.5 Mtpa. This rate is readily achievable by open cut mining but cannot be sustained by the current underground mining plan.
- Open cut cash flow scenarios run at 2.5 Mtpa show that present value is maximised if an elevated cut-off grade is used to maximise metal production in the early years, with low-grade ore being stockpiled for processing at the end of the mine life when the high grade is depleted. As a result, mine planning focuses on the higher-grade portion of the potentially economic open cut ore tonnage and the remaining low grade, which forms 12% of the open cut reserve, is stockpiled for processing at the end of the open cut mine life.
- Considering the previous factor, in order to minimise mining costs the open cut should ideally only extend to the depth where the incremental cost per tonne of ore (not including low grade) is equal to the cost of mining a tonne of ore from that depth by underground means. The incremental open cut cost referred to is the cost of stripping all the pit walls and floor to take the pit one metre deeper divided by the ore tonnes (excluding low grade) recovered in that metre.

- Once the tailings dam wall, ROM pad and pit are in place, there is very limited storage room available on the site for low-grade ore and waste rock. As the pit becomes deeper it takes up more area and generates more low grade and waste rock so the storage shortage becomes more acute.

Considering these factors, the open cut depth and lateral extent must be selected to:

- Provide high-grade feed at 2.5 Mtpa at no greater cost per ore tonne than the cost of mining comparable ore from underground.
- Establish the high-grade ore feed quickly to meet the mill startup and maintain the feed for as long as it takes to develop the underground mine to a point where it can achieve its full production rate.
- Provide sufficient waste to meet the bulk fill requirement of the tailings dam wall, but avoid generating more low-grade ore and excess waste than the available storage areas will hold.

The pit selected against these objectives has a broad base at 2540RL, taking in the high-grade zone at the northern end of the orebody. The underground stopes will eventually extract ore right up to the pit base.

#### 21.1.3.2 Open Cut Mining Objectives

The objectives of the open cut design are to:

- Provide ore of sufficient grade to average 70,000 eqAu oz (production) per year until the underground mine reaches full production;
- Minimise pre-production mining to reduce initial capital;
- Provide waste for construction of the run of mine (ROM) stockpile base and the tailings dam wall; and
- Achieve the above objectives with minimal cost and maximum present value.

#### 21.1.3.3 Factors Affecting Open Cut Mining

##### 21.1.3.3.1 Pit Wall Stability

The main factors affecting pit wall stability are:

- The Biak Shear Zone, which runs from north-west to south-east across the northern quarter of the pit;
- Zones of broken ground north of the Biak Shear, which were identified in test bores drilled to model the groundwater regime;
- Hydraulic pressure from the groundwater; and
- The depth of the pit, because all of the above factors become more important as the overall wall heights increase.

##### 21.1.3.3.2 Open Cut Water Management

Open cut dewatering will be managed by:

- A dewatering borefield installed around the perimeter to draw down the water table in the Biak Shear Zone and other minor water-bearing structures, and
- In pit sump pumps.

Total pumping rates across the two systems are expected to peak at 8 Ml/day in the first year when initial drawdown is occurring in the Biak Shear Zone and then fall to 5-7 Ml/day as the pit becomes deeper and wider. Around 4.8 Ml/day of this volume is expected to come from direct rainfall on to the pit, with the balance coming from ground water sources, predominantly the Biak Shear Zone.

### 21.1.3.4 Pit Optimisation and Design

A series of pit optimisation studies have been run on the Dinkidi orebody since 2003. The scenarios examined have been in response to improvements in the resource model, better definition of mining costs, changing gold and copper prices and testing of strategies for transition from open cut to underground mining.

The most recent set of pit optimisations was conducted in early 2009. It was designed to assess the value of the project as an open cut-only mine and it tested production rates of 1.0 to 3.0 Mtpa. Although gold and copper prices have increased significantly since these runs and mining costs have been reassessed by local contractors, the results are still relevant guides to the current design because they confirmed earlier runs that showed that, on a purely commercial basis, an open cut mine can be taken to more than 300 metres below the valley floor.

However, every detailed mine design and schedule since 2003, including checks done during 2009 and 2010, have shown that maximum project value is obtained by having an open cut and an underground mine and that the open cut base should be close to 2540RL, or 150 metres below the valley floor. The reasons for this limitation on the open cut are:

- Below 2540RL the incremental mining cost per tonne of ore (the cost to mine all the ore and waste to widen and deepen the pit by one bench divided by the tonnes of ore in that bench) starts to approach the average underground production mining cost of US$21.60 per tonne. Having decided to spend the capital to develop the underground decline and support services, it starts to become cheaper to mine ore underground below 2540RL.

- In order to maximise ore recovery from the stopes immediately below the pit floor, it is preferable to have the final pit floor covering most of the area of the top stoping panel. This avoids difficult mining shapes under and close to the pit walls. Even though the optimisation shows that a pit could go to RL2480, it is better for the open cut/underground transition if it is truncated at RL2540 to provide a wide pit base.

- The larger the pit, the more waste rock is generated. The waste volume of pits designed to wide pit bases at RL2540 is close to the maximum that can be stored in the available area between the tailings dam and the pit.

- The larger the pit, the more low-grade ore is mined. Since the elevated cut-off strategy is used to maximise early cash flows, the additional low grade must be stored. Once the pit goes below 2540RL the volume of low-grade ore exceeds the available storage room. Also, the larger tonnage means that some of the low grade will remain on stockpile longer, increasing the chance of oxidation and recovery loss.

- The underground mine plan relies on developing the decline off the final pit wall to avoid boxcut and decline development through deeply weathered surface materials adjacent to the pit. Having a larger, deeper pit means that it takes longer to reach the portal position on the final pit wall, which in turn delays the start of high-grade ore delivery from the underground mine.

- The dewatering bores become much more expensive to develop and operate below 150 metres in depth. As well as allowing relatively dry mining on each bench, these bores depressurise the pit walls. The pit geotechnical analysis assumes drained walls.

- Larger pits create additional surface area to collect rain water, which must be pumped from the pit. Some of this water also percolates into the underground mine and must be pumped out.

Given the above considerations, it was decided to use the Whittle pit optimisations to define a pit shape to maximise value but with the constraints of:

- Having a wide pit base no deeper than 2540RL, and

- Limiting waste and low-grade volumes to available storage volume.

These constraints were met by limiting the optimisation to a maximum depth at 2500RL and selecting the highest value shell truncated 40 metres above the optimised base so that it covers most of the top stoping panel area. Checks on waste volumes showed a good match with available storage in the tailings dam wall construction and the waste dump between the wall and the pit.

Although pit selection in this case was governed by factors other than simply pit value optimisation, the process is still important because it confirms the viability of the pit since the shell selected lies inside the shell that forms if no depth constraint is applied and it provides design guidance for selecting pit stages and positioning final pit walls.

The following sections describe the optimisation inputs used in early 2009 along with current estimates where they have changed significantly. This highlights the fact that if for any reason it is decided not to proceed with the underground mine, there is scope to expand and deepen the existing pit design.

#### 21.1.3.4.1 Optimisation Inputs

**Mining costs**

Based on a detailed tender submitted by Leighton Contractors with explosives quoted by Orica and diesel supplied by OGC. Average mining cost of the optimised shell was US$2.33/tonne ore and waste.

Current financial model uses all inclusive rates quoted by Filipino civil and mining contractors in November 2009. Average cost applied against current production schedule is US$2.17/tonne.

**Mill feed rate**

January 2009 optimisations were run at 1.0, 1.5, 2.0, and 3.0 Mtpa and the shell to guide the design was taken from the 2.0 Mtpa run.

Current mine plan is for 2.5 Mtpa during the open cut phase.

**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$9.50/tonne of ore at 2.5 Mtpa.

**Fixed costs**

Based on 2007 estimate of US$7,200,000 per year factored up by 20% to US$8,640,000 per year, or US$4.32 per tonne of ore at 2.0 Mtpa.

Current financial model uses US$7,500,000 per year, or US$3.00 per tonne of ore at 2.5 Mtpa.

**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 21-1).

**Table 21-1: Overall pit slopes**

| Domains | Weathered rock | Fresh rock |
|---|---|---|
| Bench height | 15 m | 20 m |
| Bench slope | 55° | 60° |
| Berm width | 10 m | 5 m |
| Inter-ramp angle | 36° | 50° |

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 21-2.

**Table 21-2: Adjusted slopes for pit optimisation**

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

**Ore loss and dilution**

A "smearing" procedure was run on the resource block model to simulate the effect of movement of material across defined ore boundaries caused by blasting and excavation. The technique and its dilution effect are more fully explained in section 17.10.2.1.

**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 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 the 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 |

**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$800/oz for gold and US$2.60 per pound for copper.

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

#### 21.1.3.4.2 Final Pit Design

Figure 21-3 shows the completed final pit. The main points to note about the final pit design are:

- The final pit floor at RL2540 matches the highest grade area of the top underground stoping panel.
- There is only one ramp. It includes switchbacks on the south-west and north-east sides to avoid having the main access cross the Biak Shear Zone.
- There are two pit exits off the ramp. The western pit exit reduces haul distance for waste and low-grade ore. The eastern pit exit reduces the haul distance for ore going to the ROM stockpile.
- Most of the ramp is 20 metres wide to allow two-way haulage for 35-tonne trucks. The ramp width reduces to 15 metres from 60 metres above the final pit floor to allow the pit to go deeper at a more narrow width. This will require one-way haulage.

- The highest wall is on the south-east corner and is 235m from floor to crest. This wall will be formed in the more competent Tunja and Dark Diorite rock. The northern wall, which is in more broken ground, is only 150m in height.
- There is 60m width between the northern crest and the start of the ridge along the north side of the Diduyan Valley. This leaves ample room for a diversion channel to keep any overflow water from the tailings area and waste dump away from the mined out pit while the underground mine is in operation.

### 21.1.3.4.3 Staged Pit Designs

Pit stages 1 and 2 (see Figure 21-1 and Figure 21-2) allow the open cut to expose high-grade ore early in the mine life and to maintain ore supply while the upper waste sections of the successive stages are being mined.

During the early stages of mining, waste from the above-ground benches of all three stages will be mined to provide construction fill for the tailings dam wall.

**Figure 21-1: Stage 1 pit**

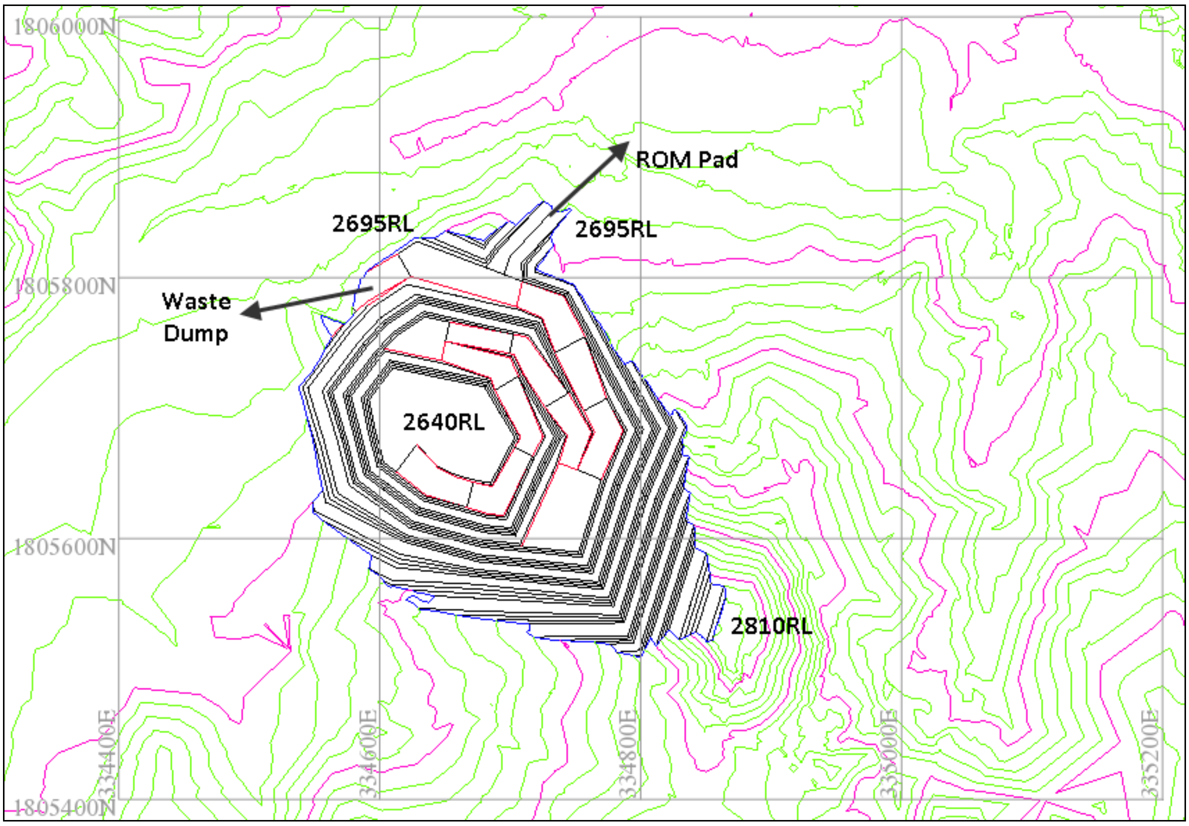


**Figure 21-2: Stage 2 pit**

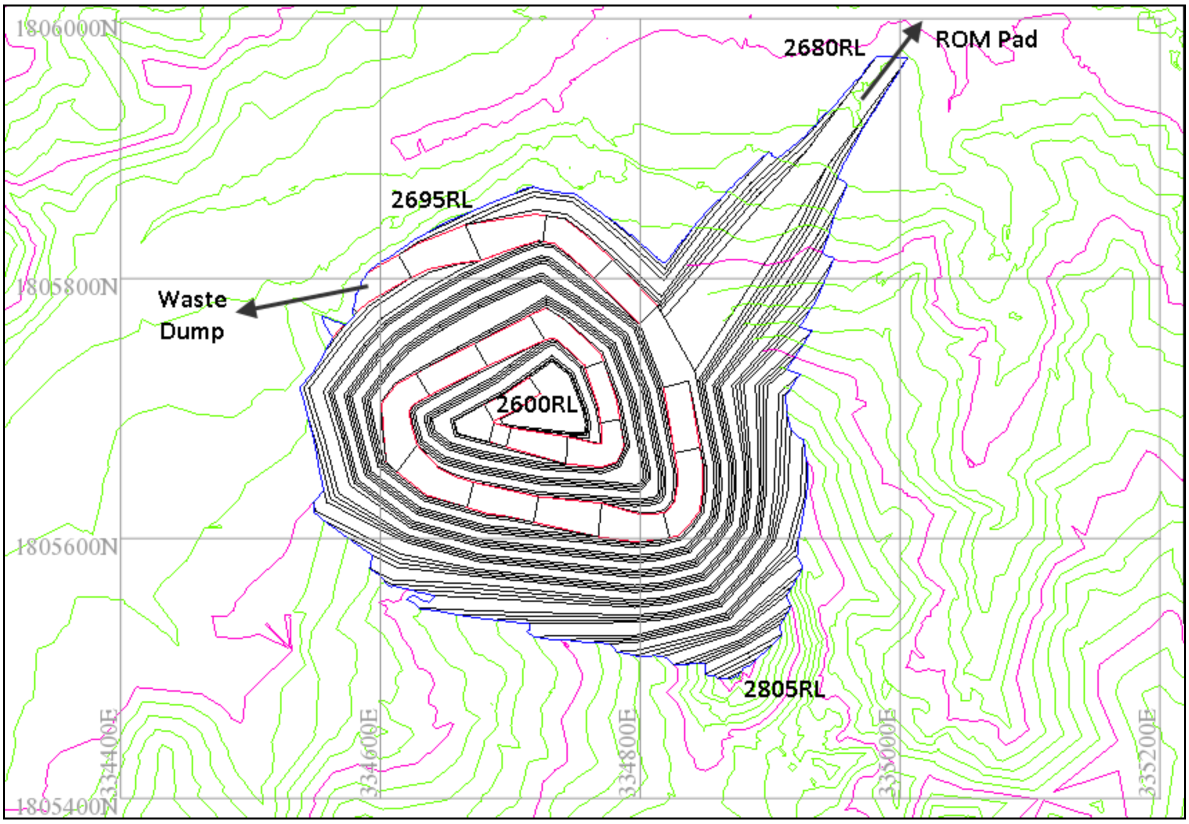
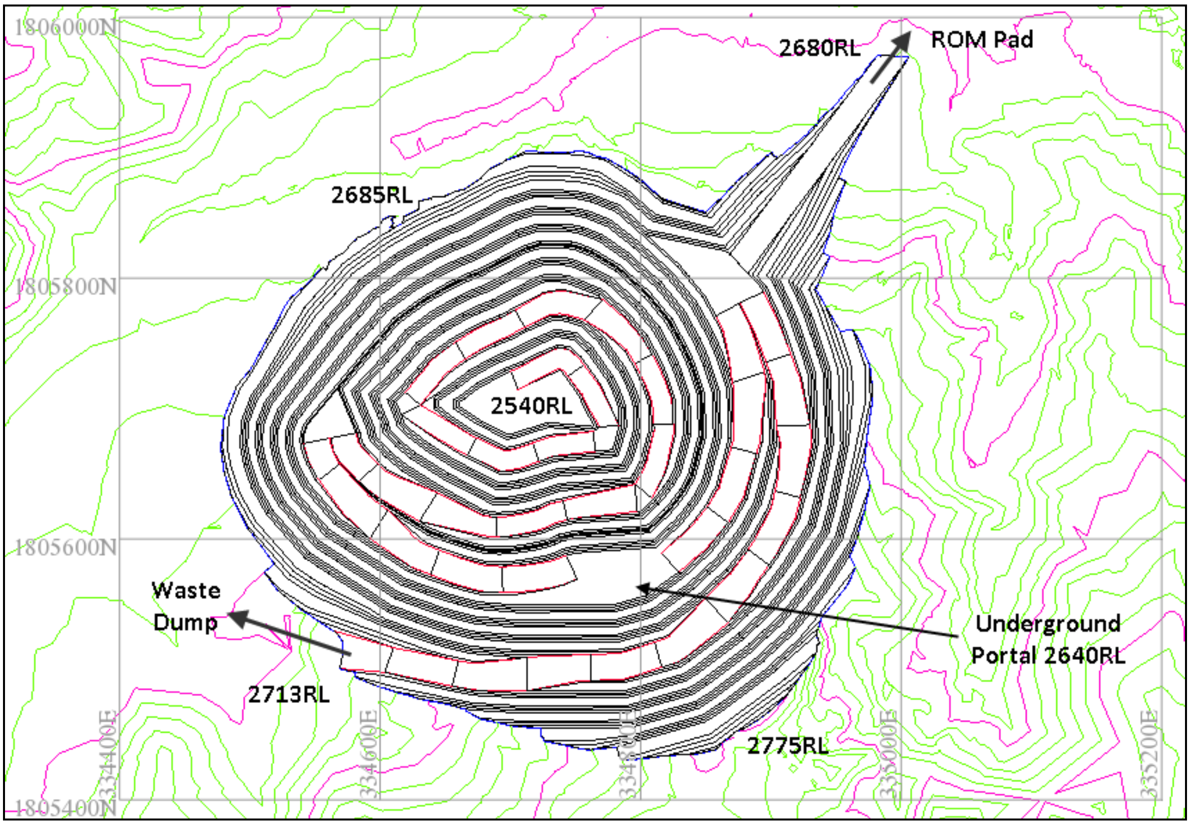



**Figure 21-3: Stage 3 pit (final)**



### 21.1.3.5 Mining Operations

#### 21.1.3.5.1 Grade Control

A dedicated reverse circulation (RC) drill rig will conduct grade control sample drilling across the mining benches. RC drilling is used in preference to blast hole sampling to ensure representivity of the samples and to keep the sampling and ore definition well ahead of mining.

#### 21.1.3.5.2 Drill and Blast

The study assumes that all ore and waste will be drilled and blasted.

#### 21.1.3.5.3 Groundwater Management

Hard-rock Consultants (Meyer 2006) created a groundwater flow model that was calibrated against the results of the long-term pump test supervised by Coffey in 1998.

Peak groundwater inflows to the pit of up to 13 to 15 million litres per day are expected during the wet season. Most of this should come from the Biak Shear Zone and the broken ground north of it. A much smaller flow is expected through the Tatts Fault, which runs north-south through the pit and is the main host structure for the mineralisation.

The pit will be kept free of groundwater by installing a system of 10 advance dewatering bores prior to mining to depress the water table in the pit area by up to 30m below the pit floor. These bores will be drilled at 450mm diameter and lined with 300mm slotted casing to house 200mm diameter downhole pumps. Most of the holes will be drilled to 180m depth to depress the water table below the pit floor over the full four-year life of the open cut. The total installed capacity of the bore-field will be 13ML/day. However, the annual abstraction rates modelled average 5-8ML/day.

In-pit sump pumps will remove any groundwater that seeps into the pit.

Analyses of the groundwater in the pit area show mild salinity and some elevation of arsenic, zinc, iron, manganese and sulphate. These naturally occurring levels appear low enough to allow discharge to the Surong River.

#### 21.1.3.5.4 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.

#### 21.1.3.5.5 Ground Support

Slope stability is a significant issue for the open cut because the Biak Shear and the broken ground on the northern walls may be prone to rubble failures and the pit only has one ramp. The measures planned to mitigate the slope stability risk include:

- Restricting all faces to a maximum height of 15m;
- Increasing berm widths in and north of the Biak Shear Zone;
- Flattening the face slopes and further widening the berms over the whole pit area above the base of total oxidation;
- Advance dewatering to keep the water table below the pit floor, thereby reducing hydrostatic pressure in the walls;

- Preventing surface water flows from entering the pit;
- Allowing for good in-pit drainage of direct rainfall to minimise erosion of the face and berm crests;
- Drilling drainage holes into the walls of the pit at regular intervals;
- Allowing for pre-split blasting of all walls;
- Allowing for meshing of the open ramp edges;
- Allowing for cable bolting and/or meshing of faces as required;
- Allowing for ongoing slopes monitoring and design through the life of the pit; and
- Applying all the above measures to both pit stages 1 and 2.

Since the pit will only be in operation for five years and it will be mined in three stages, the interim and final walls are open for shorter periods than the full mine life. This further reduces the risk of major failures.

#### 21.1.3.5.6 Waste Dumps

Waste rock mined from stage 1 during construction (Years 0) will be used to form the ROM and stage 1 of the tailings dam wall. From the start of Year 1, part of the waste will be used to heighten the tailings dam wall and 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.

Since the underground mine will extend the mine life well beyond the open cut phase and almost all of the fill for the tailings dam wall will be sourced from open cut waste, the tailings dam wall will be built to final height before completion of the open cut mine.

Waste rock characterisation tests conducted during 2006 found little potential for acid generation in the dumps. However, if potentially acid forming material is identified in the waste it will be placed against the dam wall on each lift and buried in non-acid forming waste to prevent rapid oxidation and acid formation.

#### 21.1.3.5.7 Haul Roads

Ore and waste will be hauled up 10% grade ramps within the pit stages. The ramps will be 15m wide for the first 60 vertical metres above the final floor of each stage and will then widen to 20m. A safety bund will be formed along the open side of each ramp to one-half of the wheel height of the largest truck. A two-metre wide spoon drain will be formed on the wall side of each ramp.

Ore haulage from the pit to the ROM will be via the eastern pit exit.

Waste will be hauled up a 20-metre wide, 10% grade road to be formed in the eastern face of the waste dump. The entry to this ramp is adjacent to the pit ramp exit on the western side of the pit.

#### 21.1.3.5.8 Ore Stockpiles

The stockpile strategy is designed to:

- Provide 2.5 Mtpa of ore for mill feed over the first four years at a grade that will produce at 50,000 to 80,000 ounces AuEq each year across the gold in dóre, gold in concentrate and copper in concentrate products;
- 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.

#### 21.1.3.6 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:

- Loading – two or three hydraulic excavators in the 80-110 tonne size range;
- Haulage – 7-10 off-highway dump trucks with 35-50 tonne payload capacity;
- Blast hole drills – two 110mm diameter percussion drills. One of these will have reverse circulation drilling capacity for the grade control work; and
- Support equipment – including bulldozers, graders, water carts, lighting plants, pit sump pumps, and maintenance and servicing vehicles.

### 21.1.4 Underground Mining

#### 21.1.4.1 Underground Mining Objectives

- Provide a consistent ore supply of 1.2 Mtpa with a ramp-up to production as the open pit is being completed. Development of the underground mine would commence once the open pit has been mined deep enough to establish the decline portal. Once the portal is established, the underground development and production would be independent of the open pit operations.
- Decide on a mining method that is low risk and will provide ore to the mill at the end of the pit life.
- Minimise mining costs, particularly capital costs.

#### 21.1.4.2 Factors Affecting Underground Mining

##### 21.1.4.2.1 Geotechnical Conditions

A substantial volume of geotechnical work has been completed to support a range of possible mining methods, including caving methods and sublevel open stoping (SLOS). In relation to SLOS, AMC Consultants advised that:

- Stable stopes can be mined to a height of 60m, except in areas of 'poorer' ground conditions, i.e. within the Bugoy breccia rock unit, where the stable height is limited to 30m.
- The stope size can be no larger than 20m x 20m in the plan area.

Sublevel development costs also make allowance for full passive ground support, instead of active ground support, based on the plan area of Bugoy breccia expected on each sublevel.

##### 21.1.4.2.2 Groundwater Inflows

If a caving method were to be used, large groundwater inflows would be expected from the Biak Shear Zone either through existing fractures or through new fractures generated as caving progresses. However, the SLOS extraction configuration would not intersect the Biak Shear Zone and would not be as succeptible to these groundwater inflows.

##### 21.1.4.2.3 Surface Water Inflows

If a caving method were to be used, rainfall directly on to the final open cut footprint (160,000m$^2$) would funnel large volumes of water into the caving zone during major rain events. Such inflows, in addition to the base inflow from the Biak Shear Zone, would likely result in flooding of the lowest development levels before the water could be pumped back to the surface. The use of SLOS avoids these large water inflows as the underground workings would not connect through to the ground surface, apart from small access and ventilation openings.

#### 21.1.4.3 Selection of Mining Method

Three caving methods were assessed: front caving, block caving and sublevel caving, as well as sublevel open stoping. The advantages and disadvantages of each method for possible use at Didipio are outlined below.

#### 21.1.4.3.1 Front Caving

Front caving would be cheap to establish, but would incur excessive ore loss and dilution. Like block caving, development would be required only at the base of the cave zone, which can be one, two, three or more extraction levels, depending on the 'behaviour' of the rock mass. This method does not require the intensive development and therefore ground support inherently associated with block caving, but all the 'draw-cones' within the cave would be drawn down next to a dilution wall of material.

#### 21.1.4.3.2 Block Caving

Block caving is attractive for the Didipio Gold-Copper Deposit because it would deliver high-grade from the base of the deposit, would be cheap to run, can be operated at high production rates (compared with other mining methods) and would allow more control of the draw to manage loss and dilution. However, preliminary cost estimates showed that block caving development costs during the open cut mining phase are still much greater than the other methods considered. It would also carry a higher risk of production delays through hang-ups.

#### 21.1.4.3.3 Sublevel Caving

Sublevel caving would be less risky, but more expensive, than block caving and front caving because it must blast all the ore. It would start at the lower-grade top of the Didipio Gold-Copper Deposit, below the base of the open cut, and work down. Although it would be more expensive and would incur more loss and dilution than block caving, it would be the preferred caving method due to its lower initial development cost, more rapid and cheaper establishment and lower risk of production delays. High levels of ore recovery with minimal dilution would be expected for sublevel caving because the surrounding rocks are mostly competent and metal grades continue into the zones where most of the dilution is expected to come from.

All the caving methods examined carry a high risk of potentially catastrophic flooding from rain events because the open cut mine acts as a funnel directly into the cave zone. Various water removal systems, including a 6km drainage tunnel, have been examined but they are all expensive to install and carry high operational risks.

#### 21.1.4.3.4 Sublevel Open Stoping

Open stopes would be mined in a "checker-board" sequence defined by the initial extraction of primary stopes and their subsequent filling with cemented fill, before extraction of the adjacent secondary stopes. The use of cemented fill in this way would maximise resource recovery while maintaining the integrity of the surrounding rock mass outside the stopes, thus avoiding the water inflow risks associated with the caving options. While SLOS has the lowest risk of the methods considered, the development and cemented fill requirements make this the most expensive method of those under review. Other mechanised mining methods would be significantly more expensive.

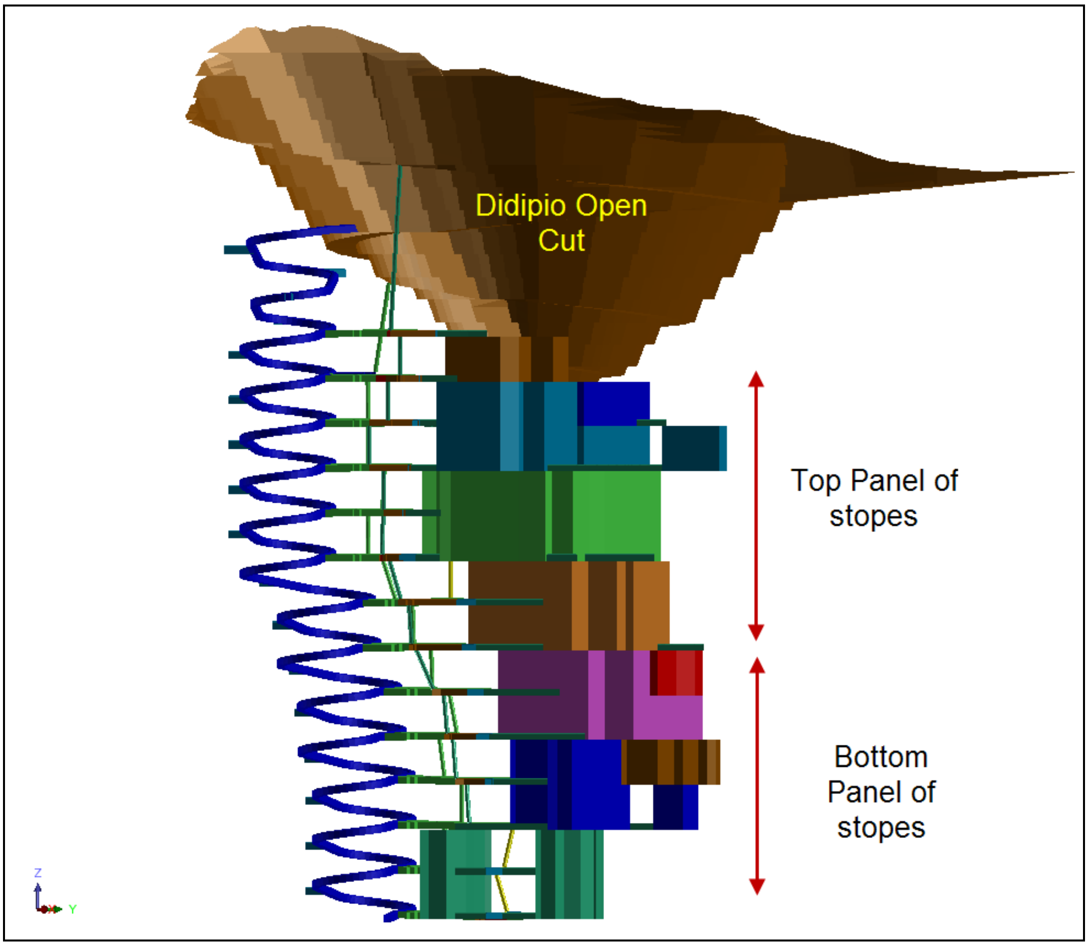
Due to the risks associated with the caving options, SLOS was selected as the preferred mining method.

### 21.1.4.4 Sublevel Open Stoping Design

The key features of the SLOS design are described.

- As shown in Figure 21-4, the orebody is divided vertically into a Top Panel and Bottom Panel, separated at a predetermined level. Within each panel, extraction will advance upwards.

**Figure 21-4: Top and bottom SLOS production panels**




- Sublevels are developed at 30m vertical intervals. Each sublevel design is based on a cut-off grade model derived for this mining method applied to the block model to determine the plan extent of each 30m lift.
- 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.
- Development headings will be mined using conventional drill and blast development equipment and all waste and ore will be hauled out of the mine using a fleet of five 20-tonne loaders and up to nine 50-tonne trucks. Development rates are based on current practices in the Philippines.
- Each stope would be extracted by first opening up an expansion void at one side of the stope. Once this expansion 'slot' has been developed, the stope would be opened up by drilling and blasting of main rings.
- The fragmented rock would be loaded out or 'mucked' from drawpoints at the base level of the stope.
- Haulage options considered included shafts, conveyors and trucks. Conventional truck haulage up a decline was selected to minimise capital costs and development lead time. Costs and production rates are based on the latest 50-tonne rigid body underground haul trucks.
- Once the stope has been mucked 'clean', it would be backfilled.
- Scheduling and costing is based on owner/operator estimates.

Details of the various design elements are provided in the following subsections.

### 21.1.4.4.1 Underground Design Parameters

Table 21-3 summarises the basic design dimensions used throughout the underground mine design.

**Table 21-3: Underground design criteria**

| Design parameter | Value |
|---|---|
| Height between extraction sublevels | 30m |
| Decline profile | 5.0mw x 5.5mh |
| Lateral access development profile | 5.0mw x 5.5mh |
| Drill drive and draw point drive profile | 5.0mw x 4.5mh |
| Stope dimensions | 20m x 20m |
| Ventilation raise profile (Alimak) | 3.5 x 3.5m |
| Decline gradient | 1 in 8 |

### 21.1.4.4.2 Mine Access

Access to the underground is by decline, as shown in Figure 21-5 and Figure 21-6, and primarily via a portal established in the side wall of the last stage of the open-cut development. The advantages of this are:

- The underground development can begin in competent rock at depth in the open cut, at an elevation that lies below the oxidised zone; and
- By starting the main underground access at a lower elevation, less development is required to reach the first ore source.

Ventilation rises will also be collared below the top of the open cut, again avoiding developing those rises through the oxidised zone.

**Figure 21-5: View looking north-west showing access decline with portal in open cut, and vent rises**

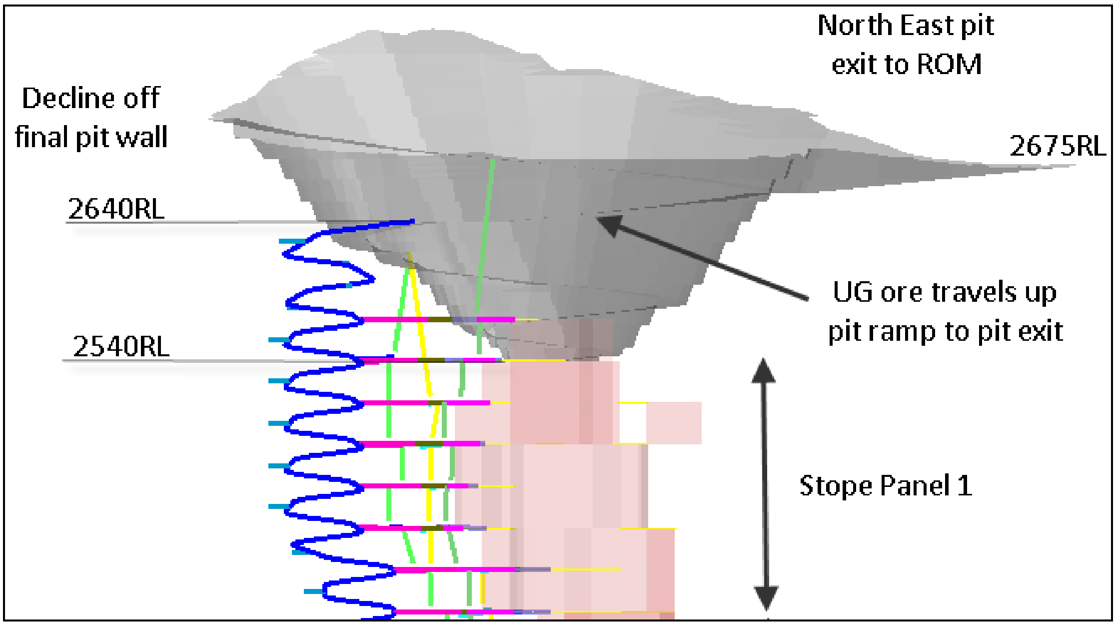
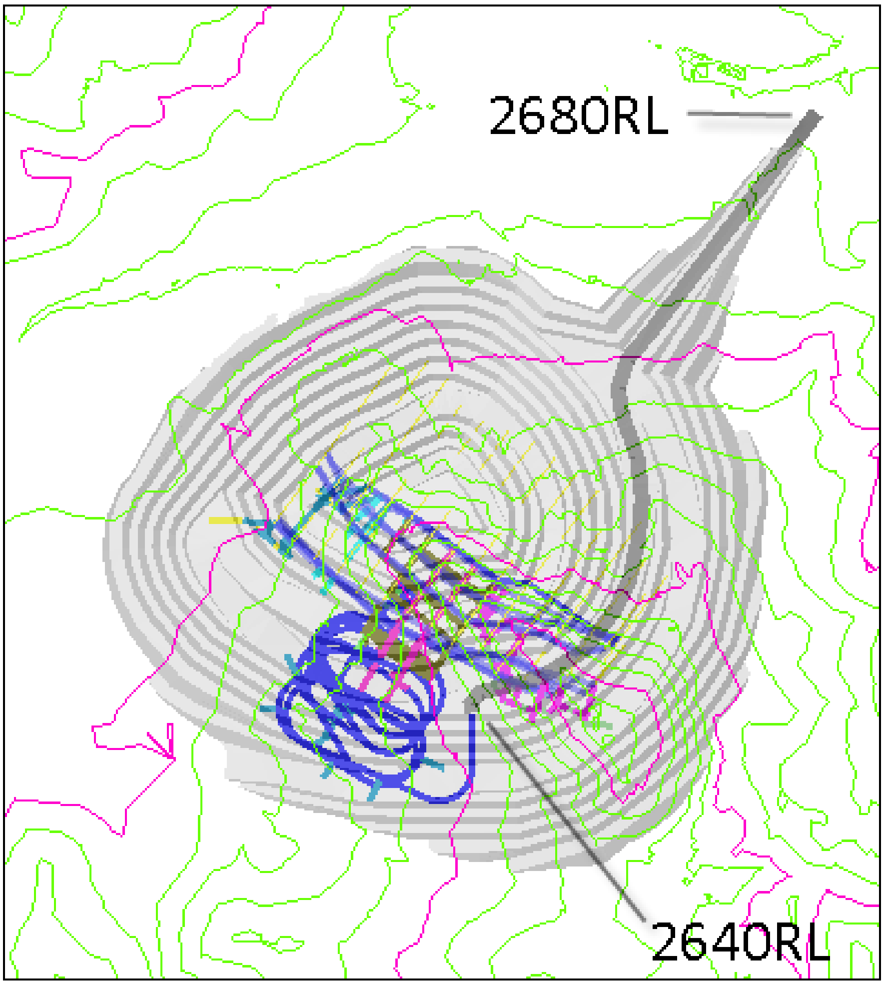


**Figure 21-6: Plan view showing access decline with portal in open cut**




The decline is designed with a one-in-eight gradient. This gradient may be conservative, with some existing operations in Australia using one-in-seven. This was used to provide some in-built contingency for future design work on the decline.

The main decline is designed with a cross-sectional profile 5m wide by 5.5m high. This profile was selected for the following reasons:

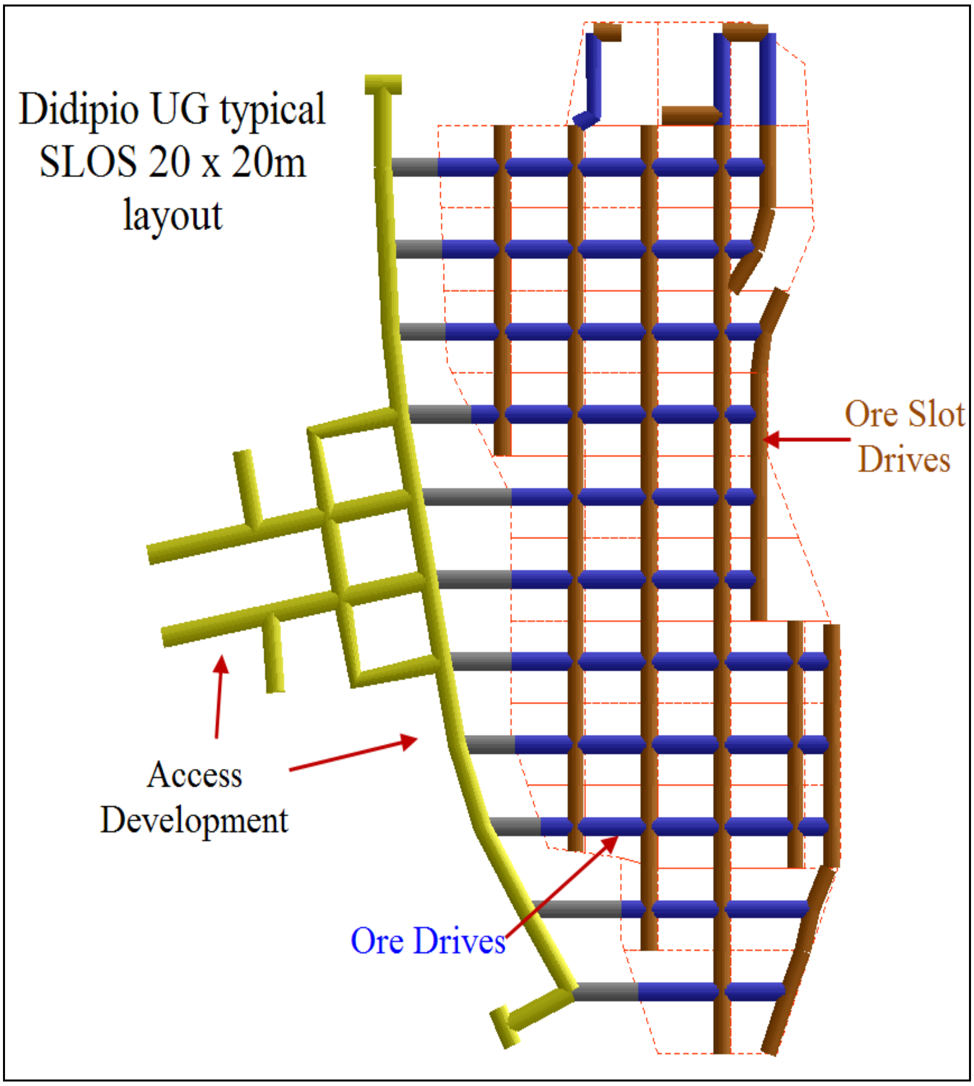
- It is sufficient for a 50t to 60t haulage truck;
- It provides adequate cross-sectional area to accommodate two large ventilation ducts, each 1.2m in diameter; and
- The cross-sectional area (approximately 26sq m) has capacity for up to 180 cubic metres of air per second and the decline will be used as one of the primary sources of fresh air into the underground workings.

Stockpile bays will be developed every 120-130m to expedite loading out of the decline face, maximise utilisation of the drill jumbo and achieve a high rate of decline advance.

#### 21.1.4.4.3 Layout of SubLevels

A 30m sublevel interval was chosen for Didipio based on the geotechnical recommendations for stope heights. This sublevel interval will accommodate both 30m and 60m high stopes. An indicative sublevel development layout is shown in the following figure. The access between the decline and the sublevel provides a loop for haul trucks to turn around without the need for reversing, enhancing productivity and safety. See Figure 21-7.

**Figure 21-7: Typical SLOS development layout**




### 21.1.4.4.4 Ground Support

All development is planned with ground support consisting of rockbolting, meshing and shotcreting where required. All ground support for decline and lateral development is subdivided into two groups, for costing purposes.

1. Active support – rockbolting, meshing and shotcreting where required.
2. Passive support – rockbolting, shotcrete, concrete and/or steel sets where required.

Within the deposit, the ground support measures would include cable-bolt support for stope walls and backs installed from the sublevel development headings. Requirements will vary from stope to stope, as determined by specific geotechnical assessment based on ground conditions.

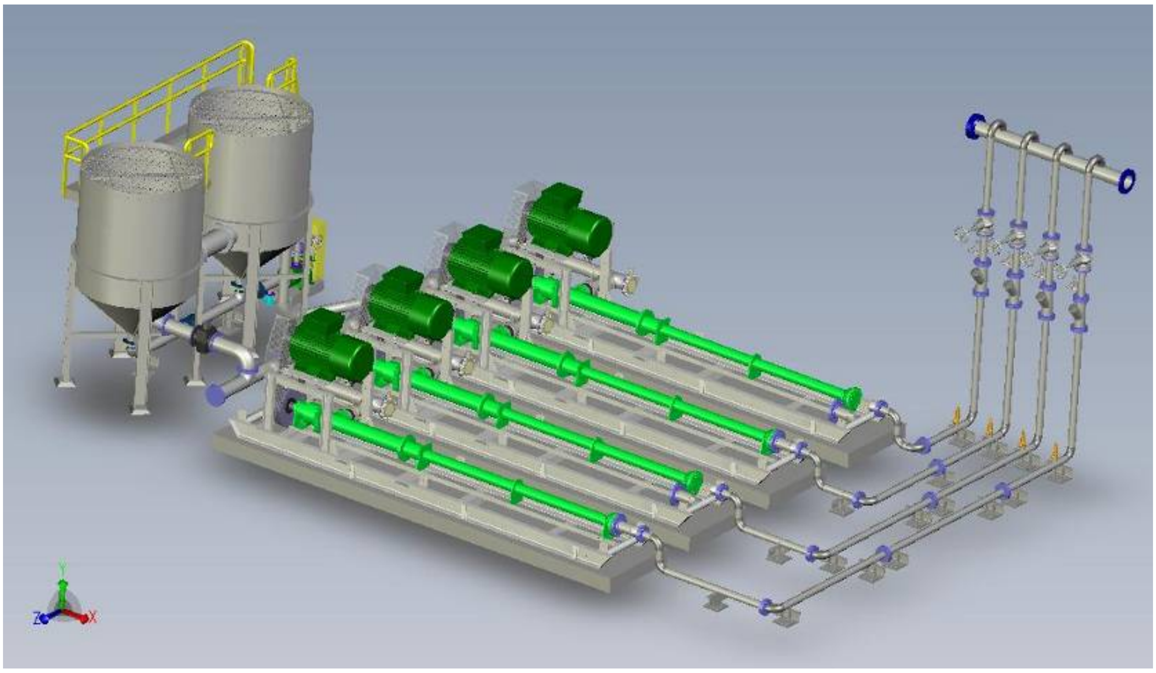
### 21.1.4.4.5 Mine Dewatering

Although the SLOS method will result in much lower water inflows than caving methods, inflows from ground water and mining activities may still be significant.

A pumping system (see Figure 21-8 for an example system) will be established to remove this water from the underground mine. The pumping will be handled by two main pumping stations. Each station will comprise four pumps of approximately 27l/sec capacity over a maximum head of 220m. The units proposed are mono pumps using the screw drive impellors that can handle dirty water pumping applications.

The main pump station will be at the base of the Top Panel. The second station will be mobile and relocated with the advancing decline face in the Bottom Panel.

**Figure 21-8: Pumping station**



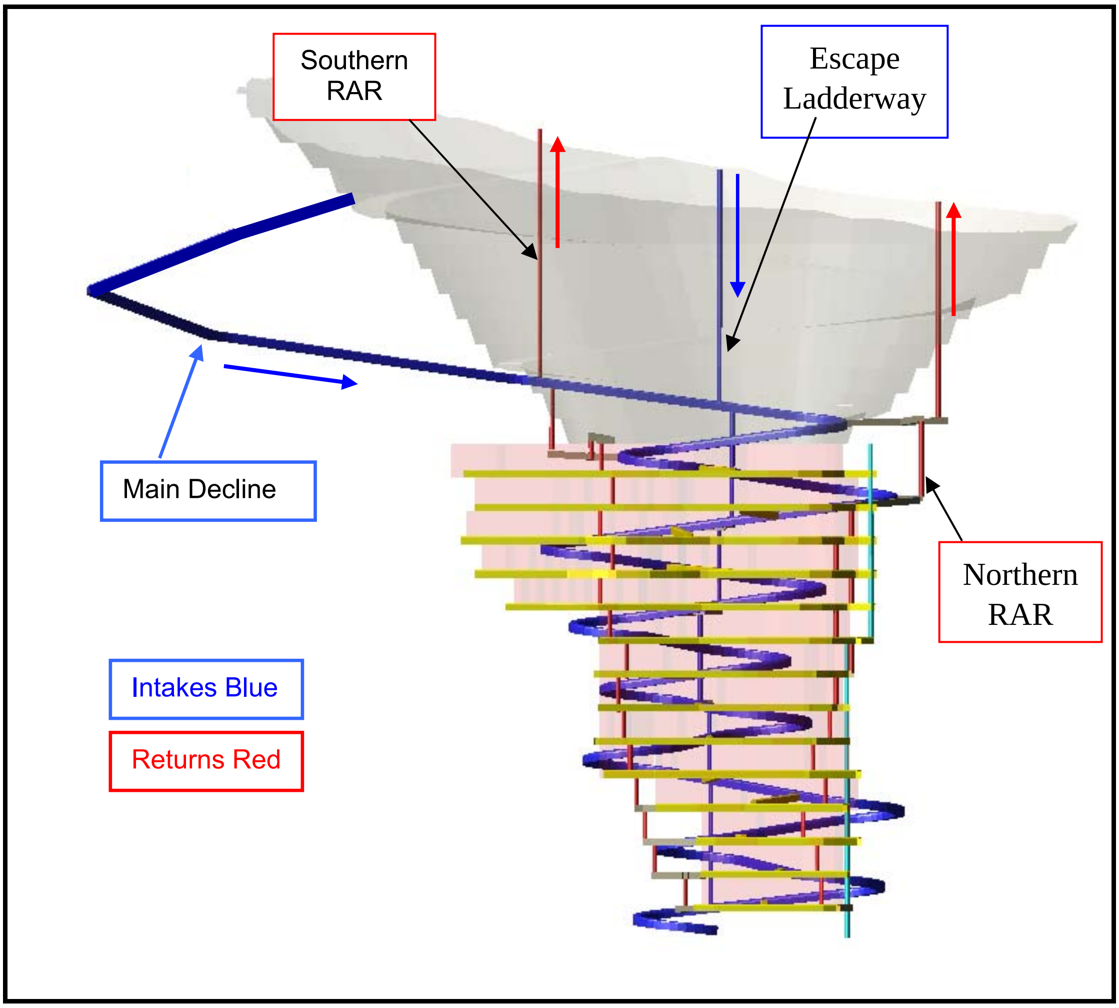
### 21.1.4.4.6 Ventilation

Airflow is required for dilution of diesel exhaust emissions. Minimum air quantities for dilution of diesel exhaust emissions from the proposed diesel equipment fleet assume a ventilation dilution factor of $0.05m^3/s$ per kW of installed diesel power.

A simple ventilation system is proposed for the underground, whereby fresh air will enter the mine via the main decline and escape ladder way and return air via the north and south Return Air Ways (RAR). Figure 21-9 shows the ventilation network concept. All ventilation rises have been designed at 3.5m x 3.5m and are assumed to be mined using an Alimak miner. This system would have a capacity of approximately 300m to $350m^3/s$.

Intake air will be distributed to each level from the decline. This air will be directed to the working areas with auxiliary fans, and then contaminated air will be exhausted to the surface primary fans via the north and south RAR. Each RAR will be equipped with a 550kW primary fan at the surface.

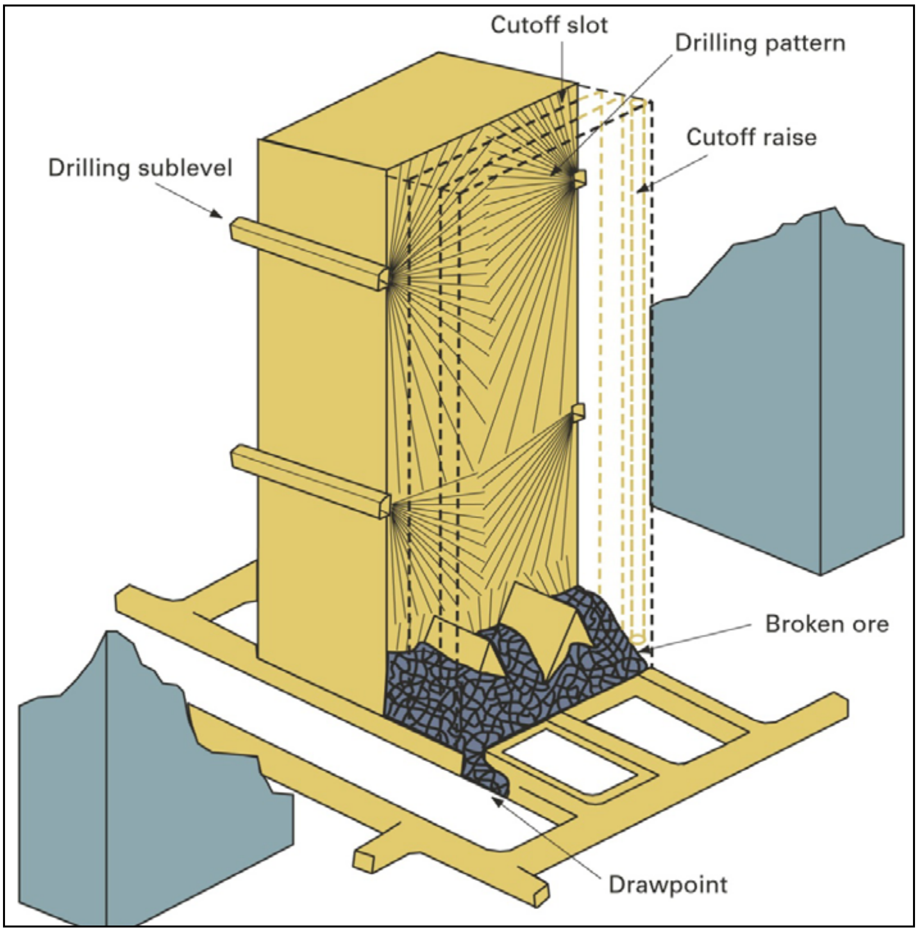
**Figure 21-9: Ventilation network concept**



#### 21.1.4.5 Mining Operations

The general concept of sublevel open stoping is shown schematically in Figure 21-10.

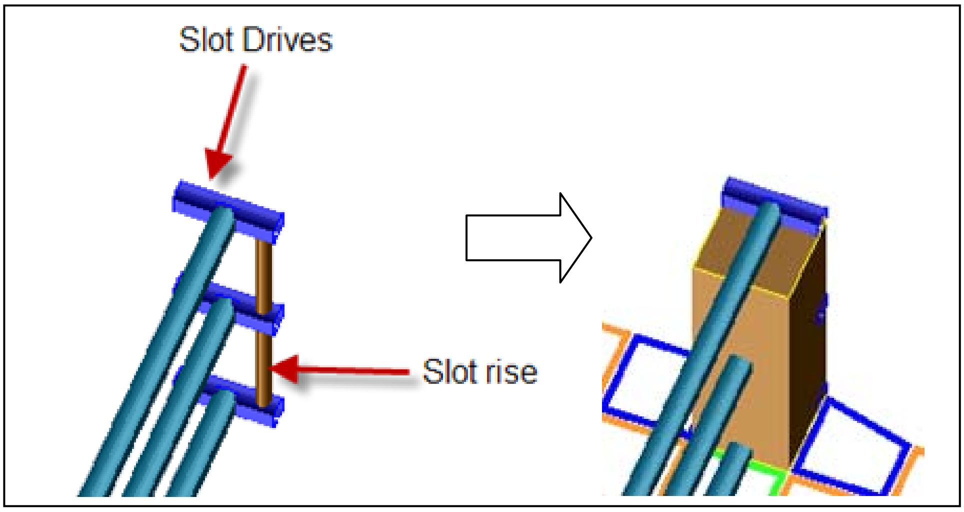
**Figure 21-10: Schematic sublevel open stoping method (from Atlas Copco)**




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

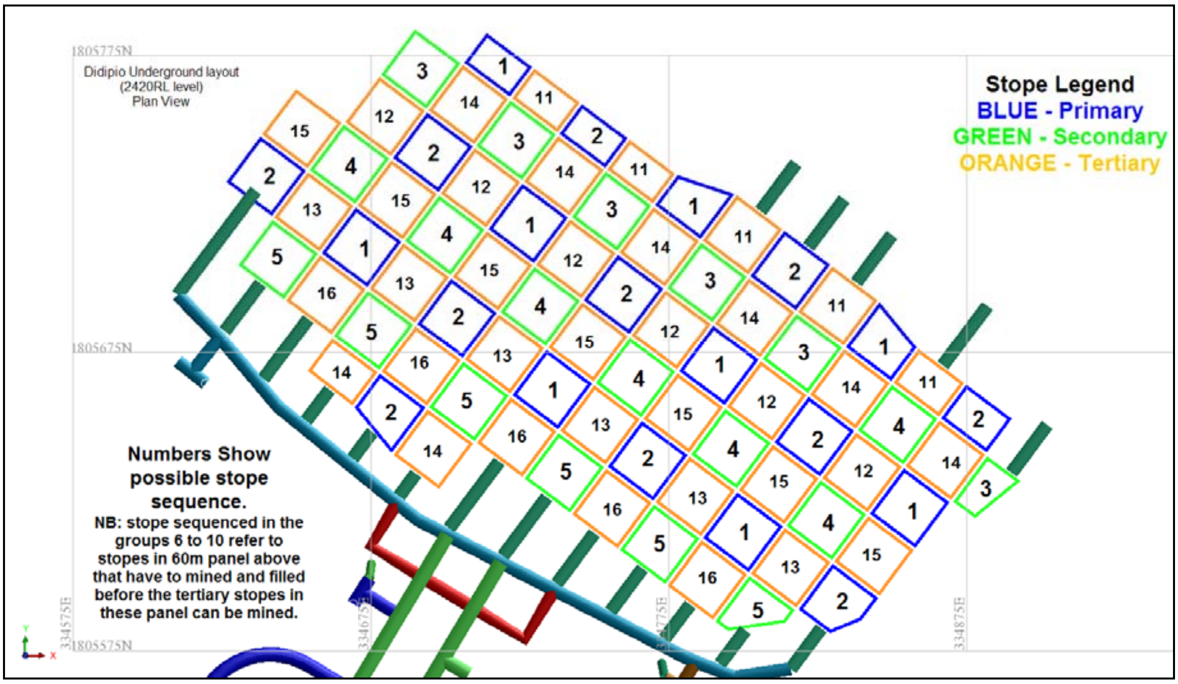
- The first step in extraction of a stope would be the progressive development of a vertical expansion slot. This creates a void over the full height of the stope that will provide expansion volume for larger production blasts. Once the expansion slot is in place, the stope would be opened up by drilling 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 21-11.

**Figure 21-11: Schematic showing development for stope expansion**




- 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.
- 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 21-12.

**Figure 21-12: Schematic showing stope sequencing**




### 21.1.4.6 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.

#### 21.1.4.6.1 Ore Recovery and Dilution

Different dilution and mining recovery factors have been applied depending on the sequence-type of the stope, i.e. primary stope, secondary stope or tertiary stope.

#### 21.1.4.6.2 Production Drilling and Blasting

All draw points and slot drives are designed at 5.0m wide x 4.5m high to accommodate both production drilling and mucking activities. The production drill required for the slot and main rings will drill up holes, using either 89mm or 102mm bits. Rates and costs for production drilling and blasting are based on the Tamrock Solo rigs used elsewhere in the Philippines.

#### 21.1.4.6.3 Production Mucking

Production mucking rates and costs are based on the 20-tonne capacity loader. Production mucking and haulage estimates assume two 12-hour shifts per day working on a continuous roster. On this basis three loaders will be required to mine 1.2 Mtpa.

#### 21.1.4.6.4 Ore Haulage

Articulated 50-tonne trucks were assumed for haulage up the decline. On each loading level a loading loop has been designed to facilitate one-way travel by loaded and unloaded trucks with minimised interference of each other.

#### 21.1.4.7 Equipment Fleet and Manning

The underground has been designed and estimated on the basis of contract mining for the first 44 months to establish the mine and commence SLOS operations followed by owner-operation thereafter. Fleet and manning numbers take into account operating conditions at other Filipino underground operations, but some greater efficiencies have been applied to avoid the extremely large numbers of personnel commonly employed in Filipino underground mines.

### 21.1.5 Mine Closure

When open cut mining finishes in Year 7 the open cut mine will be maintained in a safe condition to enable continued use of the open cut ramp above the underground entry at RL2640 and to act as a sump to collect rainwater for removal by pumping before it enters the underground mine below. The final faces of the waste rock dump will be rehabilitated in accordance with commitments made in the EPEP.

Upon completion of underground mining in Year 20 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.

### 21.1.6 Production Schedule

#### 21.1.6.1 Mine Development Sequence

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.
- After two months of pre-production mining a level bench will be established at 2790RL. Open cut mining then ceases for one month to allow grade control drilling of the top ore benches and the oxide gold zone.
- The open cut mine operates for 71 months. The first nine months are pre-production.
- The final wall of the open cut reaches the underground portal position at 2640RL in the fourth quarter of Year 4. This marks the start point of underground mine development.
- After six months of decline development the first orebody sublevel begins and the first development ore is hauled to the ROM pad in Quarter 2 of Year 5. The open cut is still in full production at this time.
- First underground production ore is mined from the stopes in Quarter 4 of Year 6, five months after the end of open cut mining.
- The underground mine continues to operate until the end of Quarter 2, Year 20.
- The mill processes ore at 2.5 Mtpa with predominantly ore-grade feed until the start of Quarter 3, Year 6. It continues at 2.5 Mtpa for a further eight months on a combination of low-grade ore off stockpile and underground development and production ore. From Quarter 2, Year 7 the feed rate drops back to 1.2 Mtpa using underground ore only.

#### 21.1.6.2 Production Schedule

The production schedule is summarised in Table 21-4.

### 21.1.7 Mining Cost Estimate

#### 21.1.7.1 Open Cut Mine

Open cut capital and operating costs are based on quotes obtained from Filipino mining and civil contractors in late 2009. These quotations were supplied on an all-inclusive basis covering:

- Mobilisation;
- Site establishment;
- Supply, operation and maintenance of all mining equipment;
- Recruitment, training, accommodation and messing for all employees;
- Holding of all necessary permits; and
- Supply of all diesel, explosives and other consumables.

#### 21.1.7.2 Underground Mine

Underground mine capital and operating costs were initially estimated from first principles by AMDAD using vendor quotes for equipment and explosives and rates supplied by OGC for labour and diesel. In order to make the underground cost model more flexible in examining different schedule scenarios, the costs per metre of development for a range of profiles were taken from the original model and used as cost drivers for subsequent mine plans.

During 2010, the costs per metre of advance were benchmarked against several Asian and African projects and adjusted to conform. Costs for capital items such as workshops and communications are still based on AMDAD's original estimates with some escalation to 2010 values.

## Table 21-4: Annual production schedule by AMDAD

| YEAR | | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **MINING** | | | | | | | | | | | | | | | | | | | | | | | |
| **Tonnes** | | | | | | | | | | | | | | | | | | | | | | | |
| OC ore | kt | 0 | 1,521 | 2,527 | 2,464 | 2,118 | 2,449 | 1,140 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 12,220 |
| OC low grade | kt | 0 | 171 | 409 | 360 | 328 | 363 | 24 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1,655 |
| Underground | kt | 0 | 0 | 0 | 0 | 0 | 40 | 285 | 1,134 | 1,230 | 1,193 | 1,169 | 1,213 | 1,216 | 1,198 | 1,196 | 1,176 | 1,212 | 1,200 | 1,151 | 971 | 133 | 15,719 |
| Total | kt | 0 | 1,692 | 2,936 | 2,824 | 2,447 | 2,852 | 1,449 | 1,134 | 1,230 | 1,193 | 1,169 | 1,213 | 1,216 | 1,198 | 1,196 | 1,176 | 1,212 | 1,200 | 1,151 | 971 | 133 | 29,594 |
| **Au g/t** | | | | | | | | | | | | | | | | | | | | | | | |
| OC ore | g/t | 0.00 | 0.46 | 0.75 | 0.64 | 0.95 | 0.99 | 2.02 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.89 |
| OC low grade | g/t | 0.00 | 0.23 | 0.26 | 0.25 | 0.25 | 0.33 | 0.34 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.27 |
| Underground | g/t | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 1.36 | 2.48 | 2.57 | 2.06 | 1.75 | 1.79 | 1.75 | 1.84 | 2.19 | 2.48 | 1.91 | 2.24 | 2.40 | 1.62 | 2.32 | 1.56 | 2.07 |
| Total | g/t | 0.00 | 0.44 | 0.68 | 0.59 | 0.86 | 0.91 | 2.08 | 2.57 | 2.06 | 1.75 | 1.79 | 1.75 | 1.84 | 2.19 | 2.48 | 1.91 | 2.24 | 2.40 | 1.62 | 2.32 | 1.56 | 1.48 |
| **Cu %** | | | | | | | | | | | | | | | | | | | | | | | |
| OC ore | | 0.00 | 0.64 | 0.73 | 0.63 | 0.72 | 0.65 | 0.76 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.68 |
| OC low grade | | 0.00 | 0.27 | 0.26 | 0.26 | 0.26 | 0.22 | 0.23 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.25 |
| Underground | | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.68 | 0.59 | 0.58 | 0.53 | 0.51 | 0.47 | 0.48 | 0.49 | 0.51 | 0.52 | 0.45 | 0.52 | 0.54 | 0.50 | 0.56 | 0.51 | 0.51 |
| Total | | 0.00 | 0.60 | 0.66 | 0.59 | 0.66 | 0.60 | 0.72 | 0.58 | 0.53 | 0.51 | 0.47 | 0.48 | 0.49 | 0.51 | 0.52 | 0.45 | 0.52 | 0.54 | 0.50 | 0.56 | 0.51 | 0.00 |
| **PROCESSING** | | | | | | | | | | | | | | | | | | | | | | | |
| **Tonnes** | | | | | | | | | | | | | | | | | | | | | | | |
| OC ore | kt | 0 | 1,333 | 2,500 | 2,500 | 2,298 | 2,363 | 1,227 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 12,220 |
| OC low grade | kt | 0 | 0 | 0 | 0 | 202 | 98 | 988 | 367 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1,655 |
| Underground | kt | 0 | 0 | 0 | 0 | 0 | 40 | 285 | 1,134 | 1,230 | 1,193 | 1,169 | 1,213 | 1,216 | 1,198 | 1,196 | 1,176 | 1,212 | 1,200 | 1,151 | 971 | 133 | 15,719 |
| Total | kt | 0 | 1,333 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 1,501 | 1,230 | 1,193 | 1,169 | 1,213 | 1,216 | 1,198 | 1,196 | 1,176 | 1,212 | 1,200 | 1,151 | 971 | 133 | 29,594 |
| **Au g/t** | | | | | | | | | | | | | | | | | | | | | | | |
| OC ore | g/t | 0.00 | 0.45 | 0.73 | 0.65 | 0.93 | 0.98 | 1.97 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.89 |
| OC low grade | g/t | 0.00 | 0.00 | 0.00 | 0.00 | 0.25 | 0.25 | 0.27 | 0.27 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.27 |
| Underground | g/t | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 1.36 | 2.48 | 2.57 | 2.06 | 1.75 | 1.79 | 1.75 | 1.84 | 2.19 | 2.48 | 1.91 | 2.24 | 2.40 | 1.62 | 2.32 | 1.56 | 2.07 |
| Total | g/t | 0.00 | 0.45 | 0.73 | 0.65 | 0.87 | 0.96 | 1.36 | 2.01 | 2.06 | 1.75 | 1.79 | 1.75 | 1.84 | 2.19 | 2.48 | 1.91 | 2.24 | 2.40 | 1.62 | 2.32 | 1.56 | 1.48 |
| **Cu %** | | | | | | | | | | | | | | | | | | | | | | | |
| OC ore | | 0.00% | 0.63% | 0.72 | 0.64 | 0.71 | 0.65 | 0.76 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.68 |
| OC low grade | | 0.00% | 0.00% | 0.00 | 0.00 | 0.26 | 0.26 | 0.25 | 0.25 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.25 |
| Underground | | 0.00% | 0.00% | 0.00 | 0.00 | 0.00 | 0.68 | 0.59 | 0.58 | 0.53 | 0.51 | 0.47 | 0.48 | 0.49 | 0.51 | 0.52 | 0.45 | 0.52 | 0.54 | 0.50 | 0.56 | 0.51 | 0.51 |
| Total | | 0.00% | 0.63% | 0.72 | 0.64 | 0.68 | 0.64 | 0.54 | 0.50 | 0.53 | 0.51 | 0.47 | 0.48 | 0.49 | 0.51 | 0.52 | 0.45 | 0.52 | 0.54 | 0.50 | 0.56 | 0.51 | 0.57 |
| **PRODUCT** | | | | | | | | | | | | | | | | | | | | | | | |
| Concentrate | | | | | | | | | | | | | | | | | | | | | | | |
| Dry tonnes | kt | 0 | 25 | 60 | 54 | 57 | 54 | 46 | 26 | 22 | 21 | 19 | 20 | 21 | 21 | 21 | 18 | 22 | 22 | 20 | 19 | 2 | 572 |
| Cu % | % | 0.00 | 27.91 | 28.27 | 27.96 | 28.10 | 27.92 | 27.44 | 27.20 | 27.38 | 27.29 | 27.02 | 27.06 | 27.18 | 27.29 | 27.33 | 26.85 | 27.33 | 27.48 | 27.23 | 27.58 | 27.25 | 27.40 |
| Au g/t | g/t | 0.00 | 13.58 | 18.34 | 18.41 | 22.83 | 26.28 | 41.28 | 59.99 | 58.19 | 52.17 | 57.27 | 55.46 | 56.24 | 62.89 | 68.83 | 63.20 | 63.49 | 64.25 | 49.72 | 60.51 | 49.14 | 40.59 |
| Gold in dore | koz | 0 | 4 | 16 | 13 | 22 | 25 | 42 | 41 | 34 | 27 | 27 | 27 | 29 | 36 | 42 | 30 | 38 | 41 | 23 | 32 | 3 | 551 |
| Gold in concentrate | koz | 0 | 9 | 35 | 32 | 42 | 45 | 59 | 50 | 42 | 35 | 35 | 36 | 38 | 43 | 47 | 37 | 44 | 46 | 32 | 36 | 4 | 747 |
| Total gold | koz | 0 | 14 | 51 | 46 | 63 | 70 | 100 | 91 | 76 | 62 | 62 | 63 | 67 | 79 | 90 | 67 | 82 | 87 | 56 | 68 | 6 | 1,298 |
| Silver in dore | koz | 0 | 0 | 1 | 1 | 1 | 2 | 3 | 3 | 2 | 2 | 2 | 2 | 2 | 2 | 3 | 2 | 2 | 3 | 1 | 2 | 0 | 34 |
| Copper in concentrate | kt | 0 | 6 | 17 | 15 | 16 | 15 | 13 | 7 | 6 | 6 | 5 | 5 | 6 | 6 | 6 | 5 | 6 | 6 | 5 | 5 | 1 | 157 |
| **OPENCUT MINING** | | | | | | | | | | | | | | | | | | | | | | | |
| Ore volume mined | kbcm | 0 | 605 | 989 | 963 | 834 | 961 | 453 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 4,805 |
| Low grade volume mined | kbcm | 0 | 67 | 154 | 135 | 126 | 136 | 9 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 626 |
| Waste volume mined | kbcm | 875 | 1,862 | 1,020 | 1,701 | 2,646 | 1,105 | 15 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 9,225 |
| Total volume mined | kbcm | 875 | 2,534 | 2,163 | 2,799 | 3,606 | 2,203 | 477 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 14,656 |
| Waste:Ore ratio volume | | 0.00 | 2.77 | 0.89 | 1.55 | 2.75 | 1.01 | 0.03 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 1.70 |

*Note: [1] The schedule added into OGC's financial model differs slightly from AMDAD's due to changes in the ramp-up profile.
[2] The timeline in this table is for illustrative purpose only, as a decision to proceed with project construction still requires completion of project execution plan, regulatory approval and OGC board approval.

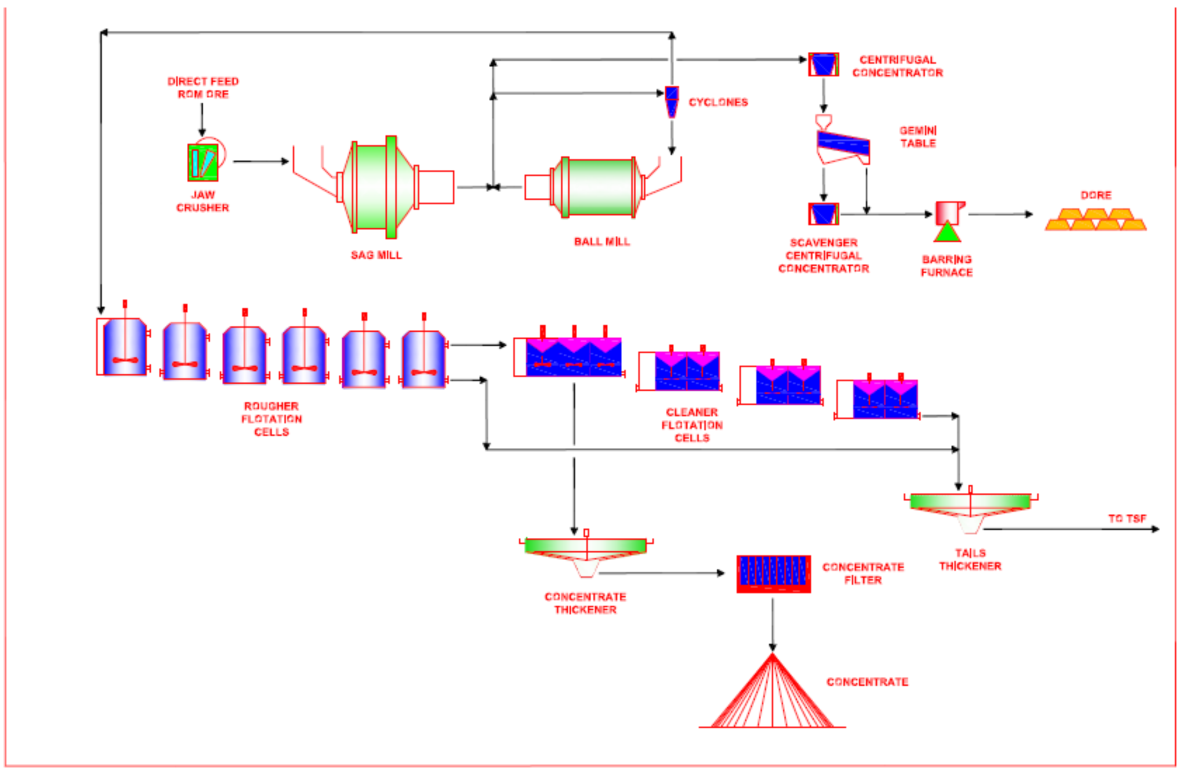
## 21.2 Recoverability

A full suite of metallurgical test work can be found in section 16.

### 21.2.1 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 21-13 presents the latest process flow designed by Arccon.

**Figure 21-13: 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 900mm run of mine (ROM) material to about 125mm. 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 majority of the coarse underflow will return to a closed circuit ball mill, with the ball mill discharge similarly pumped to the hydrocyclones for classification.

The concentrate will then be directed to a Falcon-type gravity concentrator, while the tailings return to the ball mill. The gravity concentrate will be transported to the gold room, where it will be further concentrated using a shaking table. 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 a trash screen and the cleaned pulp will then be conditioned with reagents prior to rougher flotation followed by scavenger flotation. Concentrate from the rougher cells will be retreated in a cleaner and recleaner set of cells to produce final concentrate. Tailings from the scavenger 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.

#### 21.2.1.1 Primary Crushing

The crushing circuit will be situated next to the ROM pad. Mining trucks will haul ore from the open pit and underground mines to the ROM pad. ROM ore will be fed by either 55-tonne dump truck or FEL through a 500mm 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 a brute force vibrating grizzly feeder and will be discharged on to a static grizzly into a single toggle crusher. Fines will bypass the crusher. Vibrating 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 125mm. A monorail 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 same FEL operation allows fine ROM to be separated on the ROM pad and serve as emergency feed on those rare occasions when the primary crusher is down and the mill is not. Crushed ore is conveyed to the SAG mill via a transfer bin.

#### 21.2.1.2 Grinding

The 7.3m diameter by 3.5m effective grinding length (EGL) grate discharge SAG mill will be fitted with steel liners and pulp discharges and will process 2.5 Mt/y. The SAG mill will be equipped with a 4300 kW wound rotor induction motor and liquid resistance starter including heat exchanger and capability to provide speed variation. 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 8.8m EGL ball mill will be supplied with rubber liners, 4300 kW wound rotor induction motor, liquid resistance starter, 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 the majority reporting to ball mill feed. The balance will be directed to a gravity circuit and the hydrocyclone overflow will gravitate on to a trash screen. The tailings from the gravity circuit will return to the milling circuit.

A maintenance crane will be provided to facilitate hydrocyclone replacement and maintenance on the cyclone cluster.

Trash from the screen will be directed to a trash bin at the ground level. Trash screen underflow will gravitate to the flotation circuit.

#### 21.2.1.3 Gravity Circuit

The gravity circuit will be installed to recover coarse gold. The gravity circuit comprised a 3mm aperture vibrating scalping screen surmounting a Falcon model SB2500 concentrator. Oversize from the scalping screen will report to the mill feed and the concentrator tails to the mill discharge hopper. A dedicated pump will supply the concentrator with fluidising water.

The gravity circuit components, comprising the main concentrator, a surge bin for the concentrate, a Gemini table treating all the concentrate and a further Falcon model SB250 concentrator on the table tails, will be located in the secured area gold room.

The concentrate from this unit will gravitate to the gold room for further processing. Gravity process plant tails will return to the milling circuit.

#### 21.2.1.4 Flotation Circuit

Cyclone overflow will be screened to remove trash material such as plastic and wood fibre. The screened pulp will be conditioned with reagents in the rougher/scavenger feed box. The overflow from the conditioner feeds the first of six rougher/scavenger flotation cells. Tank cells of $40m^3$ will be used for the roughers and scavengers. 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. Scavenger concentrates will be pumped to a scavenger cleaner bank with the scavenger tailings sent as final tailings to the tailings thickener.

Concentrate from the cleaner cells feeds the bank of recleaner cells. Tailings from the recleaner cells join the rougher concentrate as feed to the cleaner cells. Concentrate from the recleaner cells will be directed to the concentrate thickener. The concentrate from the scavenger cleaner cells will join the feed to the cleaner cells. Tailings from the cleaner cells join the feed to the scavenger cleaner cells.

#### 21.2.1.5 Concentrate Handling

Final copper concentrate will be thickened in a 12m diameter conventional thickener. The underflow will be pumped at about 50-60% solids to a storage tank. A 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 13 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.

#### 21.2.1.6 Tailings Handling

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.

A variable speed thickener underflow pump will pump thickened tails to the Tailing Storage Facility (TSF). TSF decant will be reclaimed and pumped to the process water tank.

#### 21.2.1.7 Gravity Gold Concentrate Treatment

The concentrates from the Falcon concentrator will be treated using a Gemini model 1000 shaking table. Concentrates from the table will be filtered and dried prior to smelting in a standard diesel-fired barring furnace. The gold/copper concentrates will be treated using acid digestion to remove some of the problematic copper sulphide minerals. 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.

#### 21.2.1.8 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 10-tonne silo, equivalent to about five days' requirements. 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, SEX and S701. The SEX will be delivered in pellet form in 120kg solid drums, mixed on site and stored in a tank with a one-day capacity. A ring main system will distribute the reagent to the addition points. The S701 will be delivered in 200L drums and stored in a head tank with five days' capacity. The reagent will be distributed to the various points in the flotation circuit using dedicated dosing pumps.

The frother comes in 200L drums and will be pumped to a five-day 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 two-day storage tank. Flocculant distribution will be by a variable speed pump.

#### 21.2.1.9 Water

Raw water will be supplied from the mine dewatering bores and tailings dam water and pumped to a raw 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 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 tanks, 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.

#### 21.2.1.10 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.

#### 21.2.1.11 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. An operator control station will be installed in the main plant control room. The compact plant will obviate 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.
- 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.

#### 21.2.1.12 Plant Buildings and Layout

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

#### 21.2.1.13 Recent Metallurgical Testwork

Table 21-5 provides confirmation of metallurgical performance and a high level of confidence that the plant will operate as designed.

**Table 21-5: 2006 confirmatory testwork results**

| Sample | Head grade | | Concentrate | | | Recovery % | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | Grade | | | Gold | | |
| | % Cu | g/t Au | Type | % Cu | g/t Au | Copper | 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 |

## 21.3 Contracts

### 21.3.1 Contracting Strategy

The project has been in care and maintenance since 2008. The project team is currently developing a project execution plan.

### 21.3.2 Health, Safety and Environmental Management

Health, safety and environmental management controls that are planned, including site security, are formalised into detailed policies and procedures for implementation by OGPI.

### 21.3.3 Initial Implementation Activities

A provisional development timeline was prepared as part of the 2010 optimisation study and used to establish the production schedules and cash flow analyses. A decision to proceed with mine construction still requires completion of the project execution plan and regulatory and board approval by OGC.

The current execution strategy for the project is to be developed in two phases: a pre-works phase (six months) followed by a construction phase (12 months).

The pre-works phase is expected to last six months and will generally include: acquisition of land required for the project; upgrading of project site support facilities including community roads, site fencing, temporary accommodation and warehousing facilities at an area to the north of the existing offices known as Boulevard; commencement of permanent accommodation facilities; dismantling of the existing exploration camp and offices; and commencement of the recruitment and mobilisation of OGC's project management team.

During the pre-works phase it is proposed to continue community relations activities, to call and assess tenders for the open-pit mining contract and to prepare for the mobilisation of the open-pit mining contractor.

The construction phase, subject to review during the project execution plan, is proposed to be managed by an OGC in-house project management team supported by mining, processing, tailings disposal and water management consultants.

BDA considers the proposed execution strategy generally reasonable. However, while the durations for the completion of the design, procurement and construction activities in the schedule appear achievable, for the overall schedule to be met the surface rights access programme targets must be met and a suitable project team must be recruited, mobilised and familiarised with the project early in the pre-works phase.

## 21.4 Environmental and Permitting

BDA has not undertaken any legal due diligence on ownership, tenement or licensing issues. The following is based on information provided by OGC.

### 21.4.1 Tenement Status – Land Ownership

#### 21.4.1.1 Minerals Agreement

On 20 June 1994, the Philippines Government entered into a Financial or Technical Assistance Agreement (FTAA No. 001) with CAMC, granting the latter the exclusive right to undertake exploration, development and utilisation of the mineral resources in an initial Contract Area of 34,992ha (exclusive of the Kasibu Forest Reserve) in the provinces of Nueva Vizcaya and Quirino.

FTAA No. 001 has a term of 25 years from the date of its execution, renewable for like period. On 23 December 1996, CAMC transferred the FTAA, with all its rights and obligations, to APMI. This transfer of the FTAA from CAMC to APMI was approved by the DENR on 9 December 2004.

On 18 March 2005, APMI filed a Partial Declaration of Mining Project Feasibility with the Mines and Geosciences Bureau covering 975ha of the remaining 21,428.76ha Contract Area under FTAA 001. APMI was deemed to have complied with the requirements for approval of this Mining Project Feasibility in an MGB document to APMI dated 11 October 2005.

Subsequently, APMI changed its name to OceanaGold (Philippines), Inc. (OGPI) and since then OGPI has applied to the Environmental Management Bureau to amend the existing ECC to allow for operations based on the current mine plan. These changes are expected to be approved in 2010 prior to the commencement of operations.

A series of Environmental Impact Statements, amendments and supplements were submitted to the DENR between 1996 and 2004, resulting in the issuance of a series of three Environmental Compliance Certificates (ECC). An Environmental Protection and Enhancement Programme was prepared in December 2004, for the development of the 975ha Didipio Project. The EPEP provides an operational link between the EPEP committees, the EIA programme and associated permit compliance programmes.

#### 21.4.1.2 Surface Ownership

The majority of the land in the Didipio Valley is designated as forest reserve with a tract of land in the centre of the valley being defined as alienable and disposable. Claims of tenure to the area vary from freehold title to land tax declaration tenure (unregistered land) to squatters occupying government land.

In accordance with the Philippine Mining Act, OGPI established a program to acquire surface rights to the land required for the project.

#### 21.4.1.3 Ownership Agreements

CAMC has an agreement (known as the “Addendum Agreement”) with a Philippine claim owner syndicate which covers that portion of the FTAA previously included in a block of mineral claims held by the syndicate. This previous syndicate holding included a core of six claims which reached the advanced stage of mining lease application ‘without adverse claims’. These lease applications essentially cover the prime area of interest.

The Addendum Agreement provides the syndicate with a contractual right to an 8% free carried interest in the operating vehicle that will be established to undertake the management, development, mining and processing of ores, and the marketing of products from, the area of the FTAA covered by the Addendum Agreement upon the satisfaction of certain conditions including completion of a bankable feasibility study and a decision to proceed to production.

The free carried interest will entitle the syndicate to a proportionate share of any dividends declared from the net profits of the operating vehicle, but not until all costs of exploration and development have been recovered. As at the date of this report, the operating vehicle has not yet been formed. The Addendum Agreement also provides for payment of a 2% net smelter return to the syndicate.

### 21.4.2 Environment and Statutory Approvals

#### 21.4.2.1 Regulatory Issues, Licences and Programmes

OGC has advised BDA that 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 has been a significant aspect of the project development. 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 21-6 lists the permits which are required to enable commencement of mine operations at the Didipio Project.

**Table 21-6: 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 2008 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 previous mine plan was lodged with EMB in December 2007. The approval was put on hold due to subsequent project development changes. A revised ECC (covering the project design changes) is being drafted for 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 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 and 2010. 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 under current care and maintenance. |
| Mine Rehabilitation Fund and Mine Waste and Tailings Fees Reserve Fund | MGB Region 2 | Agreements signed 15 Oct 2004.<br>Contingent Liability and Rehabilitation Fund (CLRF)<br>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 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.<br>As of December 2009 the amounts deposited are as follows:<br>MTF = PhP 153,474.53<br>ETF = 104,145.66<br>RCF = 5,572,976.43 |

| Permit/Programme | Regulatory agency | Comment |
|---|---|---|
| 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. OGC is preparing a new SDMP. |
| 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 around 75% of the land required for the project. Compensation packages are offered to both squatters and private landowners. Where negotiations fail, legal mechanisms exist to enable the company to obtain surface rights. |
| 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 made on 2007; the permits are yet to be granted. 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.*

#### 21.4.2.1.1 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 more than 75% complete and applications for water rights are in process.

## 21.4.3 Environmental Issues

BDA has reviewed those environmental engineering aspects that are a material part of the project and which may have significant implications for project feasibility, costs and timing. Some social aspects are inevitably incorporated in the review. The issues discussed cover the main environmental risk areas identified from BDA's review of the available documents, including the Environmental Impact Statements, government documents and approvals, design information and budget models.

#### 21.4.3.1.1 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.

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 citris 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.

### 21.4.3.2 Water Management

#### 21.4.3.2.1 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 recoded downstream in the Diduyon River resulting from illegal mining in the catchment.

The project incorporates river water diversion facilities (the Tailings Storage Facility (TSF) diversion system and the TSF decant system) that carry the drainage waters from the Dinauyan to the Didipio River.

#### 21.4.3.2.2 Water Supply

The daily water demand for the project is estimated at 1824m$^3$, of which 1440m$^3$ will be make-up water for the process plant, sourced from the 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 decant discharge 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 or discharged to the Didipio River via a settling pond, and diversion of clean surface water flows around the TSF.

Water will be recycled from the TSF 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, 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.

BDA believes the proposed water management strategy for TSF decant is appropriate to ensuring water quality compliance.

#### 21.4.3.2.3 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 against the TSF dam wall and encapsulated in non-acid forming waste. Final designs for the TSF, waste dump and the low-grade stockpile are yet to be 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.

#### 21.4.3.3 Waste Management

##### 21.4.3.3.1 Tailings Disposal

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

The TSF will be located in the Dinauyan Valley, adjacent to the processing plant. The TSF is designed as a cross-valley impoundment, located approximately 1km west of the proposed plant site. The facility is being designed to store a total of 32Mt 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 to provide storage for up to 50Mt of tailings.

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, which 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 a slightly alkaline liquor, with low levels of Pb, Cu, Zn, and Hg. Decant water is to be discharged from the TSF using a floating pump system, from which process water will be pumped to the plant for reuse in the process cycle.

Tailings waste characterisation studies are ongoing; these results will provide an important input to the design development process.

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 interval) 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.

A surface water diversion system will be constructed to control the amount of surface water discharging into the TSF catchment area (27Mm$^3$ maximum annual water flow). The Dinauyan River diversion dam and diversion system will divert the river before it reaches the tailings area. Any run-off, springs or direct precipitation or flows that exceed the diversion design criteria will accumulate at the north-western edge of the tailings beach and will be used, as required, in the process plant.

Water in excess to this requirement will be pumped into a decant system, which will be discharged to the Didipio 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 21-7: Didipio Project Tailings Storage Facility**

| TSF component | Storage | Method | Construction | Closure |
|---|---|---|---|---|
| TSF | Single cell; water to be reclaimed via floating pontoon pumping system; approx 32Mt tailings capacity at 6850tpd rate (dry tails); designed to store one in 100 year /24hr storm event. Diversion and decant systems designed to manage maximum precipitation event. An ultimate storage capacity of up to 50Mt is possible with the embankment raised (only required if other deposits processed here in the future). | Tailings pumped from the tailings thickener in pipeline approx. 1km from plant; deposited via spigots along embankment (to increase impermeability) and at other sites within the TSF. Floating pump/ pontoon decant system, directing water to the plant, via pipeline. Excess water discharged to Didipio River, via 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 diversion and decant systems for Dinauyan River flow. |
| TSF Pumping | A floating pump pontoon will allow flexibility in managing water level. | Conventional low head pumping. | . | Pumping facility may not be operated and flow directed to reinstated river channel. |
| Surface Water Diversion System | A 15km long diversion pipeline to redirect the $16Mm^3$ annual water flow. | Constructed pump and pipeline around the TSF and discharged into the Didipio River. | Pontoon pumping station and contour pipeline around the tailings dam to divert annual flow. | Pontoon, pump and pipeline to be removed and flow directed to TSF catchment. |
| Diversion Dam | The cofferdam and pipeline to divert water around the TSF. | To divert the water flowing along the Dinauyan River valley above the TSF site into the Didipio catchment. | Single embankment across creek. | |

The TSF wall will be constructed from waste rock material from the open pit. The waste rock dump adjacent to the wall will 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 and final rehabilitation profile during the first four years of mine life.

A spillway draining into the Surong River will be constructed on the north-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 restored Dinauyan River, once the river diversion and decant systems are 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. The surface water diversion system will be removed. Upon closure, if seepage water quality meets the statutory release criteria, it will be allowed to flow to the downstream river system. After decommissioning, the surface of the TSF will be revegetated or cultivated.

Engineering designs have been produced and these will be used for the TSF and Diversion Construction Permit application.

#### 21.4.3.3.2 Waste Rock Dumps

All available waste material will be used in construction of the TSF and other infrastructure. The major part of the TSF wall (downstream face) will be completed to its full height and rehabilitated by the end of the first year of operations. The buttressing of the TSF wall will occur until the end of open pit mining. No additional waste rock dumps are planned as all future mining activities will be underground. Waste generated for underground mining will be crushed and be available for road maintenance.

#### 21.4.3.3.3 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 VERY HIGH hazard classification has been assigned to the TSF. Based on the VERY 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.
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.

### 21.4.3.4 Open Pit Void

The open pit will be developed to a level 150m 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 pit rainwater dewatering system will be designed to pump out rainfall water collected in a 1:50 year storm event in 17 days. 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.

By the commencement of underground mining, the waste rock dump and the TSF wall will have reached full height but the TSF will still retain the capacity to accept underground ore tailings. The main mine closure phase will occur at the completion of underground mining and ore processing scheduled for Year 20.

The decommissioning phase will make provision for the surface and groundwater flows to enter and be retained in the pit and underground workings, eventually flooding the pit to the level of the water table which should be at, or close to, 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 area. 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. Any surface drainage from the completed pit will have to ensure excess water discharge meets the water quality discharge criteria.

However, this will be assisted both by the extent of dilution that is likely to occur and the basic nature of the geology, as well as the likelihood that the pit wall will be cleared of any ore material as well as submerged below the water table. BDA notes that 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 affects on downstream water quality and riverine users.

#### 21.4.3.5 Rehabilitation/Revegetation

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

#### 21.4.3.6 Fuel and Chemicals

Diesel storage tanks will be located adjacent to the mine equipment workshop 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 disposed of to a suitable repository, possibly in Manila.

### 21.4.4 Environmental Management

#### 21.4.4.1 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 may need to 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 management work plan (AEPEP), based on the approved EPEP strategy, is a statutory requirement.

The AEPEP makes provision for monitoring of meteorological data, noise levels, 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.

#### 21.4.4.2 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.

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 need to ensure that these structures are geotechnically and geochemically stable landforms. At this early stage, costs associated with this scenario are estimated at US$20M.

#### 21.4.4.3 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 and, based on the information provided by the supplier, is not expected to have an adverse impact.

#### 21.4.4.4 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 proposed 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, neither the actual site access route nor the final transport route (dependent on which port is finally selected) have been determined by OGPI. Therefore, the extent of the impact on affected settlements cannot be assessed. However, it is planned to ship up to 75,000tpa 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, remains an ongoing risk.

#### 21.4.4.5 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 presence of some endangered bird species.

#### 21.4.4.6 Equator Principles

OGPI has committed to designing and operating the Didipio Gold-Copper Project in a manner that is consistent with all relevant World Bank Group (WBG) policies and guidelines, in addition to adhering to Philippine legislative requirements. Accordingly, the EIS Amendment lodged with the DENR that resulted in an amended ECC (issued January 2000) was, in part, seeking to satisfy the requirements of the WBG policies and guidelines.

Today, satisfaction of these requirements is equated with meeting the requirements of the Equator Principles, which are a voluntary framework for the assessment and management of environmental and social issues associated with project financing. The framework provides a means for financial institutions to ensure that projects they finance are developed in a manner that is socially responsible and consistent with sound environmental management practices.

BDA is of the opinion that the Didipio EIA and supporting documents have adequately addressed the environmental criteria required by the Equator Principles. It should be noted that BDA has not been requested to review the social component of the project as this is being addressed in detail by others.

Having regard to the fact that the project is still at the planning stage, it is not possible to comment on compliance with the emission aspects of the project, except to note OGPI's stated commitment to designing and operating the Didipio Project in a manner consistent with relevant WBG policies and guidelines. With this conditionality, BDA is of the opinion that the collective EIA documents, together with the associated environmental management and monitoring plans included in the EPEP, meet the relevant minimum requirements set out in the numerous WBG policies and guidelines.

#### 21.4.4.7 Conclusions

BDA considers that the proposed environmental management and monitoring programmes are generally well planned. Based on the mitigation measures proposed to reduce environmental effects, BDA concludes 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 of the TSF and 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 Port Irene 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 (which appear to be being addressed appropriately), 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 Gold-Copper Project to 2.5 Mtpa.

The Environmental Management Bureau (EMB) of the DENR could have required an amendment to the environmental approval for the project due to this increase in throughput and some associated changes in the Tailings Storage Facility. However, discussions in late April 2007 with the EMB have indicated that as long as the Mining and Geosciences Bureau (MGB) of the DENR supports the increase in throughput and the relatively inconsequential changes in the Tailings Storage Facility, then no amendment to the ECC would be required.

However, as a number of other changes to the ECC are proposed an application to amend this clause of the ECC has been included in the application to amend the ECC lodged in 2007. A revised ECC covering the recent project design changes is being drafted for deliberations and endorsement by the review committee prior to approval by the DENR Secretary.

A further uncertainty for the project is the need for resolution of 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 highlights a prevailing, unresolved local governance and project beneficiary situation in the immediate vicinity of the project.

Finally, fallout from the Rapu Rapu project delays, NGO activity, caution by DENR and MGB management, and a reinforced parochial attitude are key issues considered critical to the project requiring affirmative action to address.

## 21.5 Taxes

The current corporate income tax rate in the Philippines is 30%.

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. The total allowance of VAT in the initial capital cost is US$10.8M.

The Philippines imposes an excise tax on mineral products. 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 and sea freight.

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, after which period only 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.

# 21.6 Capital and Operating Costs

## 21.6.1 Capital Costs

### 21.6.1.1 Capital Estimate of September 2010

The capital cost estimate totals US$140.1M, with a cost base of third quarter 2010. This estimate is summarised in Table 21-8.

**Table 21-8: Capital cost estimate summary, September 2010**

| Item | US$M |
|---|---|
| Mining | 9.0 |
| Process plant | 24.5 |
| Tailings Storage Facility (TSF) | 5.0 |
| Infrastructure | 28.7 |
| Owner's costs | 3.9 |
| Indirect costs | 38.4 |
| *Subtotal* | *109.6* |
| Contingency | 19.8 |
| VAT | 10.8 |
| **Total** | **140.1** |

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

- Open pit mining costs as per contract rates tendered by Filipino contractors;
- Bulk quantities for the TSF were provided by Knight Piésold and costs tendered by local contractors and estimated by OGC;
- Estimates for the process plant, port, power plant and associated infrastructure have been prepared by Arccon. In estimating these costs, Arccon has used rates for labour, equipment and locally supplied materials based on recent experience in the Philippines;
- Freight is based on information provided by DHL in 2008;
- The estimate does not include any allowance for escalation during the construction period. However, the construction period is relatively short;
- Owner's costs include the costs of first fills, spares, and supporting mobile equipment; and
- Indirect costs include the costs of engineering, project management, administrative services, land acquisition, insurance and freight.

### 21.6.1.2 Mine Development Costs

Open cut capital and operating costs are based on quotes obtained from Filipino mining and civil contractors in late 2009. These quotations were supplied on an all-inclusive basis covering:

- mobilisation;
- site establishment;
- supply, operation and maintenance of all mining equipment;

- recruitment, training, accommodation and messing for all employees;
- holding of all necessary permits; and
- supply of all diesel, explosives and other consumables.

Underground mine capital and operating costs were initially estimated from first principles by AMDAD using vendor quotes for equipment and explosives and rates supplied by OGC for labour and diesel. In order to make the underground cost model more flexible in examining different schedule scenarios, the costs per metre of development for a range of profiles were taken from the original model and used as cost drivers for subsequent mine plans.

During 2010, the costs per metre of advance were benchmarked against several Asian and African projects and adjusted to conform to these. Costs for capital items such as workshops and communications are still based on AMDAD's original estimates with some escalation to 2010 values.

#### 21.6.1.3 Working Capital

In additional to the initial capital of US$140.1M, there is a total allowance in the financial model of US$4.0M for working capital in the first two years of operation.

#### 21.6.1.4 Deferred Capital and Sustaining Capital

Ongoing deferred and sustaining capital expenditure is estimated at US$76M, relating principally to ongoing capital expenditure for the underground mine and the TSF. A further allowance is made in the financial model of US$20M for closure costs at the end of life of the mine.

The deferred and sustaining capital cost figure includes:

- US$55.0M spread over operating years three to five for underground development;
- US$4.8M for completion of the Tailings Storage Facility;
- US$8.6M for further sustaining capital; and
- US$5.9 M for further VAT.

#### 21.6.1.5 Contingency

The capital estimate includes contingency allowances of US$19.8M, inclusive of nearly US$3.0M for escalation of original estimates from December 2009 to September 2010.

#### 21.6.1.6 Accuracy

BDA considers that the capital estimate qualifies as a feasibility-standard estimate with a probable accuracy of ±15%.

#### 21.6.1.7 Adjusted Estimate

BDA considers that no further adjustments are required to the updated capital cost estimate.

In BDA's experience, projects of this type may incur cost overruns above the allocated contingency. BDA recommends that capital costs be monitored closely and estimates revised as necessary.

#### 21.6.1.8 Conclusions

Early in 2010, BDA reviewed all capital costs detailed in the preliminary optimisation study of December 2009. AMDAD has subsequently updated the mine schedules (detailed in section 2.1), mining capital and operating costs. John Wyche of AMDAD is the qualified person responsible for these updates. BDA accepts AMDAD's revisions and considers its previous conclusions to hold. These are:

- BDA considers the estimation accuracy of the estimate to be ±15%. Sensitivities are provided in Table 21-14.

- The final cost may be affected to some degree by future movements in exchange rates against the US dollar and by the current escalation trends in fuel, materials and construction rates.

In BDA's experience, projects of this type may incur cost overruns above the allocated contingency. BDA recommends that capital costs be monitored closely and estimates revised as necessary.

### 21.6.2 Operating Costs

The operating cost estimates were developed by OGC and AMDAD and are summarised in Table 21-9.

**Table 21-9: Projected operating costs**

| Item | | Life Of Mine | 2011 | 2012 | 2013 | 2014 | 2015 | 2016 | 2017 | 2018 | 2019 | 2020 | 2021-2031 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Total Material Moved | Mt | 53 | 0.9 | 5.9 | 5.6 | 6.6 | 9.2 | 6.9 | 1.8 | 1.1 | 1.2 | 1.2 | 12.2 |
| Total Ore Mined | Mt | 30 | - | 1.3 | 2.9 | 2.9 | 2.6 | 2.7 | 1.7 | 1.1 | 1.2 | 1.2 | 12.2 |
| Total Ore Milled | Mt | 30 | - | 0.8 | 2.4 | 2.5 | 2.5 | 2.5 | 2.5 | 2.0 | 1.2 | 1.2 | 12.2 |
| Concentrate Wet tonnes | Kt | 633 | - | 15 | 62 | 63 | 64 | 59 | 55 | 34 | 24 | 23 | 234 |
| Total gold | Koz | 1,302 | - | 7 | 45 | 49 | 65 | 62 | 101 | 94 | 78 | 63 | 740 |
| Copper in concentrate | Kt | 158 | - | 3 | 16 | 16 | 16 | 15 | 14 | 8 | 6 | 6 | 58 |
| Copper in concentrate | Mlb | 349 | - | 7 | 35 | 35 | 36 | 33 | 31 | 18 | 13 | 13 | 128 |
| | | | | | | | | | | | | | |
| Open Cut - Total Waste Mined | Mt | 23.0 | 0.9 | 4.6 | 2.7 | 3.7 | 6.6 | 4.2 | 0.1 | - | - | - | - |
| Open Cut - Total Material Mined | Mt | 36.8 | 0.9 | 5.9 | 5.6 | 6.6 | 9.2 | 6.9 | 1.7 | - | - | - | - |
| Open Cut - Total Ore Mines | Mt | 13.9 | - | 1.3 | 2.9 | 2.9 | 2.6 | 2.7 | 1.6 | - | - | - | - |
| | | | | | | | | | | | | | |
| Underground production | Mt | 15.8 | - | - | - | - | - | 0.0 | 0.1 | 1.1 | 1.2 | 1.2 | 12.2 |
| | | | | | | | | | | | | | |
| Mining costs | | | | | | | | | | | | | |
| Open Cut | US$/t moved | 1.6 | - | 0.8 | 1.7 | 1.7 | 1.8 | 1.9 | 2.2 | - | - | - | - |
| Underground | US$/t mined | 26.1 | - | - | - | - | - | - | 20.8 | 33.6 | 33.2 | 33.2 | 24.2 |
| Others | US$/t mined | 1.4 | - | 1.1 | 0.8 | 0.8 | 0.9 | 0.9 | 2.1 | 2.8 | 1.6 | 1.6 | 1.7 |
| | | | | | | | | | | | | | |
| Processing costs | | | | | | | | | | | | | |
| Power cost | US$/t milled | 4.3 | - | 6.0 | 5.4 | 5.1 | 5.1 | 5.2 | 3.1 | 3.3 | 3.9 | 3.9 | 4.0 |
| Reagents costs | US$/t milled | 2.3 | - | 2.3 | 2.3 | 2.3 | 2.3 | 2.3 | 2.3 | 2.3 | 2.3 | 2.3 | 2.3 |
| Spares costs | US$/t milled | 1.6 | - | 1.6 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.2 | 2.0 | 2.0 | 2.2 |
| Maintenance costs (ex owners team | US$/t milled | 0.6 | - | 0.6 | 0.4 | 0.3 | 0.3 | 0.3 | 0.3 | 0.4 | 0.7 | 0.7 | 0.8 |
| Plant Labour | US$/t milled | 0.8 | - | 1.2 | 0.6 | 0.6 | 0.6 | 0.6 | 0.5 | 0.6 | 1.1 | 1.1 | 1.1 |
| Maintenance Labour | US$/t milled | 0.2 | - | 0.3 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 | 0.2 | 0.2 | 0.2 |
| Unit Processing | US$/t milled | 9.8 | - | 12.0 | 9.8 | 9.5 | 9.5 | 9.5 | 7.3 | 8.0 | 10.2 | 10.3 | 10.6 |
| | | | | | | | | | | | | | |
| Other costs | | | | | | | | | | | | | |
| Unit Corporate & Site costs | US$/t milled | 4.0 | - | 5.1 | 3.3 | 3.2 | 3.2 | 3.2 | 3.0 | 2.9 | 4.7 | 4.7 | 4.9 |
| | | | | | | | | | | | | | |
| Freight | | | | | | | | | | | | | |
| Land Transport and Ship loading | US$/t concentrate | 63.4 | - | 71.3 | 67.1 | 67.0 | 66.7 | 67.1 | 67.0 | 58.9 | 61.2 | 62.1 | 59.2 |
| Sea Freight | US$/t concentrate | 30.0 | - | 30.0 | 30.0 | 30.0 | 30.0 | 30.0 | 30.0 | 30.0 | 30.0 | 30.0 | 30.0 |

Note:

[1] The processed volumes differ slightly from AMDAD's mine planning due to updated assumptions on processing plant rampup.

[2] The timeline in this table is for illustrative purpose only, as a decision to proceed with project construction still requires completion of project execution plan, regulatory approval and OGC board approval.

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

- Open pit mining costs as per contract rates tendered by Filipino contractors;
- Underground mining costs estimates based on first principles;
- Labour costs estimated based on OGC operating experience;
- Power costs based on operating metrics as provided by Arccon, 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;
- Corporate 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;
- Excise duty is 2% on copper net smelter revenue and 2% on gold sales;
- Royalties include a 2% net smelter return (NSR) and an additional 0.6% of NSR, which is capped at approximately A$13.5M;
- Land transport includes cost from site to Port Irene;
- Sea freight is based on cost from Philippines to China; and
- Post 2016, the forecast considers a reduction in costs due to a reduction in the number of expatriates, potential availability of power from the grid and reduction of overhead costs due to the drop in mined volumes.

### 21.6.3 Cash Costs

Cash costs average US$528/oz eqAu over the life of the mine. Cash costs after copper credits over the life of the mine are US$128/oz. Gold-equivalent ounces and copper credits are calculated at US$1050/oz gold and US$3.0lb/Cu.

### 21.6.4 Conclusions

Early in 2010, BDA reviewed all operating costs detailed in the preliminary optimisation study of December 2009. AMDAD has subsequently updated the mine schedules, mining capital and operating costs. John Wyche of AMDAD is the qualified person responsible for these updates. BDA accepts AMDAD's revisions and considers its previous conclusions to hold. These are:

- BDA considers that a 15% sensitivity should be run on the underground mining costs to examine the effect of uncertainties in the conditions to be experienced, in terms of hydrology and geomechanics.

- BDA anticipates that some operating costs will increase, based on the continuing increase in fuel and materials costs. The increased annual throughput will increase annual costs, but should reduce unit costs by a small amount. BDA considers that, based on the use of factors and budget quotations, the operating costs can be considered accurate to within ±15%.

- Mining costs, while achievable, are subject to the impact of geotechnical and hydrological conditions and in BDA's view could be under-estimated by 10-15%. The overall accuracy of the mining costs is considered ±15%. It is understood that the mine plans (and therefore costs) are under review and may be subject to change.

- A sensitivity analysis, including a ±10% increase in operating costs, is included in Table 21-14.

## 21.7 Economic Analysis

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 here.

The financial analysis methodology, discount rates, exchange rates, commodity prices and financial parameters applied in the financial model were sourced from OGC and BDA has verified and confirmed consistency of capital costs and operating costs inputs in the model.

### 21.7.1 Assumptions

The revenue-related assumptions in Table 21-10 were used in the Didipio Gold-Copper Project financial model.

**Table 21-10: Economic analysis assumptions**

| Year | TC | RC | Gold refining charge | Payable copper | Payable gold | Free gold (dore) | Refining charge (dore) |
|---|---|---|---|---|---|---|---|
| | US$/dmt | US c/lb Cu | US$/oz | % Cu | % Au | % Au | US$/oz |
| 2011 | 50 | 5.0 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2012 | 50 | 5.0 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2013 | 50 | 5.0 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2014 | 50 | 5.0 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2015 | 82 | 8.2 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2016 | 82 | 8.2 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2017 | 82 | 8.2 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2018 | 82 | 8.2 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2019 | 82 | 8.2 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |
| 2020 | 82 | 8.2 | 6.0 | 96.7% | 97.5% | 1% | 0.31 |

### 21.7.2 Cash Flow Analysis

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$800/oz for gold and US$2.60/lb for copper.

Table 21-11 to Table 21-13 indicate the NPV, Net Cash Flow and IRR sensitivity of the Didipio Gold-Copper Project to gold prices and copper prices.

**Table 21-11: NPV @ 10% – sensitivities to metal prices (US$M)**

| | Copper price - US$/lb (flat) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Gold Price - US$/oz (flat) | 2.00 | 2.20 | 2.40 | 2.60 | 2.80 | 3.00 | 3.20 | 3.40 | 3.60 | 3.80 | 4.00 |
| 800 | 36 | 53 | 70 | 86 | 102 | 117 | 132 | 146 | 161 | 175 | 189 |
| 850 | 49 | 65 | 82 | 97 | 113 | 127 | 142 | 156 | 171 | 185 | 199 |
| 900 | 60 | 77 | 93 | 108 | 123 | 138 | 152 | 167 | 181 | 195 | 209 |
| 950 | 72 | 88 | 103 | 119 | 134 | 148 | 162 | 177 | 191 | 205 | 219 |
| 1000 | 83 | 99 | 114 | 129 | 144 | 158 | 173 | 187 | 201 | 215 | 229 |
| 1050 | 94 | 110 | 125 | 140 | 154 | 168 | 183 | 197 | 211 | 225 | 239 |
| 1100 | 105 | 120 | 135 | 150 | 164 | 178 | 193 | 207 | 221 | 235 | 249 |
| 1150 | 116 | 131 | 145 | 160 | 174 | 189 | 203 | 217 | 231 | 245 | 259 |
| 1200 | 126 | 141 | 156 | 170 | 184 | 199 | 213 | 227 | 241 | 255 | 269 |
| 1250 | 137 | 151 | 166 | 180 | 195 | 209 | 223 | 237 | 251 | 265 | 279 |
| 1300 | 147 | 162 | 176 | 190 | 205 | 219 | 233 | 247 | 261 | 275 | 289 |
| 1350 | 158 | 172 | 186 | 201 | 215 | 229 | 243 | 257 | 271 | 285 | 299 |
| 1400 | 168 | 182 | 196 | 211 | 225 | 239 | 253 | 267 | 281 | 295 | 309 |
| 1450 | 178 | 192 | 207 | 221 | 235 | 249 | 263 | 277 | 291 | 305 | 318 |
| 1500 | 188 | 202 | 217 | 231 | 245 | 259 | 273 | 287 | 301 | 314 | 328 |
| 1550 | 198 | 213 | 227 | 241 | 255 | 269 | 283 | 297 | 311 | 324 | 338 |
| 1600 | 208 | 223 | 237 | 251 | 265 | 279 | 293 | 307 | 321 | 334 | 348 |

**Table 21-12: Total Net Cash Flow post Tax – sensitivities to metal prices (US$M)**

| | Copper price - US$/lb (flat) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Gold Price - US$/oz (flat) | 2.00 | 2.20 | 2.40 | 2.60 | 2.80 | 3.00 | 3.20 | 3.40 | 3.60 | 3.80 | 4.00 |
| 800 | 210 | 237 | 264 | 292 | 318 | 345 | 372 | 399 | 426 | 453 | 480 |
| 850 | 236 | 263 | 290 | 317 | 344 | 370 | 397 | 424 | 452 | 479 | 506 |
| 900 | 262 | 289 | 315 | 342 | 369 | 396 | 423 | 450 | 477 | 504 | 531 |
| 950 | 287 | 314 | 341 | 367 | 394 | 422 | 449 | 476 | 503 | 530 | 557 |
| 1000 | 313 | 339 | 366 | 393 | 420 | 447 | 474 | 501 | 529 | 556 | 583 |
| 1050 | 338 | 365 | 392 | 419 | 446 | 473 | 500 | 527 | 554 | 581 | 608 |
| 1100 | 363 | 390 | 417 | 444 | 472 | 499 | 526 | 553 | 580 | 607 | 634 |
| 1150 | 389 | 416 | 443 | 470 | 497 | 524 | 551 | 579 | 606 | 633 | 660 |
| 1200 | 414 | 442 | 469 | 496 | 523 | 550 | 577 | 604 | 631 | 658 | 686 |
| 1250 | 440 | 467 | 494 | 521 | 549 | 576 | 603 | 630 | 657 | 684 | 711 |
| 1300 | 466 | 493 | 520 | 547 | 574 | 601 | 628 | 656 | 683 | 710 | 737 |
| 1350 | 492 | 519 | 546 | 573 | 600 | 627 | 654 | 681 | 708 | 735 | 763 |
| 1400 | 517 | 544 | 571 | 599 | 626 | 653 | 680 | 707 | 734 | 761 | 788 |
| 1450 | 543 | 570 | 597 | 624 | 651 | 678 | 706 | 733 | 760 | 787 | 814 |
| 1500 | 569 | 596 | 623 | 650 | 677 | 704 | 731 | 758 | 785 | 812 | 840 |
| 1550 | 594 | 621 | 648 | 676 | 703 | 730 | 757 | 784 | 811 | 838 | 865 |
| 1600 | 620 | 647 | 674 | 701 | 728 | 755 | 783 | 810 | 837 | 864 | 891 |

**Table 21-13: IRR – sensitivities to metal prices (%)**

| | Copper price - US$/lb (flat) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Gold Price - US$/oz (flat) | 2.00 | 2.20 | 2.40 | 2.60 | 2.80 | 3.00 | 3.20 | 3.40 | 3.60 | 3.80 | 4.00 |
| 800 | 15% | 17% | 20% | 22% | 24% | 27% | 29% | 31% | 33% | 35% | 37% |
| 850 | 16% | 19% | 21% | 23% | 25% | 28% | 30% | 32% | 34% | 36% | 38% |
| 900 | 18% | 20% | 22% | 24% | 27% | 29% | 31% | 33% | 35% | 37% | 39% |
| 950 | 19% | 21% | 23% | 26% | 28% | 30% | 32% | 34% | 36% | 38% | 40% |
| 1000 | 20% | 22% | 25% | 27% | 29% | 31% | 33% | 35% | 37% | 39% | 41% |
| 1050 | 21% | 24% | 26% | 28% | 30% | 32% | 34% | 36% | 38% | 40% | 42% |
| 1100 | 23% | 25% | 27% | 29% | 31% | 33% | 35% | 37% | 39% | 41% | 43% |
| 1150 | 24% | 26% | 28% | 30% | 32% | 34% | 36% | 38% | 39% | 41% | 43% |
| 1200 | 25% | 27% | 29% | 31% | 33% | 35% | 36% | 38% | 40% | 42% | 44% |
| 1250 | 26% | 28% | 30% | 32% | 33% | 35% | 37% | 39% | 41% | 43% | 45% |
| 1300 | 27% | 29% | 31% | 32% | 34% | 36% | 38% | 40% | 42% | 44% | 46% |
| 1350 | 28% | 30% | 32% | 33% | 35% | 37% | 39% | 41% | 43% | 45% | 47% |
| 1400 | 29% | 31% | 32% | 34% | 36% | 38% | 40% | 42% | 44% | 46% | 47% |
| 1450 | 30% | 31% | 33% | 35% | 37% | 39% | 41% | 43% | 44% | 46% | 48% |
| 1500 | 30% | 32% | 34% | 36% | 38% | 40% | 42% | 43% | 45% | 47% | 49% |
| 1550 | 31% | 33% | 35% | 37% | 39% | 41% | 42% | 44% | 46% | 48% | 50% |
| 1600 | 32% | 34% | 36% | 38% | 39% | 41% | 43% | 45% | 47% | 49% | 50% |

Table 21-14 presents a sensitivity on capital costs and operating costs across the life of mine.

For every 10% change in life-of-mine capital costs, the NPV of the Didipio Gold-Copper Project at a 10% discount rate changes by approximately US$11M. For every 10% change in life-of-mine operating costs, the NPV of the Didipio Gold-Copper Project at a 10% discount rate changes by approximately US$20M.

**Table 21-14: NPV @ 10% – sensitivities on capital and operating costs (US$M)**

| Sensitivity | NPV Variation |
|---|---|
| -10% Capex | 11 |
| -10% Opex | 20 |
| +10% Capex | -11 |
| +10% Opex | -20 |

## 21.8 Mine Life

The predicted mine life of approximately 20 years is achievable based on the projected annual production rate and the estimated mineral reserves.

The Didipio FTAA is considerably under-explored and the potential for discovery of additional gold and/or gold-copper mineralised systems that will contribute to the Didipio Gold-Copper Project mining and treatment operation is interpreted to be high.

Exploration of the FTAA over the past 15 years by Climax Mining has resulted in identification of more than 25 known gold and gold-copper prospects that range from soil and rock-chip geochemical anomalies to more advanced drill targets. Following discovery of the Didipio Gold-Copper mineralisation, exploration of other prospects within the FTAA was placed on hold and was not advanced.

Many of the prospects more proximal to the Didipio Gold-Copper Project are interpreted to comprise higher-level epithermal gold mineralisation or alkalic porphyry gold-copper mineralisation associated with the large, predominantly buried alkaline Surong Intrusion, typical of most porphyry terrains. Outcropping gold and copper mineralisation in the western sector of the FTAA (i.e. Papaya) is interpreted as associated with a second large, buried alkaline intrusion.

Several prospects are being exploited by artisinal gold miners for both alluvial and hard-rock gold mineralisation. However, the majority of previously identified prospects are at an early stage of exploration and require additional layers of exploration activity to elevate to drill status. It is the company's intention to further elevate several of these prospects to a drill status during the 2011 year.

# 22 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 | 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 |
| $\mu m$ | 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 | OceanaGold 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 |
| 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 |
| SEX | sodium ethyl 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 |

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## CERTIFICATE OF AUTHOR

John S. McIntyre
Managing Director
Behre Dolbear Australia Pty Ltd
Level 9, 80 Mount St,
NORTH SYDNEY NSW 2060
AUSTRALIA

1. I John S. McIntyre (B.Eng.(Hons), FAusIMM, MMICA) am Managing Director of Behre Dolbear Australia Pty Ltd.

2. I graduated with a B.Eng. (Hons) Mining degree in mining engineering from the University of New South Wales ("UNSW") in 1971.

3. I am a Fellow of the Australian Institute of Mining and Metallurgy ("FAusIMM") and a Member of the Minerals Industry Consultants Association ("MMICA").

4. I have worked as a mining engineer for a total of 35 years since my graduation from UNSW and as a consultant mining engineer since 1987. I have been the Managing Director of Behre Dolbear Australia Pty Limited ("BDA") since 1994.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the review of sections 1.5, 1.7.2, 2, 3, 4.1-4.5, 4.7-4.9, 5, 16, 19, 20.2, and 21.2 to 21.8. of the technical report titled "Technical Report for the Didipio Gold-Copper Project" dated 29 October 2010 ("Technical Report").

7. Behre Dolbear Australia Consultants who work directly for me have visited the Property on several occasions in the past 10 years, most recently in March 2010.

8. As at 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.

9. I am independent of the issuer in accordance with section 1.4 of NI43-101.

10. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

John S. McIntyre (B.Eng. (Hons) Mining, FAusIMM, MMICA))

Managing Director

Date of Signature: 29 October 2010

## CERTIFICATE OF AUTHOR

John Wyche, MAusIMM and MMICA
Managing Director
Australian Mine Design and Development Pty Ltd
Suite 1.8, 741 Pacific Highway,
GORDON NSW 2072
AUSTRALIA

1. I John Wyche (B.Eng. (Hons), MAusIMM, MMICA) am Managing Director of Australian Mine Design and Development Pty Ltd.

2. I graduated with a B.Eng. (Hons) Mining degree in mining engineering from the University of Queensland ("UQ") in 1981. I am a Member of the Australasian Institute of Mining and Metallurgy ("MAusIMM") and a Member of the Minerals Industry Consultants Association ("MMICA").

3. I have worked as a mining engineer for a total of 26 years since my graduation from UQ and as a consultant mining engineer since 1989. I have been the Managing Director of Australian Mine Design and Development Pty Ltd ("AMDAD") since 1989.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I am responsible for the preparation of sections 1.4, 17.10, 21.1 and mining capital and operating costs for section 21.6 of the technical report titled "Technical Report for the Didipio Gold-Copper Project" dated 29 October 2010 ("Technical Report").

6. I have visited the Property on several occasions in the past 4 years, most recently in February 2008.

7. I have been involved in the Didipio Project as an Independent Technical Engineer since 2003. The nature of this involvement includes mine planning work completed in relation to reserves estimation, grade control methodology, mine designs and production scheduling, capital and operating cost estimations and mining contract preparation and tendering.

8. As at 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.

9. I am independent of the issuer in accordance with section 1.4 of NI43-101.

10. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

John Wyche (B.Eng. (Hons) Mining, MAusIMM, MMICA))

Date of Signature: 29 October 2010

**CERTIFICATE OF AUTHOR**

Jonathan Godfrey Moore
Principal Resource Geologist
22 Maclaggan Street, Dunedin
NEW ZEALAND

As a qualified person responsible for the report titled "Technical Report for the Didipio Gold-Copper Project" dated 29 October 2010, (the "Technical Report") to which this certificate applies, I, Jonathan Godfrey Moore do hereby certify that:

1. I, Jonathan Godfrey Moore, am the Principal Resource Geologist 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 in good standing of the Australasian Institute of Mining and Metallurgy.

4. I have worked as a geologist in the mining industry for a total of 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 Gold-Copper Project was in August 2008.

7. I am responsible for sections 1.1, 1.2, 1.3, 1.6, 1.7.1, 4.6, 6-15, 17.1-17.9, 18, 19, 20.1 and 20.3 of the Technical Report.

8. 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.

9. Prior to my employment with Oceana in May 1996 I had no involvement with the Didipio Gold-Copper 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 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.

________________________________
Jonathan Godfrey MOORE

Date of Signature: 29 October 2010