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OYU TOLGOI PROJECT IDOP Technical Report



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Title Page

Project Name: OYU TOLGOI PROJECT

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Southern Oyu	15 April 2005
Hugo North and Hugo North Extension	1 November 2006
Hugo South	1 November 2003
Heruga	30 March 2010

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1 SUMMARY

1.1 Project Overview

The Oyu Tolgoi copper and gold project (the Project) is located in the Southern Gobi region of Mongolia and is being developed by Oyu Tolgoi LLC (OT LLC). OT LLC was formerly called Ivanhoe Mines Mongolia Inc XXK (IMMI). The Project consists of a series of deposits containing copper, gold, silver and molybdenum. The deposits stretch over 26 km, from the Hugo North deposit in the north through the adjacent Hugo South, down to the Southern Oyu deposit and extending to the Heruga deposit in the south. Construction of the first phase (Phase 1) of the Project is well advanced and includes open pit mining of Southern Oyu, a concentrator and supporting infrastructure. Phase 2 of the Project will include production from the Hugo North Lift 1 block cave, expansion of the concentrator and infrastructure and a power station. The Project is currently in the later stages of construction for the initial phase of development.

This report, the IDOP Technical Report, was prepared for Ivanhoe Mines Ltd (Ivanhoe). The IDOP Technical Report is based on the technical, production and cost information contained in the OT LLC study titled Integrated Development and Operations Plan (IDOP). IDOP was completed by the Rio Tinto PLC (Rio Tinto) appointed management of OT LLC in March 2011 as the basis for the proposed project financing.

Independent Qualified Persons (QPs), acting on behalf of Ivanhoe, reviewed the IDOP as part of the preparation for the IDOP Technical Report and in conjunction with Ivanhoe prepared the IDOP Technical Report with costs to the end of 2011 as reported to Ivanhoe by OT LLC. The IDOP Technical Report represents a view of the Project by Ivanhoe and the QPs, and is not necessarily the view of the Rio Tinto appointed management of OT LLC or Rio Tinto.

IDOP used only Measured and Indicated Mineral Resources and is a complete study of all aspects of the Project. The IDOP Technical Report analyses a reserve case only (IDOP Reserve Case) and is based on a pre-feasibility quality level study complying with Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), although some parts of the Oyu Tolgoi Project are further advanced and are at feasibility level. The IDOP Technical Report meets the standards of US SEC Industry Guide 7 requirements for reporting Mineral Reserves.

Independent reviews completed in the preparation of IDOP, indicate a number of areas where project value can be further improved or optimized. The report also notes that management has considerable potential after initial construction and operations commence to further optimize the economic returns to stakeholders.

The analysis in the IDOP Technical Report has been called the IDOP Reserve Case in order to distinguish it from the cases in previous studies of Oyu Tolgoi. IDOP is an update of the previous comprehensive project study Integrated Development Plan 2010 (IDP10). IDP10 was published in June 2010 and was an update to the original IDP05, incorporating all summaries of the Project within the framework of a signed and effective IA with the Government of Mongolia (GOM). Both IDP05 and IDP10 were published in Technical Reports.

In July of 2009, the Mongolian Parliament voted to authorize the GOM (GOM) to conclude the Oyu Tolgoi Investment Agreement (IA) with Ivanhoe Mines and Rio Tinto, its strategic shareholder.

The Investment Agreement (IA) was signed on 6 October 2009 with ten conditions precedent required before the IA was to be considered "effective." On 31 March 2010 the comprehensive IA took full legal effect after the GOM confirmed that the conditions precedent had been satisfied.

Under the terms of the IA, the GOM agreed to become a partner in the development of the Project. The GOM has acquired a 34% interest in the Ivanhoe Mines subsidiary, Oyu Tolgoi LLC, which holds the Oyu Tolgoi mining licences. Mongolia's interest will be held through the state-owned, sovereign-wealth resources company Erdenes MGL LLC. Ivanhoe Mines will own a 66% indirect interest in OT LLC.

Given the extent of the mineral discoveries associated with the Project and the potential for additional discoveries, Ivanhoe Mines and the GOM agreed that the approved IA should conform with the provision of Mongolia's current Minerals Law specifying that certain deposits of strategic importance qualify for 30 years of stabilized tax rates and regulatory provisions, with an option of extending the term of the IA for an additional 20 years. It is important to note that the IA was negotiated on the same principles as projected in the Life of Mine (Sensitivity) Case of IDP10. Major taxes and rates stabilized for the life of the agreement include: corporate income tax, customs duty, value-added tax; excise tax; royalties; exploration and mining licences; and immovable property and/or real estate tax.

The parties to the agreement had been negotiating terms since Ivanhoe Mines prepared the 2005 Integrated Development Plan (IDP05). During that time, and subsequent to the finalization of the IA, OT LLC has advanced considerably in its planning for the Project's development, its underground shaft and lateral development program, its procurement, construction, detailed capital estimation and construction activities. The Project has also seen significant growth in its mineral resource base over this period.

The series of deposits contain a currently identified resource of almost 40 billion pounds (blb) of contained copper in concentrate and 20 million ounces (Moz) of contained gold in concentrate in the Measured and Indicated Mineral Resource categories and another 40 Blb of contained copper and 25 Moz of contained gold in the Inferred category (see the Mineral Resources summary in Table 1.2). The Oyu Tolgoi trend is still open to the north and south and the deposits have not been closed off at depth.

IDP05 laid the initial blueprint for the construction and operation of a concentrator to be fed from the mining of the Southern Oyu deposit as an open pit and Hugo North deposit as an underground block cave.

In 2006, Ivanhoe presented a NI 43-101 Technical Report stating a Mineral Reserve based on the Southern Oyu Open Pit. The analysis examined an open pit only option while underground planning work continued to be progressed. As part of the GOM's approval process, open pit reserves were registered in 2007 and updated with open pit and underground reserves in 2009 to convert the Shivee Tolgoi and Javkhlant

exploration licences to mining licences (Mongolia has its own code for registering reserves that is apart of the GOM approval process).

IDOP uses the same Mineral Resources as estimated for the previous study IDP10 and the Mineral Reserves of IDOP are very similar to those of IDP10. IDOP includes resources from the Oyu Tolgoi deposit (wholly owned by OT LLC) and Entrée Joint Venture (EJV) licence areas. Although the overall strategy for the development of Oyu Tolgoi remains the same in IDOP as it did in IDP10 and IDP05, there have been changes to several key areas which are addressed in this project update. The changes to date include:

- Updated open pit designs on Southern Oyu Tolgoi and commencement of open pit mining.
- Revised start date for first ore production from third quarter 2013 to third quarter 2012.
- Revision of production schedule to account for changed timing of the underground and plant expansion between IDP10 and the IDOP.
- Revisions to capital estimates and updates for costs expended to date.

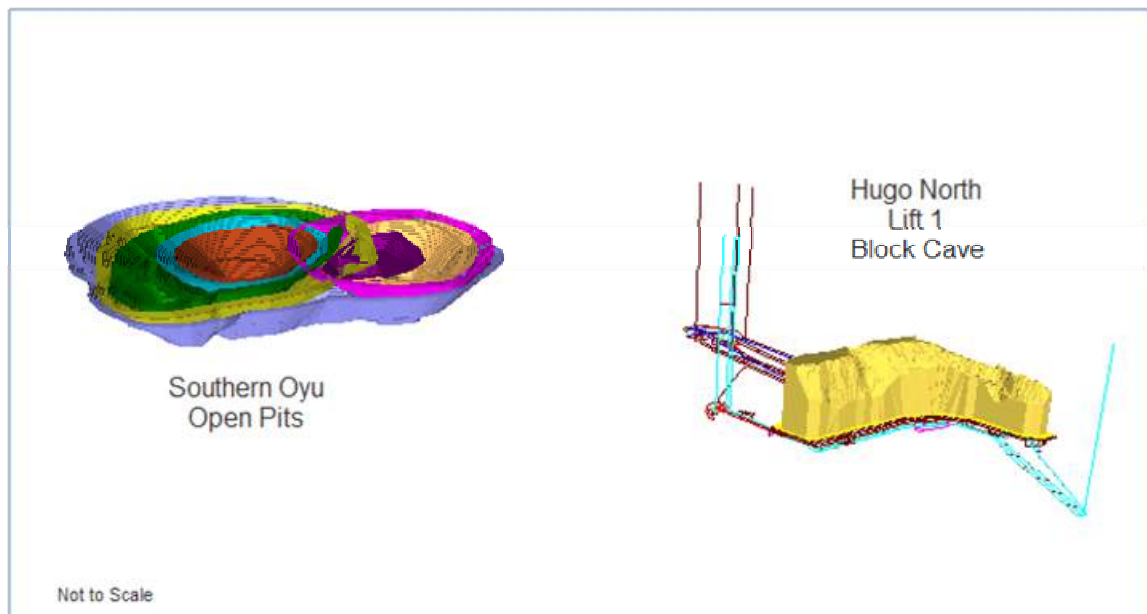
Oyu Tolgoi has a large mineral resource providing management with flexibility in studying alternative paths for mine development to match future economic conditions. Ongoing planning work using Inferred Resources has identified the potential for further expansions.

Five deposits have been identified in the mineral resource at Oyu Tolgoi; they are Southwest, Central, Hugo South, Hugo North, and Heruga. Southwest and Central comprise the Southern Oyu Tolgoi deposit, and Hugo South and Hugo North comprise the Hugo Dummett deposit. Heruga is a separate deposit south of the Southern Oyu deposit. The mine planning work to date suggests the following relative ranking for overall return from each deposit, from highest value to lowest:

- Hugo North
- Southwest
- Central
- Hugo South
- Heruga

The IDOP Reserve Case assumes processing of 1.4 bt of ore over a 27 year period, mined from the Southern Oyu open pit and the first lift in the Hugo North underground block cave. The mining areas included in the IDOP Reserve Case are shown schematically in Figure 1.1.

Figure 1.1 IDOP Reserve Case Mining Areas



The predominant source of ore at start up is the Southern Oyu Open Pit. In parallel to this surface construction, underground infrastructure and mine development is ongoing for the Hugo North underground block cave deposit. Stockpiling allows the higher grade ore from Hugo North to gradually displace the open pit ore as the underground production ramps up to reach 85 ktpd.

The ore is planned to be processed through conventional crushing, grinding and flotation circuits. The concentrate produced will be trucked to smelters in China.

The construction execution of the Project is led by the OT LLC Project Director supported by the Project Management Team. Oyu Tolgoi is a remote greenfields project and therefore requires extensive infrastructure to be constructed in addition to the concentrating facilities. The major initial infrastructure elements include:

- Water Borefields
- Water Treatment
- Housing
- Airstrip
- Supporting Facilities
- Power

In December 2010 a revised Heads of Agreement was signed by Rio Tinto and Ivanhoe Mines. The agreement established that Rio Tinto would manage the Core Operations and Ivanhoe Mines would manage the Exploration Activities within the non-core area—the area beyond a 200 m buffer from those resources included in the sensitivity cases of IDP10. The Core Operations are defined in the IA as including prospecting and exploring within the IA contract area; separating and mining minerals from land surface

and subsoil; construction and operation of infrastructure for transportation, power, water and other infrastructure facilities; processing (including crushing, grinding, floating, and filtering); producing products; marketing; mine closure and rehabilitation; ore stockpiling, waste and tailings management, and all other connected activities.

The next phase in the Project planning process after IDOP is the preparation of a Detailed Integrated Development and Operating Plan (DIDOP) that builds on IDOP. The DIDOP will integrate a number of detailed capital expansion studies into an overall project report. The key additional work to be assessed in DIDOP includes: the Phase 2 Project Expansion, infrastructure, power supply, water permitting, concentrate marketing, the underground feasibility study, and further work on mine closure and reclamation plan. OT LLC has advised that they anticipate the underground feasibility study in DIDOP will include a delay in the underground production schedule. OT LLC suggests that the underground production could be delayed by around twelve months. OT LLC plans to finalise DIDOP in late 2012.

In order to provide an analysis of IDOP for the IDOP Technical Report, costs to December 2011 were treated as sunk and escalation was applied to the operating and capital costs to prepare a schedule of expected cash flow. Metal prices used for the analysis were assumed to trend from current prices down to the long term price forecasts over five years. A summary of the production and financial results for the IDOP Reserve Case are shown in Table 1.1. Throughout this Report, measurements are in metric units and currency in United States dollars unless otherwise stated.

Table 1.1 IDOP Reserve Case Summary Production and Financial Results

Description	Units	IDOP Reserve Case
Inventory		Mineral Reserve
Production Rate (after expansion)	Mt/a	58
Total Processed	bt	1.40
NSR	\$/t	55.34
Cu Grade	%	0.94
Au Grade	g/t	0.35
Copper Recoverable	Billion lb	25.3
Gold Recoverable	Moz	12.4
Mine Life	years	27
Life of Mine Cash Cost After Credits	\$/lb Cu	0.81
Phase 1 Capital	US\$B	6.2
Expansion Capital	US\$B	5.1
NPV (8%) After Tax	US\$B	11.4
IRR After Tax	%	31
Payback Period	years	6.1

Notes:

- The NSR has been calculated with assumptions for smelter refining and treatment charges, deductions and payments terms, transport costs, metallurgical recoveries and royalties.
- Long term metal prices used in the revenue calculation, NPV, IRR and Payback economic analyses and NSR calculation are: copper US\$2.85/lb, gold US\$1,200/oz, silver US\$16.60/oz. Metal prices are assumed to fall from current prices to the long term average over five years.
- Phase 1 capital includes only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

The cash flow is shown in Figure 1.2. Concentrate and copper and gold metal production is shown in Figure 1.3. The annual production tonnages processed at the major pit stages and the block cave lifts (split between mining method) are shown in Figure 1.4.

Figure 1.2 Cumulative After Tax Cash Flow – IDOP Reserve Case

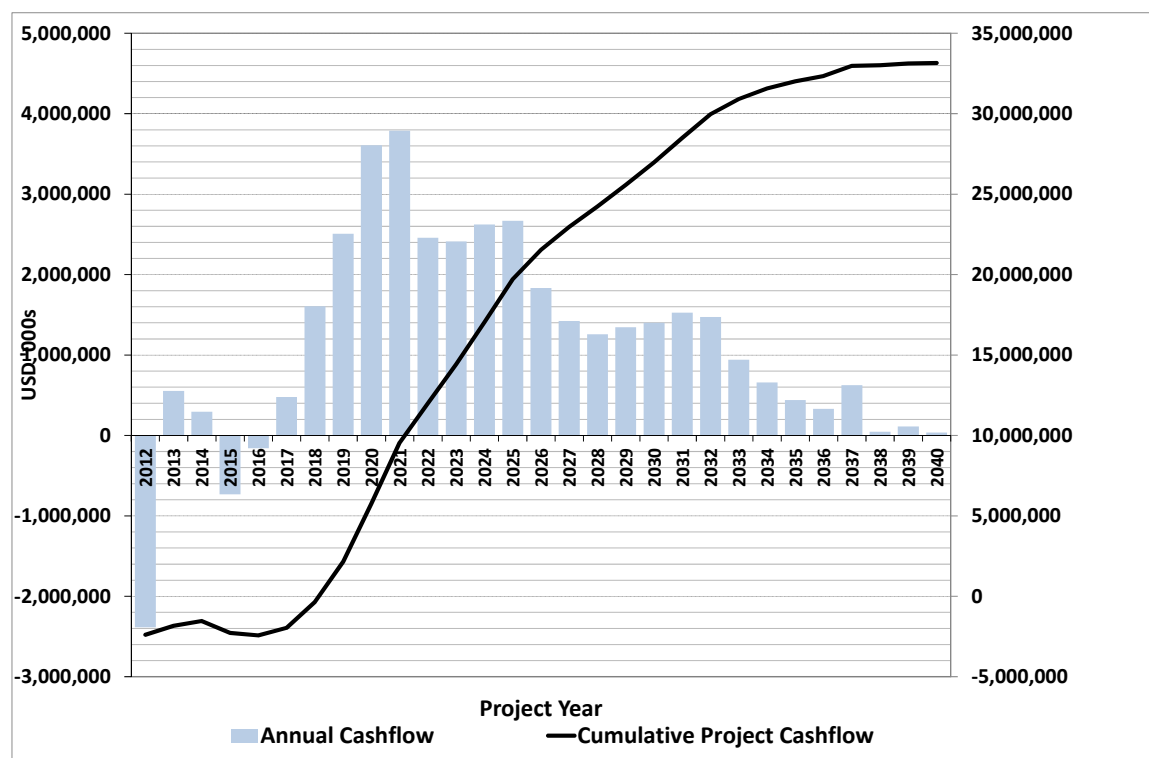


Figure 1.3 Concentrate and Metal Production – IDOP Reserve Case

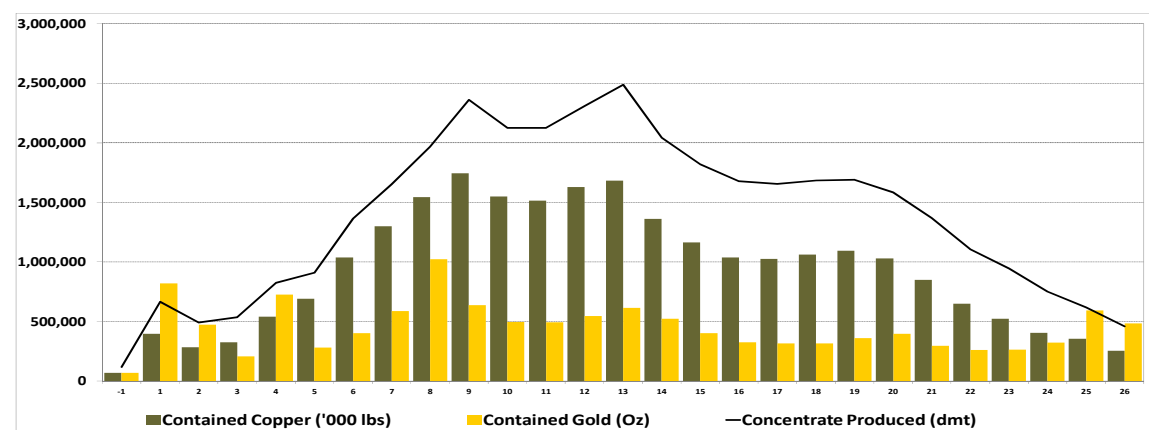
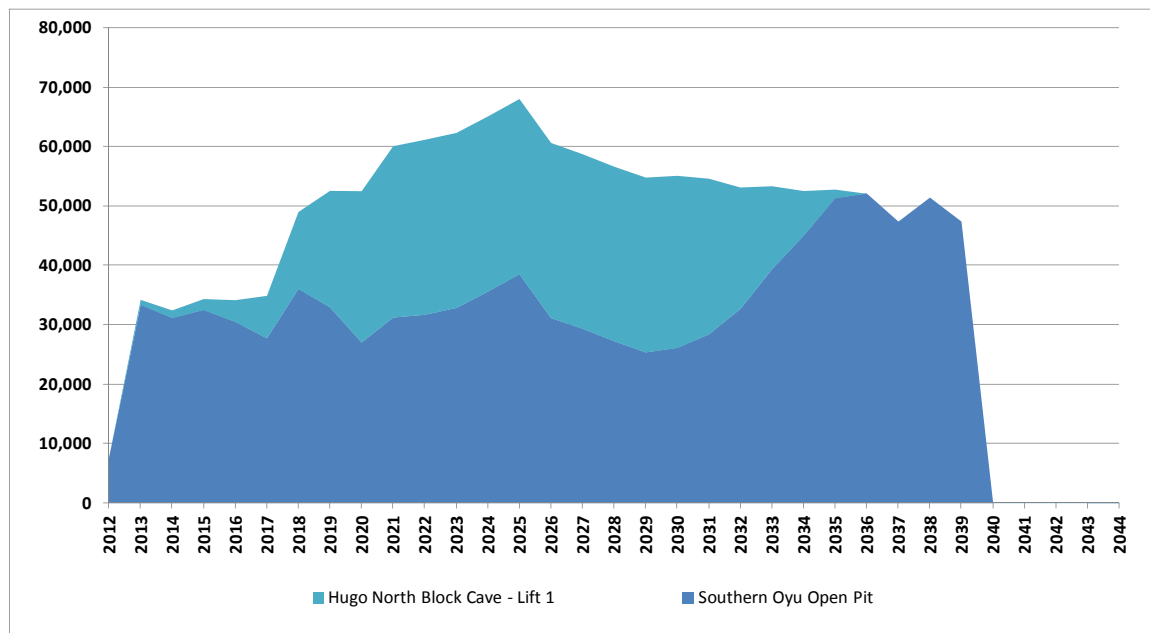


Figure 1.4 Processing by Source – IDOP Reserve Case



1.2 Qualified Persons

The following QPs were responsible for the preparation of the IDOP Technical Report:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by AMC Consultants Pty Ltd as Mining Manager/Principal Mining Engineer, was responsible for the overall preparation of the report and the Mineral Reserve estimates.
- Scott Jackson, B.Sc. (Hons), CFSG, FAusIMM (201735), employed by Quantitative Geoscience Pty Ltd (trading as “Quantitative Group” and “QG”) as Principal Consultant, was responsible for preparation of the Mineral Resource estimates.
- Albert Chance, B App Sc, Association of Professional Engineers of the Province of British Columbia (16370), an employee of Golder Associates Ltd., was responsible for preparation of the subsection on Open Pit Mine Geotechnical.
- Jarek Jakubec, C. Eng MIMMM (509147), Corporate Consultant, Rock Mechanics, an employee of SRK Consulting Inc., was responsible for preparation of the subsection on Underground Geotechnical Sections.
- Dean David, B AppSc (Metallurgy), FAusIMM (102351), employed by AMEC Minproc Limited (formerly GRD Minproc Limited) as Technical Director Process, was responsible for preparation of the sections related to and including processing.

1.3 Project Location and Ownership

The majority of the identified mineralization on the Project occurs within the Oyu Tolgoi Property at the Hugo Dummett and Southern Oyu porphyry deposits. Ivanhoe Mines operates the Project through OT LLC. OT LLC, in turn, holds its rights to the Project through mining licence 6709A (the "OT Licence"), comprising approximately 8,496 ha of property. The GOM granted the OT Licence to IMMI (now OT LLC) in 2003 along with mining licences for three properties identified as mining licences 6708A, 6710A and 6711A. Subsequently, licence 6711A has been relinquished. The OT Licence includes the right to explore, develop mining infrastructure and facilities and conduct mining operations on the Project. In 2006, the Mongolian parliament passed new mining legislation and changed the term of mining licences to a 30 year term with two 20 year extensions. Figure 1.5 shows the location of Oyu Tolgoi regionally and along the Mongolian Chinese border.

Figure 1.6 shows the deposits and licence boundaries.

Figure 1.5 Project Location

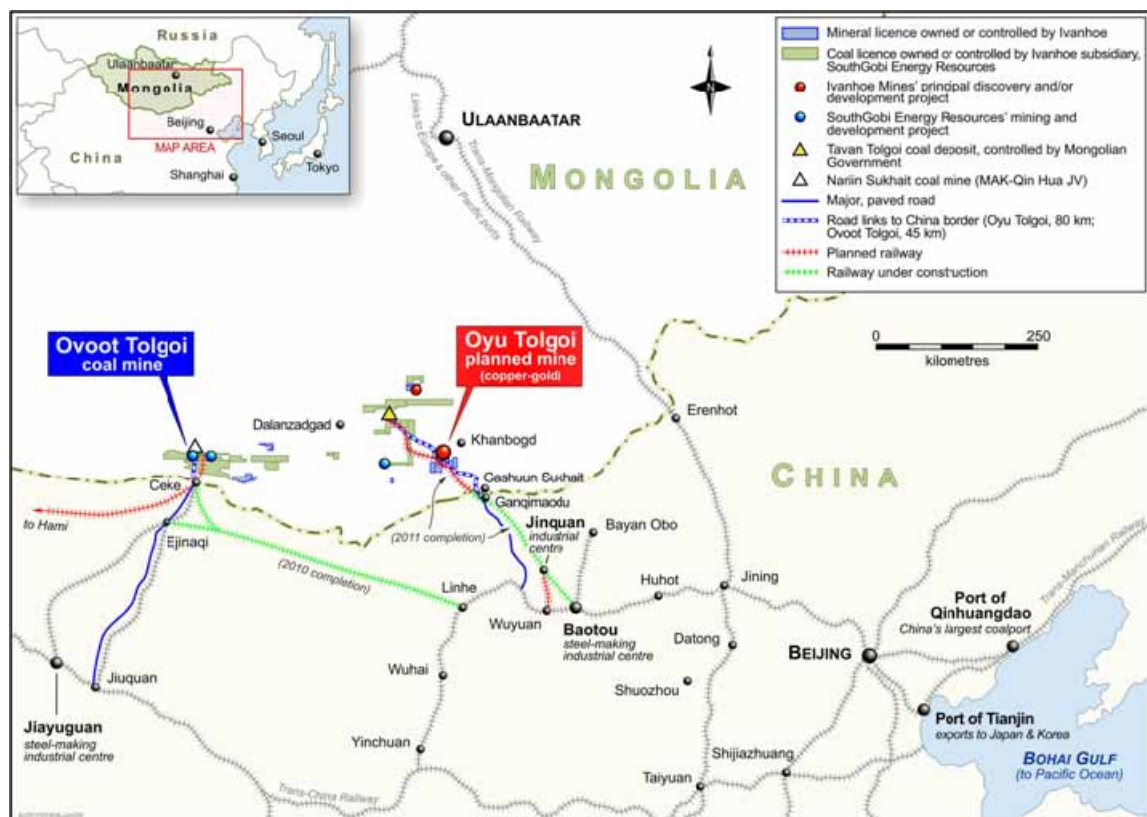
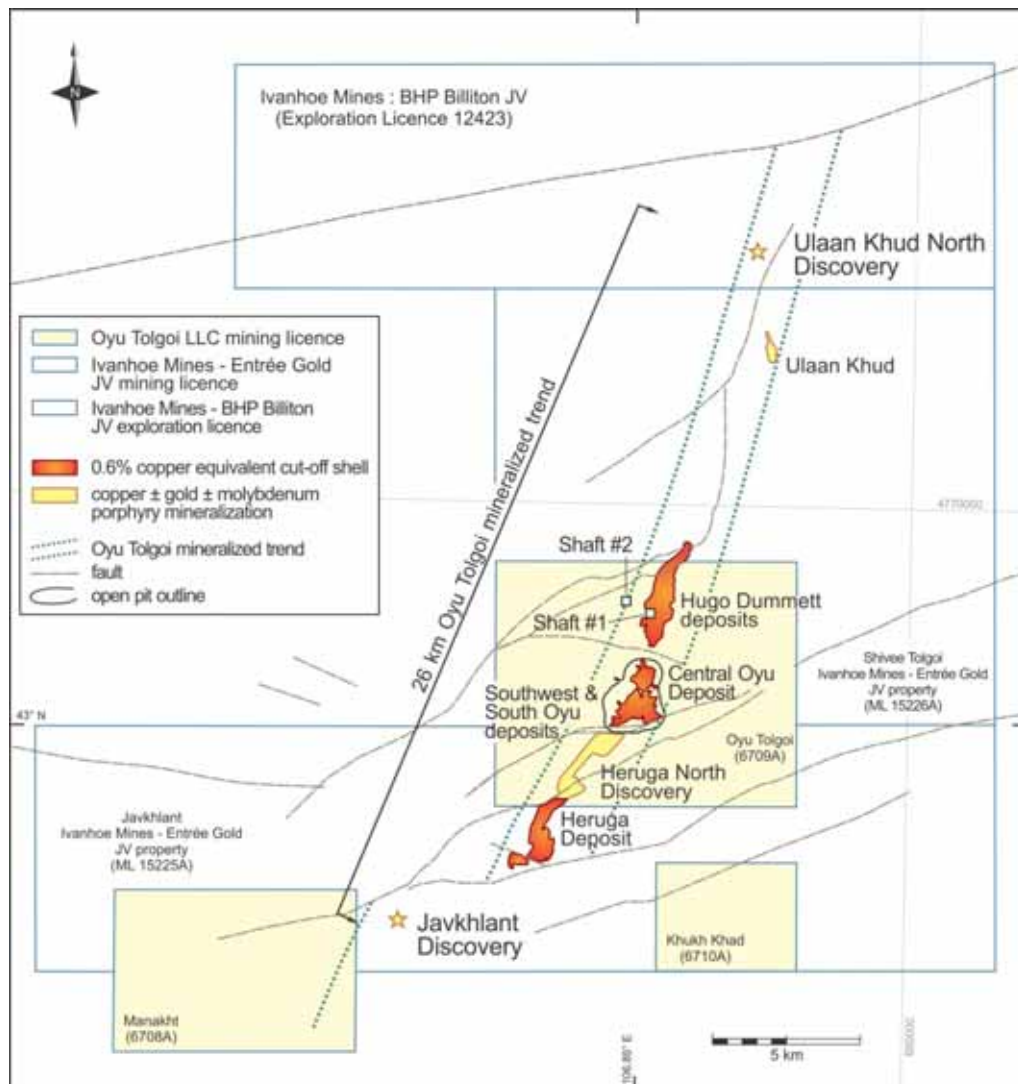


Figure 1.6 Oyu Tolgoi and Surrounding Licences



The OT Licence property was surveyed by an independent consultant in 2002 and by a qualified Mongolian Land Surveyor in 2004 to establish the legal boundaries of the OT Licence concession.

In June 2011, the GOM passed Resolution 175, the purpose of which is to authorize the designation of certain land areas for “state special needs” within certain defined areas in proximity to the Project. These state special needs areas are to be used for infrastructure facilities necessary in order to implement the development and construction of major mineral projects including the Project. Most of the land areas designated for special needs are already subject to existing mineral exploration and mining licenses issued by the GOM to third parties and, in certain cases, a mineral resource has been declared and registered with the applicable governmental authorities in respect of such licenses. OT LLC has entered into certain consensual arrangements with some of affected third parties; however, such arrangements have not been completed with all affected third parties. If OT LLC cannot enter into consensual

arrangements with an affected third party and such third party's rights to use and access the subject land area are adversely affected by application of Resolution 175, the GOM will be responsible for compensating such third parties in accordance with the mandate of Resolution 175. It is not clear at this time, whether the GOM will expect some of the compensation necessary to be paid to such third parties to be borne by OT LLC or if it will assume that obligation alone. It is also expected but not yet formally confirmed by the GOM that any consensual arrangements effected with affected third parties by OT LLC will make the application of Resolution 175 unnecessary. To the extent that consensual arrangements are not entered into with affected third parties and the GOM seeks contribution or reimbursement from OT LLC for compensation it provides such third parties, the amount of such contribution or reimbursement is not presently quantifiable and may be significant. The description of Resolution 175 has been provided by Ivanhoe and has been relied on under Item 3 of NI 43-101 Reliance on Other Experts.

1.4 Mineral Resources

The total Mineral Resources for the Project, including the updated Heruga deposit Mineral Resources, reported as of 31 March 2010 are shown in Table 1.2. Table 1.2 also includes mineralization on the Shivee Tolgoi and Javkhlant Joint Venture Properties, which are shown separately. A profile of the Oyu Tolgoi deposits is shown in Figure 1.7.

The base case CuEq cut-off grade assumptions for each deposit were determined using cut-off grades applicable to mining operations exploiting similar deposits. The CuEq cut-off applied was 0.6% CuEq.

The equivalent grade was calculated using assumed metal prices of US\$1.35/lb for copper, US\$650/oz for gold and US\$10.00/lb for molybdenum. The CuEq formula also took into account the different levels of metallurgical recovery for the metals. Gold was assumed to have 91% of copper recovery, molybdenum 72% of copper recovery. For convenience the formula is:

$$\text{CuEq\%} = \text{Cu\%} + ((\text{Au g/t} \times 18.98) + (\text{Mo g/t} \times 0.01586)) / 29.76$$

Molybdenum is used only in the CuEq formula at the Heruga deposit. The contained gold and copper estimates in Table 1.2 have not been adjusted for metallurgical recoveries.

Figure 1.7 Idealized Profile of Southern Oyu, Hugo Dummett, and Heruga Deposits (Section Looking West)

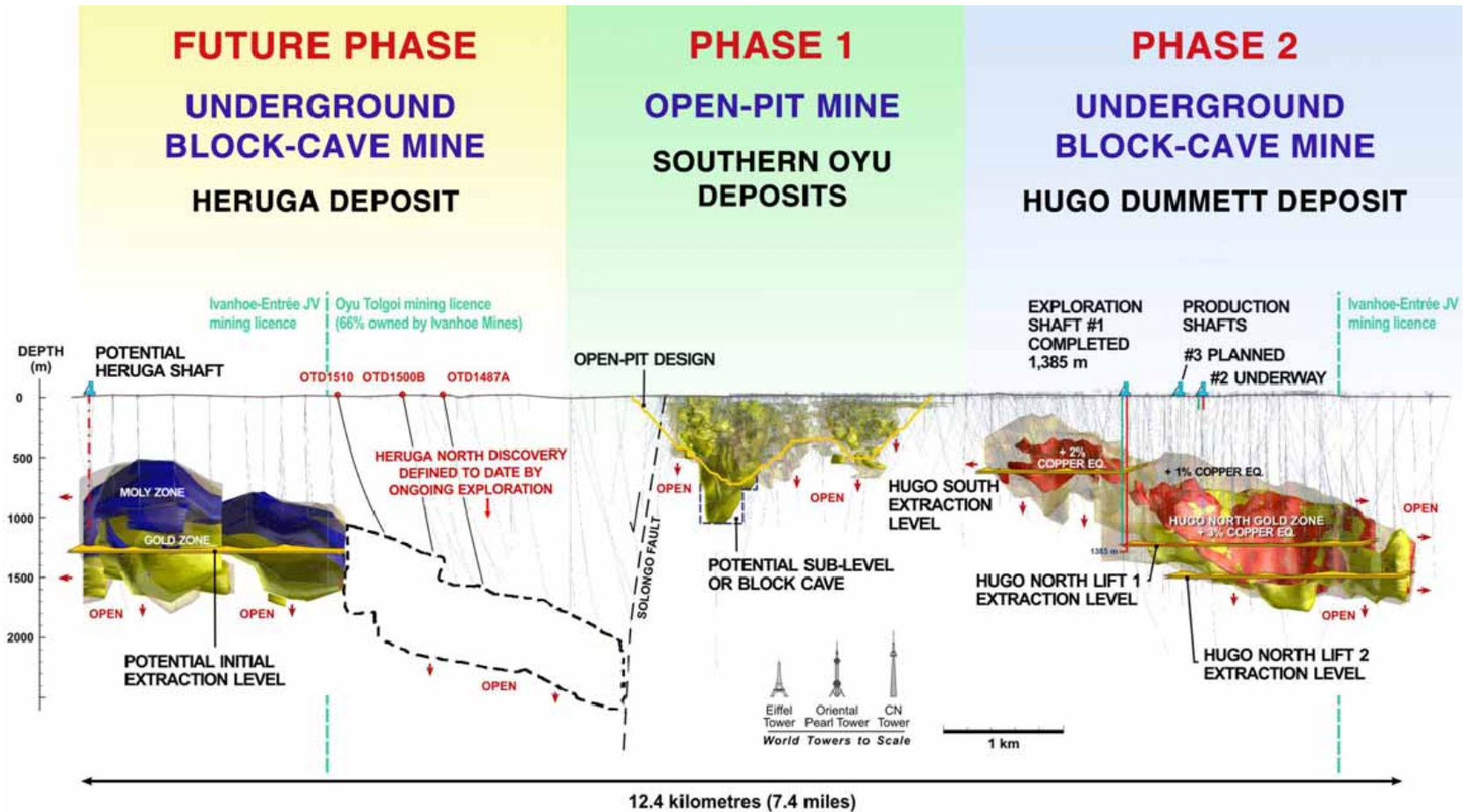


Table 1.2 Oyu Tolgoi Mineral Resource Summary, 29 March 2012

Deposit	Tonnage (t)	Cu (%)	Au (g/t)	CuEq (%)	Contained Metal		
					Cu (000 lb)	Au (oz)	CuEq (000 lb)
Southern Oyu Deposits							
Measured	101,590,000	0.64	1.10	1.34	1,430,000	3,590,000	3,000,000
Indicated	465,640,000	0.62	0.43	0.89	6,360,000	6,440,000	9,140,000
Measured + Indicated	567,230,000	0.62	0.55	0.97	7,750,000	10,030,000	12,130,000
Inferred	88,500,000	0.47	0.41	0.73	920,000	1,170,000	1,420,000
Hugo Dummett Deposits							
Indicated (Hugo North – OT LLC)	703,200,000	1.82	0.39	2.07	28,220,000	8,820,000	32,090,000
Indicated Shivee Tolgoi (Hugo North – EJV)	117,000,000	1.80	0.61	2.19	4,640,000	2,290,000	5,650,000
Indicated (All Hugo North)	820,200,000	1.82	0.42	2.08	32,910,000	11,080,000	37,610,000
Inferred (Hugo North – OT LLC)	722,800,000	0.97	0.30	1.17	15,460,000	6,970,000	18,640,000
Inferred Shivee Tolgoi (Hugo North – EJV)	95,500,000	1.15	0.31	1.35	2,420,000	950,000	2,840,000
Inferred (All Hugo North)	818,300,000	1.00	0.30	1.19	18,040,000	7,890,000	21,470,000
Inferred (Hugo South)	490,330,000	1.05	0.09	1.11	11,350,000	1,420,000	12,000,000
Inferred (Hugo North and South)	1,308 630,000	1.02	0.22	1.16	29,430,000	9,260,000	33,470,000
Heruga Deposit							
Inferred Heruga Javkhlant EJV	910,000,000	0.48	0.49	0.87	9,570,000	14,300,000	17,390,000
Inferred Heruga Ivanhoe	60,000,000	0.48	0.37	0.78	670,000	700,000	1,090,000
Inferred (All Heruga)	970,000,000	0.48	0.48	0.86	10,240,000	15,000,000	18,480,000
Oyu Tolgoi Project Grand Total							
Measured	101,590,000	0.64	1.10	1.34	1,430,000	3,590,000	3,000,000
Indicated	1,285,840,000	1.38	0.42	1.65	39,120,000	17,360,000	46,770,000
Measured + Indicated	1,387,430,000	1.33	0.47	1.63	40,680,000	20,970,000	49,860,000
Inferred	2,367,130,000	0.78	0.33	1.02	40,610,000	25,390,000	53,280,000

Notes:

- The contained gold and copper estimates in the tables have not been adjusted for metallurgical recoveries.
- The 0.6% CuEq cut-off has been used to enable comparison with previous disclosures.
- The Mineral Reserves are not additive to the Mineral Resources.
- CuEq has been calculated using assumed metal prices (\$1.35/lb for copper and \$650/oz for gold and \$10/lb for molybdenum); $CuEq\% = Cu\% + ((Au\text{ g/t} \times 18.98) + (Mo\text{ g/t} \times 0.1586)) / 29.76$. Mo grades outside of Heruga are assumed to be zero for CuEq calculations. The CuEq formula also took into account the different levels of metallurgical recovery for the metals. Gold was assumed to have 91% of copper recovery, molybdenum 72% of copper recovery.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- EJV is the Entrée Gold Joint Venture. The Shivee Tolgoi and Javkhiant licences are held by Entrée Gold. The Shivee Tolgoi and EJV Javkhiant Licences are planned to be operated by OT LLC. OT LLC will receive 80% of cash flows after capital and operating costs for material originating below 560 m, and 70% above this depth.

1.5 Mineral Reserves

Total Mineral Reserves for the Project and the OT LLC and EJV Mineral Reserves for the open pit and underground components of the Project are shown in Table 1.3. The Mineral Reserves for IDOP Technical Report are based on mine planning work prepared by OT LLC as presented in IDOP. This work was reviewed extensively and has been used as the basis for reporting the IDOP Technical Report Mineral Reserves.

The mine design work on the Southern Oyu open pits was prepared by OT LLC for IDOP and was used as the basis for the March 2012 Southern Oyu Open Pit Mineral Reserve by AMC. The mine design work on Hugo North Lift 1 was prepared by Stantec for IDP10 and was used as the basis for the 2010 Hugo North Underground Mineral Reserve. This work was reviewed by OT LLC and accepted as the basis for the underground mine planning in IDOP. AMC has agreed with this conclusion and has used the results for the March 2012 Hugo North Underground Mineral Reserve estimate.

The Mineral Reserves for the Project have been estimated using the Southern Oyu and Hugo North Mineral Resources. The Hugo North Mineral Reserve contains ore that is on the OT LLC Oyu Tolgoi licence and on the EJV Shivee Tolgoi licence. The EJV Shivee Tolgoi licence is subject to a joint venture agreement between OT LLC and Entrée Gold.

Mineral Reserves were first reported in the IDP10 Technical Report June 2010. A reconciliation of the IDP10 and IDOP Mineral Reserves is shown in Table 1.4. Although the total tonnes of the Mineral Reserve sets are very similar there is a 2.4% increase in Cu grade and a 5.1% decrease in gold grade. These are matched by a 1.5% increase in recoverable copper and a 5.6% decrease in recoverable gold. Almost all of the difference is in the IDOP Southern Oyu open pit designs. The pit optimization work on Southern Oyu is ongoing and has not yet defined the final pit limits. High grade resources remain in the floor of the IDOP pit design in Southwest that would be expected to be included in updated pit designs. Calculation of the underground Mineral Reserve by OT LLC resulted in a rounding up of the Hugo North Lift 1 tonnage on the Oyu Tolgoi licence by 1 Mt to 411 Mt and, on the EJV Shivee Tolgoi licence, the same tonnage but increased grades.

1.5.1 Southern Oyu Open Pit Mineral Reserve

In order to estimate the Mineral Reserves, AMC relied on the study work prepared by OT LLC. Pit designs were prepared using industry standard mining software, assumed metal prices as described above and smelter terms as set forth in the IDOP Technical Report. The estimate was prepared on a simplified project analysis on a pre-tax basis. Key outstanding variables noted by AMC include: marketing matters, water supply and management, underground ventilation development and power supply. The report only considers Mineral Resources in the Measured and Indicated categories, and engineering that has been carried out to a pre-feasibility level or better to estimate the open pit Mineral Reserve.

1.5.2 Hugo North Underground Mineral Reserve

Mine planning work by OT LLC has continued during since the previous Mineral Reserve estimate in 2010 but has not indicated any significant change in the Mineral Reserve.

The Hugo Dummett underground deposit will be mined by block caving; a safe, highly productive, cost-effective method. The deposit is comparable in dimension and tonnage to other deposits currently operating by block cave mining elsewhere in the world. The mine planning work has been prepared using industry standard mining software, assumed metal prices as described above and smelter terms as set forth in the IDOP Technical Report. The estimate was prepared on a simplified project analysis pre-tax basis. The Hugo North Mineral Reserve only considers Mineral Resources in the Indicated category because there are no Measured Resources in Hugo North. The engineering has been carried out to a pre-feasibility level or better to estimate the underground Mineral Reserve. To ensure that Inferred Resources do not become included in the reserve estimate, copper and gold grades on Inferred Resources within the block cave shell were set to zero and such material was assumed to be dilution. The block cave shell was defined by a \$20/t NSR. Further mine planning will examine lower shut-offs. The Hugo North Mineral Reserve is on both the OT LLC Oyu Tolgoi licence and the EJV Shivee Tolgoi licence.

Table 1.3 Oyu Tolgoi Project Mineral Reserve, 29 March 2012

Estimate	Ore (Mt)	NSR (\$/t)	Cu (%)	Au (g/t)	Recovered Metal	
					Copper (Mlb)	Gold (koz)
Southern Oyu Deposits						
Proven	125	36.42	0.58	0.91	1,374	2,892
Probable	833	18.48	0.48	0.23	7,048	4,522
Mineral Reserve (Proven + Probable)	957	20.82	0.49	0.32	8,422	7,414
Hugo Dummett Deposits						
Probable (Hugo North – OT LLC)	411	73.18	1.94	0.40	15,904	4,407
Probable (Hugo North – EJV Shivee Tolgoi)	27	79.40	1.91	0.74	1,043	536
Mineral Reserve (Probable) (All Hugo North)	438	73.57	1.93	0.42	16,947	4,943
Oyu Tolgoi Project Mineral Reserve						
Proven	125	36.42	0.58	0.91	1,374	2,892
Probable	1,270	37.46	0.98	0.30	23,995	9,465
Mineral Reserve (Proven + Probable)	1,395	37.37	0.94	0.35	25,369	12,357

Notes:

1. Metal prices used for calculating the Southern Oyu open pit NSR and the Hugo North Underground NSR are copper \$1.80/lb, gold \$750/oz, and silver \$12.00/oz based on long term metal price forecasts at the beginning of the Mineral Reserve work. Analysis indicates that the Mineral Reserve is still valid at these metal prices.
2. The NSR has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
3. For the open pit processing and general and administration operating costs have been used to determine cut-off grades are: Southwest \$4.23/t, Central Chalcocite and Central Covellite \$3.85/t and Central Chalcopyrite \$4.03/t.
4. For the underground block cave, all Mineral Resource within the shell has been converted to Mineral Reserve this includes low grade Indicated Mineral Resource and Inferred Mineral Resource assigned zero grade treated as dilution.
5. Only Measured Resources were used to report Proven Reserves and only Indicated Resources were used to report Probable Reserves.
6. The base case financial analysis has been prepared using current long term metal price estimates of copper \$2.85/lb, gold \$1200/oz, and silver \$16.60/oz. Metal prices are assumed to fall from current prices to the long term average over five years.
7. The Mineral Reserves are not additive to the Mineral Resources.
8. EJV is the Entrée Gold Joint Venture. The Shivee Tolgoi and Javkhant licences are held by Entrée Gold. The EJV Shivee Tolgoi and EJV Javkhant Licences are planned to be operated by OT LLC. OT LLC will receive 80% of cash flows after capital and operating costs for material originating below 560 m, and 70% above this depth. The Mineral Reserves are not additive to the Mineral Resources.

Table 1.4 Mineral Reserve Reconciliation IDP10 and IDOP Technical Report

Estimate	Mineral Reserve	Ore (Mt)	NSR (\$/t)	Cu (%)	Au (g/t)	Recovered Metal	
						Copper (M lb)	Gold (koz)
IDP10	Proven	127	21.38	0.58	0.93	1,399	2,994
	Probable	1,266	24.84	0.97	0.32	23,835	10,127
	Mineral Reserve	1,393	24.52	0.93	0.37	25,234	13,121
IDOP	Proven	125	36.42	0.58	0.91	1,374	2,892
	Probable	1,270	37.46	0.98	0.30	23,995	9,465
	Mineral Reserve	1,395	37.37	0.94	0.35	25,369	12,357
Difference	Proven	-2	15.04	0.00	-0.02	-25	-102
	Probable	4	12.62	0.01	-0.02	160	-662
	Mineral Reserve	2	12.85	0.01	-0.02	135	-764

Notes:

1. IDP10 Mineral Reserves have the effective date 11 May 2010.
2. IDOP Technical Report Mineral Reserves have the effective date 30 March 2012.
3. Metal prices used for calculating the Southern Oyu open pit NSR and the Hugo North Underground NSR are copper \$1.80/lb, gold \$750/oz, and silver \$12.00/oz based on long term metal price forecasts at the beginning of the Mineral Reserve work. Analysis indicates that the Mineral Reserve is still valid at these metal prices.
4. The NSR has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
5. For the open pit processing and general and administration operating costs have been used to determine cut-off grades are: Southwest \$4.23/t, Central Chalcocite and Central Covellite \$3.85/t and Central Chalcopyrite \$4.03/t
6. For the underground block cave, all Mineral Resource within the shell has been converted to Mineral Reserve this includes low grade Indicated Mineral Resource and Inferred Mineral Resource assigned zero grade treated as dilution.
7. Only Measured Resources were used to report Proven Reserves and only Indicated Resources were used to report Probable Reserves.
8. The base case financial analysis has been prepared using current long term metal price estimates of copper \$2.85/lb, gold \$1200/oz, and silver \$16.60/oz. Metal prices are assumed to fall from current prices to the long term average over five years.
9. The Mineral Reserves are not additive to the Mineral Resources.
10. EJV is the Entrée Gold Joint Venture. The Shivee Tolgoi and Javkhlant licences are held by Entrée Gold. The EJV Shivee Tolgoi and EJV Javkhlant Licences are planned to be operated by OT LLC. OT LLC will receive 80% of cash flows after capital and operating costs for material originating below 560 m, and 70% above this depth. The Mineral Reserves are not additive to the Mineral Resources.

1.6 Project and Operations Summary

Overall construction reached a 70.0% level of completion at the end of 2011. Total capital invested in the project, including exploration, initial development and the Phase 1 project by the end of 2011 was approximately \$5 billion.

Major updates since IDP10 was completed and plans for Q1 2012 include:

- Pre-stripping of overburden began in August 2011 as part of the construction of the phase-one open-pit mine to recover ore from the Southern Oyu deposits.
- The development of the first lift of the Phase 2 underground block-cave mine at the Hugo North Deposit continued successfully during 2011. Lateral mine development 1,300 metres below surface at Hugo North is on schedule and achieved an advance during 2011 of 6,460 metres, for a total of 10,241 metres completed since tunnelling started in 2008. Underground lateral development will be shut-down as planned in February 2012. Shaft 1 changeover is planned to commence in March 2012 and be complete by August 2012. Underground lateral development will recommence in September 2012.
- Shaft 2 construction is progressing well. The head frame and ancillary buildings were 81.5% complete at the end of 2011 and ahead of plan. Shaft sinking activities began in December 2011.
- Construction of the concentrator was 73% complete at the end of 2011 ahead of the plan of 69%. The assembly and alignment of SAG Mill 2 and Ball Mills 3 and 4 was completed. The primary crushing, overland conveyor and ore stockpile circuits are on track to be completed and pre-commissioned by Q2 2012. Line 1 of the concentrator is on track to be completed and pre-commissioned by 15 August 2012. Line 2 is on track to be completed and pre-commissioned by 31 October 2012. Commissioning with ore of the concentrator Line 1 circuit will be dependent on power available from China as there will be insufficient temporary power on site to undertake this activity. If power is available from China by 15 August 2012, the plan date for production of first concentrate would be achieved.
- Completion of construction and pre-commissioning of the primary crusher, overland conveyor and coarse ore stockpile circuits is expected to be in April 2012 ahead of the plan of 15 June 2012. This will be run on temporary diesel power.
- Commissioning with ore of the primary crusher, overland conveyor and coarse ore stockpile circuits is expected to begin in April 2012 as there is sufficient temporary power and the open pit pre-strip is ahead of plan to deliver ore to the primary crusher.
- Oyu Tolgoi's site-based construction workforce peaked at 14,760 in October 2011. At the end of 2011 the workforce on site has reduced to 11,500. Approximately 6,550 Mongolians were employed at the Oyu Tolgoi site, with an additional 3,500 Mongolians participating in offsite training.
- Progress of the construction of off-site facilities and infrastructure reached 66% at the end of 2011, which was behind the plan of 68%. The cumulative shortfall was due to delays with the Oyu Tolgoi–Gashuun Sukhait road to the Mongolia-China border and the Khanbumbat permanent airport, and the decision to defer until

- spring 2012 the stringing of cables on the power line to the Mongolia-China border. Facilities required for the production of the first ore are on schedule.
- Non-binding memoranda of understanding for concentrate sales to two large Chinese smelters were agreed to during Q3 2011. Contracts are expected to be finalized with these smelters over the coming months. In addition, non-binding agreements on principal sales terms have been reached with two international trading companies; conversion of these agreements to binding contracts is under discussion. Most of the concentrate initially produced at Oyu Tolgoi is expected to be delivered to customers in China.

1.6.1 Environment

Both environmental and social monitoring activities are ongoing for the Project. The revised Environmental Social Impact Assessment (ESIA) is planned for completion in April 2012 to International Finance Corporation and European Bank for Reconstruction and Development standards.

The environmental baseline assessment for the project was prepared by drawing upon the wide range of internal and independent studies that have been prepared since 2003. The existing information was reviewed and assessed for accuracy, consistency and validity. Where additional environmental baseline data became available in 2010 before the draft ESIA was produced, these were incorporated into the ESIA. Further data collection studies have been commissioned and started and commitments have been made in the corresponding management plans to ensure that collection of baseline data continues to improve and that the results of ongoing monitoring will be integrated into updated and revised management plans and procedures.

The baseline chapters presented in the ESIA are, necessarily, a summary of an extensive body of research and assessment that has been ongoing over many years.

1.6.2 Water Management

The development of a borefield to access groundwater reserves within the Gunii Hooloi aquifer basin has been established as the most cost-effective option to meet the raw water demand for the project. Water from the borefield will be required for process water supply, dust suppression in the mining areas, and potable use.

Close monitoring of aquifer behaviour, calibration of models, and monitoring of relevant surface features will become priorities as operations commence.

Entry into the region by other resource companies and increased demand due to population increases will heighten the competition and sensitivities associated with water supply.

Preliminary estimates of total site water demand for a future processing rate of 160 ktpd are 918 L/s average and 1,081 L/s peak, both figures incorporating a conservative design development contingency of 25%. These estimates assume a Phase 1 peak design processing rate of 110 kt/d at 92% plant availability, 64% tailings density, and 0% mine dewatering. Although some mine water could become available at certain stages of mining, the amount would be limited and has been conservatively excluded from design assumptions. Water consumption in the process plant represents more than 90% of the total demand. Estimates of total site water demand for the Phase 2 processing rate of 160 kt/d are 918 L/s average and 1,081 L/s peak, both figures incorporating a conservative design development contingency of 25%.

OT LLC currently has an approved GOM licence for the abstraction of 870 L/s from Gunii Hooloi and plans to seek an increase for the project expansion based on revised yield numbers that were developed after pump-testing the borefield in 2011.

The Oyu Tolgoi Environmental Management System, currently under development in accordance with ISO14001 standards, will include water conservation as one of the key areas of continuous improvement during the operations period.

1.6.3 Open Pit

Pre-stripping of overburden to gain access to ore in the Phase 1 open-pit mine began on schedule in August 2011 using the OT LLC earthworks fleet. Work began with the construction fleet hauling clay material to the tailing-storage facility (TSF) and the infrastructure projects, including roads, laydowns and access ramps. Commissioning of the initial open-pit mining fleet began in September 2011 with the release of the first Bucyrus RH340 hydraulic shovel and Komatsu 930E haul trucks. Deliveries of heavy mobile equipment, assembly and commissioning are on-going and all operational-readiness activities are on schedule. The full fleet of 28 trucks is expected to be in operation in August 2012.

At the end of 2011, 13.2 million tonnes of total material had been moved with drill/blast production at approximately 80,000 metres drilled and over 3,000 tonnes of explosives consumed for blasting.

Recruitment is ongoing with over 200 employees hired into the open-pit team including 190 Mongolian employees and 125 heavy equipment operators for the mining operations.

1.6.4 Underground Development

The development of the first lift of the Phase 2 underground block-cave at the Hugo North Deposit continued during 2011. Lateral mine development 1,300 m below surface at Hugo North achieved an advance during 2011 of 6,460 m, for a total of 10,241 m completed since tunnelling started in 2008. Underground lateral development will be shut-down as planned in February 2012 to allow for the hoist equipment upgrade at Shaft 1 which is planned to commence in March 2012 and be complete by August 2012. Underground lateral development is planned to recommence in September 2012.

The design of the underground incorporates 4 shafts and 5 raises to ventilate the mine. The intended method of construction for the 5 ventilation raises is by raise boring. This has proven difficult and has suffered delays, the primary reason for schedule slippage has been due to poor ground conditions. Completion of the ventilation raises is on the critical path to production from the underground mine as the raises are required to meet the ventilation requirements for development and provide alternative means of egress should Shaft 1 be unavailable.

In order to mitigate and minimise risk to the project production, OT LLC is investigating alternative means of establishing the vertical development required to complete the ventilation infrastructure.

Work to determine the impact of this issue is being carried out. Initial analysis by OT LLC suggests that the underground production could be delayed by around twelve months and that each month's delay could have an NPV value of \$50 M. These investigations are planned to be incorporated into DIDOP and will allow OT LLC to quantify potential costs and changes required to meet the Project schedule.

In addition possible scope changes have been indicated including:

- Additional development and excavation expenditures
- The addition of a shaft No 5
- An additional crew underground
- Increased operating cost due to design changes

1.6.5 Government and Community Relations

OT LLC intends to implement systems to manage and mitigate social and environmental impacts of the Project in cooperation with the GOM, local administration and local communities. A series of assessments and studies have been done on social and economic impacts of the Project by independent research institutions, on which Project's socio-economic management plans are based. All assessments and planning processes have included local community members, and other regional and national-level Mongolian stakeholders.

The socio-economic impacts of the Project and main risks are defined by economic and industrial growth, social sector service delivery needs, influx into the South Gobi region, public infrastructure and urban development, re-location and displacement, social aspects of environmental impacts, local administration capacity, tradition, cultural heritage, and changes of living standards.

Understanding the various socio-economic impacts and risks of Oyu Tolgoi will be the first step in developing appropriate management plans to minimize any negative developments and to maximize the many positive benefits for communities and stakeholders from Oyu Tolgoi.

1.6.6 Human Resources and Training Strategy

OT LLC intends to work together in partnership with all relevant GOM agencies and non-government organisations (NGOs) to ensure that a suitably qualified workforce is available to meet the requirements of the Project.

All staff will be covered by a comprehensive human resources policy that will be in compliance with Mongolian Labour Law.

OT LLC has made specific commitments to human resources for the Project, namely:

- OT LLC will use its best endeavours to ensure that not less than 60% of the total Project workforce consists of Mongolian nationals during the construction period and any expansion periods.
- OT LLC will use its best endeavours to ensure that not less than 90% of the total Project workforce consists of Mongolian nationals from the commencement of production.
- In direct mining operations, OT LLC will use its best endeavours to ensure that not less than 75% of mining contractors are Mongolian nationals.
- Within five years of the commencement of production, OT LLC will use its best endeavours to ensure that not less than 50% of its employed engineers are Mongolian nationals and, within 10 years of the commencement of production, that not less than 70% of its employed engineers are Mongolian nationals.
- OT LLC will establish and maintain health and safety policies, systems, and procedures at the Project to ensure a safe workplace that complies with the Law on Labour, Safety, and Health and all other applicable health and safety laws and regulations in Mongolia.

- All citizens of Mongolia employed by OT LLC under a contract will be covered by social insurance as required by Mongolian law.
- OT LLC will comply with requirements under the Mongolian Labour Law.

To facilitate meeting these commitments, OT LLC has developed numerous plans, strategies and policies related to training, recruitment, employee and labour relations, performance management, and work conditions.

As of January 2012, more than 11,000 Mongolians were working on the Project at the development site with 66 contractor companies. Approximately 360 Mongolian Nationals are currently contracted to Oyu Tolgoi as trainees and are undertaking training in four selected Mongolian technical and vocational education training (TVET) Schools and two major OEM vendor companies (Transwest and Wagner Asia) to further develop apprenticeship training. OT LLC has more than 3,300 Mongolians are participating in a special Government Employee Training Scheme, funded by Oyu Tolgoi that adds to the overall skills development pool.

1.6.7 Occupational Health, Hygiene and Safety

The Project Health and Safety Management System (H&S MS) has been developed to provide Management with clear direction on health and safety management to ensure compliance and provide the basis for driving improvements. The Project H&S MS aligns to the Rio Tinto HSEQ Management System Model, ISO 9001, 14001 and 18001 and other recognized management system standards. The Project H&S MS consists of 17 “elements” each of which describe how the Project manages each aspect of the business, with respect to health and safety, while also ensuring compliance to Rio Tinto expectations. The year end for the Project was an All Injury Frequency Rate (AIFR) of 0.67. This included all hours worked at site and in the Ulaanbaatar office. While this would normally be considered an excellent achievement, the Project did suffer one fatality in March 2009 in underground development. OT LLC continues to implement the concept of safety leadership to all Mongolian Supervisors and Managers and embedding a shared safety culture across the Project.

1.6.8 Current Exploration Activities

During 2011, Ivanhoe continued its drilling program on the Project. For the period 1 January 2011 to 1 February 2012 it completed 50,130 m of surface resource geology drilling (including geotechnical and mine-development investigation holes), 10,560 m of underground geotechnical drilling and 30,059 m of surface exploration diamond drilling. Five rigs are conducting surface exploration drilling at Oyu Tolgoi. Two of these rigs are delineating an initial resource estimate for the Heruga North Deposit, a 2.5-kilometre, mineralized extension of the Heruga Deposit, which extends northward from the southern border of the Oyu Tolgoi mining licence to the Southern Oyu deposits. Drilling in 2011 focused on the southern one-kilometre-long section of this zone, where three 300-metre-spaced sections were drilled, totalling 12,389 m.

1.7 IDOP Reserve Case Financial Summary

1.7.1 Key Assumptions

IDOP is an update of the Reserve Case previously presented in IDP10. The results of the IDOP Reserve Case show an after tax NPV₈ of US\$11.4 B. The case exhibits an after tax IRR of around 31% and a payback period of around 6 years. Capital costs of US\$6.0 B will be expended before production can commence.

Key economic assumptions in the analyses are shown in Table 1.5. Metal prices are assumed to fall from current prices to the long term average over five years.

Table 1.5 Economic Assumptions

Parameter	Financial Analysis Assumptions	
Discount Rate	8.00%	per annum
Copper Price	\$2.90 average (\$2.85 long-term)	per lb
Gold Price	\$1,272 average (\$1,200 long-term)	per oz
Silver Price	\$17.41 average (\$16.60 long-term)	per oz
Treatment Charges	\$70.00	per dmt concentrate
Copper Refining Charge	\$0.070	per lb
Gold Refining Charge	\$5.00	per oz

1.7.2 Investment Agreement and Taxation Assumptions

Both the process of negotiation and the final agreement of the IA presented an opportunity to confirm how the laws of Mongolia should be interpreted in their application to the Project and provided for some specific terms to apply to the Oyu Tolgoi Project. For OT LLC, the agreement has provided the confidence in the stability of the terms the Project will operate under and reliably assess its intended investment in the Project. The agreement itself is effective for an initial term of 30 years with an available extension of a further 20 years.

In accordance with the requirements outlined in the 2006 Minerals Law of Mongolia, upon execution of the IA and the fulfillment of all conditions precedent, the GOM has become a 34% shareholder in OT LLC through the immediate issue of OT LLC's common shares to a shareholding company owned by the GOM. Upon a successful renewal of the IA after the initial 30 year term, the GOM also has the option to increase its shareholding to 50%, under terms to be agreed with Ivanhoe Mines at the time.

A number of conditions precedent were set down in October 2009 and were required to be met before the IA terms came into effect. These were met and confirmed by the GOM in March 2010, triggering the issue of the GOM's equity share in the Project and bringing the IA into full effect.

Despite its role as an equity owner, the GOM will not be required to contribute to the initial capital cost of the Project. Ivanhoe, as the parent company, retains the right to

fund the Project by way of a combination of debt, redeemable preference shares and common shares provided the debt to equity ratios fall within the 3:1 ratio required by Mongolian law.

In the case of shareholder debt, loans (including existing shareholder loans at the time of the agreement) initially attracted an interest rate of 9.9% (real) per annum with corresponding adjustments to be made to the outstanding balance to reflect increases in US CPI during each period. The coupon rate applicable to redeemable preference shares was to be 9.9% (real) and carry the same escalation terms. All principal and interest outstanding on shareholder debt, outstanding coupon payments on redeemable preference shares and the face value of those redeemable preference shares must be paid in full prior to any dividends on common shares being paid.

In 2011 an Amended Shareholders Agreement was concluded which reduced the applicable rate from 9.9% to LIBOR plus 6.5%. In addition, an in principle agreement was reached to convert the balance of preference shares into ordinary shares. Both adjustments were to take place based on the 31st January 2011 balances, although the preference share conversion had not occurred by 31st December 2011.

Under the authority of the Shareholders Agreement, Ivanhoe has the right to act as or appoint a Management Team to oversee the construction and operation of Oyu Tolgoi. The management team is compensated with a Management Services Payment equal to 3% of total operating and capital costs prior to commencement of operations and 6% of operating and capital costs during operations. This payment is included in the economic analysis as a project expense and is confirmed as tax deductible in the IA.

OT LLC is required to achieve commencement of production within 7 years of the effective date of the IA.

Under the terms of the IA, a range of key taxes has been identified as stabilized for the term of the agreement at the rates and base currently applied. The following taxes comprise the majority of taxes and fees payable to the GOM under Mongolian law and are shown with their stabilized rates:

- Corporate income tax 25%
- Royalties 5% (gross sales value)
- Value added tax 10%
- Customs duties 5%

In accordance with the Excess Profits Tax invalidating law, as from 1 January 2011, the taxpayer will not be subject to the excess profits tax or any similar windfall tax.

OT LLC is also only subject to those taxes currently listed in the General Taxation Law and not taxes introduced at any future date. These taxes are collectively noted as non-discriminatory taxes and as such cannot be imposed on OT LLC in any manner other than that applied to all taxpayers. OT LLC may also apply to take advantage of any future law or treaty that comes into force and which would apply any rates lower than those specified in the IA.

The GOM recently enacted amendments to the legislation governing the carry forward of income tax losses. The loss carry forward period has been extended to 8 years and if sufficient, can be applied to offset 100% of taxable income. This contrasts with the previous law in which losses carried forward for 2 years and were subject to a 50% limit.

The agreement also provides OT LLC with the benefit of a 10% tax credit for all capital investment made during the construction period. The amount of this credit can be carried forward and credited in the three subsequent profitable tax years. It is noted in the agreement that if Mongolian Value Added Tax (VAT) payments, which are currently non-refundable, become refundable in the future, the availability of the investment tax credit will cease from that point. In that event, past earned investment tax credits will still be applied.

1.7.3 Operating Assumptions

Although it has a requirement to make its self discovered water resources available to be used for household purposes, it is confirmed in the agreement that OT LLC holds the sole rights to use these water resources for the Project. The contract for the utilization of water with the GOM water authority is in effect for 30 years with subsequent 20 year periods of renewal.

The supply of power has been recognized as being critical to the execution of the Oyu Tolgoi Project in the IA. OT LLC has been given the right to import power initially but must secure power from sources within Mongolia from the fourth year of operation.

OT LLC also has the right to construct roads for the transport of its product and airport facilities to suit the Project's needs. The GOM has committed to providing OT LLC with non-discriminatory access to any railway constructed between Mongolia and China if such a railway is constructed.

1.7.4 Project Results

A summary of the IDOP Reserve Case project financial results is shown in Table 1.6.

The estimates of cash flows of the Project have been prepared on a real basis based at 1 January 2012 and discounted back to a current day Net Present Value (NPV) at a rate of 8%. Metal prices used for the analysis are copper \$2.90/lb (\$2.85/lb long-term), gold \$1,272/oz (\$1,200/oz long-term) and silver \$17.41/oz (\$16.60/oz long-term).

Table 1.6 IDOP Reserve Case Financial Results

		Before Taxation	After Taxation
Net Present Value (US\$M)	Undiscounted	41,080	33,105
	5.0%	20,435	16,773
	6.0%	17,893	14,730
	7.0%	15,693	12,953
	8.0%	13,783	11,403
	9.0%	12,119	10,046
	10.0%	10,665	8,856
Internal Rate of Return	–	32%	31%
Project Payback Period (Years)	–	6.14	6.14

Mine site cash costs are shown in Table 1.7. Those costs relating to the direct operating costs of the mine site, namely:

- Mining
- Concentration
- Tailings
- General and administrative (G&A) costs
- Government fees and charges (excluding corporate taxation)
- Management fees

In addition, realization costs are shown which indicate the actual realizable value of Payable Copper produced after accounting for the transport treatment and GOM royalties payable on these sales.

Table 1.7 IDOP Reserve Case Unit Operating Costs by Copper Production

	\$/lb Payable Copper		
	IDOP Reserve Case	First 5 Years	First 10 Years
Mine Site Cash Cost	1.08	2.17	1.05
TC/RC, Royalties and Transport Cash Cost	.40	0.58	0.47
Total Cash Costs Before Gold Credits	1.48	2.75	1.52
Gold Credits	(0.62)	(1.73)	(0.81)
Silver Credits	(0.05)	(0.08)	(0.05)
Total Cash Costs After Credits	0.81	0.93	0.65

Notes: Payments made directly to Ivanhoe from the Project, principally the management fee as specified in the terms of the IA are included in the Mine Site Cash Cost per pound of payable copper. These payments due to Ivanhoe average around 0.10 \$/lb.

The revenues and operating costs have been presented in Table 1.8, along with the net sales revenue value attributable to each key period of operation.

Table 1.8 IDOP Reserve Case Operating Costs and Revenues

	US\$M	\$/t Ore Milled		
	Total IDOP Reserve Case	First 5 Years	First 10 Years	IDOP Reserve Case
Revenue				
Gross Sales Revenue	87,105	65.71	78.41	62.33
Less: Realization Costs				
Concentrate Transport	960	0.45	0.72	0.69
Treatment and Refining	4,453	2.39	3.55	3.19
Government Royalty	4,355	3.29	3.92	3.12
Total Realization Costs	9,768	6.12	8.19	6.99
Net Sales Revenue	77,337	59.58	70.22	55.34
Less: Site Operating Costs				
Mining (all sources)	7,531	6.69	6.46	5.39
Processing	6,311	6.90	5.20	4.52
Tailings	922	0.92	0.79	0.66
G&A	6,738	8.39	6.25	4.82
Entrée JV Fees	539	0.02	0.02	0.39
Government Fees and Charges	2,495	2.84	2.22	1.79
Management Fees	1,789	3.11	1.90	1.28
Total Site Operating Costs	26,325	28.86	22.82	18.84
Operating Margin	51,012	30.72	47.40	36.50

The Phase 1 direct capital investment program since project inception is depicted in Table 1.9 as the capital cost required prior to reaching commercial production at the end of 2012 (pre-production). The total cash cost estimated to be expended at that point is estimated at \$US6.0 B which includes \$1.0 B invested in underground development.

Total project direct capital costs are shown in Table 1.10 divided between the investment required to commence production at 100 ktpd and complete Shaft No 2 (Phase 1), the concentrator expansion to 160 ktpd, power station and completion of the underground mine to 54 ktpd (Phase 2), and the sustaining capital to support ongoing operations.

The Phase 1 capital shown in Table 1.9 and Table 1.10 include only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

The changes in financial results for a range of copper and gold prices are shown in Table 1.11.

Table 1.9 Pre-production Capital Cost

US\$M	Phase 1		Phase 2	Total
	2000 to 2011	Costs Remaining		
Mining				
Southern Oyu	308	201	–	509
Hugo South	–	–	–	–
Hugo North Lift 1	548	299	184	1,032
Hugo North Lift 2	–	–	–	–
Heruga	–	–	–	–
Subtotal	855	501	184	1,541
Processing and Infrastructure				
Concentrator	950	260	–	1,210
Infrastructure	945	240	–	1,186
Power Station	17	–	–	17
Tailings Storage Facility (TSF)	11	27	–	38
Subtotal	1,924	528	–	2,452
Indirects & EPCM				
Construction Indirects/Site Services	430	73	–	503
Project Management	332	134	–	466
Subtotal	762	207	–	969
Construction & Operations				
Construction Owner’s Team	137	120	–	256
Operations Owner’s Team	287	486	–	773
Subtotal	424	605	–	1,029
Mongolian Government Payments				
Mongolian Customs Duties	–	–	5	5
Mongolian VAT	–	–	14	14
Subtotal	–	–	19	19
Total Development Program	3,966	1,841	204	6,010

Notes: Phase 1 capital includes only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

Table 1.10 Total Project Capital Cost– IDOP Reserve Case

US\$M	Phase 1 100 ktpd	Expansion 160 ktpd	Sustaining	Total
Mining				
Southern Oyu	484	–	490	974
Hugo South	0	0	0	0
Hugo North Lift 1	944	2,828	830	4,601
Hugo North Lift 2	0	0	0	0
Heruga	0	0	0	0
Subtotal	1,428	2,828	1,319	5,575
Processing and Infrastructure				
Concentrator	1,310	645	121	2,076
Infrastructure	1,404	–	129	1,534
Power Station	17	1,156	96	1,270
Tailings Storage Facility (TSF)	37	–	50	86
Subtotal	2,768	1,801	397	4,966
Indirects & EPCM				
Construction Indirects/Site Services	433	–	–	433
Project Management	434	–	–	434
Subtotal	868	–	–	868
Construction & Operations				
Construction Owner's Team	305	–	–	305
Operations Owner's Team	828	–	–	828
Subtotal	1,133	–	–	1,133
Mongolian Government Payments				
Mongolian Customs Duties	–	127	43	170
Mongolian VAT	–	360	133	493
Subtotal	–	487	176	663
Total Development Program	6,197	5,115	1,892	13,204

Notes: Phase 1 capital includes only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

Table 1.11 After Tax Metal Price Sensitivity – IDOP Reserve Case

After Tax Values	Gold (\$/oz)							
Copper (\$/lb)	800	1,000	1,200	1,500	1,750	2,000	2,500	3,000
Project Net Present Value at 8% (\$M)								
2.00	4,998	5,563	6,169	7,085	7,779	8,441	9,750	11,053
2.50	8,266	8,791	9,310	10,103	10,765	11,418	12,719	14,031
2.85	10,350	10,875	11,403	12,188	12,840	13,496	14,808	16,114
3.00	11,246	11,769	12,293	13,079	13,736	14,392	15,702	17,004
3.50	14,219	14,748	15,277	16,065	16,716	17,367	18,668	19,977
4.00	17,206	17,729	18,250	19,031	19,682	20,338	21,650	22,959
4.50	20,175	20,699	21,224	22,011	22,667	23,322	24,627	25,932
5.00	23,159	23,684	24,207	24,990	25,643	26,296	27,599	28,903
Project Internal Rate of Return (IRR%) After Tax								
2.00	21%	22%	23%	25%	26%	27%	29%	31%
2.50	26%	27%	28%	29%	30%	31%	33%	35%
2.85	29%	30%	31%	32%	33%	34%	36%	37%
3.00	30%	31%	32%	33%	34%	35%	37%	38%
3.50	34%	35%	35%	36%	37%	38%	40%	41%
4.00	37%	38%	39%	39%	40%	41%	42%	44%
4.50	40%	41%	41%	42%	43%	44%	45%	46%
5.00	43%	43%	44%	45%	45%	46%	47%	48%
Project Payback (Years)After Tax								
2.00	7.48	7.31	7.16	6.95	6.74	6.55	6.25	6.00
2.50	6.76	6.61	6.48	6.31	6.18	6.06	5.79	5.53
2.85	6.35	6.25	6.15	6.02	5.88	5.75	5.51	5.31
3.00	6.21	6.12	6.04	5.88	5.75	5.63	5.41	5.23
3.50	5.81	5.72	5.63	5.51	5.41	5.32	5.16	5.03
4.00	5.48	5.41	5.34	5.25	5.18	5.11	4.98	4.75
4.50	5.25	5.19	5.14	5.07	5.01	4.92	4.72	4.55
5.00	5.08	5.03	4.98	4.86	4.77	4.69	4.53	4.39
Total Cash Costs (after Credits) \$/lb – Life of Project								
2.00	0.90	0.82	0.74	0.63	0.53	0.43	0.24	0.04
2.50	0.94	0.86	0.78	0.66	0.57	0.47	0.28	0.08
2.85	0.96	0.88	0.81	0.69	0.59	0.50	0.30	0.11
3.00	0.97	0.90	0.82	0.70	0.60	0.51	0.31	0.12
3.50	1.01	0.93	0.86	0.74	0.64	0.54	0.35	0.16
4.00	1.05	0.97	0.89	0.78	0.68	0.58	0.39	0.19
4.50	1.09	1.01	0.93	0.81	0.72	0.62	0.43	0.23
5.00	1.12	1.05	0.97	0.85	0.75	0.66	0.46	0.27

1.8 Future Work

1.8.1 Detailed Integrated Development and Operating Plan

The next phase in the project planning process is the preparation of DIDOP that builds on IDOP. The DIDOP will integrate a number of detailed capital expansion studies into an overall project report. The key additional work to be assessed in DIDOP includes: the Phase 2 Project Expansion, infrastructure, power supply, water permitting, concentrate marketing, the underground feasibility study and further work on mine closure and reclamation plan.

1.8.2 Phase 2 Project Expansion Plan

A detailed execution plan will be developed for Phase 2 that includes lessons learned and incorporates tools and advancements from the Phase 1 project execution.

OT LLC plans to undertake engineering studies of the concentrator expansion. The focus of the study will be the addition of a third grinding line and other requirements for the Phase 2 plant production.

1.8.3 Power Supply Determination

The supply of power has been recognized as being critical to the execution of the Oyu Tolgoi Project in the IA. OT LLC has been given the right to import power initially but must secure power from sources within Mongolia from the fourth year of operation.

Power is currently assumed in the IDOP Technical Report to be initially imported from Inner Mongolia and sourced from a power station constructed by OT LLC by the fourth year of operation in accordance with the terms of the Investment Agreement.

OT LLC is currently considering a range of options to ensure a reliable and efficient power supply after Year 4. Central to their considerations are two options:

- Option 1 – Build a coal-fired power plant (four 150 MW units) at site. A feasibility study and detailed drawings for power plant with three 150 MW units capacity have already been completed. This study will need to be re-worked to reflect the addition of another generator and auxiliaries.
- Option 2 – Build a coal-fired power plant (four 150 MW units) at the Tavan Tolgoi project site and erect a double-circuit 220 kV transmission line to the Project site. With the power plant at this location, the feasibility study would need to address changes in capacity, water supply, and other design issues, as well the 140 km long 220 kV transmission line.

OT LLC's preferred option is to pursue the accelerated construction of a power station at OT LLC with construction to commence as soon as all stakeholders agree. OT LLC intends to seek approval from the board for an accelerated program if GOM permits are approved.

Currently, there is no clear date for the issuing of those permits and the IDOP Technical Report remains with the assumption that capital is expended by OT LLC to bring a power station into production by the fourth year of operation.

It is expected that the timing of power station construction would be finalised in the preparation of DIDOP.

1.8.4 Water Permit

The current estimate of average water demand for the concentrator expansion to 160 kt/d is 918 L/s, which is marginally above the rate of 870 L/s that has already been approved by the GOM.

OT LLC's strategy is to obtain approval for increases to the currently approved water reserve ahead of any mine expansion plans. The objective of the study will be to assess the impact of the concentrator expansion on water demand and to determine the need for obtaining GOM approval for any substantial increase in the approved water demand from the Gunii Hooloi aquifer.

1.8.5 Concentrate Marketing

OT LLC has developed a marketing plan and currently includes consideration of the following factors:

- Location value to customer compared to imported material landed at Chinese ports.
- Precious metals recovery and payment.
- Length of contract.
- Percentage of off-take to smelters versus traders.
- Percentage of tonnage on contract versus spot.
- Percentage of feed for any one smelter.
- Number of smelters for a given scale of operation.
- Management of concentrate quality and volume during commissioning and ramp-up.
- Alternate off-shore logistics and costs.
- Delivery point and terms.
- Packaging.

A detailed timeline has been developed for marketing, logistics, and contract-to-cash functions. Non-binding memoranda of understanding for concentrate sales to two large Chinese smelters were agreed to during Q3 2011. Contracts are expected to be finalized with these smelters over the coming months. In addition, non-binding agreements on principal sales terms have been reached with two international trading companies; conversion of these agreements to binding contracts is under discussion. Most of the concentrate initially produced at Oyu Tolgoi is expected to be delivered to customers in China. OT LLC's Sales and Marketing will be supported by Rio Tinto

Copper Marketing, led by its Chief Marketing Officer. The marketing team will oversee and execute all sales and marketing activities on behalf of OT LLC.

1.8.6 Underground Feasibility Study

As part of DIDOP the underground study work is being brought to feasibility study level, the scope of work includes all activities from extraction to the shaft collars and will optimise the development and production of Hugo North Lift 1. OT LLC plans to finalise DIDOP in late 2012.

The design of the underground incorporates 4 shafts and 5 raises to ventilate the mine. The intended method of construction for the 5 ventilation raises is by raise boring. This has proven difficult and has suffered delays, the primary reason for schedule slippage has been due to poor ground conditions. Completion of the ventilation raises is on the critical path to production from the underground mine as the raises are required to meet the ventilation requirements for development and provide alternative means of egress should Shaft 1 be unavailable. In order to mitigate and minimise risk to the project production, OT LLC is investigating alternative means of establishing the vertical development required to complete the ventilation infrastructure.

Work to determine the impact of this issue is being carried out. Initial analysis by OT LLC suggests that the underground production could be delayed by around twelve months and that each month's delay could have an NPV value of \$50 M. These investigations are planned to be incorporated into DIDOP and will allow OT LLC to quantify potential costs and changes required to meet the Project schedule.

In addition possible scope changes have been indicated including:

- Additional development and excavation expenditures
- The addition of a shaft No 5
- An additional crew underground
- Increased operating cost due to design changes

1.8.7 Socio-economic Aspects of Mine Closure Plan

The preliminary mine closure and reclamation plan includes provisions to ensure that adverse socio-economic impacts of mine closure are minimized and positive impacts are maximized. To this end, OT LLC has planned that allowances will be incorporated into the annual mine operations budget starting 10 years before mine closure to address the costs of:

- Lost employment by the mine workforce.
- Adverse effects on supply chain businesses and downstream businesses, affected communities, public services, and infrastructure.
- Promoting ongoing sustainability among affected stakeholders and communities.

The details of additional socio-economic aspects of a conceptual mine closure plan have not yet been fully developed and are the subject of work to be done in the near future.

1.8.8 Infrastructure

As part of the preparation of DIDOP, work is continuing on further defining the ongoing infrastructure needs for the project. OT LLC has advised that it expects future additions to the project scope which may include:

- Operations camp expansion
- Border facilities upgrade
- Concentrate bagging plant upgrade
- Power substation expansions
- Central maintenance complex
- Central control room
- Borefield expansion
- Operations warehouse expansion
- Core storage warehouse

There may be additions to scope beyond these items and all items and updated cost estimates will be included in DIDOP.

2 INTRODUCTION

2.1 Issuer for Whom Report Prepared

This report is titled the IDOP Technical Report and has been prepared for Ivanhoe.

2.1.1 Ownership/Joint Venture

The project comprises 5 contiguous properties; three held by 100% by Oyu Tolgoi (6708A, 6709A, and 6710A) and two held by Entrée Gold LLC in bare trust for the JV between OT and Entrée (Shivee Tolgoi and Javkhlant). These are further described in Sections 4.2.1 and 4.2.2, respectively. The vast majority of the identified mineralization on the Project occurs within the Oyu Tolgoi Property at the Hugo Dummett and Southern Oyu Porphyry Deposits. Only the northernmost extension of the Hugo Dummett Deposit (Hugo North) crosses onto the Shivee Tolgoi Joint Venture Property and the majority of Heruga mineralisation occurs on the Javkhlant property (Entrée Gold Inc. refers to the Hugo North Deposit within its property limits, as the Copper Flats Deposit). AMC has relied on OT LLC for the description of the Mineral Tenure in this section.

In October 2009, Ivanhoe Mines, OT LLC, and Rio Tinto PLC signed an Investment Agreement (IA) with the GOM. The IA defines the fiscal and regulatory environment the Project will operate in and brings the GOM (GOM) into the Project as a 34% equity owner of OT LLC with the option to increase their equity holding by a further 16% after 30 years providing terms can be reached with Ivanhoe Mines at that time. At the time of signing, there remained a number of conditions precedent to the agreement becoming effective. On 31 March 2010 Ivanhoe Mines announced that the conditions precedent to the Investment Agreement had been met and confirmed by the GOM.

Ivanhoe initially acquired 100% interest in the mineral exploration rights to this Property on 5 May 2000, through an earn-in agreement with BHP on a mineral exploration licence (MEL) covering 1,120 km² (112,000 ha). This MEL named "210" was originally issued to BHP on 17 February 1997 and was later re-named MEL 66X. In September 2000, the licence was reduced in size and converted to four non-contiguous MELs, one of which represented the Oyu Tolgoi Property (the other MELs are not considered in this report). The lease boundaries along with the surface projection of the main deposits and proposed open pit outlines are shown in Figure 2.1.

The terms of this original agreement called for Ivanhoe to spend a total of US\$6 M on exploration over 7 years and to pay BHP US\$5 M (Ivanhoe press release, 8 May 2000). Upon transfer of the MEL, Ivanhoe agreed to a 2% net smelter return (NSR) based payment payable to BHP. The terms of the BHP agreement were met by Ivanhoe in April 2002, giving it 100% interest in the Oyu Tolgoi Property. The 2% NSR was subsequently acquired by Ivanhoe in February 2004 (two payments totalling US\$37 M), thereby removing any future obligations to BHP Billiton. (On 29 June 2001, BHP merged with Billiton resulting in a company name change to BHP Billiton).

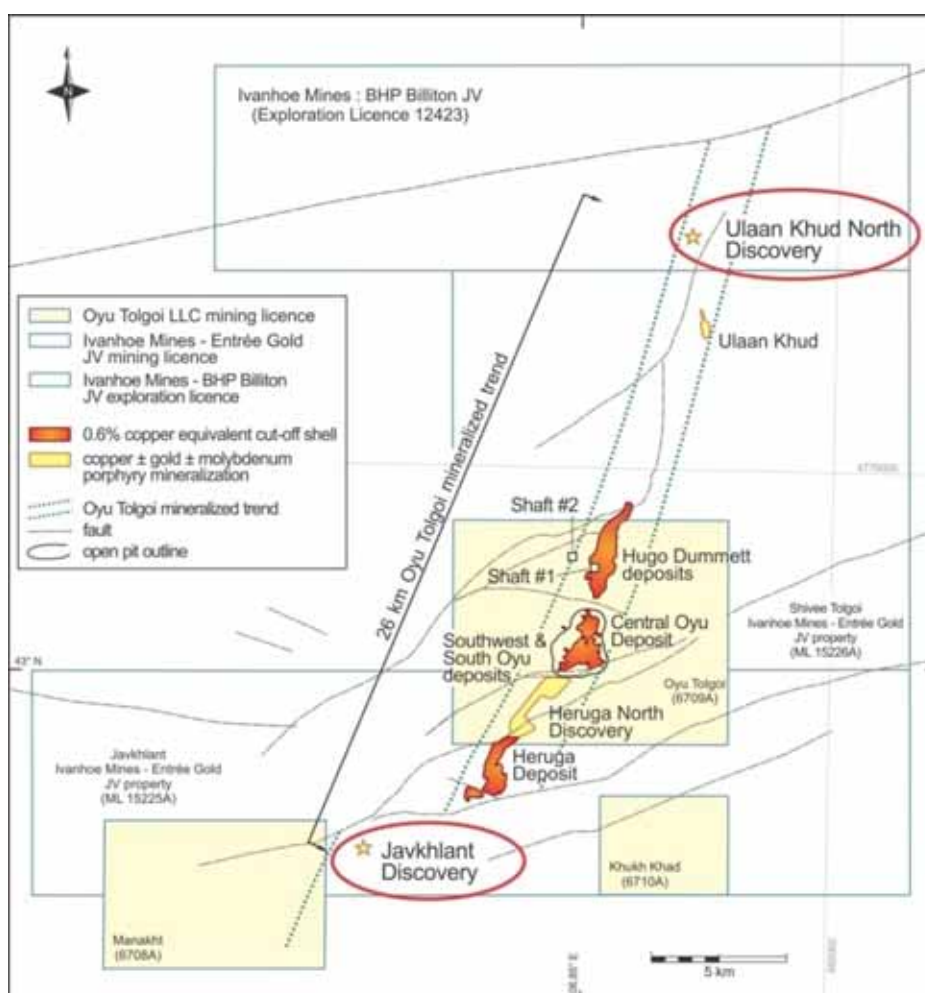
On 23 December 2003, Ivanhoe was granted 100% interest in a Mining Licence for the Oyu Tolgoi Property (Mining Licence 6709A, covering 8,496 ha; Figure 2.1), in accordance with the Minerals Law of Mongolia. When originally granted, the OT Mining Licence had a term of 60 years, with an option to extend the licence for an additional

term of up to 40 years. In 2006, the Mongolian parliament passed new mining legislation that changed the term of mining licences to 30 years with two 20 year extensions.

Given the extent of the mineral discoveries associated with the Project and the potential for additional discoveries, Ivanhoe Mines and the GOM agreed that the approved IA should conform with the provision of Mongolia's current Minerals Law specifying that certain deposits of strategic importance qualify for 30 years of stabilized tax rates and regulatory provisions, with an option of extending the term of the IA for an additional 20 years.

Mineral production from the Property is subject to a 5% royalty on the sales value of all products, except coal and common minerals, extracted from the mining licence that are sold, shipped for sale, or used. Royalties are payable to the GOM and are governed by Article 38 of the Minerals Law of Mongolia.

Figure 2.1 Oyu Tolgoi Project Land Tenure



The boundary coordinates of the Mining Licence are defined on Table 2.1 by latitude/longitude and also by UTM coordinates (WGS-84, Zone 48N).

Table 2.1 Oyu Tolgoi Property (Mining Licence 6709A) Boundary Coordinates

Point	Latitude/Longitude, MSK42		UTM, WGS-84, Zone48N	
	Latitude	Longitude	Northing	Easting
1	42 58' 30" N	106 47' 30" E	4759595.990	646096.713
2	43 03' 00" N	106 47' 30" E	4767924.920	645918.987
3	43 03' 00" N	106 55' 00" E	4768149.900	656099.637
4	42 58' 30" N	106 55' 00" E	4759820.928	656289.773

Note: Point 1 corresponds with the NW corner of the Licence, as shown on Figure 2.1, and then moving clockwise.

Payments to maintain mining and exploration licences in Mongolia are payable in advance on an annual basis according to the schedule shown on Table 2.2. An exploration licence is valid for a 3 year period with two 3 year extensions for a total of 9 years. Prior to expiry of the exploration licence, application can be made for conversion to a mining licence. The current licences can be maintained for another 3 year extension, referred to as the pre-mining period, by making application and by paying annual fees of US\$1.50/ha during this extension period as defined in Article 32.2.

Table 2.2 Exploration and Mining Licence Annual Maintenance Costs

Years of Licence	Exploration Licence Cost Per Hectare (US\$)	Mining Licence Cost Per Hectare (US\$)
1	0.10	15.00
2	0.20	15.00
3	0.30	15.00
4	1.00	15.00
5	1.00	15.00
6	1.00	15.00
7	1.50	15.00
8	1.50	15.00
9	1.50	15.00*

*Beyond Year 9, the mining licence fee remains at US\$15.00/ha.

The approximate exploration and mining licence fees paid to date by Ivanhoe for the Oyu Tolgoi Property are shown on Table 2.3. Ivanhoe advises that all payments have been made to keep the licence in good standing at the effective date of the Technical Report.

Table 2.3 Exploration and Mining Licence Fees (2000 to 2008)

Mining/Exploration Licence	Year	Licence Fees (US\$/ha)	Total Payments (US\$)
Exploration Licence	2000	1.00	8,496
	2001	1.00	8,496

Mining/Exploration Licence	Year	Licence Fees (US\$/ha)	Total Payments (US\$)
	2002	1.50	12,744
	2003	1.50	12,744
Mining Licence 6709A (Oyu Tolgoi)	2004	5.00	42,480
	2005	5.00	42,480
	2006	5.00	42,480
	2007	15.00	127,440
	2008	15.00	127,440
Total Fees Paid			424,800

The annual payment for 2009 onwards will be approximately US\$127,440 for the Oyu Tolgoi Mining Licence (8,496 ha), and US\$597,960 for the Shivee Tolgoi JV Mining Licence (39,864 ha).

OT is also contractually obliged to meet payments on the Javkhlant JV Mining licence (54,653 ha; \$819,789) as well as ML6708 (4,533 ha; \$68,000) and ML6710 (1,763; \$26,441) for a total \$1.35 M for 2012.

The Oyu Tolgoi Property was legally surveyed in August 2002 by Surtech International Ltd. using the internationally recognized survey datum WGS–84 Zone 48N. In September 2004, Geomaster Co. Ltd. (Geomaster), a licensed Mongolian land survey company, re-surveyed the mining concession's corner points based on the official Mongolian survey datum "MSK42" and marked the corners with concrete and steel pylons. In November 2004, Geomaster also surveyed the northern boundary between the Oyu Tolgoi Property and the adjacent Shivee Tolgoi JV Property (see Section 4.2.2) and marked it with wooden posts on 250 to 500 m intervals. Ivanhoe advised that the differences between these two surveys are considered insignificant.

The Minerals Council is a group of Mongolian national experts appointed from industry, academia, and government by the GOM to review and recommend on the Mongolian Commercial Minerals registration. Commercial Minerals include Mongolian Mineral Resources and Mongolian Mineral Reserves. Commercial Minerals can only be reported in Mongolia by registration by the Ministry of Mineral Resources and Energy. On 17 May 2006, an Experts Group that had been established by the GOM recommended registration of the open pit reserves under Mongolian guidelines, as part of the conclusions the Experts Group confirmed the IMMI title to the Oyu Tolgoi licence 6709A. On 1 July 2009, a new Experts Group, the Minerals Council recommended to the Ministry of Mineral Resources and Energy and Mineral Resources Authority of Mongolia that the Oyu Tolgoi Commercial Minerals be registered. Included in the recommendations was an acknowledgment of the licences and a recommendation that the Shivee Tolgoi 3148X and Javkhlant 3150X exploration licences be converted to mining licences, which now has been done as follows – Shivee Tolgoi 15226A and Javkhlant 15225A. The IA describes the exploration and mining licences relating to the Project and confirms their ownership by OT LLC. It is considered from this that, under the guidelines for NI 43-101, it is reasonable to conclude that Ivanhoe and Entrée have title of the licences.

2.1.2 Rio Tinto Agreement

On 18 October 2006, Ivanhoe entered into a private placement agreement (the "Agreement") with Rio Tinto International Holdings Limited (Rio Tinto), a wholly owned subsidiary of Rio Tinto PLC. Under the terms of the Agreement, Rio Tinto acquired 37.1 M or 9.95% of the issued and outstanding common shares of Ivanhoe at US\$8.18 per common share for an aggregate subscription price of approximately US\$303.4 M. In October 2009, Rio Tinto executed Tranche 2 of their investment and acquired a further 46.3 M common shares, or 9.95% of Ivanhoe at US\$8.38 per common share for an aggregate subscription price of approximately US\$388 M. The conclusion of this second tranche of the private placement completed one of the conditions precedent to the IA.

These documents can be accessed at www.sedar.com.

2.2 Terms of Reference and Purpose of Report

The Oyu Tolgoi copper and gold project (the "Project") is located in the southern Gobi region of Mongolia and is being developed by Oyu Tolgoi LLC (OT LLC). OT LLC was formerly called Ivanhoe Mines Mongolia Inc XXK (IMMI). Oyu Tolgoi consists of a series of deposits containing copper, gold, silver and molybdenum. The deposits stretch over 12 km, from the Hugo North deposit in the North through the adjacent Hugo South, down to the Southern Oyu deposit and extending to the Heruga deposit in the South.

This report, the IDOP Technical Report, was prepared for Ivanhoe. The IDOP Technical Report is based on the technical, production and cost information contained in the OT LLC study titled Integrated Development and Operations Plan (IDOP). IDOP was completed by the Rio Tinto appointed management of OT LLC in March 2011 as the basis for the proposed Oyu Tolgoi Project financing.

Independent Qualified Persons (QP), acting on behalf of Ivanhoe, reviewed the IDOP as part of the preparation for the IDOP Technical Report and in conjunction with Ivanhoe prepared the IDOP Technical Report with costs to the end of 2011 as reported to Ivanhoe by OT LLC. The IDOP Technical Report represents a view of the Project by Ivanhoe and the QPs, and is not the view of the Rio Tinto appointed management of OT LLC or Rio Tinto.

IDOP used only Measured and Indicated Mineral Resources and is a complete study of all aspects of the Project. The IDOP Technical Report analyses a reserve case only (IDOP Reserve Case) and is based on a pre-feasibility quality level study complying with NI 43-101, although some parts of the Oyu Tolgoi Project are further advanced and are at feasibility level. The work of the IDOP Technical Report meets the standards of US SEC Industry Guide 7 requirements for reporting Reserves.

2.3 Units of Measure and Currency

Throughout this Report, measurements are in metric units and currency in United States dollars unless otherwise stated.

2.4 Sources of Information and Study Participants

This report was compiled by the Qualified Persons listed on the Title Page. Original authors and companies are listed throughout the text.

2.5 Personal Site Inspections

The following site visits were carried out by the Qualified Persons:

- Bernard Peters visited the property in March 2003, July 2003, April 2006, April 2009, July 2010 and October 2011. Meetings were also attended in Ulaanbataar with OT LLC (formerly IMMI) and Mongolian authorities to discuss the Project from 2003 to 2011. Some of these meetings did not include site visits. Other visits were made to Mongolia and China as part of work on the Project.
- Scott Jackson visited the property from 25 November 2011 to 28 November 2011, 26 July 2010 to 30 July 2010, 20 February 2008 to 22 February 2008 and 2 September 2008 to 12 September 2008.
- Albert Chance visited the property from 15 June 2009 to 19 June 2009 and from September 14 to 16, 2010.
- Jarek Jakubec visited the property in February 2003, June 2004, October 2004, July 2007 and April 2009.
- Dean David visited the property from 1 June 2009 to 5 June 2009. Whilst on site, inspection was made of the core library, site preparations for mining and milling facilities, geological facilities, administration facilities local infrastructure, and site accommodation.

3 RELIANCE ON OTHER EXPERTS

The authors of this report state that they are Qualified Persons for those areas as identified in the appropriate "Certificate of Qualified Person" attached to this report. The authors have relied upon, and believe there is a reasonable basis for this reliance, the following experts and reports have contributed information regarding legal, land tenure, corporate structure, permitting, environmental, and other issues in portions of this Technical Report in the Sections as noted below.

Reports used in Section 4, Property Description and Location:

The following reports and documents affirm the ownership of the explorations and mining licences related to Oyu Tolgoi.

- Ministry of Mineral Resources and Energy, Mineral Resources Authority of Mongolia, Conclusion Of The Minerals Council, 1 July 2009. The Minerals Council is a group of Experts appointed by the GOM to review and recommend on the Mongolian Commercial Minerals registration.
- Investment Agreement. The IA describes the licence areas and confirms the ownership by OT LLC.
- Ivanhoe Mines Ltd. Annual Information Form (AIF) For the year ended 31 December, 2011 Dated 19 March, 2012. The AIF describes the circumstances and potential impact of the GOM Resolution 175.
- Integrated Development and Operations Plan, (IDOP)OT LLC April 2011.
- The information on approval status of the Detailed Environmental Impact Assessment Report was provided by OT LLC (email 23 March 2012, OT LLC National Compliance).

Reports used in Section 5, Accessibility, Climate, Local Resources, Infrastructure, and Physiography:

- Ivanhoe Mines Mongolia Inc. Oyu Tolgoi Project, Project No. A2MW Process Design Criteria Section:0.05 Site Conditions. Fluor Canada.
- The information in Section 20 Environment was prepared by OT LLC and in IDOP.

Reports used in Section 17, Mineral Processing and Metallurgical Testing:

- JKTech, October to December 2004. Drop-Weight Test Report on Twenty Six Samples from Oyu Tolgoi Deposit, Tested at AMMTEC Pty Limited, Perth, Western Australia for Ivanhoe Mines Ltd. JKTech Job No. 04001.
- AMMTEC Ltd., April 2005. Metallurgical Testwork for the Oyu Tolgoi Copper Gold Project Conducted for Ivanhoe Mines Mongolia. Report No. A9141.
- Metso Minerals Industries, Inc., 8 April 2005. Copper Concentrate (South West Rougher Concentrate) – Jar Mill Test. Test Plant Report No. 65045.
- MinnovEX Technologies Inc., October 2002. Preliminary Grinding Circuit Design for the Project, Mongolia.

- MinnovEX Technologies Inc., 26 September 2003. Review of Grinding Design for Oyu Tolgoi and Indicative CEET Assessment.
- MinnovEX Technologies Inc., March 2004 – Assessment of grinding circuit throughput for suggested mill sizings supplied by Ivanhoe Mines for Oyu Tolgoi Project (South-West ore zone) as part of the conceptual design. Rev. 2.
- MinnovEX Technologies Inc., July 2004. Ivanhoe Mines Ltd., Oyu Tolgoi Project, Grinding Circuit Evaluation Interim Report.
- MinnovEX Technologies Inc., 10 September 2004. Grinding Laboratory Test Results.
- MinnovEX Technologies Inc., 14 February 2005. Summary of Oyu Tolgoi CEET runs conducted in Vancouver, Jan. 31 - Feb. 4, 2005 – Final Report.
- MinnovEX Technologies Inc., February, 2005. Ivanhoe Mines, Oyu Tolgoi Project, MFT Testwork and Fleet Study, Revision 0.
- MinnovEX Technologies Inc., 20 April 2005. Oyu Tolgoi Drill Core Samples, Grade Correlations with SPI.
- MinnovEX Technologies Inc., April 2005. Column Flotation Pilot Plant on Oyu Tolgoi South West Ore Conducted at SGS Lakefield.
- MinnovEX Technologies Inc., 25 May 2005. MFT Testwork and Fleet Study Report No. 2, Revision 1.
- AAJV MTR029 – Summary of Metallurgical Testwork Conducted by Lakefield Research (Report 1 of 5).
- AAJV MTR030 – Summary of Metallurgical Testwork Conducted at Lakefield Research (Report 2A of 5).
- AAJV MTR031 – Summary of Metallurgical Testwork Conducted at Lakefield Research (Report 2B of 5).
- AAJV MTR032 – Summary of Metallurgical Testwork Conducted at Lakefield Research Report 3 of 5 – South Deposit.
- AAJV MTR033 – Summary of Metallurgical Testwork Conducted by Lakefield (Report 4 of 5). (Note Report 5 of 5 is not for Southern Oyu material).
- Knight Piésold Report PE601-0001/16, December 2004. Site Water Management.

Reports used in Section 20, Environmental Studies, Permitting, And Social Or Community Impact:

- Integrated Development and Operations Plan, (IDOP)OT LLC April 2011
- EGI Report 8550/589, May 2003. Acid Rock Damage (ARD) Assessment of the Turquoise Hill (Oyu Tolgoi) Gold and Copper Project.
- EGI Report 8550/633, June 2004. Manual for Turquoise Hill Waste Rock Leach Columns Operated by Ivanhoe Mines Mongolia.
- EGI Report 8550/638, June 2004. Preliminary Cover Design Investigations for Control of ARD at Turquoise Hill.
- EGI Report 8550/641, October 2004. Assessment of Geochemical Data for the

Turquoise Hill Project.

- EGI Report 8550/677, June 2005. Geochemical Assessment of Pilot Plant and Bench Test Tailings for the Oyu Tolgoi (Turquoise Hill) Project.
- EGI E-mail, 16 June 2005. Oyu Tolgoi Waste Rock Column Update.
- EGI Appendix A – Assessment of Acid Forming Characteristics.
- Aquaterra Report, 15 June 2005. Oyu Tolgoi Integrated Development Plan, Water Supply and Dewatering Sections.
- Aquaterra Report, September 2004. Oyu Tolgoi Camp and Construction Water Supply.
- Aquaterra Report, October 2004. Feasibility Study, Oyu Tolgoi Dewatering Investigation – Open Pit and Block Cave Mining.
- Aquaterra Report, November 2004. Groundwater Conditions at the Hugo Shaft.
- Aquaterra Memo, 12 July 2004. Open Pit and Underground Mine Dewatering Predictions.
- Sustainability Report, 10 January 2005. Training System Development Report.
- Eco-Trade Ltd., October 2002. Environmental Baseline Study.
- Eco-Trade Co., April 2003. Preliminary Community Consultation Report for the Project.
- The Mongolian Academy of Sciences Institute of Archaeology, 2002. The Report for the Archaeological Excavation Conducted on the Licensed Field of the "Ivanhoe Mines Mongolia INCII".
- The Mongolian Academy of Sciences Institute of Archaeology, 2003. A Summary Report of the Archaeological Excavation Works Conducted through Roads from Oyu Tolgoi, an Exploitation Site of "Ivanhoe Mines Mongolia Inc". Company, To Gashuun Sukhait Frontier Crossing Point in the Territory of Khanborg Soum, South Gobi Aimak.
- Eco-Trade Ltd. Environmental Consultants, Mongolia, 2004. Oyu Tolgoi Project Environmental Impact Assessment.
- Volume 1, Report of Oyu Tolgoi to Gashuun Sukhait Road and Infrastructure Corridor, Environmental Protection Plan and Environmental Monitoring Plan.
- Sustainability 2005, Coordinate environmental, archaeological, and socioeconomic assessments and 2007 updates.

Reports used in Section 24.2 Risk Assessment:

- Integrated Development and Operations Plan, (IDOP)OT LLC April 2011 for political, tax and environmental risks.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Project is in the Aimag of Omnogovi, located in the South Gobi region of Mongolia. The property is approximately 550 km south of the capital city, Ulaanbaatar, and 80 km north of the border with China (Figure 4.1).

Figure 4.1 Project Location Map



The Project comprises 5 contiguous properties; three held by 100% by Oyu Tolgoi (6708A, 6709A, and 6710A) and two held by Entrée Gold LLC in bare trust for the JV between OT and Entrée (Shivee Tolgoi and Javkhlant). The vast majority of the identified mineralization on the Project occurs within the Oyu Tolgoi Property at the Hugo Dummett and Southern Oyu Porphyry Deposits. Only the northernmost extension of the Hugo Dummett Deposit (Hugo North) crosses onto the Shivee Tolgoi Joint Venture Property and the majority of Heruga mineralisation occurs on the Javkhlant property. (Entrée Gold Inc. refers to the Hugo North Deposit within its property limits, as the Copper Flats Deposit.) AMC has relied on OT LLC for the description of the Mineral Tenure in this section.

4.2 Permits and Agreements

4.2.1 Legal – Mongolia

The Mongolian Minerals Law (2006) and Mongolian Land Law (2002) govern OT LLC's exploration, mining and land use rights for the Project. Water rights are governed by the Mongolian Water Law and the Mongolian Minerals Law. These laws allow licence holders to use the land and water in connection with exploration and mining operations, subject to the discretionary authority of Mongolian national, provincial and regional governmental authorities as granted under Mongolian law. OT LLC expects that it will have to negotiate with all three levels of government to ensure access to appropriate land and water rights prior to the commencement of any mining operations.

OT LLC has studied, and continues to study the permitting and approval requirements for the development of the Project and maintains a permit and licensing register. OT LLC personnel work with the Mongolian authorities and have developed descriptions of the permitting processes and procedures for Project permitting in Mongolia. Some permits have already been obtained and others are in the process of being submitted. OT LLC has advised that it expects that all permits can be obtained in a suitable time frame for the Project development. Under the terms of the IA, a working group consisting of company and GOM representatives will be formed to assist in the permitting process.

With the adoption of the new Minerals Law in 2006, the GOM increased the royalty of 2.5% to 5.0% on the sales value of all minerals, except coal and common minerals, extracted from a mining licence area which have been sold, shipped for sale or used. Ivanhoe holds a 2% net smelter returns based payment over the property which was purchased from BHP Exploration in 2004.

For areas covered by exploration licences, an environmental plan accompanied the annual work plans submitted to the relevant soum, or district (Khanbogd Soum). The original environmental performance bond was posted in 1998 by BHP and is still retained by the soum for the ongoing work. Further requirements for environmental impact assessment are discussed below.

4.2.2 Investment Agreement and Taxation Assumptions

Both the process of negotiation and the final agreement presented an opportunity to confirm how the laws of Mongolia should be interpreted in their application to the Oyu Tolgoi Project and provided for some specific terms to apply to the Project. For OT LLC, the agreement has provided the confidence in the stability of the terms the Project will operate under and reliably assess its intended investment in the Project. The agreement itself is effective for an initial term of 30 years with an available extension of a further 20 years.

In accordance with the requirements outlined in the 2006 Minerals Law of Mongolia, upon execution of the IA and the fulfilment of all conditions precedent, the GOM became a 34% shareholder in OT LLC through the immediate issue of OT LLC's common shares to a shareholding company owned by the GOM. Upon a successful renewal of the IA

after the initial 30 year term, the GOM also has the option to increase its shareholding to 50%, under terms to be agreed with Ivanhoe Mines Ltd at the time.

The IA has been signed by all parties and became effective at the 31 March 2010 as a number of conditions precedent before terms come into effect were completed to the satisfaction of all parties. The main conditions precedent to the agreement were as follows:

- The submitted Mongolian feasibility study of the Project has been considered in accordance with the laws and regulations of Mongolia.
- Balance of existing income tax losses, capitalized expenses, outstanding tax liabilities or credits, had been confirmed by the tax office.
- The balance of shareholder loans had been agreed and audited.
- Company restructuring required to execute the agreement has been completed.
- A standing working committee being established with members of the GOM and OT LLC to provide a means to expedite permits, customs clearance or general GOM administration.
- Transfer of the Ivanhoe interests in licenses 15225A and 15226A held by Entrée LLC to OT LLC.
- Payment of the second tranche equity investment in Ivanhoe Mines Ltd by Rio Tinto PLC.

Despite its role as an equity owner, the GOM will not be required to contribute to the initial capital cost of the Project. Ivanhoe Mines Ltd, as the parent company, retains the right to fund the Project by way of a combination of debt, redeemable preference shares and common shares provided the debt to equity ratio falls within the 3:1 ratio required by Mongolian Law.

In the case of shareholder debt, loans (including existing shareholder loans at the time of the agreement) will attract an interest rate of 9.9% (real) with corresponding adjustments to be made to the outstanding balance to reflect increases in US CPI during each period. The coupon rate applicable to redeemable preference shares will be 9.9% (real) and carry the same escalation terms. All principal and interest outstanding on shareholder debt, outstanding coupon payments on redeemable preference shares and the face value of those redeemable preference shares must be paid in full prior to any dividends on common shares being paid.

In 2011 an Amended Shareholders Agreement was concluded which reduced the applicable rate from 9.9% to LIBOR plus 6.5%. In addition, an in principle agreement was reached to convert the balance of preference shares into ordinary shares. Both adjustments were to take place based on the 31 January 2011 balances, although the preference share conversion had not occurred by 31 December 2011.

Under the authority of the Shareholders Agreement, Ivanhoe Mines has the right to act as or appoint a Management Team to oversee the construction and operation of Oyu Tolgoi. The management team is compensated with a Management Services Payment equal to 3% of total operating and capital costs prior to commencement of operations and 6% of operating and capital costs during operations. This payment is

included in the economic analysis as a project expense and is confirmed as tax deductible in the IA.

OT LLC is required to achieve commencement of production within 7 years of the effective date of the agreement.

Under the terms of the IA, a range of key taxes have been identified as stabilized for the term of the agreement at the rates and base currently applied. The following taxes comprise the majority of taxes and fees payable to the GOM under Mongolian Law and are shown with their stabilized rates:

- Corporate income tax 25%
- Royalties 5% (gross sales value)
- Value added tax 10%
- Customs duties 5%

In accordance with the EPT invalidating law, as from 1 January 2011, the taxpayer will not be subject to the excess profits tax or any similar windfall tax.

OT LLC is also only subject to those taxes currently listed in the General Taxation Law and not taxes introduced at any future date. These taxes are collectively noted as non-discriminatory taxes and as such cannot be imposed on OT LLC in any manner other than that applied to all taxpayers. OT LLC may also apply to take advantage of any future law or treaty that comes into force and which would apply any rates lower than those specified in the IA.

The GOM recently enacted amendments to the legislation governing the carry forward of income tax losses. The loss carry forward period has been extended to 8 years and, if sufficient, can be applied to offset 100% of taxable income. This contrasts with the previous law in which losses carried forward for 2 years and were subject to a 50% limit.

The agreement also provides OT LLC with the benefit of a 10% tax credit for all capital investment made during the construction period. The amount of this credit can be carried forward and credited in the three subsequent profitable tax years. It is noted in the agreement that if VAT payments, which are currently non-refundable, become refundable in the future, the availability of the investment tax credit will cease from that point. In that event, past earned investment tax credits will still be applied.

Although it has a requirement to make its self-discovered water resources available to be used for household purposes, it is confirmed in the agreement that OT LLC holds the sole rights to use these water resources for the Project. The contract for the utilization of water with the GOM water authority is in effect for 30 years with subsequent 20 year periods of renewal.

The supply of power has been recognized as being critical to the execution of the Oyu Tolgoi Project in the agreement. OT LLC has been given the right to import power initially but must secure power from sources within Mongolia for the fourth year of operation.

OT LLC also has the right to construct roads for the transport of its product and airport facilities to suit the Project's needs. The GOM has committed to providing OT LLC with non-discriminatory access to any railway constructed between Mongolia and China if such a railway is constructed.

4.2.3 Shivee Tolgoi Property

On 9 November 2004 (the Shivee Tolgoi Effective Date) an Earn-In and Equity Participation Agreement (the Earn-In Agreement) was finalized between Ivanhoe and Entrée Gold Inc. (Entrée) giving Ivanhoe the right to earn an interest in a portion (39,864 ha) of the overall area of Entrée's Shivee Tolgoi property and was initially referred to as the Shivee Tolgoi Earn-in Property (the Earn-in Property; Figure 2.1). On 11 March 2008, Ivanhoe Mines notified Entrée that it had incurred sufficient expenditures (>\$27.5 M) to earn a 60% interest as was outlined in the initial agreement. At the end of June 2008, Ivanhoe Mines notified Entrée that it had incurred sufficient expenditures (>\$35 M) to earn an 80% participating interest and thereby forming a joint venture. The property is now referred to as the Shivee Tolgoi Joint Venture ("Shivee Tolgoi JV").

The JV property is adjacent to the north, east and south of the Oyu Tolgoi Property and includes Entrée's Javkhant Mining Licence (15225A) and the eastern portion of the Shivee Tolgoi Mining Licence (15226A). The details of the two licences are provided in Table 4.1. The boundaries of the two mining licences are defined by latitude and longitude coordinates and by UTM coordinates with datum set to WGS-84, Zone 48N and these are listed in Table 4.2.

Table 4.1 Details of Shivee Tolgoi Joint Venture Property

Mining Licence	Mining Licence Name	Area (ha)	Area of Agreement (ha)	Licence Date	Date of Licence Extension
15226A	Shivee Tolgoi	54,760	19,518	27 October 2009	27 October 2039
15225A	Javkhant	20,346	20,346	27 October 2009	27 October 2039
Total			39,864		

The two exploration licences (3148X and 3150X) considered in the JV property were granted in April 2001 to a private Mongolian company, Mongol Gazar Co. Ltd. (Mongol Gazar) and were originally acquired by Entrée (through its wholly owned Mongolian subsidiary Entrée LLC) in July 2002 via a 5 year option agreement. In September 2003, the option agreement was replaced by a purchase agreement and Entrée completed the 100% purchase of the licences in November 2003.

In October 2009, Entrée received notice that its application for conversion of the Shivee Tolgoi and Javkhant exploration concessions to mining licences had been approved by the Mineral Resources Authority of Mongolia and the licences have been received, as shown in Table 4.2.

Successful conversion of these exploration licences to mining licences was a condition precedent to the IA signed earlier in October 2009, by Ivanhoe Mines, the parent company of OT LLC and Rio Tinto PLC, minority partner in the Project, with the GOM.

These mining licences are contained within the Contract Area covered by the IA. Under the terms of the Entrée-Ivanhoe Mines joint venture, Entrée retains either a 20% or 30% carried interest on these licences, dependent on the depth of mineralization. The duration and financial obligations to the GOM for the mining licences are described in the introduction to Section 4.2.2.

Table 4.2 Shivee Tolgoi Joint Venture Property Boundary Coordinates

Mining Licence	Point	Latitude/Longitude, MSK		UTM, WGS-84, Zone 48N	
		Latitude	Longitude	Easting	Northing
15226A (Shivee Tolgoi eastern portion only)	1	43,08,00 N	106,47,30 E	645752.90	4777222.00
	2	43,08,00 N	107,00,00 E	662698.13	4777605.65
	3	43,00,00 N	107,00,00 E	663051.38	4762798.00
	4	43,00,00 N	106,55,00 E	656258.54	4762639.53
	5	43,03,00 N	106,55,00 E	656131.70	4768192.34
	6	43,03,00 N	106,47,30 E	645950.77	4767967.25
15225A (Javkhlant)	1	43,00,00 N	106,36,00 E	630445.98	4762098.85
	2	43,00,00 N	106,47,30 E	646069.34	4762414.47
	3	42,58,30 N	106,47,30 E	646128.58	4759638.09
	4	42,58,30 N	106,55,00 E	656321.92	4759863.13
	5	43,00,00 N	106,55,00 E	656258.54	4762639.53
	6	43,00,00 N	107,00,00 E	663051.38	4762798.00
	7	42,55,30 N	107,00,00 E	663249.69	4754468.83
	8	42,55,30 N	106,55,00 E	656448.59	4754310.38
	9	42,57,30 N	106,55,00 E	656364.16	4758012.21
	10	42,57,30 N	106,51,30 E	651605.97	4757905.31
	11	42,55,30 N	106,51,30 E	651687.83	4754203.49
	12	42,55,30 N	106,44,00 E	641486.25	4753985.55
	13	42,57,00 N	106,44,00 E	641428.99	4756761.90
	14	42,57,00 N	106,38,00 E	633271.07	4756598.47
	15	42,55,30 N	106,38,00 E	633325.02	4753822.13
	16	42,55,30 N	106,36,00 E	630604.62	4753769.81

Note: Point 1 for each Licence corresponds with the NW corner of the Licence and then moving clockwise (see Figure 4.1)

4.3 Environmental Considerations and Liabilities

Holders of a mining licence in Mongolia must comply with environmental protection obligations established in the Environmental Protection Law of Mongolia, Law of Environmental Impact Assessment and the Minerals Law. These obligations include preparation of an environmental impact assessment (EIA) for mining proposals, submitting an annual environmental protection plan, posting an annual bond against completion of the protection plan and submitting an annual environmental report.

OT LLC has posted environmental bonds to the Mongolian Ministry for Nature and Environment (MNE) in accordance with the Minerals Law of Mongolia for restoration and environmental management work required for exploration and the limited development work undertaken at the site. OT LLC pays to the Khanbogd Soum annual fees for water and road usage, while sand and gravel use fees are paid to the Aimag government in Dalanzadgad.

In March 2003, IMMI (now OT LLC) completed and submitted to the GOM an environmental baseline study regarding the Project. The study, together with a brief project description, was subject to screening under the Mongolian EIA procedures and guidelines were issued for a "Detailed EIA" of the Project. The EIA was prepared by Eco Trade LLC, a licensed Mongolian EIA consultancy, in accordance with statutory procedures and the screening guidelines. The EIA work was supervised and coordinated by Sustainability Pty Ltd, of Australia, to apply international standards and methods of assessment.

The EIA for the Project consists of three volumes: (i) road, (ii) water supply, and (iii) mine and processing facilities. The first EIA document, for the transport corridor south of the Oyu Tolgoi property to the Chinese border, was submitted in April 2004 and approved in May 2004. An amendment to the approved EIA was submitted in December 2006 to allow for an alternative road alignment to the Chinese border. The EPP for this alignment was recently rejected by the MNET due to a resolution stating that certain roads in the South Gobi region need to be bitumized. The EPP will be re-submitted referencing the bitumizing of the road.

The second volume of the Project EIA, covers the supply of water from nearby aquifers, was submitted in June 2005. Approval was obtained for the use of the Gunii Hooloi water reserve in September 2005. Since approval, changes have been proposed to the mine process, which has resulted in an increase to the Project water demand. In addition, further hydrogeological investigations at Gunii Hooloi have occurred, which has allowed a revision to be made to the aquifer capacity, characteristics and exploitable reserves. This revision is provided in the Report on Groundwater Exploration (IMMI, 2007) submitted by IMMI (now OT LLC) to the MNE in January 2008. This document was reviewed by the Water Sector Science and Technology Council who approved registration (January 2009) of the available water resource at Gunii Hooloi of 870L/s.

An explanatory amendment document to the approved 2005 EIA is to be submitted to the MNE in June 2010. The document assesses any potential changes to the environmental impacts associated with the use of the resource as a result of the latest hydrogeological assessments and approved exploitable water reserves.

The third volume, incorporating the results of the 2005 IDP, was first submitted in January 2006 and, after review by the GOM, the submission was amended in May 2006. An expert committee review of the mine and processing facilities volume of the EIA was completed in September 2006, and further information was submitted in November 2006 to address the issues raised. The third volume of the EIA was approved by the MNE in December 2007.

Further volumes have been established to accommodate the Power Plant (Volume IV) and the temporary Domestic Airport Re-location (Volume V – approved 2007), EPP approved May 2010: and a number of supplementary assessments (Volume VI) which cover current development work associated with underground Shafts 1 and 2, waste water treatment, diesel power supply, emulsion plant, chemicals import and usage, the quarry, and batch plant for construction materials. The Power Plant Screening and EIA reports are prepared for a 3 x 150 MW coal fired steam power plant located on the mining licence at the Project. Submission to the MNE is pending the exact location if on site. The water supply pipeline and borefield infrastructure EIA was approved May 2010.

Table 4.3 Oyu Tolgoi EIA Approvals

Nº	Name of DEIA	DEIA Report Development	Local Approval Stage	State Approval Stage
1	Amendment of OT GS road 2011	Completed	Completed	Completed
2	Amendment of GH Hooloi water recourse	Completed	Completed	Completed
3	Amendment of Mining and Processing	Completed	Completed	Completed
4	Permanent Airport – Khanbumbat	Completed	Completed	Completed
5	Javkhlant – Entrée Lease area	Completed	Completed	Completed
6	Shivee Tolgoi –Entrée Lease area	Completed	Completed	Completed
7	Umdai River Diversion	Completed	Pending	Pending
8	Amendment of Chemical	Completed	Completed	Completed
9	Power plant	Completed	Completed	Completed
10	Amendment of OT GS road DEIA 2012	Ongoing	–	–
11	Amendment of Gunii Hooloi Pipeline DEIA	Ongoing	–	–
12	Dalan Shar lime stone DEIA	Completed	Ongoing	–

OT LLC has, through qualified independent consultants, prepared an EIA for Oyu Tolgoi consisting of three parts:

- Road
- Water supply
- Mine and processing facilities

The first EIA document, for the transport corridor south of the Oyu Tolgoi property to the Chinese border, was submitted in April 2004 and approved in May 2004. An amendment to the approved EIA was submitted in December 2006 to allow for an alternative road route to the Chinese border. The second volume of the EIA, covering the supply of water from nearby aquifers, was submitted in June 2005 and approved in September 2005. The third document, incorporating the results of the 2005 IDP was first submitted in January 2006 and, after review by the GOM, the submission was amended in May 2006. An expert committee review of the mine and processing facilities volume of the EIA was completed in September 2006, and further information was submitted in November 2006 to address the issues raised. The third volume of the EIA was approved by the MNE in November 2007.

OT LLC has received approval for supplementary environmental assessments and management plans covering current development work associated with the underground shafts, temporary site accommodation facilities, waste water treatment, diesel power supply and the excavation of construction materials. A further EIA report is being prepared for a 3 x 150 MW coal fired steam power plant located on the mining licence at the Project.

OT LLC has retained the Institute of Archaeology at the Mongolian Academy of Science to complete archaeological studies of the Project. The studies have resulted in the excavation and removal of sites of historical and cultural significance within the Project are in accordance with the relevant Mongolian Laws and custom.

To date, no environmental issues have been identified that cannot be managed through normal mining.

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

5.1 Topography, Elevation, and Vegetation

5.1.1 Topography and Elevation

The topography of the project area largely consists of gravel-covered plains, with low hills along the northern and western lease borders. Small, scattered rock outcrops and colluvial talus are widespread within the northern, western, and southern parts of the property.

The Project is centered at approximately latitude 43°00'45"N, longitude 106°51'15"E or UTM coordinates 4764000 N and 651000 E with datum set to WGS-84, Zone 48N. The Southern Oyu and Hugo Dummett Deposits are the principal zones of mineralization defined on the Project and these occur within a north-north-east trending, 8 km long and 1 km wide mineralized "corridor" in the central portion of the Project at elevations ranging from approximately 1,140 masl to 1,215 masl.

5.2 Property Access

5.2.1 Property Access – General

Access to the property from the Mongolian capital, Ulaanbaatar, is possible either by an unpaved road, via Mandalgovi, a 12 hour drive under good conditions, or by air. Oyu Tolgoi LLC has constructed a 2,000 m long gravel airstrip at the site and routinely receives flights from Ulaanbaatar. The Trans Mongolian Railway crosses the Mongolia-China border approximately 420 km east of the property, traversing the country from south-east to north-west through Ulaanbaatar to the border with Russia. At the Mongolian-Chinese border the rail gauge changes from the Russian standard to the Chinese standard. A desert trail connects the rail line at Sajn Sand to the site.

The Chinese government has upgraded 226 km of road from Ganqimaodao to Wuyuan, providing a direct road link between the Mongolian border crossing at Gushaan Suhait, 80 km south of Oyu Tolgoi, and the Trans-China Railway system. A rough track/trail crosses the desert from the Mongolian border to the site.

Ulaanbaatar has an international airport, and Mandalgovi and Dalanzadgad have regional airports. There is currently charter air service between the site and Ulaanbaatar. The closest regional airport in China is at Hohhot. There are no airport facilities at Wuyuan or Bayan Obo.

5.2.2 Property Access – Protected Areas

The Small Gobi Strictly Protected Area (SGSPA) is approximately 80 km south of the mine licence area, on the Mongolia-China border. The access road corridor traverses through 13 km of the protected area. With the acceptance of the EIA for the corridor in June 2004, OT LLC has received approval to cross through this area.

5.3 Regional Population Centres and Infrastructure

There are a number of communities in the South Gobi region. The most prominent is Dalanzadgad, population 15,000, which is the center of the Omnogovi Aimag and is 220 km north-west of the Oyu Tolgoi property. Facilities at Dalanzadgad include a regional hospital, tertiary technical colleges, a domestic airport, and a 6 MW capacity coal-fired power station. OT LLC envisions that Dalanzadgad may be suitable as a regional center for recruiting and training. The closest community to the property is Khanbogd, the center of the Khanbogd soum. Khanbogd has a population of approximately 2,500 and is 35 km to the east. Other communities relatively near to the Project include Mandalgovi (population 13,500), the capital of the Dundgovi Aimag, 310 km north of the Project on the road to Ulaanbaatar, Bayan Ovoo (population 1,600), 55 km to the west, and Manlai (population 2,400), 150 km to the north.

5.4 Climate and Length of Operating Season

5.4.1 Climate and Operating Season

The South Gobi region has a continental, semi-desert climate. The spring and autumn seasons are cool, summers are hot, and winters are cold. Typical of desert climates, the site has low average humidity and significant variations in daily temperatures.

Knight Piésold conducted an extensive evaluation of the available climatic information for the Project area using regional data from bibliographical sources and local data from nearby climate stations.

The climatic conditions are such that the operating season for the Project will cover the entire year on a continuous shift basis. Minor disruptions are expected and have been allowed for in the Project operating hours estimates.

5.4.2 Data Sources

Data were derived primarily from climatic records for Byan Ovoo, approximately 75 km west of Oyu Tolgoi, and from 2 years of available Oyu Tolgoi site data. Although these data have some limitations they are considered adequate for use in design. Data were also obtained from Khanbogd, approximately 45 km north-east of the site, Dalanzadgad, 220 km north-west, and Hailutu, 200 km south-west, but the information from Byan Ovoo was deemed the most representative of conditions at Oyu Tolgoi.

5.4.3 Air Temperature

Temperatures range from an extreme maximum of about 50°C to an extreme minimum of about -34°C. The typical air temperature in winter fluctuates between 6°C and -21°C. In the coldest month, January, the average temperature is -12°C. Data from Byan Ovoo are shown in Table 5.1.

Table 5.1 Monthly Temperatures (°C) Based on Byan Ovoo Data

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Minimum	-34	-33	-25	-22	-13	0.4	4	2	-4	-20	-26	-33
Average	-12	-8	-0.4	9	17	23	25	22	16	7	-2	-10
Maximum	9	16	24	31	38	49	40	39	39	30	25	14

5.4.4 Relative Humidity

The average relative humidity ranges from 18.7% in May to 53.3% in January. Daily relative humidity is dependent on current temperature and varies considerably. Table 5.2 shows monthly relative humidity statistics using the calculated hourly averages from the site weather station. The design relative humidity for summer is based on an analysis of the July 2002 and 2003 hourly temperatures and corresponding relative humidity values. The design relative humidity for a 1 July temperature of 34.5°C is 15.1%.

Table 5.2 Monthly Relative Humidity

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Minimum	19	12	3	2	1	1	4	8	1	2	5	10
Average	53	38	23	24	18	31	37	36	34	30	40	43
Maximum	81	67	88	89	100	96	100	100	100	81	85	80

5.4.5 Ground Temperature

From the data available to date, the minimum recorded ground temperature is -39°C and the maximum is 70°C. Table 5.3 shows the design freezing depths at the site based on the Mongolian Climate Data and Geophysical Parameters for Use in Construction Design Code Document No. CNR 2.01-01-93.

Table 5.3 Design Soil Freezing Depths

Soil Type	Freezing Depth(m)
Clayey soil	1.5
Sandy soil	1.9
Gravely soil	2.2

5.4.6 Solar Radiation

Solar radiation data have been collected at the Oyu Tolgoi site station since 2002. Solar radiation is measured in W/m² and fluctuates during the day, ranging from 0 W/m² at night and peaking soon after mid-day. The average daily maximum for the 2 years of data available is 655 W/m², the highest daily maximum is 1,030 W/m², and the lowest daily maximum is 76 W/m².

Maximum levels of solar radiation are lower during the winter. The average daily maximum is 429 W/m² for January and 859 W/m² for July.

5.4.7 Precipitation

Average annual precipitation is 57 mmpa, 90% of which falls as rain and the rest as snow. Snowfall accumulations rarely exceed 50 mm. Maximum rainfall events of up to 43 mmph for a 1-in-10 year, 10 minute storm event have been recorded. In an average year, rainfalls occur on only 19 days, and snow falls on 10 to 15 days. The ground snow load is 0.1 kPa. Monthly rainfall data are shown in Table 5.4 and Table 5.5. Both tables are derived from Byan Ovoo data for 1975 to 2002.

Table 5.4 Rainfall Summary(mm)

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Maximum daily	2.1	3.8	4.4	10.4	19.0	16.2	29.5	102.0	19.2	4.0	4.3	1.5	–
Average monthly	0.4	0.4	0.8	1.4	3.1	8.1	18.1	17.8	5.0	0.9	0.6	0.2	56.8
Average rain days per month	0.6	0.6	1.0	0.8	1.5	3.0	4.5	3.9	1.4	0.6	0.7	0.4	19.0

Table 5.5 Rainfall Intensities(mmph)

Return Interval Duration	1 in 2 Years	1 in 10 Years	1 in 20 Years	1 in 50 Years	1 in 100 Years	1 in 500 Years
10 minutes	15.4	44.2	63.5	99.8	138.3	284.2
30 minutes	10.0	28.7	41.3	64.8	89.9	187.7
60 minutes	6.8	19.5	28.0	44.0	60.9	125.2
2 hours	4.3	12.3	17.7	27.8	38.6	79.3
3 hours	3.2	9.2	13.3	20.9	28.9	59.4
6 hours	1.9	5.5	7.9	12.4	17.2	35.4
12 hours	1.1	3.2	4.6	7.3	10.1	20.7
24 hours	0.7	1.9	2.7	4.2	5.9	12.0
48 hours	0.4	1.1	1.5	2.3	3.2	6.3
72 hours	0.3	0.8	1.0	1.6	2.2	4.2

5.4.8 Thunderstorms and Lightning

Local records indicate that thunderstorms are likely to occur from 2 to 8 days each year at Oyu Tolgoi. Electrical activity generally totals about 29 hours each year. An average storm will have up to 83 lightning flashes a minute.

5.4.9 Evaporation

Given the importance of this variable for determining project water requirements, a number of different methods were used to generate and analyse evaporation data to determine design levels. The results are summarized in Table 5.6. It should be noted that site measurements are ongoing to confirm these results.

Table 5.6 Design Evaporation Data

Month	Sublimation (Water bodies Frozen in Winter) (mm)	Evaporation	
		Winter	Summer
		(Open Water bodies) (mm)	
January	22	82	–
February	41	101	–
March	–	–	142
April	–	–	256
May	–	–	439
June	–	–	378
July	–	–	382
August	–	–	285
September	–	–	192
October	–	–	132
November	53	11	–
December	27	88	–
Total Water Loss: Sublimation + Evaporation = 2349 mm			

5.4.10 Wind Loading and Dust Generation

Wind is usually present at the site, predominantly from the north. Very high winds are accompanied by sandstorms that often severely reduce visibility for several hours at a time. At present, site-specific wind monitoring data are available for only a short period of time, less than a year. Based on regional information, windstorms can have gusts up to 50 m/s. Snowstorms and blizzards with winds up to 40 m/s occur in the Gobi region for 5 to 8 days each Winter. Spring dust storms are far more frequent and can continue through June and July. The average storm duration is 6 to 7 hours. An average of 120 hours of dust storm activity and 220 hours of drifting dust are recorded each year.

Based on the Mongolian Code, the Basic Wind Speed is 34 m/s. Maximum 1 hour speeds recorded at Byan Ovoo are shown in Table 5.7. The number of dust storms per month is shown in Table 5.8.

Table 5.7 Maximum One-Hour Wind Speeds (m/s) at Byan Ovoo

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Maximum wind speed	13.4	14.0	15.4	18.1	16.6	16.2	16.3	14.8	16.0	18.6	19.3	14.5

Table 5.8 Frequency of Dust Storms in the Gobi Desert

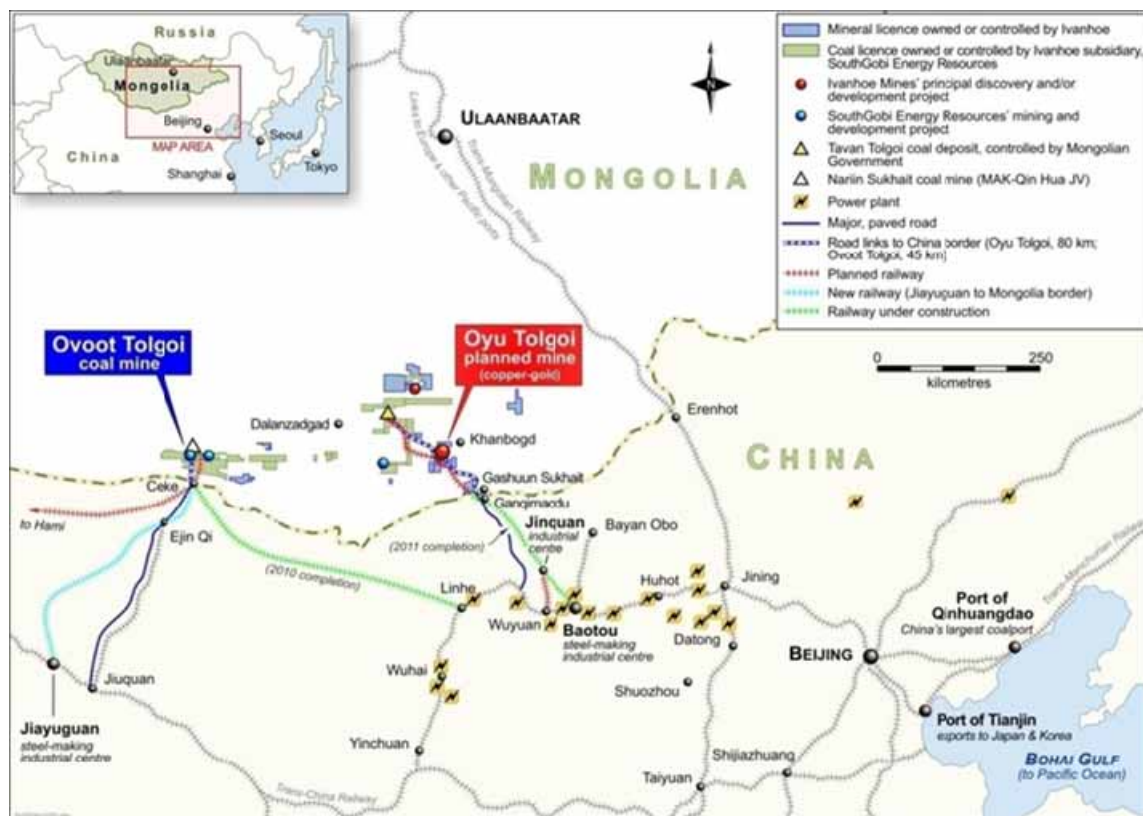
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
No. of days	0.7	1.0	2.4	4.7	4.1	1.5	1.0	0.4	0.6	0.7	1.9	0.7

5.5 Site Infrastructure and Local Resource Considerations

5.5.1 Power

Since 2005, OT LLC conducted a study into construction of a power station at the Oyu Tolgoi site, it was agreed in the IA that power may be sourced externally from sources outside Mongolia until Year 4 at which time power to the Project will be sourced from a power station within Mongolia, whether operated by OT LLC or an external party. A final decision on the source of initial power needs to be progressed, power supply contracts negotiated with associated utilities and planning/approvals done for the power transmission corridor. In addition to this, options for the location and ownership of the Mongolian power station should be studied and identified. The locations of power providers in northern China is shown in Figure 5.1.

Figure 5.1 Power Northern China Grid



5.5.2 Water

The Project area is located within the closed Central Asian drainage basin and has no outflow to the ocean. Most riverbeds in this drainage basin are ephemeral creeks that remain dry most times of the year. The Umdai River is the most significant hydrological feature of the Project area. A tributary of the river passes through the site.

Flows after heavy summer rainstorms often result in very turbulent, high-velocity mud flows, locally termed “Gobian wild floods.” These floods have been known to destroy road crossings and to carry away vehicles caught in the riverbeds. No surface flow data are available for these isolated and episodic flood events. During exploration, only two such events were experienced from 2002 to 2009. Further discussions with locals indicate these events can occur yearly (excluding current drought conditions).

Shallow springs are used by wildlife and livestock as drinking water sources. Migratory wildlife movements during summer months in the Gobi are likely to be dictated by the presence of surface water in natural springs.

Water quality baseline data for surface waters throughout the Project area, access road corridor, and aquifer areas are being collected through current monitoring programs. This information will be established prior to project development as a basis for assessing potential project-related impacts on surface water quality during routine monitoring.

Potential impacts on surface water systems in the Project area will include local changes to natural flow paths and depletion of springs, ephemeral wetlands, and salt-marshes from project development and operation activities. The mitigation and design work with regard to surface water focused on the potential impacts to surface water quality include increased sedimentation and risk of pollution of springs, ephemeral wetlands, and salt-marshes from increased erosion, contaminated dust fallout, contaminant spills, and accidents associated with project construction and operational activities.

Fugitive dust control management plans and spill management systems are being developed to avoid and mitigate potential impacts to air and surface water quality. These studies will be used to mitigate impacts that may result in loss of wildlife habitat, decrease in wildlife health, and decrease in wildlife population because of higher mortality rates.

On and off-site infrastructure associated with the upgrading of road facilities at the site and along the corridor include the formation of dedicated stream crossings, which may reduce the number of undefined and informal crossings that now exist along tracks within the corridor.

5.5.2.1 Hydrogeology and Groundwater Quality

Detailed groundwater investigations to date have been concentrated in the Gunii Hooloi, Galbyn Gobi, and Nariin Zag aquifer areas to assess the potential to meet the Oyu Tolgoi Project’s estimated water demand. Groundwater investigations conducted in the mine licence area focused on assessment of required dewatering rates for mine works and the potential to meet the Project’s camp and construction water demands. Process water supply has been registered and will be piped from the Gunii Hooloi basin to the north-west of the Project area. Current studies are ongoing at site to confirm groundwater model predictions with respect to dewatering of the pit and underground and groundwater environmental impacts.

5.5.3 Site Infrastructure

Oyu Tolgoi is a remote greenfields project and therefore requires extensive infrastructure to be constructed in addition to the concentrating facilities. The major initial infrastructure elements include:

- Water Borefields

Water demand will be supplied from the Gunii Hooloi basin which extends 35 to 75 km north of the Oyu Tolgoi site. Bores will be developed in the south-west and the north-east areas of the Gunii Hooloi borefield with storage lagoons along the pipeline designed to provide for emergency use without impacting site water needs.

- Water Treatment

A permanent water treatment facility and bottling plant will be constructed to treat raw water from the Gunii Hooloi borefield to drinking (potable) and domestic water standards.

- Housing

Accommodation facilities will be constructed to support the operations phase of the Project. Temporary facilities will also be constructed throughout the Project to support additional manpower requirements for construction and expansion demands.

- Airstrip

The airport will be located approximately 7 km north of the Oyu Tolgoi mine site and will facilitate the transport of people and goods to the site from Ulaanbaatar and other points of departure.

- Supporting Facilities

Administration, training, mine equipment maintenance, gatehouse, medical center, fire station, heating plant, fuel storage, and warehouse facilities, among others will also be constructed to support operational requirements over the life-of-mine.

- Railroads

During the initial 3 years of operation, the transport of bulk supplies and the delivery of copper concentrate to China will be by access road to the railhead. However, direct rail transport is considered a long term transportation solution after this initial development period.

- Power

The IDOP Technical Report assumes initial power for the Project is to be sourced from Inner Mongolia, China. By the fourth year, in accordance with the terms of the recently completed IA with the GOM, power must be sourced from a power station within Mongolia; although it is not required that OT LLC must build a power station. This report, however, assumes that a power station sufficient for ongoing power requirements is constructed by OT LLC by the fourth year of operations.

5.5.4 Other

5.5.4.1 Land Use

The land surrounding the mine licence area is predominantly used for nomadic herding of goats, camels, and sheep by small family units. Use is based on informal traditional Mongolian principles of shared grazing rights with limited land tenure for semi-permanent winter shelters and other improvements. Initiation of the herder support program has reduced the incidence of land use conflict between current mineral exploration and grazing practices. The Project intends to maintain co-existence of traditional grazing practices and mineral development except where there is a risk to public safety or livestock.

5.5.4.2 Risk Assessment

The Law of Mongolia on Environmental Impact Assessment (2001) and the guidelines issued for the IMMI EIA (2001) require the inclusion of a risk assessment in project documentation. "Risk assessment" means identification and prediction of the possible emergencies and accidents that could occur during the production process or natural disasters, and elimination and mitigation of their consequences.

5.5.4.3 Ongoing Work

A complete list of outstanding work to be completed at various stages of development, and current status, is included in the report. Key work includes baseline studies and assessment of project-related impacts along the access road corridor from Mongolia-China border to Wuyuan, China; wildlife population dynamics, habitat use, ecology and migratory habits of key species within the region; transboundary issues, cumulative effects, human health risks, and noise effects; and continued evaluation of acid mine drainage and metal leachate potential, hydrology, water quality of surface water occurrences, groundwater resources, air quality, soil chemistry; and projected impacts. Completion of this work will aid in customizing and improving existing environmental management and monitoring plans as part of an Adaptive Environmental Management System.

5.5.4.4 Closure and Reclamation

As part of overall project planning, OT LLC has prepared a preliminary reclamation and closure plan. Certain features of the mine, such as the open pit, waste dumps, and tailings impoundment, will create permanent changes to the current landscape that cannot be completely remedied through reclamation. The closure plan will; however, ensure that these disturbed areas are seismically and chemically stable as to limit the ecological impacts to the surrounding water, air, and land.

The closure plan for the Project will address the socio-economic impacts of mine closure considering that the existence and economic survival of some communities may have become dependent on the Project. Issues include ongoing environmental management during and after reclamation, loss of jobs, and socio-economic impact to the region.

The primary reclamation objective is to develop the mine in a manner that prevents leaving an unsustainable environmental legacy and that considers community input and values. Other key objectives are as follows:

- Protect public health and safety during all stages of project development.
- Prevent or mitigate environmental degradation caused by mining-related activities.
- Return the maximum amount of disturbed land to pre-mining conditions suitable for nomadic herdsman and their grazing animals.
- Secure the open pit areas, subsidence zones, waste dumps, and tailings storage facilities to ensure public and environmental safety.
- Plan and implement reclamation techniques that eliminate the need for long-term maintenance presence on site and permit OT LLC to “walk away” from the reclaimed mine with no environmental or social encumbrances.

OT LLC will develop environmental monitoring plans, including proposed activities and schedules, to ensure that environmental parameters meet the criteria, standards, and limits laid out in the EIA and Environmental Protection Plan. In accordance with Mongolian law, OT LLC will undertake monitoring at its own expense using approved methods and accredited facilities. The monitoring will permit procedures and activities to be adjusted and/or modified as necessary to ensure optimal environmental protection.

Parameters to be monitored during the closure and post-closure phases of the mine include the following:

- Surface water and groundwater quality.
- Physical stability of tailings deposits.
- Physical stability of the river water diversion dike, waste rock dumps, drainage ditches, and concrete shaft/raise caps.
- Isolation of open pit voids and unfilled subsidence zones, including status of open water and erosion controls.
- Success of indigenous revegetation, including remediation as required until proven to be self-sustaining.
- Condition of groundwater monitoring wells, piezometers, survey monuments, and other instrumentation.
- Effectiveness of dust-control measures on waste rock, tailings storage facility, and other waste areas with specific attention to potential wind-blown contaminant sources.

5.5.4.5 Seismic Zone and Risk

OT LLC reported in IDOP that a seismic hazard assessment of the Oyu Tolgoi site was completed. The seismicity of the Oyu Tolgoi site was determined to be low, and the seismicity of eastern Mongolia is generally low. However, to the west of Oyu Tolgoi lies the Mongolian Altai – a tectonically active mountain range stretching 1,700 km from south-west Siberia to the Gobi Desert. The easternmost extension of the Mongolian Altai

is known as the Gobi Altai, which dies out approximately 50 km to 100 km west of Oyu Tolgoi.

The Research Center for Astronomy and Geophysics of Academy of Science (Seismology Center) is responsible for assessing seismology in Mongolia. OT LLC appointed the Seismology Center to perform a seismic assessment for the project that was incorporated into the assessment.

6 HISTORY

6.1 History of Oyu Tolgoi Property

The existence of copper in the Oyu Tolgoi area has been recognized since the Bronze Age, but contemporary exploration for Mineral Resources did not begin until the 1980s, when evidence of alteration and copper mineralization at South Oyu was noted. In September 1996 Magma Copper team began exploring the area. The outcrops of stock-worked quartz veining at Central Oyu were identified as a leached cap of a porphyry copper deposit and exploration tenements were secured in late 1996. Magma was subsequently acquired by BHP Billiton (BHP).

Geophysical surveying at Oyu Tolgoi was first initiated by BHP in 1996. An airborne magnetometer survey was flown at a height of approximately 100 m on 300 m spaced, east-west oriented lines over approximately 1,120 km² of BHP's mineral concession. The survey provided good resolution of the magnetic features to facilitate geological and structural interpretation across the concession areas. BHP also undertook an induced polarization (IP) survey utilizing a single gradient array with a 2,000 m AB and a ground magnetometer survey. Both surveys were conducted on east-west-oriented lines surveyed by a local Mongolian surveying team at 250 m spacing. The surveys covered Southern, Southwest, Central, and North Oyu exploration targets but did not extend into the Far North region that ultimately became the Hugo Dummett deposit.

BHP carried out geological, geochemical, and geophysical surveys and diamond drilling programs (23 holes total) in the Central and South deposits in 1997 and 1998. Copper and gold were encountered at depths from 20 m to 70 m below surface. Based on the results of this drilling, BHP estimated a preliminary resource of 438 Mt, averaging 0.48% copper and 0.25 g/t gold, early in 1999. The historical resource estimate pre-dates NI 43-101 and is not necessarily in accordance with the resource classification categories prescribed therein.

BHP then halted its exploration in Mongolia and offered its properties for joint venture. IMMI (now OT LLC) visited Oyu Tolgoi in May 1999 and made an agreement to acquire 100% interest in the property, subject to a 2% NSR based payment.

In 2000, IMMI (now OT LLC) completed 8,000 m of reverse circulation (RC) drilling, mainly at the Central deposit, to explore the chalcocite blanket discovered earlier by BHP. Based on this drilling, IMMI (now OT LLC) estimated an Indicated Resource of 31.7 Mt at 0.80% copper and an additional Inferred Resource of 11.2 Mt grading 0.78% copper for the chalcocite blanket. The historical resource estimate pre-dates NI 43-101 and is not necessarily in accordance with the resource classification categories prescribed therein.

In 2001, IMMI (now OT LLC) continued RC drilling, mostly in the South deposit area, to test possible oxide resources, and then drilled three diamond core holes to test the deep hypogene copper-gold potential. Hole 150 intersected 508 m of chalcopyrite-rich mineralization grading 0.81% copper and 1.17 g/t gold; hole 159 intersected a 49 m thick chalcocite blanket grading 1.17% copper and 0.21 g/t gold, followed by 252 m of hypogene covellite mineralization grading 0.61% copper and 0.11 g/t gold. Holes 150 and 159 are part of the IMMI (now OT LLC) drillhole database and has been subject to

the same Quality Assurance Quality Control (QA/QC) and other validation as the other holes drilled at Oyu Tolgoi.

These results encouraged IMMI (now OT LLC) to mount a major follow-up drill program. In late 2002, drilling in the far northern section of the property intersected 638 m of bornite-chalcopryrite-rich mineralization in hole 270, starting at a depth of 222 m. This hole marked the discovery of the Hugo Dummett deposit.

IMMI (now OT LLC) completed all of its earn-in requirements by June 2002 and acquired the 2% NSR based payment retained by BHP in November 2003. As a result, IMMI (now OT LLC) became the sole owner of the property. The licences are valid for 60 years, with an option for IMMI (now OT LLC) to acquire a 40 year extension. The licences were converted from exploration licences originally issued on 17 February 1997.

IMMI (now OT LLC) was granted mining licences for the Oyu Tolgoi property and three satellite properties on 23 December 2003 in accordance with the Minerals Law of Mongolia. Also in accordance with this law, IMMI (now OT LLC) submitted a Scoping Study in February 2004 to support these licences, which give IMMI (now OT LLC) the right to mine within the bounds of the licence area. The Scoping Study was published as a 43-101 Preliminary Assessment Report.

The Southern Oyu open pit Mineral Reserves were first reported in January 2006 in a 43-101 pre-feasibility study. The study reported only open pit production and was considered to be a stage on the path to full development of the deposits.

6.2 History of the Shivee Tolgoi Property

Hugo North Extension represents the extension of the Hugo North deposit into the Shivee Tolgoi licence. IMMI (now OT LLC) initiated exploration work on the Shivee Tolgoi licence in November 2004 following the signing of the earn-in agreement with Entrée.

Prior to that time, Entrée had undertaken soil geochemistry, ground magnetics, Bouger gravity and pole-dipole geophysical surveying, and geological mapping.

IMMI (now OT LLC) commenced drilling north from the northern boundary of the Oyu Tolgoi mining licence in 2005, stepping out from the existing drilling in the Oyu Tolgoi Mining Licence. A first-time resource estimate for the Hugo North Extension deposit was completed in 2006.

Starting at the northern boundary of the OT License, an IP survey was run on 100 m spaced lines oriented east-west to trace the northern projection of the Hugo North Deposit. This initial IP survey used gradient array with 11,000 m AB electrode spacing, and covered an area extending 5.6 km north of the boundary and 10 km in width. Additional IP chargeability targets were also revealed 2.5 km to 3 km west of the Hugo North trend and are referred to as the Eagle anomalies, and at Ulaan Khuud South.

6.3 History of the Javkhlant Property

In 2005 and 2006, IMMI (now OT LLC) conducted IP surveying on 100 m spaced, east-west lines across Entrée's Javkhlant licence which is the joint venture "Earn-In" property south of the Oyu Tolgoi ML. This resulted in the discovery of three significant chargeability IP anomalies subsequently named the Sparrow South, Castle Rock, and SW Magnetic anomalies. Diamond drilling was initiated to test these IP anomalies in early 2007. Up to 2009, a total of 42 holes have been on the Javkhlant licence with 40 holes into the Sparrow South Anomaly, now renamed the Heruga Deposit two holes were also drilled on the Castle Rock anomaly and one hole on the south-west Magnetic anomaly (included in other drilling in Table 10.1). A first-time Mineral Resource estimate was completed for the Heruga deposit, in 2008.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

7.1.1 Regional Geology

The Oyu Tolgoi camp in southern Mongolia lies within the Palaeozoic Gurbansayhan Terrane, which consists of highly-deformed accretionary complexes and oceanic island arc assemblages (Badarch et al., 2002). The Gurbansayhan Terrane is itself a component of the Altaid orogenic collage, a continental-scale belt dominated by collisional tectonics related to Late Palaeozoic convergence and rotation of Neoproterozoic and pre-0.6 Ga cratonic blocks. In the region surrounding Oyu Tolgoi, the Altaid orogenic collage forms a broad corridor of major strike-slip faults, contractional fault and fold belts, and fault-controlled Mesozoic sedimentary basins. Major structures in this area include the Gobi Tien Shan sinistral strike-slip fault system, which splits eastward into a number of splays in the Oyu Tolgoi area, and the Gobi Altai Fault system, which forms a complex zone of sedimentary basins over thrust by basement blocks to the north and north-west of Oyu Tolgoi (Figure 7.1). To the east of Oyu Tolgoi, regional structures are dominated by the north-east-striking East Mongolian Fault Zone, which forms the south-eastern boundary of the Gurbansayhan Terrane. This regional fault may have formed as a major suture during Late Palaeozoic terrane assembly, with Mesozoic reactivation leading to the formation of north-easterly-elongate sedimentary basins along the fault trace.

The present morphology and structure of southern Mongolia largely reflect the influences of Mesozoic to Cenozoic intra-plate deformation events, superimposed on late Paleozoic orogenic features. The younger deformation episodes include:

- Late Jurassic to Cretaceous north-south extension, accompanied by the intrusion of granitoid bodies, unroofing of metamorphic core complexes, and formation of extensional and transpressional sedimentary basins.
- Superimposed north-east–south-west shortening associated with major strike-slip faulting and folding within the Mesozoic basins.

Figure 7.1 Location of Oyu Tolgoi, Gurvansayhan Terrane and Major Cenozoic Faults in Southern Mongolia

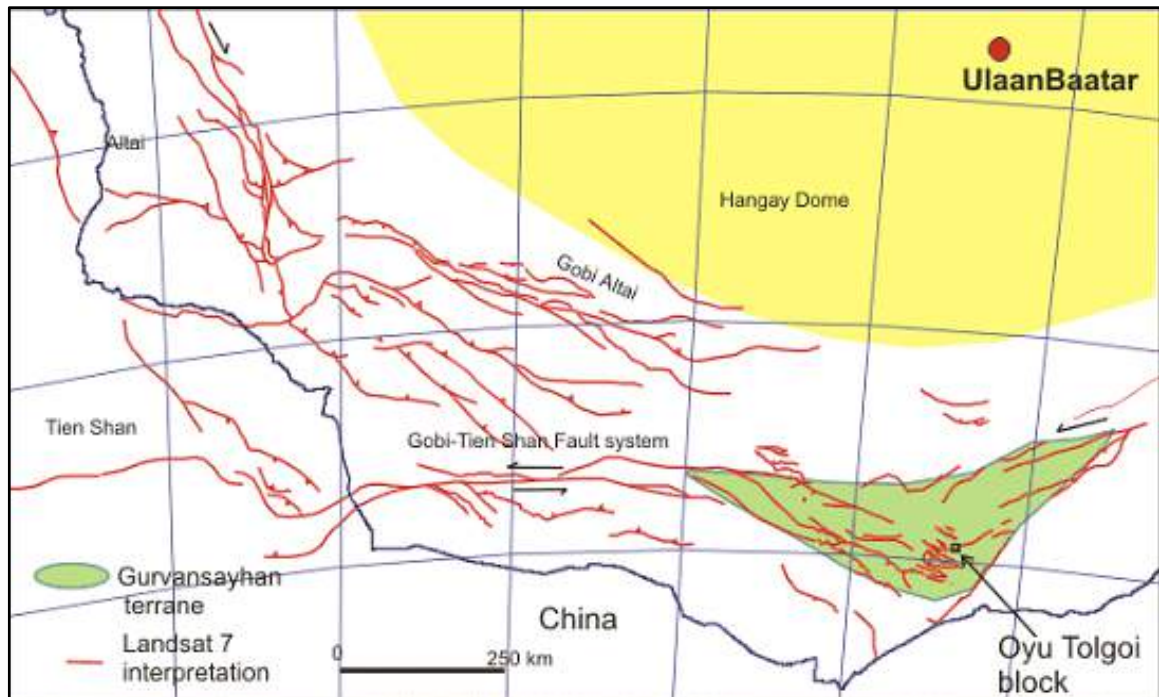


Figure provided by Ivanhoe.

7.1.2 Local Geology

7.1.2.1 Sedimentary and Volcanic Rocks

The Oyu Tolgoi area lies within an east-west trending belt of Devonian-Carboniferous volcanic and sedimentary rocks of continental margin and island arc affinities, constituting the South Mongolia Volcanic Belt. Two major stratigraphic sequences are recognized in the Project area (Figure 7.2):

- Tuffs, basaltic rocks, and sedimentary strata of probable island arc affinity, assigned to the Upper Devonian Alagbayan Formation.
- An overlying succession containing conglomerates, fossiliferous marine siltstones, sandstones, water lain tuffs, and basaltic to andesitic flows and volcaniclastic rocks, assigned to the Carboniferous Sainshandhudag Formation. The two sequences are separated by a regional unconformity that, in the Oyu Tolgoi area, is associated with a time gap of 10 Ma to 15 Ma.

A thin covering of gently-dipping to horizontal Cretaceous stratified clays and clay-rich gravels overlies the Palaeozoic sequence, infilling paleochannels and small fault-controlled basins.

Devonian Alagbayan Formation

The oldest rocks identified at Oyu Tolgoi are correlated with the Upper Devonian Alagbayan Formation. They subcrop in the area around and to the west of the South and Southwest Oyu deposits, and are intersected in drillholes in all of the deposit areas. Four major lithologic divisions are present within the Alagbayan Formation, DA1 to DA4, and each of these divisions comprises two or more mappable subunits. The lower most unit (DA1) is commonly strongly altered and a significant host to mineralization throughout the Oyu Tolgoi district. DA2 and DA3 unconformably overly DA1 and are weakly altered and mineralized only at the Heruga deposit. Unit DA4 is an overturned allochthonous unit that has been structurally emplaced over DA1 to 3, subsequent to mineralization.

Unit DA1: Basaltic Flows and Volcaniclastic Rocks

The stratigraphically lowest rocks identified to date at Oyu Tolgoi consist of mafic volcanic flows and volcanogenic sedimentary rocks, forming a sequence at least several hundred metres in thickness. These rocks are commonly strongly altered and host much of the contained copper in the Southern Oyu, Hugo Dummett, and Heruga deposits.

Three main lithotypes are present in the lower sequence. The lowest is a subunit consisting of grey to green finely-laminated volcanogenic siltstone and interbedded fine sandstone. These rocks are overlain by and partially interbedded with dark green, massive porphyritic (augite) basalt, characterized by up to 35% phenocrysts in a fine-grained pilotaxitic matrix of plagioclase and augite. The third principal rock type is a monolithic to slightly poly lithic basaltic lapilli tuff to volcaniclastic conglomerate/breccia, occurring above or partially interbedded with the massive augite basalt.

Unit DAba: Basaltic-Dacitic Hyaloclastite.

Unit DAba is within the DA4 sequence probably near the base. It is a hyaloclastite with intrusive basaltic trachyandesite sills (BTa), extrusive hyaloclastite breccias (HCBx) and Hyaloclastite tuffs (HCTu). A pelagic or turbiditic facies is interbedded with the extrusive parts. DAba is well developed in the Javkhant and east Heruga areas as well as west of the West Bat fault in the shaft farm area.

Unit DA2: Dacite Tuff/Volcaniclastic Rocks

DA2 unconformably overlies the basaltic rocks of unit DA1 and consists of dacitic welded and unwelded lapilli tuffs, ash tuffs, volcaniclastic breccias, polymictic breccias and conglomerates.

DA2a (CgIm) consists of an unsorted polymictic conglomerate, dominated by porphyritic andesitic volcanic clasts and is characterized by the presence of quartz vein fragments and less commonly altered and mineralized feldspar porphyritic intrusive clasts. Clasts at the base of DA2a are commonly highly angular and may represent an eruptive breccia that has been reworked and incorporated into conglomerate.

DA2b1 and DA2b2 represent pyroclastic deposits of dacitic composition, corresponding to dacitic block and ash tuff and dacitic ash flow tuff respectively. DA2b1 dacitic block and ash tuff is dominated by dacitic volcanic clasts that range to several 10s of cms in size. The DA2b2 dacitic ash flow tuff is characterized by prominent fiamme textures and a general lack of lithic fragments. DA2b1 and DA2b2 interfinger locally but DA2b1 generally occurs lower in the section.

DA2a overlies the unconformity at Heruga, the Wedge Zone and parts of Hugo South but is absent at Hugo North where DA2b1 directly overlies the unconformity.

A zircon U/Pb date of 367 ± 3 Ma from Unit DA2b2 rock overlying the Hugo Dummett deposit constrains the dacite sequence to Upper Devonian (Wainwright, 2008).

Unit DA3: Clastic Sedimentary Sequence

A clastic sedimentary sequence that overlies the dacite marks an important change in stratigraphic style that is recognizable throughout the Project area. This clastic sequence is only weakly altered and can be up to approximately 100 m in thickness. Two main rock types are present. A finer subunit consisting of rhythmically interbedded carbonaceous siltstone and fine brown sandstone forms a lithologically distinctive unit overlying the top of the dacite. This unit is ubiquitous in drillholes in Hugo North and is also discontinuously distributed in the more southerly deposits. A second subunit, characterized by poly lithic conglomerate, sandstone, and siltstone is abundant in the Southern Oyu deposits and parts of the Hugo South deposit. The conglomerate typically has a muddy matrix, and is transitional downward to boulder conglomerate and volcanic breccia at the top of the underlying dacite sequence.

Cross-cutting relationships with radiometrically-dated units constrain the age of these clastic strata to the Upper Devonian.

Unit DA4 Basaltic Flows/Fragmental Rocks, Siltstone

The structurally uppermost strata of the Alagbayan Formation at Oyu Tolgoi comprise a sequence of basaltic flows and volcaniclastic rocks, thinly-bedded siltstone, and massive sandstone (Unit DA4). These strata form a weakly altered to unaltered cover sequence to the Hugo Dummett deposit, averaging around 600 m thick, but do not presently overlie the more deeply eroded Southern Oyu deposits. Unit DA4 is separated from the underlying Alagbayan Group units by a contact-parallel fault. Sedimentary facing

direction indicators within the unit face downwards, implying that it is allochthonous with respect to the underlying, upright sequence. It has not been directly dated, and the stratigraphic relationship to the lower rocks is uncertain.

Basaltic composition, dark green volcanic breccia with vesicular, fine-grained to coarsely porphyritic basaltic clasts is the dominant lithotype in unit DA4. These breccias are commonly interstratified with volcanogenic sandstones and conglomerates that appear to be directly derived from the basalts. Middle parts of unit DA4 are dominated by thinly-interbedded red and green siltstone, which contain subordinate basalt layers in their lower levels. Structurally highest (stratigraphically lowest) strata of the unit are massive green to grey sandstone with rare siltstone interbeds.

(2) Lower Carboniferous Sainshandhudag Formation

The Lower Carboniferous Sainshandhudag Formation forms the upper of the two major stratigraphic packages at Oyu Tolgoi. The unit post-dates porphyry mineralization and is separated from the underlying Devonian rocks by a regional unconformity. Radiometric dates and biostratigraphic correlations suggest the unconformity spans a time gap of 10 Ma to 15 Ma, omitting most of the Famennian (Upper Devonian) and Tournaisian (Lower Carboniferous) stages.

The Sainshandhudag Formation is divided into three major units at Oyu Tolgoi: a lowermost tuffaceous sequence, an intermediate clastic package, and an uppermost volcanic/volcaniclastic sequence.

Unit CS1: Andesitic Lapilli Tuff and Volcaniclastic Rocks

Lowest strata within the Sainshandhudag Formation in the Oyu Tolgoi area consist of andesitic lapilli tuff with abundant fiamme, and subordinate block tuff to breccia. These rocks are characterized by a crowded feldspar phenocryst-rich matrix and angular lithic clasts, which are invariably fine-grained and non-porphyritic.

The andesitic tuff is widely distributed in the Oyu Tolgoi area and reaches up to 200 m in thickness. It is absent from the base of the Sainshandhudag Formation in several drillholes along the east side of the Hugo Dummett deposit, where higher units in the Sainshandhudag Formation unconformably overlie Devonian strata.

A U/Pb zircon date obtained from the Hugo Dummett deposit area indicates an age of 354 ± 2 Ma for the andesitic tuff/volcaniclastic unit (Wainwright, 2008).

Unit CS2: Conglomerate, Sandstone, Tuff, and Coal

A clastic sedimentary sequence overlies the lower andesitic package and is up to 200 m in thickness. The sequence typically shows a progression from a lower conglomerate–sandstone–siltstone-dominant unit to an overlying siltstone–waterlain tuff unit. Carbonaceous siltstone and coal beds occur in the lower part of the sequence.

Abundant marine and plant fossils constrain an Upper Tournaisian–Lower Visean (Lower Carboniferous) age for the unit.

Unit CS3: Basaltic and Andesite Lava and Volcaniclastic Rocks

The uppermost division of the Sainshandhudag Formation consists of a thick sequence of andesitic to basaltic flows and volcaniclastic rocks comprising several subunits. These subunits include, in order of superposition: a basal thin volcanic sandstone, a discontinuous porphyritic basaltic andesitic lava sequence, a thick basaltic breccia-to-block tuff unit, and an interstratified– to overlying-porphyritic basalt flow sequence. Together, these units can attain thicknesses of over 800 m.

7.1.2.2 Intrusive Rocks

Intrusive rocks are widely distributed through the Oyu Tolgoi area and range from large batholithic intrusions to narrow discontinuous dykes and sills. At least seven classes of intrusive rocks can be defined on the basis of compositional and textural characteristics. Copper–gold porphyry mineralization is related to the oldest recognized intrusive suite, comprising large Devonian quartz monzodiorite intrusions that occur in all of the deposit areas. Many of the older intrusive units found on the property are strongly to intensely altered (e.g. quartz monzodiorite suite), and the compositional classifications used for these units should therefore be considered only as field terms.

Basalt/Dolerite

Mafic dykes are the youngest intrusive rocks recognized on the Oyu Tolgoi property, and intrude all stratified units. They are typically aphanitic to fine-grained, locally vesicular, and contain variable amounts of plagioclase phenocrysts. In the deposit area, they are limited to dykes from metres to at most a few tens of metres wide, but in the south-west part of the property they also occur as large, sill-like intrusive masses. This class of intrusions includes a strongly magnetic, north-striking subvertical dolerite dyke that cuts across the Hugo Dummett deposit.

Rhyolite

Rhyolite dykes and sills are abundant throughout the property and measure up to a few metres to tens of metres wide, with strike extents of hundreds of metres. Except where emplaced as sills, they typically have steep dips, and strike orientations are variable. Texturally they are aphanitic and aphyric. Intrusive breccias are common along dyke contacts, commonly incorporating both rhyolitic and wall rock fragments within a flow-banded groundmass. U/Pb zircon dates obtained from the rhyolite dykes are 340 ± 3 Ma (Wainwright, 2008).

Hornblende–Biotite Andesite, Dacite

Hornblende–biotite andesite and dacite dykes occur in all of the deposit areas, but usually are less volumetrically significant than other intrusive units. Both units are part of a trachyte suite, which may be genetically related to flows and pyroclastic units of the Sainshandhudag Formation. They are typically strongly porphyritic with feldspar, hornblende, and biotite. Quartz phenocrysts are common within the dacite dykes. U/Pb zircon dates obtained from a hornblende biotite andesite dykes are 345 ± 2 Ma (Wainwright, 2008).

Biotite Granodiorite

Late- to post-mineral biotite granodiorite intrusions form a voluminous dyke system along the western side of the Hugo Dummett deposit, and more restricted dykes and sills elsewhere. The intrusions are compositionally and texturally varied and probably include several intrusive phases. Typically, they contain large plagioclase phenocrysts with lesser small biotite phenocrysts, within a fine-grained to aphanitic brown groundmass. In the Hugo Dummett deposit, the age of the biotite granodiorite is constrained by U/Pb dating of zircon to ~366 Ma (Wainwright, 2008).

Quartz Monzodiorite

Porphyry-style mineralization at Oyu Tolgoi is genetically linked to Late Devonian quartz monzodiorite to monzodiorite intrusions, which form the most voluminous intrusions in the deposit area. These intrusions are texturally and compositionally varied, and several distinct phases can be distinguished within the deposits. They are typically phenocryst-crowded, with >40% plagioclase phenocrysts up to 5 mm long, and 10 to 15% biotite and hornblende. The crystallization age of quartz monzodiorite has been interpreted at 372 ± 2 Ma (Wainwright, 2008). The form and characteristics of the quartz monzodiorite subunits are described in more detail in the individual deposit descriptions.

7.1.2.3 North-west Granitic Complex

The Oyu Tolgoi deposit area is bounded on the north-west by a large, polyphase granitic complex. Intrusive phases recognized in this complex include syenite, granite, quartz monzonite, quartz diorite, and quartz syenite. The temporal relationships between these different compositional phases have not been documented, and the only radiometric date obtained to date is a U/Pb zircon age of 350 Ma (Wainwright, 2008), implying that the complex post-dates mineralization at Oyu Tolgoi. The Northwest Granitic Complex is separated from older rocks of the deposit area by the north-east-striking, steeply north-west-dipping North Boundary Fault. The movement history of this fault is uncertain, and it may largely be the result of movement focused along the intrusive contact during post-intrusion deformation.

7.1.2.4 Khanbogd Complex

The Early Permian Khanbogd alkaline granite complex is a large, circular intrusion exposed just east of the Oyu Tolgoi property. The complex has a concentric structure defined by abundant pegmatite dykes enriched in rare earth elements and Zr. The Khanbogd Complex has a flat roof, as indicated by numerous basaltic wall rock roof pendants, and may therefore, have a “pancake” or lopolithic intrusive form.

7.1.3 Property Geology

The Oyu Tolgoi deposits are grouped into three areas from north to south: Hugo Dummett, Oyu South and Heruga. The following section is based on geological descriptions completed by AMEC (see Juras, 2003a, 2003b, 2003c, 2004, 2005; Hodgson 2004; and Blower, 2006a, 2006b), updated by Ivanhoe independent

consultant, Peter Lewis, P.Geo. in 2006 to 2007, and Ivanhoe Senior Geologist, Cyril Orssich, P.Geo. in 2009.

7.1.3.1 Structural Geology

The Project area is underlain by complex networks of faults, folds, and shear zones (Figure 7.3). Most of these structures are poorly exposed on surface and have been defined through integration of detailed exploration data (primarily drillhole data), property-scale geological mapping, and geophysical data. Ivanhoe has made extensive use of oriented core drilling at Oyu Tolgoi, and the structural data collected has been invaluable in helping determine the subsurface morphology and structural history of the Project area. Major structures in the Project area strongly influence the distribution of mineralization by both controlling the original position and form of mineralized bodies, and modifying them during post-mineral deformation events.

There is evidence for several phases of deformation and reactivation of early formed faults during later deformational events. North-north-east extension is evidenced in the west-north-west trending moderate north dipping 110 and Central faults. It has been suggested that thick accumulations of DA2b1 dacitic block and ash tuff on the north side of the 110 fault provides evidence of late Devonian syndepositional displacement.

The Contact fault and Lower Fault are low angle thrust faults that developed in a west-north-west–east-south-east compressive setting during late Devonian.

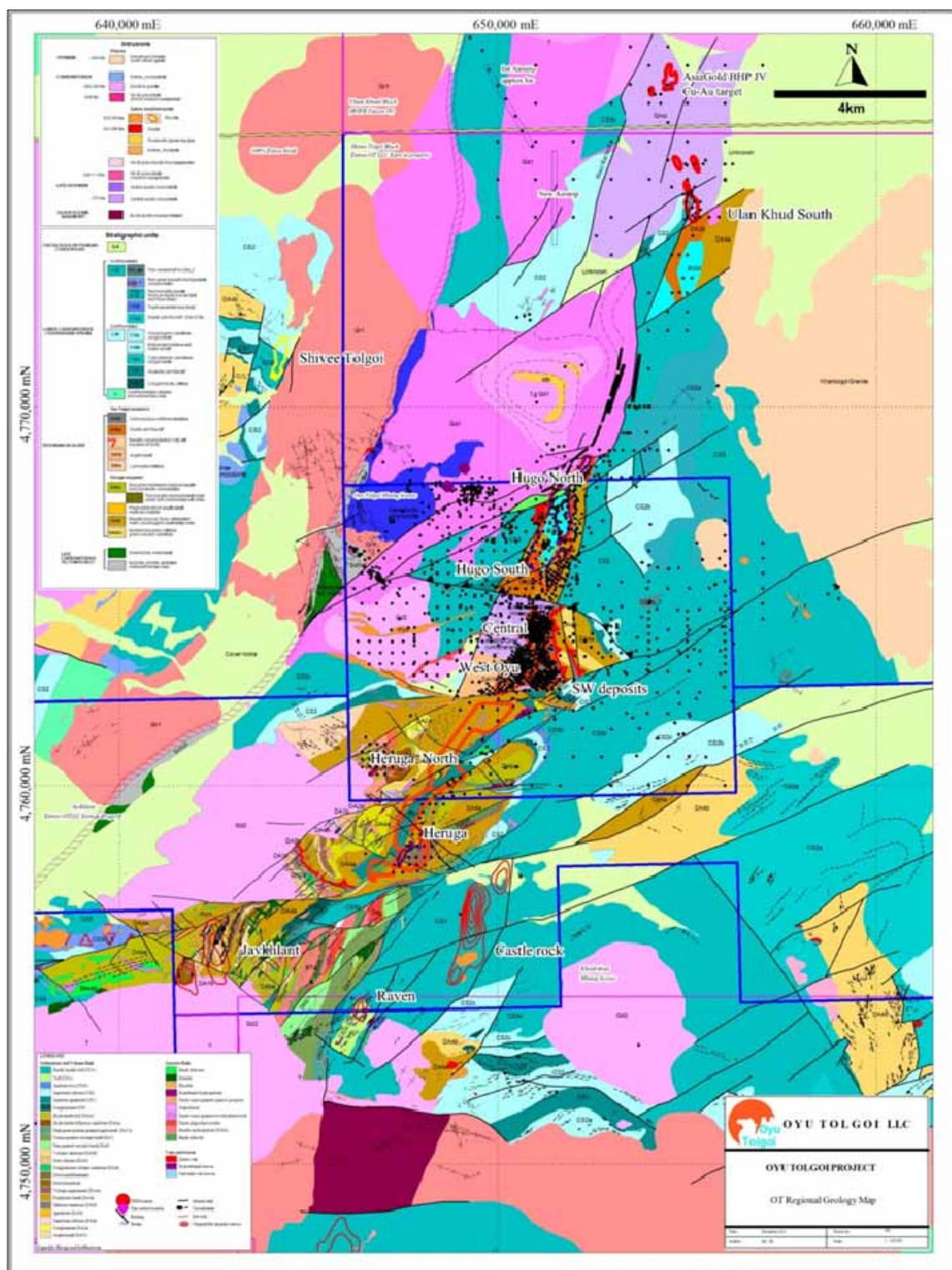
The north-north-east trend of the Oyu Tolgoi deposits is suggestive of a deep seated arc transverse zone of crustal weakness that focused magmatism and associated hydrothermal activity. Evidence of such a structure is not evident at the current crustal level of exposure or has been healed and concealed by alteration and mineralization. Nevertheless, there is ample evidence of syn to post mineral north-north-east striking structures that are interpreted to have formed within a west-south-west-east-north-east extensional setting and exhibit a sinistral transtensional sense of displacement.

Numerous east-north-east striking faults dissect the OT trend with dextral and variable vertical displacement and are assumed to be late post mineral structures.

(1) Central Fault

The Central Fault is a west-north-west-striking, moderately north-dipping structure that lies between the Hugo South and Central Oyu deposits. It is intersected by numerous drillholes in Hugo South, and its surface trace coincides with a strong linear magnetic anomaly. Stratigraphic distribution across the Central Fault shows conflicting stratigraphic relationships at different levels. The fault consists of several splays and may have experienced multiple periods of displacement. The simplest interpretation of geological relationships is that early fault displacement resulted in north-side-down apparent offset, followed by a later apparent reverse displacement of lesser magnitude. Stratigraphic evidence suggests the Central Fault may have been active as a Late Devonian growth fault. However, the fault is clearly visible as a linear magnetic feature cutting the overlying Sainshandhudag Formation, implying that movement continued into the Carboniferous or later.

Figure 7.3 Geological Map of the Project



(2) Contact Fault

The Contact fault is a low angle thrust fault generally parallel to bedding. It occurs throughout the OT district from the Heruga deposit in the south to the Hugo deposits in the north. It places overturned upper Devonian sedimentary and volcanic rocks belonging to the DA4 unit over upright Devonian sediments of unit DA3b. The thrust fault occurs completely in the unmineralized Devonian sequence overlying the deposits and nowhere does it truncate mineralization.

(3) North-north-east Striking Faults

(3).1 Axial Fault, West Bat, East Bat Faults

The alignment of the Southwest Oyu, Central Oyu, and Hugo Dummett deposits, along with the elongate form of the Hugo Dummett deposit itself, strongly implies that an underlying north-north-east-striking fault or fault zone (referred to as the Axial Fault) controlled emplacement of porphyry intrusions and related hydrothermal activity. For example, the tabular, north-north-east-elongate form of the biotite granodiorite dyke complex that follows the west side of the Hugo deposit may reflect fault-controlled intrusion. The axial fault is at this point hypothetical, and has not been specifically identified in drillholes. At the deposit scale, evidence for such a fault would have been obliterated by the porphyry intrusions exploiting it.

The Hugo Dummett deposit lies within a north-north-east-trending structural high bounded by the West Bat and East Bat Faults. Although the latest movement on these bounding faults displaces the Carboniferous post-mineral strata, they may represent the shallow expression of a longer-lived, deposit-controlling fault zone. Offsets of post-mineral stratigraphic contacts measure at least a kilometre (east-side-up) for the West Bat Fault, and 200 to 300 m (west-side-up) for the East Bat Fault. For both of these structures, the true slip direction is uncertain, and if there is a large strike-slip component of movement, total displacement may be significantly greater than stratigraphic offset.

(3).2 East Bounding, West Bounding, Bor Tolgoi, and Bor Tolgoi West Faults

Magnetic and satellite images show a strong north-east to north-north-east linear structural fabric cutting across parts of the Oyu Tolgoi property. Many of these lineaments can be correlated with faults identified in drillholes (e.g. East Bounding and West Bounding faults at the Southwest Oyu deposit).

The East Bounding and West Bounding Faults at the Southwest Oyu deposit form a primary control on the distribution of copper and gold mineralization, and the presence of mineralized clasts within the fault zones implies that they were also active following mineralization. Other north-east-striking faults displace stratigraphic contacts in the Lower Carboniferous Sainshandhudag Formation near the South Oyu deposit.

The Bor Tolgoi and Bor Tolgoi West Faults at the Heruga deposit, display 300 to 500 m of west side down apparent offset of stratigraphic contacts. Kinematic indicators have not been observed and the true displacement direction and magnitude is unconstrained.

(4) East-north-east Striking Faults

(4).1 Boundary Fault System at Hugo North

Roughly coincident with the northern boundary of the Oyu Tolgoi concession, an east-north-east-striking fault zone juxtaposes the Devonian sequence hosting and overlying the Oyu Tolgoi deposits against the Carboniferous granitic complex to the north. Faults within this system include the North Boundary Fault, an unnamed splay of the North Boundary Fault, and the Boundary Fault. These faults dip steeply to the north or north-west, and occur as strongly-developed, foliated gouge and breccia zones ranging from decimetres to several tens of metres wide. They juxtapose younger rocks (north block) over older (south block), but true displacement direction and magnitude are poorly constrained.

(4).2 Northwest Shear Zone

The Northwest Shear Zone is a wide, ductile shear zone that cuts across the far north-west corner of the Project area (Figure 7.3). This shear zone consists of mylonitic to ultramylonitic rocks in the center, grading outward over about 200 m to rocks lacking visible ductile strain. Together with the Boundary Fault, the shear zone marks the break between the volcanic and sedimentary strata hosting and surrounding the Oyu Tolgoi deposits (Alagbayan and Sainshandhudag Formations), and the Carboniferous granitic complex exposed on the north-western corner of the property. A ubiquitous subhorizontal stretching lineation, asymmetric mesoscopic fabrics (shear bands, asymmetric porphyroclasts), and map-scale deflections of dykes indicate that the Northwest Shear Zone accommodated dominantly dextral strike-slip movement. The magnitude of displacement is not closely constrained, but the geometry of deformed dykes in the shear zone indicates that at least several kilometres of displacement have occurred.

Deformed quartz-feldspar porphyry dykes in the shear zone are geochemically identical to the rhyolite intrusions in the deposit area, and have produced a U/Pb age of 335 ± 1 Ma, placing a maximum age on shear zone deformation; no minimum age is constrained.

(4).3 Solongo Fault

The Solongo Fault is an east- to east-north-east-striking, subvertical structure that cuts across the Oyu Tolgoi property just south of the Southwest Oyu and South Oyu deposits. The Solongo Fault typically occurs as a strongly tectonized, foliated zone up to several tens of metres wide. Rhyolite dykes commonly intrude the fault zone, and themselves have tectonically brecciated margins. On ground magnetic data, the fault forms a major linear anomaly that can be traced across the entire width of the Oyu Tolgoi property.

The Solongo Fault forms a major structural break, and there is a minimum of approximately 1,600 m of south-side-down stratigraphic offset where it juxtaposes mineralized basalt (unit DA1) in the South Oyu deposit against sediments correlated with the upper Alagbayan Formation (unit DA4) to the south. Depending on movement direction, net slip may be considerably greater. The displacement history of the Solongo

Fault is poorly constrained. The fault clearly existed prior to emplacement of rhyolite dykes, which are part of a suite dated at 340 ± 3 Ma. Rhyolite dykes locally cross the fault with little or no apparent displacement, implying that movement since their emplacement was negligible.

(4).4 East-north-east Striking Bounding Faults at Heruga

The Heruga deposit area lies between two regional, east-north-east-striking faults that form prominent features on both magnetic and satellite images. Detailed geological mapping shows a 500 m apparent dextral displacement of dykes and stratigraphic contacts across the faults.

(5) Folds

Variations in bedding attitude recorded in both oriented drill core and surface outcrops define two orientations of folds on the Oyu Tolgoi property: a dominant set of north-east-trending folds, and a less developed set of north-west-trending folds. These folds are well defined in bedded strata of both the Sainshandhudag and Alagbayan formations. They may be present in stratified rocks throughout the property, but outcrop and drillhole data are insufficient to define them in many areas.

Together, the two orientations of folds form a dome-and-basin interference pattern, but it is not possible to determine their relative ages. Both of the dominant fold orientations occur in Lower Carboniferous strata, indicating that both folding events post-date mineralization.

Sedimentary facing direction indicators in drillholes along the east flank of the Hugo Dummett deposit indicate that parts of the Alagbayan Formation are overturned. The overturning is interpreted to be associated with tight to isoclinal folds and contact-parallel thrust faults. These folds are cut by dykes of the 366 Ma biotite granodiorite suite, and therefore were formed within the Late Devonian.

7.1.3.2 Southern Oyu Deposits

The Southern Oyu deposits include the Southwest Oyu or Southwest, South Oyu or South, Wedge Zone or Wedge, and Central Oyu or Central deposits. These deposits form contiguous zones of mineralization representing multiple mineralizing centres, each with distinct styles of mineralization, alteration, and host rock lithology. The boundaries between the individual deposits coincide with major fault zones (Figure 7.3). Strong, high-sulfidation mineralization and associated advanced argillic alteration, hosted by basaltic fragmental volcanics and quartz monzodiorite, are characteristic of the Central and Wedge deposits. The mineralization grades downward into chalcopyrite–gold mineralization with associated biotite–chlorite alteration hosted within massive augite basalt. The overlying fragmental basalt sequence was previously thought to be dacitic in composition. Whole rock geochemistry has shown it to be similar in composition to the underlying basalt and has been affected by advanced argillic alteration to give it the appearance of a dacitic tuff. At Southwest, the basaltic fragmental volcanic and overlying strata have been removed by erosion, exposing deeper-level chalcopyrite–gold mineralization with associated biotite–chlorite alteration, hosted within massive augite basalt. Mineralization at the South deposit is chalcopyrite–bornite dominant with

associated biotite–chlorite alteration, and is hosted within quartz monzodiorite, massive and fragmental basaltic volcanic.

(1) Southwest Oyu Deposit

(1).1 Host Rocks

The Southwest Oyu or Southwest deposit is a gold-rich porphyry system characterized by a south-west-plunging, pipe-like geometry with over a 700 m vertical extent. Over 80% of the deposit is hosted by massive to fragmental porphyritic basalt of the Upper Devonian Alagbayan Formation, with the remainder hosted by intra-mineral, Late Devonian quartz monzodiorite intrusions. The quartz monzodiorite intrusions form irregular plugs and dykes related to several distinct phases. These include:

- Early strongly-altered quartz veined dykes mainly limited to the high-grade central deposit core (informally referred to as OT–Qmd).
- Superimposed younger fragmental dykes entraining early quartz vein clasts but lacking strong sulfide mineralization (informally referred to as xQmd).
- Voluminous massive quartz monzodiorite containing weaker mineralization, flanking and underlying the high-grade deposit core.

Several phases of post-mineral dykes cut the Southwest deposit. Most of the dykes belong to the rhyolite, hornblende–biotite andesite, or biotite granodiorite intrusive phases. They commonly have steep dips, and many are localized along faults. The rhyolite dykes tend to have west to west-north-west strikes in the deposit core and north-east strikes when emplaced along major faults. Hornblende–biotite andesite dykes strike east-north-east except where they intrude along the major north-east-trending faults.

(1).2 Structural Geology

Most of the Southwest deposit, and the entire high-grade, gold-rich core of the deposit, lies between two north-east-striking faults, the West Bounding Fault and the East Bounding Fault. Both faults are clearly defined on ground magnetic images, and their positions and orientations are well constrained by numerous drillhole intersections.

The bounding faults consist of foliated cataclasite, gouge/breccia, and mylonitic bands in zones ranging from a few metres to a few tens of metres wide. Both faults have subparallel splays locally. Correlation of drillhole intersections constrains an average fault dip of 80° towards 310° for both faults. The East Bounding Fault juxtaposes younger rocks to the south-east against the Alagbayan Formation rocks (augite basalt) hosting the deposit, while the West Bounding Fault is mainly intraformational within the augite basalt. The West Bounding Fault is commonly intruded by hornblende–biotite andesite dykes, whereas rhyolite dykes are more common along the East Bounding Fault.

The cataclasite within the fault zones contains abundant quartz, quartz sulfide, and sulfide (pyrite, chalcopyrite, sphalerite, and galena) clasts in a comminuted matrix that is locally overprinted by fine-grained pyrite and chalcopyrite. These relationships imply that at least some of the fault movement was contemporaneous with mineralization.

Kinematic indicators within the fault zones imply dominantly subhorizontal, sinistral movement on the bounding faults.

Quartz-dominant veins with variable amounts of sulfide (pyrite, chalcopyrite, bornite), K-feldspar, chlorite, and carbonate are ubiquitous in the Southwest deposit, and there is a general correlation between vein density and copper and gold grades. Most veins have widths of several millimetres to several centimetres, although within the core of the deposit veins up to a metre or more thick occur. Vein contacts can be either planar or variably deformed, and folded and/or faulted veins are common. Veins within the high-grade deposit core display subparallel to sheeted forms with a preferred south-west-dipping orientation. These pass into more irregularly oriented stockwork veins in peripheral mineralized zones, where subvertical north to north-west-striking orientations are most common.

Fault geometry and kinematics, vein orientations, and deposit geometry at Southwest support a structural model invoking deposit formation in a dilational fault transfer zone. This zone is delineated by the West Bounding Fault on the north-west and the East Bounding Fault on the south-east. The preferred vein orientation within the core of the deposit reflects the local stress geometry within this zone of dilation. The Southwest deposit probably formed as a subvertical cylindrical body and attained its present west-south-west plunge during post-mineral regional deformation. This post-mineral rotation is consistent with the easterly stratigraphic dips of both pre- and post-mineral rocks in the deposit area.

(2) South Oyu Deposit

(2).1 Host Rocks

The South Oyu zone occurs within an east to north-east-dipping sequence of Alagbayan Formation strata (basalt and basaltic tuff units), intruded on the south-west by an irregular quartz monzodiorite body. Much of the basalt sequence contains fragmental textures with juvenile pyroclasts and is texturally similar to the overlying basalt tuff sequence. The overlying basaltic tuff sequence was previously interpreted to be dacitic in composition. However, whole rock geochemistry has shown the basaltic tuff sequence to be similar in composition to the underlying basalt and that the unit has been affected by advanced argillic alteration to give it the appearance of a dacitic tuff.

To the north-east, the altered and mineralised rocks are overlain by mudstones and conglomerates of the upper Alagbayan Formation, which pass up-section into the overlying basalt and sediment sequence and ultimately into rocks of the Sainshandhudag Formation. The deposit area is cut by numerous barren dykes, most of which belong to the post-mineral rhyolite and basalt intrusive suites. These dykes typically have widths of only a few metres, with the exception of a major, east-west rhyolite dyke that cuts through the middle of the South deposit and attains widths of up to a few tens of metres. This wide dyke commonly balloons into larger intrusive masses where it intersects the South and Solongo faults. Although irregular in form, the rhyolite dykes have roughly west to west-north-west strikes and steep dips. In contrast, the basalt dykes have moderate north-easterly dips, subparallel to contacts within the stratified host rocks.

(2).2 Structural Geology

The South deposit lies within a faulted block bound on the north-west by the north-east-striking South Fault, and on the south by the east-north-east-striking Solongo Fault. The South Fault forms a zone with several strands over a width of up to 90 m, which juxtapose progressively younger strata on the north-west against older strata to the south-east. Drillhole intersections of these faults typically consist of gouge and breccia zones up to several metres wide. To the west, the faults strike into a large quartz monzodiorite intrusion. The faults are difficult to trace through the intrusion and offset of the intrusive contact is minimal, implying that most movement pre-dated emplacement of the quartz monzodiorite.

The Solongo Fault truncates the southern edge of the South deposit. It forms a wide, strongly tectonized zone. Stratigraphic offset on the Solongo Fault is at least 1,600 m. No significant mineralization has been identified on the south side of the fault.

Copper mineralization in the South deposit is associated with stockworks of thin (usually <10 cm) quartz-sulfide veins. In surface exploration pits and trenches, veins occur as steep, north-west-striking, strongly sheeted sets. However, veins intersected in drillholes have a stockwork style and lack the strong preferred orientation visible in surface exposures.

(3) Wedge Deposit

(3).1 Host Rocks

The Wedge deposit occurs within a north-east-dipping sequence of Upper Devonian Alagbayan Formation strata similar to that hosting the adjacent South deposit. However, in the Wedge deposit, the basaltic tuff unit is significantly thicker (up to 180 m) than at South and forms the dominant host to copper mineralization. On the north-east, structurally overlying non-mineralized rocks of Alagbayan Formation unit DA4 and the lower Sainshandhudag Formation form the upper limit of mineralization.

Mineralized rocks in the Wedge deposit are cut by numerous barren dykes, including biotite granodiorite, hornblende-biotite andesite, and rhyolite compositions. Biotite granodiorite and hornblende-biotite andesite are more common along the north-west margin of the deposit and typically have north-east strikes, parallel to the East Bounding Fault. These intrusions are also common as sills, frequently intruding along the stratigraphic contact between the basaltic tuff and the overlying sedimentary strata. Rhyolite dykes are common throughout the deposit. They typically have steeply-dipping contacts but varied strike orientations.

(3).2 Structural Geology

The Wedge deposit occupies a rectangular fault block bounded on the west by the north-east-striking East Bounding Fault and on the south by the east-north-east-striking South Fault. Within this block, stratigraphic contacts are continuous and relatively planar, showing little evidence of structural disruption.

Movement on the East Bounding and South Faults has juxtaposed younger strata within the fault block hosting the Wedge deposit against older strata on the adjacent blocks containing the Southwest and South deposits. Stratigraphic contacts are relatively continuous between the Wedge deposit and the Central deposit, implying that displacement on the East Bounding Fault is largely transferred to the Rhyolite Fault (between Southwest and Central), leaving the Wedge and Central deposits as a structurally intact block that has been displaced downward relative to the Southwest and South deposits.

Fault disruption is common along the contact between the Alagbayan Formation basaltic tuff and the overlying sedimentary strata. However, there is no evidence of significant stratigraphic omission or repetition associated with this faulting, and the movement may be relatively minor.

(4) Central Oyu Deposit

(4).1 Host Rocks

The Central Oyu or Central deposit is hosted within a swarm of feldspar-phyric quartz monzodiorite intrusions, emplaced into porphyritic augite basalt and overlying basaltic tuff of the Alagbayan Formation. The basaltic tuff is in turn overlain by unmineralized sedimentary and mafic volcanic rocks of the Alagbayan Formation unit DA4, which currently dip moderately to the east.

Several phases of intra-mineral and late-mineral quartz monzodiorite intrusions have been distinguished in the Central deposit based on textural variations and intensity of mineralization and alteration. Most have dyke forms, emanating from a larger intrusive mass to the north and west of the deposit area. The quartz monzodiorite dykes terminate within the base of the sedimentary units of the upper Alagbayan Formation.

Basalt flows and basaltic tuffs of the Alagbayan Formation are preserved as a series of isolated, irregular, moderately north- to north-east-dipping bodies within the quartz monzodiorite dyke swarm. These volcanic windows are up to 200 m thick and extend several hundred metres down dip to the limit of drilling. The contact between the basaltic tuff and the overlying sedimentary sequence is commonly faulted and forms the upper limit to mineralization, as elsewhere in the Oyu Tolgoi district.

Post-mineral dykes are common in the Central deposit and comprise rhyolite, biotite granodiorite, hornblende–biotite andesite, and dacite dykes. The rhyolite dykes are most abundant, with the majority occurring as west- and west-north-west-striking bodies in the southern half and on the periphery of the deposit. Biotite–granodiorite dykes occur along the eastern margin of the deposit and tend to strike north to north-north-east. East-north-east striking hornblende–biotite andesite dykes occur mainly along the north-eastern margin of the deposit.

(4).2 Structural Geology

Drillholes through the Central deposit show little evidence of significant post-mineral faulting, and the mineralogical zoning, grade distribution, and continuity of contacts are consistent with the deposit being contained in a structurally intact block. Most contacts in the deposit are intrusive or stratigraphic, although minor faulting occurs locally. Post-mineral faults form minor zones of breccia and cataclasite in some drillholes, but it is not possible to correlate these intersections between drillholes to define continuous fault surfaces. Pre- or syn-mineral faults, if present, are largely obscured by intrusive and hydrothermal overprinting.

The Central deposit area is overlain to the east by non-mineralized conglomerate, mudstone, and siltstone of Alagbayan Formation unit DA4. Wide zones of breccia and foliated breccia occur along the basal contact of and within the lower portion of these sedimentary strata. The displacement history of these faults is uncertain, and they may be related to minor post-mineralization movement between the two rheologically contrasting rock packages.

Along its southern margin, the Central deposit is juxtaposed against the Southwest deposit area by an east-west-striking fault that is now occupied by a rhyolite dyke (the Rhyolite Fault). The basaltic tuff and overlying sedimentary units have been uplifted and eroded from the block south of this fault.

Mineralized veins within the Central deposit show a range in orientations, the most common including south-west-, west-, and north-west-dipping attitudes. Vein orientations are similar to those documented in the Southwest deposit, although the degree of preferred orientation in the deposit core is weaker at Central. Similar preferred vein orientations at Central and Southwest suggest that the two deposits were formed in a similar structural regime. However, the Central deposit lacks the strong bounding fault control that is fundamental to the form and geometry of the Southwest deposit. This lack of bounding fault control may account for the more irregular form of the mineralized body at Central.

Post-mineral tilting of the Central deposit is implied by bedding dips in the enclosing and overlying stratigraphic sequence. Rotating the structural data for the Central deposit sufficiently to restore bedding to horizontal indicates a strong preference for subvertical veins within the deposit at the time of formation.

7.1.3.3 Hugo Dummett Deposits

The Hugo Dummett deposits contain porphyry-style mineralization associated with quartz monzodiorite intrusions, concealed beneath a deformed sequence of Upper Devonian and Lower Carboniferous sedimentary and volcanic rocks. The deposits are highly elongated to the north-north-east and extend over 3 km (Figure 7.3). Although mineralization is continuous over this entire length, it thins markedly and decreases in grade where the host strata are displaced by an east-west-striking, north-dipping fault, termed the 110 Fault. This fault defines the boundary between the Hugo South and the Hugo North deposits. The depth to the top of the high-grade (>2.5% Cu) zone varies from 300 m at Hugo South to about 900 m at Hugo North.

(1) Hugo South Deposit

(1).1 Host Rocks

The Hugo South deposit is hosted by an easterly-dipping sequence of volcanic strata correlated with the lower part of the Devonian Alagbayan Formation, and by quartz monzodiorite intrusive rocks. Stratigraphically lowest rocks in the sequence consist of porphyritic (augite) basalt flows and minor volcanoclastic strata. These are overlain by basaltic tuffs and breccias forming a sequence varying from 100 m to 200 m thick. The basaltic fragmental sequence was previously thought to be dacitic in composition. Whole rock geochemistry has shown it to be similar in composition to the underlying basalt and has been affected by advanced argillic alteration to give it the appearance of a dacitic tuff. Weakly altered to unaltered sedimentary and volcanic rocks of the upper Alagbayan Formation and Sainshandhudag Formation structurally overlie the mineralized sequence along the eastern flank of the Hugo South deposit. The thickness of the non-mineralized Alagbayan Formation sequence commonly exceeds 600 m, although structural thickening may occur within the sequence. The Sainshandhudag Formation strata unconformably overlie, and are locally faulted against, the Alagbayan Formation.

Several phases of intrusive rocks occur in the Hugo South deposit. The oldest recognized intrusions are quartz monzodiorite bodies, which underlie the entire deposit area and contain low copper grades. Quartz monzodiorite contacts are irregular, but overall show a preferred easterly dip, subparallel to contacts in the enclosing stratified rocks. The quartz monzodiorite is broadly contemporaneous with alteration and mineralization, and two varieties are distinguished on the basis of alteration characteristics and position within the deposit:

- An intensely quartz veined phase that occurs along the upper margin of the main intrusive body or as a separate east-dipping tabular body in the overlying strata.
- A lower-grade, more weakly veined variety, which makes up the large intrusive body forming the lower part of an underlying the entire deposit.

Late- to post-mineral biotite granodiorite intrusions form a north-north-east-striking dyke complex cutting across the western edge of the deposit. Correlations between drillhole intersections and measurements of individual contacts indicate that dyke contacts have a moderate to steeply west-dipping preferred orientation.

Younger intrusions include rhyolite, hornblende–biotite andesite, dacite, and basalt/dolerite compositional varieties. These intrusions usually occur as dykes with subvertical orientations, or less commonly as easterly-dipping sills emplaced along stratigraphic contacts. They are non-mineralized and not volumetrically significant except locally in the deposit.

A typical section through the Hugo South Deposit is included as Figure 7.4.

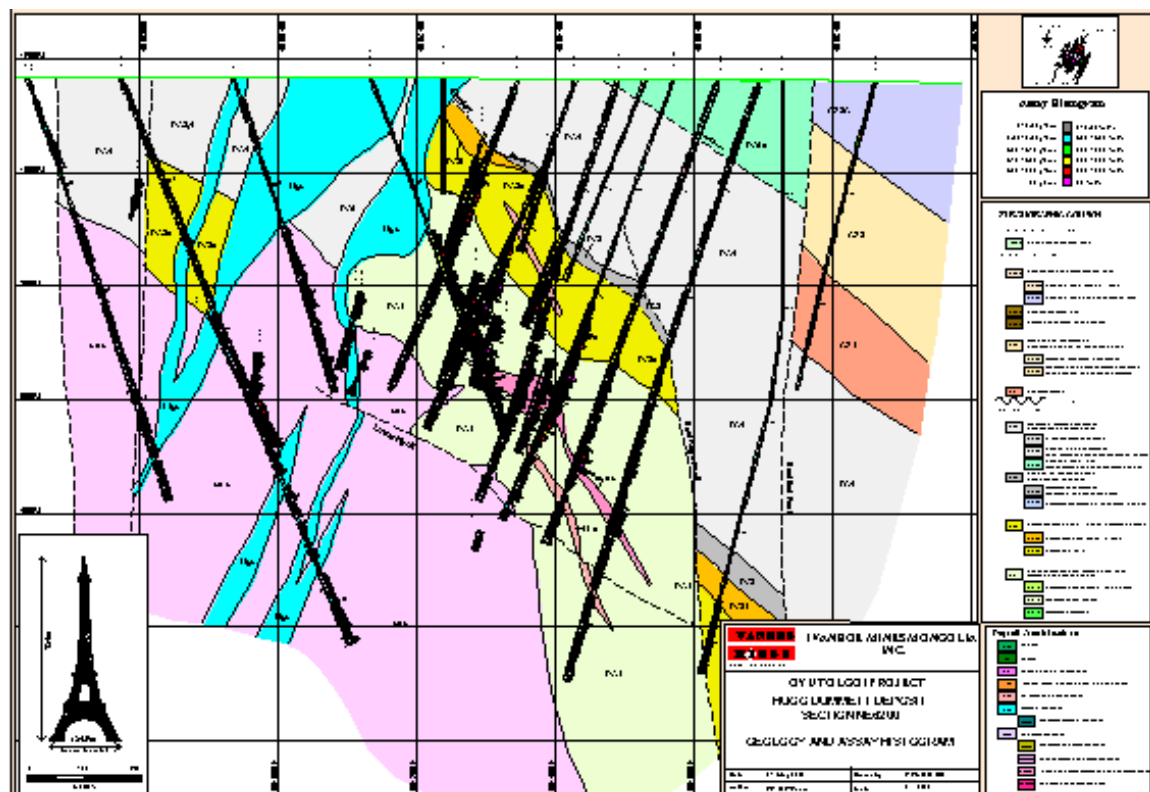
(1).2 Structural Geology

The Hugo South deposit occurs within a north-north-easterly elongate block bounded on the north and south by moderately north-dipping faults, and on the east and west by steep, north-north-east-striking faults. Strata within the block form a homoclinal sequence dipping moderately to the east-south-east.

Deformation of the Hugo South deposit is dominated by brittle faulting. Major faults cutting the deposit can be grouped on the basis of orientation into four sets:

- East-west-striking, moderately north-dipping faults (110, Central Faults).
- Steep north-north-east-striking faults (East and West Bat, East Hugo, and Axial Faults).
- North-north-east-striking faults that dip moderately east, subparallel to lithologic contacts (Contact, Lower Faults).
- East-west-striking, subvertical faults (East-West Fault).

Figure 7.4 Hugo South Deposit Geology, Section 6200 N (Looking North)



Note: Shows distribution of major lithologic units, principal faults, and Cu mineralization. Figure provided by Ivanhoe.

110 Fault

The 110 Fault defines the division between the Hugo North and Hugo South deposits, although mineralization is continuous across the fault. The fault strikes east-west and dips northerly at approximately 45° to 55°. In drillhole intersections, the 110 Fault consists of zones of non-cohesive gouge and breccia up to several metres thick. Tectonic fragments within the fault zone are derived from adjacent wall rocks and include mineralized quartz veins within strongly foliated clay gouge.

The 110 Fault juxtaposes younger strata on the hanging wall block (north) against older strata on the footwall block (south). The stratigraphic offset of the base of the upper member of the Alagbayan Formation varies between approximately 100 m and 200 m. Depending on the net slip direction, the true offset may be greater. The fault is truncated up-dip by the East-West Fault, and the intersection between the two fault surfaces is subhorizontal. Because it is truncated by the East-West Fault, the 110 Fault does not have a surface trace in the Hugo Dummett deposit area.

Stratigraphic evidence suggests that earliest movement on the 110 Fault had a large north-side-down component and may have been syn-depositional with respect to the strata immediately overlying the ignimbrite unit (unit DA2a). In particular, in several areas, coarse volcanic breccias (block and ash tuff unit DA2b) are thicker in drillholes immediately north of the fault than on the southern fault block. Additional evidence for early movement on the 110 Fault is the lack of offset of the biotite granodiorite dyke contact where the two intersect, implying that most of the stratigraphic displacement pre-dates dyke emplacement.

Kinematic indicators in the foliated gouge indicate oblique sinistral–reverse movement on the 110 Fault, contrary to the apparent stratigraphic displacement. These fabrics likely record a late period of movement that is of lesser apparent magnitude than earlier displacements. Cataclasite developed during this stage of movement contains tectonic fragments of veins with copper sulfides, indicating post-mineral timing. However, the late fault movement does not significantly displace mineralization.

Central Fault

The Central Fault is a shallowly to moderately north-dipping structure that lies beneath the southern portion of the Hugo Dummett deposit. It projects to surface between the Hugo South and Central deposits. Drillhole intersections indicate a strike of approximately 90° to 100°, and its surface trace coincides with a strong linear magnetic anomaly.

At shallow levels, stratigraphic contacts are offset across the Central Fault with apparent normal displacement. At deeper levels, the opposite stratigraphic relationships occur: dacite tuff of the Alagbayan Formation is faulted over sedimentary strata correlated with the upper Alagbayan Formation. In drill core, the Central Fault consists of a zone of fault breccia and gouge that can be up to several metres thick.

Contradictory stratigraphic relationships (stratigraphic repetition at deeper levels, stratigraphic omission at shallow levels) imply that the Central Fault may have experienced multiple periods of displacement, similar to the history proposed for the

110 Fault. The simplest interpretation is that early displacement resulted in north-side-down apparent offset, followed by later reverse displacement during a period of roughly north-south contractional deformation. The magnitude of reverse displacement was insufficient to restore the entire apparent normal offset. At depth, the reverse sense reactivation may have been localized along a splay in the hanging wall of the principal fault surface, resulting in local stratigraphic repetition.

East-West Fault

The East-West Fault cuts across and displaces the northern end of the Hugo South deposit. Drillhole intersections constrain the fault orientation to subvertical with a strike of approximately 080° to 090°. It forms an abrupt boundary to alteration and mineralization in several sections. The East-West Fault also cuts biotite granodiorite dyke contacts, the Contact Fault, the East Hugo Fault, the 110 Fault, and the East Bat Fault. In drillhole intersections, the East-West Fault occurs as a zone of clay-rich breccia and locally foliated gouge up to several metres in width, similar in character to the Central and 110 faults. Narrow basaltic dykes commonly occur within the fault zone.

Offsets of a late basaltic dyke, stratigraphic contacts, and the axis of the Hugo South deposit constrain an oblique dextral–north-side-down net slip of roughly 150 to 200 m on the fault, along a gently eastward-plunging slip vector.

Deposit-Parallel Faults

Moderately east-dipping (deposit-parallel) faults that occur within and immediately adjacent to the Hugo Dummett deposit include the Contact Fault and the Lower Fault. The Contact Fault is a bedding-parallel detachment zone that normally occurs at the contact between tectonized mudstones stratigraphically overlying the deposit (Unit DA3), and overlying basalt flows and volcanoclastic rocks (unit DA4). The hangingwall sequence contains downward-facing sedimentary structures, indicating that the fault juxtaposes a large, overturned fold limb over the upright deposit rocks. The intensity of fabric development along this fault is highly variable: in some drillholes it occurs as a wide zone of anastomosing foliation and brecciation in carbonaceous mudstones while elsewhere there is little tectonic disruption along the contact. The Contact Fault is often intruded by dykes or sills of basaltic, andesitic, or rhyolitic composition. Bedding above and below the faulted contact is concordant.

In the western part of the Hugo Dummett deposit, the biotite granodiorite dyke complex cuts across the Contact Fault at a high angle. The fault continues as a narrow zone of breccia and clay gouge in the biotite granodiorite, but is difficult to identify in many drillholes. Although there is probably some displacement of the intrusive contact, the magnitude is too small to be calculated from the present drillhole spacing.

The Lower Fault occurs as an intensely brecciated, clay gouge-rich zone within the middle or lower portion of the mineralized body, typically 200 to 400 m below the Contact Fault. Similar to the Contact Fault, the Lower Fault can be traced westward through at least part of the biotite granodiorite dyke as a narrow zone of breccia and gouge.

Latest movement on the deposit-parallel faults post-dates both mineralization in the Hugo South deposit and intrusion of the biotite granodiorite dykes. It is difficult to clearly demonstrate geological offsets on the Lower Fault, in part because the fault orientation is close to that of both stratigraphic and alteration contacts. However, in the Hugo North deposit, the Lower Fault appears to displace the Hugo North gold zone and biotite granodiorite dykes by up to 400 m.

The contact-parallel nature of the Contact and Lower Faults suggests that they may have formed as gently-dipping thrusts during regional contractional deformation. These faults were localized along contacts between units with differing competencies, or along relatively weak layers within the stratigraphic sequence.

Axial Fault

The linear mineralized trend defined by the Central, Southwest, and Hugo Dummett deposits likely reflect the presence of a deep, north-north-east-striking crustal fault or fault zone controlling magma emplacement and mineralization. This inferred structure probably also controlled emplacement of the north-north-east-striking late-mineral biotite granodiorite dyke. Although the controlling structure is largely conceptual, and direct evidence for it would have been overprinted in many areas by the biotite granodiorite dyke, it is referred to for convenience as the Axial Fault.

Apparent offsets of stratigraphic contacts across the main biotite granodiorite dyke have been interpreted by Ivanhoe and provide supporting evidence for the existence of the Axial Fault. Depending on how contacts are projected through areas with low drillhole density, the basal contact of the upper member of the Alagbayan Formation shows an apparent stratigraphic displacement of approximately 200 to 300 m across the biotite granodiorite dyke. Because the slip direction is unknown, this represents the minimum amount of movement on the fault. There is no clear evidence for post-dyke movement on the Axial Fault. This restricts the timing of movement to the narrow time interval between deposition of Alagbayan Formation and intrusion of the biotite granodiorite.

West Bat Fault

The West Bat Fault is a north-north-east-striking, subvertical structure that extends along the west side of the Hugo Dummett deposit. It cuts the north-western edge of the Hugo North deposit, but is well to the west of the main part of the Hugo South deposit. The structural character of the fault in drillhole intersections varies with both depth and stratigraphic level. In deeper intersections, where the fault juxtaposes massive volcanic conglomerates (upper Sainshandhudag Formation) against quartz monzodiorite, biotite granodiorite, or lower Alagbayan Formation, it occurs as a weakly- to moderately-foliated gouge-and-breccia zone up to a few metres wide. At shallower levels, the fault forms a much broader tectonized zone and commonly splits into several subparallel splays.

The West Bat Fault juxtaposes significantly lower stratigraphic levels to the east against higher stratigraphic levels to the west. If the base of the upper Sainshandhudag Formation is used as a marker, stratigraphic offset, and therefore the minimum net slip, is over 1,500 m. Determination of true net slip amount and direction on the West Bat

Fault is complicated by the probable multiple periods of movement on the fault, and available data do not uniquely constrain the fault movement history.

East Bat Fault

The East Bat Fault is a north-north-east-striking, subvertical structure occurring along the east side of the Hugo Dummett deposit. Although contact positions adjacent to the fault imply a minimum displacement of a few hundred metres, the fault expression in drill core can be subtle, often amounting to only decimetres of foliated gouge or cataclasite. The East Bat Fault occurs well to the east of the mineralization defined within the Hugo South deposit.

Stratigraphic contacts within the Sainshandhudag Formation show around 200 to 300 m of east-side-down stratigraphic offset across the East Bat Fault. There is no kinematic information available for the fault, so the true slip direction and amount are uncertain. Much, if not all, movement would have been post-mineral: the fault cuts Carboniferous strata of the Sainshandhudag Formation, but is itself cut by the East-West Fault.

East Hugo Fault

The East Hugo Fault occurs as a north- to north-north-west-striking, steeply east-dipping zone of strong to intense brecciation and clay gouge occurring along the east limb of the Hugo South and Hugo North deposits. At Hugo South it cuts across stratigraphic contacts at moderate angles and forms a sharp break in alteration intensity and copper grade. It displaces mineralization, and there is no evidence in grade distribution or alteration to suggest that the fault was present at the time of mineralization. Drillhole data suggest that the East Hugo Fault cuts the 110 Fault but is displaced dextrally by the East-West Fault. Sections where fault and contact geometry are best constrained show apparent east-side-down displacement of the Alagbayan Formation contacts ranging from about 200 m to as much as 400 m. The true slip direction, and thus the amount of net slip, is not constrained.

(1).3 Folding History

Bedding measurements obtained from oriented drill core, mainly from bedded intervals within the upper Alagbayan Formation, define two orientations of folds in the Hugo South deposit area: a dominant set of north-north-east-trending folds, and a subordinate set of north-west-trending folds. Both of the dominant fold orientations also occur in Carboniferous post-mineral strata, indicating that both events post-date mineralization and may have modified the form of the deposit.

Within the upper Alagbayan Formation, north-north-east-trending folds have wavelengths and amplitudes on the scale of metres to tens of metres. The far greater abundance of east-dipping bedding measurements implies that the fold geometry is strongly asymmetric, characterized by west-verging folds. In contrast, north-west trending folds have wavelengths of hundreds of metres and open, symmetric forms. Together, the two fold sets define an elongate dome and basin interference pattern. Because the axial surfaces and fold axes of the two sets are at a high angle to one another, and no penetrative cleavage fabrics formed during either event, it is not possible to determine their relative ages in the deposit area.

Reversals in sedimentary facing direction occur locally in the upper part of the Alagbayan Formation along the east flank of the Hugo South deposit. These reversals suggest the presence of tight to isoclinal folds affecting at least the Alagbayan Formation. These same overturned folds occur in the Hugo North deposit, where they are cut by the Late Devonian biotite granodiorite dyke. These cross-cutting relationships effectively bracket the timing of folding to Late Devonian, roughly contemporaneous or close in age to mineralization.

(2) Hugo North Deposit

(2).1 Host Rocks

The Hugo North deposit is virtually contiguous with the Hugo South deposit and occurs within a similar geological setting. The two deposits are however separated by a 110° striking, 45 to 55° north dipping fault that displaces Hugo North vertically down a modest distance from Hugo South. Ivanhoe has divided the mineralization into these two deposits for the purposes of resource estimation, and future mine planning development. Host rocks at Hugo North are an easterly-dipping sequence of volcanic and volcanoclastic strata correlated with the lower part of the Devonian Alagbayan Formation and quartz monzodiorite intrusive rocks that cut them. Stratigraphically lowest rocks in the host sequence are basalt flows and minor volcanoclastic strata, overlain by a basaltic tuff and breccia sequence. The basaltic tuff sequence was previously thought to be dacitic in composition. Whole rock geochemistry has shown it to be similar in composition to the underlying basalt and has been affected by advanced argillic alteration to give it the appearance of a dacitic tuff. The host sequence basaltic volcanics are overlain by dacitic block and ash tuff and dacitic ash flow tuff. Weakly altered to unaltered sedimentary and volcanic rocks of the upper Alagbayan Formation and Sainshandhudag Formation structurally overlie the mineralized sequence along the eastern flank of the deposit.

Intrusive rocks at Hugo North are dominated by a quartz monzodiorite body that underlies the entire deposit area and hosts a significant portion of the copper and gold mineralization. Intrusive contacts of the quartz monzodiorite are irregular, but overall show a preferred easterly dip, subparallel to stratification in the overlying rocks. The quartz monzodiorite is contemporaneous with alteration and mineralization, and several varieties are distinguished on the basis of alteration characteristics and position within the deposit:

- An intensely quartz veined phase that occurs along the upper margin of the main intrusive body.
- A gold-rich phase, restricted to the western part of the main intrusion in the northern part of the Hugo North deposit.
- The main intrusive body, which typically has lower vein density and lower copper and gold grades. Cross-cutting relationships between the different phases are ambiguous, and it is uncertain whether they represent temporally distinct intrusive events or simply variations in alteration intensity related to position within the hydrothermal system.

Late- to post-mineral biotite granodiorite intrusions form a voluminous, northerly-striking dyke complex cutting across the western edge and deeper levels of the deposit.

Although these intrusions locally contain elevated copper grades adjacent to intrusive contacts or within xenolithic zones, they are essentially non-mineralized. The positions and orientations of dyke contacts are well established in the Hugo North deposit area on the basis of correlations between drillhole intersections and measurements of individual contacts using oriented drill core. Dominant dyke orientation varies with depth. At levels above approximately 250 m elevation, where it cuts through the non-mineralized hanging wall strata, the biotite granodiorite occurs as a single intrusive mass with contacts dipping moderately to steeply to the west. Below this level, the biotite granodiorite is more complex, occurring as multiple, subparallel to anastomosing dykes that cut through the quartz monzodiorite intrusion and mineralized Alagbayan Formation strata. Drillhole correlation suggests that this lower portion contains a wide, steeply-dipping central dyke, from which emanate numerous moderately- to steeply-dipping apophyses.

Younger intrusions include rhyolitic, hornblende-biotite andesite, dacite, and basalt-dolerite compositional varieties. These intrusions usually occur as dykes with subvertical orientations or less commonly as easterly-dipping sills emplaced along stratigraphic contacts. They are non-mineralized and not volumetrically significant in most of the deposit.

Figure 7.5 to Figure 7.8 are representative cross-sections across the Hugo North Deposit, starting at the south end of Hugo North and continuing to the northern end on the Shivee Tolgoi Joint Venture Property.

(2).2 Structural Geology

The Hugo North deposit occurs within easterly-dipping homoclinal strata contained in a north-north-easterly elongate fault-bounded block. The northern end of this block is cut by several north-east-striking faults near the northern boundary of the Oyu Tolgoi property, but the deposit remains open along trend north of these faults. Other than these north-easterly faults, the structural geometry and deformation history of the Hugo North deposit are similar to those of the Hugo South deposit.

Deformation of the Hugo North deposit is dominated by brittle faulting. Major faults cutting the deposit can be grouped on the basis of orientation into the following sets:

- East-west-striking, moderately north-dipping faults (110 Fault).
- Steep, north-north-east-striking faults (East and West Bat Fault, 160 Fault, and Axial Faults).
- North-north-east-striking, moderately east-dipping faults subparallel to lithologic contacts (Contact, Lower faults).
- East-north-east-striking faults (Boundary Fault System; Rhyolite Fault).

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Deposit-Parallel Faults

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The Lower Fault at Hugo North occurs as an intensely brecciated, locally foliated, clay-rich gouge zone within the middle or lower portion of the high-grade mineralized body, typically at a level 200 to 400 m below the Contact Fault. It often coincides with the upper contact of the easterly-dipping portion of the biotite granodiorite dyke, and extends up dip through the main body of the dyke as a variably-developed fracture zone as well. In many drillholes, the Lower Fault occurs at an abrupt down hole discontinuity in copper and/or gold grades, indicating that latest movement displaced mineralized zones. Cross-sections through the Hugo North gold zone show apparent thrust offsets of around 300 to 400 m, based on displaced high gold grades.

West Bat Fault

The subvertical, north-north-east-striking West Bat Fault occurs along the west side of the Hugo North deposit and cuts the western edge of the northern part of the deposit. Its orientation and structural character are similar to its character at Hugo South, except that it splays upwards and northwards into several subparallel strands. Drillhole intersections in the northern part of Hugo North imply an irregular fault surface, although this irregularity may be in part due to inaccurate surveys, compounded by the great depths of the fault-piercing points. The West Bat Fault is truncated at its northern end by the north-east-striking Boundary Fault System.

The stratigraphic offset of the base of the upper Sainshandhudag Formation across the West Bat Fault is over 1,500 m. The fault movement direction is unknown, and the total offset may be significantly greater than the stratigraphic offset.

East Bat Fault

The north-north-east-striking East Bat Fault follows the east flank of the Hugo Dummett deposit, well east of the known deposit extents. Based on constraints in the Hugo South area, the East Bat Fault has several hundred metres of apparent east-side-down displacement. The East Bat fault has been intersected by a number of geotechnical holes drilled to the east of Hugo North.

160 Fault

The 160 Fault can be traced through the southern part of the Hugo North deposit, between sections 4766700N and 4767100N. It cuts across stratigraphic contacts at moderate angles and forms a sharp break in alteration intensity and copper grade. The fault gradually loses expression northward, suggesting a gradual decrease in displacement.

The 160 Fault strikes north to north-north-west (subparallel to the deposit trend) and dips steeply to the east. The fault cuts and displaces mineralization, which occurs at several tens of metres higher elevation on the western fault block. The true slip direction of the fault is unknown. There is no evidence that the fault was present at the time of mineralization.

North-east-Striking Faults (Boundary Fault, Others)

Several east-north-east to north-east-striking faults cut across the northern end of the

Hugo North deposit near where it crosses the Oyu Tolgoi property boundary. These include the North Boundary Fault, which juxtaposes Carboniferous granitic rocks against Carboniferous strata to the south, an unnamed, more gently-dipping splay of the North Boundary Fault, and the Boundary Fault, which occurs slightly to the south.

The North Boundary Fault has been intersected in several drillholes and usually occurs as a zone of intensely developed, foliated clay gouge up to several tens of metres wide. Its strike varies from east-north-east to nearly north-south, but dips are consistently around 70° to the north-west. The Boundary Fault follows the intrusive contact of the granitic complex in the north-west part of the Oyu Tolgoi Property and likely formed through reactivation of that surface during regional deformation.

A moderately north-west-dipping splay of the North Boundary Fault juxtaposes Carboniferous strata over the deposit sequence in several drillholes in the northern part of the property. This splay diverges upward from the North Boundary Fault well above the level of mineralization, and thus does not cut the deposit.

The steep, north-east-striking Boundary Fault juxtaposes strongly-mineralized rocks against post-mineral Carboniferous strata near the northern property boundary. This fault truncates the West Bat Fault and may displace it and the northern end of the Hugo North deposit to the north-east. At higher levels, the Boundary Fault is cut by the splay of the North Boundary Fault.

(3) Rhyolite Fault

The subvertical, east-north-east-striking Rhyolite Fault cuts across the southern part of the Hugo North deposit between sections 4766500N and 4766700N. It coincides with a wide rhyolite dyke zone, and usually is manifested as strong to intense breccia and clay gouge zones along dyke contacts. Stratigraphic contacts show several tens of metres of apparent dextral or south-side-down displacement across the fault, but the absolute slip direction is unknown. The Rhyolite Fault cuts and displaces the Contact Fault.

(4) Folding History

Bedding measurements indicate that fold style and orientation in the Hugo North deposit are similar to those at Hugo South, with most folding restricted to the upper part of the Alagbayan and overlying Sainshandhudag Formation.

The inverted sedimentary facing directions found in the upper part of the Alagbayan Formation at Hugo South occur at similar stratigraphic levels at Hugo North. However, at Hugo North, they define tight folds that are clearly truncated by the biotite granodiorite dyke. This narrowly restricts the age of folding to Late Devonian, in roughly the same time period as mineralization.

7.1.3.4 Heruga Deposit

The Heruga Deposit is the most southerly of the known deposits at Oyu Tolgoi. The deposit is a Cu, Au, Mo porphyry deposit and is zoned with a Mo rich carapace at higher elevations overlying Au rich mineralization at depth. It has been drilled over a 2.3 km length, elongated in a north-north-east direction and terminates to the north on an east-north-east trending regional fault with 500 m of apparent dextral displacement. The deposit is transected by a series of north-north-east trending vertical fault structures that step down 200 to 300 m at a time to the west and have divided the deposit into two structural blocks. The most recent discovery at Oyu Tolgoi is the faulted offset of Heruga mineralization to the north of the east-north-east bounding fault. The deposit is open to the east, west, and south.

(1) Host Rocks

Host rocks at Heruga are an easterly-dipping sequence of volcanic and volcanoclastic strata correlated with the lower part of the Devonian Alagbayan Formation and quartz monzodiorite intrusive rocks that intrude them.

(1).1 Devonian Sedimentary Volcanic Rocks

Mineralization at Heruga is hosted by unit DA1b porphyritic augite basalt, overlying DA2a conglomerate and quartz monzodiorite intrusions. Mineralization and alteration at Heruga contrasts with that at Hugo and the Southern Oyu in that it extends to a higher level in the stratigraphy, transgresses the unconformity at the top of unit DA1 and into overlying unit DA2a conglomerate. The bulk of significant mineralization however occurs in unit DA1b augite basalt.

The host sequence DA1b basaltic volcanic rocks and DA2a conglomerates are overlain by unmineralized Alagbayan formation rocks consisting of DA2b dacitic pyroclastic rocks, and DA3 interbedded conglomerate, sandstone, siltstone, and mudstone.

Unmineralized unit DABa, basaltic andesite breccia to conglomerate, is only seen in the western block at Heruga and in the shaft farm area west of Hugo. Its stratigraphic position is uncertain but has been suggested by Lewis (2008), to conformably overlie DA1b at Heruga, and DA4 in the shaft farm area, thereby providing the missing link between unit DA1b and the allochthonous unit DA4. As seen throughout the Oyu Tolgoi district unit DA4, sedimentary and volcanic rocks of the upper Alagbayan Formation form an overturned allochthonous sequence that has been structurally emplaced over the lower Alagbayan stratigraphy. The Sainshandhudag Formation unconformably overlies the Alagbayan sequence along the eastern flank of the deposit.

(1).2 Intrusives

Biotite Granodiorite

Biotite granodiorite at Heruga intrudes Devonian rocks as it does throughout the Oyu Tolgoi region. According to Forster et al (2008), it occurs only below the unconformity at Heruga, where it commonly forms sills intruding along bedding-parallel structures near the Contact Fault. It also occurs less commonly as subvertical dykes.

Quartz Monzodiorite

Quartz Monzodiorite at Heruga is similar to that at the Southern Oyu Tolgoi and Hugo Dummett deposits. The Quartz Monzodiorite at Heruga is relatively fine grained and may possibly be present at two distinct phases (one mineralized and one unmineralized). Much of the higher grade portion of the deposit is localized adjacent to contacts of the mineralized quartz monzodiorite intrusion. The interpretation of Lewis (2008) shows clearly that this intrusion is nearly confined to a single section at current drill spacing, indicating that its contacts are nearly section parallel. QG concur with Lewis (2008) that any infill drilling will require specific holes drilled perpendicular to cross-section to better define its geometry.

Andesite and Dacite Dykes

Post-mineralization porphyritic hornblende-biotite dykes of andesitic to dacitic composition are widespread at Heruga and strike north to north-east, dipping steeply. Forster et al (2008) report that these dykes are similar to those dated as Carboniferous in the Oyu Tolgoi deposits.

Basalt and Rhyolite Dykes

Fine grained basaltic dyke, interpreted as often intruding both steep and low-angle structures (Forster et al, 2008) are common at Heruga. These dykes also post-date mineralization and are interpreted as coeval with similar dykes assumed to be Carboniferous in the Oyu Tolgoi deposits. Rhyolite dykes are relatively uncommon. Forster et al (2008) note that some dolerite dykes of possibly Mesozoic age are encountered at Heruga.

(2) Structure

Heruga lies at the southern end of a 20 km north-north-easterly trend of porphyry style deposits, extending to Shivee Tolgoi (Hugo North Extension) in the north. Forster et al (2008) postulate that this alignment reflects deep crustal structure, exploited by Late Devonian magmatism and mineralizing systems.

The interpretation of Lewis (2008) shows Heruga cut by several major brittle fault systems, partitioning the deposit into discrete structural blocks. Internally, these blocks appear relatively undeformed, and consist of south-east-dipping volcanic and volcanoclastic sequences. The stratiform rocks are intruded by quartz monzodiorite stocks and dykes that are probably broadly contemporaneous with mineralization, as well as subvertical hornblende-biotite andesite dyke swarms discussed in the preceding section of this report.

The deposit-scale faults identified at Heruga displace mineralized zones, with mineralization and alteration zones not localized by the faults, implying they post-date mineralization. Lewis (2008) concludes that it is likely the Heruga porphyry formed within a relatively intact structural block, with most faulting and disruption of contacts related to post-mineralization deformation.

(2).1 Contact Fault

As at Oyu Tolgoi, volcanoclastic strata of unit DA4 are structurally juxtaposed over the Heruga deposit along a major, south-east-dipping fault zone. Lewis (2008) describes this fault being "... almost certainly the southerly extension of ..." the Contact Fault at the Hugo Dummett and Southern Oyu deposits.

The Contact Fault at Heruga varies from tens of centimetres to 40 m in thickness, and has an average orientation striking 110° dipping 45° east-south-east. Lewis (2008) also reports that kinematic indicators such as shear bands and drag folds record up-dip (thrust) displacement.

In the northern part of the deposit, facing direction reversals and repetition of stratigraphic units define a large-scale recumbent anticline in the hangingwall sequence to the Contact Fault. Although the magnitude of displacement on the Contact Fault is poorly constrained, the scale of the overturned folds in the hangingwall, vertical stacking of dissimilar stratigraphic sequences, and the fault continuity throughout the Oyu Tolgoi area all suggest that displacement of kilometres to tens of kilometres is probable.

(2).2 East-North-East Striking Bounding Faults

The Heruga deposit area lies between two regional, east-north-east-striking faults that form prominent features on both magnetic and satellite images referred to as the Heruga North and South Sparrow faults. The surface trace of the Heruga North fault has been mapped in surface outcrops. Drillholes collared on section 4759700N to the east of the fault, were drilled to the west, passing beneath the surface trace of the fault, thus defining a moderate northerly dip to the fault. Surface geological mapping suggests a 500 m apparent dextral displacement. If the Heruga North between Southwest Oyu and Heruga is the offset continuation of Heruga mineralization to the north of the North Heruga fault then a dextral displacement in the order of 1,000 m as well as a north side downward displacement in the order of 600 m is required.

The southern bounding fault ("South Sparrow Fault") crosses the deposit trend approximately 250 m south of the southernmost drillhole at the time of writing (EJD-0019). One drillhole has been completed in the Heruga area south of this fault (EJD-0016) which intersected a thick sequence of weakly-altered to unaltered volcanoclastic rocks of probable Carboniferous age, suggesting south-side-down apparent offset.

(2).3 Bor Tolgoi Fault System

Lewis (2008) divides the Heruga Deposit into four discrete structural blocks by subvertical, north-north-east-striking faults he refers to as the "Bor Tolgoi fault system". These faults occur in drill core as intervals of breccia and gouge, sighted by QG during the site visit, and also reportedly form subtle linear anomalies on ground magnetic images.

In east-west sections, both the West Bor Tolgoi Fault and the better constrained Bor Tolgoi Fault display between 300 m and 500 m west-side-down apparent offset of

stratigraphic contacts (Lewis, 2008). Kinematic indicators have not been observed in either fault, and consequently the true slip direction and magnitude are unconstrained.

Lewis (2008) interprets fault movement as dominantly strike-slip.

(2).4 Southern Fault

The Southern Fault is an interpreted steep, west-north-west-striking structure that cuts across the southern end of the Heruga deposit. Although multiple interpretations are possible, Lewis (2008) concludes that the simplest fault geometry compatible with the apparent offsets of contacts in these drillholes requires only a single section-parallel fault. Surface maps show an interpreted fault roughly coincident with the interpreted structure, and the fault trace also coincides with a weak linear magnetic feature.

7.1.3.5 Heruga North

Heruga North occurs in the 3 km gap between Heruga and Southwest Oyu in a structural block limited by the east-north-east trending Solongo fault to the north and the Heruga North fault to the south, within the Oyu Tolgoi licence area. It appears to be the northern structurally offset continuation of the Heruga deposit to the south. The area is cut by north-north-east trending west side down faults similar to the Bor Tolgoi fault structures at Heruga and the West Bat fault at Hugo Dummett. To date, only two drillholes on sections 1.5 km apart have intersected the Heruga North zone and the actual deposit geometry is poorly constrained.

(1) Host Rocks

(1).1 Devonian Sedimentary and Volcanic Rocks

Host rocks at Heruga North are an easterly-dipping sequence of volcanic and volcanoclastic strata that correlate with the lower part of the Devonian Alagbayan Formation and quartz monzodiorite intrusive rocks. The host sequence DA1b basaltic volcanic rocks and DA2a conglomerates are overlain by unmineralized Alagbayan formation rocks consisting of DA2b dacitic pyroclastic rocks, and DA3 interbedded conglomerate, sandstone, siltstone and mudstone. Unit DA4 sedimentary and volcanic rocks of the upper Alagbayan Formation form an overturned allochthonous sequence that has been structurally emplaced over the lower Alagbayan stratigraphy. The Sainshandhudag Formation unconformably overlies the Alagbayan sequence along the eastern flank of the deposit.

(1).2 Intrusives

Quartz Monzodiorite

Three varieties of quartz monzodiorite are recognized at Heruga North, all occur within discovery hole OTD1487 and 1487a. The discovery hole passed through several hundred meters of sericite altered quartz monzodiorite with disseminated pyrite and only traces of chalcopyrite mineralization before passing through a fault (continuation of Bor Tolgoi) and into strongly mineralized basaltic volcanics (unit DA1b) intruded by strongly mineralized quartz monzodiorite dykes. Late xenolithic quartz monzodiorite intrudes the

strongly mineralized basaltic volcanics. The xenolithic quartz monzodiorite is pervasively sericite altered with moderate disseminated pyrite and weak chalcopyrite. Xenoliths include mineralized clasts of the enclosing wallrock.

Biotite Granodiorite

Biotite Granodiorite forms late to post mineral narrow dikes and sills that intrude lower Alagbayan stratigraphic units DA1, DA2 and DA3 as well as the allochthonous unit DA4. A number of biotite granodiorite dykes cut a large body of weakly mineralized quartz monzodiorite. A 30 m thick biotite granodiorite sill occur within the DA2a conglomerate unit.

Andesite, Dacite and Basalt Dykes

Post-mineralization porphyritic hornblende-biotite dykes of andesitic to dacitic composition and basalt dykes intrude Alagbayan and Sainshandhudag Formation stratigraphy as well as the quartz monzodiorite and biotite granodiorite intrusive rocks. Similar andesitic dykes at the more northerly deposits have been dated as Carboniferous. Rhyolite dykes have not been encountered in Heruga North drillholes.

(2) Structure

(2).1 Contact Fault

The Contact fault is a low angle thrust fault generally parallel to bedding that places overturned upper Alagbayan Devonian sedimentary and volcanic rocks belonging to the DA4 unit over upright Devonian sediments of unit DA3b. The thrust fault occurs completely in the unmineralized Devonian sequence overlying the deposits and nowhere does it truncate mineralization.

(2).2 Solongo Fault

Heruga North occurs between two major east-north-east striking regional faults, consisting of the Solongo Fault to the north and the Heruga North fault to the south. Solongo Fault is an east— to east-north-east-striking, subvertical structure that cuts across the Oyu Tolgoi property just south of the Southwest Oyu and South Oyu deposits. The Solongo Fault typically occurs as a strongly tectonized, foliated zone up to several tens of metres wide. Rhyolite dykes commonly intrude the fault zone, and themselves have tectonically brecciated margins. On ground magnetic data, the fault forms a major linear anomaly that can be traced across the entire width of the Oyu Tolgoi property. The Solongo Fault forms a major structural break, and there is a minimum of approximately 1,600 m of south-side-down stratigraphic offset where it juxtaposes mineralized basalt (unit DA1) in the South Oyu deposit against sediments correlated with the upper Alagbayan Formation (unit DA4) to the south. Depending on movement direction, net slip may be considerably greater. The displacement history of the Solongo Fault is poorly constrained. The fault clearly existed prior to emplacement of rhyolite dykes, which are part of a suite dated at 340 ± 3 Ma. Rhyolite dykes locally cross the fault with little or no apparent displacement, implying that movement since their emplacement was negligible.

(2).3 North Heruga Fault

The surface trace of the North Heruga fault has been mapped in surface outcrops. Drillholes collared on section 4759700N to the east of the fault, were drilled to the west, passing beneath the surface trace of the fault, thus defining a moderate northerly dip to the fault. Surface geological mapping suggests a 500 m apparent dextral displacement. If the Heruga North between Southwest Oyu and Heruga is the offset continuation of Heruga mineralization to the north of the North Heruga fault then a dextral displacement in the order of 1,000 m as well as a north side downward displacement in the order of 600 m is required.

(2).4 Bor Tolgoi Fault System

The Heruga North occurs in a fault bounded block between north-north-east trending near vertical faults with west side down displacement. The western most fault in this series however has a displacement an order of magnitude larger than those of Bor Tolgoi fault series, at least 1,500 m, similar to the displacement seen on the West Bat Fault to the north, on the west side of the Hugo Dummett deposits.

7.1.3.6 References

The information for this section is in the report Oyu Tolgoi Technical Report, March 2008, with the exception of the following additional references:

Wainwright, J.A., 2008. Volcanostratigraphic framework and magmatic evolution of the Oyu Tolgoi porphyry Cu-Au district, South Mongolia. Unpublished PhD thesis, University of British Columbia.

7.2 Mineralization

7.2.1 Southern Oyu Deposits

7.2.1.1 Southwest Deposit

Copper–gold porphyry style mineralization at the Southwest deposit consists of a cylindrical high-grade core roughly 250 m in diameter enclosed within a broad zone of lower-grade mineralization. The high-grade core is centered on a 10 m – 30 m wide, vein-rich quartz monzodiorite dyke and extends for over 100 m into the adjacent massive porphyritic augite basalt. The high-grade core is characterized by 1 to 50 cm wide contorted milky white quartz veins in sericite, albite, minor tourmaline-altered quartz monzodiorite and biotite–magnetite-altered augite basalt, overprinted by chlorite and sericite. Chalcopyrite with subordinate pyrite, bornite, and molybdenite occur as late veinlets filling fractures in quartz veins and disseminated through wall rocks.

Low-grade copper mineralization peripheral to the high-grade core is characterized by lower vein densities, hosted in chlorite and epidote altered basalt and lesser sericite- and albite-altered quartz monzodiorite. Magnetite veinlets post-date the quartz veins but predate the main sulfide event. Chalcopyrite, bornite, and pyrite are mainly disseminated, with fracture- or vein-controlled sulfides being less prominent. These peripheral zones include; the informally defined Far South zone, which encompasses

mineralized basalt with 1:1 gold:copper (ppm:%) ratios on the south-west margin of the deposit area, and the Bridge zone, consisting of copper-mineralized basalt and quartz monzodiorite between the Southwest and Central deposits. Although these two subzones were used in 2005 as domain boundaries in resource modelling (see Juras, 2005), there is no clear geological boundary distinguishing them from the adjacent peripheral zone mineralization.

Gold mineralization in the Southwest deposit is closely associated with chalcopyrite, and occurs intergrown with chalcopyrite, as inclusions and fracture infills within pyrite, or on grain boundaries of pyrite. Less frequently, gold occurs on grain boundaries with bornite or as inclusions in bornite, quartz or carbonate. The gold to copper (ppm:%) ratios range from 2:1 to 3:1 within the high-grade core, decreasing to 1:1 in the low-grade margins of the deposit.

The Southwest deposit is capped by an oxidized zone that varies from 50 to 60 m thick, and consists of black copper oxide (neotocite or tenorite) as fracture coatings, and speckled throughout the oxidized limonite-stained basalt.

Alteration styles at the Southwest deposit are typical of copper–gold porphyry systems. Augite basalt in the high-grade core of the deposit contains biotite and magnetite alteration, overprinted by chlorite and sericite. Biotite alteration occurs pervasively in the core of the deposit and grades outwards to selvage controlled within pervasive chlorite and epidote alteration. Minor albite alteration occurs as selvages along veins or fractures. Locally, brown carbonate alteration is present in the basalt.

Vein-rich quartz monzodiorite (OT-Qmd and xQmd phases) in the high-grade core contains sericite–biotite–albite alteration with minor tourmaline and montmorillonite. Pink albite alteration commonly occurs as selvages on veins or fractures, and sericite overprints biotite and albite.

In the low-grade peripheral portions of the deposit, augite basalt is pervasively chlorite–magnetite altered, with epidote occurring in patches and sericite and pink albite on vein or fracture selvages. Pink albite may form reaction rims around irregularly-shaped epidote patches. Biotite alteration occurs locally. Late calcite or ankerite veins cross-cut the assemblage. Quartz monzodiorite within the low-grade margin contains pervasive sericite alteration, with albite occurring along quartz vein or fracture margins. Spotty biotite alteration occurs locally.

7.2.1.2 South Oyu Deposit

Mineralization in the South Oyu Deposit is hosted dominantly in quartz monzodiorite in the south-western portion of the deposit, in basalt throughout the central portion of the deposit, and in a minor zone of basaltic tuff and breccia on the northern margin. Contorted quartz veins are present, but there is no clearly defined zone of high quartz vein density such as at the Southwest Deposit. Consequently, fracture-controlled sulfide veins are minor, and sulfides occur dominantly as disseminated chalcopyrite, bornite, and molybdenite. Chalcopyrite is the principal copper sulfide, but in higher-grade areas bornite locally exceeds chalcopyrite in abundance. Magnetite occurs as disseminations and as veins. Small zones with elevated gold values occur locally.

A small zone of high-sulfidation mineralization occurs within a quartz monzodiorite breccia in the western part of the deposit, adjacent to the South Fault. Mineralization here consists of pyrite, chalcopyrite, bornite, covellite, and primary chalcocite in quartz-sericite-kaolinite alteration, with late dickite veins.

An oxide zone approximately 60 m thick overlies the South Oyu deposit, and consists of malachite, azurite, cuprite, chrysocolla, neotocite, or tenorite hosted within basalt and quartz monzodiorite.

Alteration within the basaltic rocks at South Oyu consists of moderate chlorite, biotite, hematite-magnetite, weak sericite, and pink albite fracture and vein selvages. Hematite overprints magnetite. Quartz monzodiorite is typically pervasively altered with quartz, sericite, and pyrite, as well as albite within vein selvages, small radiating clusters of tourmaline, and fluorite in quartz veins. Advanced argillic alteration, consisting of quartz, sericite, and kaolinite with late dickite veins, is associated with the high-sulfidation mineralization in the quartz monzodiorite breccia.

7.2.1.3 Wedge Deposit

The Wedge deposit contains a zone of high-sulfidation mineralization hosted principally in basaltic tuff and breccia, grading downward and southward into chalcopyrite mineralization in basalt and quartz monzodiorite host rocks.

High-sulfidation mineralization consists of pyrite, chalcopyrite, bornite, enargite, covellite, and primary chalcocite in advanced argillically altered host rocks. Higher grades of copper ($>0.8\%$ Cu) occur in a shallowly east-dipping zone in the upper hundred metres of basaltic tuff/breccia unit. Gold is absent, except locally in drillholes adjacent to the South Fault. Mineralization is open down dip and to the north.

High-sulfidation mineralization grades downward into chalcopyrite, with lesser bornite within massive augite basalt host rocks, and pyrite and chalcopyrite mineralization in quartz monzodiorite.

Basaltic tuff and breccia within the Wedge deposit is characterized by advanced argillic alteration consisting of kaolinite, zunyite, pyrophyllite, muscovite, illite, topaz, diaspore, alunite, montmorillonite, late dickite, and fluorite. A barren, specular, hematite-rich zone occurs marginal to advanced argillic alteration and is progressively overprinted by advanced argillic alteration assemblages with increasing copper grades towards the center of the deposit. The advanced argillic alteration grades downward into biotite and chlorite alteration with hematite overprinting magnetite, mainly within massive augite basalt host rocks underlying the basaltic tuff/breccia. In the southern part of the Wedge deposit, sericite and pyrite alteration occurs within the quartz monzodiorite.

7.2.1.4 Central Deposit

Mineralization in the Central deposit is characterized by an upward-flaring, high-sulfidation zone that overprints and overlies porphyry-style chalcopyrite-gold mineralization. A secondary-enriched supergene chalcocite blanket tens of metres in thickness overlies the high-sulfidation covellite-pyrite zone.

Chalcopyrite–gold mineralization is dominant on the south and western margins of Central within either basalt or quartz monzodiorite adjacent to intrusive contacts with basalt. Higher grades are associated with zones of intensely contorted quartz stockwork veins, where the gold (ppm) to copper (%) ratios reach 2:1. Peripheral, lower-grade mineralization has gold to copper ratios of less than 1:1. Hematite, pyrite, chalcopyrite, bornite, magnetite, and gold occur disseminated in the zone and as fracture fillings. Hematite is pervasive and overprints magnetite.

The high-sulfidation part of the Central deposit lacks significant gold and contains a mineral assemblage of pyrite, covellite, chalcocite/digenite, enargite, tennantite, cubanite, chalcopyrite, and molybdenite. Dominant host rocks are dacite tuff and quartz monzodiorite. Higher-grade mineralization is associated with disseminated and coarse-grained fracture-filling sulfides in zones of intense contorted quartz stockwork veins and anastomosing zones of hydrothermal breccias. Hydrothermal breccia consists of quartz vein and quartz monzodiorite fragments within an intensely sericitized matrix. The sulfide-filled fractures cut both the quartz veins and enclosing wall rock. High-grade mineralization grades outward to a broad, weakly veined, low-grade halo of dominantly disseminated sulfides. Pyrite, chalcopyrite, bornite, and enargite occur here as relic grains replaced by chalcocite, and covellite, and pyrite also hosts small inclusions of covellite. Covellite, chalcocite and enargite occur as intimate intergrowths or as free disseminations. Cubanite and tennantite occur intergrown with or replacing enargite, and molybdenite occurs locally in quartz.

A supergene enrichment zone overlies the high sulfidation assemblage and underlies a 20 to 60 m thick, hematitic limonite, goethite-rich leached cap. The supergene zone consists of pyrite, hematite, and chalcocite/digenite, with lesser amounts of colusite, enargite, tenorite, covellite, bornite, chalcopyrite, cuprite, and molybdenite. Pyrite is the dominant sulfide and occurs as disseminated crystals. Sooty chalcocite occurs as rims or microveinlets in pyrite and covellite, and as independent disseminations. Colusite occurs as single grains or intergrown with chalcocite/digenite and/or pyrite. Tenorite occurs interstitial to silicate-iron oxide grain boundaries. Micrograins of chalcopyrite replaced by bornite and covellite occur as small inclusions within pyrite.

Minor exotic copper oxide mineralization occupies a bedrock depression on the north-eastern flank of Central. Chrysocolla, malachite, and neotocite mineralization occur over a 400 m x 300 m area as a thin 2–4 m thick layer at the base of the gravels. The leached cap is generally devoid of mineralization except on the edges of the eastern and southern flanks of the deposit, where patchy malachite and neotocite occur.

Alteration in the Central deposit shows a close spatial relationship to mineralization and original host lithology. Biotite–chlorite and intermediate argillic alteration coincide with chalcopyrite–gold mineralization within basalt. Advanced argillic and sericite alteration coincides with the high-sulfidation mineralization within quartz monzodiorite and basaltic tuff/breccia.

The biotite–chlorite zone consists of an assemblage of biotite, chlorite, epidote, sericite, albite, carbonate, and anhydrite. Hematite and minor magnetite occurs in veins and as disseminations. Biotite has been overprinted by chlorite and sericite, and magnetite has been altered to hematite. Anhydrite and carbonates occur as late veins. K-feldspar

alteration increases at depth beneath Central, occurring as vein selvages within biotite-altered basalt.

Intermediate argillic alteration forms a narrow zone separating the advanced argillic and sericite alteration from the biotite chlorite alteration. Intermediate argillic alteration is characterized by a creamy yellow to pale green coloured assemblage of kaolinite, chlorite, pyrophyllite, and illite.

Advanced argillic and sericite alteration are associated with high-sulfidation mineralization, hosted primarily within dacite and quartz monzodiorite. The advanced argillic assemblage consists of topaz, quartz, zunyite, diaspore, alunite, illite, and andalusite, late kaolinite, and dickite. There is a zonation from an advanced argillic assemblage of zunyite, andalusite, and alunite associated with higher-grade hydrothermal breccia-hosted mineralization, to a muscovite, sericite-dominant peripheral zone associated with lower-grade disseminated mineralization.

Alteration within the supergene zone is characterized by illite, muscovite, kaolinite, alunite and pyrophyllite. Montmorillonite, smectite, kaosmectite, illite and kaolinite are the dominant clay minerals in the leached cap.

7.2.2 Hugo Dummett Deposits

7.2.2.1 Hugo South Deposit

Copper mineralization at the Hugo South deposit is centered on a high-grade (typically >2% Cu) zone of intense quartz stockwork veining, which in much of the deposit is localized within narrow quartz monzodiorite intrusions and extends into the enclosing basalt and dacite tuff. The intense stockwork zone has an elongate tabular form, with a long axis plunging shallowly to the north-north-west, and an intermediate axis plunging moderately to the east. Copper grades gradually decrease upwards from the stockwork zone through the upper part of the massive augite basalt and the basaltic tuff, and a broader zone of lower grades occurs below and to the west in basalt and quartz monzodiorite.

Dominant sulfide minerals at Hugo South are chalcopryite, bornite, chalcocite, and pyrite, with minor molybdenite, enargite, tennantite, and covellite. Rarely, sphalerite and galena occur. Sulfides are zoned (Figure 7.9) with bornite ± chalcopryite, chalcocite, and tennantite comprising highest grades (>2.5% Cu), grading outwards to chalcopryite (1 to 2% Cu). Pyrite–chalcopryite ± enargite, tennantite, bornite, chalcocite, and rarely covellite occur with low-grades (<1% Cu), mainly in advanced argillically-altered basaltic tuff.

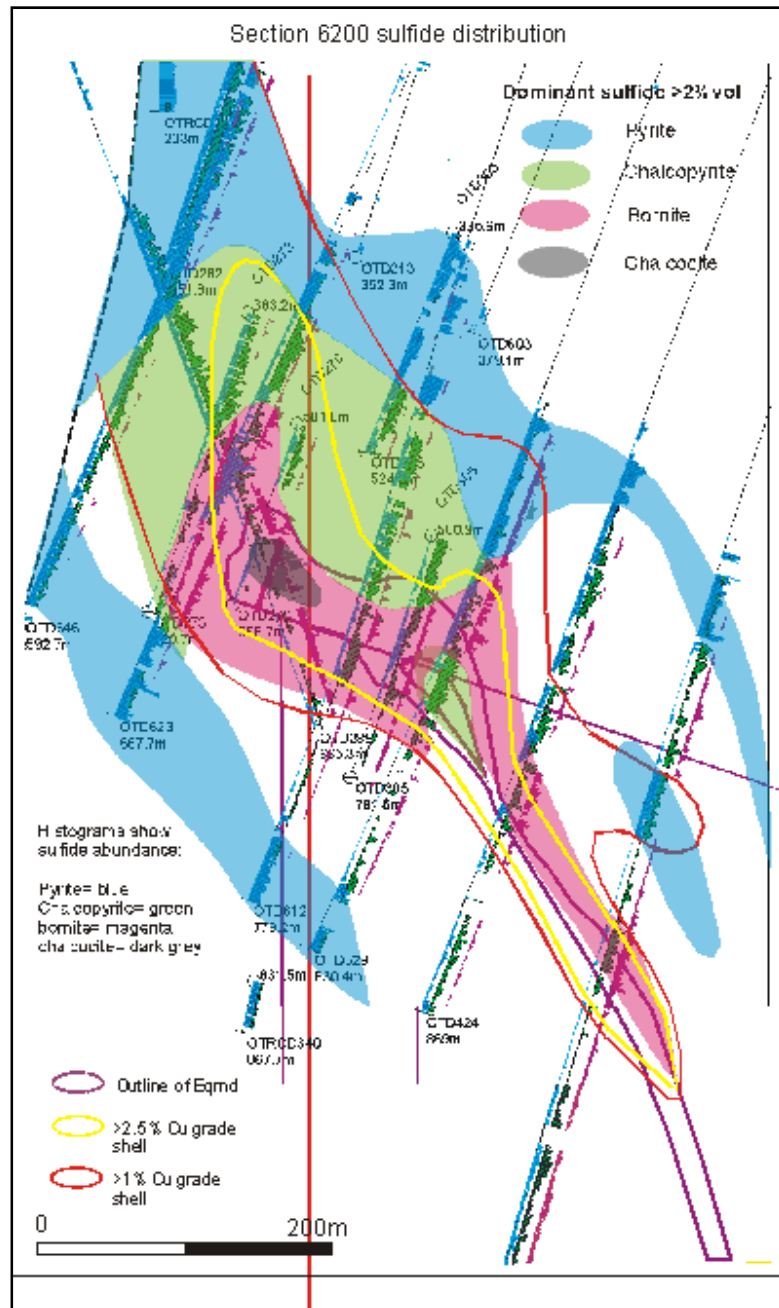
Alteration in the Hugo South deposit is typical of copper porphyry systems, including K-silicate (minor), advanced argillic, muscovite/sericite, and intermediate argillic styles. The mineral groupings used to define individual zones are not necessarily true assemblages that formed contemporaneously, but are associations that may represent several paragenetic stages.

The following alteration zones have been defined at Hugo South (Figure 7.10):

- Chlorite–illite alteration occurs in the uppermost part of the basaltic tuff. This alteration is not texturally destructive, and primary volcanic textures are clearly visible. No mineralization is associated with this weak outermost alteration zone.
- Advanced argillic alteration is dominant in the basaltic tuff and is characterized by the minerals quartz, pyrophyllite, kaolinite, topaz, diaspore, zunyite, alunite, and dickite. Because the advanced argillic alteration is widespread and consists of a complex mineral assemblage, it is divided into subzones:
 - Pyrophyllite–kaolinite: This is the most widespread subgroup of the advanced argillic alteration. It occurs dominantly in the basaltic tuff, but also in the upper parts of the basalt and quartz monzodiorite. Late white to pink dickite on fractures is ubiquitous.
 - Quartz–alunite: The quartz–alunite zone typically occurs in the basaltic tuff in bedding-parallel lenses. The lenses are parallel to, and occur just above and east of, the high-grade, vein-rich deposit core. It is whitish–pink to brown in colour.
 - Topaz alteration occurs as an intense, completely texturally destructive zone and is typically of limited extent. Topaz-altered zones are vuggy and whitish–brown in colour. They are most common in the basaltic tuff, but occur locally in the augite basalt.
- Pyrophyllite–kaolinite–dickite–muscovite/illite (intermediate argillic) alteration occurs in the upper parts of the massive augite basalt at the contact with the basaltic tuff and is the dominant alteration type in the high-grade deposit core. It is brownish–yellow in colour but turns pinkish where pyrophyllite is the dominant mineral. It is not texturally destructive. Hematite is common as fine specularite.
- Chlorite–muscovite/illite–hematite alteration is the dominant alteration in the augite basalt. It is characterized by its green colour, and the original texture of the basalt is still visible (e.g. relict pseudo-hexagonal augite crystals). Locally, particularly at depth, biotite occurs in the zone. Relict magnetite occurs either disseminated or in veins.

Muscovite–illite alteration occurs mainly in quartz monzodiorite intrusions. It is pale green to grey in colour and in most cases texturally destructive, although in places original feldspar crystals can still be observed. Locally, topaz occurs with muscovite, and a subgroup of muscovite–topaz alteration was recognized.

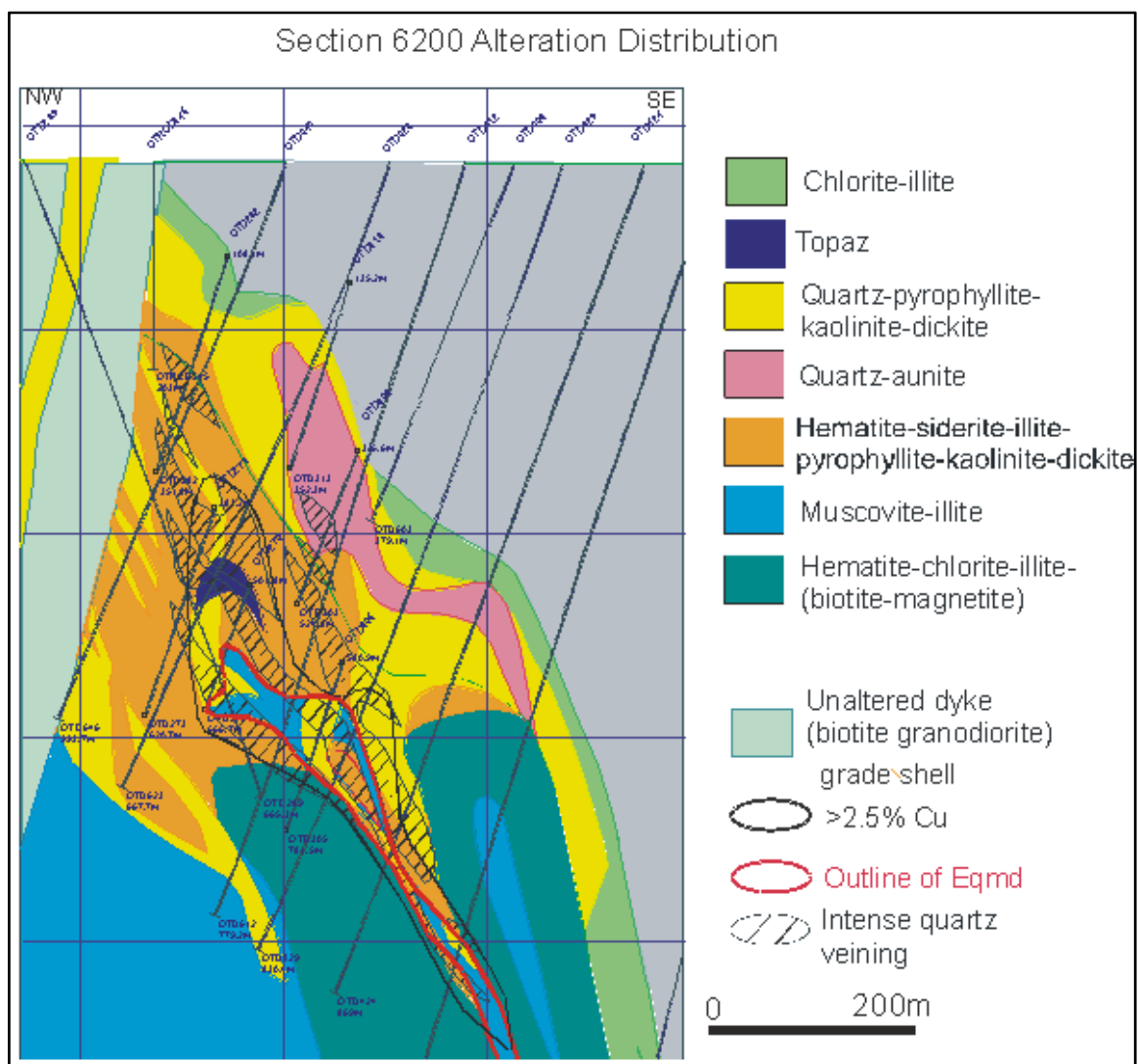
Figure 7.9 Hugo South Deposit, Sulfide Distribution (Section 6200N, Looking NE)



Note: Figure provided by Ivanhoe.

Shows bornite + chalcocite core zone, grading outward through bornite, chalcopyrite + bornite, chalcopyrite, and pyrite zones.

Figure 7.10 Hugo South Deposit Alteration, Cross-Section 6200N (Looking NE)



Note: Figure provided by Ivanhoe

Shows distribution of alteration zones. Grades within shells have been previously reported, see Juras, 2005.

The distribution of the alteration is strongly lithologically controlled: basaltic tuff typically shows strong advanced argillic alteration, whereas basalt tends to be chlorite-muscovite-hematite-altered with pyrophyllitic advanced argillic alteration in its uppermost parts. Pockets of advanced argillic alteration occur locally in the high-grade zone in the quartz monzodiorites.

7.2.2.2 Hugo North Deposit

The highest-grade copper mineralization in the Hugo North deposit is related to a zone of intensely stockworked to sheeted quartz veins. The high-grade zone is centered on thin, east-dipping quartz monzodiorite intrusions or within the upper part of the large quartz monzodiorite body, and extends into the adjacent basalt. In addition, moderate to high-grade copper and gold values occur within quartz monzodiorite below and to the

west of the intense vein zone, in the Hugo North gold zone. This zone is distinct in its high Au (ppm) to Cu (%) ratios (0.5:1). In other respects, the Hugo North and Hugo South deposits have similar mineralogy and zonation patterns.

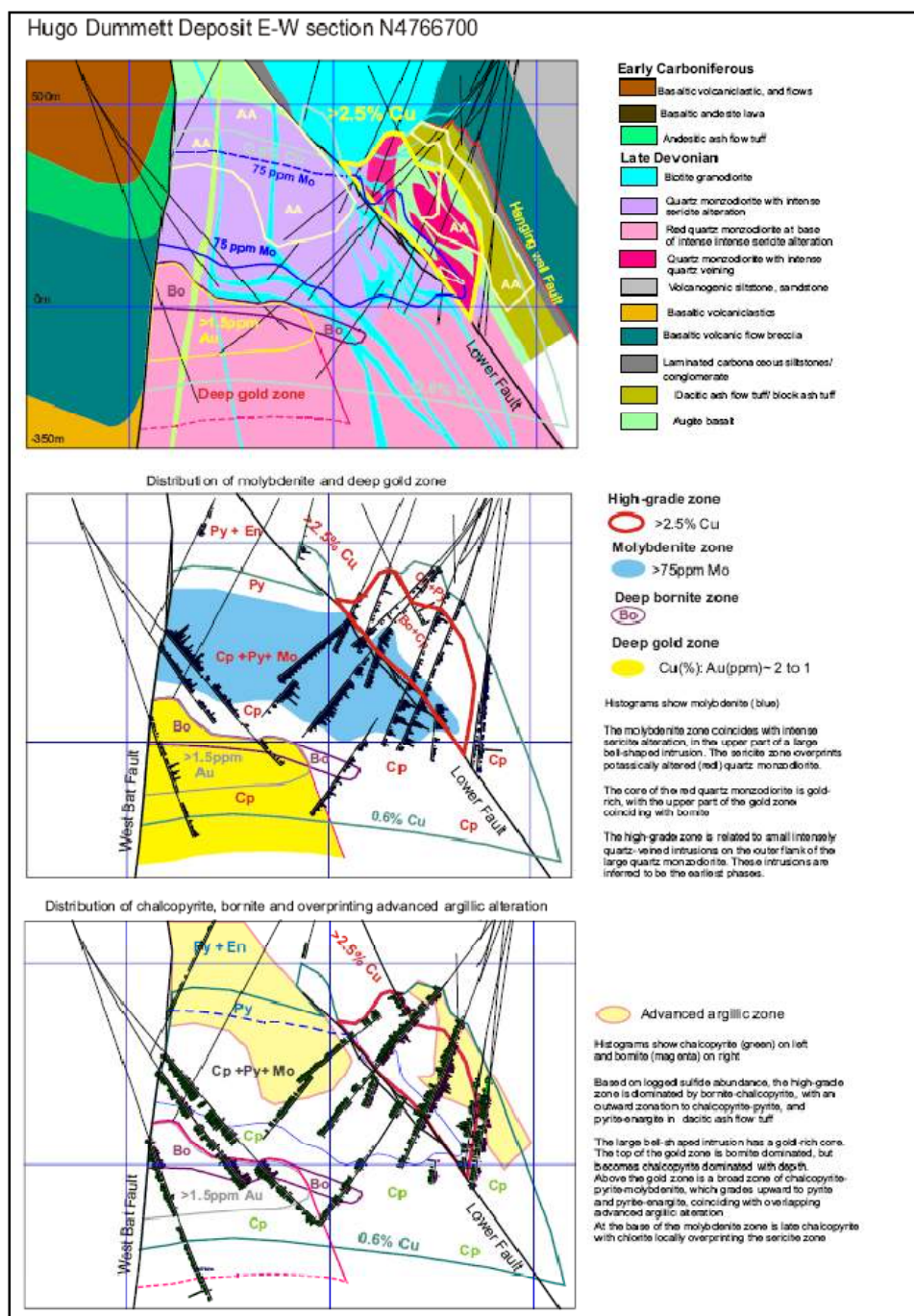
Bornite is dominant in highest-grade parts of the deposit (3-5% Cu) and is zoned outward to chalcopyrite (2%). At grades of <1% Cu, pyrite–chalcopyrite ± enargite, tennantite, bornite, chalcocite, and rarely covellite occur, hosted mainly by advanced argillically-altered basaltic tuff.

The high-grade bornite zone comprises relatively coarse bornite impregnating quartz and disseminated in wall rocks, usually intergrown with subordinate chalcopyrite. Pyrite is rare or absent, except in local areas where the host rocks are advanced argillically altered. Whereas chalcocite is commonly found with bornite at Hugo South, it is less common at Hugo North. In addition, high-grade bornite is associated with minor amounts of tennantite, sphalerite, hessite, clausthalite, and gold. These minerals occur as inclusions or at grain boundaries.

Elevated gold grades in the Hugo North deposit occur within the up-dip (western) portion of the intensely veined, high-grade core, and within a steeply-dipping lower zone cutting through the western part of the quartz monzodiorite. Quartz monzodiorite in the lower zone exhibits a characteristic pink to buff colour, with a moderate intensity of quartz veining (5 to 25% by volume). This zone is characterized by finely disseminated bornite and chalcopyrite, although in hand specimen the chalcopyrite is usually not visible. The sulfides are disseminated throughout the rock in the matrix as well as in quartz veins. The fine-grained bornite has a black “sooty” appearance. The red coloration is attributed to fine hematite dusting, mainly associated with albite.

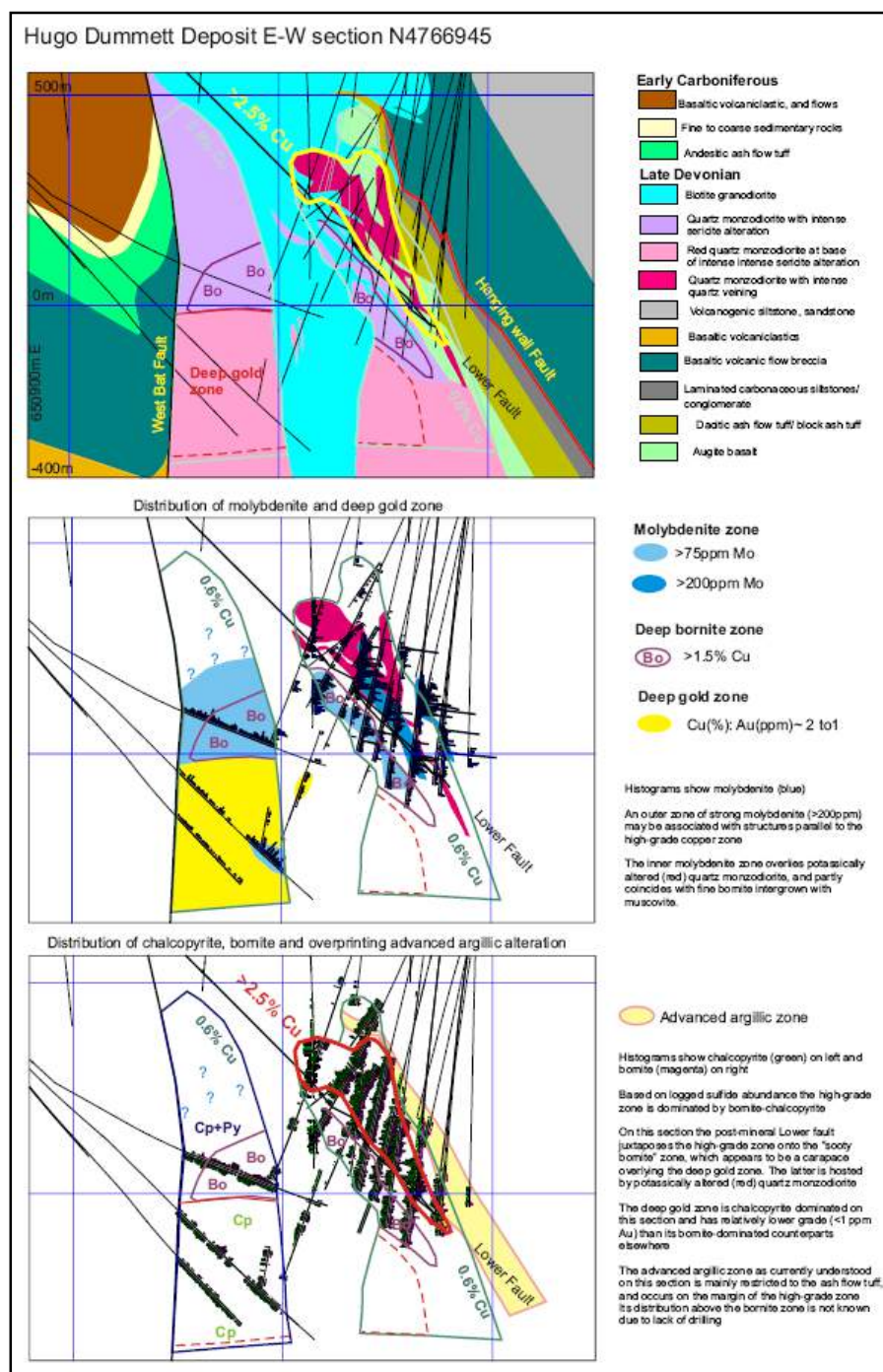
The geology and distribution of molybdenite, the deep gold zone, chalcopyrite, bornite and advanced argillic alteration from three representative sections at Hugo North are shown in Figure 7.11 to Figure 7.20.

Figure 7.11 Hugo North Deposit Geology and Mineralization, Section 4766700N



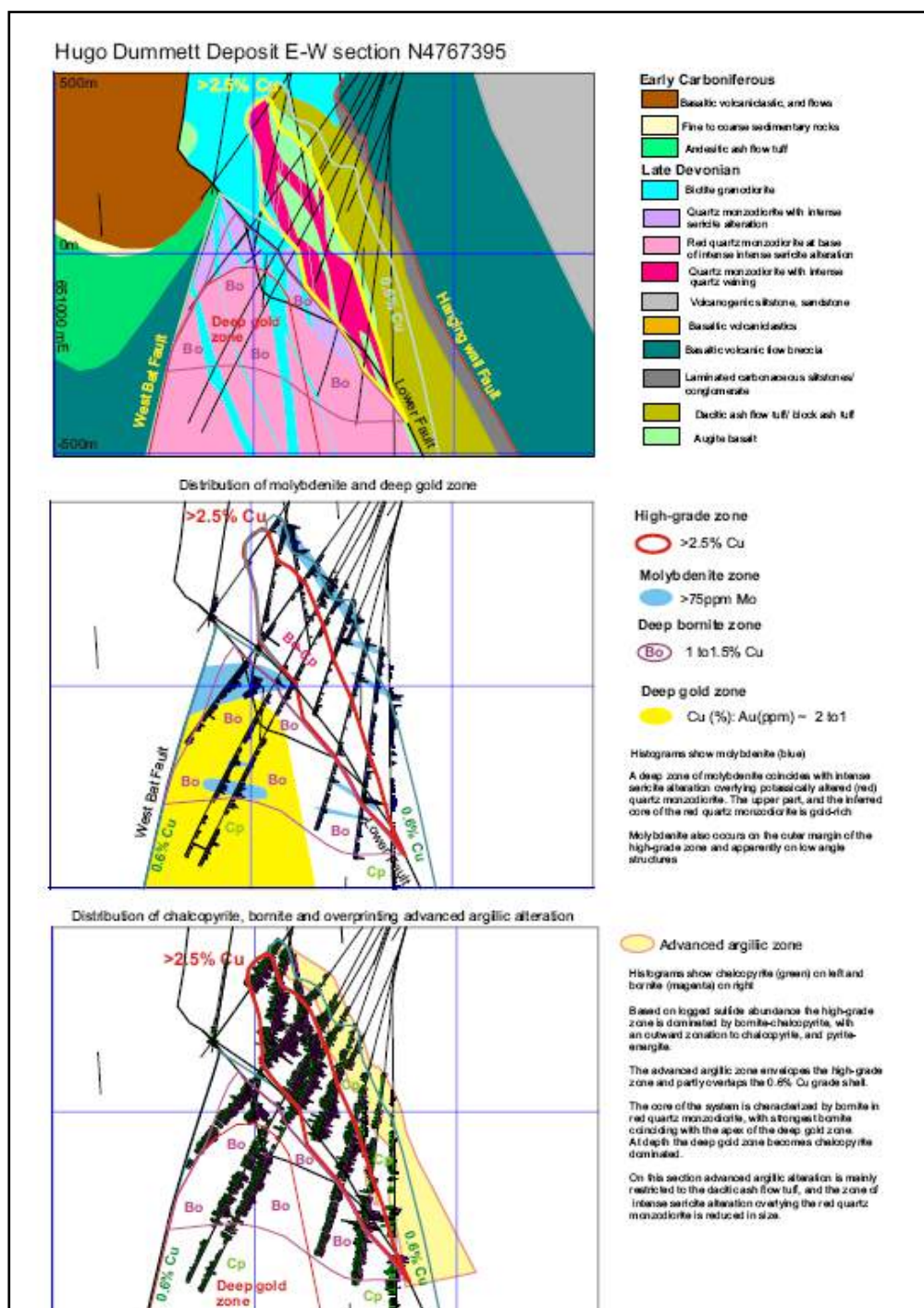
Note: Figure provided by Ivanhoe.

Figure 7.12 Hugo North Deposit Geology and Mineralization, Section 4766945N



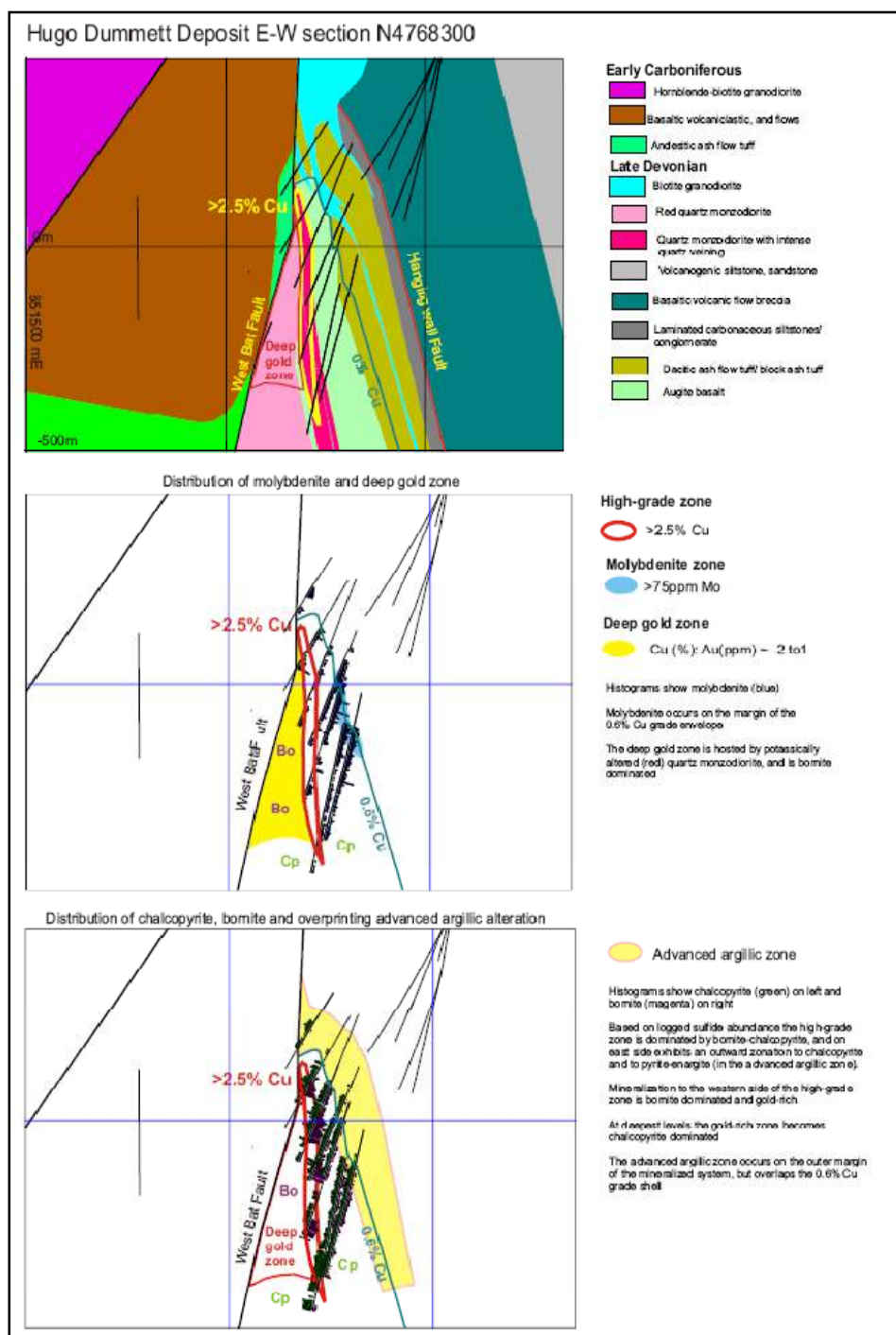
Note: Figure provided by Ivanhoe.

Figure 7.13 Hugo North Deposit Geology and Mineralization, Section 4767395N



Note: Figure provided by Ivanhoe.

Figure 7.14 Hugo North Deposit Geology and Mineralization, Section 478300N



The Hugo North Deposit is characterized by copper–gold porphyry and related styles of alteration similar to those at Hugo South. This includes biotite–K-feldspar (K-silicate), magnetite, chlorite–muscovite–illite, albite, chlorite–illite–hematite–kaolinite (intermediate argillic), quartz–alunite–pyrophyllite–kaolinite–diaspore–zunyite–topaz–dickite (advanced argillic), and sericite–muscovite zones.

The distribution of alteration zones is similar to that in the Hugo South Deposit, except that the advanced and intermediate argillic zones are more restricted and occur mainly along the outer margin of the intrusive system.

Chlorite–illite marks the outer boundary of the advanced argillic zone. It occurs mainly in the coarse, upper part of the basaltic tuff/breccia.

Quartz–pyrophyllite–kaolinite–dickite (advanced argillic) is hosted mainly in the lower part of the basaltic tuff, although on some sections at Hugo North it extends into strongly veined quartz monzodiorite. The advanced argillic zone is typically buff or grey, and late dickite on fractures is ubiquitous. Within the advanced argillic zone, a massive quartz–alunite zone forms a pink–brown bedding-parallel lens.

Topaz is widespread as late alteration controlled by structures cutting both the advanced and intermediate argillic zone. In certain areas topaz appears to replace parts of the quartz–alunite zone. In addition topaz may also occur disseminated with quartz–pyrophyllite–kaolinite. Strong topaz zones are mottled buff or light brown and sometimes vuggy.

Hematite–siderite–illite–pyrophyllite–kaolinite–dickite (intermediate argillic) is an inward zonation from the advanced argillic zone, and is commonly hosted by augite basalt but may also occur in basaltic ash flow tuff. Hematite usually comprises fine specularite and may be derived from early magnetite or Fe-rich minerals such as biotite or chlorite.

Hematite–chlorite–illite–(biotite–magnetite) (chlorite) is transitional to the intermediate argillic zone and is commonly hosted by augite basalt. It is characterized by a green colour, and relict hydrothermal magnetite, either disseminated or in veins.

Muscovite–illite (sericite) generally occurs in the quartz monzodiorite intrusions and is a feature of the strongly mineralized zone. Alteration decreases with depth in the quartz monzodiorite.

7.2.3 Heruga Deposit

The Heruga deposit is a Cu, Au, Mo porphyry deposit with overlapping zones of mineralization consisting of a Mo-rich carapace at higher elevation with increasing Au-rich mineralization at depth. Copper-rich mineralization occurs at intermediate elevations and overlaps both the Mo rich and Au rich zones. At a 100 ppm Mo cut-off the Mo zone has a vertical extent of 300 to 400 m. At a 0.3% Cu cut-off the top of the copper zone roughly correlates with the top of the Mo zone but extends to deeper depths, for a total vertical extent in the order of 600 m. At a 0.3 g/t Au cut-off the top of the gold zone is generally about 280 m below the top of the Cu and Mo zones, creating a small overlap with the Mo zone in areas, and a more significant overlap of the Cu zone. The bottom of the gold zone generally extends below the bottom of the Cu zone.

by about 200 m, giving the gold zone a total vertical extent of about 450 m. The three zones together combine for a continuous zone of mineralization with a vertical extent of close to 800 m. Post mineral north-north-east trending subvertical faults have offset mineralization into two discrete structural blocks. Mineralization remains open down dip to the east of the eastern block although trending towards lower grades in that direction. The western block is assumed to terminate against a parallel north-north-east trending structure, intersected at higher elevations in only a few drillholes, and remains poorly defined at the depth of the deposit. The high grade gold core in the western block remains poorly defined and requires further definition.

There is a rough spatial correlation between metal and alteration zonation and the two may be genetically linked. Generally speaking at deeper levels mineralization consists of chalcopyrite and pyrite in veins and disseminated within biotite-chlorite-albite-actinolite altered basalt or sericite-albite altered quartz monzodiorite. Selvages of salmon pink albite alteration occur with chalcopyrite associated with higher grade mineralization in both basalt and quartz monzodiorite. The higher levels of the orebody are overprinted by strong quartz-sericite-tourmaline-pyrite alteration where mineralization consists of disseminated and vein controlled pyrite, chalcopyrite and molybdenite.

Gold may correlate with strong biotite alteration at the deeper levels of the deposit. Au to Cu ratios (ppm:%) are generally less than one throughout the upper Mo-rich part of the deposit, i.e. within the >100 ppm Mo shell. Below this, the gold to copper ratios reflect an inverse relationship between Au and Cu. The Au to Cu ratio increases rapidly with depth from one to greater than 10, reflecting decreasing copper grades and increasing gold grades. In the deepest parts of the deposit chalcopyrite diminishes to only trace amounts, and is replaced by pyrite as the predominant sulfide. This would suggest that Au does not correlate with chalcopyrite, but is perhaps associated with pyrite at the deepest levels. East of the Bor Tolgoi Fault high Au:Cu values occur within and marginal to a quartz monzodiorite intrusion. However; west of the Bor Tolgoi Fault, a small quartz monzodiorite dyke was intersected in only one drillhole and a possible association is less apparent.

A paragenetic framework suggested by Hopper (2008) is simplified as follows:

- Pre to early mineral alteration facies predating intrusion of Qmd (dark coloured alteration):
 - Early pervasive biotite magnetite actinolite alteration with quartz magnetite-chalcopyrite-pyrite.
 - Retrograde chlorite-epidote-sericite.
- Intermineral alteration facies:
 - Quartz-magnetite-chalcopyrite-bornite-pyrite-molybdenite and associated quartz-albite, biotite, actinolite alteration.
- Inter to late mineral alteration facies (light coloured alteration):
 - Pyrite sericite pyrophyllite alteration (molybdenite shell seems to coincide with this stage).
- Post Mineral:
 - Quartz-calcite veinlets.

Mineralized veins have a much lower density at Heruga than in the more northerly Southern Oyu and Hugo Dummett deposits. Lewis (2008) reports that some quartz veins show a weak preferred orientation, but in general most occur as stockworks with no visible preferred orientation.

High grade copper and gold intersections show a strong spatial association with contacts of the mineralized quartz monzodiorite porphyry intrusion in the southern part of the deposit, occurring both within the outer portion of the intrusion and in adjacent enclosing basaltic country rock.

There is no oxide zone at Heruga, nor is there any high-sulfidation style mineralization known to date. Figure 7.15 to Figure 7.22 show drilling plans and typical sections for the Heruga deposit.

Figure 7.15 Heruga Deposit Metal Zonation, Section 4758400N

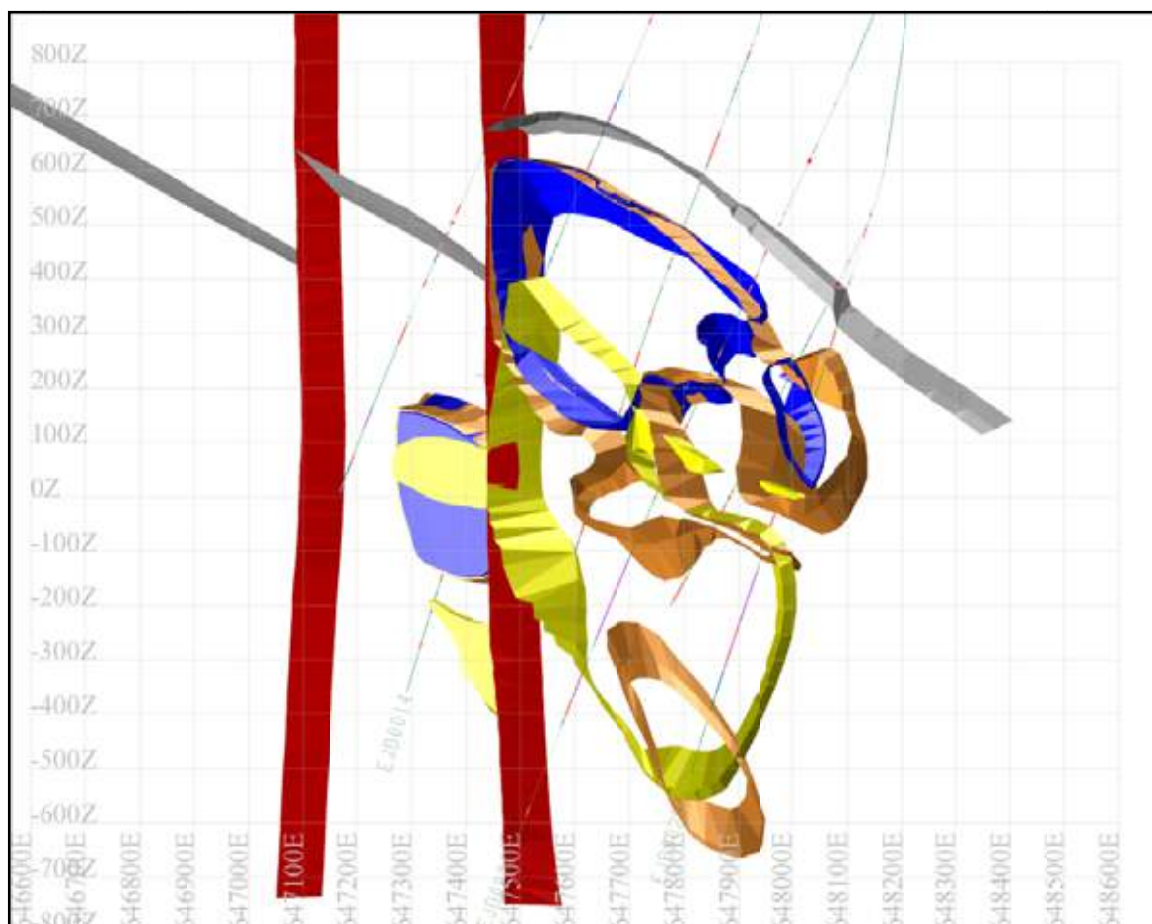
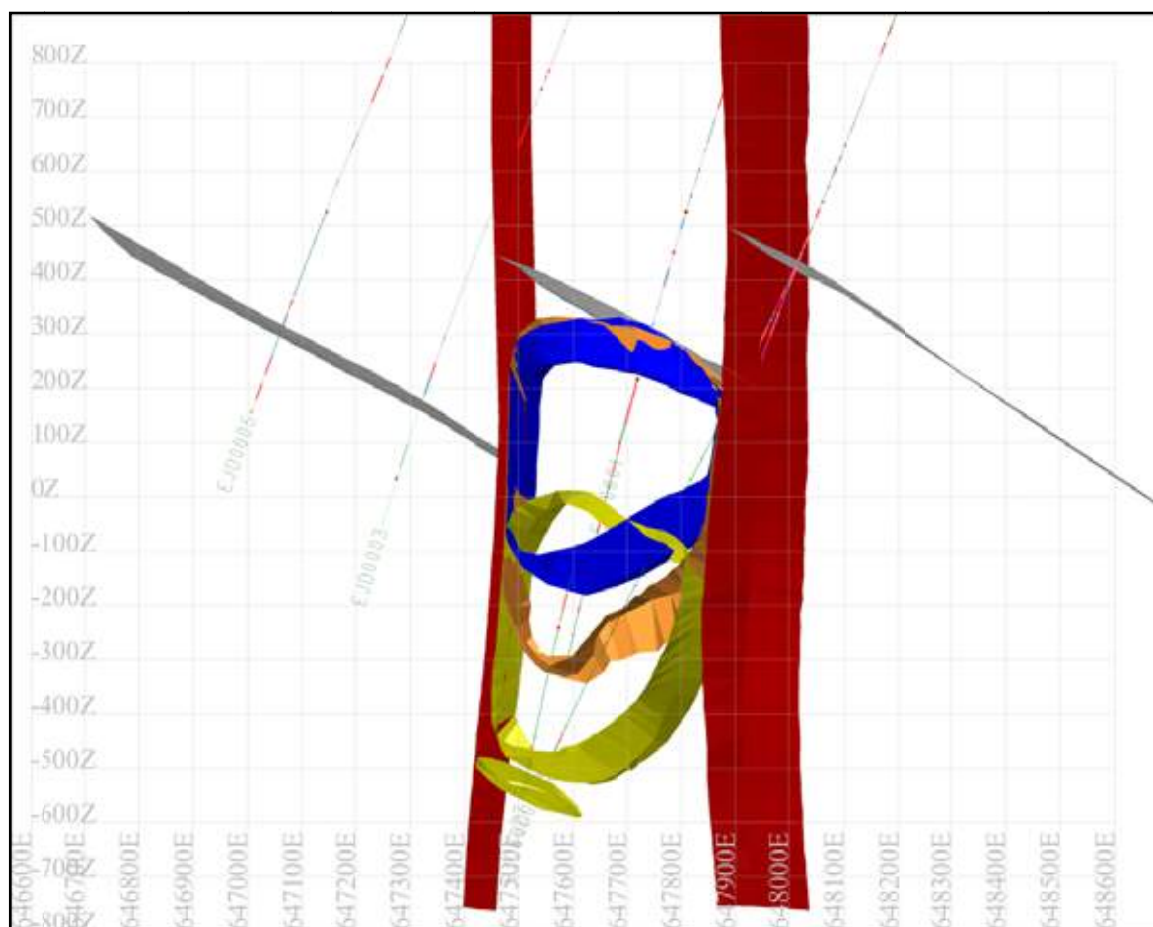


Figure 7.16 Heruga Deposit Metal Zonation, Section 4759300N



HERUGA DEPOSIT
SECTION N473440
GEOLOGY AND ASSAY INTERCEPT

Scale: 1:100,000
Datum: NAD 83
Units: Meters

Assay Intercept

- > 3% Cu or > 3 g/t Au
- > 2% Cu or > 2 g/t Au
- > 1% Cu or > 1 g/t Au
- < 1% Cu or < 1 g/t Au

344 : 0.87, 3.8, 3.8 + m : g/t Au, % Cu, Cu/gt

see previous and/or next sections

Assay Histogram

- 0 - 0.25 g/t Au
- 0.25 - 0.50 g/t Au
- 0.50 - 0.75 g/t Au
- 0.75 - 1.00 g/t Au
- 1.00 - 2.00 g/t Au
- > 2 g/t Au
- 0 - 0.25 % Cu
- 0.25 - 0.50 % Cu
- 0.50 - 0.75 % Cu
- 0.75 - 1.00 % Cu
- 1.00 - 2.00 % Cu
- > 2 % Cu

Deposit Area Intrusions

- diabase
- basalt
- diabase porphyry, microdykes
- phyllite, felsic, quartzite/diorite porphyry dykes
- hornblende breccia andesite
- igneous granulite
- sericitic diorite granulite
- quartz monzonite
- sericitic quartz monzonite
- strongly-veined quartz monzonite
- fine-grained intensely-veined quartz monzonite
- gabbro-rich quartz monzonite

DEPOSIT AREA

Heruga deposit area

- 1.00 g/t Au
- 0.50 g/t Au
- 0.25 g/t Au
- 0.125 g/t Au
- 0.0625 g/t Au
- 0.03125 g/t Au
- 0.015625 g/t Au
- 0.0078125 g/t Au
- 0.00390625 g/t Au
- 0.001953125 g/t Au
- 0.0009765625 g/t Au
- 0.00048828125 g/t Au
- 0.000244140625 g/t Au
- 0.0001220703125 g/t Au
- 0.00006103515625 g/t Au
- 0.000030517578125 g/t Au
- 0.0000152587890625 g/t Au
- 0.00000762939453125 g/t Au
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IVANHOE MINES MONGOLIA INC.

HERUGA DEPOSIT

SECTION N4759700

GEOLOGY AND ASSAY INTERCEPT

Date: 08 June 2008 Drawn by: Q7/Dean
Author: Q7/Dean Scale: 1:2500

OTD1489 OTD1485 OTD1490

124m: 0.08 g/t Au, 0.43% Cu, 96 ppm Mo, 0.53% CuEq
42m: 0.18 g/t Au, 0.65% Cu, 92 ppm Mo, 0.82% CuEq
36m: 0.17 g/t Au, 0.47% Cu, 78 ppm Mo, 0.61% CuEq
35m: 0.82 g/t Au, 0.39% Cu, 72 ppm Mo, 0.82% CuEq
68m: 0.27 g/t Au, 0.50% Cu, 288 ppm Mo, 0.83% CuEq
78m: 0.34 g/t Au, 0.56% Cu, 175 ppm Mo, 0.74% CuEq
222m: 0.66 g/t Au, 0.43% Cu, 77 ppm Mo, 0.89% CuEq
92m: 0.94 g/t Au, 0.42% Cu, 63 ppm Mo, 1.05% CuEq
335m: 0.28 g/t Au, 0.53% Cu, 154 ppm Mo, 0.79% CuEq
76m: 0.71 g/t Au, 0.53% Cu, 90 ppm Mo, 1.06% CuEq

Assay Intercept

- > 3% Cu eq > 3 g/t Au
- > 2% Cu eq > 2 g/t Au
- > 1% Cu eq > 1 g/t Au
- > 0.5% Cu eq > 0.5 g/t Au

344: 0.87, 3.8, 3.8 ± m: g/t Au, % Cu, CuEq
see previous and/or next sections

Assay Histogram

- 0 - 0.25 g/t Au
- 0.25 - 0.50 g/t Au
- 0.50 - 0.75 g/t Au
- 0.75 - 1.00 g/t Au
- 1.00 - 2.00 g/t Au
- > 2 g/t Au
- 0 - 0.25 % Cu
- 0.25 - 0.50 % Cu
- 0.50 - 0.75 % Cu
- 0.75 - 1.00 % Cu
- 1.00 - 2.00 % Cu
- > 2 % Cu

Deposit Area Intrusions

- diabase
- basalt
- basalt porphyry, trachyte
- hyaline, felsic, quartz-hyaline porphyry dykes
- hornblende-biotite gneiss
- quartz gneiss
- hornblende-biotite gneiss
- quartz monzonite
- amphibole-quartz monzonite
- strongly altered quartz monzonite
- fine-grained felsic-sand quartz monzonite
- pinkish quartz monzonite

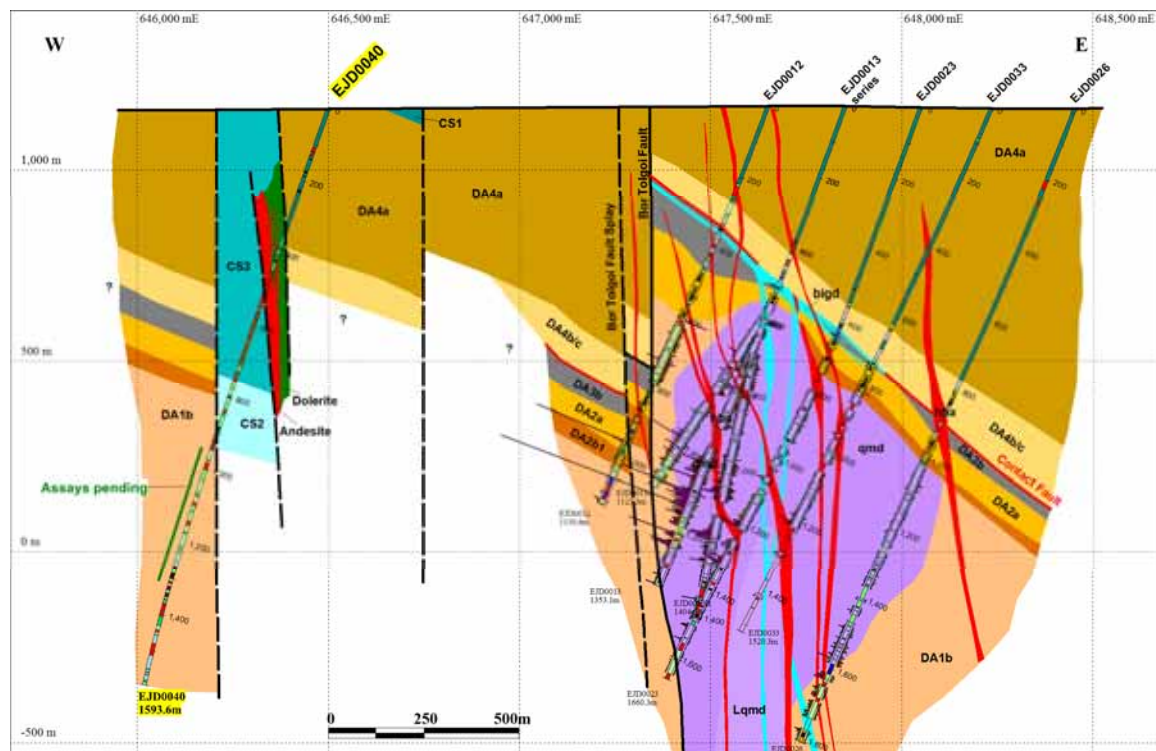
DEPOSIT AREA

Location map of the Heruga Deposit

Legend

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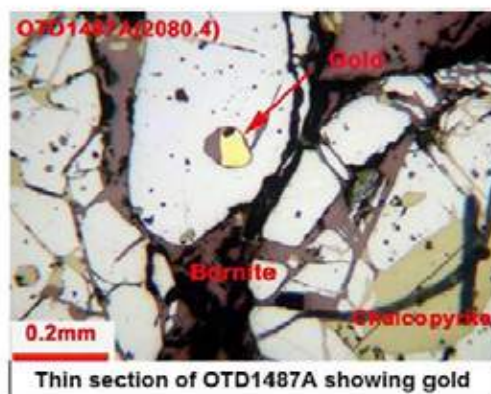
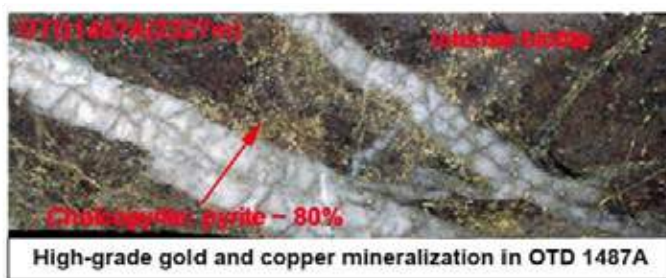
Figure 7.21 Heruga SW EJD0040 Section



7.2.4 Heruga North Mineralization

Heruga North mineralization consists of strong chalcopyrite-pyrite mineralization with trace molybdenite occurring in veinlets and disseminated. Host rock consists of intensely quartz veined massive augite basalt with strong pervasive biotite magnetite alteration and minor pink albite alteration on vein and fracture selvages. It represents the northward extension of the mineralization at Heruga and is similar in its characteristics.

Figure 7.23 Heruga North Mineralization Photos



7.2.5 References

The information for this section is in the report Oyu Tolgoi Technical Report, March 2008, with the exception of the following additional references:

Hopper, D., 2008. Preliminary Paragenetic Framework Based on Drill Holes EJD0013, 21, 24, 27 and 34. Rio Tinto Exploration Power Point Presentation.

8 DEPOSIT TYPES

8.1 Deposits

The Oyu Tolgoi deposits display copper–gold porphyry and related high-sulfidation copper–gold deposit styles. Copper–gold porphyry deposits are low-grade bulk tonnage deposits where copper sulfides are finely disseminated or deposited in anastomosing veins and fractures in a large volume of rock. These deposits are amenable to large-scale open-pit or underground bulk mining methods.

The Oyu Tolgoi porphyry copper deposits display a range of mineralization styles, alteration characteristics, and deposit morphologies despite having formed in close spatial and temporal association. These distinctions probably reflect differences in structural controls, host rock lithology, and depth of formation. Structural influences account for most differences in shape and distribution of mineralization within the deposits. In general, high-sulfidation mineralization and associated advanced argillic alteration are most common within the basaltic tuff and breccia, in the upper part of the quartz monzodiorite where it intrudes to levels high in the stratigraphic succession, and in narrow structurally controlled zones. In contrast the more typical copper–gold porphyry style alteration and mineralization tend to occur at deeper levels, predominantly within massive augite basalt and quartz monzodiorite.

The Southwest deposit, particularly the gold-rich core zone, occurs as a steep cylindrical body typical of many porphyry copper deposits. Southwest is localized within a dilational transfer zone linking movement between the north-east-striking West Bounding and East Bounding faults. The strong structural control exerted by these faults is reflected in the abundant sheeted veins that show a high degree of preferred orientation, the pipe-like form of the gold-rich core to the deposit, and the spreading of low-grade copper and gold values along the bounding fault zones.

The Central deposit occupies a structurally intact block within which no significant internal fault disruption has been identified. It forms an irregular bowl- to slightly funnel-shaped zone of mineralization characterized by high-sulfidation (pyrite–covellite–chalcocite–enargite) and copper–gold (chalcopyrite–gold) porphyry styles, as well as a chalcocite enrichment blanket. Although the preferred orientation of quartz veins implies strong structural control within the deposit, the geometry of intrusive phases is irregular. Weathering of the Central deposit has produced an oxide zone 40–60 m thick, usually devoid of mineralization, overlying a chalcocite enrichment zone up to 80 m thick.

The South and Wedge deposits are located on the eastern flank of the north-north-east-trending mineralized corridor at Oyu Tolgoi, within several structural blocks cut and bounded by minor north-east-striking subvertical faults. Despite the numerous faults cutting these deposits, the lesser degree of preferred vein orientation relative to other deposits implies that structural control was weaker during mineralization.

The Hugo Dummett deposits have several features unusual to porphyry copper systems, including the following:

- Anomalously high copper and gold grades, particularly in the northern part.

- An unusual, weakly altered, pre-mineral sedimentary cover sequence that lies just above the porphyry system.
- Quartz + sulfide vein contents always exceeding 15%, and commonly over 90%, in the high-grade portion of the deposit.
- A highly-elongate, gently-plunging, tabular shaped, high-grade stockwork system.

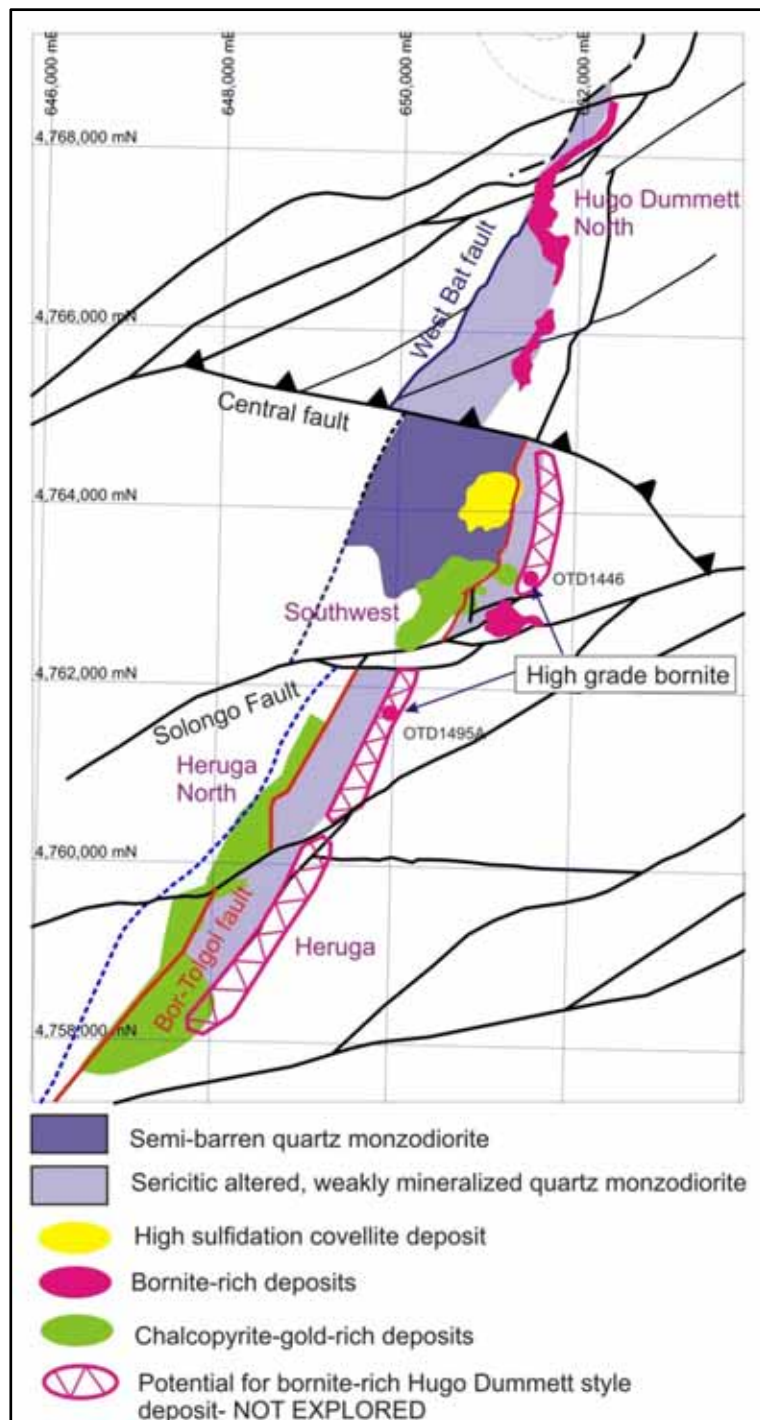
The formation of the high-grade portion of the Hugo Dummett deposits as a tabular, intensely veined, subhorizontal body contrasts markedly with most porphyry copper deposits, which tend to have steep, roughly cylindrical or elongate forms. This unusual form could be the result of emplacement within a passive to slightly compressional horizontal stress regime, synchronous with thrust faulting. This would inhibit the upward migration of hydrothermal fluids, leading to the formation of a broad subhorizontal zone of fluid over-pressuring with attendant hydraulic fracturing and vein formation. This fluid focusing also may have contributed to the unusually high copper grades in these deposits.

The main Oyu Tolgoi deposits are described above as “porphyry copper-gold” and “high sulfidation copper-gold”. Heruga is distinctly higher in molybdenum, and is best described as a “porphyry copper-gold-molybdenum” system with a Mo rich carapace at higher elevations overlying Au rich mineralization at depth. Section 7.2.3 provides further explanation of this distinction.

Heruga East Exploration Model

Figure 8.1 shows the relationship of mineralisation along the OT trend with quartz monzodiorite intrusions. It is evident that bornite-rich mineralisation generally occurs on the eastern side of the quartz monzodiorite, as observed at the Hugo North, South Oyu, and Heruga North (OTD1495A). The eastern side of Heruga and Heruga North are the current focus of exploration drilling.

Figure 8.1 Plan View of Bornite-rich Exploration Targets along the Eastern Flank of Qmd



9 EXPLORATION

9.1 Oyu Tolgoi

9.1.1 Introduction

9.1.1.1 Oyu Tolgoi Property

Exploration on the Oyu Tolgoi Property has been conducted principally by two companies; BHP from late 1996 to mid-1999 and Ivanhoe since 2000. Early-stage exploration by BHP was focused on property-wide target generation and was included airborne geophysical surveys, remote sensing studies, regional mapping and sampling, and 23 core holes (3,902 m) in the Southern Oyu Deposits. After acquiring the Property in 2000, Ivanhoe embarked on a major exploration program, which has continued to the present date. Over this period, Ivanhoe completed over 2,315 drillholes comprising 1,002,500 m, most of which are core holes, and through this, outlined a series of world-class Cu–Au porphyry deposits referred to as the Southern Oyu, Hugo Dummett, and Heruga Deposits. Accompanying the drilling were surface mapping programs, regional rock, soil and stream sediment geochemical surveys, remote sensing studies, regional airborne and detailed ground geophysical programs, and studies, including metallurgical test work, environmental and geotechnical, and exploration shaft sinking, designed to eventually bring the deposits through to feasibility level.

The principal deposits that were discovered on the Oyu Tolgoi Property are the Southern Oyu (Southwest, Central, South Oyu, Wedge), Hugo Dummett (Hugo North, Hugo South), and Heruga Deposits. A majority of the exploration work has been focused on these areas. Drilling of magnetic and IP/resistivity anomalies has been the principal technique used by Ivanhoe to advance the Property to its current level.

9.1.1.2 Shivee Tolgoi Joint Venture Property

Early exploration over the current Shivee Tolgoi Property was initiated by BHP during its 1997 regional exploration program. Since Entrée acquired the mineral rights in 2001 and Ivanhoe signed the Earn-in agreement in 2004, exploration has been mainly by geochemical, remote sensing, geophysical, and drilling methods. These include satellite image interpretation, detailed ground magnetics, Bouguer gravity, pole-dipole and gradient array induced polarization (IP) as well as core and RC drilling. Gradient array IP was conducted on 200 m spaced lines, with electrode spacings of up to 20 km. Drillholes were targeted to test strike projections of the Hugo North deposit, IP chargeability targets, or structural zones. Similar to the Oyu Tolgoi Property, drilling, followed by geophysical methods have been the most important exploration tools. The principal deposit that has been discovered on the Shivee Tolgoi Property is the northerly extension of the Hugo North Deposit Extension. The Ulaan Khud prospect has also been discovered some 10 km to the north near the northern boundary of the property.

9.1.2 Exploration Activities

Various exploration methods, using a range of parameters have been utilized at both the Oyu Tolgoi and Shivee Tolgoi JV Properties. These have been described in varying detail in previous Technical Reports (Juras 2003a, 2003b, 2003c, 2004, 2005; Hodgson

2004; Gingrich 2005; Peters 2006; Blower, 2006a, 2006b) and are briefly summarized in the following subsections.

9.1.2.1 Coordinates and Datum

Coordinates used by Ivanhoe for exploration on the Project are mostly UTM coordinates with the datum set to WGS-84, Zone 48N. The boundary coordinates of the Mining and Exploration Licences are defined by latitude and longitude coordinates. The official Mongolian survey datum is "MSK42", using the Baltic mean sea level as the elevation datum. Ivanhoe confirms that the coordinates obtained using the MSK42 data are almost identical to those using WGS-84, Zone 48N.

9.1.2.2 Remote Sensing and Satellite Imagery

In 2001, Ivanhoe commissioned Pacific Geomatics from Vancouver to produce 1:100,000 scale Satellite LandSat TM images and a structural and alteration interpretation over a 1,500 sq. km area centered on the Project. These data are integrated into a GIS database and have been used to aid in the structural interpretation of the Property and for alteration mapping.

In 2003, Ivanhoe commissioned Pacific Geomatics to also provide Quickbird imaging over the entire Oyu Tolgoi mining licence.

9.1.2.3 Prospecting and Lithogeochemistry

Although a great deal of prospecting and lithogeochemical sampling has been completed by Ivanhoe and previous companies on the Oyu Tolgoi Property. Much of this early stage work was completed over the surface exposures of the Southern Oyu deposits and has been superseded by the drilling and on-going advanced level studies.

Entrée began exploration on the Shivee Tolgoi Property during the 2002 and 2003 field seasons with prospecting and reconnaissance rock chip sampling. In total, 75 rock chips were taken. Although numerous, up to 2 m wide, quartz veins and vein breccia systems were identified along the margin of the "Copper Flats" intrusive complex; none carried any appreciable mineralization. However; a sample of epidote-magnetite vein taken in July 2002 (ST015 – center of Copper Flats Grid) assayed 0.39 g/t Au and 105 ppm Mo (Blower, 2006b).

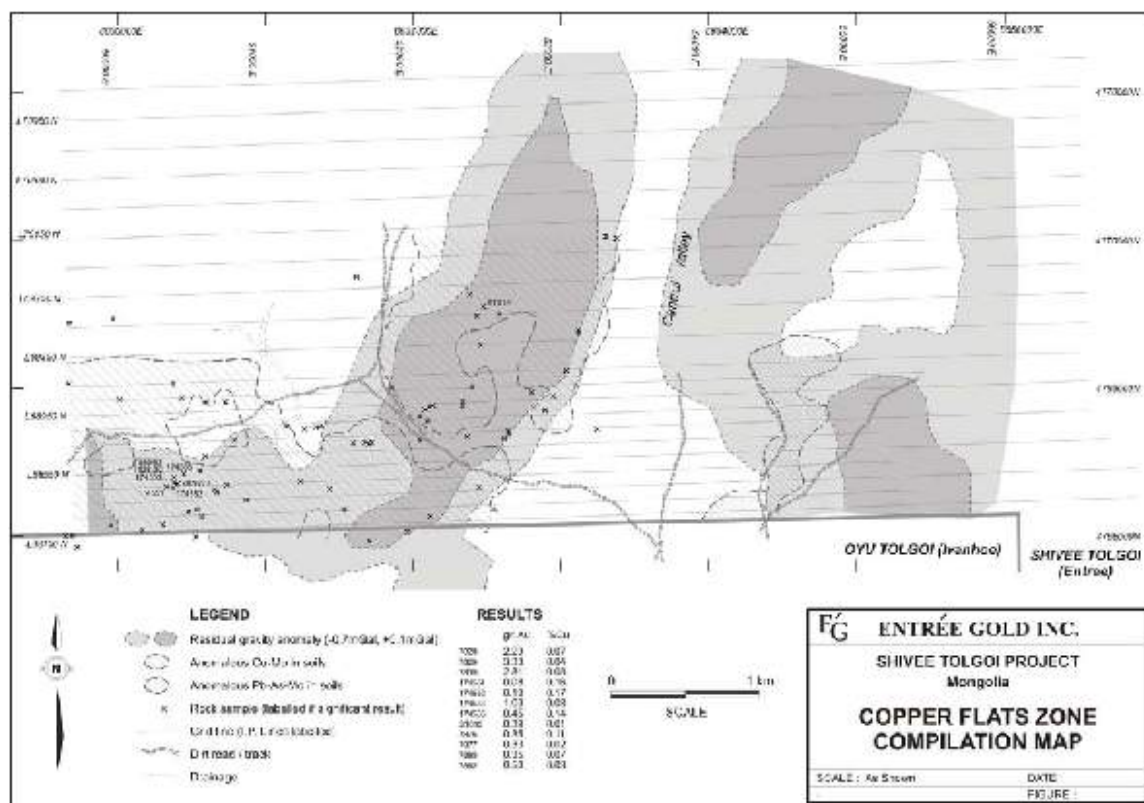
9.1.2.4 Soil Geochemistry

Ivanhoe reports that early-stage soil geochemical sampling was completed over various grids on the Oyu Tolgoi Property; however, this early-stage work has been superseded by the drilling and ongoing advanced level studies and is not discussed in this report.

Within the Shivee Tolgoi Property, Entrée completed test soil sampling in 2002 at 50 m intervals along four test lines (68150N, 68350N, 68550N and 68950N; grid is truncated UTM). Based on this initial survey, in 2003 the grid was expanded to 70150N (11 lines) and was sampled at 50 m intervals. The Cu-Mo geochemical response roughly correlates with the surface projection of Hugo North mineralization and a -0.7 mGal Bouguer gravity anomaly and increases in proximity to the contact of the previously

defined intrusive rocks (Figure 9.1). In the central part of the grid, where overburden dominates, responses were more subtle. The eastern part of the grid shows irregular W–As anomalies that appear to be spatially related to the Khanbogd intrusive complex.

Figure 9.1 Copper Flats Rock Sample, Soil and Bouguer Gravity Anomaly Plan



After Cann, 2004 in Blower 2006a.

9.1.2.5 Geophysics

(1) Airborne Geophysics

Geophysical surveying at Oyu Tolgoi Project was initiated by BHP in 1996. An airborne magnetometer survey was flown by High-Sense Geophysics Limited (now part of Fugro Airborne Surveys) at a height of approximately 100 m on 300 m spaced, east-west lines over approximately 1,120 km² of BHP's mineral concession. The survey provided good resolution of the magnetic features to facilitate geological and structural interpretation across the concessions. The results of this survey have been superseded by more recent surveys and are not described in this report.

The Oyu Tolgoi test area, which is approximately 25 km north-south by 20 km east-west, was flown on 400 m spaced, north-south lines that included all of the Oyu Tolgoi, (6709A), the Shivee Tolgoi (15226A) and Javkhlant (15225A) mining licences in the Entrée–Ivanhoe Mines Joint Venture Agreement. None of the 100% Entrée-owned concessions were flown and any data that may have been captured by overflight lines was clipped from survey results provided to Ivanhoe Mines.

The survey also incorporates airborne magnetics and shows the Southwest Oyu deposit as a gravity-low, magnetic-high response, whereas the Hugo Dummett deposits coincide with a gravity high and a magnetic-low response. BHP Billiton, using advanced computer processing algorithms developed in-house, combined the gravity and magnetic responses to produce a “Solid Earth Model” or SEM representative of the Oyu Tolgoi deposits. This SEM was applied to regionally-acquired data to search for similar features that could be representative of a buried Oyu Tolgoi-style deposit.

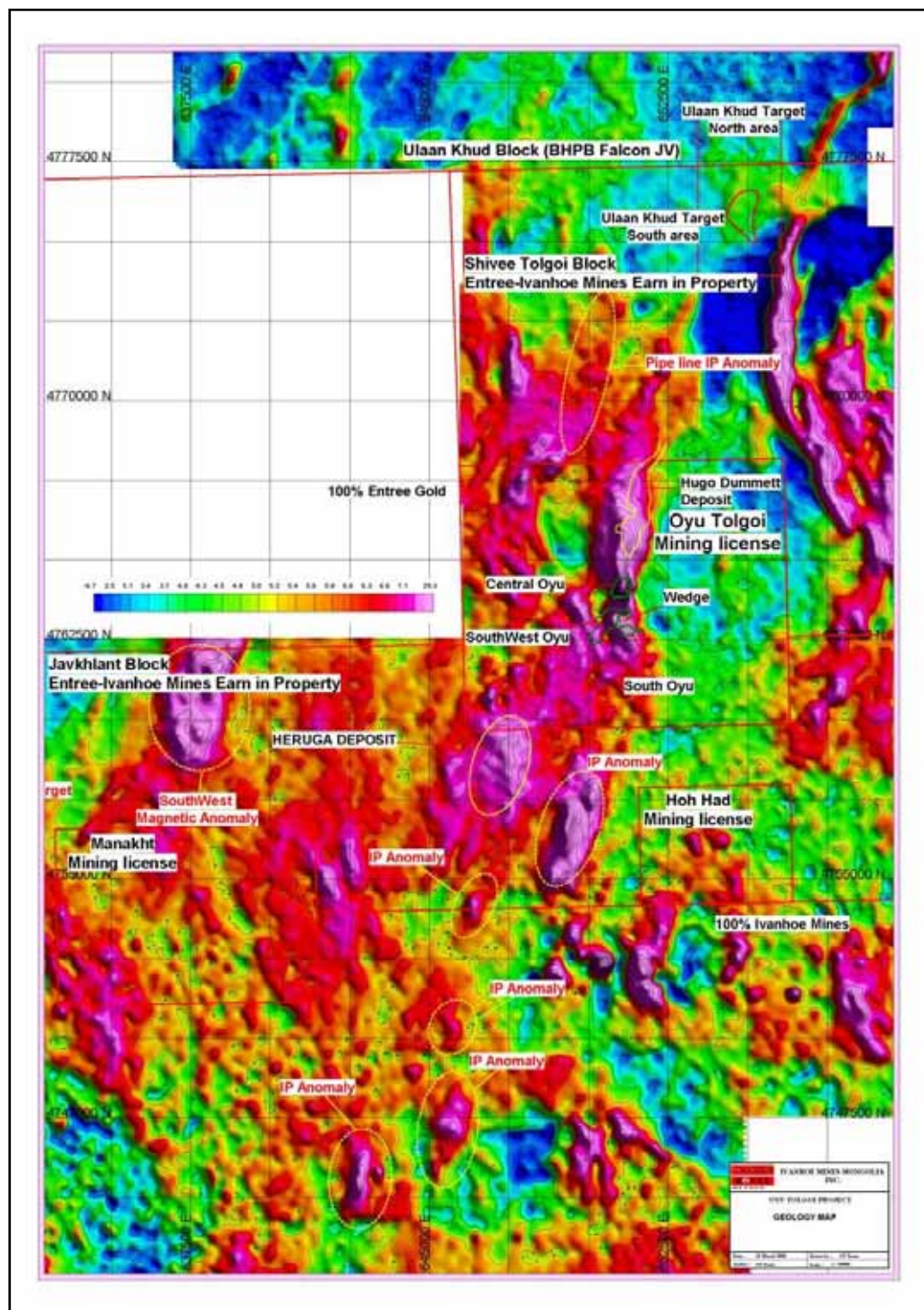
(2) Ground Geophysics – Oyu Tolgoi Property

In 1996, BHP undertook a ground magnetometer survey and an induced polarization (IP) survey utilizing a single gradient array with a 2,000 m AB spacing. Both surveys were conducted on east-west lines surveyed by a local Mongolian surveying team at 250 m spacing. The surveys covered what eventually became the Southern Oyu deposits, but did not extend into the “Far North” region where the Hugo Dummett deposits were discovered.

Ivanhoe initiated geophysical exploration after the discovery of the high-grade, gold-rich portion of the Southwest Oyu deposit in June 2001. The gold-rich, chalcopyrite-dominant mineralization discovered in drillhole OTRCD150 was determined to be the source of the IP chargeability anomalies previously delineated by the BHP survey. The presence of hydrothermal magnetite with the copper–gold mineralization was the probable source of the intense magnetic anomalies defined over Southwest Oyu and possibly the South Oyu Deposits.

Delta Geoscience of B.C., Canada, was mobilized to the property in late 2001 to undertake detailed IP surveying utilizing gradient arrays with multiple electrode spacings, to delineate the vertical extent of the sulfide mineralization being defined by drilling in Southwest Oyu (Figure 9.2).

Figure 9.2 Oyu Tolgoi Project – IP Chargeability



The detailed IP surveys initially covered a 3 km east-west by 4 km north-south block encompassing the Southwest, South, Central Oyu, and Far North (Hugo Dummett) target areas.

Ivanhoe acquired two magnetometers and initiated very detailed ground magnetometer surveying using 5 m spaced readings along 25 m spaced, north-south oriented lines over the Southern Oyu target areas. An Ulaanbaatar geophysical team expanded this magnetic survey to cover the entire 8 km x 10 km concession. This team covered the northern half of the concession with east-west oriented lines on 50 m intervals with 25 m spaced readings.

In 2002, the geophysical program was further expanded to include a gravity survey over the Oyu Tolgoi concession block. The survey was controlled by GPS with readings taken on 50 m centers over the core of the concession and 100 m centers over the extremities. The Bouguer map was reduced to residual gravity for contouring. No additional processing or modelling of the data has been completed.

Telluric Electromagnetic (TEM) surveying was conducted over the eastern half of the concession in conjunction with extensive TEM surveying used to define the Cretaceous-aged, semi-consolidated sedimentary basins along the Galbyn Gobi and Gunii Hooloi valleys, south-east and north-east respectively, from Oyu Tolgoi. These basins developed along the East Mongolian Fault system and a splay off of the fault, and form reservoirs for extensive water resources. On the Oyu Tolgoi concession, the TEM work was designed to delineate smaller drainage basins that could have channelled copper-rich surficial waters from the exposed copper deposits during the Cretaceous period. These “pregnant waters” could potentially have precipitated copper into river gravels downstream to form secondary “exotic” copper deposits although to date no such deposits have been discovered.

(3) Zeus Ground Geophysics

A “Zeus” high power gradient array IP survey started at Oyu Tolgoi on 19 June 2009 and terminated on 11 November 2009. During this period 64% of the survey time was spent on 100% OT LLC owned lands with the balance (36%) spent on the Ivanhoe Mines/Entree joint venture lands. The joint venture work time can be further broken down to 26% on Heruga and 10% on Hugo North Extension.

(4) Ground Geophysics – Shivee Tolgoi Joint Venture Property

In 2002, Entrée surveyed the eastern half of lines 68150N and 68950N over the Copper Flats area using IP/resistivity pole-dipole methods and 50 m electrode spacing. Because of highly conductive overlying sediments and deeper than expected basement (>150 m) this initial survey provided only partial information in the main area of interest. Despite these problems, the results were considered to indicate at least two structures that could in part bound the north-trending valley along the eastern side of the Copper Flats grid. These structures were interpreted to potentially be important in controlling porphyry mineralization further south on the Oyu Tolgoi Property (Blower, 2006b).

In 2003, the survey was expanded significantly to 16 lines (68150N to 71150N). This work was mainly conducted using a 50 m electrode spacing; however, this configuration also had problems penetrating the conductive sediments. Therefore, lines 68150N, 68350N and 68750N were resurveyed using a 100 m electrode spacing. This configuration was partially successful and the work clearly:

- Contrasts the sediments underlying the eastern flats area with western intrusive complex.
- Suggests at least two north-north-east-trending structures underlying and bounding the flat Central Valley.
- Partly defines a moderate chargeability anomaly in the north-east corner of the grid marginal to the Khanbogd Complex (Blower, 2006b).

Gravity surveys were completed in 2003 over the central portion of the “Copper Flats” grid by Entree, where, Ivanhoe had noted a strong spatial correlation between positive residual gravity anomalies and mineralized zones (Ivanhoe Mines., 2003), which was a similar response to that shown by the known mineralization on the Oyu Tolgoi Property. Results of the survey are summarized in Figure 9.1; the conclusions being (Bérubé, 2003a, 2003b in Blower, 2006b) that the mineralization identified by Ivanhoe at the Oyu Tolgoi Property continued northwards into the EJV Shivee Tolgoi licence.

In 2003, magnetic readings were taken at 12.5 m intervals (Cann, 2004 in Blower, 2006b) along lines 68150N to 69150N (100 m spacing) and 69350N to 71150N (200 m spacing). Where significant sedimentary cover existed, magnetic relief was subdued. Areas underlain by mainly intrusive rocks showed significant magnetic relief and east-north-east-trending structures were outlined near the south center of the grid. A linear magnetic anomaly trending north-north-west, marginal to the Khanbogd intrusive complex, was interpreted to be related to contact hornfels.

Ivanhoe commenced work on the Property in November 2004, following the signing of the Earn-in Agreement with Entrée. Starting at the northern boundary of the Oyu Tolgoi Property, which is the southern boundary of the Shivee Tolgoi JV Property, an IP/resistivity survey was run on 100 m spaced lines oriented east-west to trace the northern projection of the Hugo North deposit across the Oyu Tolgoi–Shivee Tolgoi Property boundary. In 2005 and 2006, Ivanhoe continued the gradient array IP and ground magnetometer surveying over the EJV Javkhant Licence that is situated south and south-west of the Oyu Tolgoi Property. Results of this work delineated strong north-south-trending chargeability anomalies that possibly represented fault displaced extensions of the Southern Oyu Deposits.

The initial IP/resistivity survey, using a gradient array with an 11,000 m electrode spacing, covered an area that extended 5.6 km north of the JV boundary and was 10 km in width. Subsequent IP surveys covering smaller areas within the larger area were carried out with gradient arrays using 1,200 m, 2,000 m, 3,100 m, 5,000 m and 6,600 m-spaced AB electrodes. Completed by February 2005, the IP surveys resulted in the delineation of a significant chargeability feature extending for approximately 4 km north along strike of the Hugo North deposit (Figure 9.2). Additional IP chargeability targets were also revealed 2.5 km to 3 km west of the Hugo North trend.

A detailed ground magnetometer survey was carried out in October 2008 over the postulated extension of the Hugo North Deposit. A 5 km by 5 km area starting 800 m north of Shivee licence boundary was surveyed on 25 m east-west lines. A similar survey also was carried out over the proposed airstrip north-east of this area in March 2010; the portion of the property surveyed was 4 km by 2 km and continued into the adjacent licence to the north.

Telluric Electromagnetic (TEM) surveying also was conducted in 2005 to explore for water. This survey covered the 12 km length of the Cretaceous palaeochannel that extends north to north-east through the eastern part of the property.

GoCad Targeting Program 2011

Ivanhoe Mines utilized Targeting Workflow module within the Gocad® 3D modelling software developed by Mira Geoscience. The Targeting Workflow utilizes statistical analysis to quantify geological targeting decisions based on the physical characteristics of known mineralisation. The Workflow then provides tools for mapping out and ranking prospective areas based on the targeting criteria. The analysis is based on a “Common Earth Model” (CEM) which represents a volume of rock and is populated with the geological and geophysical parameters. These parameters were derived from a variety of sources including, geochemical assays, geophysical inversions, 3D fault maps, distances to key mineral system components (e.g., faults, host rocks, intrusions), geological maps, and distances to drillholes.

Mira Geoscience consultants Nigel Phillips and Shannon Frey compiled the available magnetic, chargeability, conductivity inversion models and populated them into a CEM for Oyu Tolgoi. The CEM was defined to cover the area from 640350 mE – 656950 mE and 4753100 mN – 4777100 mN from RL +1250 m to RL -1750 m. All cells within the model are 50 m x 50 m x 25 m. Ivanhoe Mines added additional parameters as described below.

Mira Geosciences imported Gocad® 3D ore shells at 0.6 wt. % Cu Eq. for each of the known deposits, inflated those shells by 100 m in all directions to capture lower grade mineralization and alteration haloes and then truncated them by key faults (such as the West Bat fault) and the Devonian-Carboniferous unconformity surface. These 3D volumes provided training regions for analysing correlations between the geological and geophysical parameters. They also provided training volumes for defining the most common values of each CEM parameter within mineralisation to assist in identifying new zones with similar parameters that may contain similar mineralisation.

Based on qualitative evaluation of the geophysical responses of known mineralization within the Oyu Tolgoi structural corridor, and comparison of correlation coefficients between geochemical assays and geophysical inversion model properties within the ore shells, the following seven parameters were selected for use in the CEM:

- 3D geologically-constrained density inversion model from Falcon gravity data
- 3D chargeability inversion model V2 from Zeus IP survey
- 3D chargeability inversion model V3 from Zeus IP survey
- 3D conductivity inversion model

- 3D magnetic amplitude model from aeromagnetic inversion
- Distance to V2 chargeability inversion model >12 ms isosurface
- Distance to V3 chargeability inversion model >12 ms isosurface

Two versions of the 3D chargeability model were included because each works well in different deposits. The V2 model commonly matched the known mineralisation better; however, V3 had better resolution and worked better for the Hugo Dummett deposit. The two models also include different depth weighting functions and so may place the chargeability anomalies at different vertical positions, resulting in offsets of the chargeability anomalies from the actual mineralisation. These offsets are considered to be artifacts of the inversion but can be accommodated in the targeting by including a tolerance on the positions of the chargeability anomalies relative to mineralisation. All known deposits lie within 600–800 m of either the V2 or V3 >12 ms chargeability isosurface; most of that offset is in the vertical direction. Some deposits lie much closer, or within that isosurface depending on whether the V2 or V3 inversion model is used.

The Targeting Workflow created a 3D prediction model from the weighted sum of all the input weights based on the property values for each parameter in every cell of the model. The prediction models have higher values in those cells where more of the parameters fall in the preferred ranges that match known mineralization styles within the CEM. The model has lower values in those cells with more divergent properties.

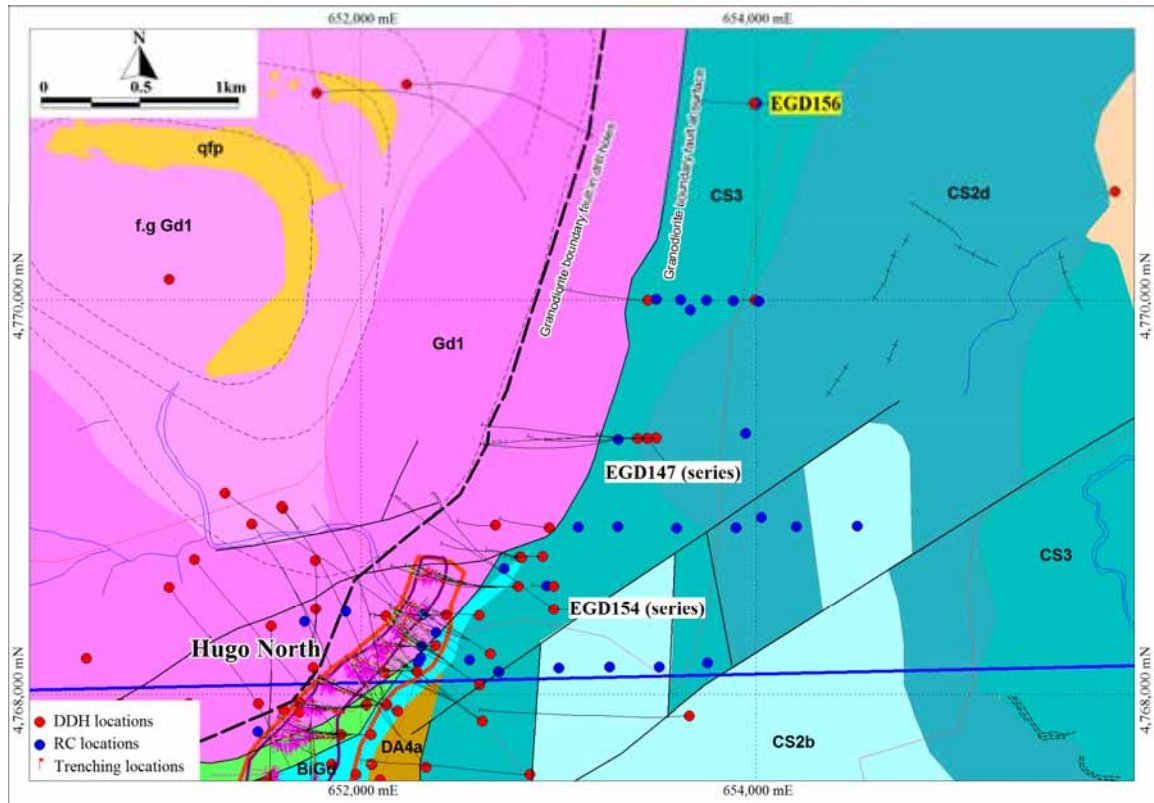
9.1.2.6 Drilling - Shivee Tolgoi License

Three exploration sections have been drilled across the projected extension of the OT Trend in the Shivee Tolgoi Licence (Plan: Figure 9.3).

- The EGD154 section 350 m north of the Hugo Dummett deposit (Figure 9.4).
- The EGD147 section 800 m north of the Hugo Dummett deposit reported in previous quarterly reports (Figure 9.5).
- The EGD156 section 2.4 km north of the Hugo Dummett deposit(Figure 9.6).

Combined, the three sections suggest that there is little potential for mineralization to continue north of the Hugo Dummett deposit through the three sections drilled. However, it remains unclear how mineralization re-appears at shallow depths further north in the Ulaan Khud area.

Figure 9.3 Shivee Licence - Location of EGD154, 147, and 156 Sections



EGD154

This hole was the first in a series of three planned holes on the section, with the other holes planned to cut off a navi-bend between 880 m and 1,000 m.

From the base of the Cretaceous clay at 48 m to a depth of 1,346 m the hole drilled through CS2, CS1 Carboniferous sequence then DA4 Heruga sequence rocks before passing through the contact fault at 1,268 m and into fresh DA3b and DA2a. This sequence is the same as for the EGD053 section to the south.

A major fault (Entrée Boundary Fault) was then encountered at 1,346 m and DA4 lithologies were again intersected. The fault is the same as the east-west oriented structure evident on surface and confirms its significance as the cause of a major down dropping of the prospective geology to the north.

Below the Entrée Boundary Fault, the drillhole intersected DA4 lithologies to 1,531 m, then passed through the contact fault into DA3b to 1,568 m and finally through a thick unmineralised sequence of ignimbrite and block ash tuff to 1,924 m.

A major brittle fault was intersected followed by a series of sheared alternations of what appeared to be DA1b augite basalt and Da2b tuff to 1,998 m, followed by sheared biotite granodiorite to 2,067 m with thin slivers of what appeared to be magnetite-altered augite basalt. Hornfels and dykes were then intercepted to 2,088 m then fresh Carboniferous

granodiorite with minor hornfels was intercepted till the end of the hole at 2,122.4 m. The granodiorite had an intrusive contact with the hornfels.

The interval from 1,924 m to the end of the hole at 2,122.4 m is probably a sliver up against the granodiorite boundary fault. Similar slivers have been intercepted in most holes at the northern end of Hugo North, and it is common to intercept Biotite granodiorite and fragments of Ignimbrite and even DA1b in the slivers, even at shallow depths. Usually the Carboniferous granodiorite is in faulted contact with the slivers, but EGD154 seems to be an exception with an intrusive contact.

It is unlikely that the West Bat Fault has been intercepted in EGD154; usually the West Bat Fault is characterized by a single healed fault with fresh Carboniferous or Devonian lithologies on the western side, very different to what was intercepted.

EGD154A

EGD154A cut off the mother hole at 999 m near the end of the navi-bend and was successfully made to drop. From 999 m to 1,317.3 m the hole drilled through DA4, DA3 and then fresh DA2b ignimbrite, the same as the EGD053 section to the south.

The Entrée Boundary Fault was then encountered at 1,317.3 m and DA4 lithologies were again intersected.

Below the fault the hole intersected DA4 lithologies to 1,637.8 m and then passed through the contact fault into DA3b to 1,681 m then through a thick unmineralised sequence of ignimbrite and block ash tuff to 2,371 m. This sequence was generally weakly altered but there were zones of quartz-sericite alteration up to 50 m wide with some pyrite. The hole then intersected the West Bat Fault between 2,371 m and 2,406 m, which consisted of a series of ductile shear zones within DA4a basaltic tuffs and slices of sheared ignimbrite. The hole ended at 2,415.7 m in fresh DA4a basalt.

The 731 m of ignimbrite intersected was anomalously thick; approximately twice as thick as in sections both to the north and south. The thickness of this unit was most likely due to the hole drilling at low angle to the dip direction.

EGD156

EGD156 combined with EGD143 and EGD005 drilled in previous years tested the section 2.4 km to the north of the Hugo Dummet deposit. Only carboniferous rocks were intersected despite drilling to 1,450 m. Hornfelsing was common due to the proximity to the Khanbogd granite.

Figure 9.4 Shivee Tolgoi Section EGD154

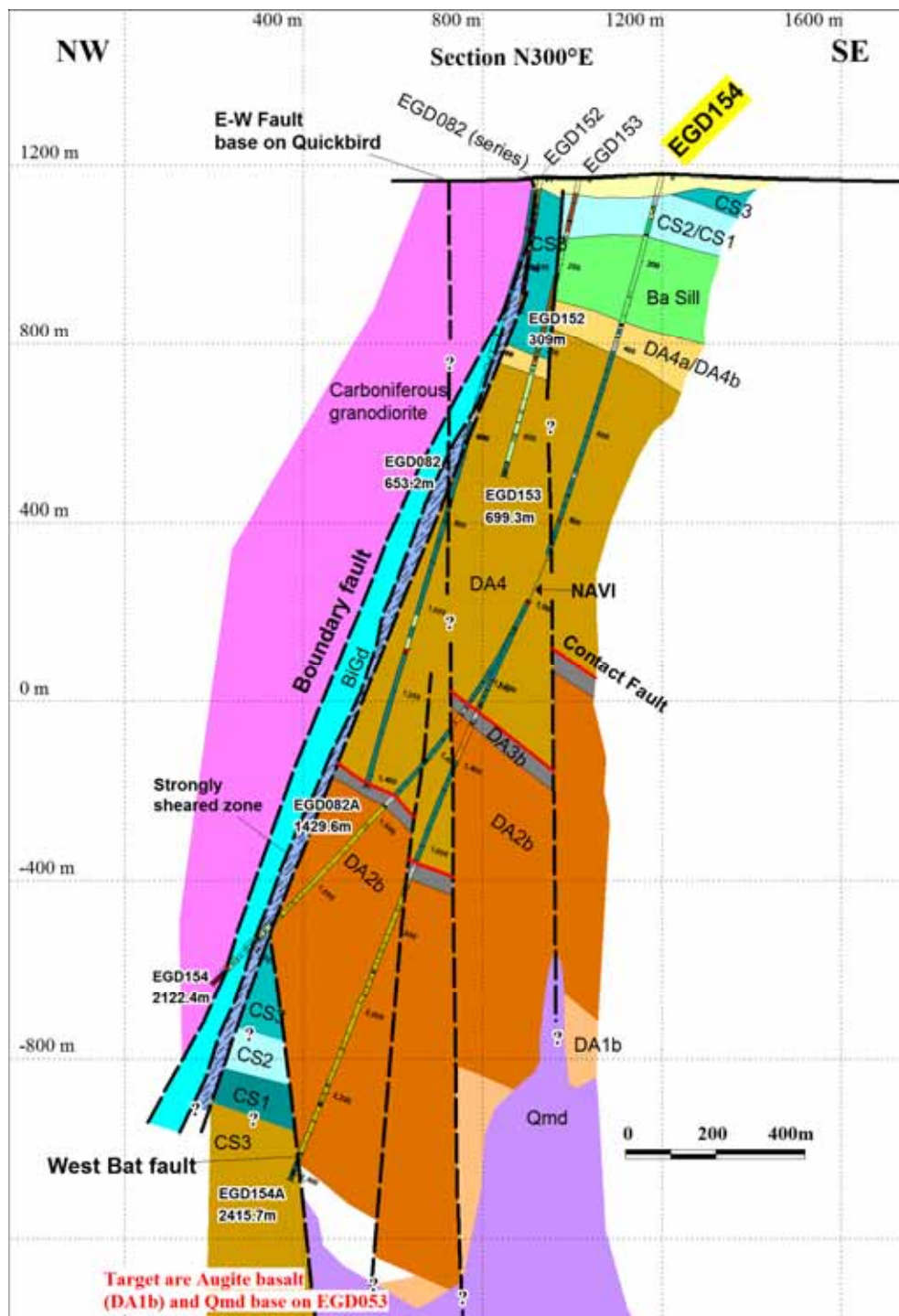


Figure 9.5 Shivee Tolgoi Section EGD147

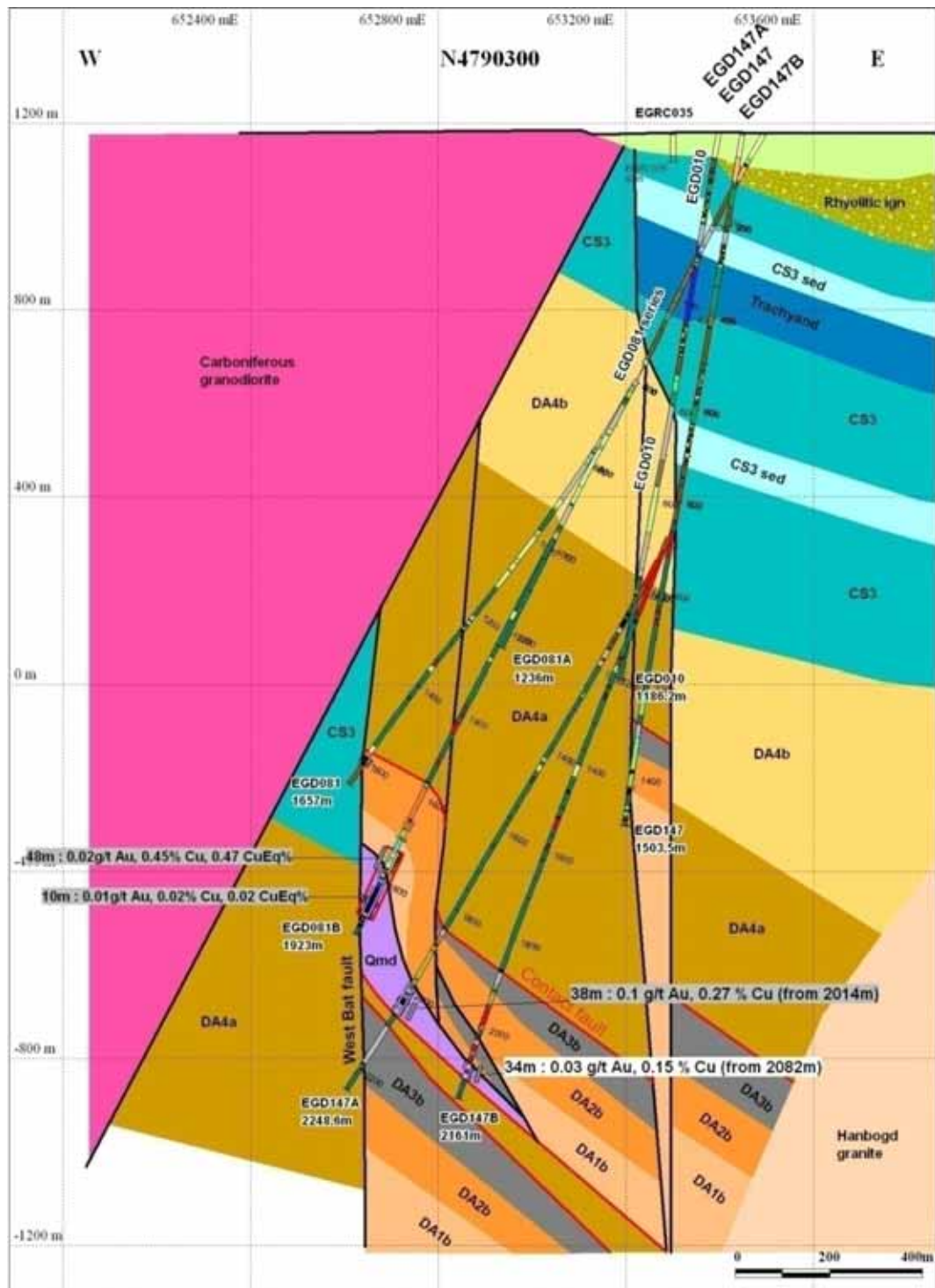
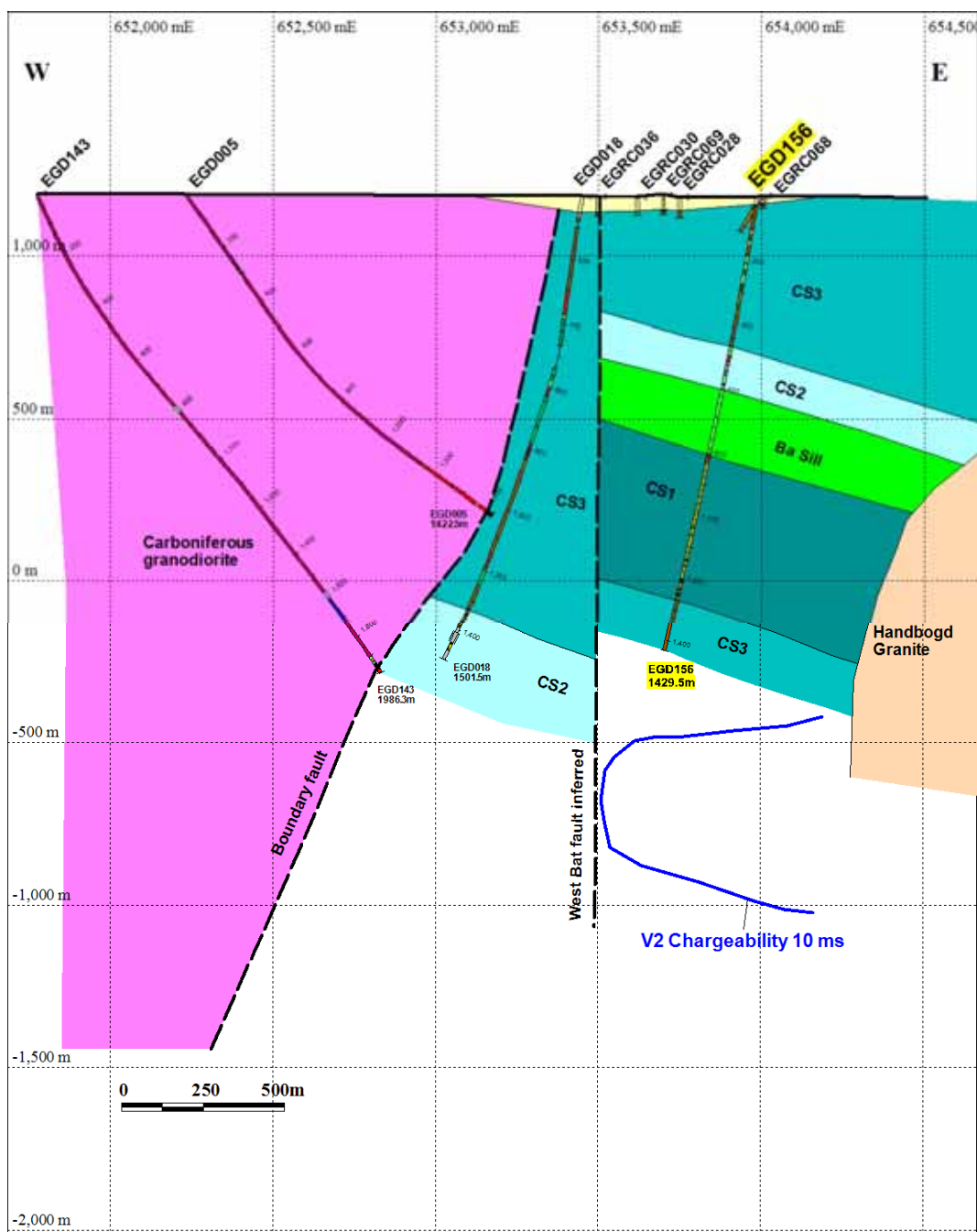


Figure 9.6 Shivee Tolgoi Section EGD156



9.1.2.7 Topography

Various topographic surveys have been completed on the Project area, the most recent of which was completed by Geomaster covering a 10 km by 10 km area using an electronic total station instrument. This instrument has an accuracy of 2". Contour intervals are 1 m, as presented on the digital topography map.

9.1.2.8 Geological Mapping and Related Studies

Outcropping prospects on the Oyu Tolgoi Property (Southwest, South, Central Deposits) were mapped at 1:1,000 scale in 2001. The central part of the exploration block was mapped at 1:5,000 scale in 2001, and the entire block was mapped at 1:10,000 scale in 2002. The understanding of the surface geology in areas where thin gravel and clay cover obscured outcrops was greatly improved as a result of the extensive trenching and mapping program that was carried out in 2003 and 2004 (see Section 9.1.2.9). Figure 7.3 in Section 7 shows a recent compilation of the geology over the Project area.

In 2004, the Copper Flats area within the Shivee Tolgoi JV Property was geologically mapped (Panteleyev, 2004a, 2004b), using the Copper Flats grid adjacent to the northern and north-eastern boundary of the Oyu Tolgoi Property. The geological mapping covered a region of approximately 15 km² (Blower, 2006b).

Mapping results were plotted on existing 1:10,000-scale topographic base maps. In addition, diamond drillhole data were tabulated, as well as the assay values from grab and chip samples collected during the mapping program. Alteration minerals were determined by short wave infrared spectrometry ("SWIR" or "PIMA" analysis) on typical specimens from a number of alteration zones (Blower, 2006b).

Figure 7.3 shows a compilation of the geological mapping over the Oyu Tolgoi and Shivee Joint Venture Properties.

9.1.2.9 Trenching

During 1998, two trenches were completed by BHP over surface exposures of the Southwest Oyu Deposit. Both trenches were approximately 60 m in length and provided early-stage geological and assay information regarding the deposit.

During 2003 and 2004, an extensive 8,000 m trenching program was carried out over the South and Southwest Oyu Deposits, plus an additional 20 km of trenches were completed in various other locations throughout the licence. Trenches at the South Oyu Deposit, range in length from 280 to 1,177 m and average around 600 m. Trenches were sampled over 2 m intervals and assayed for Cu, Au, Mo, As, and Ag.

No trench data are used for resource estimation.

9.1.2.10 Drilling

(1) Core Drilling

Diamond drilling has been the primary tool of exploration at the Project. As of October 2009, more than 1,390 drillholes were completed on Oyu Tolgoi, Shivee Tolgoi, and Javkhlant Properties, with many from Hugo North Deposit having multiple “daughter holes” navi-drilled from a single parent (see Section 10).

(2) RC Drilling

The proportion of RC drilling completed on the Property is minimal (5%) in comparison to the amount of core drilling. Most of the RC drilling was completed by Ivanhoe in early campaigns during 2000 and 2001. In the first year 109 holes (8,828 m) were completed at Central Oyu, to explore the chalcocite blanket discovered previously by BHP. In 2001, Ivanhoe completed 16 RC holes totaling 2,091 m, mostly in the South Oyu area, to test its potential to host oxide copper resources. Near the end of this campaign Ivanhoe drilled three combined RC/core drillholes to test the deep hypogene Cu-Au potential at the South Oyu deposit, each of which intersected significant widths of Cu and Au mineralization. Subsequent to the completion of these three core holes, Ivanhoe switched exploration drilling to core methods. RC drilling is still occasionally used as a pre-collar for deep core holes, and for condemnation and water drilling purposes.

The RC drilling campaigns are discussed in more detail in Section 10.

9.1.2.11 Mineralogical and Petrographic Studies

Ivanhoe maintains an in-house petrology laboratory in the Oyu Tolgoi exploration camp. An ASD spectrometer, binocular transmitted and reflected light microscope, plus an automated thin section cutting machine and polished section lapidary equipment for making polished mineral specimen blanks and polished thin sections, are currently in use. A program of preparing mineralization samples and making metallurgical index estimates from all of the Oyu Tolgoi deposits was initiated by Giovanni de Prisco in 2002. The program has continued through to the present with two Mongolian National petrologists employed on a full-time basis to examine core samples to produce a tabulated metallurgical index on the deposits.

Other exploration groups within the Ivanhoe group of companies also send samples to the laboratory for petrographic descriptions.

9.1.2.12 Exploration Shaft Sinking

IMMI (now OT LLC) sank a 6.7 m diameter exploration shaft (Shaft 1) into the Hugo North Deposit. The Shaft 1 headframe, hoisting plant and associated infrastructure were completed in January 2006. The shaft was completed to a depth of 1,385 m in January 2008 by the Redpath Group of North Bay, Canada. Shaft 1 is designed to provide access to the Hugo North and Hugo South deposits. Development from the shaft will enable additional delineation drilling and rock characterization for proposed mining operations. Lateral development work is currently in progress.

9.2 Heruga

Geophysical exploration of the Heruga property commenced in 2005 with a reconnaissance gradient array IP survey followed by dipole-dipole IP surveying in 2007. Drilling at Heruga commenced in 2007 using the gradient array IP anomaly to help target diamond drillholes.

9.2.1 Initial Exploration

In the period 2002 to 2004 Entrée Gold collected approximately 1,400 soil samples over six east-west lines spaced 200 m apart. No significant anomalies were found. Rock chip sampling also found nothing of significance.

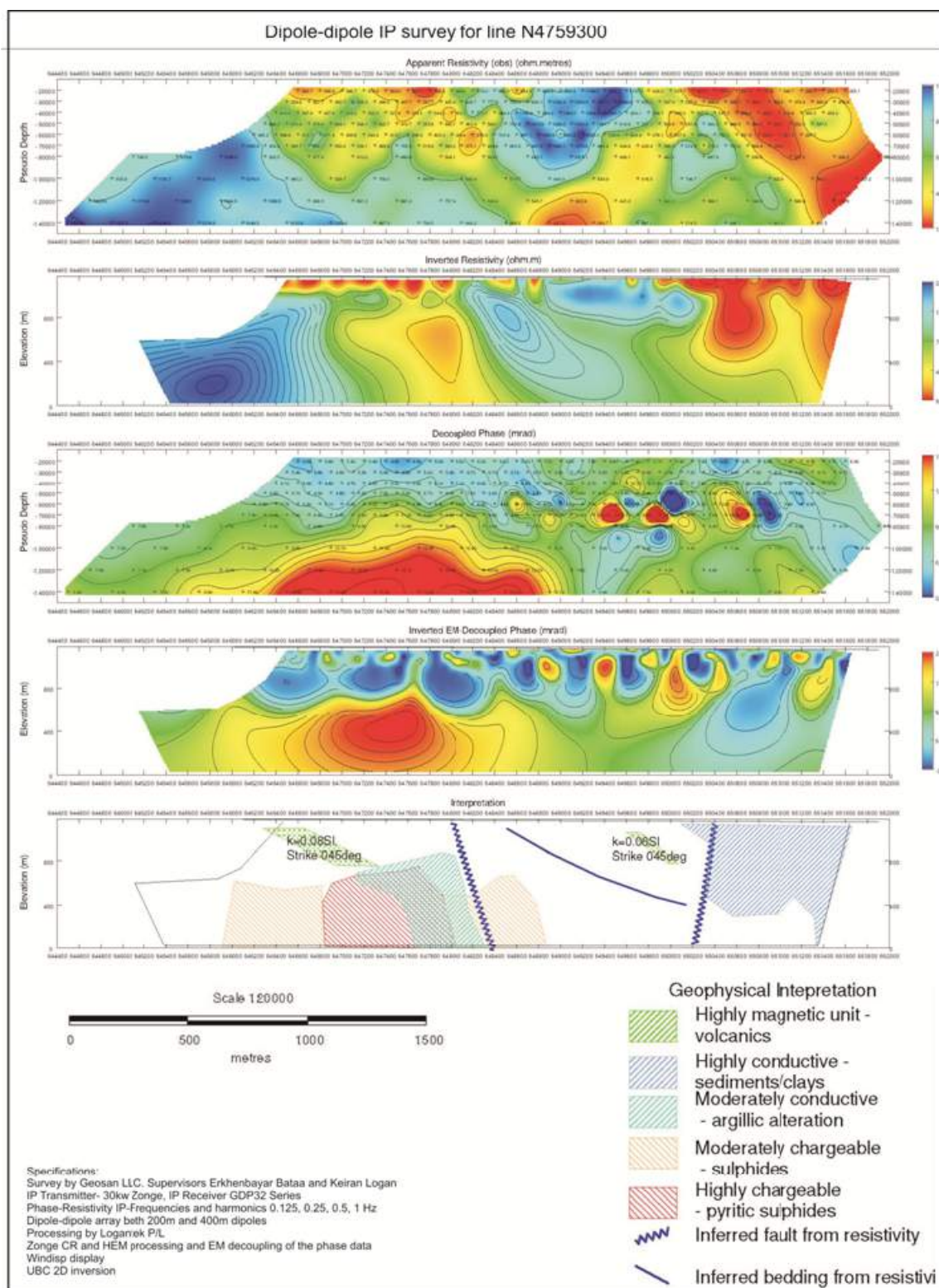
In 2003, two lines of dipole-dipole IP totalling 10 km were completed. Dipole spacing was 50, 100, and 200 m with both lines crossing over the Heruga area. The results were reported as being “generally low and unresponsive” (Reid et al., (2004), Cann (2004).

In October 2004, IMMI entered into an earn-in agreement with Entree Gold to explore the Javkhlant licence, which included the Heruga area, and the eastern half of the Shivee Tolgoi licences, which included the extension of Hugo Dummett deposit.

Because the primary focus of the agreement was the Hugo Dummett deposit extension, Javkhlant was a lower priority, so it was not until late 2005 that the regional IP and magnetometer surveys were completed over the Heruga area. An anomaly was recognised for the first time as one of three north-south trending chargeability anomalies of similar magnitude to the Oyu Tolgoi anomalies. Heruga proved to be the most interesting because outcropping rocks were Devonian, or older, basalts.

In February 2006, IMMI's geophysical team re-surveyed the anomalous chargeability features using a gradient array. A later version of the results using the Zeus IP transmitter is shown in Figure 9.7. A third survey was completed by Geoscan LLC in 2007. Results were sufficiently positive for drill testing.

Figure 9.7 Geoscan Dipole-dipole IP Survey for Line N4759300



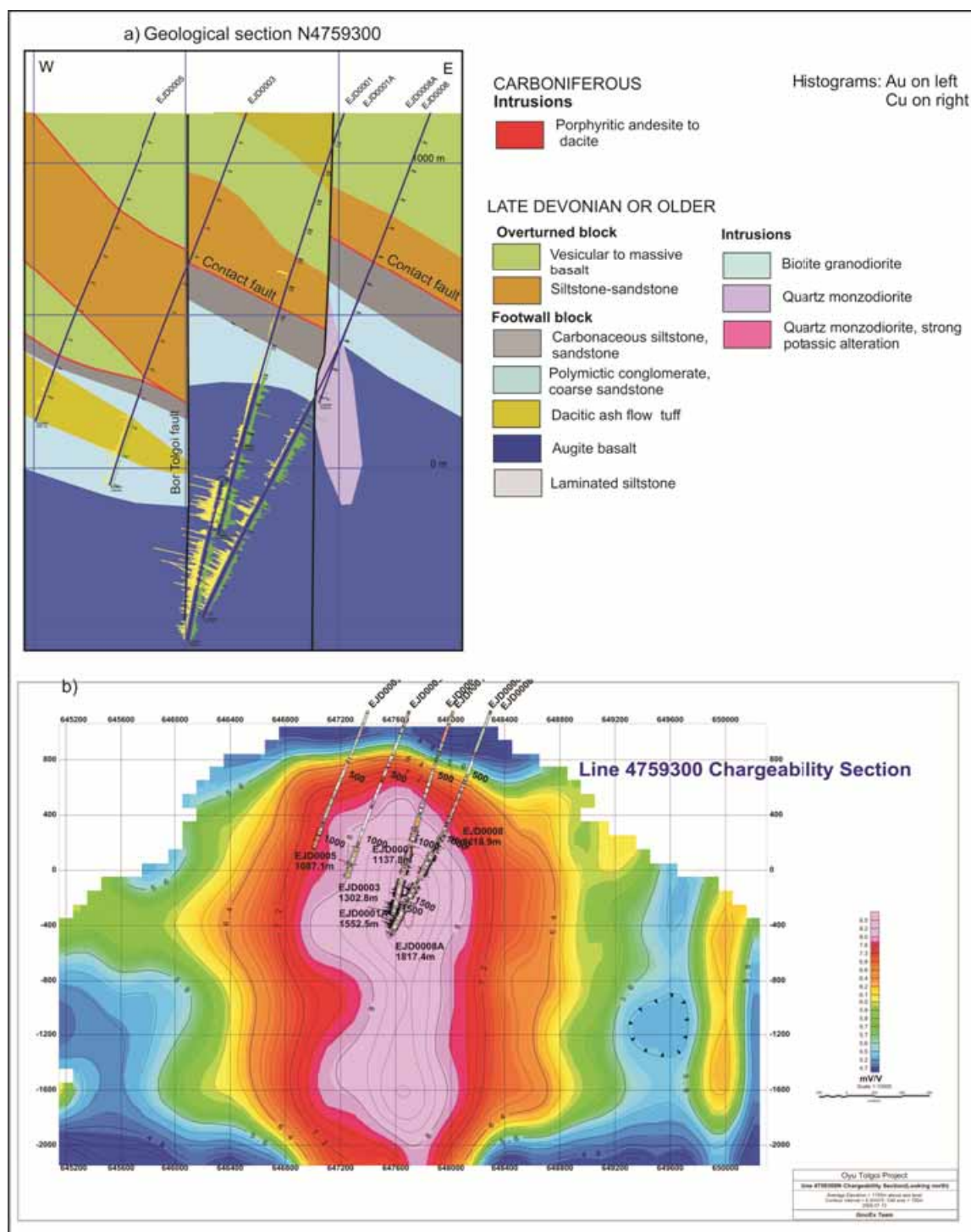
9.2.2 Heruga Drilling

Based on the results of the IP, drillhole EJD0001 was collared on line N4759300 (Figure 9.8), over the strongest IP anomaly. The hole commenced on 1 March 2007 and was completed on the 25 March 2007 at a depth of 1,137 m. It intersected Carboniferous ignimbrite, drilled through Devonian (or older) basalts and sediments before encountering the typical stratigraphy that overlies mineralisation at Hugo Dummett. The hole intersected approximately 150 m of 0.3-0.7% Cu and 0.1-0.3g/t Au, and very significant Mo (60–160 ppm), in augite basalt below the conglomerate. The hole was therefore not high grade, but was extremely significant. The hole was stopped because the dipole-dipole pseudo section suggested that the anomaly had been drilled through and this seemed to correspond with a decrease in grade, although the gradient array IP results strongly suggested that only the top of the mineralisation had been drilled.

A second drillhole (EJD004) was drilled, 1.4 km south of line N4759300. This intersected 114 m of 0.2 g/t Au, 0.39% Cu and 135 ppm Mo before the hole was lost at 922 m due to poor drilling conditions and the relatively small drill rig. Geology was very similar to the northern section and demonstrated that the IP anomaly represented one big system. With the establishment of similar geology and mineralisation at either end of the anomaly, it was then just a simple matter of filling in between the lines.

Since the initial discovery holes, a total of 79,000 m of drilling has been carried out at Heruga and its extension to the north using the gradient array IP pseudo sections as a targeting guide with good success. EJD0013 intercepted an overall intercept of 454 m of 1.43g/t Au and 0.5% Cu and 178 ppm Mo within a quartz monzodiorite. Gold was much higher, compared to earlier drillholes, which encountered about 0.3 g/t or 0.4g/t gold and 0.5% Cu but always with significant Mo. Before EJD0013 the prospect was looking like a very large low grade Cu-Mo deposit with some gold. It became apparent with further drilling that early holes that were drilled in Heruga terminated in the Mo shell and needed to be deepened. Hole EJD0001A was extended from the previous depth at 1,137 m to 1,724 m, and intersected additional mineralisation that included 162 m of 1.41g/t Au and 0.42%Cu. This confirmed what was predicted from the previous gradient array IP survey.

Figure 9.8 Drill Section and Gradient Array IP Survey for Line N4759300



a) Geological cross section N4758100

W N E

Bor Tolgoi fault

Contact fault

1000 m

0 m

CARBONIFEROUS Intrusions

Porphyrific andesite to dacite

LATE DEVONIAN OR OLDER

Overtaken block

Vesicular to massive basalt

Siltstone-sandstone

Footwall block

Carbonaceous siltstone, sandstone

Polymictic conglomerate, coarse sandstone

Dacitic ash flow tuff

Augite basalt

Laminated siltstone

Intrusions

Biotite granodiorite

Quartz monzodiorite

Quartz monzodiorite, strong potassic alteration

Histograms: Au on left Cu on right

b)

Line 4758100 Chargeability Section

644400 644800 645200 645600 646000 646400 646800 647200 647600 648000 648400 648800 649200 649600 650000 650400

800
400
0
-400
-800
-1200
-1600
-2000

EJ00013 1133.8m
EJ00013 1123.3m
EJ00013 1241.8m
EJ00013 1353.1m
EJ00013 1404m
EJ00013 1520.3m
EJ00013 1587.3m

647600 648000 648400 648800 649200 649600 650000 650400

Chargeability scale (mV/V)

0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
1.1
1.2
1.3
1.4
1.5
1.6
1.7
1.8
1.9
2.0
2.1
2.2
2.3
2.4
2.5
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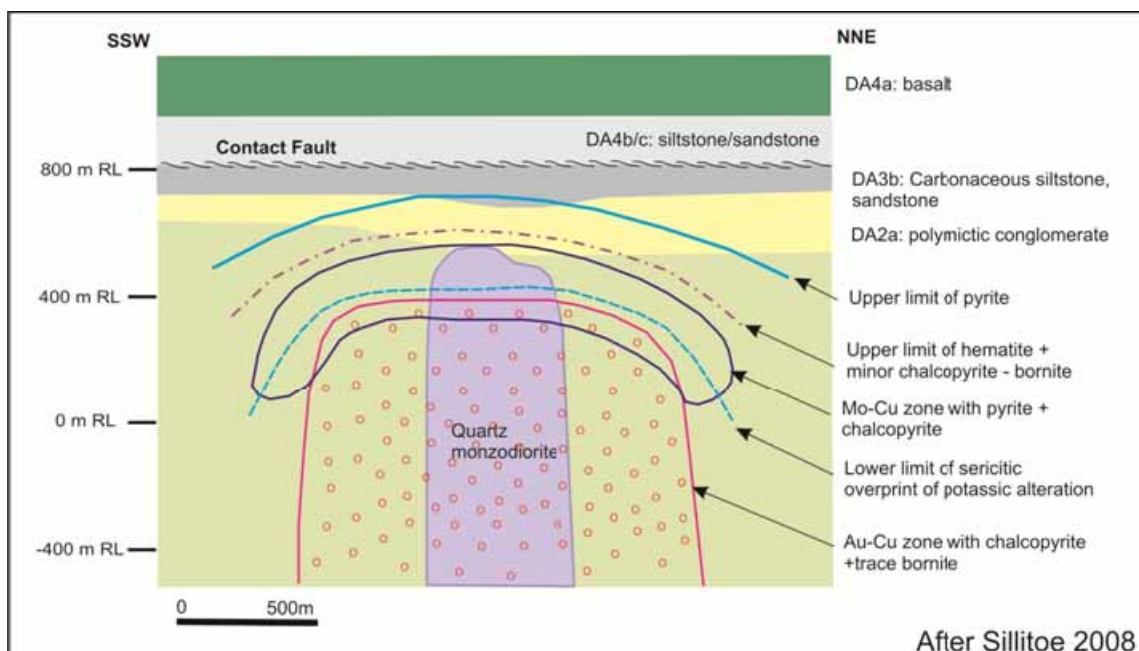
Scale = 1:10000

Geological Project
Line 4758100 Chargeability Section (looking north)
Project location: 1:25000 Scale map sheet
Data source: 1:25000 Scale map sheet
Date: 2008

9.2.3 Heruga Mineralisation, Alteration, and Schematic Model

The Heruga deposit is characterized by chalcopyrite-gold mineralization hosted by mainly augite basalt. At deep levels bornite is also found, and at high levels pyrite and sericite alteration forms an outer shell above strong biotite alteration. A zone of Mo (>150 ppm) 200–300 m thick partly corresponds to the sericite zone. The Heruga porphyry system, like others in the Oyu Tolgoi district is related to quartz monzodiorite intrusions, but at Heruga the possible source intrusions, and the centre of the system are not well understood. Heruga may be comparable to Southwest Oyu, where small (metres to 10's of metres wide) highly altered quartz monzodiorite dykes occur in the core of the deposit. Eighty percent of the mineralization is hosted by augite basalt wall rock, and the highest grades, and highest Au(g/t) to Cu(%) ratios (up to 3) occur with the most intense biotite alteration. Whereas Southwest Oyu is an eroded porphyry system, Heruga appears to be intact (Figure 9.10). Alteration and mineralization rapidly disappear upward in sedimentary rocks overlying the augite basalt, and do not reach the Contact Fault.

Figure 9.10 Schematic Section of Heruga Genetic Model



9.2.4 Mineralogical and Petrographic Studies

Mineralogical and petrographic studies of the Heruga core samples were completed, using methodology as for previous Ivanhoe exploration at Oyu Tolgoi.

9.3 Heruga North

9.3.1 Initial Exploration

Initial exploration on Heruga North targeted deep seated IP anomalies in the Gap between Heruga and south-west Oyu.

9.3.2 Heruga North Drilling

The holes drilled in Heruga north are in six north-west-south-east-oriented sections 300–500 m apart. The mineralization at Heruga North is similar to Heruga and the South West deposit, with the exception of OTD1495A which was drilled on the eastern side of the quartz monzodiorite and is rich in bornite and appears to be very similar to that in the Hugo Dummett Deposit.

The holes drilled in the northern end of Heruga North had a low success rate of hitting mineralisation: four holes (OTD1487C, OTD1487D, OTD1492 and OTD1496) targeted induced polarization (IP) anomalies. All were terminated after intersecting a major north-north-east-oriented fault, thought to be an extension of the West Bat Fault that terminates the western side of the Hugo Dummett deposit. Minor sulfides near the fault are thought to explain the IP. Four other holes (OTD1493, OTD1493A, OTD1495 and OTD1498) reached the top of the mineralized zone but were lost due to drilling difficulties. Two other holes in the northern-most section drilled (OTD1497 and OTD1499) also hit a major fault, with OTD1499 reaching the top of the mineralized section before being faulted off. In contrast, holes at the southern end of Heruga North have been more successful, due to the shallower depth of mineralisation. Cross sections are shown in Figure 9.12 to Figure 9.16.

Drilling in 2011 focused on infilling the southern portion of Heruga North with the intention of extending the defined resources at Heruga. Mineralization in the target area is between 1,100 m and 1,700 m below surface and the additional resources defined would form a lateral extension to Heruga mineralized body. Table 9.1 shows the drill results at Heruga North. 2011 drill sections include the 1510 series, 1569 series and 1570 series.

Table 9.1 Heruga North Exploration Drill Results

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Cu (%)	Mo (ppm)	CuEq (%)
OTD1487	1,978	1,994	16	0.1	1.65	38	1.73
	2,028	2,053.7	25.7	0.85	0.89	62	1.47
OTD1487A	1,978	1,994	16	0.09	1.55	38	1.63
	2,028	2,126	98	0.96	0.88	115	1.56
	2,258	2,336.3 (EOH)	78.3	2.13	0.82	126	2.24
	1,978	2,336.3	358.3	0.85	0.54	66	1.12
OTD1495A	2,034	2,314	280	0.06	0.84	13	0.88
	2,330	2,377.2 (EOH)	47.2	0.1	1.32	8	1.39
	2,034	2,377.2 (EOH)	343.2	0.06	0.87	12	0.91
OTD1498A	1,978	2,100	122	0.05	0.42	54	0.48
including	2,240	2,318	78	0.07	0.61	12	0.66
	2,346	2,418	72	0.07	0.44	41	0.51
	2,464	2,478	14	0.31	0.21	28	0.42
	2,546	2,562	16	0.19	0.27	72	0.43
	2,582	2,632(EOH)	50	0.17	0.32	75	0.47
OTD1500	1,800	1,910	110	0.17	0.44	189	0.65
OTD1500A	1,462	1,506	44	0.09	0.55	55	0.64
OTD1501	1,662	1,682	20	0.12	1.06	12	1.14
including	1,724	1,730	6	0.07	1.23	30	1.29
	1,754	1,776	22	0.03	0.59	39	0.63
OTD1500B	2,066	2,094	28	2.3	1.35	200	2.92
	2,182	2,276	94	0.91	0.6	60	1.21
OTD1510	1,460	2,398	938	0.42	0.46	72	0.76
including	1,460	1,492	32	0.08	0.66	22	0.72
	1,504	1,742	238	0.11	0.41	37	0.5
	1,758	1,780	22	0.39	0.59	140	0.91
	1,792	1,976	184	0.55	0.59	74	0.98
	1,986	2,006	20	0.6	0.53	83	0.96
	2,012	2,074	62	1.08	1.11	235	1.93
	2,090	2,162	72	0.18	0.36	139	0.55
	2,166	2,184	18	0.13	0.57	225	0.77
	2,214	2,232	18	0.19	0.55	166	0.75
	2,246	2,264	18	0.21	0.46	132	0.67
	2,286	2,398	112	1.36	0.34	14	1.21
	2,272	2,312	40	0.87	0.49	92	1.09
	2,326	2,354	28	1.32	0.39	8	1.23
	2,364	2,398	34	2.44	0.47	6	2.03
	2,376	2,396	20	3.78	0.64	6	3.06
	2,388	2,394	6	8.44	0.66	5	6.05

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Cu (%)	Mo (ppm)	CuEq (%)
OTD1502A	2,156	2,172	16	0.84	0.67	152	1.29
	2,206	2,402	196	0.7	0.58	117	1.09
OTD1502B	2,090	2,104	14	0.99	0.69	30	1.34
	2,160	2,202	42	0.15	0.75	67	0.88
OTD1502C	1,966	2,020	54	0.08	0.66	40	0.73
	2,078	2,090.4	12.4	0.11	0.48	133	0.62
OTD1510B	1,436	1,478	42	0.09	1.55	46	1.64
	1,714	1,776	62	0.51	0.72	69	1.09
	2,204	2,216	12	1.04	0.88	200	1.65
	2,282	2,296	14	0.91	0.44	14	1.03
	2,332	2,370	38	1.79	0.4	22	1.55
	2,388	2,428	40	1.59	0.39	12	1.41
OTD1510C	1,486	1,496	10	0.08	1.57	18	1.63
	1,510	1,550	40	0.09	0.66	34	0.74
	1,608	1,782	174	0.23	0.53	25	0.68
OTD1510D	1,498	1,876	378	0.26	0.55	86	0.76
OTD1567	30	454	424	0.13	0.41	38	0.51
and	550	1,184	634	0.76	0.33	21	0.82
including	892	940	48	1.14	0.49	16	1.22
including	966	1,030	64	1.02	0.35	41	1.02
including	1,084	1,184	100	1.73	0.63	15	1.74
OTD1569	1,496	1,848	352	0.38	0.60	116	0.88
including	1,528	1,746	218	0.34	0.75	151	1.05
OTD1569A	1,665	1,862	197	0.38	0.56	101	0.86
including	1,665	1,726	61	0.40	1.03	234	1.40
including	1,778	1,862	84	0.51	0.46	54	0.81
OTD1569B	1,508	1,698	190	0.18	0.49	168	0.70
and	1,758	1,806	48	1.25	0.31	5	1.11
and	1,944	2,108	164	0.26	0.42	145	0.67
OTD1569C	1,468	1,644	176	0.22	0.61	167	0.84
and	1,728	2,204	476	0.70	0.33	50	0.80
including	1,924	1,952	28	4.23	1.16	56	3.88
OTD1569D	1,206	2,346	1,140	pending	–	–	–
OTD1570	1,598	1,958	360	0.74	0.26	25	0.75
including	1,696	1,846	150	1.30	0.30	9	1.13
and	2,010	2,116	106	0.33	0.23	21	0.45
OTD1570A	1,078	1,624	546	pending	–	–	–
including	1,084	1,184	100	1.73	0.63	15	1.74

Figure 9.11 Drill Collar Locations Updated with Heruga North

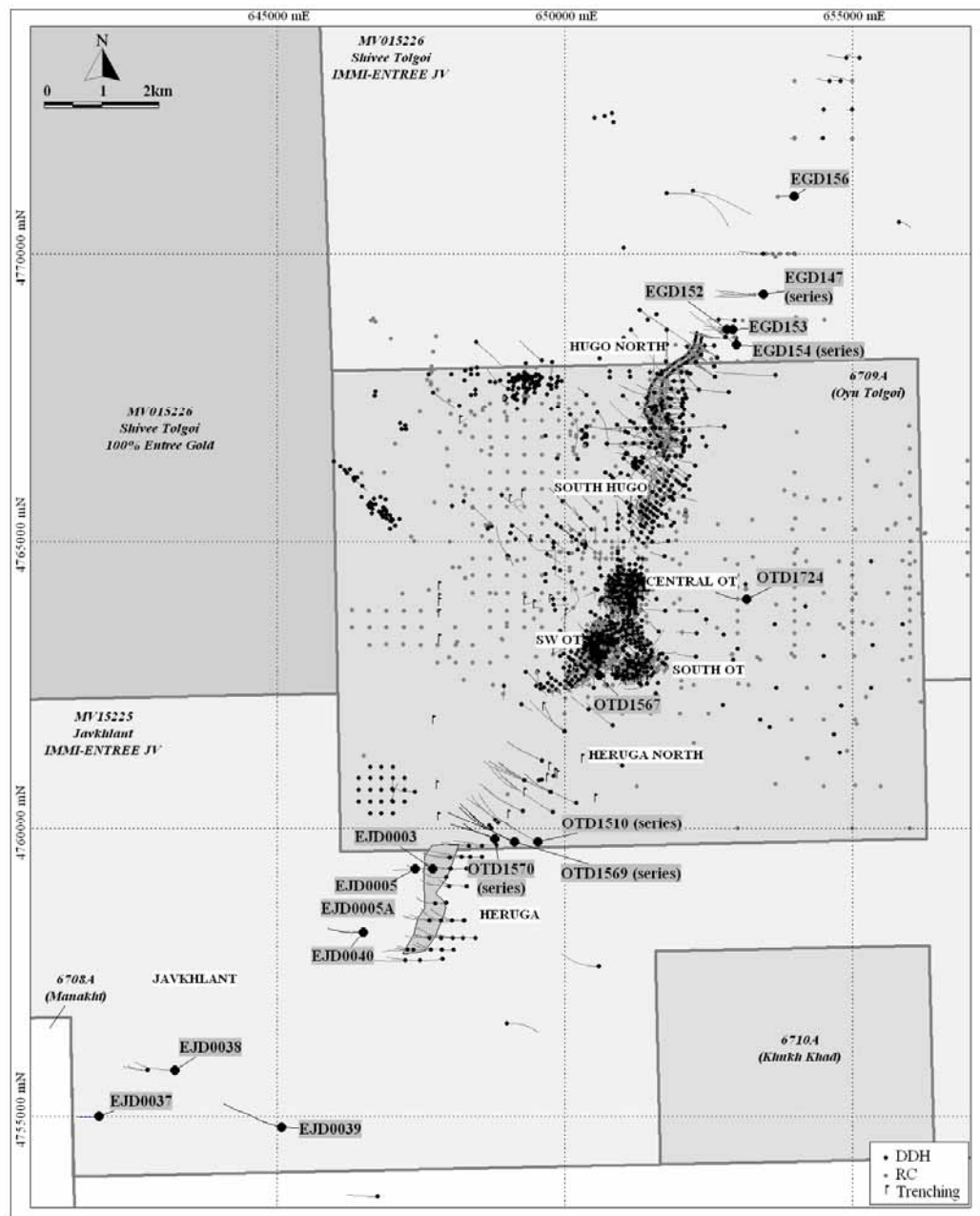


Figure 9.12 Heruga North Geology, Section OTD1487

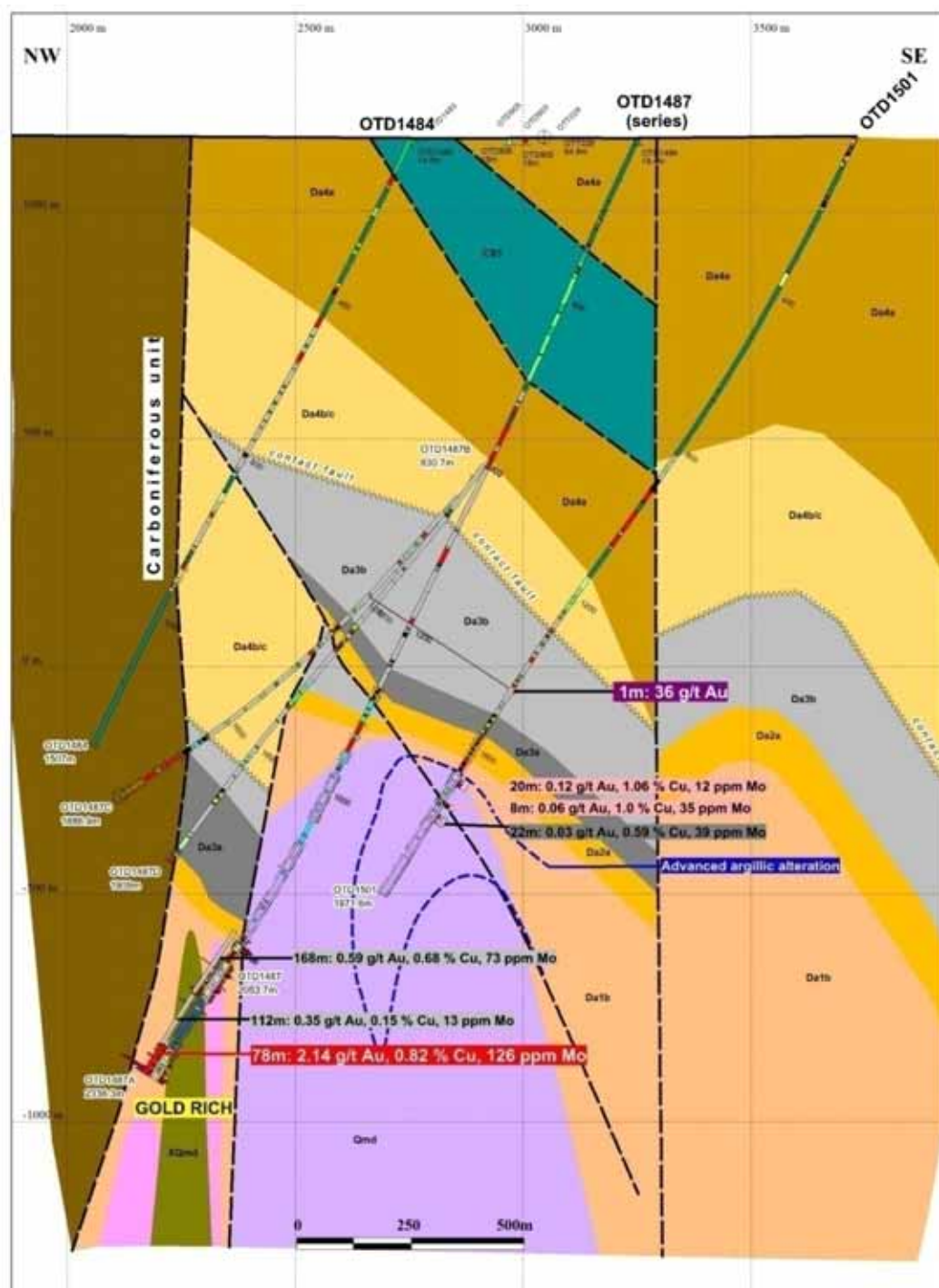


Figure 9.13 Heruga North Geology, Section OTD1502

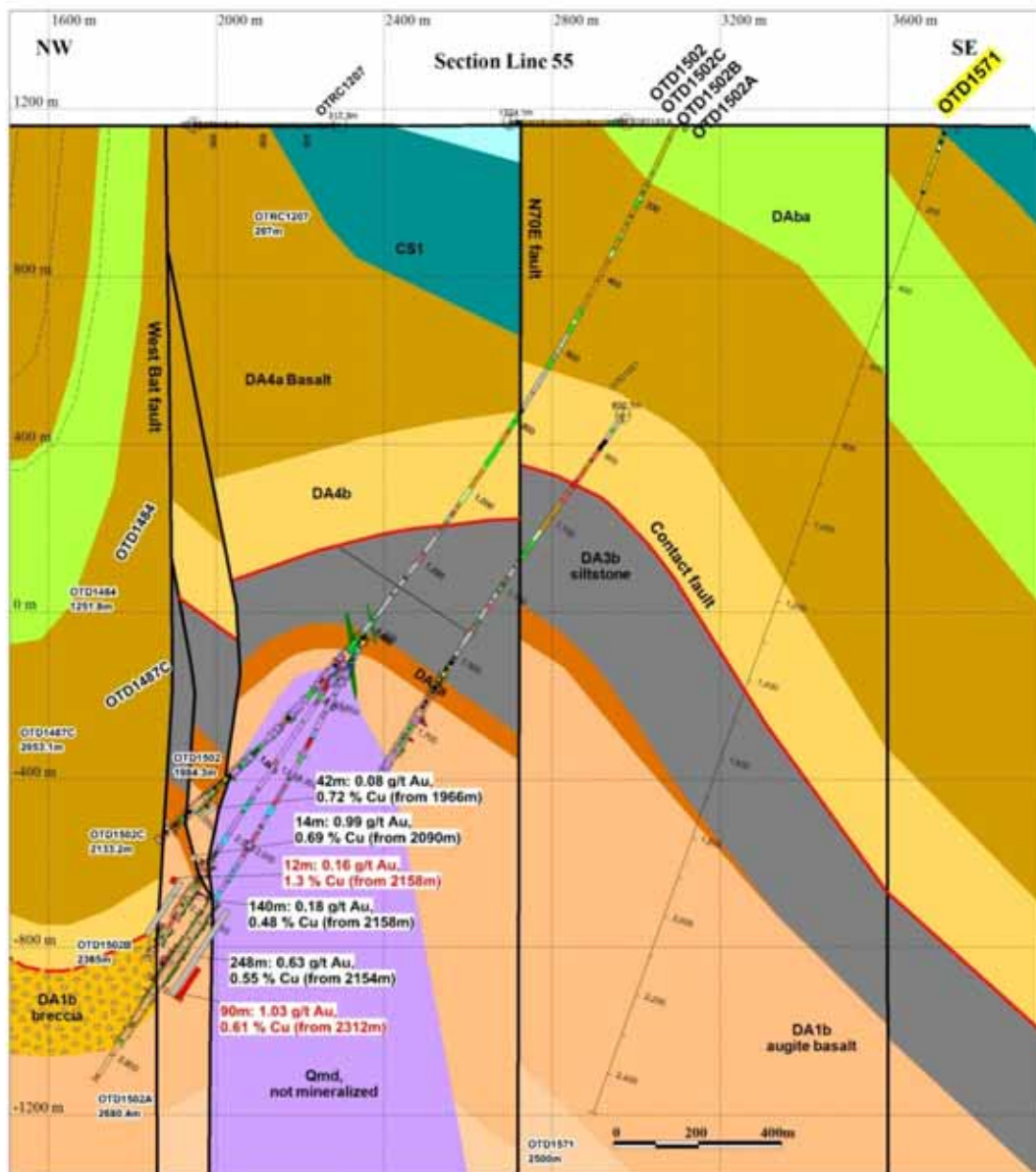


Figure 9.14 Heruga North Geology, Section OTD1500

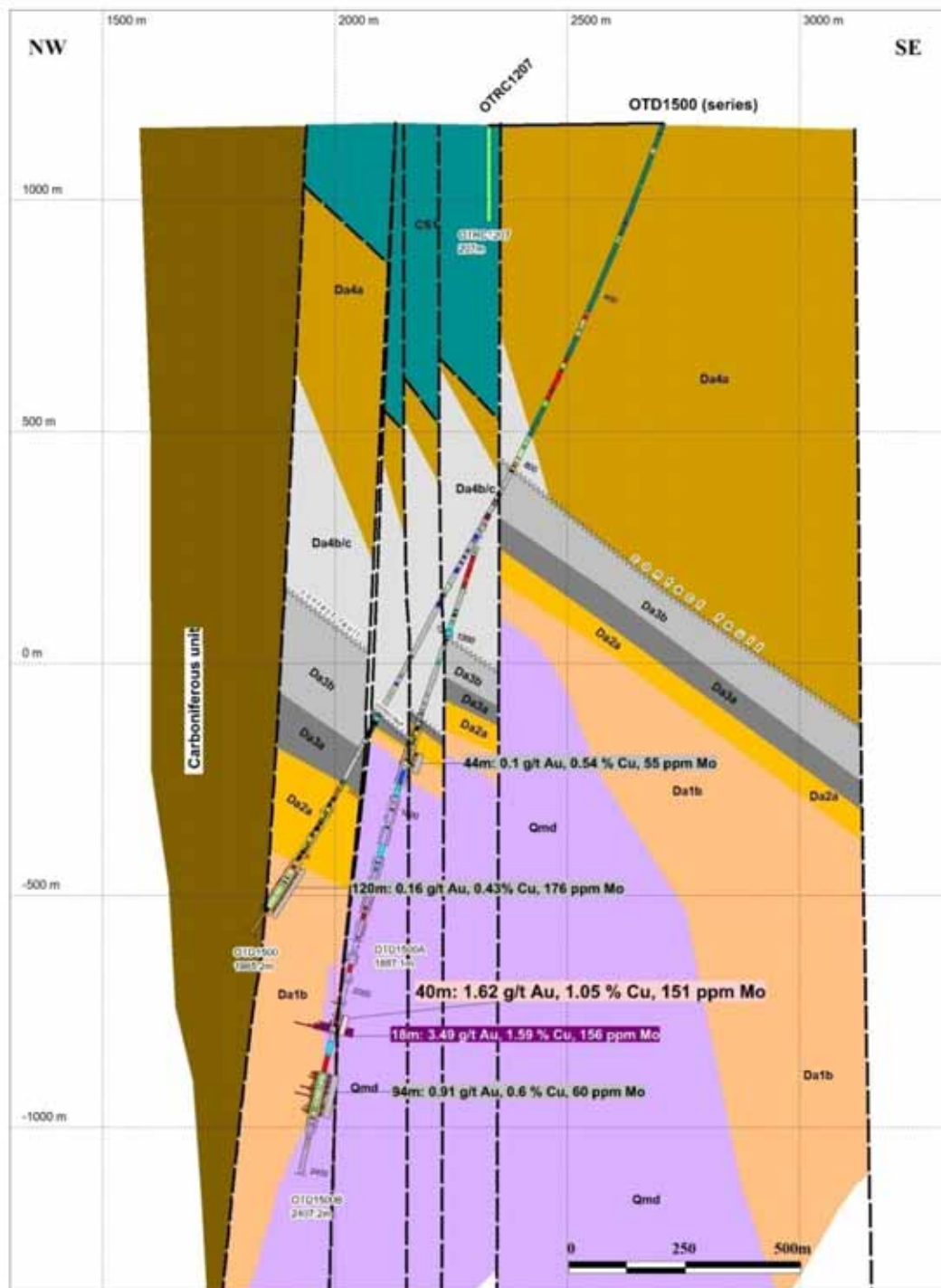


Figure 9.15 Heruga North Geology, Section OTD1570

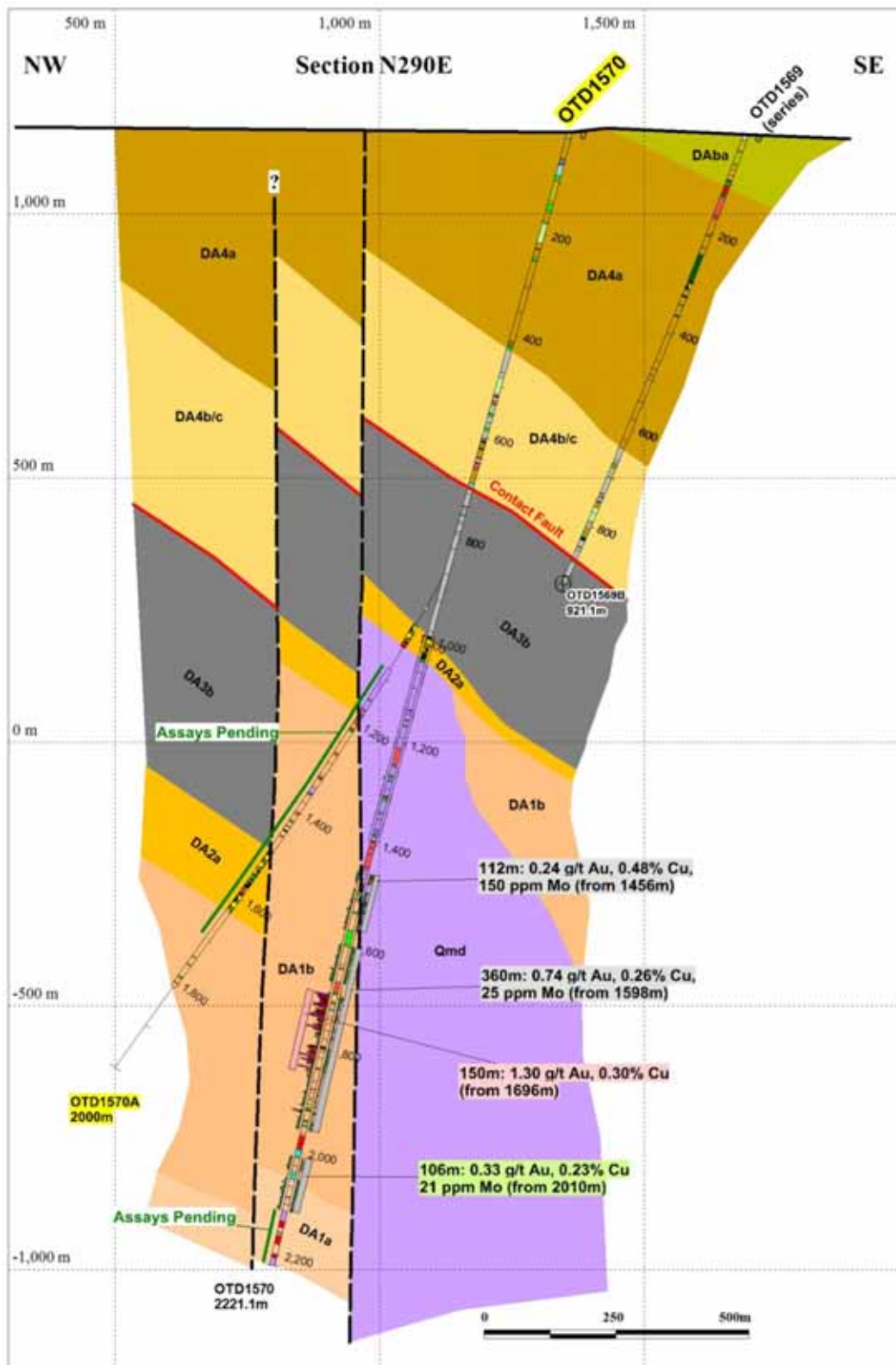
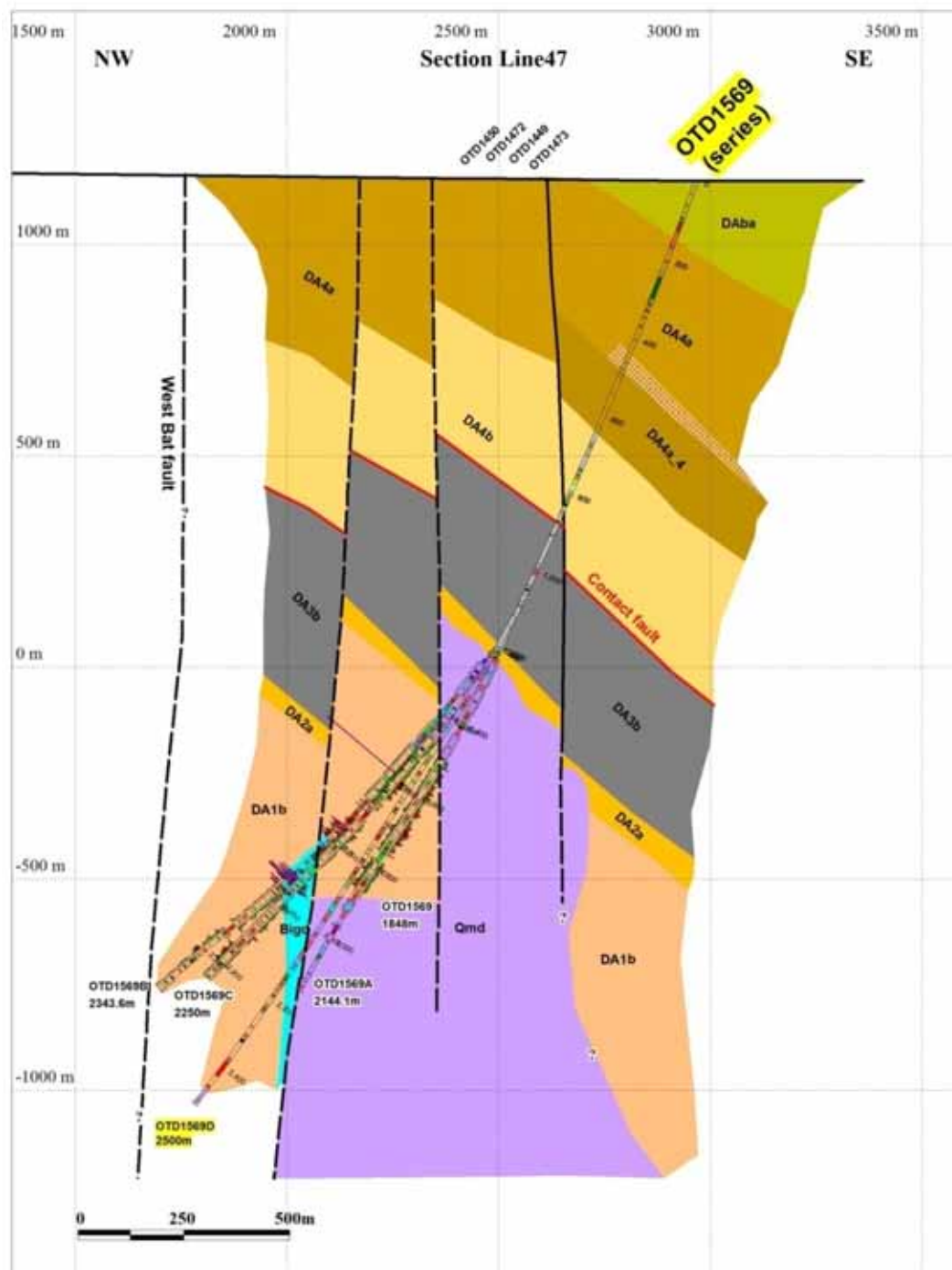


Figure 9.16 Heruga North Geology, Section OTD1569



9.4 Javkhlant

9.4.1 Introduction

The Javkhlant Area is located at the southern side of Heruga across a major east-west fault; and continues southward to the southern border of the Entrée-OT LLC Javkhlant licence. The focus of exploration at Javkhlant has been the Javkhlant I and Javkhlant II IP anomalies that were located during the regional gradient array Iris IP survey carried out by Delta Geoscience. Since that time the IP anomalies have been drilled and the area mapped at 1:2,500 scale. A detailed ground magnetometer survey has also been carried out over the area, and further Zeus IP surveying has been done over most of the area (Figure 9.17 to Figure 9.18).

Figure 9.17 Javkhlant Geology

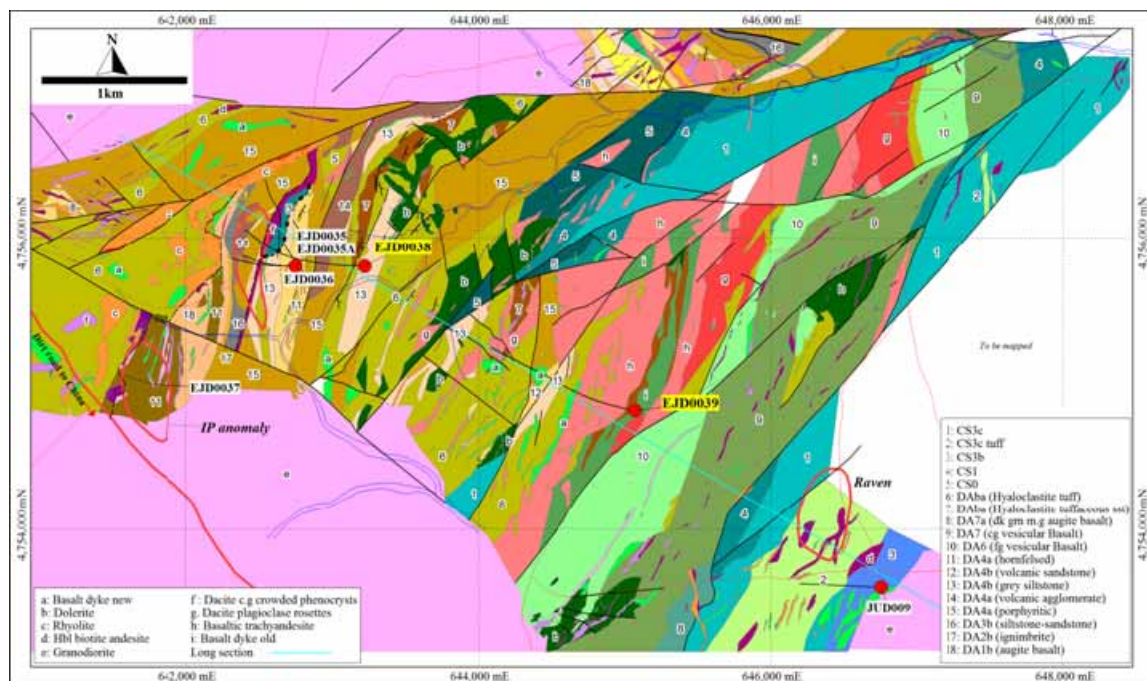
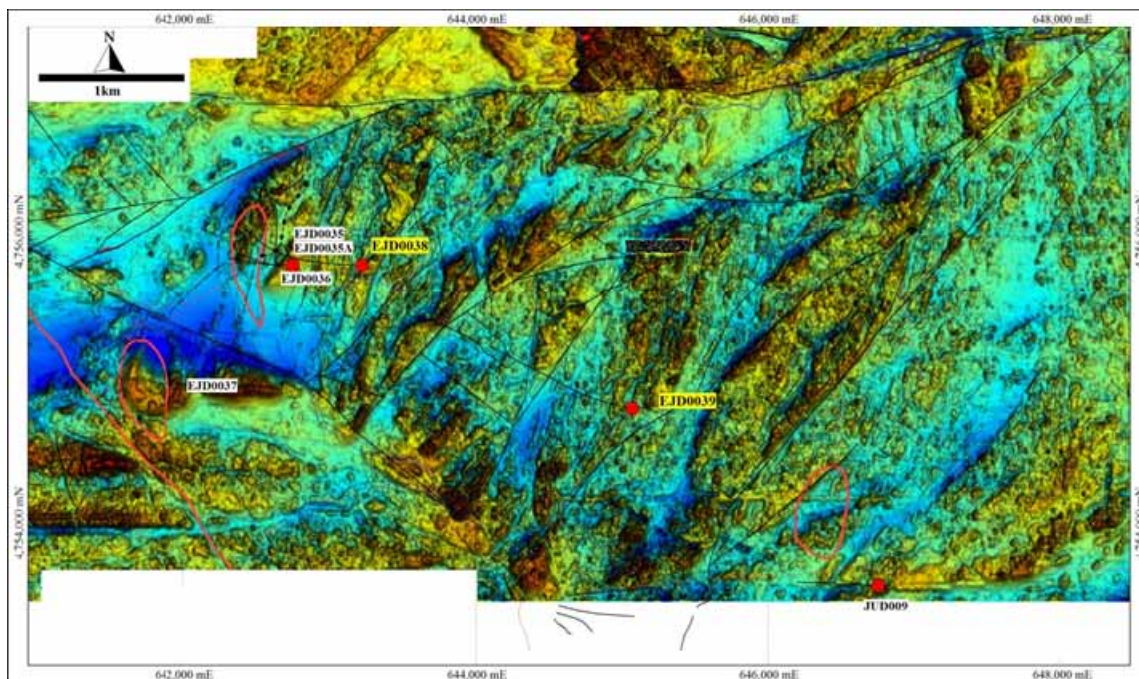


Figure 9.18 Javkhlant Ground Magnetics



9.4.2 Javkhlant Drilling

The Javkhlant target (Geology, magnetics, IP Figure 9.17) consists of two IP anomalies Javkhlant I to the north-east and Javkhlant II to the south-west in Devonian rocks located some 5 km to the south-west of Heruga. The IP anomalies were discovered during regional IP surveying in 2005. Javkhlant II, the stronger of the anomalies was followed up by detailed IP profiling in 2005. Geological mapping since that time suggests that the area may be the faulted offset of Heruga mineralization, displaced 5 km to the west on an east-north-east trending fault. This is considered to be the southernmost known mineralization on the OT trend. Mineralization at Javkhlant I was discovered in May of 2010 by targeting the IP anomaly. Hole EGD0035A went into mineralized dacitic ignimbrite (Da2b) at a down hole depth of 1,384 m and stayed in mineralization to 1,679 m before going into unmineralized intrusive. Although the overall interval was low grade, a 32 m interval from 1,422 m to 1,454 m down hole averaged 0.92% Cu. The style of mineralization with advanced argillic alteration is typical of the shallower parts of mineralized systems at Oyu Tolgoi, which suggests potential for higher grade mineralization at Javkhlant I exists deeper within the system.

Subsequent to the initial discovery, detailed geological mapping, ground magnetic and Zeus IP surveying have better defined the IP anomalies and their geological context. Javkhlant I and Javkhlant II may have been a continuous zone of mineralization that has been cut and displaced by a NW trending fault. At the Javkhlant II anomaly the presence of DA1b augite basalt, possible altered quartz monzodiorite, and a prominent hillock of strongly altered rock with anomalous gold values (J. Forster pers comm.) all suggest that mineralization is subcropping. On the western side of the anomaly, a fault defines the boundary with Carboniferous rocks which may be a southward continuation of the

West Bat Fault. A ground magnetic anomaly (Figure 9.18) corresponds with the IP anomaly.

During the quarter EJD0037 was drilled targeting the Javkhant II anomaly. The entire drillhole was strongly hornfelsed due to its close proximity to the Javkhant granodiorite batholith, relected in abundant secondary biotite, and sparse garnet. Hornfelsing has destroyed a lot of the primary textures but clearly the first half of the hole down to around 500 m is largely DA1b augite basalt intruded by andesitic dykes. A coarse conglomerate assigned to unit DA2a was also intersected containing some DA1b clasts. The DA1b basalt contains disseminated and vein pyrite/pyrrhotite with pyrite up to 8% and trace chalcopyrite. There are rare porphyry style quartz veinlets. From 500 m to the end of the hole at 877.4 m are unmineralised trachyandesite volcanoclastics and fine siltstones, most likely correlatable to unit CS3. There is a characteristic tuffaceous sandstone near the end of the hole containing quartzite clasts similar to surface outcrops directly above and probably part of unit CS2. The hole intersected four zones averaging 30 metres thickness from 232 m to 504 m with a grade averaging 0.15% copper, 0.05g/t Au, and 30 ppm molybdenum.

The large amount of pyrite intersected adequately explains the IP anomaly. The Zeus IP section is essentially vertical sided and corresponds well with the older IRIS IP data.

Figure 9.19 Javkhant Section EJD0039

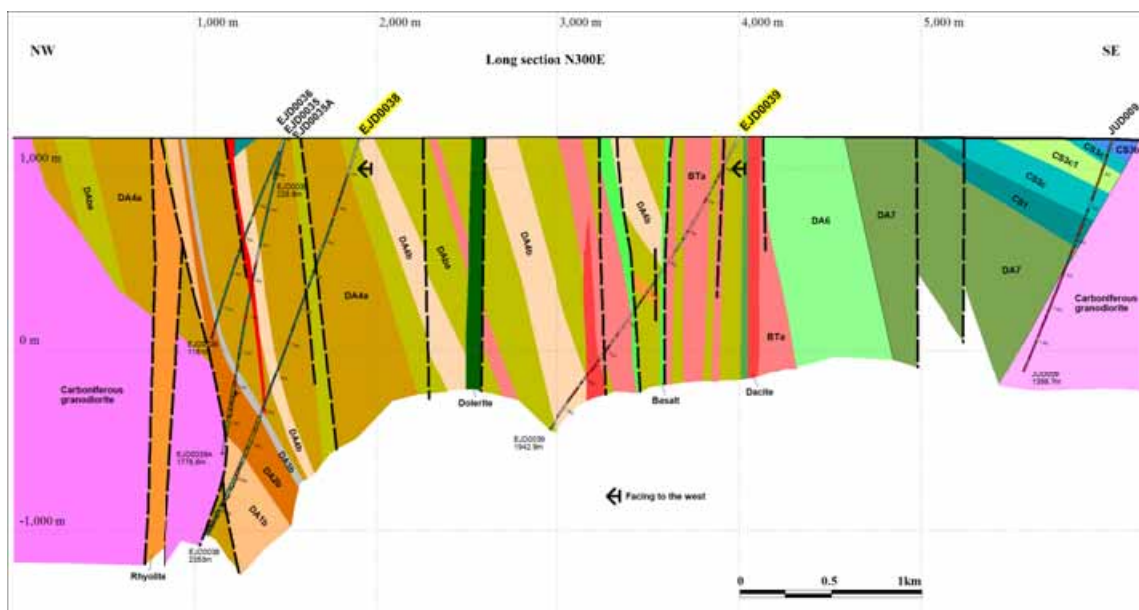


Figure 9.20 Javkhlant Section EJD0039 with GoCad Targets

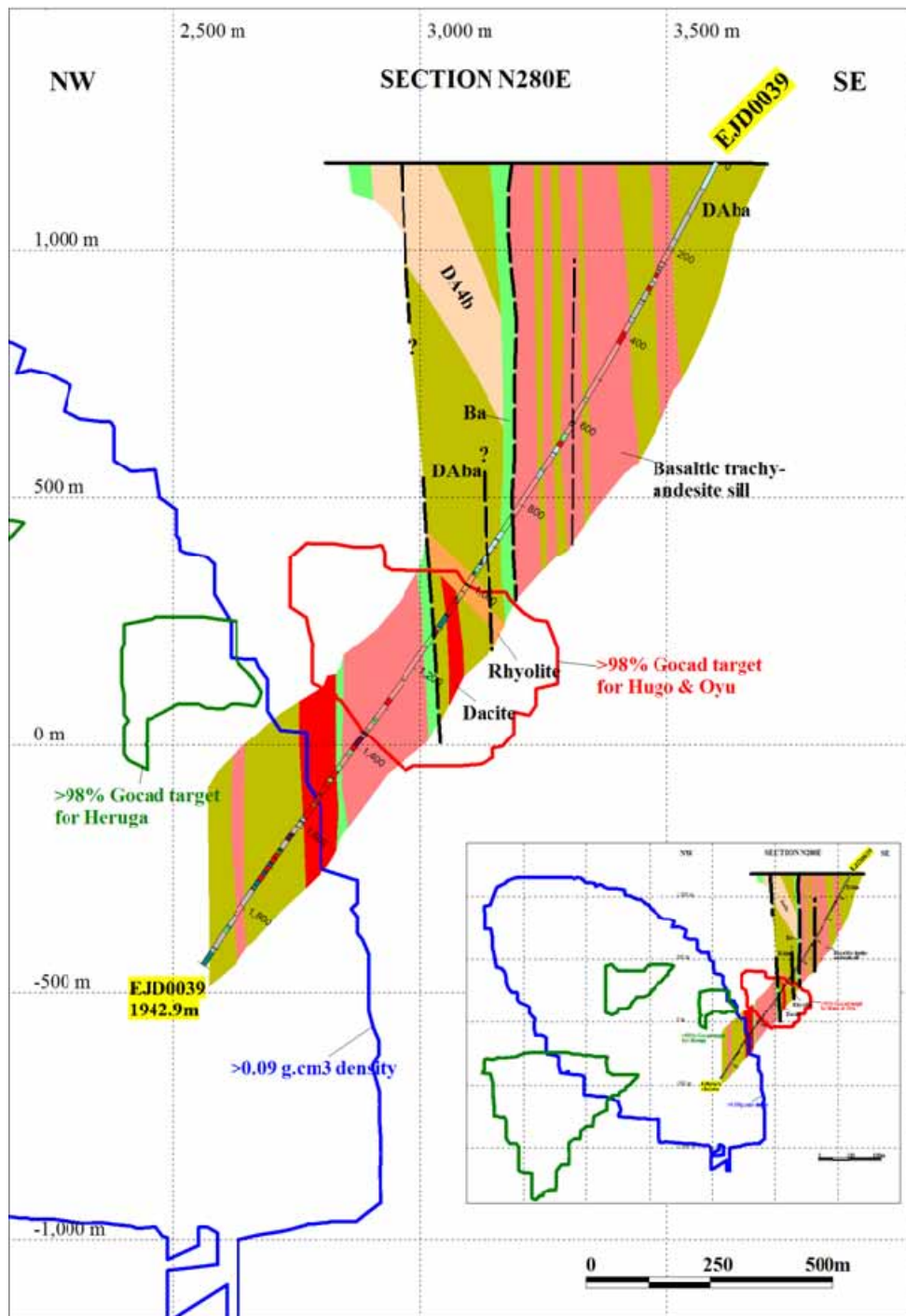
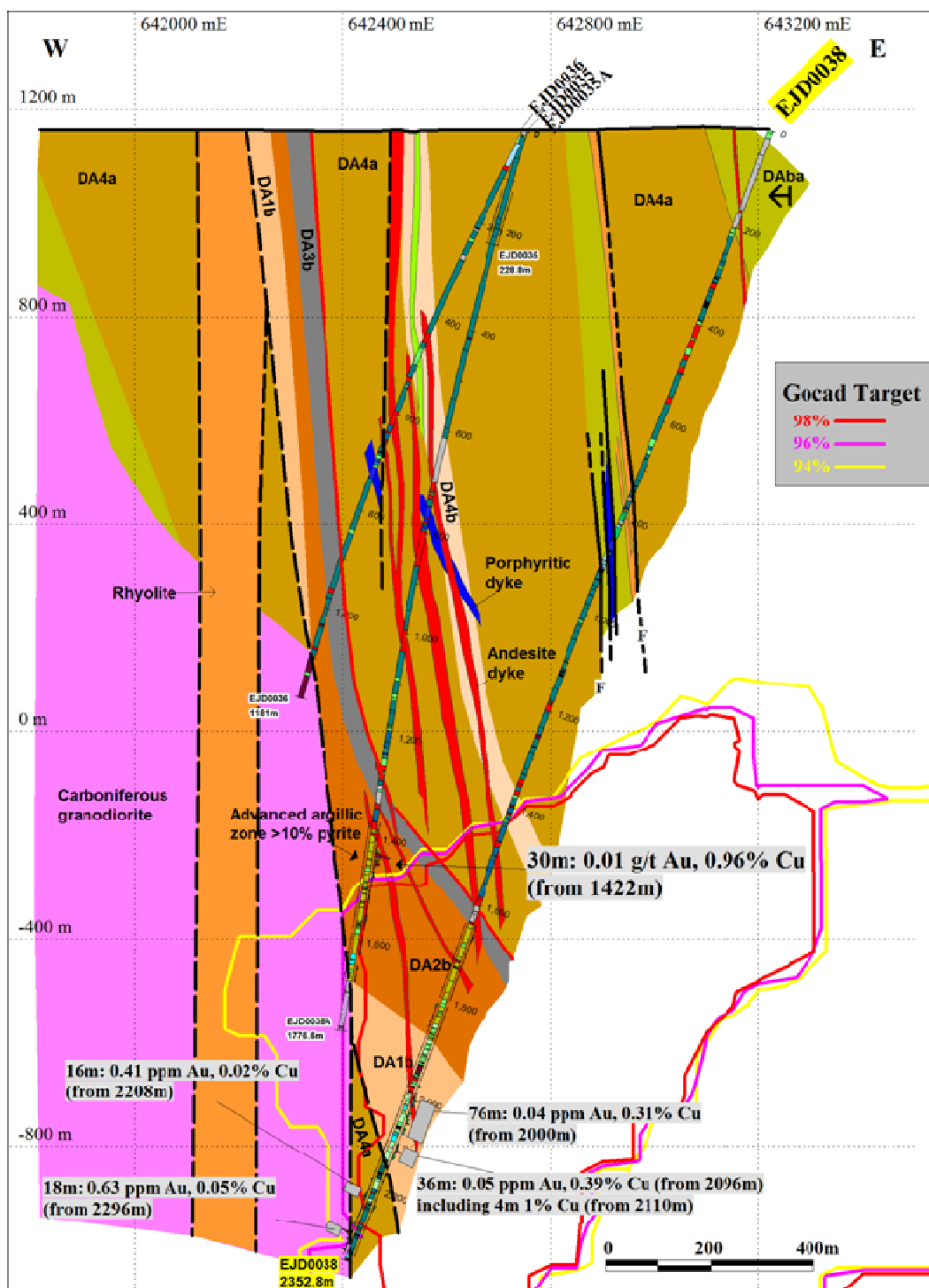


Figure 9.21 Javkhlant Section EJD0038 with GoCad Targets



9.5 References

The information for this section is in the report Oyu Tolgoi Technical Report, March 2008.

10 DRILLING

10.1 Introduction

IMMI (now OT LLC) sampling programs on the Project have included stream sediment, soil, rock chip, drill core and RC samples. All of the early-stage sampling methods have been superseded by the drilling, which forms the basis of the Mineral Resource estimates discussed in this report and, therefore, the early-stage sampling methods are not discussed in this report.

No documentation was provided for any of the previous exploration by BHP, including diamond drill programs, therefore the BHP sampling methods are not discussed in this section.

10.1.1 Diamond Drill Core Sampling

Sampling for resource estimation has been conducted on diamond drill core obtained from IMMI (now OT LLC) holes drilled between 2002 and March 2012. The current core cutting protocol is given as follows:

- The uncovered core boxes are transferred from the logging area to the cutting shed by forklift on wooden pallets.
- Long pieces of core are broken into smaller segments with a hammer.
- Core is cut with a diamond saw, following the line marked by the geologist. The rock saw is flushed regularly with fresh water.
- Both halves of the core are returned to the box in their original orientation with one piece on top of the other.
- The uncovered core boxes are transferred from the cutting shed to the sampling area by fork lift carrying several boxes on a wooden palette; constant 2 m sample intervals are measured and marked on both the core and the core box with permanent marker; a sample tag is stapled to the box at the end of each 2 m sample interval, and sample numbers are pre-determined and account for the insertion of QA/QC samples (core twins, standards, blanks).
- Samples are bagged. These are always half core samples collected from the same side of the core. Each sample is properly identified with inner tags and outside marked numbers. Samples are regularly transferred to a sample preparation facility, operated by SGS Mongolia LLC (SGS Mongolia), which is located approximately 50 m from the sample bagging area.
- The unsampled half of the core remains in the box, in its original orientation, as a permanent record. It is transferred to the on-site core storage area.

Barren dykes that extend more than 10 m along the core length are generally not sampled. Figure 7.5 to Figure 7.8 are representative sections across the Hugo North Deposit and show significant length-weighted intervals across the main mineralized zones.

QG has reviewed the sampling and handling of core during four site visits between 2007 and 2011. QG has observed photography and core being cut, and reviewed the quality of photos. The procedure and equipment for core re-construction, mark-up, logging, cutting, and sampling observed by QG are fit for purpose. Samples are taken on regular 2 m intervals and post-mineralization dykes over 10 m thick down hole were not sampled. The observed core was cut in half correctly.

QG concludes that the sampling is sufficient for the purposes of resource estimation on porphyry style deposits.

10.1.2 RC Sampling

This method of drilling forms a very small part of the support for the Mineral Resource estimates at the Southern Oyu Deposits, representing approximately 1% of the total drilling used for resource estimation. RC drilling is also used for condemnation, geotechnical and water drilling purposes.

10.2 Oyu Tolgoi

10.2.1 Introduction

Diamond drillholes are the principal source of geological and grade data for the Oyu Tolgoi Project. A small percentage (~3%) of the drilling total comes from reverse circulation (RC) or combined RC/core drilling. Most of the RC holes were drilled in the early days of exploration (2000 and 2001) at the Southern Oyu deposits, whereas the combined RC/core holes, which have RC at the top of the hole and core drilling in the bottom, are a small percentage of the total number of holes on the Project. RC holes drilled in the early stage of exploration (pre-2005) are not used in any of the resource estimates at Oyu Tolgoi, a total of seven RC holes drilled in February 2005 were included in the Southern Oyu resource estimate.

The first drilling on the Property was done by BHP in 1997 and 1998, during which time 23 core holes (3,902 m) were drilled at the Southern Oyu deposits. IMMI (now OT LLC) carried out approximately 109 holes (8,828 m) of RC drilling in 2000, mainly at Central Oyu, to explore the chalcocite blanket discovered earlier by BHP. In 2001, IMMI (now OT LLC) continued RC drilling (16 holes totalling approximately 2,091 m), mostly in the South Oyu area; however, on hole number OTRCD149 they tested a combined RC collar/core tail drilling method. Two additional holes were drilled using this method (OTRCD50 and 52), along with an additional seven RC holes totalling 801.5 m (up to hole OTRC158), before Ivanhoe switched to core drilling methods for all of their exploration. In 2011, as part of the mine development program, OT LLC drilled 19 holes for approximately 11,000 m.

The majority of drilling on the Hugo Dummett deposit was carried out from 2003 to mid 2008. Drilling at the extensions of the Hugo North deposit (in the Shivee Tolgoi Joint Venture Property) started in March 2004 and continued to mid 2007. Although resource definition drilling terminated in April of 2006, geotechnical drilling continued into 2008. Geotechnical drill core is released for sampling and assayed once it is no longer required for geotechnical purposes. Core from a number of geotechnical drillholes was released for sampling and assayed during 2008. Some geotechnical holes remain

unsampled, (OTD1486, OTD1491). The above work has been the subject of previous Technical Reports including: Juras, 2003, 2004a, 2004b and 2005; and Blower 2006a, 2006b. In 2010 and 2011, OT LLC commenced a program of underground and surface drilling aimed at infilling the area mostly around the proposed Lift 1 of the Hugo North cave. This drilling is ongoing in 2012.

As of March 2012, a total of approximately 1,000,000 m of drilling in 2,339 holes has been completed on the Project (see Table 10.1). Of this, 926,446 m was core drilling in 1,679 holes; 59,059 m was completed in 629 RC holes. This drilling has been spread mostly over the Hugo Dummett, Southern Oyu, and Heruga deposits; however, smaller reconnaissance programs have been drilled by IMMI (now OT LLC) at various other targets as well.

Table 10.1 Oyu Tolgoi Project – Drilling Summary

Deposit	DDH Holes	Length of DDH (m)	RC Holes	Length of RC (m)	Combined RC/DDH Holes	Length of Combined RC/DDH (m) ⁴	All Holes ¹	Total Length (m) ¹
Hugo Dummett Deposits								
Hugo South	140	96,133.66	45	3,263	12	7,988.95	197	107,385.61
Hugo North	426	276,280.54	4	319	4	2,417.5	434	279,017.04
Total Hugo Dummett	566	372,414.2	49	3,582	0	0	631	386,402.65
Southern Oyu Deposits								
Central	240	90,511.63	71	6,694	0	0	311	97,205.63
Southwest	225	128,256.13	28	4,576	3	2,092	256	134,924.13
South	92	34,432.89	28	3,515	2	890.9	122	38,838.79
Wedge	47	27,186.85	12	1,337.5	0	0	59	28,524.35
Total Southern Oyu	604	280,387.5	139	16,122.5	0	0	748	299,492.9
"Other" Drilling								
West Oyu	47	17,614.35	115	4,929	0	0	162	22,543.35
Shaft Farm (exploration and geotechnical)	25	15,637.2	28	846.8	0	0	53	16,484
X-Grid	6	571	0	0	0	0	6	571
East Side Licence	26	6,300.85	157	19,413	8	2,400	191	28,113.85
"Other" ³	179	13,706.96	54	6,016	0	0	233	19,722.96
Total "Other" Drilling	283	53,830.36	354	31,204.8	0	0	645	87,435.16
Far South	58	60,303.55	14	3,272	0	0	72	63,575.55
Shivee Tolgoi Earn in property-Hugo North	117	93,448.45	73	4,868	2	736	192	99,052.45
Heruga-Javkhant	51	66,062.15	0	0	0	0	51	66,062.15
Total Drilling	1,396	872,616	275	27,845	0	0	1,694	914,586
Total "Other" Drilling	283	53,830.36	354	31,204.8	0	0	645	87,435.16

Notes:

1. Includes all holes drilled to 27 February 2012.. Not all of this drilling is relevant to the Mineral Resource estimate, described in Section 14 of this report.
2. A portion of these holes were collared in the Shivee Tolgoi Joint Venture Property and drilled into the Oyu Tolgoi property.

3. Includes holes drilled for geotechnical, water, and condemnation purposes.
4. Approximately 79% of the combined RC/core drilling is by core methods.
5. Includes one drillhole at Castle Rock and two on South Sparrow IP anomaly.

The Hugo Dummett deposit resources were last estimated in 2007 and reported in the 2007 Technical Report (AMEC Minproc, 2007). Between cut-off dates for the 2007 technical report and the 2008 technical report, Ivanhoe completed 27 new exploration drillholes totalling approximately 27,000 m in the Hugo North and Shivee Tolgoi area. In 2008, the result of this drilling was compared to the existing resource model for Hugo Dummett and it was felt that the results would not materially affect the current resource (AMEC Minproc, 2008). Since the 2011 technical report, approximately 10,700 m additional metres from 19 holes were drilled in part of Lift 1 of the planned Hugo North cave. Drilling is ongoing for and a new estimate has not been completed yet. To check that the Mineral Resource estimate reported in this Technical Report is still relevant and valid, QG compared the 2011 drilling with the pre-2011 drilling and the previously reported model. Figure 10.1 to Figure 10.6 show in section and plan that the new drilling has generated copper results in line with previous drilling and that a new model is unlikely to be materially different.

Figure 10.1 Hugo North Section 4767500N with Pre-2011 Drilling (Cu Grades Shown)



Figure 10.2 Hugo North Section 4767500N with 2011 and Pre-2011 Drilling (Cu Grades Shown)

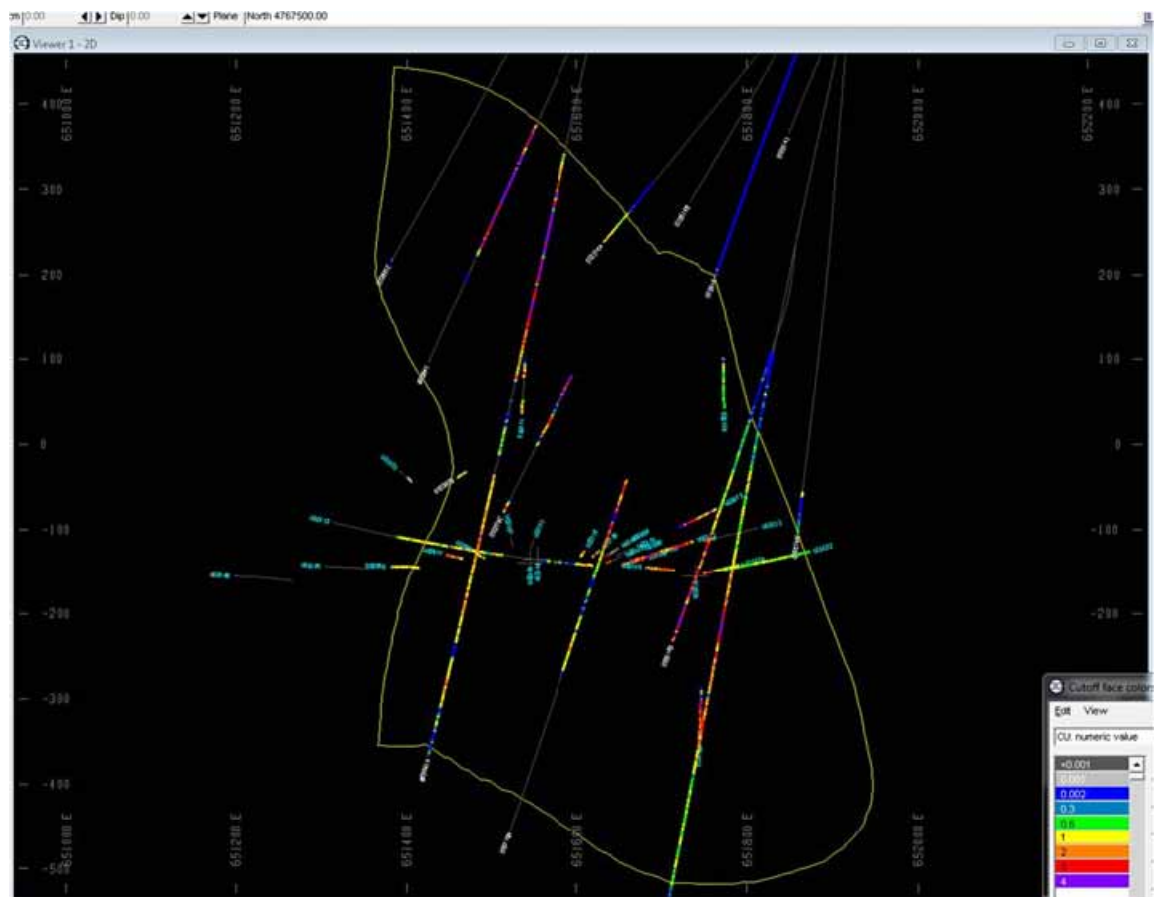


Figure 10.3 Hugo North Section 4767500N with all Drilling and Currently Reported Block Grades (Cu Grades Shown)

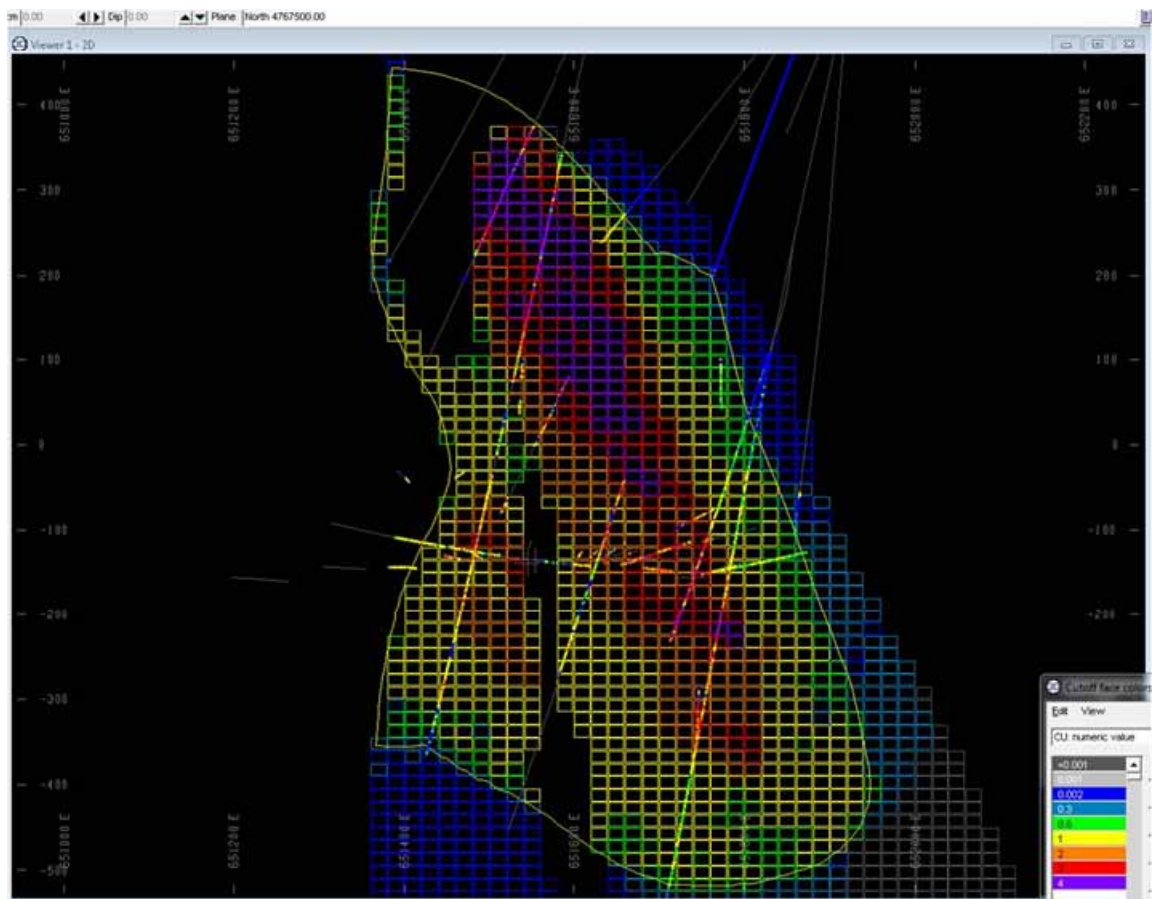


Figure 10.4 Hugo North Plan -150RL with Pre-2011 Drilling (Cu Grades Shown)

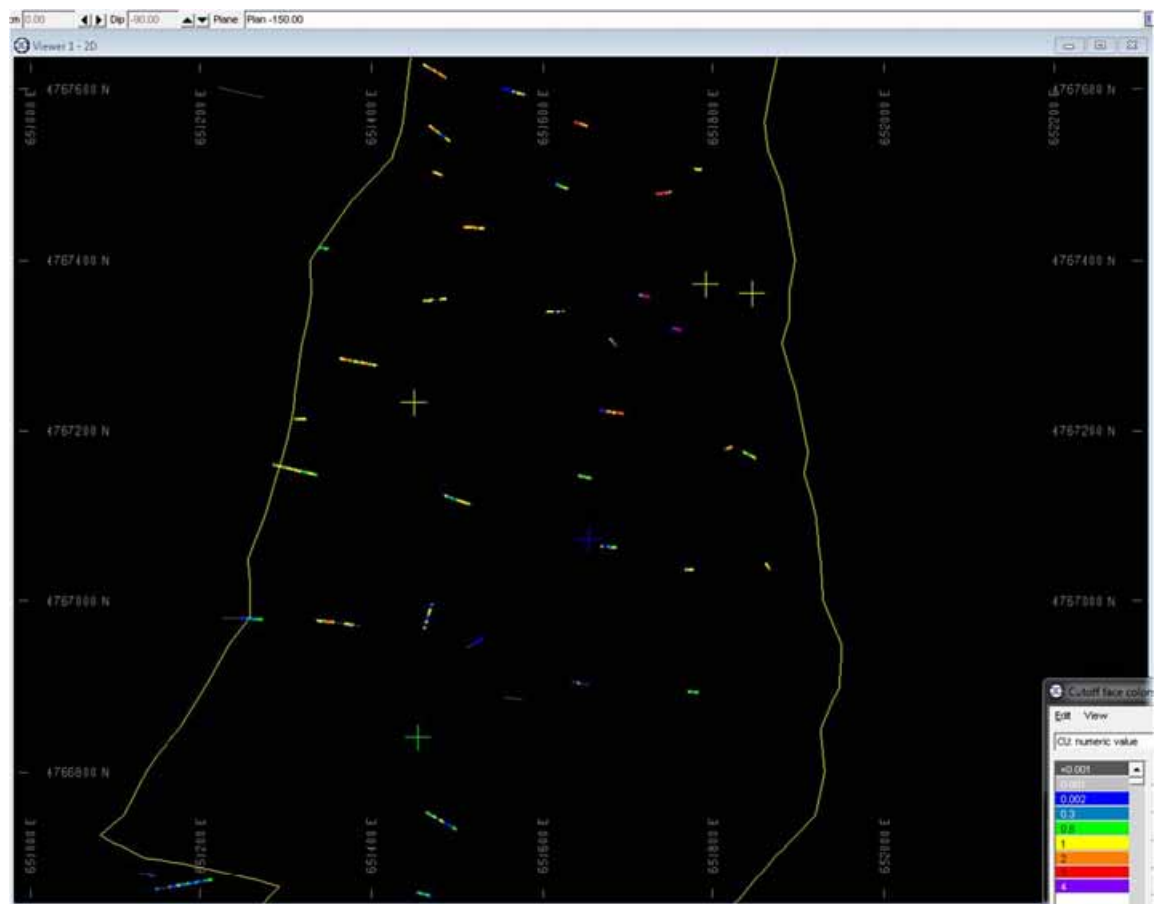


Figure 10.5 Hugo North Plan -150RL with 2011 and Pre-2011 Drilling (Cu Grades Shown)

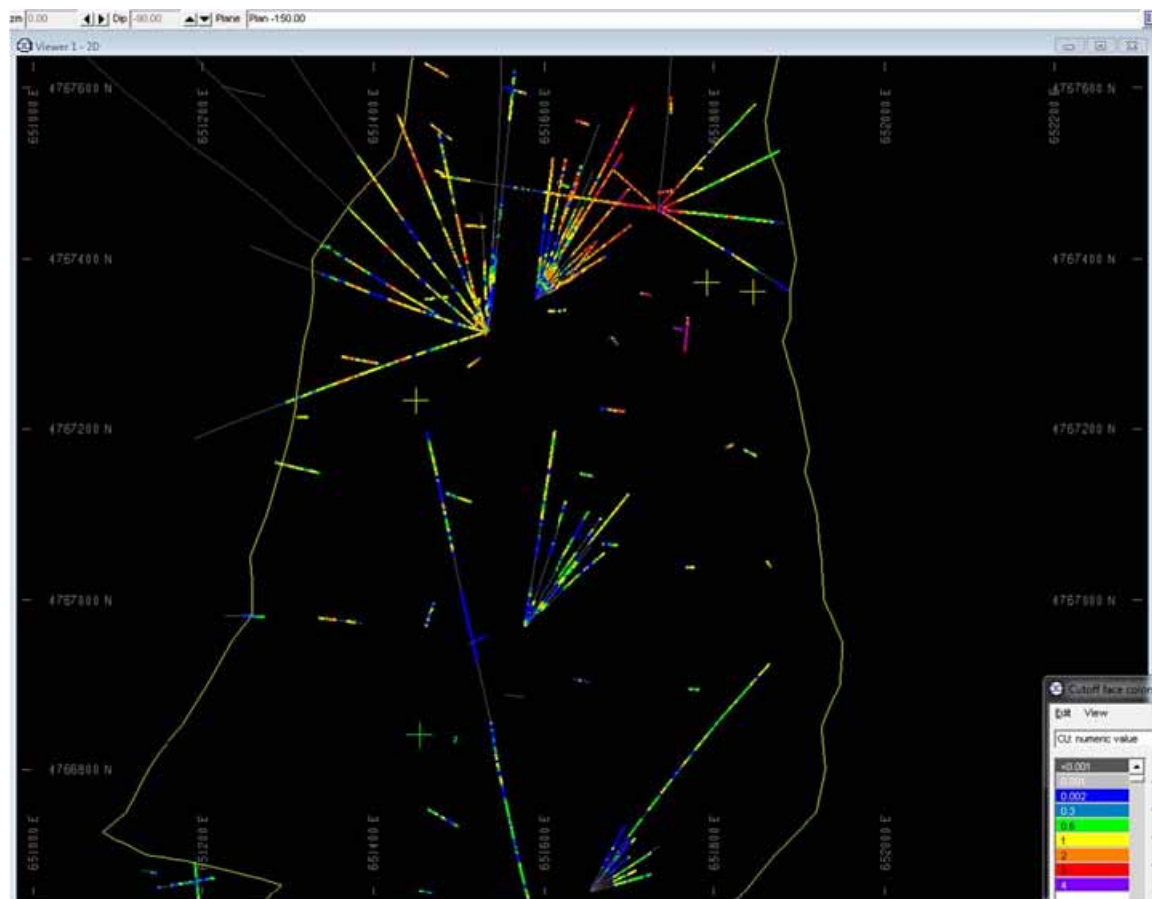
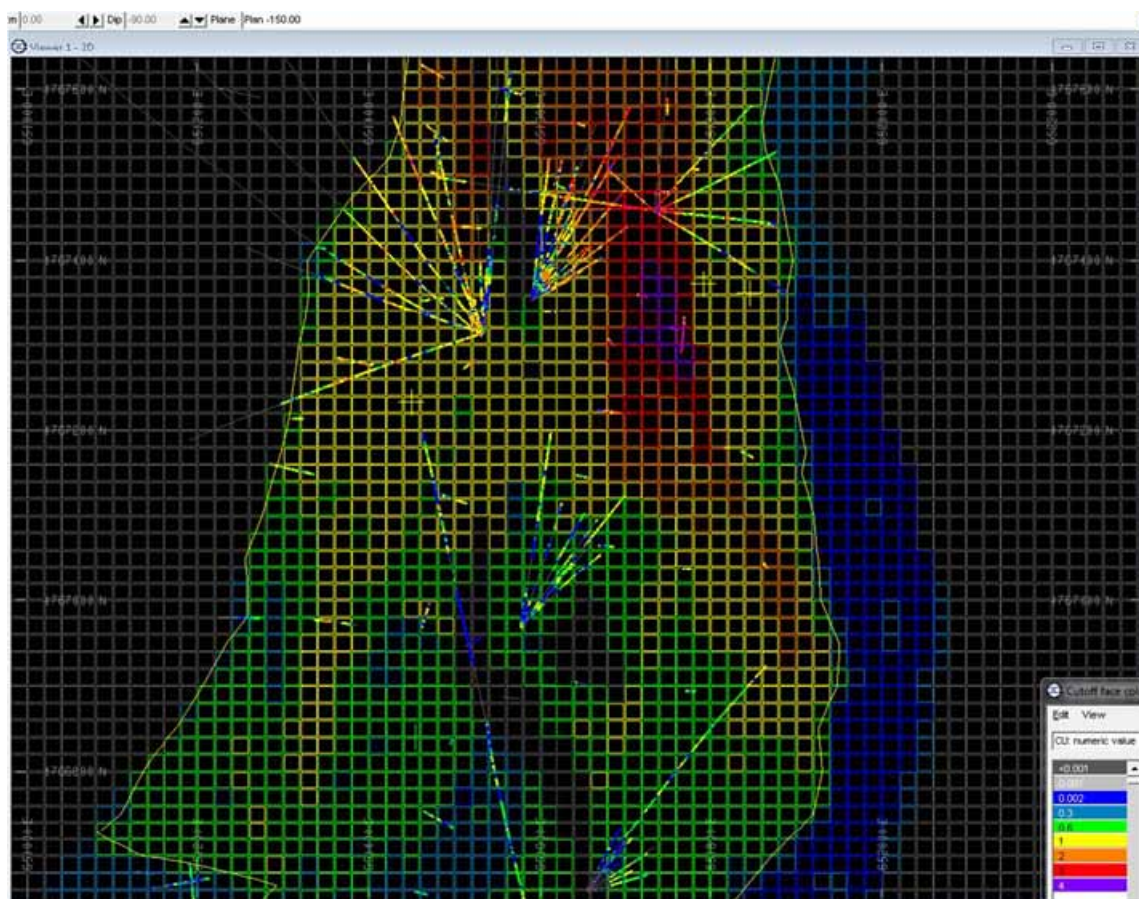


Figure 10.6 Hugo North Plan -150RL with all Drilling and Currently Reported Block Grades (Cu Grades Shown)



Ivanhoe has drilled five new holes into the Heruga Deposit since the last resource update reported in the 2008 technical report, (AMEC Minproc, 2008), consisting of two on the EJV Javkhant Licence and three on the adjacent Oyu Tolgoi Licence. In Heruga North, there are insufficient holes to generate a Mineral Resource estimate at present.

Similar to Hugo North, additional drilling has been completed in Southern Oyu in 2011. A new Mineral Resource estimate is in progress and should be ready for reporting in 2013. Preliminary investigations suggest that the new model is not materially different to the previous model. QG consider therefore that the current model is still valid and appropriate to use in mine planning studies up to when the new model is finalised.

10.2.2 Exploration Diamond Drilling

Exploration diamond drilling on the Project has been almost continuous since 2001, using a variety of different contractors. The majority of the more recent drilling was contracted to Major Drilling Mongolia LLC (based out of Australia, who have used a variety of rigs, some with depth capabilities close to 2,000 m. Rigs which have recently been, or are currently on site, include UDR 1000, 1500, and 5000 and Major 50 units. Other drilling campaigns have been completed by Gobi Drilling, Can Asia, Mongolia Drilling Services, Australian Independent Diamond Drillers and Soil Trade.

The vast majority of core diameters at the Southern Oyu and Hugo Dummett deposits have either been PQ-size (85 mm nominal core diameter), HQ-size (63.5 mm nominal core diameter) or NQ-size (47.6 mm nominal core diameter), with a small percentage drilled with BQ-size core (35.5 mm nominal core diameter). Most holes at Hugo North were collared with PQ core and were reduced to HQ at depths of around 500 m prior to entering the mineralized zone. A few holes have continued to depths of about 1,300 m using PQ diameter. Some holes were reduced to NQ-size at varying depths depending on drilling conditions.

Many of the deeper holes, especially at Hugo North (including Copper Flats), include multiple “daughter” holes (wedges) drilled from a PQ diameter “parent” drillhole. A bend is placed in the parent hole at the location where the planned daughter holes are to branch off. The bend is achieved by means of a Navi-Drill® (navi) bit, which is lowered down the hole to the desired depth and aligned in the azimuth of the desired bend. As the navi bit advances, a bend is achieved at the rate of 1° every 3 m. No core is recovered from the navi-drilled interval and the core diameter is reduced, generally to HQ size.

The majority of core has been drilled using “Ball Mark” or “Ace” oriented core marking systems to assist with geological and structural interpretations and for geotechnical purposes.

Upon completion of a drillhole, the collar and anchor rods are removed, and a PVC pipe is inserted into the hole. The hole collar is marked by a cement block inscribed with the hole number (e.g. OTD663). Drillhole collars are located relative to a property grid using UTM coordinates with datum set to WGS-84 Zone 48N (see Section 4.2.3). Proposed hole collars and completed collars are surveyed by a hand held GPS unit for preliminary interpretations. After the hole is completed it is re-surveyed using a Nikon Theodolite instrument. The two collar readings are compared and if any significant differences are noted the total station reading is re-surveyed, otherwise it is adopted as the final collar reading.

The holes are drilled at a wide range of azimuths and dips depending upon the orientation of the mineralization (see Section 7.2). Average drillhole lengths at the Hugo Dummett and Southern Oyu deposits range from 316 m (Oyu South) to 894 m (Hugo North), and average around 525 m (see Table 10.2). Down hole surveys are systematically conducted along each hole.

Table 10.2 Oyu Tolgoi Project – Average Hole Lengths

Deposit	Average Hole Length (m)*
Hugo Dummett Deposits	
Hugo South	545
Hugo North	642
Total Hugo Dummett	612
Southern Oyu Deposits	
Central	312
Southwest	219
South	318
Wedge	483
Total Southern Oyu	400
“Other” Drilling	
Far South	822
West Oyu	139
Shaft Farm	311
X-Grid	95
East Side Licence	147
Condemnation, water, geotechnical, etc.	84
Total “Other” Drilling	210
EJV Shivee Tolgoi licence (includes Hugo North Extension)	515
EJV Javkhlant Licence (includes Heruga)	1,295
Total Drilling	539

10.2.3 Core Drilling Procedures

Current core drilling and sample collection procedures are given as follows.

At the drill rig core is removed from the core barrel by the drillers and placed directly in wooden core boxes. Individual drill runs are identified with small wooden blocks, where the depth (m) and hole number are recorded. In general, the core-handling procedures at the drill site appeared to be very good. Unsourced core is never left unattended at the rig; boxes are transported to the OT LLC core logging facility twice a day under a geologist’s or technician’s supervision. Core is transported in open boxes in the back of a truck.

Upon arrival at the core shed, the core is subject to the following procedures:

- Quick review.
- Box labelling: core boxes are identified with the hole number, metres “from–to”, and box number written with a permanent marker on the front.
- Core “re-building” (core is rotated to fit the ends of the adjoining broken pieces).
- Core photography.

- Geotechnical logging, using pre-established codes and logging forms, includes: length of core run, recovered/drilled ratio, RQD, and maximum length, structural data and oriented core data (orientated core measurements were logged as interval data using standardized codes for structural and vein data only; the orientated core measurements usually did not commence until the hole was within the mineralized zone).
- Geological logging: this is completed on paper logging forms, in accordance with the company protocol, which includes: header information, lithology description and litho code, graphic log, coded mineralization and alteration. The geologist marks a single, “unbiased” cutting line along the entire length of the core.
- Core cutting and sample collection is discussed in Section 10.1.1.

All diamond core is stored in a secure location at the main camp. Core is stacked on pallets in a stable, 3 x 3 box configuration to a height of about 15 boxes. Core logging facilities are indoors and logging takes place on sturdy steel racks, each of which is capable of holding upwards of 25 or more boxes.

10.2.4 Down Hole Surveys

Ivanhoe uses down hole survey instruments to collect the azimuth and inclination at specific depths of the diamond drillholes for the majority of the diamond drilling programs. Six principal types of survey method have been used over the duration of the drilling programs including Eastman Kodak, Flexit, Ranger, Gyro, and N-seeking gyro methods.

No down hole survey data was collected during first 149 holes drilled on the Project, including the initial diamond drill program by BHP in 1998 and the 125 RC holes completed by IMMI (now OT LLC) in 2001 and 2002. The first surveys initiated by Ivanhoe were for holes OTRCD149, 150 and 152, which were surveyed by the Eastman Kodak method. This method, along with gyro and Ranger methods were interchangeably used by Ivanhoe as the principal methods to measure deviations until approximately hole OTD397. Beyond this, gyro, N-seeking gyro, Flexit and Ranger have been the principal methods used. A small percentage of the holes in the database remain unsurveyed. It should be noted that the Eastman Kodak, Pontil, Flexit and Ranger methods derive azimuth measurements using a magnet and are therefore subject to potential problems that can be caused by magnetic minerals, common at some of the deposits on the Project.

Since January 2006, the procedure has been to measure deviations initially using a Flexit instrument along approximate 50 m intervals in order to monitor the drillhole progress. At completion, all holes are resurveyed with a north-seeking gyro or “SRG”-gyro instrument over approximate 5 to 20 m intervals. The gyro instruments are not dependent on magnetic readings and are therefore considered to be more appropriate methods for this style of deposit and the depth of the holes. Gyro methods have been used for the majority of holes drilled into the Hugo North Deposit, which are incorporated into this current resource estimate.

Ivanhoe has a detailed validation program built into their database to reveal any moderate kinks or deviations in the down hole data. All of these are checked, and if required, adjusted, prior to finalization of the database.

10.2.5 Recoveries and RQD

In general, Ivanhoe reports that core recoveries obtained by the various drilling contractors have been very good, averaging between 97% and 99% for all of the deposits. In localized areas of faulting and/or fracturing the recoveries decrease; however, this is a very small percentage of the overall mineralized zones. In addition, Ivanhoe notes decreased recoveries near-surface in overlying non-mineralized Cretaceous clays and to a lesser extent in a portion of the oxidized rocks (generally above 100 m depth). Table 10.3 shows the recovery by year for all drilling and for drilling below 100 m depth.

Table 10.3 Core Recovery by Drilling Year – All Deposits

Year	All Drilling		Recoveries Below 100 m	
	Recovery (%)	Number of Measurements	Recovery (%)	Number of Measurements
1998	75.6	19	N/A	N/A
2001	97.4	5,784	98.4	3,876
2002	97.8	33,964	98.6	26,359
2003	97.4	61,182	98.8	48,722
2004	97.7	66,116	98.6	54,605
2005	98.6	25,224	99.1	21,927
2006	98.5	21,570	99.1	17,909
2007	98.3	17,986	98.4	15,867
2008	99.5	8,905	99.6	8,151
2009	99.8	1,956	99.8	1,845

Core recovery and RQD are measured at the core logging area by IMMI's (now OT LLC) Mongolian geology staff where the following measurements are recorded:

- Block interval
- Drill run (m)
- Measured length (m)
- Calculated recovery (%)
- RQD measured length (m)
- Calculated RQD (%)

Recovery and RQD measurements are recorded in the geology database and have been collected during all core programs. The methodology used for measuring recovery is standard industry practice.

10.2.6 Bulk Densities

Ivanhoe has collected an extensive database of bulk density determinations from core samples, dating back to drilling completed in 2002. Currently, there are over 52,000 determinations in the database and these are divided among the main deposits on the Project as shown in Table 10.4.

Table 10.4 Distribution of Bulk Density Determinations

Deposit	Number of Determinations
Hugo North	16,440
Hugo South	8,803
Total Hugo Dummett	25,243
Central Oyu	6,156
South West Oyu	9,757
South Oyu	1,682
Wedge Zone	1,517
Total Southern Oyu	19,112
EJV Shivee Tolgoi licence (includes Hugo North Extension)	4,001
EJV Javkhlant Licence (includes Heruga)	2,632
Plantsite-ConCamp	371
Shaft Farm	1,200
East Side Licence	18
West Oyu	801
Far South (includes Heruga North)	469

Samples for specific gravity determination are taken at approximately 10 m intervals per drillhole and tabulated by rock type. The specific gravity for non-porous samples (the most common type) is calculated using the weights of representative samples in water (W2) and in air (W1). The bulk density is calculated by the formula:

$$\frac{W1}{(W1-W2)}$$

Less common porous samples were dried and then coated with paraffin before weighing. Allowance was made for the weight and volume of the paraffin when calculating the specific gravity.

10.2.7 Database Management

In 2011, all geological, geochemical geotechnical drillhole data was transferred from an MS Access relational database to an Acquire database management system. An audit of the transfer process was completed by AMEC consultants in mid 2011 but reporting had not been completed by March 2012. That aside, the two most important projects (in terms of the mining schedule) - Southern Oyu and Hugo North - are currently being re-estimated by OT LLC with the assistance of AMEC. Preliminary checks show no differences in drill data when comparing to data used (and extracted from the old Access

database) to the data from the new Acquire database. On this basis, QG accepts that the data are likely to be suitable for ongoing use but an independent audit report should be forthcoming.

Prior to 2011, drillhole data was initially manually recorded in the field or in the core logging shed on paper logging sheets. Now information is recorded directly into Acquire based forms on laptops. Geological logging is carried out by the logging geologist who collects information including lithology, alteration and mineralization. When entered, both systems had/have a series of embedded checking programs to look for obvious errors. Formational names are subsequently assigned, according to the accepted geological interpretation and position within the stratigraphic column.

Once the assay data have been received from the laboratory in Ulaanbaatar, the digital assay results are also imported to the Acquire database. None of the assay data are manually entered.

Final surveyed collars (total station) are entered manually into the database and visually checked against the preliminary, hand-held GPS readings. No double data entry is applied during the entry of the final collar coordinates.

IMMI (now OT LLC) presents the data in up-to-date 50 to 100 m spaced MapInfo drill sections in two directions (north-west and north-east) and these are reconciled to 50 m spaced level plans to ensure that domains (solids) are properly constructed and interpretations are sound. Tight control is maintained through up-to-date sections and levels, which are reviewed on a regular basis to ensure that all holes have crossed the target where planned and that sufficient data density exists to make an appropriate interpretation. The solids of all lithologies and mineralization types are present on the interpretations and if significant deviations in the holes are noted or they appear to miss their targets, additional holes are added to fill in un-tested areas in the model. Ivanhoe's ultimate objective has been to develop a comprehensive and coherent geological (geometric) model based on sound and accurate geological information, as the basis for resource estimates.

The various databases, which form the basis of the resource estimates, appear to be well-organized, transparent, and duplicated in several locations off site for security purposes. This is crucial as part of any audit trail.

10.2.8 Exploration RC Drilling and Combined RC/Core Drilling

The exploration RC drilling program was carried out during 2001 to 2002 at the Southern Oyu Deposits. Drilling was completed using a truck mounted, center-sample return, triple-wall system UDR-1000 combination drill, which was owned and operated by Major Drilling Mongolia LLC.

None of the RC drilling completed during the 2001 and 2002 programs are used for resource estimation.

A total of 32,057 m (approximately 3% of all drilling) has been drilled in 484 RC holes on the Project to date. Most of this drilling occurred during the Ivanhoe 2001-2002 exploration programs and more recently for drill testing regional prospects

(e.g. Ulaan Khud and East Side Licence). Also included in this total are an additional 194 holes (10,191 m) that were drilled mostly for condemnation and water drilling in various parts of the Project.

Reverse circulation drilling methods have also been used as pre-collars for 23 combined RC/core holes (see Table 10.1). These holes comprise 2,941 m of RC and 11,184 m of core drilling for a total of 14,125 m.

10.2.9 Collar Surveys

Refer to Section 10.2.2 for a description of the collar survey practices and results.

10.2.10 Down Hole Surveys

None of the RC drillholes have down hole survey data and the holes are assumed to be without deviation from the collar survey. In general, most RC holes are not more 100 m in depth and therefore are not expected to experience excessive deviations. RC drillholes are not considered in this current resource estimate; however, RC holes are incorporated into the Southern Oyu estimates (Section 14).

A variety of methods were used to collect deviation measurements of the azimuth and dip for most of the combined RC/core drillholes.

10.2.11 Logging

Logging procedures for combined RC Core drilling are as per core drilling.

10.2.12 Recoveries

No recovery data are available for the historical RC programs. As discussed previously, the historical RC drill data are not used for resource estimation.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Split core samples are prepared for analysis at the on-site sample preparation facility operated by SGS Mongolia. The prepared pulps are then shipped by air under the custody of Ivanhoe to Ulaanbaatar, where they are assayed at a laboratory facility operated by SGS Mongolia.

The facility is well-equipped and the staff well-trained by SGS Mongolia. All sample preparation procedures and QA/QC protocols were established by IMMI (now OT LLC) in consultation with SGS Mongolia. The facility currently processes approximately 100–200 samples per month. The maximum sample preparation capacity has been demonstrated to be around 600 samples per day when fully staffed.

The facility has one large drying oven, two Terminator jaw crushers, and two LM2 pulverisers. The crushers and pulverizers have forced air extraction and compressed air for cleaning. All equipment appeared to be in good working order.

The samples are initially assembled into groups of 15 or 16 samples, and then 4 or 5 quality control samples are interspersed to make up a batch of 20 samples. The quality control samples comprise one duplicate split core sample, one uncrushed field blank, a reject or pulp preparation duplicate, and one or two standard reference material (SRM) samples (one <2% Cu and one >2% Cu if higher-grade mineralization is present based on visual estimates). The two copper SRMs are necessary because SGS Mongolia uses a different analytical protocol to assay all samples >2% Cu. The split core, reject, and pulp duplicates are used to monitor precision at the various stages of sample preparation. The field blank can indicate sample contamination or sample mix-ups, and the SRM is used to monitor accuracy of the assay results.

The SRMs are prepared from material of varying matrices and grades to formulate bulk homogenous material. Ten samples of this material are then sent to each of at least seven international testing laboratories. The resulting assay data are analyzed statistically to determine a representative mean value and standard deviation necessary for setting acceptance/rejection tolerance limits. Blank samples are also subjected to a round-robin program to ensure the material is barren of any of the grade elements before they are used for monitoring contamination.

The sample preparation protocol for Ivanhoe samples is given as follows:

- Coding: an internal laboratory code is assigned to each sample at reception.
- Drying: the samples are dried at 75°C for up to 24 hours.
- Crushing: the entire sample is crushed to obtain nominal 90% at 3.35 mm.
- Splitting: the sample passes twice through a nominal 1 inch (approximately 2.5 cm) Jones Splitter, reducing the sample to approximately 1 kg. The coarse reject is stored.
- Pulverization: the sample is pulverized for approximately 5 minutes to achieve nominal 90% at 75 µm (200 mesh). A 150 g sample is collected from the pulveriser and sealed in a Kraft envelope. The pulp rejects are stored on site.

- The pulps are put back into the custody of Ivanhoe personnel and SRM control samples are inserted as required.
- Shipping: the pulps are stored in a core box and locked and sealed with “tamper-proof” tags. Sample shipment details are provided to the assaying facility both electronically and as paper hard copy accompanying each shipment. The box is shipped by air to Ulaanbaatar where it is picked up by SGS personnel and taken to the analytical laboratory. SGS confirms to Ivanhoe staff by electronic transmission that the seal on the box is original and has not been tampered with.
- Storing and submitting: The pulp rejects are stored on site at the laboratory for several months and then returned to Ivanhoe in Ulaanbaatar for storage.

All equipment is flushed with barren material and blasted with compressed air between each sample that is processed. Screen tests are done on crushed and pulverized material from one sample taken from the processed samples that comprise part of each final batch of 20 samples to ensure that sample preparation specifications are being met.

Reject samples are stored in plastic bags inside the original cloth sample bags and are placed in bins on pallets and stored at site. Duplicate pulp samples are stored at site in the same manner as reject samples.

11.1.1 Review of the On-Site Preparation Laboratory

QG has performed four inspections of SGS Mongolia sample preparation facility on site. These visits do not constitute audits. A more comprehensive review was undertaken by Dale A. Sketchley (Ivanhoe Mines) - Sketchley, 2011). IMMI (now OT LLC) retained an independent geologist/geochemist, Barry Smee, to conduct audits of preparation and analytical facilities.

During the most recent visit to the facility, which was in the process of being moved there were a number of minor comments made to the lab manager for correction related to contamination. The only two points worthy of comment were:

- The catch trays from the Terminator crushers are supposed to be used with the riffle splitters as they are sized correctly; however, the larger riffle splitter catch trays were being used instead. The riffle splitter catch trays should be re-designed so they are the same as the riffle splitter hopper.
- The riffle splitters are coming to the end of their useful life. They have been welded a number of times now and there is potential for contamination on the splitters. QG recommends they are replaced.

When complete, QG recommends the new preparation facility be properly audited.

The security measures in place for shipment of pulps to the SGS analytical laboratory in Ulaanbaatar were reviewed (including observing the dispatch of samples) and are considered by Scott Jackson to be appropriate and adequate.

11.2 Analysis

11.2.1 SGS Mongolia

All samples are routinely assayed by SGS Mongolia for gold, copper, and molybdenum. Gold is determined using a 30 g fire assay fusion, cupelled to obtain a bead, and digested with Aqua Regia, followed by an AAS finish, with a detection limit of 0.01 g/t. Prior to 2011, copper and molybdenum were determined by four acid digestion of a 2 g subsample, followed by an AAS finish. Samples are digested with nitric, hydrochloric, hydrofluoric, and perchloric acids to dryness before being leached with hydrochloric acid to dissolve soluble salts and made to volume with distilled water. The detection limits of the copper and molybdenum are 0.001% and 10 ppm, respectively. The same acid digestion is also used for analyses of Ag and As with detection limits of 1 ppm and 100 ppm. In 2011, following concerns related to lower level precision of Ag, As and Mo assaying, OT LLC decided to switch methods for Cu, Mo, Ag, As and a suite of other elements to ICP-OES/MS.

Ivanhoe also conducts a Trace Elements Composites (TEC) program, in addition to routine copper gold and molybdenum analyses. Ten metre composites of equal weight are made up from routine sample pulp reject material. The composites are subject to multi-element analyses comprising a suite of 48 elements determined by ICP-OES/MS after four acid digestions. Additional element analyses include mercury by Cold Vapour AAS, fluorine by KOH Fusion/Specific Ion Electrode, and carbon/sulfur by Leco furnace. Ivanhoe uses results from the TEC program for deleterious element modeling.

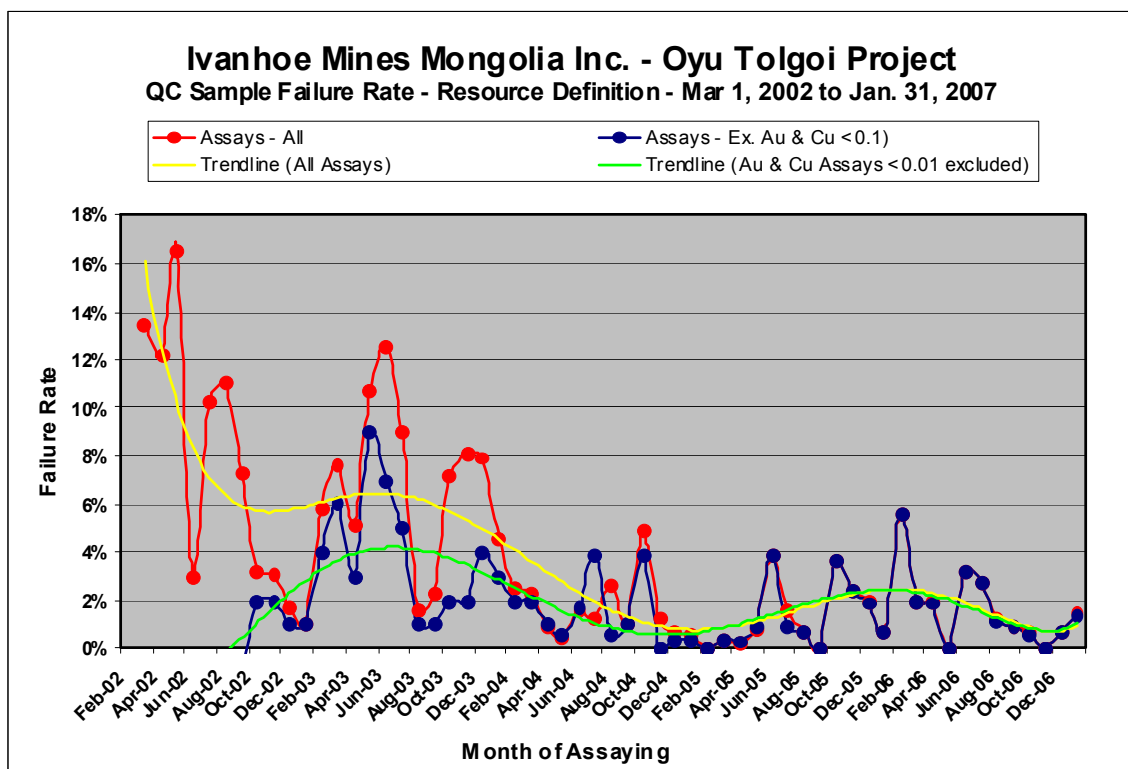
According to the most recent audit in 2008 (Smee 2008), the analytical laboratory is using procedures and equipment that are consistent with industry "Best Practices" and therefore can be used for resource estimating purposes. SGS Mongolia reports the results digitally to Ivanhoe via email and submits signed paper certificates. General turnaround is approximately seven days. All hard copy certificates are stored in a well-organized manner in a secure location on site.

11.3 QA/QC Program

IMMI (now OT LLC) strictly monitors the performance of the SRM samples as the assay results arrive at site. The ability of the laboratories to return assay values in the prescribed SRM ranges has steadily improved to greater than 99%. All samples are given a "fail" flag as a default entry in the Project database. Each sample is re-assigned a date-based "pass" flag when assays have passed acceptance criteria. The failure rate chart is presented in Figure 11.1 a selection of SRM charts in Figure 11.2.

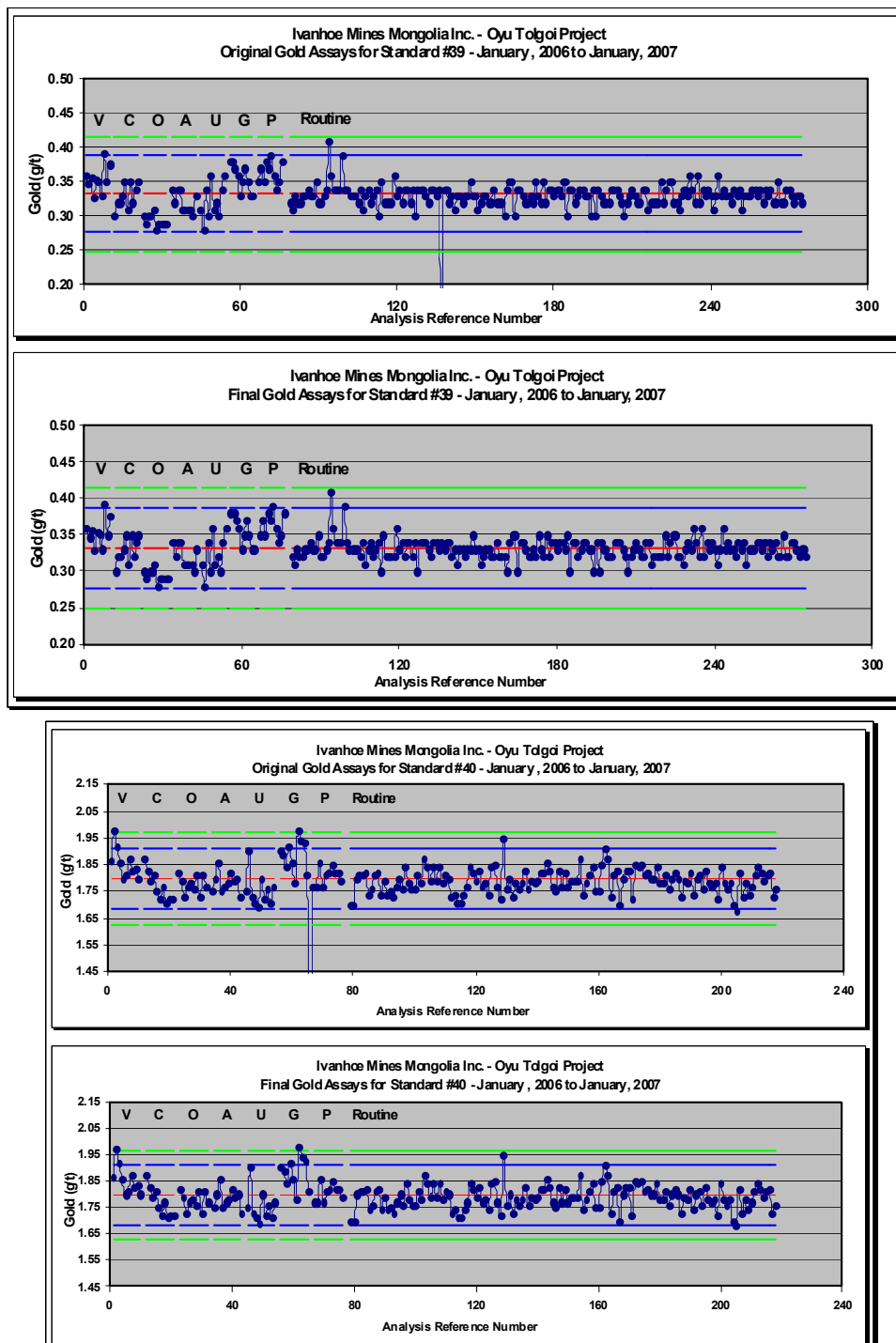
All QA/QC is now monitored by OT LLC through the Acquire database and the system is currently being set up for reporting the various results. Results from the drilling over the period that impacts the resource estimate are provided below.

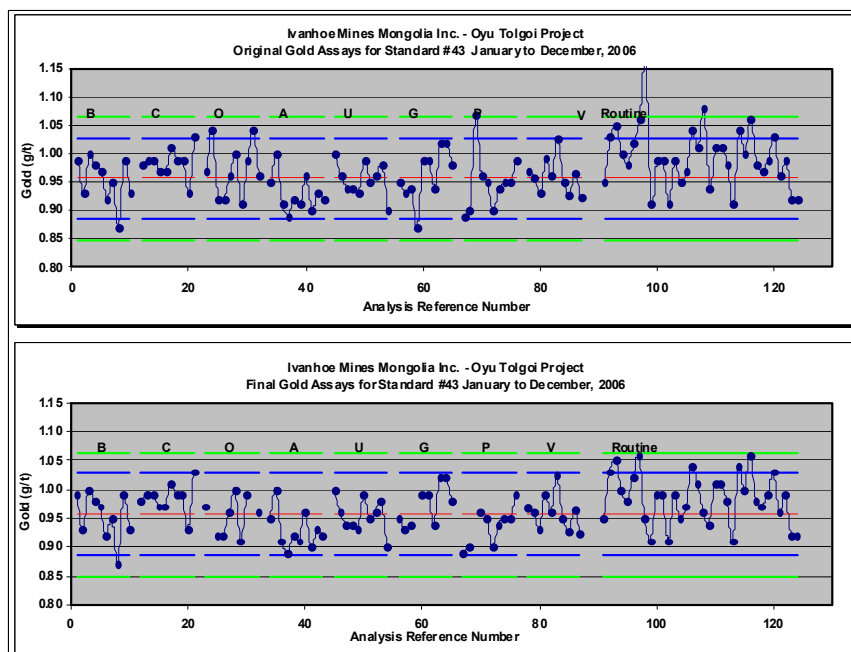
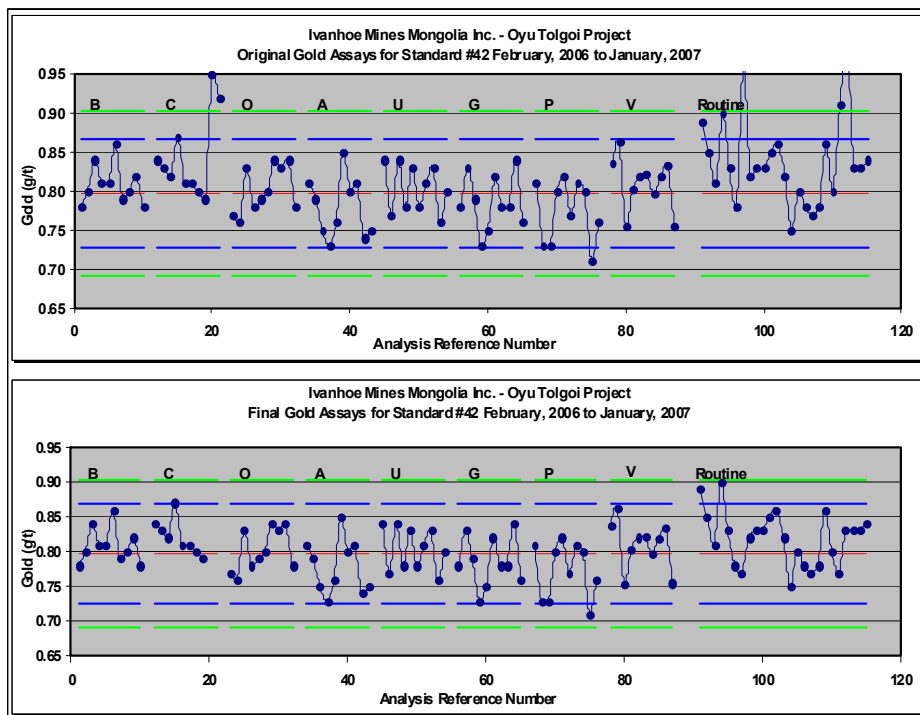
Figure 11.1 Quality Control Sample Failure Rate, 2002 to 2007



Analytical results shown on each of the QC monitoring charts (Figure 11.2 and Figure 11.3) have been divided into two parts with subtitles, “RR Assays” and “Routine Assays”. The subtitle “RR Assays” denotes analytical results obtained from each of the participating laboratories that were used to determine the mean and +2SD/3SD tolerance limits marked on each chart. The subtitle “Routine Assays” denotes analytical results obtained from SGS-UB, the lead analytical laboratory.

Figure 11.2 A to E SRM Example Charts – Gold Original and Final – 2006 and 2007





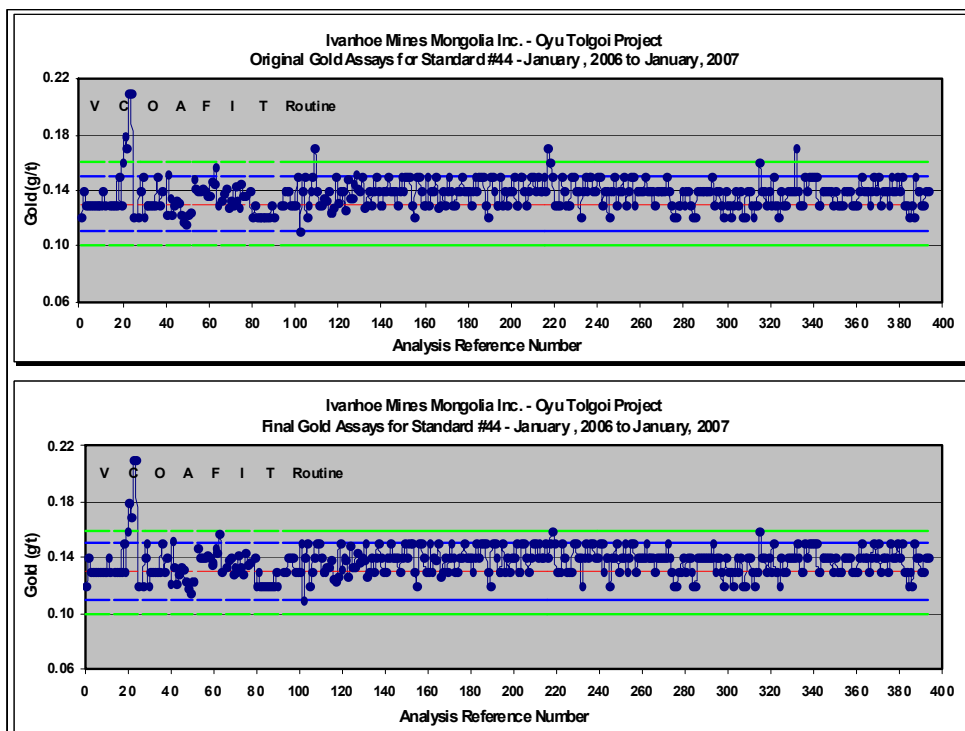
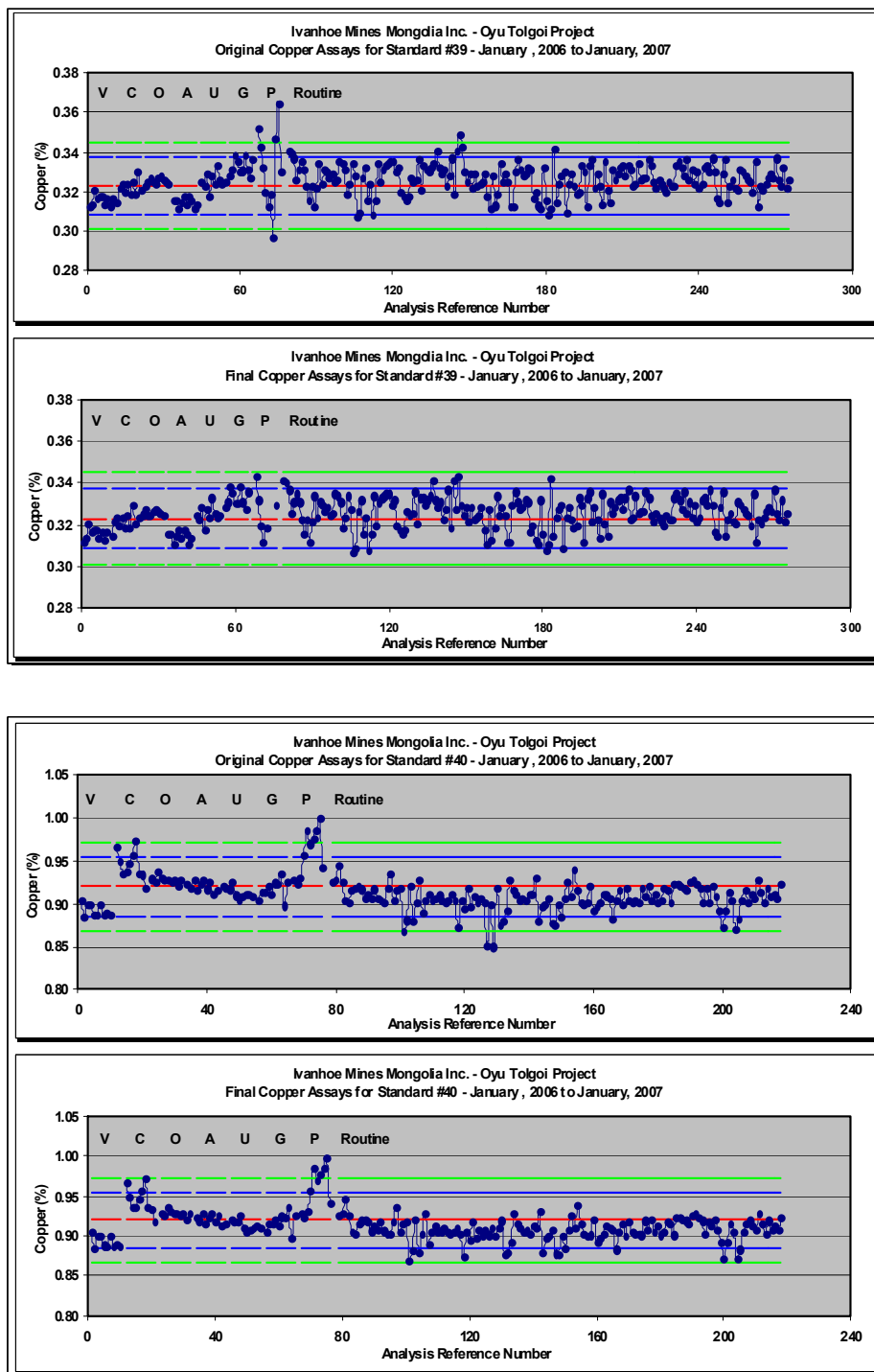
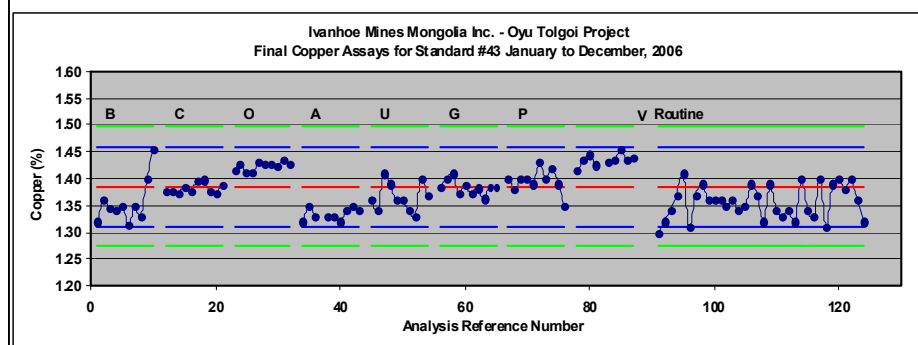
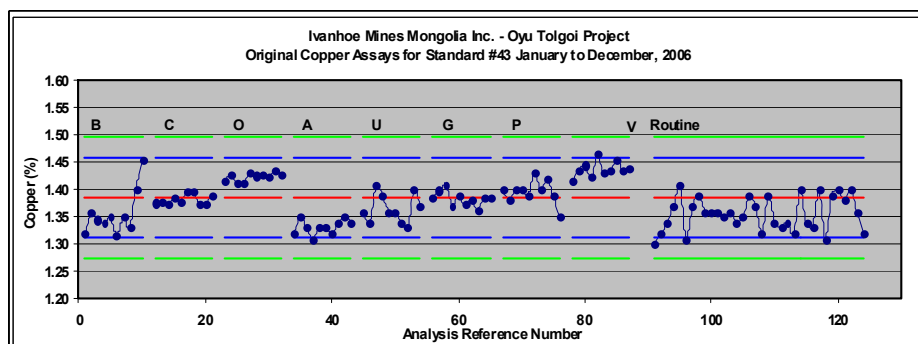
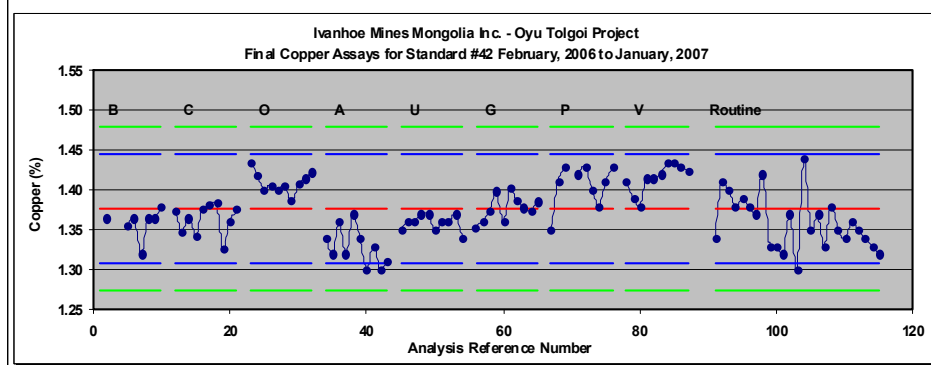
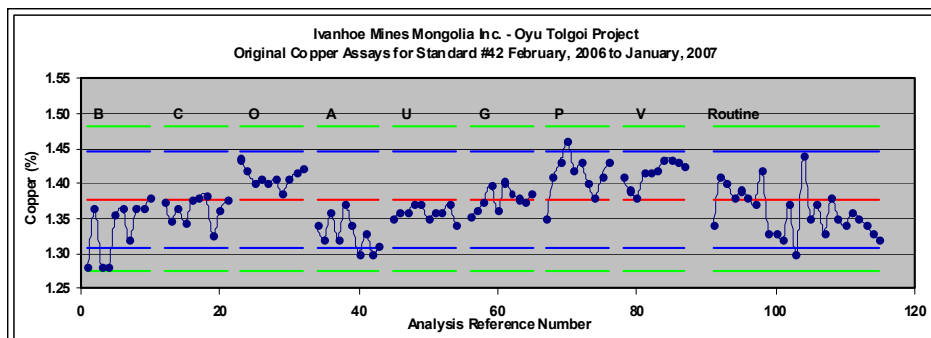
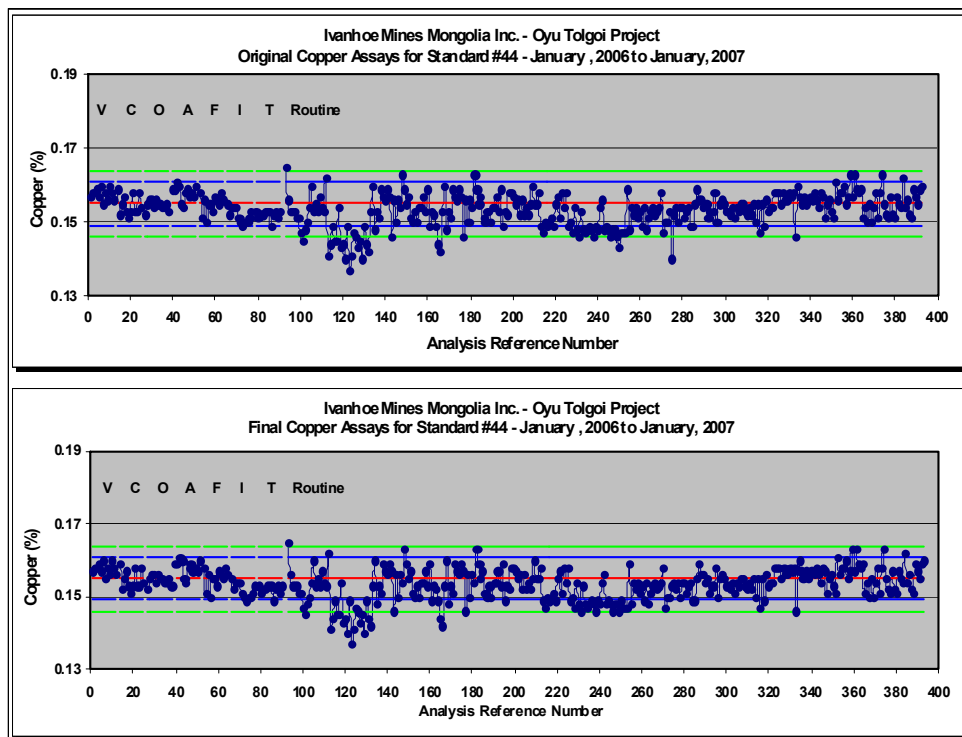


Figure 11.3 A to E SRM Example Charts – Copper Original and Final – 2006 and 2007







11.3.1 Blank Sample Performance

Figure 11.4 and Figure 11.5 show typical assay performance of field blanks for gold and copper. In these figures, the lower blue horizontal line represents the analytical detection limit (ADL) of the respective metal, and the upper yellow horizontal line represents the analytical rejection threshold (ART). The gold ADL is 0.01 g/t with an ART of 0.06 g/t; copper ADL was initially 0.01% and is now 0.001% with an ART of 0.06%. The results show a low incidence of contamination and a few cases of sample mix-ups, which were investigated at site and corrected.

Figure 11.4 Field Blank Assay Performance – Gold

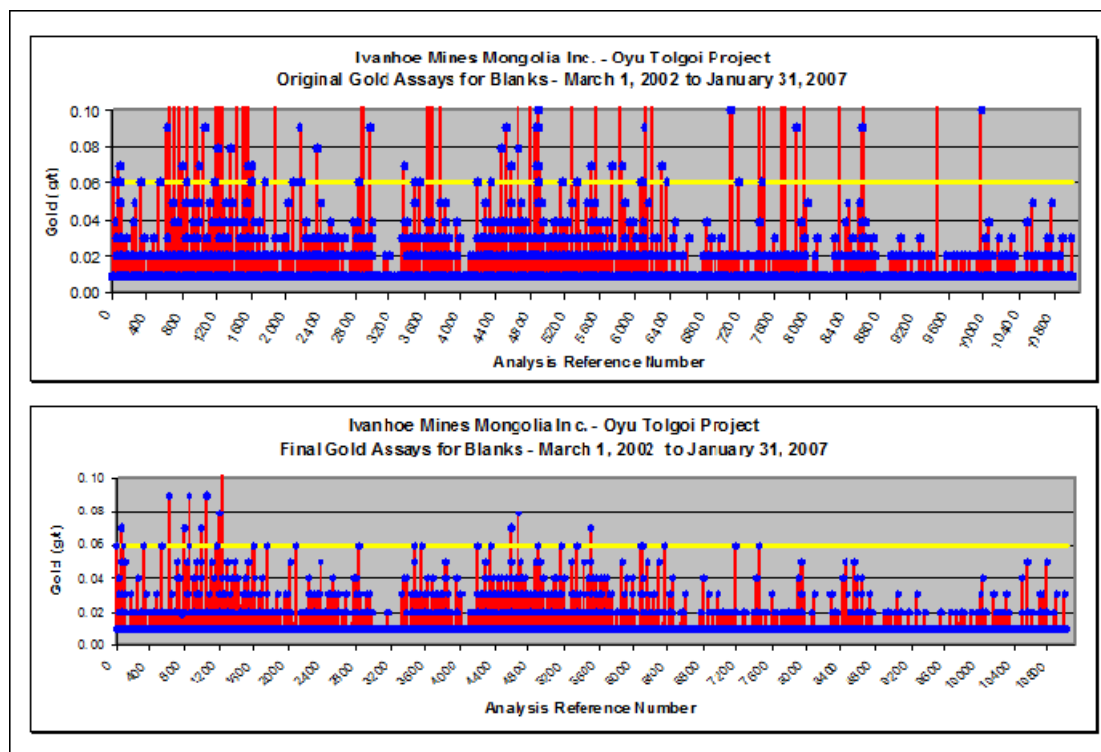
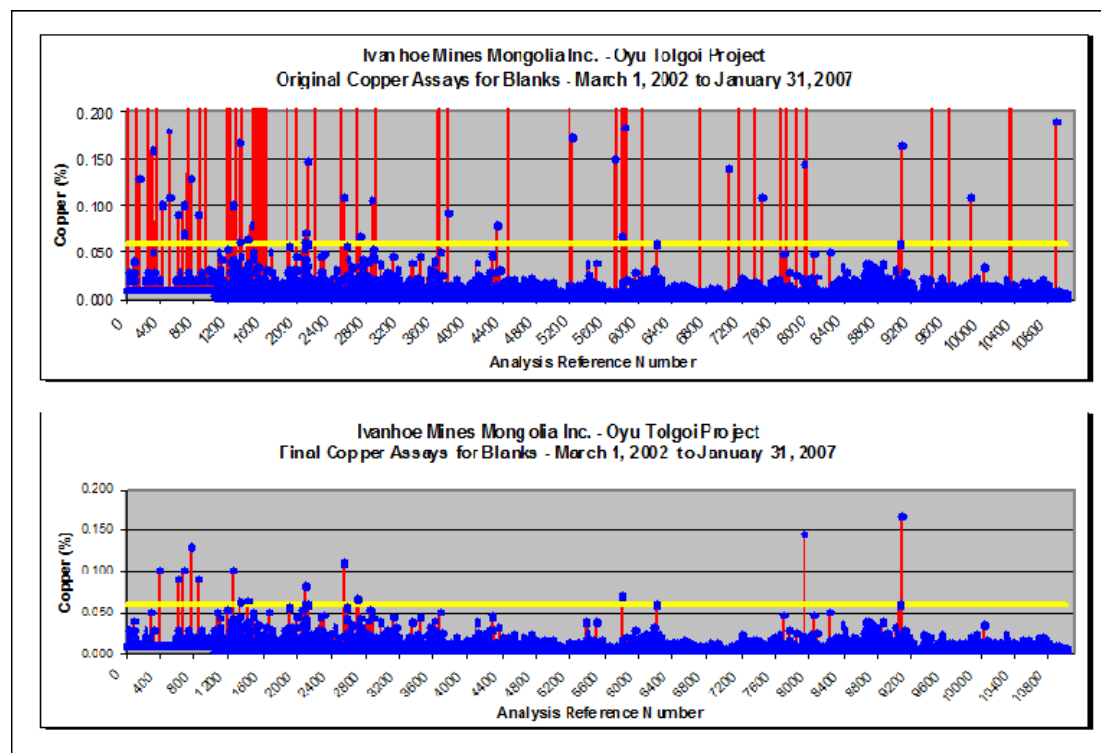


Figure 11.5 Field Blank Assay Performance – Copper



11.3.2 Duplicate Sample Performance

The QA/QC program currently uses three different types of duplicate samples: core, coarse reject, and pulp; laboratory check pulp samples sent to an umpire laboratory were only used up to the end of 2005 for the Southern Oyu and Hugo Dummett deposits.

Table 11.1 summarizes the results of the Percentile Rank statistical analyses for each type of duplicate sample with charts of the results for Hugo North shown in Figure 11.6 and Figure 11.7. Table 11.1 summarizes the results previously published by Juras 2005 and Cinits 2007.

Table 11.1 Results of Percentile Rank Statistical Analyses

Duplicate Type	% Diff. Limit	Cu		Au	
		No.	% Diff.	No.	% Diff.
Southwest, South and Wedge Deposits					
Core	—	2,898	38	2,771	60
Coarse Reject	20	1,371	14	1,095	25
Pulp	10	1,368	5	1,092	18
Central Deposit					
Core	—	2,898	38	2,771	60
Coarse Reject	20	1,371	14	1,095	25
Pulp	10	1,368	5	1,092	18
Hugo North Deposit (>0.2 g/t)					
Core	—	1,969	40	848	44
Coarse Reject	20	996	5	445	17
Pulp	10	934	4	434	15

For the 90th percentile of the population, a maximum difference of 10% is recommended for the pulp duplicates and 20% for the coarse reject duplicates because these duplicate types can be controlled by the subsampling protocol. The same criteria do not apply to core duplicates because these differences cannot be controlled by the subsampling protocol.

Table 11.1 shows that copper generally performs very well with results well within expected limits, gold results are higher than copper but considered reasonable.

Figure 11.6 Gold Duplicate Samples

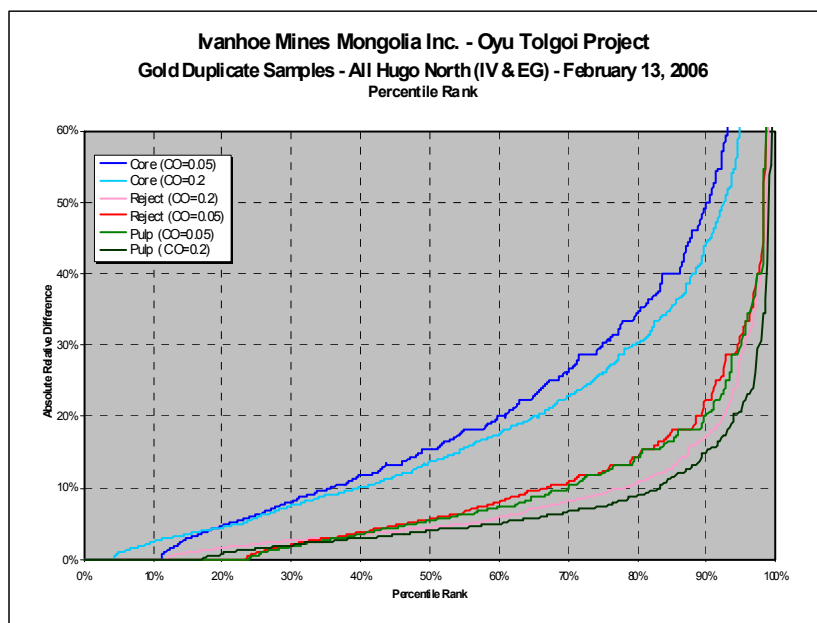
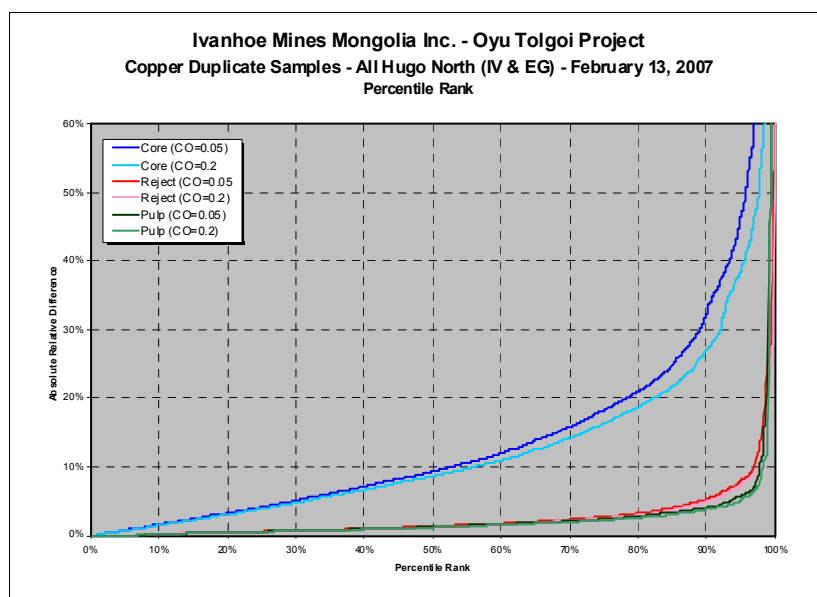


Figure 11.7 Copper Duplicate Samples



11.3.3 Check Assay Program

For most of the drill programs, Ivanhoe has maintained a check assay program sending approximately 5% of assayed pulps to external (secondary) laboratories. SGS Welshpool in Perth was designated as the secondary laboratory until May 2005 (drillhole OTD900), when Ivanhoe switched to Genalysis in Perth. Mr. Smee's January 2006 audit recommended that the check assay program be stopped as long as the internal Oyu Tolgoi QA/QC program continued to produce accurate and precise assay results.

11.4 QG Comments on Sampling and QA/QC

John Vann (QG) spent time with Dale A. Sketchley (Ivanhoe Manager QA/QC Advanced Projects), who presented the current systems of quality management to QG. These systems are also summarized in the following article (Sketchley, D.A., and Forster, C.F., 2007, QA-QC at Oyu Tolgoi Cu-Au Project, South Gobi, Mongolia: Meeting and Exceeding Technical Disclosure Requirements, CIM Magazine, February 2007, Vol. 2, No. 9).

All sampling, assaying, and QA/QC work up to 2010 at Oyu Tolgoi was overseen by Dale A. Sketchley, assisted by Ariunaa Tuvshintsengel (QA/QC Data Manager, Ivanhoe).

QG focussed its attention on the elements being estimated in the resource, i.e. copper, gold, molybdenum and silver, and make the following specific comments and recommendations:

- QG confirms that the steps described in Section 11.2 of this report are continuing on the Project.
- The general level of diligence and supervision of sample preparation and analytical quality control is good.
- The frequency of insertion of SRM, blanks and pulp duplicates is considered by QG to be sufficient to manage the quality of data.
- There are some biases evident in many of the standard run-charts, mostly minor, and in many instances, reversing over time (thus not contributing to global bias).

QG considers that the maintenance of a check assay program is good practice. QG was advised by OT LLC that a check assaying procedure was reinstituted in early 2007, but suspended again during 2009 to 2011 when drilling was almost stopped.

The duplicates for assay pulps and coarse rejects for OT are performing satisfactorily.

Scott Jackson concludes that IMMI's (now OT LLC) past and current sample preparation, analytical and QA/QC procedures, as well as the security measures in place, are adequate and appropriate and consider that the data for the Project are acceptable as inputs to resource estimation.

11.5 References

The references for this section can be found in the report Oyu Tolgoi Technical Report, March 2008, with the exception of the following additional references:

Smee, B., 2008. A Review of Quality Control Data and Commercial Laboratories used for the Heruga Deposit Resource Calculation, Mongolia. Consultants report prepared for OT LLC by Smee and Associates Consulting Ltd (dated April 2008).

12 DATA VERIFICATION

12.1 Property Visits – Oyu Tolgoi and Shivee Tolgoi

Scott Jackson completed four site visits in September 2008. The site visits usually entailed the following activities.

- Overview of the geology and exploration history of the Project with site geologists.
- A complete review of geological interpretation on paper sections and plans as well as independent examination of the same sections and plans.
- Limited review of sections via the computer screen.
- Review of existing drilling data, including brief checking of drillhole orientation, depth, number of holes, etc.
- Update on changes in sample collection procedures.
- Rapid review of core recovery for the Oyu Tolgoi drill data.
- Collection of required data to enable checking of QA/QC measures including standard reference materials, blanks, duplicates, etc. Discussions were often held with Dale A. Sketchley (and later Keenan Jennings) on these issues. Initial checking of collected data indicated no material errors.
- Update on bulk density sampling procedures.
- Review of diamond drill core, core logs, core recovery, collar survey, down hole surveys, core photography for ten selected holes.
- Prior to 2012, checks ensuring that the paper logs matched the logging in the database.
- Review of construction of geological wireframes used as the basis of estimation domains (reviewed with Cyrill Orssich, Senior Geologist, Ivanhoe Mines, on screen).

12.1.1 Oyu Tolgoi, Heruga Shivee Core Review

Over four site visits Scott Jackson has examined in detail almost 30 drillholes from Shivee Tolgoi through to Heruga. Drillholes were chosen from cross-sections and plans by QG and the core examined to establish if:

- The geological descriptions logged in the database broadly concur with the sighted core.
- Specifically, important contacts, structures etc. are independently confirmed.
- To obtain a better understanding of geology/mineralization allowing QG to make comment on validity (or otherwise) of geological domains.
- To examine the condition of core.

From the reviews, Scott Jackson concludes that the quality of logging of important features in the core is generally of a high standard, is considered adequate and appropriate and is sufficient as a basis for resource estimation to be reported under the auspices of NI 43-101.

12.2 Data Verification

In September 2008, QG collected 15 core samples. The intent of these samples was to subject them to independent analysis in order to confirm the general range of gold and copper grades, especially the high copper and gold values, reported in the Oyu Tolgoi database. A similar exercise was carried out previously by AMEC and is reported in the March 2008 Ivanhoe Technical Report.

QG consultants selected the intervals, collected the core and, where applicable, specified cutting required for quarter core sampling. All core was digitally photographed pre-collection and post-collection. Scott Jackson then bagged and numbered samples. Immediately after collection the sample bags were sealed with security zip-tags, placed into two sealed plastic drums on site at Oyu Tolgoi. Due to logistical difficulties and the time needed to get sample export approval from the GOM, it was not feasible for QG to carry the samples out of the country. Instead, Ivanhoe Mines agreed to organize air transport of the sealed drums to Ulaanbaatar, then courier shipping of the samples (via an international courier service) from Ulaanbaatar to QG in Perth. Upon their arrival in Perth, the seals were checked.

These samples were analysed by a reputable laboratory in Perth and the results are presented in Table 12.1.

Table 12.1 Comparison Between Original Ivanhoe Assays and Independently Sampled Assays Collected by QG

Area	Hole No.	Sampled Interval (m)	Ivanhoe Au (ppm)	QG Au (ppm)	Ivanhoe Cu (%)	QG Cu (%)
SW Oyu	OTD 185	602-604	2.45	1.89	0.79	0.61
SW Oyu	OTD 288	304-306	2.00	1.90	0.55	0.54
SW Oyu	OTD 288	420-422	1.81	1.77	1.35	1.21
SW Oyu	OTD 292	242-244	2.16	2.02	0.08	0.69
SW Oyu	OTD 755	204-206	1.65	1.49	0.77	0.72
Hugo South	OTD 653	518-520	0.03	0.03	2.59	2.69
Hugo South	OTD 653	620-622	0.13	0.10	2.57	2.62
Hugo South	OTD 470	202-204	0.03	0.06	1.05	1.31
Hugo South	OTD 470	472-474	0.07	0.14	2.72	3.02
Hugo North	OTD 576C	1,026-1,028	3.45	2.09	5.25	5.45
Hugo North	OTD 576C	1,078-1,080	0.52	0.68	4.23	4.39
Hugo North	OTD 576C	1,112-1,114	0.56	0.53	4.53	4.48
Shivee Tolgoi	EGD006A	1,240-1,241	1.87	1.49	5.27	4.84
Shivee Tolgoi	EGD006A	1,464-1,466	2.99	3.23	5.42	5.18
Shivee Tolgoi	EGD006A	1,564-1,566	0.42	0.40	1.41	1.44

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

The Project under consideration consists of three main feed sources:

- Southwest
- Central
- Hugo North

The Mineral Reserve considers Southwest and Central ores will be extracted by open cut mining. IDP05 which was a Preliminary Assessment Report considered Hugo North mining by block caving.

The Southwest deposit is a gold-rich porphyry system characterized by pipe-like geometry approximately 250 m in diameter and extending over 700 m vertically. The copper mineralization is dominated by chalcopyrite, with minor bornite (less than 20%).

The South deposit outcrops as copper oxides underlain by secondary sulfides and then chalcopyrite. The copper grade is lower than in the other deposits, and there is little gold. Some of the South deposit will fall within the bounds of the Southwest pit. For this reason, both the South and Southwest ores are hereafter referred to as Southwest.

The Central deposit is “funnel” shaped, with a chalcocite enrichment blanket overlying a large covellite zone. A chalcopyrite/gold zone lies at the base of the covellite zone, although this is expected to contribute less than 5% of the material mined from the Central deposit.

Hugo North dips away and has very high copper grades at a depth of 1,000 m or more. Gold appears to increase with depth, with some coarse, visible particles observed in the drill core. Copper mineralization consists of chalcopyrite, bornite, and chalcocite.

In 2001, OT LLC initiated and supervised a program of work to investigate the metallurgical response of samples of drill core from the Southwest, Central, and Hugo Dummett deposits. Testwork carried out by SGS Lakefield Research Limited (SGS Lakefield), A.R. MacPherson Consultants Ltd. (ARM), and Terra Mineralogical Services established basic comminution parameters, the amenability of gold recovery by gravity concentration, and the amenability of copper and gold recovery by flotation. Limited testwork was also conducted on samples from the South deposit.

In 2003, the then study managers, AMEC Ausenco Joint Venture (AAJV), together with OT LLC, determined that an additional phase of metallurgical testwork was necessary to support a feasibility study. The 2004 to 2005 program was designed to establish the flotation and comminution response of ores from Southwest, Central, Hugo South, and Hugo North. Laboratory batch-scale and pilot-plant flotation testwork programs were conducted at AMMTEC Ltd. (AMMTEC) in Perth. Additional testwork to define fundamental flotation and comminution parameters was executed by MinnovEX. Laboratory-scale comminution testwork programs were also conducted at AMMTEC. SGS Lakefield carried out a SAG pilot-plant test program to confirm the laboratory-scale

testwork. Laboratory batch-scale and pilot-plant flotation testwork programs were conducted at AMMTEC Ltd. (AMMTEC) in Perth. Additional testwork to define fundamental flotation and comminution parameters was executed by MinnovEX. Laboratory-scale comminution testwork programs were also conducted at AMMTEC. SGS Lakefield carried out a SAG pilot-plant test program to confirm the laboratory-scale testwork.

In 2006 and 2007, SGS (MinnovEX) conducted confirmatory metallurgical cleaner testwork on the Southwest and Central deposits. During the same period, core became available from additional drilling of the Hugo North Sub-Level Cave (SLC) and Entrée deposits, and Process Research Associates (PRA) was contracted to conduct the metallurgical testwork of these new reserves.

13.2 Sample Representation

The main source of IDP05 feed samples for the metallurgical testwork program was diamond drill core from the resource drilling program.

Samples were taken from the Southwest, Central, Hugo South, and Hugo North deposits. The samples collected from Southwest and Central were based on 100 m x 100 m x 45 m vertical height “metallurgical blocks” generated from the 20 m x 20 m x 15 m high resource model blocks that fell within the NSR \$2.81 pit shell during the first 10 years of production (based on the current mining schedule, at 9 June 2004). Samples were selected from metallurgical blocks lying more than 25% within the NSR shell. Each metallurgical block was defined by a simple three-coordinate system of Level [L], Row [R], and Column [C].

Quarter core was taken from drillhole intercepts that passed through the blocks. If more than one hole passed through a block, then the intercept with the grade closest to that of the average grade of the metallurgical block was chosen. If no hole passed through the block, then the hole in an adjacent block closest to the resource model block grade was chosen.

Samples from Hugo South and Hugo North were taken from the holes that passed through the block-cave envelopes identified in the mining schedule at 9 June 2004. The block level [L] designation is the same for all deposits at Oyu Tolgoi. Therefore, nominal mining blocks were superimposed at each level to delineate the block-cave mining areas. Quarter core was taken from each 2 m geological assay interval along the core from the point where it entered each metallurgical block to the point where it left. These samples were kept in a separate bag until compositing so that location and geological assay data could be referenced. If all of the 2 m sample intervals that constituted the upper, middle, or lower vertical 15 m of a metallurgical block were outside the NSR shell, then they were rejected as barren material. Otherwise, the 2 m samples were added to the hole composite that represented the metallurgical block.

Because the Hugo North orebody is very deep and a relatively small number of holes were available at the time of the sampling program, every available drillhole that could be sampled was sampled. With the non-selective nature of block-cave mining, even obviously barren dyke material, which would be part of the internal dilution, was sampled and included in composite preparation.

The Hugo North, Hugo South, and Central deposits have not been sampled to their limits but only to the limits of zones corresponding to the early years of production, e.g. the end of Year 7 in the case of Hugo South.

13.2.1 Comminution

(1) Origin of Core Samples for 2003 – 2005 Grindability Testwork – IDP05

The IDP05 comminution samples represented the complete set of samples used for the current 2009 comminution plant design. Samples tested post 2005 are used to populate the geostatistical block model to determine the processing rate provided in the IDOP Reserve Case production schedule given a fixed plant design.

The core samples delivered for grindability testwork over the 2003 to 2005 period covered Southwest, Hugo North, Central and Hugo South.

Table 13.1 lists the number of samples taken for comminution testwork per deposit.

Table 13.1 Number of Samples taken per Deposit for 2003-2005 Testwork(Samples for Comminution Testwork from All Deposits)

	Years of Scheduled Production*	Production First Ten Years* (Mt)	No. of Met Blocks	No. of Holes in Met Blocks	No. of Holes Sampled	No. of Comminution Samples Collected	No. of Samples Tested
Southwest	0-15	231	424	363	106	198	169
Central	6-14	30	–	51	45	56	81
Hugo North	4-20+	71	–	–	28	97	36
Hugo South	9-20+	4	–	71	71	81	10

*Based on the IDP05 Production Schedule.

For Southwest, a comminution sample was taken for every metallurgical block representing the first three years of mine production. From then on, samples were selected to provide an adequate sample distribution for a given level. For all comminution samples, quarter core was taken from core diameters greater than 55 mm, and half core for core diameters less than 55 mm. Most samples taken were in the weight range 5.0 to 5.5 kg but, for approximately every tenth sample, a larger sample of two 6.0–6.5 kg samples was taken and combined to make up a nominally 12 kg sample on which a full Bond Work Index test could be performed.

The purpose of the MinnovEX testwork was for nominal throughput determination based on the tonnage calculated for each metallurgical block represented by a drill core sample. To achieve this objective, Ci (Crusher Index); SPI (SAG Power Index); BWi and Mod-Bond (Bond Work Index and Mod-Bond Work Index) tests were carried out. Table 13.2 provides a list the number of SPI, Ci, Mod-Bond and full BWi tests performed per deposit, between 2003 and 2005.

Table 13.2 Number of Comminution Tests Performed at MinnovEX per Deposit

Deposit/Sample	SPI Tests	Ci Tests	Mod-Bond Tests	BWi Tests	Production First Ten years (Mt)	SPI Representation (per Mt)
Southwest	169	124	167	31	231	0.68
Hugo South	10		10	3	4	2.50
Hugo North	36	36	36	2	71	0.51
Central	81	18	78	5	30	2.70
Entree	9	9	9	–	–	–
Composite	11	11	–	–	–	–
Unallocated	7	7	–	7	–	–
Total	350	206	319	48	336	–

In addition to the above comminution tests, 26 samples were tested at AMMTEC, for the CEET-JK model comparison study, and 4 samples of Heruga were tested for BWi at G&T Metallurgical.

(2) Origin of Bulk Sample for 2005 Pilot Campaign

An exploratory hole (OTD725) was drilled in the Southwest orebody to intercept a cross-section of typical plant feed types, guided by geological information gained from the previous exploration program. Down hole grades and SPI measurements were taken. An interval from 68 to 72 m down hole was selected based on SPI, lithology, and grade being near the orebody average.

A 3 m diameter shaft was sunk in the vicinity of this hole, at UTM coordinates 650718E, 4763089N, centred on previously drilled diamond drillhole OTD189. A 250 t bulk sample was taken from the sides of the bottom 6 m of this shaft at a depth 68 to 74 m below surface. Care was taken to blast minimally to ensure that the material taken from the shaft was not over-fractured. The sample was shipped to SGS Lakefield for pilot testing.

A 250 t bulk sample was shipped to SGS Lakefield for the 2005 pilot plant campaign. A subsample of the pilot plant SAG feed material was submitted to MinnovEX for SPI and Mod-Bond measurement. A pilot plant campaign was conducted to confirm the suitability of Oyu Tolgoi Southwest material for semi-autogenous grinding and to validate the CEET Model.

(3) CEET Validation with JKSimmet

The 26 samples were selected in order to cover each of the significant feed types throughout the deposits. Sample mass and top size requirements dictated that PQ drill core was needed for these validation samples. Of the 26 samples, 22 of the samples were selected from three PQ holes through the Southwest deposit, and four samples were selected from one PQ hole in the Central deposit.

(4) Origin of Core Samples for 2006-2007 Grindability Testwork

A minimum of 80 samples from the Hugo North SLC region was required to undertake geostatistical calculations for this region; therefore additional core samples from this region were located, shipped and tested at SGS Lakefield in 2006. The 82 samples comprised 16 core samples already tested during the 2003 – 2005 program, and 66 new core samples shipped from the Oyu Tolgoi site in October 2006. Nine Entrée samples were tested in early 2007.

Sampling was based on 10 cm/m over a 15 m interval and the core sample was provided as a combination of half-NQ and half-PQ core. Most of the samples were from the bornite core and were split between the two main rock types; the remainder of the samples were from the HW argillic alteration zone and the FW disseminated Cpy mineralization in Qmd.

13.2.2 Flotation

Table 13.3 and Table 13.4 provide a summary of the flotation sample representation and testwork for the 2004, and the 2006–2007 periods, respectively. Drillhole locations are provided in the geology section.

Table 13.3 Flotation Test Representation(List of Samples for 2004 Test Program)

	Production First Ten Years* (Mt)	Composites	No. of Met Blocks	No. of Holes in Met Blocks	No. of Variability Samples
Southwest	231	Periodic Composites Months 0-6	424	363	285
		Months 7-12 Year 2 Year 3 Year 4 Year 5 Years 6-10 2006 Minnovex			7
Central	30	Chalcocite Covellite	–	–	30
		Chalcocite/Covellite 2006 Minnovex			5
Hugo North SLC	71	Upper Phase 1 2007 SLC	–	–	30 55
Hugo South	4	Phase 1	–	–	20
Entrée	–	Rougher Cleaner	–	–	6 6
Heruga	–	–	–	–	9 comp.

*Tonnages refer to the IDP05 Schedule.

Table 13.4 2007 Summary of Flotation Testwork 2006-2007 Summary of Laboratory Scale Flotation Tests

Test	SW	HN	SLC	Central	HS	Entrée	Heruga
Rougher							
Batch	149	8	–	25	10	–	4
pH/Reagent	20	–	11	–	–	–	–
% Sol.	–	–	3	–	–	–	–
P. Grind	133	45	9	41	21	6	–
FKT/MFT	7	4	6	1	1	–	–
Variability	266	47	55	41	–	–	–
Cleaner							
Batch	50	–	–	11	–	–	4
pH	3	–	9	–	–	–	–
Regrind	51	–	10	–	–	6	–
FKT	2	–	2	2	–	–	–
Column	8	–	2	–	–	–	–
Locked Cycle	21	8	3	22	3	–	–
Pilot Plant	2	1	–	–	–	–	–

(1) IDP05 2001-2003 – Early Lakefield

The early Lakefield work consists primarily of seven composites: two Southwest, one South, three Central and one Hugo North. The total drillhole representation is:

- Southwest – 3,076 m
- Hugo North – 852 m
- Central – 1,284 m

(2) IDP05 2003-2004

The 2004-2005 IDP05 work consisted primarily of period composites: seven Southwest, three Central, three Hugo North and one Hugo South. There were also two pilot plant trials: one Southwest and one Hugo North. The total drillhole representation was:

- Southwest – 17,280 m; pilot – 16,351 m
- Hugo North – 1,210 m; pilot – 1,650 m
- Central – 779 m
- Hugo South – 1,506 m

(3) 2006-2007 Southwest, Central, SLC

The confirmatory work conducted at SGS-Lakefield was on the same period composites for Southwest and Central. The Hugo North SLC work was on three composites consisting 122 m of core.

(4) Entrée and Heruga

Entrée scoping consisted of three composites with 630 m of core and Heruga scoping consisted of four composites of 80 m of core.

(5) Plant Feed Representation and Mine Planning

Note that where reference is made to blocks being treated at certain times in the production schedule (for example, Table 13.3) these references refer to the mine plans current at the time the test program was being planned. As mine plans have changed over time, so too has the relationship between the test samples and the time when the represented material is likely to be processed.

13.3 Test Programs

In late 2003, AAJV and IMMI (now OT LLC) determined that additional testwork was necessary to support the feasibility study. The objective of the new testwork was to confirm the amenability of the flow sheet adopted from the early testwork done for the Preliminary Assessment and to characterize the potential plant feed featured in the new mine plans in terms of flotation and comminution response. The testwork scope included:

- Additional bench-scale batch and locked-cycle flotation testing on Southwest and Central production composites to confirm the previous results and to test flow sheet alternatives.
- Bench-scale batch and locked-cycle flotation testing on Hugo South and North production composites to develop pre-feasibility level metallurgical parameters.
- Variability testing on Southwest samples to finalize the plant design parameters.
- Bulk flotation testwork on Southwest samples to confirm plant design parameters and to produce sufficient concentrate and tailing samples for vendor testwork and concentrate marketing.
- Bench-scale comminution testwork to develop design parameters.
- Pilot-scale comminution testwork to confirm the circuit design and throughput.

During the course of the testwork, potential risks and opportunities were to be identified and information was to be gathered to develop strategies to minimize smelter penalties.

The flotation testwork data forms the basis for all metallurgical modelling. A substantial results database exists for the Southwest ores and moderate databases exist for Central, Hugo North, and Hugo South ores.

The flotation tests summarized in Table 13.5 (approximately 1,137 tests) have been conducted since the beginning of the Oyu Tolgoi test program and the majority of them have been used in model development and in the review process. These tests are detailed in reports from Ammtec, SGS and PRA and are discussed at length in reports by Aminpro and in this review report.

Table 13.5 Oyu Tolgoi Project Flotation Testwork (Quantity of tests on each sample type)

Deposit	SW	C-Cv	C-Cc	C-Cpy	C-4,5,6	HS	HN	HN Entrée	Orebody Comps
Roughers	213	36	20	1	94	–	87	6	–
Cleaners and Roughers	390	13	12	–	110	42	74	6	4
Lock Cycle Tests	6	1	1	–	–	1	5	–	–
Column	8	–	–	–	–	–	2	–	–
Pilot	4	–	–	–	–	–	1	–	–

The comminution testwork for each deposit is summarized in Table 13.6. This work has been analyzed and used to prepare the throughput recommendations for each of the ores.

Table 13.6 Summary of Comminution Samples Dispatched to Testwork Facilities

Facility	Location	Scale of Testwork	Testwork Conducted	Quantity
MinnovEX Technologies	Toronto, Canada	Bench scale	SAG Power Index [SPI] (measured in minutes)	350
			Bond Ball Mill Work Index [BWi] (measured in kWh/t)	48
			Modified Bond Index [Mod-Bond] (measured in kWh/t)	319
			MinnovEX Crusher Index [Ci]	206
			specific gravity	7
AMMTEC	Perth, Australia	Bench scale	JKTech FAG/SAG Mill parameters	26
			JKTech Rod/Ball Mill Appearance function	26
			Unconfined Compressive Strength [UCS]	26
			Bond Impact Crushing Work Index	26
			Bond Abrasion Index	26
			Bond Rod Mill Work Index	26
			specific gravity	26
SGS Lakefield	Lakefield, Canada	Pilot scale	Pilot-plant runs at a series of circuit options	12 tests on one bulk sample
		Bench scale	JKTech FAG/SAG Mill parameters	Tests on seven composite samples (2003)
			JKTech Rod/Ball Mill Appearance function	
			Unconfined Compressive Strength [UCS]	
			Bond Impact Crushing Work Index	
			Bond Abrasion Index	
			Bond Rod Mill Work Index	
			specific gravity	

13.3.1 AMMTEC Bench-Scale Flotation Test Program

AMMTEC Perth, an experienced Australian metallurgical laboratory, was selected to perform the bench-scale flotation work. In early 2004, the testwork program parameters were finalized, and a campaign was initiated to collect samples from existing drill core at site. Concurrent with this sampling program, composite samples from the 2003 program were shipped to Perth for AMMTEC to conduct a series of validation and calibration tests. The 2004 samples began arriving in Perth in May 2004, and testwork commenced soon after. This program was completed in early 2005.

13.3.2 MinnovEX Comminution Testing

The MinnovEX Comminution Economic Evaluation Tool (CEET) was used as the primary methodology for estimating throughput rates. CEET uses grinding parameters measured from the geological drill core sample set. Samples were selected from the core at the same time as the flotation sample set and were sent to the MinnovEX laboratory in Toronto.

13.3.3 MinnovEX FLEET Testwork Program

To provide a secondary confirmation of flotation parameters, portions of the composite samples developed in Perth were sent to MinnovEX for MinnovEX Flotation Tests (MFT). These MFT parameters were used in conjunction with the MinnovEX Flotation Economic Evaluation Tool (FLEET) to provide alternative kinetic information intended to translate the laboratory tests into plant design criteria and to validate circulating loads.

13.3.4 AMMTEC Comminution Testing

A secondary, or validation, assessment of the milling rates was undertaken with the JKSimMet simulator method, which uses a suite of parameters obtained from testing of large-diameter (PQ or larger) core. Four PQ holes were drilled to intercept as many feed types as possible. Selected core from these holes was shipped to AMMTEC in late 2004 for measurement of JK grinding parameters. Splits from these samples were also sent to MinnovEX for SPI determinations.

13.3.5 SAG Pilot Plant

A bulk 250 t sample of mined rock, typical of Southwest ore, was shipped to SGS Lakefield in early 2005. A series of pilot-scale plant tests, intended to confirm the prediction of the bench scale comminution work, was conducted in April 2005.

13.3.6 Hugo Far North (Hugo North Extension)

While the 2004 test program was in progress, IMMI (now OT LLC) identified potentially large reserves of copper-rich material to the north of the previously sampled area of Hugo North. Additional samples were taken from that part of the orebody for bench-scale flotation work at AMMTEC and MinnovEX and bench-scale comminution work at MinnovEX. Flotation testwork focused on the kinetics of the roughers and cleaners by PRA was also carried out on samples from this area.

13.3.7 Bulk Flotation Test

Large composites of Southwest and Hugo North samples were made up from surplus sample at the AMMTEC laboratory. These samples were processed through pilot-scale equipment to generate large samples of concentrate and tailings for further testing. Concentrate was required for marketing analysis and to measure the thickening and filtration design parameters. Tailings material was required to confirm the design parameters for the thickeners, transportation pumping, and tailings deposition method. As well as physical plant design data, tailings were also evaluated to define environmental parameters.

13.3.8 Concentrate Upgrading Program, Lakefield

The bulk flotation test at AMMTEC on Southwest material did not produce concentrate to specification because the material had not been reground properly. Unfortunately, the physical characteristics of the test equipment were not adjusted until the Southwest material sample had been consumed. The off specification sample was shipped to SGS Lakefield, where it was reground to the correct size and upgraded in a pilot-scale flotation column brought in from MinnovEX. The Hugo North bulk flotation test was satisfactory and produced concentrate and tailings consistent with the design criteria.

13.3.9 Final Product Concentrate Assay Analysis

Final concentrates from the locked-cycle tests were assayed for a range of minor elements, as shown in Table 13.7. Fluorine and arsenic are the only significant deleterious elements identified.

Table 13.7 Concentrate Assay Analysis

Element	Unit	Southwest				Hugo North		Hugo South 0-5 Yrs	Central Covellite
		Range		Average	Median	0-5 Yrs	6-9 Yrs		
Ag	ppm	34	49	44	45	73	63	62	40
Al	ppm	5,125	9,150	6,845	7,160	8,415	10,950	11,750	6,175
As	ppm	20	1,290	431	140	400	800	2 320	3,180
Au	ppm	10	31	23	25	9	4	2	2
Ba	ppm	20	36	28	27	45	50	51	60
Be	ppm	<0.1	0	0	0	0	<0.1	<0.1	<2
Bi	ppm	<10	<10	<10	<10	<10	<10	<10	<10
Ca	ppm	1,715	3,260	2,197	1,973	880	777	812	1,370
Cd	ppm	8	65	24	15	7	8	25	100
Cl	ppm	<50	1,600	464	100	<50	<50	<50	<50
Co	ppm	85	178	127	128	19	64	80	163
Cr	ppm	18	40	26	22	130	100	44	182
Cu	%	20.4	25.7	23.4	24.5	41.6	31.8	32.8	20.5
F	ppm	250	410	320	310	330	420	510	130
Fe	%	26.8	31.3	29.5	30.2	17.3	21.1	22.9	26.9
Ge	ppm	1	25	6	1	<0.5	1	3	30

Element	Unit	Southwest				Hugo North		Hugo South 0-5 Yrs	Central Covellite
		Range		Average	Median	0-5 Yrs	6-9 Yrs		
Hg	ppm	0	2	1	0	3	4	<0.1	1
K	ppm	1,587	2,600	2,121	2,200	3,005	2,500	1,423	1,110
Li	ppm	<5	5	4	<5	<5	<5	<5	<5
Mg	ppm	1,700	3,015	2,299	2,090	930	765	691	335
Mn	ppm	60	260	123	91	81	85	116	27
Mo	ppm	1,031	3,071	1,987	1,850	548	286	798	130
Na	ppm	980	1,404	1,147	1,090	399	300	161	206
Ni	ppm	56	138	103	105	125	78	53	235
P	ppm	<100	100	100	100	100	100	235	200
Pb	ppm	248	1,182	546	441	247	184	165	45
Pd	ppm	0	0	0	0	<0.1	<0.10	<0.10	<0.10
Pt	ppm	<0.05	0	0	<0.05	<0.1	<0.10	<0.10	<0.10
Re	ppm	<0.05	3	1	0	0	0	1	0
S	%	34.4	38.0	36.0	36.1	27.4	28.7	32.8	45.3
Sb	ppm	10	65	31	30	142	380	10	10
Se	ppm	180	275	227	235	390	280	30	90
SiO2	%	3.8	7.5	4.7	4.2	8.4	8.7	8.3	4.7
Sn	ppm	1.0	4.7	2.3	1.8	1.2	4.1	10.1	19.3
Sr	ppm	18	58	32	27	69	109	122	114
Te	ppm	5	10	7	7	30	25	24	10
Ti	ppm	570	1,360	890	735	515	665	908	230
Tl	ppm	<0.5	1,160	129	<0.5	<0.5	<0.5	1	4
V	ppm	22	71	45	43	26	73	62	508
Y	ppm	4	7	5	4	<1	1	<1	2
Zn	ppm	488	2,931	1,808	2,069	444	425	853	1,455
Zr	ppm	255	437	350	346	485	451	353	479

13.4 2006-2007 Confirmatory Testwork

13.4.1 2006 SGS-Lakefield – Southwest and Central – Rougher/Cleaner Kinetics Verification

In 2006, confirmatory rougher and cleaner kinetics testwork was conducted at SGS-Lakefield. The testwork was conducted on the 2004 – 2005 IDP05 period composite samples to verify the IDP05 kinetics. The cleaner tests included variations in pH and regrind. As the kinetics of the roughers was developed under an FKT test, the parameter extractions include a relationship between kinetics and primary P₈₀.

13.4.2 Hugo North

A complete suite of rougher and cleaner flotation work was carried out at Process Research Associates Ltd (PRA) Laboratory in Vancouver. Variability work was conducted to determine the effect of head grade on recovery, and rougher and cleaner

tests were conducted to determine the effect of pH and grind. Samples were selected in the area of the deposit that is to be mined first.

13.5 2008 Heruga Scoping Study

Four composite samples of the Heruga deposit were sent to G&T laboratory for initial scoping testwork to assess, the mineralogical characteristics of feed and flotation products, the flotation response of each sample, the bond work index and to analyze the exit streams from the float work to identify opportunities for further improvement in metallurgical performance.

The Heruga metallurgical scoping study was conducted at G&T Metallurgical Services Ltd. and is reported in "Preliminary Metallurgical Assessment of Metallurgical Composites", Ivanhoe Mines.– Oyu Tolgoi Project Mongolia, KM2133, 5 August 2008. Overall, the composites responded well to the applied Oyu Tolgoi flow sheet.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

The total Oyu Tolgoi Project Mineral Resources, including the updated Heruga deposit Mineral Resources, reported as of 29 March 2012 are shown in Table 14.1. These Mineral Resources include material that is classified in Measured, Indicated and Inferred categories. Table 14.1 also includes mineralization on the Shivee Tolgoi and Javkhlant Joint Venture Properties, which are shown separately. A profile of the Oyu Tolgoi deposits is shown in Figure 14.1.

The base case CuEq cut-off grade assumptions for each deposit were determined using cut-off grades applicable to mining operations exploiting similar deposits. The CuEq cut-off applied was 0.6% CuEq.

The equivalent grade was calculated using assumed metal prices of US\$1.35/lb for copper, US\$650/oz for gold and \$10.00/lb for molybdenum. The CuEq formula also took into account the different levels of metallurgical recovery for the metals. Gold was assumed to have 91% of copper recovery, molybdenum 72% of copper recovery. For convenience, the formula is:

$$\text{CuEq\%} = \text{Cu\%} + ((\text{Au g/t} \times 18.98) + (\text{Mo g/t} \times 0.01586)) / 29.76$$

For consistency with previous reports, the actual ratio of gold to copper has remained constant which has meant that previous tabulations of resources are still valid at the new metal prices. Molybdenum is used only in the CuEq formula at the Heruga deposit. The contained gold and copper estimates in Table 14.1 have not been adjusted for metallurgical recoveries.

The Mineral Resources were estimated under the direction of Mr Scott Jackson, B.Sc. (Hons) employed by Quantitative Geoscience Pty Ltd ("QG") as Principal Consultants. Mr Jackson is independent of OT LLC or Ivanhoe Mines.

Figure 14.1 Idealized Profile of Southern Oyu and Hugo Dummett Deposits (Section Looking West)

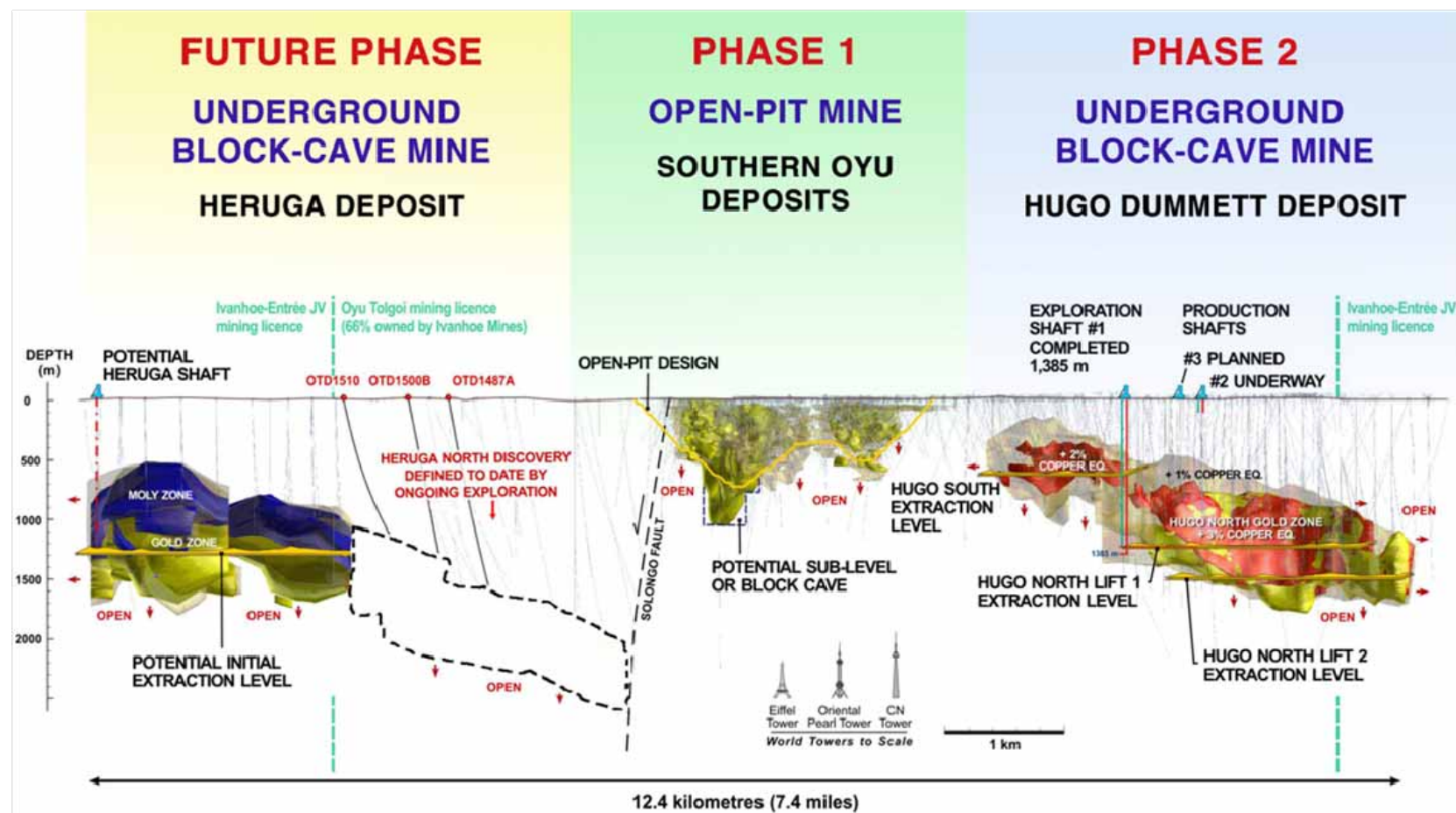


Table 14.1 Oyu Tolgoi Mineral Resource Summary, 29 March 2012

Deposit	Tonnage (t)	Cu (%)	Au (g/t)	CuEq (%)	Contained Metal		
					Cu (000 lb)	Au (oz)	CuEq (000 lb)
Southern Oyu Deposits							
Measured	101,590,000	0.64	1.10	1.34	1,430,000	3,590,000	3,000,000
Indicated	465,640,000	0.62	0.43	0.89	6,360,000	6,440,000	9,140,000
Measured + Indicated	567,230,000	0.62	0.55	0.97	7,750,000	10,030,000	12,130,000
Inferred	88,500,000	0.47	0.41	0.73	920,000	1,170,000	1,420,000
Hugo Dummett Deposits							
Indicated (Hugo North – OT LLC)	703,200,000	1.82	0.39	2.07	28,220,000	8,820,000	32,090,000
Indicated Shivee Tolgoi (Hugo North – EJV)	117,000,000	1.80	0.61	2.19	4,640,000	2,290,000	5,650,000
Indicated (All Hugo North)	820,200,000	1.82	0.42	2.08	32,910,000	11,080,000	37,610,000
Inferred (Hugo North – OT LLC)	722,800,000	0.97	0.30	1.17	15,460,000	6,970,000	18,640,000
Inferred Shivee Tolgoi (Hugo North – EJV)	95,500,000	1.15	0.31	1.35	2,420,000	950,000	2,840,000
Inferred (All Hugo North)	818,300,000	1.00	0.30	1.19	18,040,000	7,890,000	21,470,000
Inferred (Hugo South)	490,330,000	1.05	0.09	1.11	11,350,000	1,420,000	12,000,000
Inferred (Hugo North and South)	1,308,630,000	1.02	0.22	1.16	29,430,000	9,260,000	33,470,000
Heruga Deposit							
Heruga Javkhlant EJV	910,000,000	0.48	0.49	0.87	9,570,000	14,300,000	17,390,000
Heruga Ivanhoe	60,000,000	0.48	0.37	0.78	670,000	700,000	1,090,000
Inferred (All Heruga)	970,000,000	0.48	0.48	0.86	10,240,000	15,000,000	18,480,000
Oyu Tolgoi Project Grand Total							
Measured	101,590,000	0.64	1.10	1.34	1,430,000	3,590,000	3,000,000
Indicated	1,285,840,000	1.38	0.42	1.65	39,120,000	17,360,000	46,770,000
Measured + Indicated	1,387,430,000	1.33	0.47	1.63	40,680,000	20,970,000	49,860,000
Inferred	2,367,130,000	0.78	0.33	1.02	40,610,000	25,390,000	53,280,000

Notes:

- The contained gold and copper estimates in the tables have not been adjusted for metallurgical recoveries.
- The 0.6% CuEq cut-off has been used to enable comparison with previous disclosures.
- The Mineral Reserves are not additive to the Mineral Resources.
- CuEq has been calculated using assumed metal prices (\$1.35/lb. for copper and \$650/oz for gold and \$10/lb for molybdenum); $CuEq\% = Cu\% + ((Au\ g/t \cdot 18.98) + (Mo\ g/t \cdot 0.1586)) / 29.76$. Mo grades outside of Heruga are assumed to be zero for CuEq calculations. The CuEq formula also took into account the different levels of metallurgical recovery for the metals. Gold was assumed to have 91% of copper recovery, molybdenum 72% of copper recovery.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- EJV is the Entrée Gold Joint Venture. The EJV Shivee Tolgoi and EJV Javkhant Licences are held by Entrée Gold. The EJV Shivee Tolgoi and EJV Javkhant Licences are planned to be operated by OT LLC. OT LLC will receive 80% of cash flows after capital and operating costs.

14.2 Oyu Tolgoi and Shivee Licence Mineral Resources

Mineral Resource estimates for the Hugo North, Hugo South and Southern Oyu deposits were estimated by AMEC and are described in previous technical reports by Juras (2003), Juras (2005), Blower (2006) and AMEC Minproc (2008). They include estimates for trace and impurity elements including arsenic, fluorine, sulfur and silver as well as copper and gold.

14.2.1 Geologic Models

The resource estimates produced for the Oyu Tolgoi deposits have various close off dates associated with data incorporated into the resource estimates. These are listed below:

- 15 April 2005 Southern Oyu (South-West, Central, Southern Oyu, Bridge and Wedge Deposits)
- 1 November 2006 Hugo North and Hugo North Extension
- 1 November 2003 Hugo South
- 30 March 2010 Heruga

In each case, Ivanhoe produced 3-dimensional geological models of the major structures and lithological units as well as copper and gold shells at various cut-offs. Details of the geology used for each of the resource estimates are contained in the relevant technical reports. The major geological features at Oyu Tolgoi are summarized in Table 14.2.

Traditional grade shells were drawn manually for the lower-grade copper mineralization at Hugo North, as well as for Hugo North gold mineralization. The copper shell was made at a grade threshold at 0.6% Cu, whereas the gold shells (Main and West) were drawn at a grade threshold of 0.3 g/t Au. In addition, a third high grade gold shell using a 1.0 g/t Au threshold was constructed for Hugo North and Hugo North Extension.

The grade and mineralized shapes were checked for interpretational consistency on section and plan, and were found to have been properly constructed. The shapes honour the drill data and interpreted geology, and appear well constructed.

The solids and surfaces were used to code the drillhole data. A set of cross-sections and plans with drillholes colour-coded by domain were plotted and inspected to determine the proper assignment of domain.

Table 14.2 Lithology and Structural Solids and Surfaces, Oyu Tolgoi Project

Surfaces – General	Comment
Topography	Project-wide
Base of Oxide	Project-wide, but relevant only for OTS
Solids/Surfaces – Lithology	
Top of quartz monzodiorite (Qmd)	Hugo South only
quartz monzodiorite (Qmd) solid	OTS and Hugo North, Hugo North Extension
Base of ash flow tuff (DA2a - ign)	Project-wide
Base of unmineralized volcanic and sedimentary units(DA2b or DA3 or DA4)	Project-wide - used as a hanging wall limit to grade interpolation
Quartz-veined and mineralized Qmd – OT-Qmd	Southwest deposit in OTS only
Xenolithic Qmd (xQmd) containing quartz-veined and mineralized fragments	Southwest deposit in OTS only
Xenolithic biotite granodiorite (xBiGd) containing quartz-veined and mineralized fragments	Hugo North deposit only
Biotite granodiorite (BiGd) dykes	Project wide, most important in Hugo deposits. Unmineralized unit
Rhyolite (Rhy) dykes	Project wide, most important in OTS. Unmineralized unit
Hornblende biotite andesite and dacite dykes (HbBi)	OTS only, most important in Southwest deposit. Unmineralized unit
Basalt dykes (Bas)	Project wide; unmineralized unit.
Surfaces – Faults	
East Bat Fault	Hugo area: used to define Hugo North eastern limit
West Bat Fault	Hugo area: used to define Hugo North western limit
110 Fault	Hugo area: forms boundary between Hugo South and Hugo North deposits
North Boundary Fault	Hugo North area: used to define north western limit
East Bounding Fault	OTS area: forms eastern boundary to the Southwest deposit and western boundary to the Wedge deposit
West Bounding Fault	OTS area: forms informal western boundary to the Southwest deposit (generally marks contact between unmineralized Qmd and mineralized Va)
“Rhyolite” Fault	OTS area: marks boundary between Southwest and Central deposits
South Fault	OTS area: marks boundary between Wedge and South deposits
Solongo Fault	OTS area: defines the southern edge of the South deposit

14.2.2 Data Analysis

The lithologic, structural, and mineralized domains were reviewed in Hugo North and OTS to determine appropriate estimation or grade interpolation parameters. Several different procedures were applied to the data to discover whether statistically distinct domains could be defined using the available geological objects. For each zone, key lithologic categories were investigated within and outside grade shells. The mineralized volcanic and volcanoclastic units (basalt – DA1 or Va, and dacite ash flow tuff – DA2 or lgn) were separated in Hugo North but combined in the OTS deposits. The Qmd was treated as a distinct unit in the analysis, as was the quartz-veined and variable mineralized phases of the Qmd (OT-Qmd and the xenolithic Qmd) in the Southwest deposit. The mineralized xenolithic BiGd was also investigated in Hugo North.

Descriptive statistics, histograms and cumulative probability plots, box plots and contact plots, and X-Y scatter plots have been completed for copper and gold in each area. Results obtained were used to guide the construction of the block model and the development of estimation plans. The data analyses were conducted on composited assay data: 5 m down hole composites.

14.2.2.1 Histograms and Cumulative Frequency Plots

The statistical properties of the copper and gold data are summarized in Table 14.3 and Table 14.4 for OTS, and Table 14.5 and Table 14.6 for Hugo North.

Table 14.3 OTS Statistics for 5 m Composites – Cu% Data

Lithology/Zone	Shell	Mean	CV	q25	q50	q75	Max	No. of Comps
Southwest – Far South								
Va	–	0.29	0.68	0.16	0.25	0.38	1.99	2,828
Southwest								
Va	–	0.49	0.69	0.27	0.42	1.64	3.80	9,143
Qmd	–	0.10	1.60	0.02	0.04	0.10	1.58	1,904
xQmd	–	0.31	0.97	0.13	0.20	0.40	1.78	131
OT-Qmd	–	0.75	0.60	0.36	0.74	1.06	2.50	540
Southwest – Bridge								
Va + Qmd	Cu	0.46	0.69	0.26	0.39	0.56	2.87	1,373
Va	Bkgd	0.30	0.70	0.17	0.27	0.39	2.78	740
Qmd	Bkgd	0.12	1.21	0.03	0.06	0.17	1.28	611
Central								
Va + Ign	Cu	0.67	0.56	0.42	0.60	0.83	2.83	2,235
Qmd	Cu	0.64	0.65	0.38	0.56	0.80	4.97	4,526
Va + Ign	Bkgd	0.21	1.15	0.04	0.12	0.28	2.29	590
Qmd	Bkgd	0.12	1.45	0.03	0.06	0.14	2.44	3,431
South								
Va + Ign	Cu	0.52	0.71	0.29	0.44	0.64	2.59	1,961
Qmd	Cu	0.49	0.59	0.30	0.44	0.61	2.41	804
Va + Ign	Bkgd	0.22	0.56	0.14	0.21	0.29	0.78	816
Qmd	Bkgd	0.18	0.78	0.09	0.14	0.23	1.05	645
Wedge								
Ign + Va	Cu	0.55	0.66	0.33	0.46	0.68	2.55	1,366
Qmd	Cu	0.46	0.45	0.32	0.43	0.56	1.45	716
Ign + Va	Bkgd	0.24	0.93	0.05	0.20	0.35	1.52	478
Qmd	Bkgd	0.16	0.96	0.06	0.12	0.21	1.33	1,250

Table 14.4 OTS Statistics for 5 m Composites – Au g/t Data

Lithology/Zone	Shell	Mean	CV	Q25	q50	q75	Max	No. of Comps
Southwest – Far South								
Va	–	0.33	0.92	0.16	0.27	0.42	8.97	2,828
Southwest								
Va	Au	1.51	0.81	0.77	1.22	1.95	32.0	3,564
xQmd	Au	0.55	0.77	0.24	0.38	0.82	2.00	130
OT-Qmd	Au	1.20	0.79	0.57	0.95	1.60	6.83	509
Va	Bkgd	0.34	0.74	0.19	0.29	0.44	3.63	5,579
Qmd	Bkgd	0.12	2.48	0.01	0.03	0.09	4.15	1,855
Southwest – Bridge								
Va	–	0.12	1.46	0.05	0.08	0.13	3.38	1,573
Qmd	–	0.06	1.58	0.02	0.04	0.07	1.83	1,151
Central								
Va + Ign	Au	0.69	0.76	0.38	0.56	0.81	7.70	722
Qmd	Au	0.53	0.97	0.19	0.38	0.71	4.13	651
Va + Ign	Bkgd	0.12	1.31	0.04	0.07	0.13	1.86	2,103
Qmd	Bkgd	0.08	1.31	0.02	0.05	0.09	1.87	7,306
South								
Va + Ign	Au	0.55	4.79	0.09	0.23	0.43	51.1	438
Qmd	Au	0.42	1.82	0.16	0.27	0.42	7.88	372
Va + Ign	Bkgd	0.09	2.29	0.02	0.04	0.09	5.36	2,339
Qmd	Bkgd	0.11	1.37	0.03	0.07	0.13	1.93	1,077
Wedge								
Ign + Va	–	0.07	2.40	0.02	0.03	0.06	2.95	1,844
Qmd	–	0.06	1.83	0.02	0.03	0.06	2.24	1,966

Table 14.5 Hugo North Statistics for 5 m Composites – Cu% Data

Lithology	Cu Shell	Mean	CV	q25	q50	q75	Max	No. Of Comps
Va	Qtz	3.264	0.442	2.255	3.120	4.087	12.480	1,224
Ign	Qtz	2.433	0.466	1.683	2.239	2.872	5.859	53
Qmd	Qtz	3.249	0.437	2.193	3.124	4.068	11.544	2,515
xBiGd	Qtz	1.084	0.581	0.592	0.890	1.409	3.027	84
Va	0.6	1.040	0.478	0.694	0.974	1.304	5.933	1,780
Ign	0.6	0.993	0.507	0.664	0.931	1.284	4.000	1,176
Qmd	0.6	1.125	0.500	0.728	1.071	1.470	4.691	3,958
xBiGd	0.6	0.777	0.592	0.486	0.700	0.988	4.065	210
Va	Bkgd	0.339	0.644	0.100	0.377	0.514	0.852	101
Ign	Bkgd	0.205	1.012	0.061	0.155	0.295	3.004	1,504
Qmd	Bkgd	0.317	0.689	0.162	0.306	0.444	2.165	489

Table 14.6 Hugo North Statistics for 5 m Composites – Au g/t Data

Lithology (*)	Au Shell	Mean	CV	q25	q50	q75	Max	No. of Comps
Va	1 g/t	1.45	0.49	0.97	1.28	1.79	4.67	130
Ign	1 g/t	0.72	0.05	0.67	0.72	0.75	0.75	2
Qmd	1 g/t	1.61	0.86	0.95	1.35	1.92	28.49	1480
xBiGd	1 g/t	1.05	1.09	0.32	0.62	1.48	8.40	52
Va	Main	0.46	0.70	0.28	0.39	0.57	2.95	641
Ign	Main	0.40	0.75	0.18	0.38	0.54	1.19	23
Qmd	Main	0.52	0.77	0.30	0.44	0.65	7.43	1656
xBiGd	Main	0.32	0.73	0.17	0.26	0.39	1.20	105
Qmd	West	0.51	0.70	0.28	0.43	0.64	4.43	846
xBiGd	West	0.49	0.61	0.25	0.46	0.74	1.06	10
Va	Bkgd	0.08	1.11	0.03	0.05	0.10	1.58	2330
Ign	Bkgd	0.03	1.54	0.01	0.02	0.03	1.12	2716
Qmd	Bkgd	0.16	0.96	0.08	0.14	0.20	4.52	2984
xBiGd	Bkgd	0.14	0.74	0.06	0.12	0.19	0.69	127
Qmd	West	0.51	0.70	0.28	0.43	0.64	4.43	846

14.2.2.2 Southern Oyu Deposits

Copper grades in the mineralized volcanic units (dominantly basalt or Va) show essentially single lognormal distributions. CDF patterns show no distinct break for Southwest units except in the Bridge area. There, and in the remaining OTS zones, a noticeable bend in the distribution pattern occurs around 0.2% Cu. This is supportive of the choice of the 0.3% Cu threshold grade for these areas. Consistent copper mean values and coefficients of variation (CV) numbers occur in the Southwest Va and in Va inside the copper grade shell domains. Mean values range from 0.3 to 0.6, whereas CVs are low at 0.55 to 0.70. The mineralized copper domains all show about 7 to 15% included lower-grade material.

Copper values in quartz monzodiorite units (Qmd) show variable differences in mean grade values with adjacent mineralized volcanic units. The differences depend on which intrusive phase was predominant in an area, e.g. early, altered and mineralized Central, South, and Wedge deposit Qmd units versus late, poorly altered and essentially unmineralized Southwest Qmd. Exception to the latter are the unique quartz-veined and mineralized phases (OT-Qmd and xQmd) in the Southwest gold zone. Coefficient of variation (CV) values range from low inside copper grade shells (0.45 to 0.65) to over 1 in the Southwest and Central background areas. South and Wedge background Qmd show the lowest CV values (0.8 to 1.0). CDF plots commonly show double to triple lognormal distributions (in analyses ignoring grade shell boundaries). Usually one of the grade distributions is at very low values, accounting for 5% to 15% of the data. Another group represents about 30 to 70% of the remaining data. Threshold grades range from 0.15% to 0.30% Cu. These populations likely correspond to the three recognized intrusive Qmd phases in the Southern deposits. Occurrence of grade breaks in the grade trends support the use of grade shells in areas where mineralized Qmd units are present (Southwest – Bridge, Central, South, and Wedge).

Gold grades are highest in the Southwest gold shell domain, where every lithology has been enriched. The mineralized basalt (Va) units contain the highest gold values here, averaging 1.5 g/t Au, followed by the OT-Qmd at 1.2 g/t Au. Background Va units in Southwest are still considered to be enriched, averaging 0.33 g/t Au. CV values are relatively low, ranging from 0.7 to 0.9. Elsewhere, gold only occurs at elevated values in Central and South where defined by a gold grade shell. Va and Qmd units are similarly mineralized in these shells, averaging 0.4 to 0.7 g/t Au. CVs are low in Central (0.7 to 0.9) but high in South because of the occurrence of ubiquitous extreme values. Background Qmd in Southwest and Qmd and Va in Central and South contain quite low gold values (around 0.1 g/t) and corresponding high CVs. The Wedge deposit represents a gold-depleted area and contains the lowest average values (0.06 to 0.07 g/t) in the OTS.

CDF diagrams for gold show typical positively skewed trends. For Southwest, the plots display single lognormal distribution populations in the high-gold shell and background Va. In the rest of the zones, double lognormal populations are shown with a variable but high included low-grade component. In plots ignoring grade shell boundaries, the Southwest Va units CDF patterns show a double lognormal population, with the threshold between the two populations at around 0.7 g/t Au (representing the 60th percentile). Similar trends are seen at Central and South but with a lower threshold grade between the two populations of between 0.3 and 0.5 g/t Au. Unlike the Southwest case, the threshold at Central and South occurs at a high percentile (95th). These observations lend support to the use of gold grade shells at Southwest, Central, and South.

14.2.2.3 Hugo North Deposit

Copper grades in the mineralized units (Va, Ign, Qmd, and xBiGd) show single lognormal to near normal distributions inside each domain (0.6% Cu and quartz-vein shells). Coefficient of variation values are low at 0.4 to 0.6. There are small variations in grade as a result of lithological differences within the copper domains: generally, Qmd and Va have the highest values, followed by Ignimbrite, and xenolithic biotite

granodiorite (xBiGd) has the lowest grades of all lithologies. The CDF pattern of copper data for all domains shows evidence for three populations: a higher-grade population (above a copper threshold value of 2.0% to 2.5% Cu), a lower-grade zone (threshold value of 0.4% to 0.5% Cu), and a background lowest-grade domain. The pattern supports the construction of the Quartz-Vein shell (2% Cu is approximately coincident) and the 0.6% Cu shell.

Gold grade distributions at Hugo North show typical positively skewed trends. The distributions are slightly more skewed than those for copper, but the level of skewness can still be described as only mild to moderate within each domain. The Qmd shows higher average gold values than the Va, unit, which in turn is higher than the ignimbrite. CV values for the host lithologies are moderate, varying from 0.6 to 0.9. The CDF pattern of gold data of all domains and the background domain shows evidence for three populations: a higher-grade population (above a gold threshold value of 1 g/t Au), a lower-grade zone (threshold value of 0.2 to 0.3 g/t Au), and a background lowest-grade domain. The pattern supports the construction of the 1 g/t and 0.3 g/t gold shells.

14.2.3 Estimation Domains

14.2.3.1 Southern Oyu Deposits

The data analyses demonstrated that all of the copper and gold grade shells in the Southern deposits should be treated as separate domains with respect to copper and gold. Additionally, the mineralized volcanic (Va) and Qmd units will be treated as separate subdomains in Southwest for both copper and gold. Grades for blocks within the respective domains in each deposit or zone will be estimated with a hard boundary between them; only composites within the domain will be used to estimate blocks within the domain. Soft boundaries will be used between Qmd units and mineralized volcanic units in Central, South, and Wedge deposits. The various copper and gold grade shells for OTS deposits are shown in relation to each other and key structural features in Figure 14.2 and Figure 14.3.

14.2.3.2 Hugo North Deposit

Contact plot and box plot analysis led to a matrix of hard and soft contacts being developed for Hugo North; these are highlighted in Table 14.7. In all cases, the contacts between grade shells were treated as hard contacts.

Figure 14.2 Southern Oyu Copper Estimation Domains

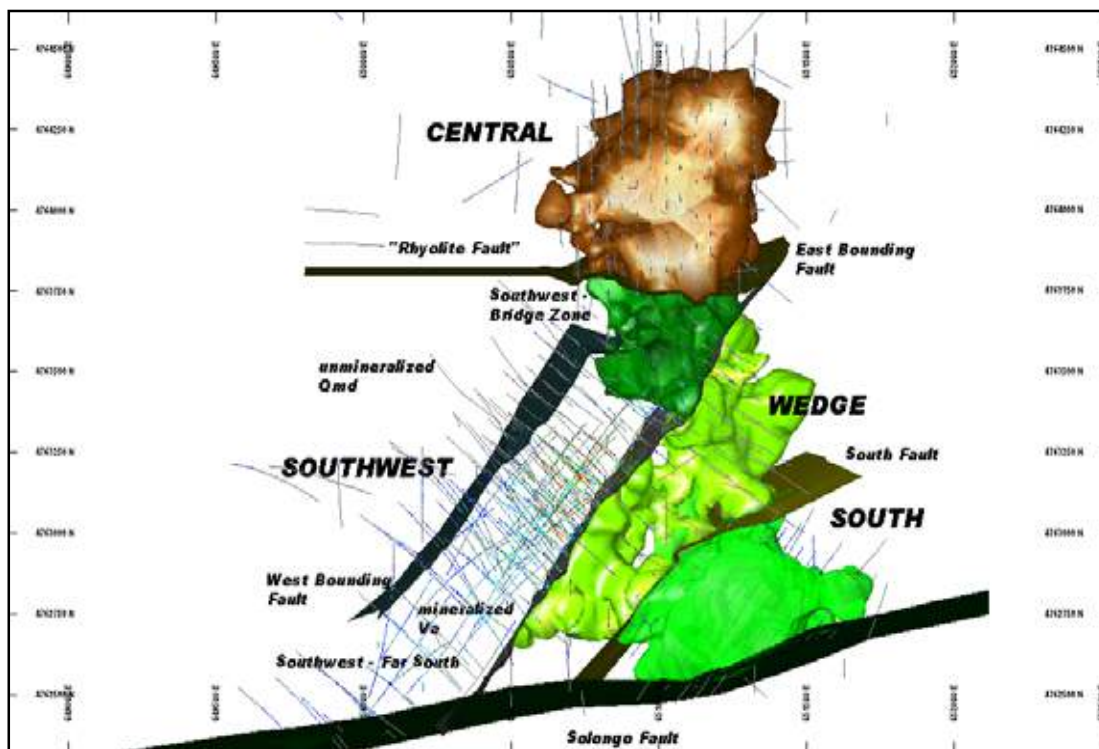


Figure 14.3 Southern Oyu Gold Estimation Domains

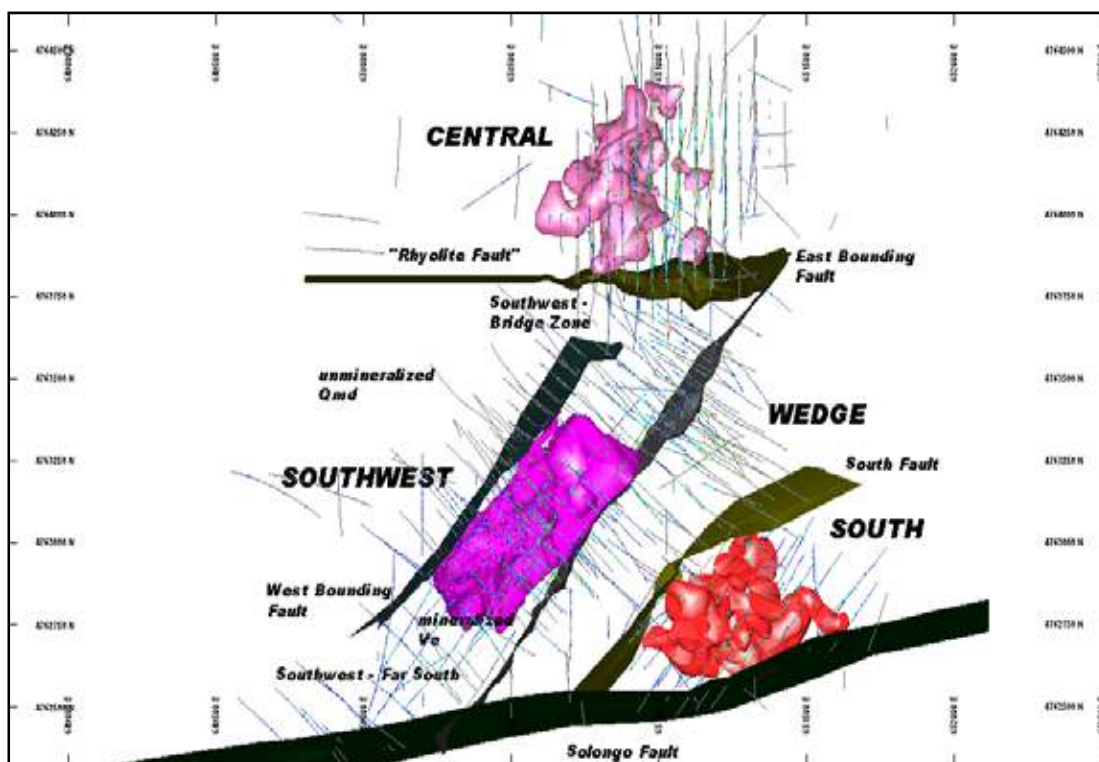


Table 14.7 Hugo North Copper Intra-domain Boundary Contacts

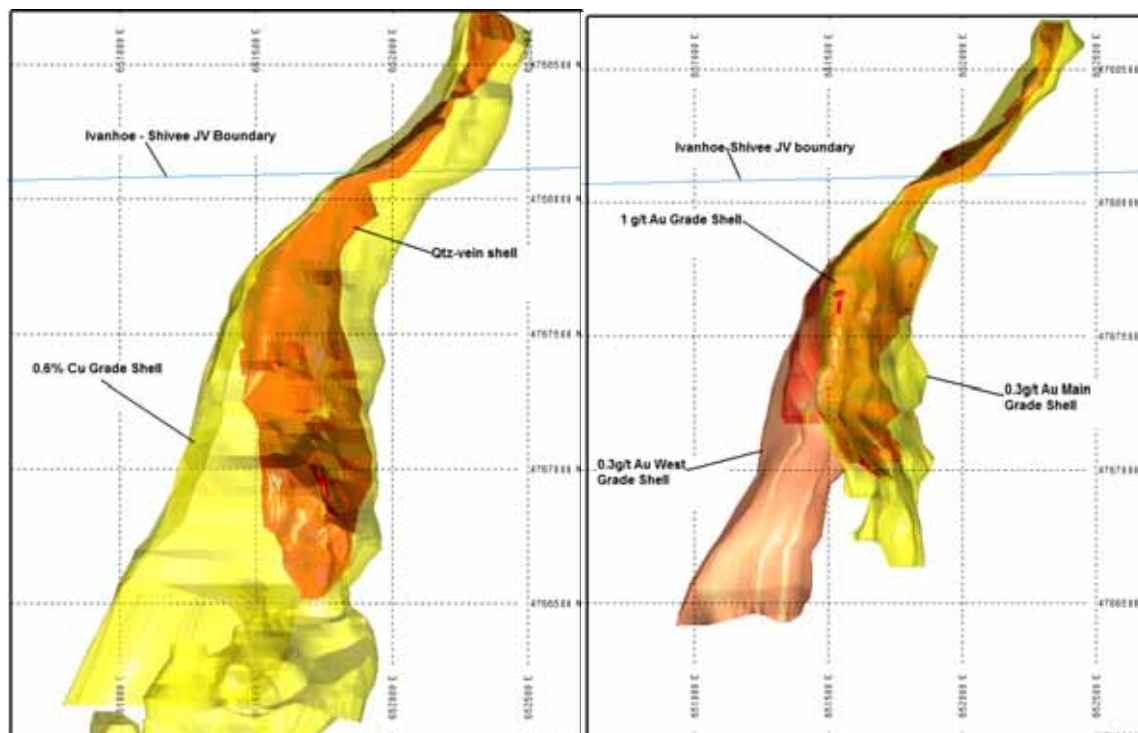
	Va	Qmd	Ignimbrite	xBiGd
Background Shell				
Va	Soft	Hard	Hard	Hard
Qmd	Hard	Soft	Hard	Hard
Ignimbrite	Hard	Hard	Soft	Hard
xBiGd	Hard	Hard	Hard	Soft
0.6% Shell				
Va	Soft	Hard	Soft	Hard
Qmd	Hard	Soft	Hard	Hard
Ignimbrite	Soft	Hard	Soft	Hard
xBiGd	Hard	Hard	Hard	Soft
Qtz Shell				
Va	Soft	Soft	Soft	Hard
Qmd	Soft	Soft	Hard	Hard
Ignimbrite	Soft	Hard	Soft	Hard
xBiGd	Hard	Hard	Hard	Soft

Table 14.8 Hugo North Gold Intra-domain Boundary Contacts

	Va	Qmd	Ignimbrite	xBiGd
Background Shell				
Va	Soft	Hard	Soft	Soft
Qmd	Hard	Soft	Hard	Hard
Ignimbrite	Soft	Hard	Soft	Hard
xBiGd	Soft	Hard	Hard	Soft
Main Shell				
Va	Soft	Soft	Soft	Hard
Qmd	Soft	Soft	Hard	Hard
Ignimbrite	Soft	Hard	Soft	Hard
xBiGd	Hard	Hard	Hard	Soft
West Shell				
Va	Soft	Hard	Hard	Hard
Qmd	Hard	Soft	Hard	Soft
Ignimbrite	Hard	Hard	Soft	Hard
xBiGd	Soft	Hard	Hard	Soft
1 g/t Au Shell				
Va	Soft	Soft	Soft	Hard
Qmd	Soft	Soft	Hard	Hard
Ignimbrite	Soft	Hard	Soft	Hard
xBiGd	Hard	Hard	Hard	Soft

The various copper and gold grade shells for Hugo North deposits are shown in relation to each other and key in Figure 14.4.

Figure 14.4 Hugo North Copper and Gold Estimation Domains



14.2.4 Evaluation of Extreme Grades Southern Oyu

Extreme grades were examined for copper and gold, mainly by histograms and CDF plots. Generally, the distributions do not indicate a problem with extreme grades for copper. OTS coppers values did not receive a cap. For gold, capped grades were selected in domains with high CVs and/or where trends defined in the cumulative probability plots began to become discontinuous. In the latter case, this generally occurred between the 98% to 99.5% level. Cap grade values for the OTS are shown in Table 14.9. The capped grades were applied to the assay data prior to compositing.

Table 14.9 Cap Grades (Assays), Southern Oyu and Hugo North Deposits

Zone and Domain	Cap Grade	Uncapped Mean	Uncapped CV	Capped Mean	Capped CV	Assays Capped
Southern Oyu – Au (g/t)						
SW – Au shell: Va, OT-Qmd	10	1.48	1.01	1.47	0.85	18
SW – Au shell: xQmd, Qmd	2	0.53	1.29	0.50	1.04	16
SW – background: Va	3	0.34	0.91	0.34	0.86	28
SW – background: Qmd	2	0.11	3.30	0.10	2.48	35
CO – Au shell	3	0.62	1.04	0.61	0.88	22
CO – background	0.7	0.09	1.59	0.08	1.29	224
SO – Au shell: Va	4	0.55	7.32	0.38	1.66	12
SO – Au shell: Qmd	2	0.42	2.63	0.34	1.13	24
SO – background	1	0.10	2.91	0.09	1.56	78
Wedge	1	0.07	2.49	0.06	1.81	51

14.2.5 Evaluation of Extreme Grades Hugo Dummett

At Hugo Dummett, rather than capping of extreme grades, outlier restriction was applied during grade estimation. The thresholds applied are shown in Table 14.10 and Table 14.11.

Table 14.10 Outlier Thresholds Applied to Cu Grade Domains

Grade Domain	Va Outlier Threshold (%)	Qmd Outlier Threshold (%)	Ign Outlier Threshold (%)	xBiGd Outlier Threshold (%)
Cu Qtz vein	7.5	7.0	5.5	3.5
Cu 0.6%	2.5	2.5	2.5	2.0
Background	0.7	0.85	0.85	0.7

Table 14.11 Outlier Thresholds Applied to Au Grade Domains

Grade Domain	Outlier Threshold (g/t)
1 g/t Gold Zone	5.0
W Gold Zone	2.0
Main Gold Zone	2.0
Background QMD, XBiGd	0.4
Background Va	0.5
Background Ignimbrite	0.3

14.2.6 Variography Southern Oyu

The spatial variability characteristics of the various deposits at Southern Oyu are summarized in Table 14.12 and Table 14.13.

Table 14.12 Cu and Au Variogram Parameters for Southern Oyu Deposits

		Nugget	Sills		Rotation Angles						Ranges					
	Model	Co	C1	C2	Z1	X1'	Y1''	Z2	X2'	Y2''	Z1	X1'	Y1''	Z2	X2'	Y2''
Cu – Southwest																
Mineralized volcanics (Va)	SPH	0.297	0.162	0.541	-3	87	34	85	5	-70	36	68	76	400	525	300
Qmd (Background)	SPH	0.179	0.518	0.303	-34	0	-60	31	13	85	52	175	21	225	350	125
Cu – Central																
Cu Shell – all units	SPH	0.229	0.552	0.218	15	-2	7	50	13	2	51	80	31	130	216	447
Background – all units	SPH	0.195	0.567	0.238	12	-37	2	21	13	11	17	88	26	67	237	182
Cu – South																
Cu Shell – all units	SPH	0.306	0.485	0.208	38	41	-68	-14	17	-38	61	308	47	373	66	413
Background – all units	SPH	0.421	0.300	0.279	8	20	98	32	-36	-7	79	142	12	384	33	221
Cu – Wedge																
Cu Shell – all units	SPH	0.342	0.318	0.340	-43	-9	-74	62	-39	47	19	60	23	344	82	230
Background – all units	SPH	0.163	0.408	0.429	40	51	4	21	-7	-42	9	46	25	75	125	200
Au – Southwest																
Au Shell – Va and OT-Qmd	SPH	0.295	0.340	0.366	53	-74	15	33	3	35	50	78	135	123	29	350
Background –Va	SPH	0.418	0.246	0.335	54	30	-66	28	-6	60	17	58	101	325	400	275
Background - Qmd	SPH	0.273	0.271	0.456	43	-56	-18	28	-6	9	20	100	135	275	400	150
Au – Central																
Au Shell – all units	SPH	0.209	0.414	0.377	62	21	-78	-25	-49	8	111	36	12	75	150	100
Background – all units	SPH	0.335	0.398	0.267	-22	5	25	53	58	-72	31	72	40	200	300	225
Au – South																
Au Shell and Background – all units	SPH	0.406	0.440	0.154	253	58	-70	253	58	-70	90	85	140	140	155	280
Au – Wedge																
all units	SPH	0.351	0.348	0.302	-37	29	88	51	-4	-10	38	110	9	225	125	75

Notes: Models are spherical (SPH) or exponential (EXP). The first rotation is about Z, left hand rule is positive. The second rotation is about X', right hand rule is positive. The third rotation is about Y'', left hand rule is positive.

Table 14.13 Azimuth and Dip Angles of Rotated Variogram Axes, Southern Oyu Deposits

	Axis Azimuth						Axis Dip					
	Z1	X1	Y1	Z2	X2	Y2	Z1	X1	Y1	Z2	X2	Y2
Cu – Southwest												
Mineralized volcanics (Va)	211	120	357	177	162	85	3	2	87	20	-69	5
Qmd (Background)	56	57	326	300	188	31	30	-60	0	5	76	13
Cu – Central												
Cu Shell – all units	299	105	16	237	140	50	83	7	-2	77	2	13
Background – all units	9	100	12	243	114	21	53	1	-37	73	11	13
Cu – South												
Cu Shell – all units	143	70	38	96	64	346	17	-44	41	49	-36	17
Background – all units	281	210	8	44	126	32	-7	69	20	54	-6	-35
Cu – Wedge												
Cu Shell – all units	45	75	317	2	117	62	16	-71	-9	32	34	-39
Background – all units	225	133	40	104	117	21	39	3	51	47	-42	-7
Au – Southwest												
Au Shell – Va and OT-Qmd	37	128	53	299	125	33	15	4	-74	55	35	3
Background – Va	156	96	54	301	108	28	21	-52	30	30	59	-6
Background – Qmd	64	148	43	330	117	28	33	-10	-56	80	9	-6
Au – Central												
Au Shell – all units	157	93	62	324	59	335	11	-66	21	41	6	-49
Background – all units	236	71	338	159	75	53	65	24	5	10	-31	58
Au – South												
Au Shell and Background – all units	180	276	253	180	276	253	-10	-29	59	-10	-29	59
Au – Wedge												
All units	232	139	323	117	142	51	1	61	29	79	-10	-4

Notes: Azimuths are in degrees. Dips are positive up and negative down.

14.2.7 Variography Hugo Dummett

The spatial variability characteristics of Hugo Dummett are summarized in Table 14.14 and Table 14.17.

Table 14.14 Copper Variogram Parameters

Model		Zone				
		Background	0.6% Shell - N	0.6% Shell - S	Qtz Shell - N	Qtz Shell - S
		SPH	SPH	SPH	SPH	SPH
Sills	Nugget	0.20	0.13	0.15	0.30	0.25
	C1	0.55	0.35	0.40	0.20	0.45
	C2	0.25	0.52	0.45	0.50	0.30
Rotation Angles	Z	0	45	25	45	-15
	X	0	0	-30	0	-30
	Y	0	0	20	0	20
Ranges	Y1	17	10	15	15	30
	X1	17	15	15	15	10
	Z1	17	20	15	15	15
	Y2	200	140	220	150	200
	X2	180	100	120	100	120
	Z2	220	175	200	150	200

Notes: Models are spherical (SPH) or exponential (EXP). Traditional ranges are used for the exponential variograms. Axis rotations are left-hand, right-hand, left-hand for the Z, X and Y axis, respectively.

Table 14.15 Azimuth and Dip Angles of Rotated Variogram Axes for Copper

Zone	Axis Azimuth			Axis Dip		
	Y	X	Z	Y	X	Z
Background	0	0	0	0	0	90
0.6% Shell – N	45	135	0	0	0	90
0.6% Shell – S	25	105	349	-30	17	54
Qtz Shell – N	45	135	0	0	0	90
Qtz Shell – S	345	65	309	-30	17	54

Notes: Azimuths are in degrees. Dips are positive up and negative down.

Table 14.16 Gold Variogram Parameters

Model		Zone				
		Background - N	Background - S	0.3 g/t Main Shell	0.3 g/t West Shell	1 g/t Shell
		EXP	SPH	EXP	EXP	EXP
Sills	Nugget	0.151	0.200	0.653	0.500	0.549
	C1	0.557	0.373	0.347	0.500	0.451
	C2	0.292	0.427	–	–	–
Rotation Angles	Z	-29	50	-2	-6	-81
	X	13	-9	38	22	70
	Y	-14	-51	-72	-7	2
	Z	-9	8	–	–	–
	X	45	-31	–	–	–
	Y	24	31	–	–	–
Ranges	Y	27	70	61	469	12
	X	15	54	139	30	149
	Z	93	30	25	49	22
	Y	1333	977	–	–	–
	X	210	400	–	–	–
	Z	919	969	–	–	–

Notes: Models are spherical (SPH) or exponential (EXP). Traditional ranges (short ranges) are used for the exponential variograms. Axis rotations are left-hand, right-hand, left-hand for the Z, X and Y axis, respectively.

Table 14.17 Azimuth and Dip Angles of Rotated Variogram Axes for Gold

Zone	First Structure						Second Structure					
	Axis Azimuth			Axis Dip			Axis Azimuth			Axis Dip		
	Y	X	Z	Y	X	Z	Y	X	Z	Y	X	Z
Background – N	58	-13	331	13	105	71	351	45	99	17	204	40
Background – S	151	-50	50	-9	133	39	8	-31	81	26	319	48
0.3 g/t Main Shell	26	-48	358	38	100	14	–	–	–	–	–	–
0.3 g/t West Shell	82	-6	354	22	157	67	–	–	–	–	–	–
1 g/t Shell	11	1	279	70	101	20	–	–	–	–	–	–

Notes: Azimuths are in degrees. Dips are positive up and negative down.

14.2.8 Model Setup

The block size for the OTS model was selected based on mining selectivity considerations (open pit mining). It was assumed the smallest block size that could be selectively mined as feed or waste, referred to the selective mining unit (SMU), was approximately 20 m x 20 m x 15 m. For Hugo North, mining selectivity was less of an issue because the mining method envisioned, block cave mining, does not easily yield to any selectivity. The same block was used for ease of integrating the block models for mine planning considerations.

The assays were composited into 5 m down hole composites in the OTS and Hugo North. The compositing honoured the lithology domain zone by breaking the composites on the domain code values. The composites included any post-mineral dyke material intervals that were deemed too small to be part of a post-mineral dyke geology model. The capping limits were applied to the assay data prior to compositing. Also, assay data in older drillholes (pre-OTD231) were adjusted for bias (Juras 2003, Hodgson 2004). The composite data were back-tagged by the grade and mineralized shells (on a majority code basis). Review of the compositing process and subsequent back-tagging found all performed as expected.

Bulk density data were assigned to a unique assay database file. These data were composited into 15 m fixed-length down hole values in both areas.

Various coding was done on the block models in preparation for grade interpolation. The block model was coded according to zone, lithologic domain, and grade shell. Percent below topography was also calculated into the model blocks. Post-mineral dykes were assumed to represent zero grade waste cutting the mineralized rock. The shapes were used to calculate an ore-remaining percent for each block by subtracting the volume percent dyke that intersects a block from 100. This percentage was used in the resource tabulation procedure to properly account for mineralized material and to account for internal or non-separable dilution.

Only the hypogene mineralization was estimated (with the Central chalcocite blanket being the only exception). The base of sulfide oxidation surface defined the top of the hypogene mineralization in the OTS deposits.

14.2.9 Estimation Southern Oyu

Modelling consisted of grade interpolation by ordinary kriging (KG). The chalcocite blanket in Central was interpolated by grade averaging because of the small data population in this domain. Only capped grades were interpolated. Nearest-neighbour (NN) grades were also interpolated for validation purposes. Blocks and composites were matched on estimation domain. To reduce the impact of locally inaccurate block grades due to conditional bias at the grade shell boundaries, all blocks straddling those contacts were estimated twice with each of the composite sets on either side of the contact. The final block grade was calculated with a volume-weighted average of the two domain grades in that block. The effect is to slightly smooth the grades at the hard grade shell boundary so that the distribution of block grades more closely approximates the shape of the composite distribution.

The search ellipsoids were oriented preferentially to the orientation of the respective zone as defined by bounding structures or to the attitude of the relevant copper or gold grade shell. Search ranges for OTS deposits were 175 to 250 m along the respective long axis, 125 to 175 m down the dip direction, and 150 to 200 m vertically. Block discretization was 4 m x 4 m x 3 m.

A two-pass approach was instituted for interpolation. The first pass allowed a single hole to place a grade estimate in a block, and the second pass required a minimum of two holes from the same estimation domain. This approach was used to enable most blocks to receive a grade estimate within the domains, including the background domains.

Blocks received a minimum of 5 and maximum of 4 composites from a single drillhole (for the two-hole minimum pass) in the OTS model. Maximum composite limit was 20.

For both metals, an outlier restriction was used to control the effects of high-grade composites within each of the domains, particularly in background domains and poorly mineralized units (e.g. Southwest Qmd). The threshold grades were generally set as the grade of the relevant grade shell, or the distinct break in the CDF curves in the case of poorly mineralized units. In Southwest (OTS), an outlier of 1.0 g/t Au was used in background domains for Va and Qmd. Other OTS gold outlier cases were 1.5 g/t implemented within the South gold shell (to minimize grade “blow-outs” from extreme grade composites at depth) and 0.5 g/t within the Wedge. OTS zones that contained copper grade shells had a 0.5% Cu outlier value implemented in the background domains, except for the South deposit, where it was set to 0.8%. The restricted distances were 40 m in OTS.

Bulk density values were estimated into the resource model by an averaging of composites. A maximum of six and minimum of two 15 m composites were used for the averaging. A rectangular search was used, measuring 200 m north x 200 m east x 50 m elevation. The assignment was constrained by matching composite values and model blocks that shared the same lithology domain in both the OTS and Hugo North models. In the event a block was not estimated, default density values were assigned based on lithology code.

Final resource grade values were adjusted to reflect likely occurrences of internal dilution from the unmineralized post-mineral dykes. The estimated mined for Southern Oyu deposits were tabulated and reported using these adjusted or diluted grade values. The planned mining method for Hugo North, underground block caving, allows for virtually no segregation of diluting material. Therefore at Hugo North, model blocks that contained less than 85% and greater than 15% mineralized material or feed by volume had final block grades recalculated with volume-weighted averaging of the interpolated grade and dyke grade (taken as zero) to 100% ore. Blocks containing more than 85% feed were allowed to use the existing interpolated grades. For block with less than 15% feed by volume, the blocks were reset to 100% but at zero grade values.

The Southern Oyu deposits will be mined by open-pit methods. This allows for potential segregation of unmineralized dyke material. However, contact dilution and feed loss will be inevitable during mining. For this reason, contact block grades and feed percent values were also adjusted. Model-blocks containing greater than 85% mineralized material or feed by volume, were reset to 100% ore, but the interpolated grades were adjusted by a volume-weighted averaging of the interpolated grade and dyke grade (taken as zero). Model blocks containing less than 15% feed by volume were reset to 0% feed and zero grade values. Blocks containing less than 85% and greater than 15% feed by volume retained the interpolated grade value. However, in this case, the feed percent values were adjusted downward by 15%.

14.2.10 Estimation Hugo Dummett

Interpolation was limited to the mineralized lithological units (Va, Ign, Qmd, and xBiGd). Only blocks within those units were interpolated, and only composites belonging to those units were used. Metal values within blocks belonging to all other units (post-mineral dykes and sediments) were set to zero.

Modelling consisted of grade interpolation by ordinary Kriging (KG) except for bulk density that was interpolated using inverse distance to the power of three (ID3). Both restricted and unrestricted grades were interpolated to allow calculation of the metal removed by the outlier restriction. Nearest-neighbour grades were also interpolated for validation purposes. Blocks and composites were matched on estimation domain.

The search ellipsoids were oriented preferentially to the general orientation of the grade shells. The ranges and the rotation angles for the various search ellipsoids are highlighted in Table 14.18 and Table 14.19. The search strategy employed concentric expanding search ellipsoids. The first pass used a relatively short search ellipse relative to the long axis of the variogram ellipsoid. For the second pass, the search ellipse was increased by 50% (up to the full range of the variogram) to allow interpolation of grade into those blocks not estimated by the first pass. A last third pass was performed using a larger search ellipsoid.

Table 14.18 Copper Search Ellipsoids for Hugo North

		Zone					
		Background – N	Background – S	0.6% Shell– N	0.6% Shell– S	Qtz Shell– N	Qtz Shell– S
Rotation Angles	Z	45	0	45	25	45	-15
	X	0	0	0	-30	0	-30
	Y	0	0	0	20	0	20
Ranges – First Pass	Y	150	150	150	150	150	150
	X	100	100	50	50	35	35
	Z	150	150	150	150	150	150
Ranges – Second Pass	Y	225	225	225	225	225	225
	X	150	150	75	75	50	50
	Z	225	225	225	225	225	225
Ranges – Third Pass	Y	400	400	400	400	400	400
	X	200	200	125	150	100	100
	Z	400	400	400	400	400	400
Number of Comps	Min	4	4	4	4	4	4
	Max	15	15	15	15	15	15
	Max per DDH	3	3	3	3	3	3

Note:

- MIN = minimum number of composites.
- MAX = maximum number of composites.
- MAX DDH = maximum number of composites derived from a single borehole.
- Axis rotations are left-hand, right-hand, left-hand for the Z, X and Y axis, respectively.

Table 14.19 Gold Search Ellipsoids for Hugo North

		Zone					
		Background	0.3 g/t Main Shell – N	0.3 g/t Main Shell – S	0.3 g/t West Shell	1 g/t Shell– N	1 g/t Shell –S
Rotation Angles	Z	0	30	0	0	30	0
	X	0	0	0	0	0	0
	Y	0	0	10	0	0	10
Ranges – First Pass	Y	200	150	150	375	150	150
	X	200	25	25	50	25	25
	Z	200	150	150	50	150	150
Ranges – Second Pass	Y	300	170	170	470	170	170
	X	300	45	45	75	45	45
	Z	300	170	170	75	170	170
Ranges – Third Pass	Y	500	250	250	600	175	175
	X	500	250	250	300	175	175
	Z	500	250	250	300	175	175
Number of Comps	Min	5	5	5	5	5	5
	Max	20	20	20	20	20	20
	Max per DDH	4	4	4	4	4	4

Note:

- MIN = minimum number of composites; MAX = maximum number of composites.
- MAX DDH = maximum number of composites derived from a single borehole.
- Axis rotations are left-hand, right-hand, left-hand for the Z, X and Y axis, respectively.

To ensure that at least two boreholes were used in the estimate, the number of composites from a single drillhole that could be used was set to one less than the minimum number of composites.

These parameters were based on the geological interpretation, data analyses, and variogram analyses. The number of composites used in estimating grade into a model block followed a strategy that matched composite values and model blocks sharing the same feed code or domain. The minimum and maximum number of composites was adjusted to incorporate an appropriate amount of grade smoothing.

Blocks that fall along grade domain boundaries were assigned two or more values, one for each of the grade domains present within the block. A final undiluted value was calculated by averaging the values for each domain within the block, weighed by the percentage of each domain within the block. This resulted in slightly smoothed metal grades along grade shell boundaries.

For both metals, an outlier restriction of 50 m was used to control the effects of high-grade samples within the domains, particularly in the background domains where unrestricted high-grade composites tended to result in “blow outs” from extreme grade composites. In outlier restricted kriging, outliers (i.e. values above the specified cut-off) are cut down to the specified threshold value if their distance to the interpolated block is greater than 50 m. If the distance to the interpolated block is less than 50 m, then

outliers are used at their full value. The outlier thresholds applied were defined at the 99th percentile of their respective population. The thresholds are shown in Table 14.10 and Table 14.11.

14.2.11 Validation

14.2.11.1 Visual Inspection

Detailed visual validation of the Hugo North and OTS resource models was done in plan and section, comparing resource block grades to original drillhole data. The checks showed good agreement between drillhole composite values and model cell values. The hard boundaries between grade shells appear to have constrained grades to their respective estimation domains. The addition of the outlier restriction values succeeded in minimizing grade smearing.

14.2.11.2 Nearest Neighbour Comparisons

The block model estimates were checked for global bias by comparing the average metal grades (with no cut-off) from the model (KG) with means from nearest-neighbour estimates. (The nearest-neighbour estimator declusters the data and produces a theoretically unbiased estimate of the average value when no cut-off grade is imposed and is a good basis for checking the performance of different estimation methods). Results show no problems with global bias in the estimates and are summarized in Table 14.20.

14.2.11.3 Swath Plots

The models were also checked for local trends in the grade estimates (grade slice or swath checks). This was done by plotting the mean values from the nearest-neighbour estimate versus the kriged results for benches (in 30 m swaths) and for northings and eastings (both in 40 m swaths). The kriged estimate should be smoother than the nearest-neighbour estimate, thus the nearest-neighbour estimate should fluctuate around the kriged estimate on the plots. The two trends behave as predicted and show no significant trends of copper or gold in the estimates in both OTS and Hugo North models. Swath plots from the Hugo North model are shown in Figure 14.5 to Figure 14.8.

Table 14.20 Global Model Mean Grade Values by Domain in Each Zone

Domain/Zone	NN Estimate	Kriged Estimate	% Difference
Cu (%) – Southern Oyu Deposits			
Southwest – Far South Va	0.217	0.220	+1.4
Southwest – Va	0.386	0.395	+2.3
Southwest – Qmd	0.040	0.042	+5.0
Southwest – Bridge Cu Shell	0.414	0.426	+2.9
Central Cu shell	0.611	0.613	+0.3
Central background	0.107	0.105	-1.9
South Cu Shell	0.474	0.470	-0.8
South background	0.172	0.172	-
Wedge Cu Shell	0.501	0.485	-3.2
Wedge background	0.113	0.116	+2.6
Au (g/t) – Southern Oyu Deposits			
Southwest Au Shell	1.350	1.358	+0.6
Southwest background (Va)	0.323	0.321	-0.6
Southwest background (Qmd)	0.044	0.046	+4.5
Central Au Shell	0.603	0.579	-4.0
Central background	0.074	0.073	-1.4
South Au Shell	0.345	0.344	-0.3
South background	0.077	0.077	-
Wedge	0.054	0.052	-3.8
Cu (%) Hugo Dummett			
All Zones	1.677	1.674	-0.2
Qtz-vein Domain	3.212	3.192	-0.6
0.6% Cu Domain	1.061	1.070	0.8
Cu background	0.222	0.229	3.2
Au (g/t) Hugo Dummett			
All Zones	0.39	0.38	-2.6
1 g/t Au Zone	1.42	1.39	-2.1
Main 0.3 g/t Au Zone	0.54	0.52	-3.7
West 0.3 g/t Au Zone	0.50	0.50	0.0
Au background	0.11	0.10	-9.1

Figure 14.5 Comparison of Kriged and Nearest Neighbour Copper Estimates with Increasing Depth – Cu Quartz-vein Domain

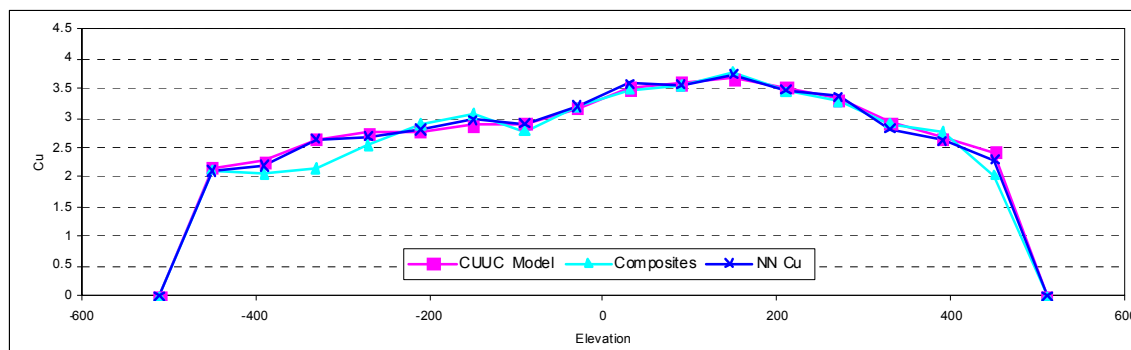


Figure 14.6 Comparison of Kriged and Nearest Neighbour Copper Estimates With Increasing Northing – Cu Quartz-vein Domain

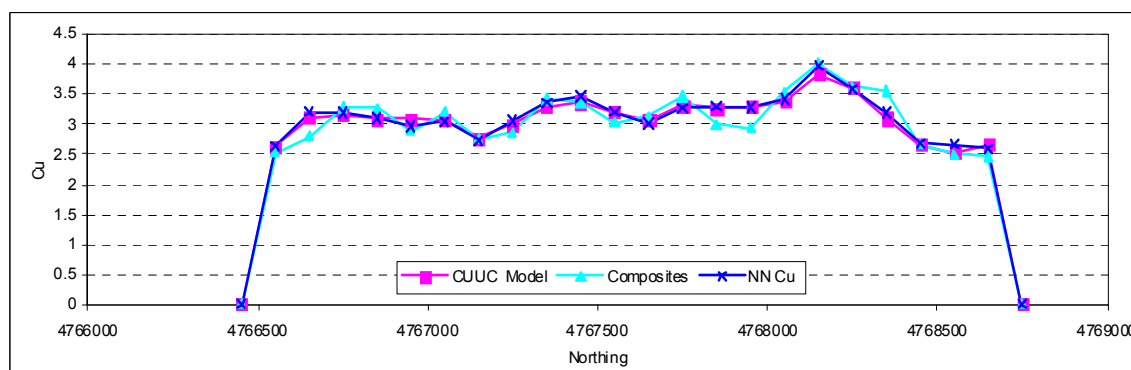


Figure 14.7 Comparison of Kriged and Nearest Neighbour Gold Estimates With Increasing Depth – Au Main + 1 g/t Domains

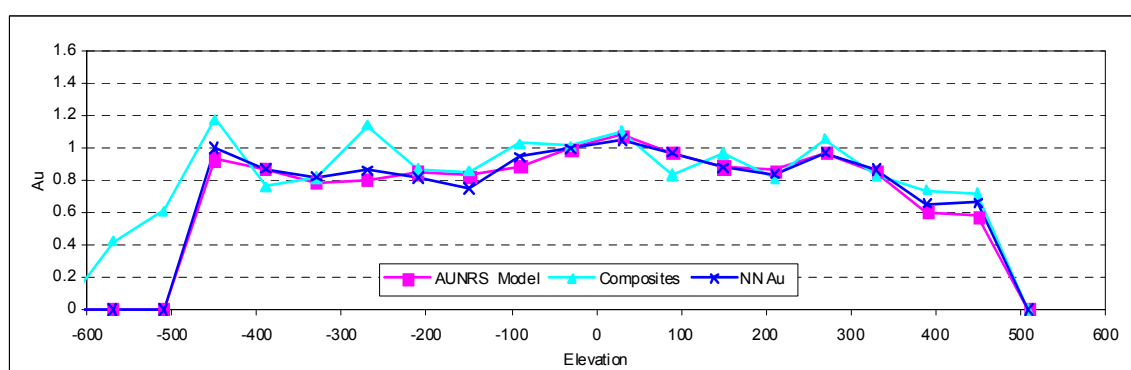
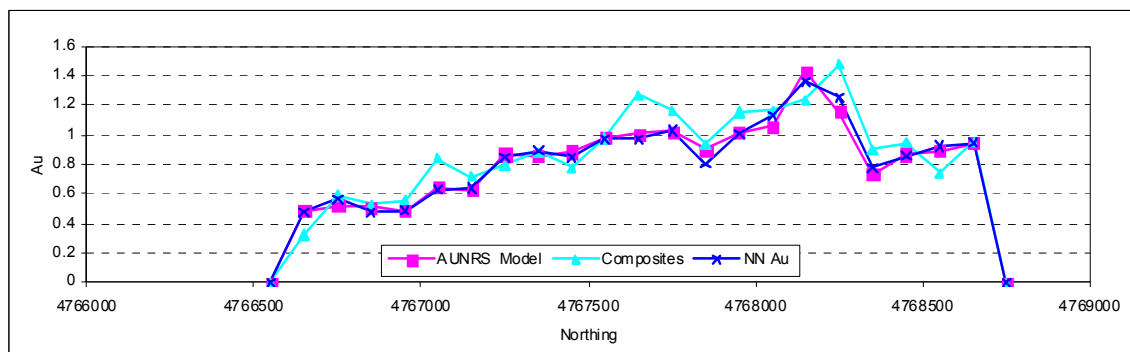


Figure 14.8 Comparison of Kriged and Nearest Neighbour Gold Estimates With Increasing Northing – Au Main + 1 g/t Domains



14.2.12 Additional Drilling at Hugo Dummett Since the Last Estimate

Subsequent to the Hugo Dummett estimate in 2006, Ivanhoe completed 27 drillholes totalling approximately 27,000 m in the Hugo North deposit and Hugo North Extension. The holes have been drilled for hangingwall and footwall structure delineation, resource infill, and geotechnical purposes. Not all holes were assayed. Figure 14.9 is a drill plan showing the location the 2007 drillholes and Figure 14.10 and Figure 14.11 are typical cross-sections showing these drillhole results relative to the February 2007 estimated block grades and geology wireframes. Pre-2007 holes are removed in the plan and sections for clarity. While individual contacts may have moved up to 50 m in distance for individual holes, the overall effect on tonnes and grade for the Hugo Dummett resource was considered negligible and did not warrant an updated resource estimate.

Figure 14.9 2007 Drillhole Location Plan Additional Drilling Since Last Resource Estimate

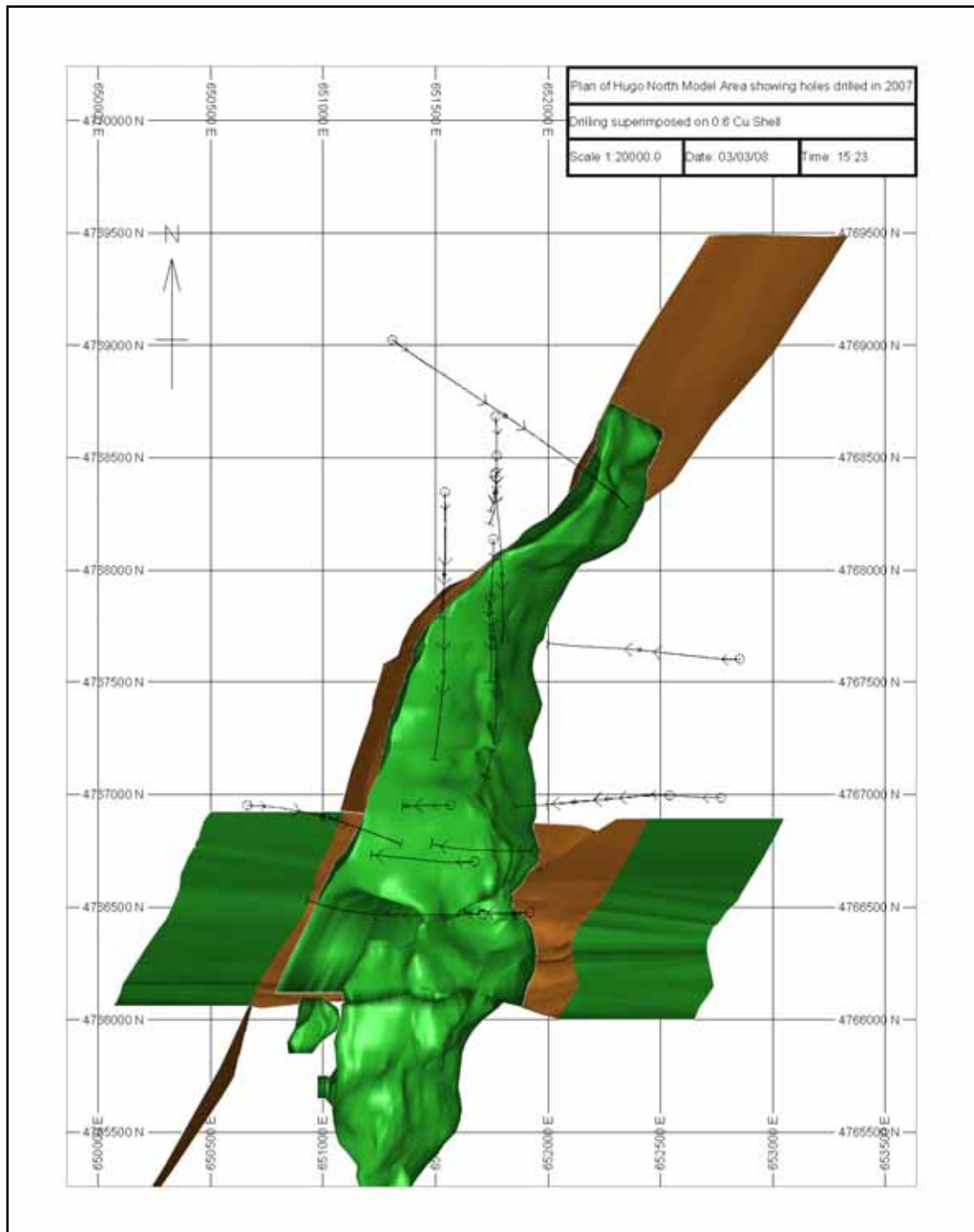


Figure 1 is a map of the study area, showing the location of the study area (OTD1435) and the location of the study area (OTD1435). The map displays a network of roads and a color-coded area representing the study area. A legend on the right indicates the color scale for the study area, ranging from 0.0 (blue) to 4.0 (red). The map includes a scale bar (0 to 1000) and a north arrow.

Figure 1 is a 3D visualization of the spatial distribution of the first principal component (PC1) scores. The map shows a grid of points colored according to PC1 scores, with a color scale ranging from -250 (blue) to 250 (red). The map includes a coordinate system with latitude (51° 17' 00" to 51° 18' 00" N) and longitude (7° 56' 00" to 7° 57' 00" E) labels. A legend on the right indicates the color scale for PC1 scores.

14.2.13 Mineral Resource Classification

The Mineral Resources of the Project were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the Project satisfies sufficient criteria to be classified into Measured, Indicated and Inferred Mineral Resource categories.

14.2.13.1 Measured Mineral Resources

The Measured Mineral Resource category was supported at the Southwest deposit where drill spacing was about 50 m. In classifying the resource model, a three-hole rule was used, where blocks containing an estimate resulting from three or more samples from different holes (all within 55 m and at least one within 30 m) were classified as Measured Mineral Resource.

14.2.13.2 Indicated Mineral Resources

The Indicated Mineral Resource category is supported at Hugo North where the drill spacing is approximately 125 m x 75 m centers. Geologic and grade continuity is demonstrated by inspection of the model and drillhole data in plans and sections over the various zones.

The Indicated Mineral Resource category in the southern Oyu Deposits is supported where the drill spacing is at a nominal 70 m on and between sections. A two-hole rule was used, where blocks contained an estimate resulting from two or more samples from different holes. For the Southwest deposit the two holes needed to be within 75 m, with at least one hole within 55 m. For the remaining Southern Oyu deposits, both holes needed to be within 65 m, with at least one hole within 45 m, to be classified as Indicated Mineral Resource.

14.2.13.3 Inferred Mineral Resources

All interpolated blocks that did not meet the criteria for Indicated Mineral Resource were classified as Inferred Mineral Resource if they fell within 150 m of a drillhole composite.

14.2.14 Independent Verification of Resource Estimation

The previous report sections (Sections 14.2.1 to 14.2.13) describe the processes by which resources were modelled by AMEC. In order to fulfil their responsibilities as independent QPs, Scott Jackson undertook a program of work to verify these resource estimates. The checking processes employed, and conclusions reached are outlined below.

14.2.14.1 Southern Oyu

The Southern Oyu deposits comprise the Southwest, South, Wedge and Central deposits. QG conducted an audit-level review of the current Southern Oyu estimate. This model was built by AMEC in 2005 (see Juras, 2005). All of the Southern Oyu resource models (Southwest, South, Wedge and Central) were built by using the same method and work flow. QG examined the integrity of the Oyu South deposit in detail.

The work covered included:

- Data loading and review – ensure the inputs used were as they reported.
- Domaining – review logic and construction.
- Data coding (drilling and model) – check coding.
- Statistical analysis – replicate statistics.
- Estimation validation – re-estimate key domains and variables.
- Resource Table validation.

The key findings are reported below:

(1) Data

QG obtained raw, unmodified database extractions, and compared these to the data used in AMEC's estimate. In the Oyu South estimate, all unsampled intervals were assigned a default grade of zero, rather than leaving them as null. This ensures that the volume corresponding to lithologies that were both not sampled and not separately domained, are informed with appropriate grades. QG is comfortable with this practice:

A number of minor discrepancies were observed between original data and AMEC data, such as:

- 10 intervals from the database were not used in the estimate.
- 79 intervals had different Cu values.
- 33 intervals had different Au values.

QG suspects that some minor adjustments were made by AMEC to account for laboratory biases (Juras, 2005). Unfortunately, the process followed or corrections used are not documented. Overall, the number of discrepancies is small, and the impact will be negligible.

(2) Domain Checks, Coding and Compositing

Domains for estimation were based on combinations of lithology and grade shells for the main variables (Cu and Au). A detailed description of the AMEC domaining strategy is given in Juras (2005). QG agrees with the overall domaining strategy and considers that it properly constrains volumes of similar material in terms of grade and/or lithology.

QG checked the coding of drillholes from domain wireframes, by re-coding and comparing against the AMEC coding. Coding is generally consistent and correct, although a few minor differences were found.

(3) Estimation

As part of QG's checking, all key parameters were tested and replicated. Minor differences were noted in the variograms QG generated but these were likely to be related to subtle differences in software implementation.

Several different check estimates were built to test the veracity of the original Oyu South estimate:

- Estimate using original composites, original parameters.
- Estimate using QG generated composites and original parameters.
- Estimate using QG generated composites and original parameters.

As expected, there were minor differences between the three models listed above and the current reported model. After exhaustive checking, QG is comfortable that any differences are not material.

(4) Resource Tables

Resource tables reported in the various technical reports were replicated by QG.

14.2.14.2 Hugo North

Similar to the process undertaken for the Southern Oyu, QG re-estimated the Hugo North deposit to check there were no material errors in the currently reported estimated and provide assurance that QG can act as QPs for the Hugo Dummett estimates.

The work flow was the same as that for the checks on Oyu South as described above.

In summary, QG examined the current Cu and Au estimates for Hugo North and conclude that these are robust and suitable for public reporting under the auspices of the NI 43-101. Scott Jackson agree to sign off as QPs for Hugo North under NI 43-101.

QG noted minor problems and discrepancies related to implementation and software issues but does not consider these to have materially impacted on the estimates.

Re-estimation of Cu and Au in Hugo North by QG has independently generated models that are within close limits of the model most recently reported publicly by Ivanhoe Mines.

14.2.15 Mineral Resource Summary

The Mineral Resources of Hugo North, Hugo South and the Southern Oyu are shown in Figure 14.8 to Figure 14.10 respectively. In addition, the total project resources are summarized at a 0.6% CuEq cut-off in Table 14.21.

The base case CuEq cut-off grade assumptions for each deposit were determined using cut-off grades applicable to mining operations exploiting similar deposits. For open pit resources, a base case cut-off for resources of 0.3% CuEq was applied, for underground block cave resources a 0.6% CuEq cut-off was applied.

The equivalent grade was calculated using assumed metal prices of US\$1.35/lb for copper, US\$650/oz for gold and US\$1/lb for molybdenum. For convenience, the formula is:

$$\text{CuEq\%} = \text{Cu\%} + ((\text{Au g/t} \times 18.98) + (\text{Mo g/t} \times 0.01586)) / 29.76$$

Molybdenum was only included in the copper equivalent formula for Heruga. At Hugo Dummett and Southern Oyu, molybdenum occurs in concentrations considered too low to justify the capital involved to add a molybdenum recovery circuit.

The contained gold and copper estimates in the tables have not been adjusted for metallurgical recoveries however the differential recoveries was taken into account when calculating the equivalent formula. The various recovery relationships at Oyu Tolgoi are complex and relate both to grade and Cu:S ratios. For the purposes of calculating equivalence, gold recovery is assumed to be 91% of copper recovery, molybdenum is assumed to be 72% of copper recovery.

The CuEq formula now applied to resources at Oyu Tolgoi has the same ratio of copper to gold as previously used (Juras 2005 and Cinits 2007). This ensures the tabulations of resources previously disclosed are consistent with the new equivalence formula.

Table 14.21 Mineral Resource Inventory, Hugo North, Based on Drilling Completed to 1 November 2006, and Reported by Ivanhoe (20 February 2007)

Class Hugo North Deposit	CuEq Cut-off	Tonnage (t)	Cu (%)	Au (g/t)	CuEq (%)	Contained Metal		
						Cu ('000 lb)	Au (oz)	CuEq ('000 lb)
Indicated (Ivanhoe)	3.5	125,300,000	3.74	0.93	4.34	10,331 000	3,750,000	11,989,000
	3	175,400,000	3.49	0.84	4.03	13,496 000	4,740,000	15,584,000
	2.5	228,100,000	3.25	0.76	3.73	16,343 000	5,570,000	18,757,000
	2	276,900,000	3.03	0.69	3.47	18,497 000	6,140,000	21,183,000
	1.5	356,000,000	2.69	0.61	3.08	21,112 000	6,980,000	24,173,000
	1.25	432,800,000	2.43	0.54	2.78	23,186 000	7,510,000	26,526,000
	1	541,600,000	2.15	0.46	2.44	25,672 000	8,010,000	29,134,000
	0.9	585,100,000	2.05	0.44	2.33	26,443 000	8,280,000	30,055,000
	0.8	626,000,000	1.97	0.42	2.24	27,188 000	8,450,000	30,914,000
	0.7	666,700,000	1.89	0.40	2.14	27,780 000	8,570,000	31,454,000
	0.6	703,200,000	1.82	0.39	2.07	28,215 000	8,820,000	32,091,000
	0.5	735,700,000	1.76	0.38	2.00	28,546 000	8,990,000	32,439,000
	0.4	765,100,000	1.71	0.36	1.94	28,844 000	8,860,000	32,723,000
	0.3	798,200,000	1.65	0.35	1.87	29,036 000	8,980,000	32,907,000
Indicated (Shivee)	3.5	22,300,000	3.68	1.43	4.59	1,809 000	1,030,000	2,257,000
	3	32,000,000	3.36	1.29	4.18	2,370 000	1,330,000	2,949,000
	2.5	42,400,000	3.08	1.17	3.83	2,879 000	1,590,000	3,580,000
	2	52,300,000	2.84	1.09	3.53	3,275 000	1,830,000	4,070,000
	1.5	65,400,000	2.56	0.96	3.17	3,691 000	2,020,000	4,571,000
	1.25	74,300,000	2.39	0.88	2.96	3,915 000	2,100,000	4,849,000
	1	84,800,000	2.22	0.80	2.73	4,150 000	2,180,000	5,104,000
	0.9	89,700,000	2.14	0.77	2.63	4,232 000	2,220,000	5,201,000
	0.8	96,700,000	2.04	0.72	2.50	4,349 000	2,240,000	5,330,000
	0.7	107,400,000	1.91	0.66	2.33	4,522 000	2,280,000	5,517,000
	0.6	117,000,000	1.80	0.61	2.19	4,643 000	2,290,000	5,649,000
	0.5	123,900,000	1.73	0.58	2.10	4,726 000	2,310,000	5,736,000
	0.4	130,300,000	1.67	0.55	2.02	4,797 000	2,300,000	5,803,000
	0.3	137,900,000	1.59	0.52	1.92	4,834 000	2,310,000	5,837,000
Indicated (All Hugo North)	3.5	147,600,000	3.73	1.01	4.38	12,138 000	4,790,000	14,253,000
	3	207,400,000	3.47	0.91	4.05	15,866 000	6,070,000	18,518,000
	2.5	270,500,000	3.22	0.82	3.75	19,202 000	7,130,000	22,363,000
	2	329,200,000	3.00	0.76	3.48	21,773 000	8,040,000	25,257,000
	1.5	421,400,000	2.67	0.67	3.09	24,805,000	9,080,000	28,707,000
	1.25	507,100,000	2.43	0.59	2.80	27,167,000	9,620,000	31,303,000
	1	626,400,000	2.16	0.51	2.48	29,829,000	10,270,000	34,248,000

Class Hugo North Deposit	CuEq Cut-off	Tonnage (t)	Cu (%)	Au (g/t)	CuEq (%)	Contained Metal		
						Cu ('000 lb)	Au (oz)	CuEq ('000 lb)
	0.9	674,800,000	2.06	0.48	2.37	30,646,000	10,410,000	35,258,000
	0.8	722,700,000	1.98	0.46	2.27	31,547,000	10,690,000	36,167,000
	0.7	774,100,000	1.89	0.44	2.17	32,255,000	10,950,000	37,033,000
	0.6	820,200,000	1.82	0.42	2.08	32,910,000	11,080,000	37,611,000
	0.5	859,600,000	1.76	0.40	2.01	33,354,000	11,050,000	38,091,000
	0.4	895,400,000	1.70	0.39	1.95	33,558,000	11,230,000	38,493,000
	0.3	936,200,000	1.64	0.38	1.88	33,849,000	11,440,000	38,803,000
Inferred (Ivanhoe)	3.5	3,600,000	3.06	1.41	3.96	243,000	160,000	314,000
	3	12,900,000	2.80	0.98	3.43	796,000	410,000	975,000
	2.5	29,000,000	2.43	0.96	3.04	1,554,000	900,000	1,944,000
	2	54,700,000	2.08	0.91	2.66	2,508,000	1,600,000	3,208,000
	1.5	127,800,000	1.69	0.65	2.10	4,762,000	2,670,000	5,917,000
	1.25	229,800,000	1.45	0.51	1.78	7,346,000	3,770,000	9,018,000
	1	385,500,000	1.25	0.41	1.51	10,624,000	5,080,000	12,833,000
	0.9	451,700,000	1.18	0.38	1.43	11,751,000	5,520,000	14,240,000
	0.8	530,600,000	1.12	0.35	1.34	13,101,000	5,970,000	15,675,000
	0.7	612,100,000	1.05	0.33	1.26	14,169,000	6,490,000	17,003,000
	0.6	722,800,000	0.97	0.30	1.17	15,457,000	6,970,000	18,644,000
	0.5	841,800,000	0.90	0.28	1.08	16,703,000	7,580,000	20,043,000
	0.4	974,900,000	0.83	0.26	0.99	17,839,000	8,150,000	21,278,000
	0.3	1,108,200,000	0.76	0.24	0.92	18,568,000	8,550,000	22,477,000
Inferred (Shivee)	3.5	1,400,000	3.32	1.03	3.98	102,000	50,000	123,000
	3	3,600,000	2.97	0.88	3.53	236,000	100,000	280,000
	2.5	5,900,000	2.68	0.87	3.23	349,000	170,000	420,000
	2	11,000,000	2.20	0.86	2.75	534,000	300,000	667,000
	1.5	29,100,000	1.73	0.58	2.10	1,110,000	540,000	1,347,000
	1.25	45,000,000	1.55	0.46	1.84	1,538,000	670,000	1,825,000
	1	62,200,000	1.39	0.39	1.64	1,906,000	780,000	2,249,000
	0.9	70,000,000	1.33	0.37	1.56	2,053,000	830,000	2,407,000
	0.8	78,300,000	1.27	0.34	1.49	2,192,000	860,000	2,572,000
	0.7	87,000,000	1.21	0.32	1.42	2,321,000	900,000	2,724,000
	0.6	95,500,000	1.15	0.31	1.35	2,421,000	950,000	2,842,000
	0.5	105,200,000	1.09	0.29	1.27	2,528,000	980,000	2,945,000
	0.4	127,600,000	0.96	0.26	1.13	2,701,000	1,070,000	3,179,000
	0.3	152,400,000	0.85	0.23	1.00	2,856,000	1,130,000	3,360,000
Inferred (All Hugo North)	3.5	5,000,000	3.13	1.30	3.96	345,000	210,000	437,000
	3	16,500,000	2.84	0.96	3.45	1,033,000	510,000	1,255,000
	2.5	34,900,000	2.47	0.94	3.07	1,900,000	1,050,000	2,362,000
	2	65,700,000	2.10	0.90	2.68	3,042,000	1,900,000	3,882,000

Class Hugo North Deposit	CuEq Cut-off	Tonnage (t)	Cu (%)	Au (g/t)	CuEq (%)	Contained Metal		
						Cu ('000 lb)	Au (oz)	CuEq ('000 lb)
	1.5	156,800,000	1.70	0.64	2.10	5,877,000	3,230,000	7,259,000
	1.25	274,800,000	1.47	0.50	1.79	8,906,000	4,420,000	10,844,000
	1	447,700,000	1.27	0.41	1.53	12,535,000	5,900,000	15,101,000
	0.9	521,800,000	1.20	0.38	1.45	13,804,000	6,370,000	16,680,000
	0.8	608,900,000	1.14	0.35	1.36	15,303,000	6,850,000	18,257,000
	0.7	699,100,000	1.07	0.33	1.28	16,491,000	7,420,000	19,728,000
	0.6	818,300,000	1.00	0.30	1.19	18,040,000	7,890,000	21,468,000
	0.5	947,000,000	0.92	0.28	1.10	19,208,000	8,530,000	22,966,000
	0.4	1,102 500,000	0.84	0.26	1.01	20,417,000	9,220,000	24,549,000
	0.3	1,260 500,000	0.77	0.24	0.93	21,398,000	9,730,000	25,844,000

Table 14.22 Mineral Resource Inventory, Hugo South (1 November 2003)

Hugo South Deposit	CuEq Cut-off	Tonnage (t)	Cu (%)	Au (g/t)	CuEq (%)	Cu ('000 lb)	Au (oz)	CuEq ('000 lb)
Inferred	3.5	5,440,000	3.71	0.25	3.87	440,000	40,000	460,000
	3.0	11,950,000	3.38	0.21	3.51	890,000	80,000	920,000
	2.5	22,760,000	3.02	0.18	3.14	1,520,000	130,000	1,580,000
	2.0	38,900,000	2.67	0.15	2.77	2,290,000	190,000	2,380,000
	1.5	69,620,000	2.21	0.12	2.29	3,390,000	270,000	3,510,000
	1.25	128,100,000	1.80	0.10	1.86	5,080,000	410,000	5,250,000
	1.0	203,590,000	1.53	0.09	1.59	6,870,000	590,000	7,140,000
	0.9	243,160,000	1.42	0.09	1.48	7,610,000	700,000	7,930,000
	0.8	317,940,000	1.27	0.09	1.33	8,900,000	920,000	9,320,000
	0.7	413,620,000	1.14	0.09	1.20	10,400,000	1,200,000	10,940,000
	0.6	490,330,000	1.05	0.09	1.11	11,350,000	1,420,000	12,000,000
	0.3	1,105,600,000	0.67	0.07	0.72	16,330,000	2,490,000	17,550,000

Table 14.23 Mineral Resource Inventory, Southern Oyu (15 April 2005)

Southern Oyu Deposits	CuEq Cut-off	Tonnage (t)	Cu (%)	Au (g/t)	CuEq (%)	Cu ('000 lb)	Au (oz)	CuEq ('000 lb)
Measured	1.0	59,550,000	0.77	1.55	1.76	1,011,000	2,970,000	2,311,000
	0.7	84,140,000	0.69	1.25	1.49	1,280,000	3,380,000	2,764,000
	0.6	101,590,000	0.65	1.09	1.34	1,456,000	3,560,000	3,001,000
	0.5	115,180,000	0.61	1.00	1.25	1,549,000	3,700,000	3,174,000
	0.4	123,440,000	0.59	0.95	1.20	1,606,000	3,770,000	3,266,000
	0.3	126,690,000	0.58	0.93	1.17	1,620,000	3,790,000	3,268,000
	0.25	127,550,000	0.58	0.92	1.17	1,631,000	3,770,000	3,290,000
	0.2	127,800,000	0.58	0.92	1.17	1,634,000	3,780,000	3,296,000
Indicated	1.0	102,330,000	0.85	0.82	1.38	1,918,000	2,700,000	3,113,000
	0.7	279,850,000	0.71	0.50	1.02	4,380,000	4,500,000	6,293,000
	0.6	430,830,000	0.63	0.40	0.89	5,984,000	5,540,000	8,453,000
	0.5	617,530,000	0.57	0.35	0.79	7,760,000	6,950,000	10,755,000
	0.4	827,050,000	0.51	0.30	0.70	9,299,000	7,980,000	12,763,000
	0.3	992,400,000	0.47	0.27	0.64	10,283,000	8,610,000	14,002,000
	0.25	1,067,830,000	0.45	0.26	0.61	10,594,000	8,930,000	14,360,000
	0.2	1,143,710,000	0.43	0.25	0.59	10,842,000	9,190,000	14,877,000
Measured+Indicated	1.0	161,880,000	0.82	1.09	1.52	2,926,000	5,670,000	5,425,000
	0.7	363,990,000	0.70	0.67	1.13	5,617,000	7,840,000	9,068,000
	0.6	532,420,000	0.64	0.54	0.98	7,512,000	9,240,000	11,503,000
	0.5	732,710,000	0.57	0.45	0.86	9,207,000	10,600,000	13,892,000
	0.4	950,490,000	0.52	0.38	0.76	10,896,000	11,610,000	15,926,000
	0.3	1,119,100,000	0.48	0.35	0.70	11,843,000	12,590,000	17,270,000
	0.25	1,195,370,000	0.46	0.33	0.67	12,123,000	12,680,000	17,657,000
	0.2	1,271,510,000	0.45	0.32	0.65	12,614,000	13,080,000	18,221,000
Inferred	1.0	3,750,000	0.91	0.48	1.22	75,000	60,000	101,000
	0.7	19,420,000	0.62	0.39	0.87	265,000	240,000	372,000
	0.6	47,390,000	0.51	0.35	0.74	533,000	530,000	773,000
	0.5	103,190,000	0.43	0.31	0.63	978,000	1,030,000	1,433,000
	0.4	181,700,000	0.38	0.26	0.55	1,522,000	1,520,000	2,203,000
	0.3	266,820,000	0.34	0.23	0.48	2,000,000	1,970,000	2,824,000
	0.25	318,380,000	0.32	0.21	0.45	2,246,000	2,150,000	3,159,000
	0.2	394,850,000	0.29	0.19	0.40	2,524,000	2,410,000	3,482,000

Notes to Table 14.21 to Table 14.23:

- The contained gold and copper estimates in the tables have not been adjusted for metallurgical recoveries.
- The Mineral Reserves are not additive to the Mineral Resources.
- CuEq has been calculated using assumed metal prices (\$1.35/lb. for copper and \$650/oz for gold and \$10/lb for molybdenum); $CuEq\% = Cu\% + ((Au\ g/t \cdot 18.98) + (Mo\ g/t \cdot 0.1586)) / 29.76$. Mo grades outside of Heruga are assumed to be zero for CuEq calculations
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The Mineral Resource estimates include estimates for trace and impurity elements including arsenic, fluorine, sulfur and silver as well as copper and gold.

- The Mineral Resource is inclusive of Reserves at Southern Oyu.
- The Effective Date for the Southern Oyu estimate was 15 April 2005 but this was re-tabulated with metal pricing assumptions more consistent with 2006.
- The difference in the Mineral Resource shown in this table at a 0.6% Cu Equivalent cut-off and the resource shown in Table 1.2 reflect that this table reports open pit mineable resource and has therefore been constrained by a pit shell.

14.3 Heruga Mineral Resources

The Mineral Resource estimate for the Heruga deposit was prepared by Stephen Torr of Ivanhoe Mines under the supervision of Scott Jackson of Quantitative Group. The estimates were in the form of three-dimensional (3D) block models utilizing commercial mine planning software (Datamine). The Project area limits and the block sizes used are shown in Table 14.24.

Table 14.24 Project Area Limits and Block Size

	Minimum	Maximum	Block Size(m)
X (UTM)	646500	648550	20
Y (UTM)	4757500	5759900	20
Z (AMSL)	-700	1250	15

14.3.1 Geologic Models

A close-off date of 21 June 2009 was used for the drillhole data used in the Heruga Resource update. The Effective Date for reporting the resource was 30 March 2010.

Ivanhoe created three dimensional shapes or wireframes of the major geological features of the Heruga deposit and these are listed in Table 14.25. The key geological features impacting on resource estimation are:

- Subvertical post-mineralization faults.
- Devonian host lithologies, primarily augite basalt and quartz monzodiorite.
- Poorly mineralized "late" quartz monzodiorite.
- Poorly mineralized hornblende-biotite-andesite and biotite-granodiorite dykes.

To assist in the estimation of grades in the model, Ivanhoe also manually created three dimensional grade shells (wireframes) for each of the metals to be estimated. Construction of the grade shells took into account prominent lithological and structural features, in particular the four major subvertical post-mineralization faults. For copper, a single grade shell at a threshold of 0.3% Cu was used. For gold, wireframes were constructed at thresholds of 0.3 g/t and 0.7 g/t. For molybdenum, a single shell at a threshold of 100 ppm was constructed. These grade shells took into account known gross geological controls in addition to broadly adhering to the abovementioned thresholds.

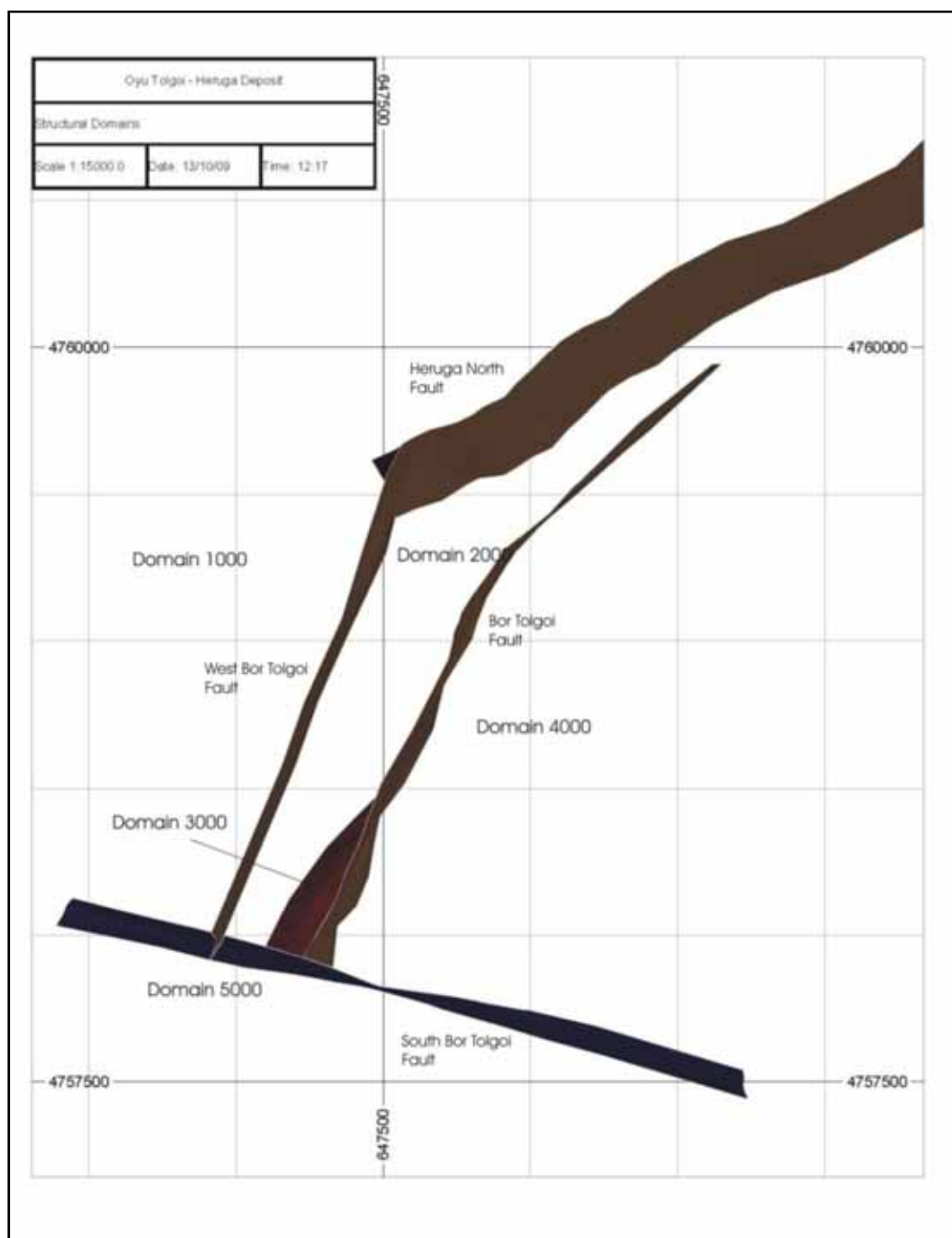
Table 14.25 Lithology and Structural Solids and Surfaces, Heruga Deposit

Surfaces – General
Topography
Solids / Surfaces – Lithology
Quartz Monzodiorite
Late Quartz Monzodiorite
Base of unmineralized volcanic and sedimentary units (DA2 or DA3 or DA4)
Biotite granodiorite (BiGd)
Hornblende-Biotite Andesite dykes (Hb-Bi An)
Surfaces – Faults
Bor Tolgoi Fault
West Bor Tolgoi Fault
Central Bor Tolgoi Fault
South Bor Tolgoi Fault
Heruga North Fault

QG checked the grade and mineralized shapes for interpretational consistency on section, in plan and in three dimensions, and found them to have been properly constructed. The shapes were found to honour the drill data and interpreted geology, and QG accepted them as an appropriate basis for the estimation process.

The solids and surfaces were used by Ivanhoe to code the drillhole data. All the drillhole intervals outside those shells were assigned to a background domain. A set of plans and cross-sections that displayed colour-coded drillholes was plotted and inspected to ensure the proper assignment of domains to drillholes. The faulting at Heruga is interpreted to have had considerable movement as demonstrated by the displacement of the overlying sedimentary and volcanic rocks. For this reason, the fault surfaces were used to define five separate structural domains for grade estimation. The five structural domains are shown in Figure 14.12.

Figure 14.12 Plan View of Heruga Structural Domains



14.3.2 Composites

The drillhole assays were composited into fixed-length, 5 m down hole composites, a size which was considered appropriate when taking into account estimation block size, required lithological resolution and probable mining method. This compositing honoured the domain zones by breaking the composites on the domain boundary. The domains used in compositing were a combination of the grade shells and lithological domains. Intervals less than 5 m in length represented individual residual composites from end-of-hole or end-of-domain intervals. Composites that were less than 2 m long were removed from the dataset that was used in interpolation.

The composites included any post-mineral dyke material intervals that were deemed too small to be part of a post-mineral dyke geology model. Any unsampled material included in the composites was set to 0.001% for copper and 0.01 g/t for gold and 10 ppm for molybdenum.

Figure 14.13 Plan View of Heruga Copper Grade Shell

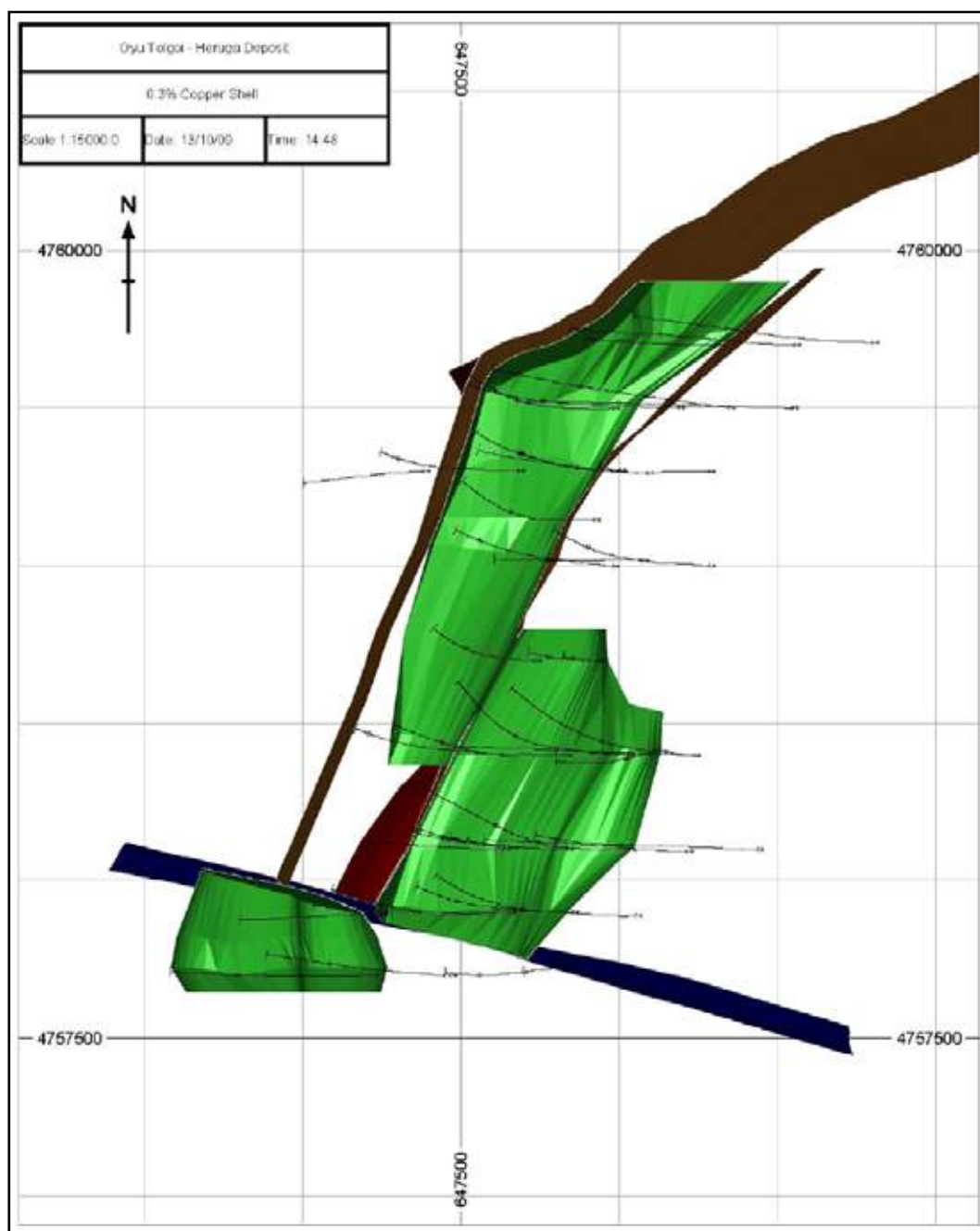


Figure 14.14 Plan View of Heruga Gold Grade Shells

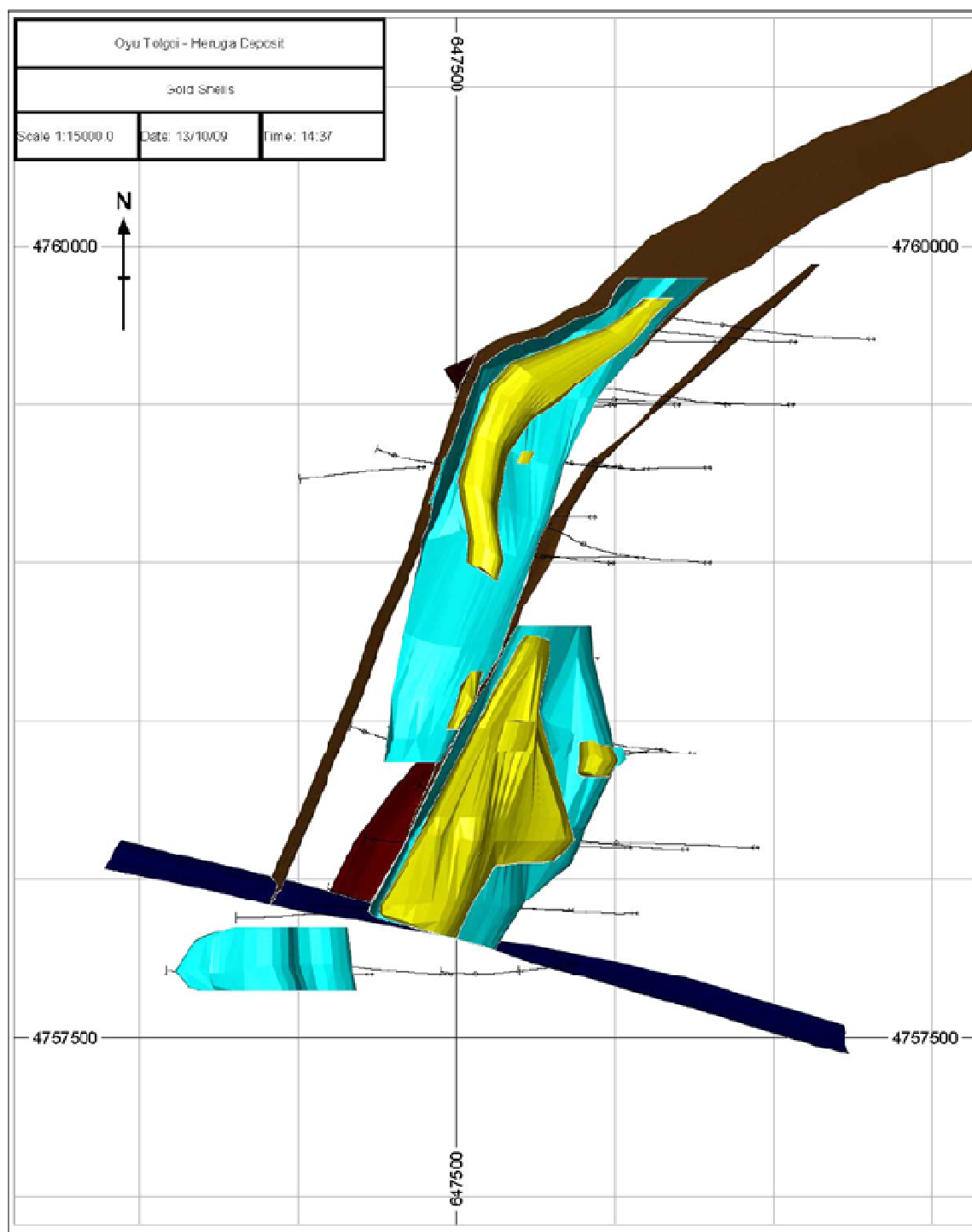
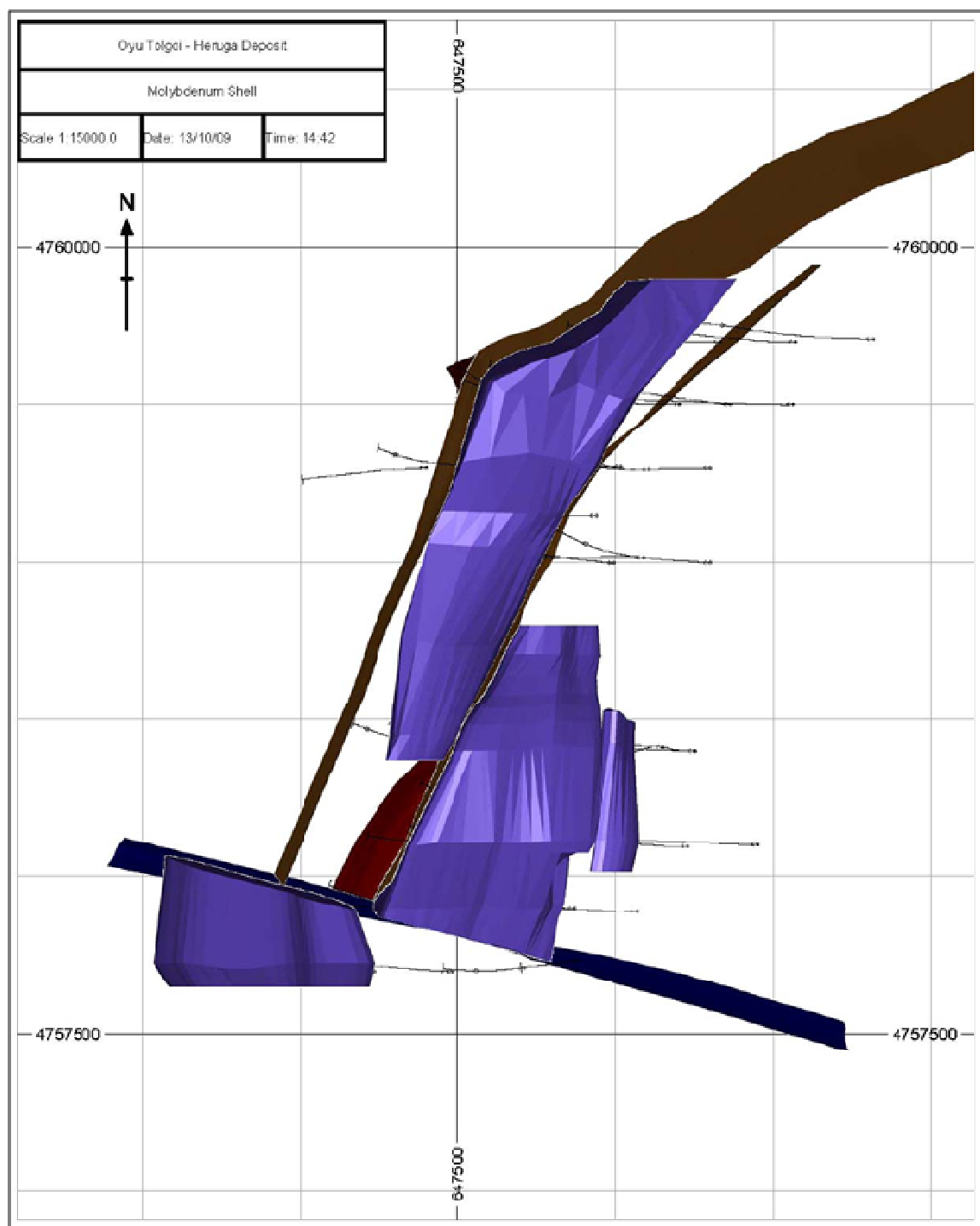


Figure 14.15 Plan View of Heruga Molybdenum Grade Shells



Bulk density data were assigned to a unique drillhole file and composited to honour lithological contacts on a 5 m interval.

14.3.3 Data Analysis

Data analysis was completed on both the raw data (original 2 m assays) and the 5 m composites. In both cases, analysis was done on individual lithologies inside and outside the grade shells, as well as independently of the grade shells. The relationship between grade and lithology was studied using descriptive statistics, box plots, histograms, CDFs, and grade contact plots.

14.3.3.1 Histograms and Cumulative Frequency Plots

Histograms and cumulative probability plots display the frequency distribution of a given variable and demonstrate graphically how frequency changes with increasing grade. With histograms, the grades are grouped into bins, and a vertical bar on the graph shows the relative frequency of each bin. Cumulative frequency or cumulative distribution function (CDF) diagrams demonstrate the relationship between the cumulative frequency (expressed as a percentile or probability) and grade on a logarithmic scale. They are useful for characterizing grade distributions and identifying whether or not there are multiple populations within a data set.

The statistics of the copper, gold and molybdenum are summarized in Table 14.26, Table 14.27, and Table 14.28 respectively, for the 5 m composites at Heruga. In these tables, CV = coefficient of variation (standard deviation/mean), a measure of relative variability. Extreme values or outliers were capped (or top-cut) prior to compositing the data which has reduced both the skewness and the CV of the populations.

Table 14.26 Heruga Statistics for 5 m Composites – Cu % Data

Lithology(*)	Cu Shell	Structural Domain	No. of Comps	Minimum	Maximum	Mean	CV
Va	Bkg	1000	114	0.001	0.37	0.03	1.55
Va	Bkg	2000	1,058	0.001	3.11	0.19	0.93
LQmd	Bkg	2000	245	0.001	0.17	0.01	1.73
Va + Qmd	0.3 %	2000	1,246	0.001	2.91	0.52	0.57
Va	Bkg	3000	92	0.001	0.67	0.06	2.54
Va	Bkg	4000	579	0.001	1.21	0.17	0.97
Va	0.3 %	4000	1,051	0.001	1.88	0.52	0.51
LQmd	0.3 %	4000	12	0.001	0.04	0.01	1.86
Va	Bkg	5000	175	0.001	0.51	0.10	0.85
Va	0.3 %	5000	253	0.001	1.53	0.35	0.54

(*) Va : basalt; Qmd: quartz monzodiorite; LQmd Late quartz monzodiorite.

Table 14.27 Heruga Statistics for 5 m Composites – Au g/t Data

Lithology(*)	Au Shell	Structural Domain	No. of Comps	Minimum	Maximum	Mean	CV
Va	Bkg	1000	116	0.01	1.36	0.10	1.72
Va	Bkg	2000	1,381	0.01	2.55	0.19	1.01
LQmd	Bkg	2000	208	0.01	0.87	0.04	2.55
Va + Qmd	0.3 g/t	2000	540	0.01	2.90	0.45	0.63
LQmd	0.3 g/t	2000	14	0.01	0.01	0.01	0.00
Va + Qmd	0.7 g/t	2000	385	0.01	7.46	1.21	0.88
LQmd	0.7 g/t	2000	29	0.01	0.27	0.02	2.53
Va	Bkg	3000	91	0.01	0.47	0.09	1.43
Va	0.3 g/t	3000	2	0.41	1.29	0.85	0.52
Va	Bkg	4000	978	0.01	2.05	0.17	0.89
LQmd	Bkg	4000	13	0.01	0.03	0.01	0.50
Va	0.3 g/t	4000	402	0.01	2.18	0.49	0.59
Va	0.7 g/t	4000	258	0.09	6.64	1.28	0.67
Va	Bkg	5000	307	0.01	0.93	0.16	0.80
Va	0.3 g/t	5000	120	0.01	0.93	0.34	0.50

(*) Va : basalt; Qmd: quartz monzodiorite; LQmd Late quartz monzodiorite.

Table 14.28 Heruga Statistics for 5 m Composites – Mo ppm Data

Lithology(*)	Mo Shell	Structural Domain	No. of Comps	Minimum	Maximum	Mean	CV
Va	Bkg	1000	116	10	19	10	0.09
Va	100 ppm	1000	1	10	10	10	0.00
Va	Bkg	2000	1,556	10	522	51	1.10
LQmd	Bkg	2000	251	10	43	11	0.38
Va + Qmd	100 ppm	2000	755	10	1,570	232	0.84
Va	Bkg	3000	93	10	274	21	1.86
Va	Bkg	4000	918	10	666	41	1.31
LQmd	Bkg	4000	13	10	10	10	0.00
Va	100 ppm	4000	713	10	1,391	208	0.90
Va	Bkg	5000	249	10	752	54	1.36
Va	100 ppm	5000	180	10	717	129	0.74

(*) Va : basalt; Qmd: quartz monzodiorite; LQmd Late quartz monzodiorite.

(1) Descriptive Statistics

Copper grades within the 0.3% Cu shell generally display single distributions with some evidence of a lower grade population due to the presence of unmineralized post mineral dykes that have not been captured by wireframes. Coefficients of variation (CV) values are relatively low at 0.5 to 0.6. Background domains for both Qmd (quartz-monzodiorite) and Va (augite basalt) show higher CVs, generally close to or above 1 and the CDF plots also show evidence of a higher grade population not captured by the 0.3% Cu

grade shells. The CDF plot for the entire population supports the construction of a grade shell in the 0.3% to 0.4% Cu range. There are no coherent zones of higher grade copper mineralization that are easily constrainable with the current drill coverage.

Gold grades were observed to display moderate positive skewness and multiple populations with evidence of lower grade populations in the 0.2 to 0.3 g/t range. Within the 0.3 g/t and 0.7 g/t gold domains the Va (augite basalt) and Qmd (quartz monzodiorite) display similar properties although the Qmd has a slightly higher mean of 1.31 g/t (vs. 1.21 g/t for Va) in the 0.7 g/t domain. CVs within the grade shells are generally low to moderate, in the 0.6 to 0.9 range; this increases to as high as 1.7 for some of the background domains. The high CV's in the background domains highlights the requirement for some outlier restriction during estimation.

Overall, the molybdenum grades within the 100 ppm shell display low to moderate positive skewness and single population distribution. Within individual structural domains there is some evidence of multiple populations with breaks at both 50 ppm (possible mixing with waste dykes) and 200 ppm. Augite basalt and Qmd show similar distributions. CVs are low to moderate, in the 0.6 to 0.8 range. The background domains are more positively skewed with somewhat higher CVs, in the 0.8 to 1.3 range.

14.3.3.2 Box-Plot and Contact Grade Profile Analyses

Contact profiles (or contact plots) were generated by Ivanhoe to explore the relationship between grade and:

- Lithological domains
- Grade shell domains

The rationale of the studies was to identify lithological or grade shell contacts that should be treated as hard during estimation. A domain with a hard contact will not share composites with other domains during grade interpolation.

(1) Results

Box-plots for copper showed that both within the grade shells and in the background domains there was little evidence of lithological control on copper grades. Augite basalt (Va) and quartz-monzodiorite (Qmd) have very similar overlapping distributions. The post mineral dykes and the late quartz-monzodiorite both show up as distinct, very low grade distributions. There are marked separations between the upper and lower quartiles of the distributions within and outside of the grade shells suggesting the distributions should be modelled separately.

Gold box plots show similar overlapping relationships for Va and Qmd to those seen for copper. The mean of the Qmd is slightly elevated compared to Va in the 0.7 g/t shell while the reverse is true in the 0.3 g/t shell. Post mineral dykes and the late Qmd show similar relationships to those observed for copper. As with copper, the distributions within the grade shells show a marked separation with those outside.

Molybdenum box-plots show the same relationships as those displayed for copper and gold.

Contact profile analysis confirmed the soft nature of the lithological contacts both inside and outside of the grade shell domains. As expected, contacts of the individual grade shells displayed a marked change in grade supporting the use of the grade shell boundaries as hard contacts. The late Qmd, post mineral dykes and sediment surface – all of which are to be treated as “waste” domains (zero grade) – showed sharp breaks in grade across the contact which were therefore treated as hard boundaries. There is some evidence of increasing grade across the Va Qmd contact inside of the grade shells but, at this stage, the contact was left as a soft contact for grade estimation.

14.3.3.3 Cross Correlation of Copper Golf and Molybdenum

The relationship between gold and copper was investigated through cross plots for both the main copper and gold estimation domains. Generally the correlations are poor with correlation co-efficients of 0.2 to 0.4 with a wide spread of data. The ratio of Gold to copper is generally 1:1. Inside the 0.7 g/t gold shell, there was clear evidence of two distinct populations, the first with a gold to copper ratio of 1:1 similar to the rest of the deposit, the second with a ratio of greater than 4:1. Spatially, this higher Au:Cu ratio distribution is associated with the deeper parts of the system and with Biotite alteration, a similar style of mineralization is seen in the high grade core of Southwest Oyu.

Copper shows a stronger relationship with molybdenum than with gold with correlation co-efficients in the 0.4 to 0.6 range. The overall ratio of Mo:Cu was of the order of 300 to 1 (ppm Mo to % Cu).

14.3.3.4 Estimation Domains

Data analysis showed no discernable difference between the two main host lithologies; augite basalt and quart-monzodiorite. For estimation purposes, therefore, these two lithologies were grouped into a single lithological domain.

It should be noted that the lack of demonstrated lithological control on grade distribution at this early stage in the development of the Project is not unexpected. Similar relationships were observed at Oyu Tolgoi, where the controls on mineralization became apparent only once detailed infill drilling was completed. However, there does appear to be a spatial association of higher copper and gold grades with the Qmd intrusion.

The post mineralization lithologies (LQmd, BiGd, HbBiAn), while represented in the block model, were not estimated but rather were assigned zero grade. Within each structural block, the model was therefore split according to whether or not it was mineralized or unmineralized and by grade shell. The final estimation domains used are shown in Table 14.29 to Table 14.31.

Table 14.29 Gold Estimation Domains – Mineralized Lithologies Only

SDM	GDM	Lithology	GDOMAIN	Description
1000	0	Va, Qmd	1000	Background
2000	0	Va, Qmd	2000	Background
2000	100	Va, Qmd	2100	0.3 g/t
2000	200	Va Qmd	2200	0.7 g/t
3000	0	Va, Qmd	3000	Background
4000	0	Va, Qmd	4000	Background
4000	100	Va, Qmd	4100	0.3 g/t
4000	200	Va, Qmd	4200	0.7 g/t
5000	0	Va, Qmd	5000	Background

Table 14.30 Copper Estimation Domains – Mineralized Lithologies Only

SDM	CDM	Lithology	CDOMAIN	Description
1000	0	Va, Qmd	1000	Background
2000	0	Va, Qmd	2000	Background
2000	100	Va, Qmd	2100	0.3%
3000	0	Va, Qmd	3000	Background
4000	0	Va, Qmd	4000	Background
4000	100	Va, Qmd	4100	0.3%
5000	0	Va, Qmd	5000	Background
5000	100	Va, Qmd	5100	0.3%

Table 14.31 Molybdenum Estimation Domains – Mineralized Lithologies Only

SDM	MDM	Lithology	MDOMAIN	Description
1000	0	Va, Qmd	1000	Background
2000	0	Va, Qmd	2000	Background
2000	100	Va, Qmd	2100	100 ppm
3000	0	Va, Qmd	3000	Background
4000	0	Va, Qmd	4000	Background
4000	100	Va, Qmd	4100	100 ppm
5000	0	Va, Qmd	5000	Background
5000	100	Va, Qmd	5100	100 ppm

14.3.3.5 Evaluation of Extreme Grades

Capping (or top-cutting) was applied to the raw assays prior to compositing. Various capping studies including decile analysis, CV plots and use of CDF plots were used to assign caps. As well as top cutting of extreme grades, some outlier restriction was also applied, particularly in the background domains.

Table 14.32 Grade Capping

Domain	Metal	Domain	Cap	Distance	Outlier Cap
Background	Au	1000 to 4000	3 g/t	50 m	1.0 g/t
Background	Au	5000	3 g/t	50 m	0.3 g/t
Background	Mo	All	1,000 ppm	100 m	500 ppm
0.3 Au Shell	Au	2000	3 g/t	–	–
0.3 Au Shell	Au	4000	5 g/t	–	–
0.7 Au Shell	Au	2000	10 g/t	–	–
100 Mo Shell	Mo	All	3,000 ppm	–	–

Top cutting was generally applied at values close to or above the 99th percentile for gold and molybdenum. No cap was felt warranted for copper. QG considers the approach taken to metal reduction was appropriate and applied reflecting the Inferred status of mineralization. A number of iterations of grade estimation were done and during these the outlier caps were lowered from those initially selected to better control potential smearing in the background domains. This was particularly the case in the 5000 domain where a quite aggressive outlier was applied to prevent excessive smearing of grades in the background.

14.3.4 Variography

Variography was undertaken by Ivanhoe on copper, gold and molybdenum for the capped and tagged composites using Visor software. Despite limited data Ivanhoe decided to attempt to model directional variograms for gold, copper and molybdenum. This approach differed to that used in 2008 where isotropic variograms were applied. For variography the Qmd and Va lithologies were combined and the raw data composited into 5 m lengths using the domain coding highlighted in Table 14.29 to Table 14.31.

Down hole variograms were initially modelled for each of the domains followed by three dimensional modelling of the variogram, this allowed accurate delineation of the nugget for each of the domains. Due to a lack of data, particularly in the strike plane, the 1st structure of the variogram model was mostly taken to be isotropic due to a lack of resolution in that distance range.

Table 14.33 Variogram Parameters

Model		Copper		Gold			Molybdenum	
		Bkg	0.3 %	Bkg	0.3 g/t	0.7 g/t	Bkg	100 ppm
		SPH	SPH	Exp	Exp	Exp	Exp	Exp
Sills	Nugget	0.32	0.4	0.33	0.37	0.25	0.41	0.41
	C1	0.27	0.37	0.42	0.2	0.53	0.23	0.37
	C2	0.41	0.23	0.25	0.43	0.22	0.36	0.32
Rotation Angles	Z	105	127	-153	180	-155	-160	-175
	X	49	68	-10	0	-70	-5	-9
	Y	168	26	80	-110	90	90	60
Ranges	Y1	80	40	50	15	250	100	100
	X1	80	40	50	15	40	100	100
	Z1	40	40	40	15	30	100	103
	Y2	447	352	250	200	450	418	461
	X2	510	274	200	200	180	445	265
	Z2	204	257	200	200	130	252	207

Notes: Models are spherical (SPH).

In defining the second structure, cognisance was taken of the variography observed in Southern Oyu and Hugo Dummett, where drill spacing is closer. In these deposits, variograms usually have elongated anisotropy both along strike and down dip of the mineralized zone. Because drilling at Heruga is perpendicular to the dip of the mineralized zone it is likely that the current isotropic 1st structure of the variogram models underestimate the range of continuity. Drilling direction is thought to lie between the y and z axis of the model variogram. This short isotropic first structure will have had an impact on the degree of smoothing in the model.

Copper and gold show relatively low nuggets of 25 to 35% of the total variance, molybdenum is moderate to high at 40% of the sill. All three metals show relatively short first structures and long second structures of 250 to 300 m. Due to the relatively wide drill spacing of 150 to 300 m between sections the first structure is essentially the down hole 1st structure which in reality is likely to be parallel to the z axis of the ellipse. In modelling the variograms cognisance was taken of the variography observed in Southern Oyu and Hugo Dummett, where drill spacing is closer. In these deposits, variograms usually have elongated anisotropy both along strike and down dip of the mineralized zone.

14.3.5 Model Setup

The block size selected was 20 m x 20 m x 15 m. This allowed consistency with previous modelling at the Oyu Tolgoi deposits and is considered a suitable block size for mining studies using the block cave approach.

To accurately account for the volume of post mineral dyke material, subcelling of the larger blocks was used when tagging the model with dyke wireframes. Each block was allowed to be split into four smaller cells in the x, y, and z dimensions. Subcells were

also permitted at grade shell boundaries to allow some smoothing of grades across the contact.

Various coding on the block models was performed by Ivanhoe in preparation for grade interpolation. The block model was coded according to zone, lithological domain, and grade shell. Post-mineral dykes and the late quartz-monzodiorite were assumed to represent zero-grade waste cutting the mineralized lithologies.

14.3.6 Estimation

Only the mineralized lithologies were estimated; i.e. Qmd and Va. All other units in the model were set to zero grade (hornblende-biotite-andesite dykes, biotite-granodiorite dykes, late quartz-monzodiorite intrusion and overlying sediments).

Primary grade interpolation for the three metals was by ordinary kriging of capped 5 m composites. As part of the model validation, grades were also interpolated using nearest neighbour, ID3 and ordinary kriging of uncapped composites. Blocks and composites were matched on specific domain key fields that ensured hard boundaries were respected between structural domains and between grade shells. Only full cell estimation was used during kriging, i.e. subcelling was used for geometry/volume purposes only.

The search ellipsoids were oriented preferentially to the general trend of the grade shells. Studies at other Oyu Tolgoi deposits have shown that the anisotropy of variograms tends to mirror the shape of the grade shells. There are many similarities between Heruga and Hugo Dummett in terms of structural geological setting that support using a similar approach at Heruga.

A staged search strategy was applied by Ivanhoe with the first pass at 200 m, and a second at 400 m. A minimum two hole rule was applied to both passes. Any blocks not interpolated by the first two passes were filled with a third pass that removed the two hole constraint. Table 14.34 shows the search strategy. The same ellipse was applied to all metals. QG independently ran additional checks to confirm that the search strategy was appropriate.

Table 14.34 Search Ellipsoids for Heruga

	Z	X	Y	XDIST	YDIST	ZDIST	Min Comps	Maximum Comps	Maximum DDH
1st Pass	-65	-80	0	200	200	100	8	30	6
2nd Pass	-65	-80	0	400	400	200	8	30	6
3rd Pass	-65	-80	0	400	400	200	6	30	6

(1) Outlier Restriction

To avoid excessive smearing of isolated high grades, particularly in the background domains, outlier restriction was applied during kriging. Outlier restriction was applied as a second cap whereby grades over a particular threshold were only used in blocks within a specified distance from a drillhole (50 m to 100 m). Outside of this distance the

lower capped value was used. The caps and the outlier distances used are shown in Section 14.3.3.5.

(2) Bulk Density

Bulk density values were interpolated into the model using ID³ estimation. The ranges and the rotation angles for the various search ellipsoids are highlighted in Table 14.35. As with copper, gold and molybdenum, a concentric search method was applied.

Table 14.35 Bulk Density Search Ellipsoids for Heruga

		All Domains
Rotation Angles	Z	-65
	X	-80
	Y	0
	Y	200
	X	200
	Z	100
Number of Comps	Min	5
	Max	20
	Max DDH	4

The process utilized lithological domaining, since statistical analysis showed bulk density variation is primarily controlled by host lithology. Any blocks in the model that were not interpolated were assigned a mean density based on rock type. The mean bulk density of each lithology is shown in Table 14.36.

Table 14.36 Average Bulk Density

Lithology	Bulk Density
Va	2.86
QMD	2.73
Late QMD	2.75
Sediment	2.81
Hornblende-biotite andesite dyke	2.75
Biotite granodiorite dyke	2.73

(3) Full Cell Model

The final subcell model was regularized to a full cell model after estimation was complete. Subcells were combined into full cells using volume weighting. Regularization results in smoothing across grade shell boundaries and the incorporation of internal dilution from post mineral dykes where full model cells contained dyke subcells. In the full cell model the lithology is regenerated based upon majority coding by volume. This means blocks can be majority coded as post mineral dyke but may contain some grade.

14.3.6.1 Validation

(1) Visual Inspection

A detailed visual validation of the Heruga resource model was undertaken by Ivanhoe and QG and it was found that tagging of the drill data file and the block model was done correctly. The model was also checked in plan and section to ensure that the grade interpolation accurately reflected the original drill assays. The grade shells appear to have adequately constrained the high values and no evidence of excessive smearing of high grades in the background was observed. In some areas the model showed evidence of nearest neighbour striping due to a lack of data although these areas were excluded when tagging the resource model for classification.

QG also built a model from scratch using the same wireframes and drill data used in the Ivanhoe model. Gold, copper and molybdenum were interpolated using independently generated variograms and search parameters. QG compared the two estimates and consider they are well within acceptable limits thus adding additional support to the estimate built by Ivanhoe. Minor differences noted in the resultant models were attributed to slight differences in variograms and search parameters.

(2) Model Checks for Bias

The block model estimates were checked for global bias by comparing the average metal grades from the model with means from unrestricted nearest-neighbour estimates. (The nearest-neighbour estimator declusters the data and produces a theoretically unbiased estimate of the average grade when no cut-off grade is imposed and is a good basis for checking the global performance of different estimation methods). Results, summarized in Table 14.37, show no problems with global bias in the estimates.

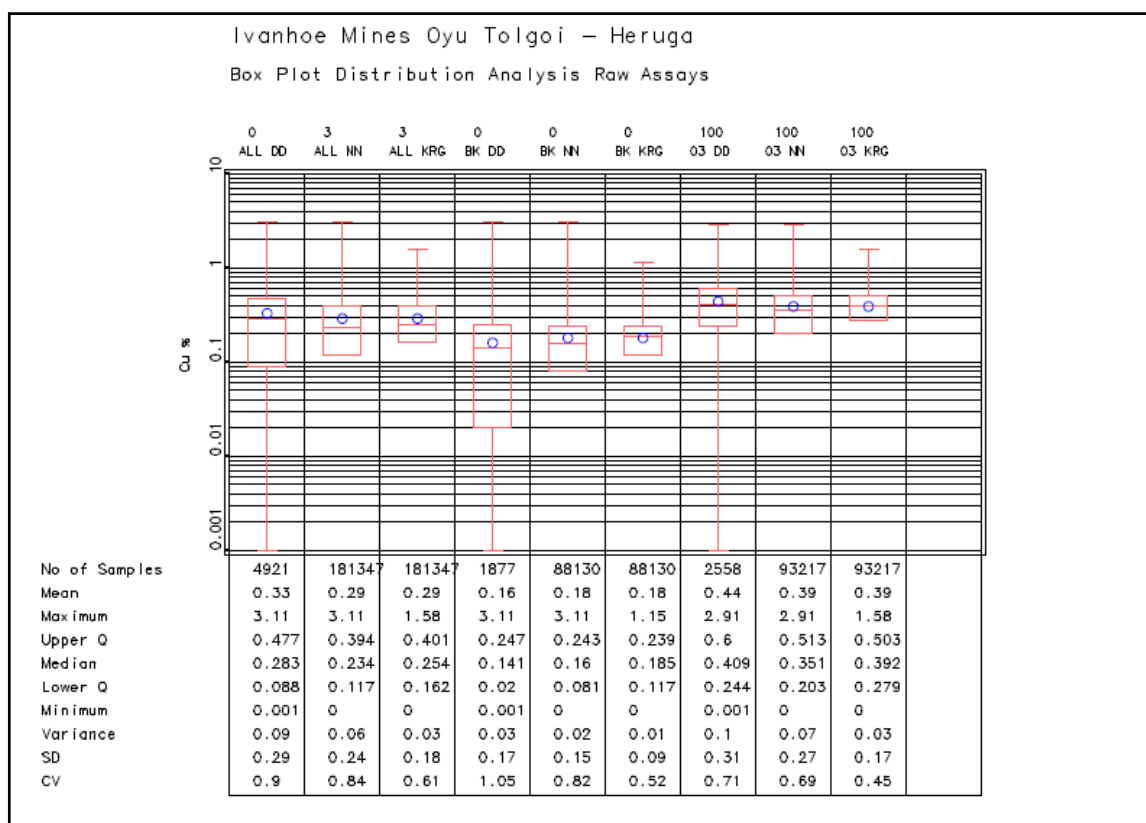
Table 14.37 Model Mean Grade Values by Domain in Each Zone –Inclusive of Post Mineral Dyke

Domain/Zone	Kriged Estimate	Nearest Neighbour Estimate	Unrestricted Kriged Estimate	Metal Reduction
Copper				
Background	0.18	0.18	0.18	0%
0.3% Shell	0.39	0.39	0.39	0%
All Blocks	0.29	0.29	0.29	0%
Gold				
Background	0.15	0.16	0.18	-16 %
0.3 g/t Au Shell	0.36	0.35	0.36	-2%
0.7 g/t Au Shell	0.94	0.92	1.0	-5%
All Blocks	0.27	0.27	0.29	-9%
Molybdenum				
Background	46	46	48	-3 %
100 ppm shell	165	163	165	0%
All Blocks	83	83	85	-1%

(3) Distribution Comparisons

The distribution of the grades in the model was compared to the distribution of the original drillhole data, the composites used to build the model and the declustered nearest neighbour model. The distributions were compared using box and whisker plots as well as cumulative distribution plots and lognormal and normal histograms. In all cases, the model was found to reflect the underlying data used to build it albeit it a more smooth distribution, this smoothing is expected and is a natural function of kriging. The degree of smoothing occurring within the model was considered reasonable for the type of deposit and the likely mining method (Block Caving). Figure 14.16 shows an example of the comparison of distributions for the copper domains, the entire model, the nearest neighbour model and the kriged estimate. The reduction in grade from composites to nearest neighbour reflects both declustering of data and the effect of incorporating post mineral dyke into the final estimate.

Figure 14.16 Copper Boxplots



(4) Local Bias Checks

The resource model was also checked for trends and local bias using 50 m swath plots that compared the restricted kriged estimates to nearest neighbour estimates. The nearest neighbour estimates act as an unbiased de-clustered sample population and comparison should highlight areas of potential bias in the kriged estimate. The plots also display the number of model blocks in each 50 m swath.

Figure 14.17 Gold Swath Plots

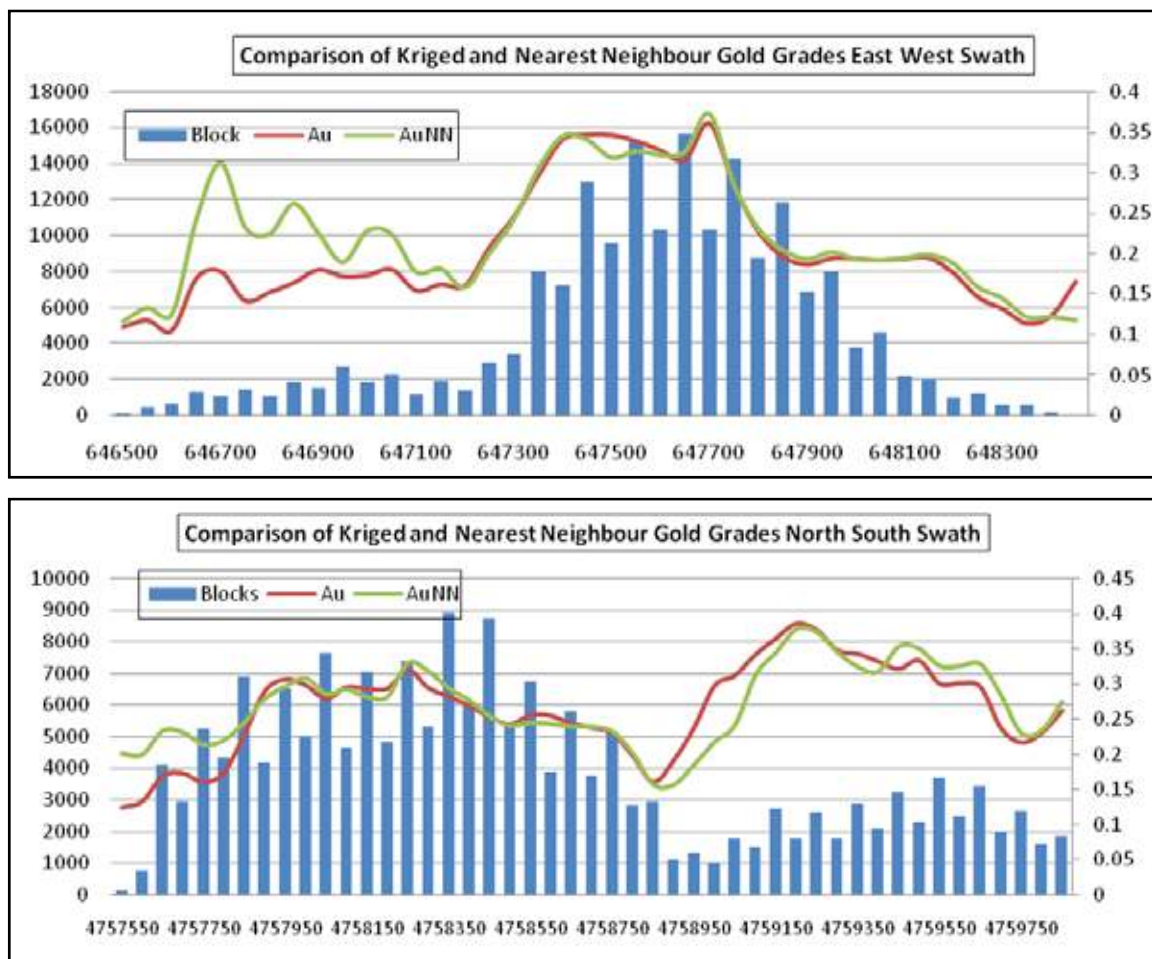


Figure 14.18 Copper Swath Plots

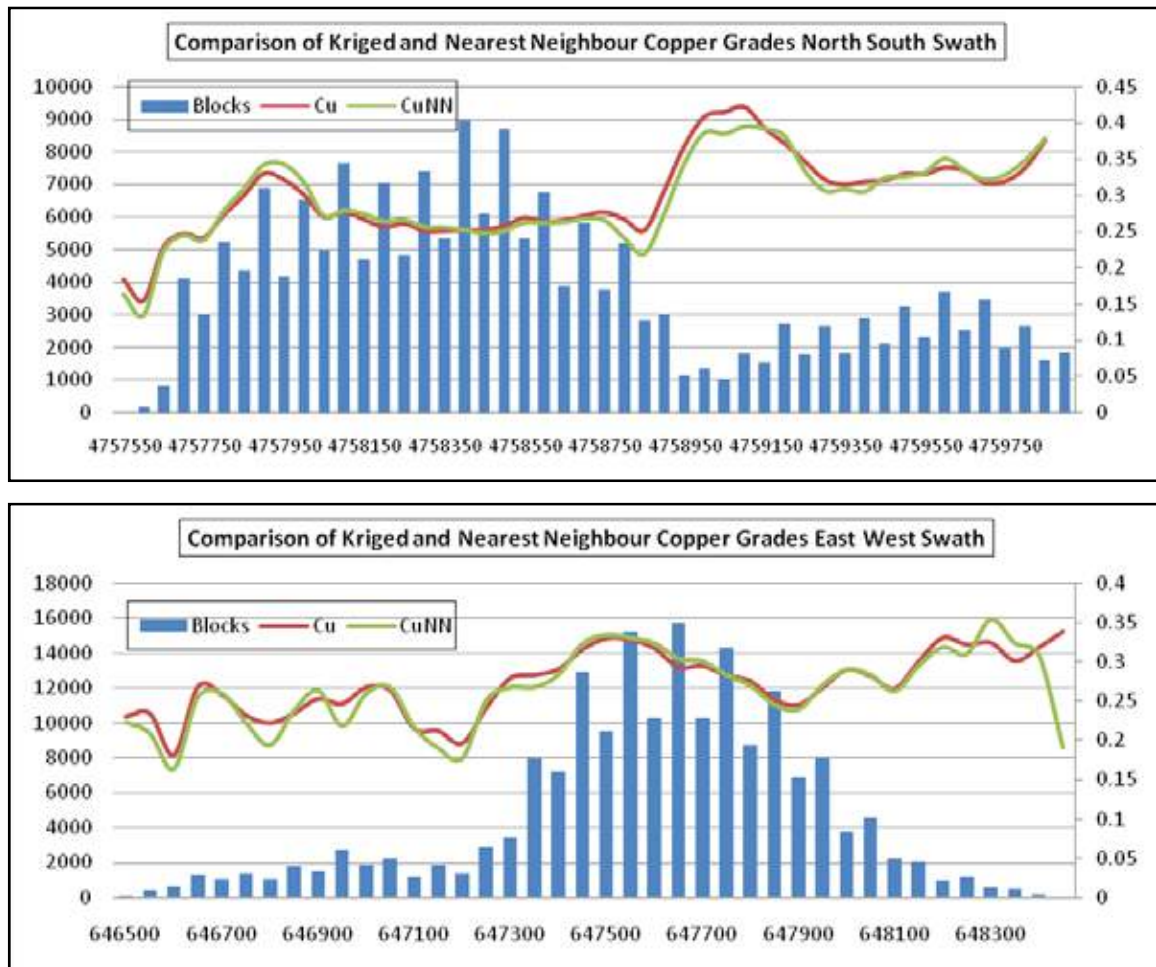
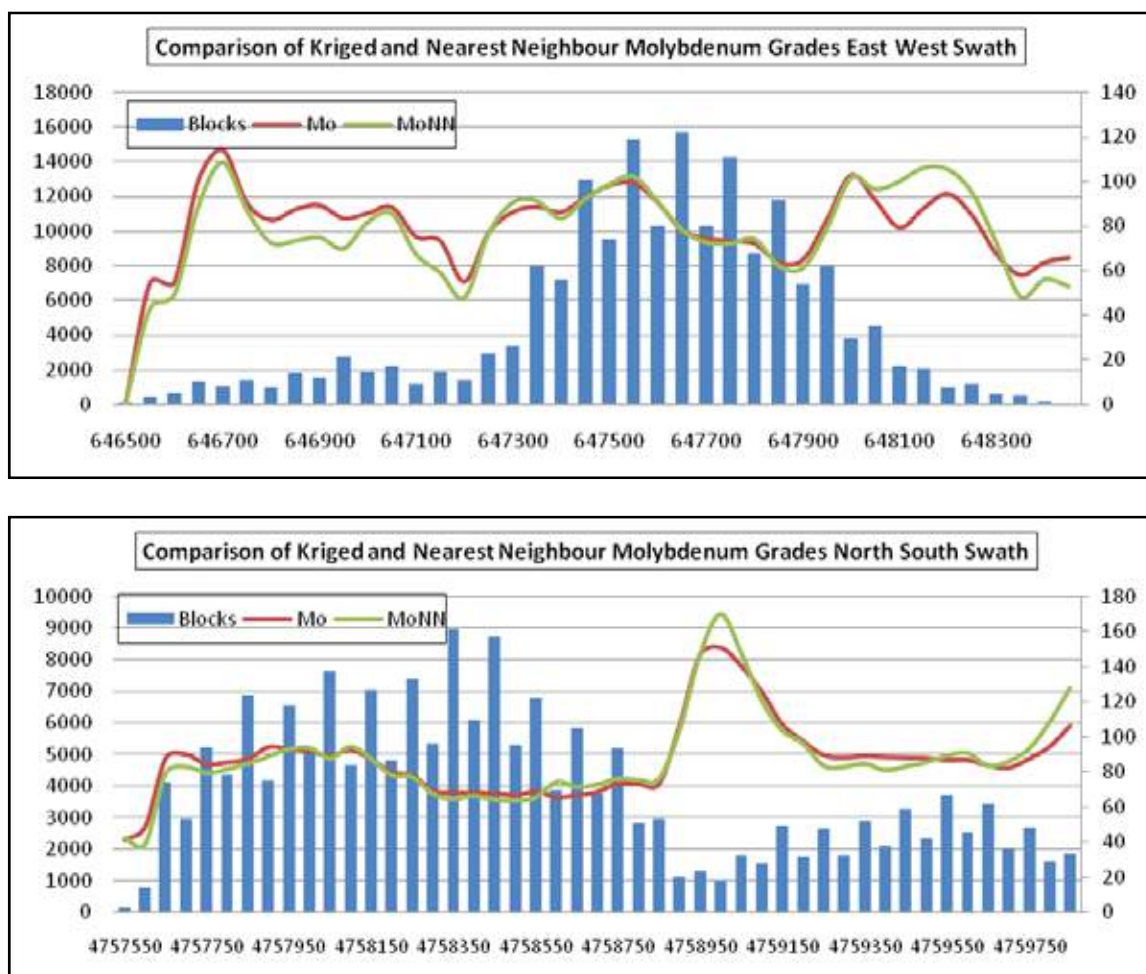


Figure 14.19 Molybdenum Swath Plots



14.3.7 Heruga Mineral Resource Classification

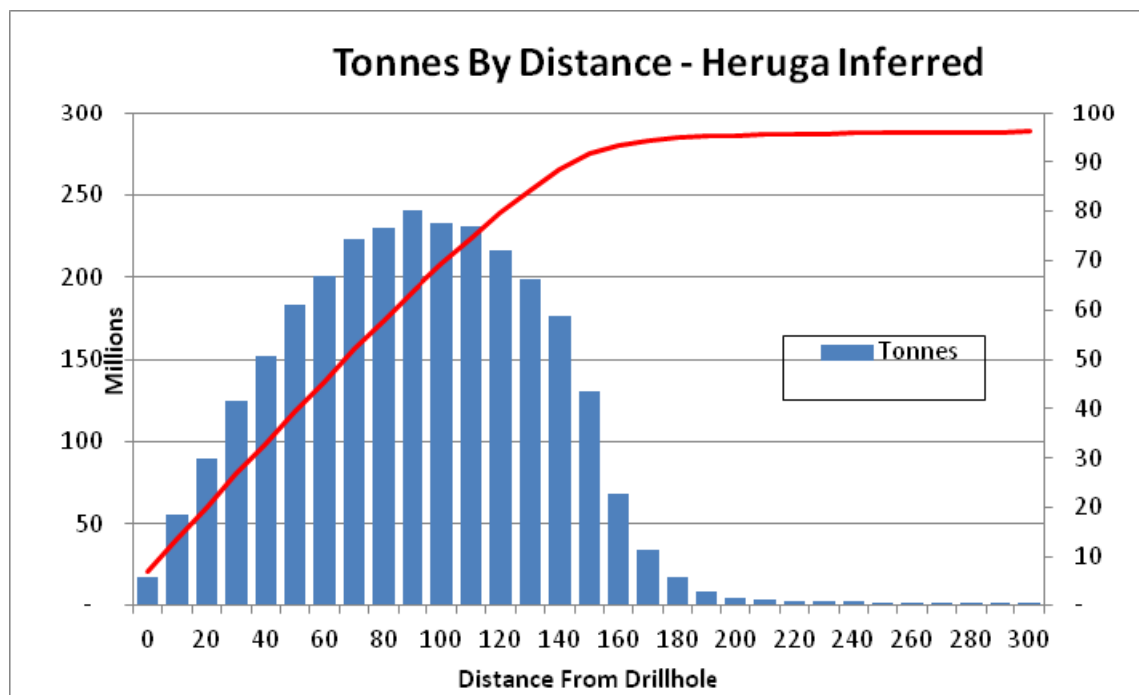
The Mineral Resources of the Heruga deposit were classified using logic consistent with the CIM definitions referred to in NI 43-101. At present, the mineralization of the Oyu Tolgoi Project satisfies sufficient criteria to be classified as an Inferred Resource.

14.3.7.1 Inferred Mineral Resources

Blocks within 150 m of a drillhole were initially considered to be Inferred. A three dimensional wireframe was constructed inside of which the nominal drill spacing was less than 150 m. The shape aimed to remove isolated blocks around drillholes where continuity of mineralization could not be confirmed. Within the 150 m shape there were a small number of blocks that were greater than 150 m from a drillhole. These were included because it was considered that geological and grade continuity could be reasonably inferred within the main part of the mineralized zone. The average distance of all the Inferred blocks in the resource model is displayed in the plot below. Of the total tonnes classified as Inferred, approximately 92% are within 150 m of a drillhole while the average distance of the Inferred blocks is approximately 95 m.

QG is satisfied that the resultant Inferred Resources are reasonable and meet the criteria set out in the CIM definitions referred to in NI 43-101.

Figure 14.20 Total Inferred Resource Tonnes by Distance in Heruga



14.3.8 Heruga Mineral Resource Summary

The total Heruga deposit Mineral Resource inventory, based on drilling as of 21 June 2009 at various copper equivalent (CuEq) cut-offs is shown in Table 14.38.

The base case reporting CuEq cut-off grade of 0.6% CuEq was determined using cut-off grades applicable to mining operations exploiting similar deposits and is consistent with the cut-offs applied to Hugo North and Hugo South. At Heruga, the presence of molybdenum in potentially economic concentrations has led to a revision of the CuEq formula used for the other Oyu Tolgoi deposits.

Table 14.38 Copper Equivalent Assumptions

	Copper	Gold	Molybdenum
Price	\$1.35 lb	\$650 oz	\$10 lb
Recovery relative to Cu Recovery	1.0	0.908	0.719

Note: The recovery stated is not absolute recovery but recovery relative to copper recovery. The gold recovery factor is based on test work at Oyu Tolgoi while Molybdenum is based on the operational Mo recoveries at other similar Gold-Copper-Molybdenum porphyry deposits.

$$\text{CuEq}\% = \text{Cu}\% + ((\text{Au g/t} \times 18.98) + (\text{Mo g/t} \times 0.01586)) / 29.76$$

While the equivalence formula accounts for differential recoveries, the contained metal shown in the table is in situ metal.

Table 14.39 shows the inventory for material on the Entrée JV property (Javkhlant), Table 14.40 shows the material on the 100% OT LLC property (Oyu Tolgoi) and Table 14.41 gives the total Heruga inventory.

Table 14.39 Mineral Resource Inventory, Heruga Deposit – Entrée JV Only, 30 March 2010

Cut-off(CuEq %)	Tonnage('000 t)	Cu(%)	Au(g/t)	Mo(pp m)	CuEq (%)	Contained Metal		
						Cu ('000 lb)	Au ('000 oz)	CuEq ('000 lb)
>1.50	30,000	0.57	1.86	124	1.83	360,000	2,000	1,150,000
>1.25	70,000	0.56	1.45	118	1.55	840,000	3,000	2,340,000
>1.00	190,000	0.57	0.96	155	1.26	2,370,000	6,000	5,260,000
>0.90	290,000	0.56	0.80	160	1.15	3,610,000	8,000	7,450,000
>0.80	450,000	0.54	0.66	160	1.05	5,310,000	10000	10,320,000
>0.70	660,000	0.51	0.56	151	0.95	7,390,000	12,000	13,780,000
>0.60	910,000	0.48	0.49	141	0.87	9,570,000	14,000	17,390,000
>0.50	1,210,000	0.44	0.44	130	0.79	11,780,000	17,000	21,060,000
>0.40	1,670,000	0.39	0.38	115	0.69	14,430,000	20,000	25,540,000
>0.30	2,180,000	0.35	0.33	102	0.61	16,730,000	23,000	29,470,000

Table 14.40 Mineral Resource Inventory, Heruga Deposit – OT LLC 100% Only, 30 March 2010

Cut-off(CuEq %)	Tonnage('000 t)	Cu(%)	Au(g/t)	Mo(pp m)	CuEq (%)	Contained Metal		
						Cu ('000 lb)	Au ('000 oz)	CuEq ('000 lb)
>1.50	–	0.58	1.46	52	1.54	–	–	–
>1.25	–	0.53	1.28	52	1.37	10,000	–	40,000
>1.00	10,000	0.54	0.86	107	1.15	70,000	–	140,000
>0.90	10,000	0.54	0.68	131	1.04	130,000	–	250,000
>0.80	20,000	0.53	0.55	136	0.95	250,000	–	450,000
>0.70	40,000	0.50	0.43	133	0.85	460,000	1 000	780,000
>0.60	60,000	0.48	0.37	128	0.78	670,000	1 000	1,090,000
>0.50	90,000	0.45	0.33	122	0.72	840,000	1 000	1,360,000
>0.40	110,000	0.41	0.29	111	0.66	1,020,000	1 000	1,640,000
>0.30	130,000	0.38	0.27	105	0.62	1,110,000	1 000	1,770,000

Table 14.41 Mineral Resource Inventory, Heruga Deposit – Total, 30 March 2010

Cut-off(CuEq %)	Tonnage('000 t)	Cu(%)	Au(g/t)	Mo(pp m)	CuEq (%)	Contained Metal		
						Cu ('000 lb)	Au ('000 oz)	CuEq ('000 lb)
>1.50	30,000	0.57	1.86	124	1.83	360,000	2,000	1,160,000
>1.25	70,000	0.56	1.45	117	1.55	860,000	3,000	2,380,000
>1.00	190,000	0.57	0.96	153	1.26	2,430,000	6,000	5,400,000
>0.90	300,000	0.56	0.79	159	1.15	3,730,000	8,000	7,700,000
>0.80	470,000	0.54	0.66	158	1.04	5,560,000	10,000	10,770,000
>0.70	700,000	0.51	0.56	150	0.94	7,850,000	13,000	14,560,000
>0.60	970,000	0.48	0.48	140	0.86	10,240,000	15,000	18,480,000
>0.50	1,300,000	0.44	0.43	130	0.78	12,620,000	18,000	22,420,000
>0.40	1,780,000	0.39	0.37	114	0.69	15,450,000	21,000	27,180,000
>0.30	2,310,000	0.35	0.33	102	0.61	17,840,000	24,000	31,250,000

Notes:

- Based on drilling completed as of 21 June 2009 and reported by Ivanhoe 30 March 2010.
- CuEq was calculated using the formula $\text{CuEq\%} = \text{Cu\%} + ((\text{Au g/t} \times 18.98) + (\text{Mo g/t} \times 0.01586)) / 29.76$.
- The contained gold, copper and molybdenum in the tables has not been adjusted for recovery.
- The 0.6% CuEq cut-off is highlighted as the base case resource for underground bulk mining.

14.4 References

The information for this section is in the report Oyu Tolgoi Technical Report, March 2008.

15 MINERAL RESERVE ESTIMATES

Total Mineral Reserves for the Project and the OT LLC and EJV Mineral Reserves for the open pit and underground components of the Project are shown in Table 15.1. The Mineral Reserves for IDOP Technical Report are based on mine planning work prepared by OT LLC as presented in IDOP. This work was reviewed extensively and has been used as the basis for reporting the IDOP Technical Report Mineral Reserves.

The mine design work on the Southern Oyu open pits was prepared by OT LLC for IDOP and was used as the basis for the March 2012 Southern Oyu open pit Mineral Reserve by AMC. The mine design work on Hugo North Lift 1 was prepared by Stantec for IDP10 was used as the basis for the 2010 Hugo North Underground Mineral Reserve. This work was reviewed by OT LLC and accepted as the basis for the underground mine planning in IDOP. AMC has agreed with this conclusion and has reported the results for the March 2012 Hugo North Underground Mineral Reserve estimate.

The Mineral Reserves for the Project have been estimated using the Southern Oyu and Hugo North Mineral Resources. The Hugo North Mineral Reserve contains ore that is on the OT LLC Oyu Tolgoi licence and on the EJV Shivee Tolgoi licence. The EJV Shivee Tolgoi licence is subject to a joint venture agreement between OT LLC and Entrée Gold.

Mineral Reserves were first reported in the IDP10 Technical Report June 2010. A reconciliation of the IDP10 and IDOP Mineral Reserves is shown in Table 15.2. Although the total tonnes of the Mineral Reserve sets are very similar there is a 2.4% increase in Cu grade and a 5.1% decrease in gold grade these are matched by a 1.5% increase in recoverable copper and a 5.6% decrease in recoverable gold. Almost all of the difference is in the Southern Oyu open pit designs. The pit optimization work on Southern Oyu is ongoing and has not yet defined the final pit limits. High grade resources remain in the floor of the current pit design in Southwest that would be expected to be included in updated pit designs. Calculation of the underground Mineral Reserve by OT LLC resulted in a rounding up of the Hugo North Lift 1 tonnage on the Oyu Tolgoi licence by 1 Mt to 411 Mt and on the EJV licence the same tonnage but increased grades.

Table 15.1 Total Oyu Tolgoi Project Mineral Reserve, 29 March 2012

Deposit	Ore (Mt)	NSR (\$/t)	Cu (%)	Au (g/t)	Recovered Metal	
					Copper (Mlb)	Gold (koz)
Southern Oyu Deposits						
Proven	125	36.42	0.58	0.91	1,374	2,892
Probable	833	18.48	0.48	0.23	7,048	4,522
Mineral Reserve (Proven + Probable)	957	20.82	0.49	0.32	8,422	7,414
Hugo Dummett Deposits						
Probable (Hugo North – OT LLC)	411	73.18	1.94	0.40	15,904	4,407
Probable (Hugo North – EJV Shivee Tolgoi)	27	79.40	1.91	0.74	1,043	536
Mineral Reserve (Probable) (All Hugo North)	438	73.57	1.93	0.42	16,947	4,943
Oyu Tolgoi Project Mineral Reserve						
Proven	125	36.42	0.58	0.91	1,374	2,892
Probable	1,270	37.46	0.98	0.30	23,995	9,465
Mineral Reserve (Proven + Probable)	1,395	37.37	0.94	0.35	25,369	12,357

Notes:

1. Metal prices used for calculating the Southern Oyu open pit NSR and the Hugo North Underground NSR are copper \$1.80/lb, gold \$750/oz, and silver \$12.00/oz based on long term metal price forecasts at the beginning of the Mineral Reserve work. Analysis indicates that the Mineral Reserve is still valid at these metal prices.
2. The NSR has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
3. For the open pit processing and general and administration operating costs have been used to determine cut-off grades are: Southwest \$4.23/t, Central Chalcocite and Central Covellite \$3.85/t and Central Chalcopyrite \$4.03/t
4. For the underground block cave, all Mineral Resource within the shell has been converted to Mineral Reserve this includes low grade Indicated Mineral Resource and Inferred Mineral Resource assigned zero grade treated as dilution.
5. Only Measured Resources were used to report Proven Reserves and only Indicated Resources were used to report Probable Reserves.
6. The base case financial analysis has been prepared using current long term metal price estimates of copper \$2.85/lb, gold \$1200/oz, and silver \$16.60/oz. Metal prices are assumed to fall from current prices to the long term average over five years.
7. The Mineral Reserves are not additive to the Mineral Resources.
8. EJV is the Entrée Gold Joint Venture. The Shivee Tolgoi and Javkhlant licences are held by Entrée Gold. The EJV Shivee Tolgoi and EJV Javkhlant Licences are planned to be operated by OT LLC. OT LLC will receive 80% of cash flows after capital and operating costs for material originating below 560 m, and 70% above this depth. The Mineral Reserves are not additive to the Mineral Resources.

Table 15.2 Mineral Reserve Reconciliation IDP10 and IDOP Technical Report

	Mineral Reserve	Ore (Mt)	NSR (\$/t)	Cu (%)	Au (g/t)	Recovered Metal	
						Copper (M lb)	Gold (koz)
IDP10	Proven	127	21.38	0.58	0.93	1,399	2,994
	Probable	1,266	24.84	0.97	0.32	23,835	10,127
	Mineral Reserve	1,393	24.52	0.93	0.37	25,234	13,121
IDOP	Proven	125	36.42	0.58	0.91	1,374	2,892
	Probable	1,270	37.46	0.98	0.30	23,995	9,465
	Mineral Reserve	1,395	37.37	0.94	0.35	25,369	12,357
Difference	Proven	-2	15.04	0.00	-0.02	-25	-102
	Probable	4	12.62	0.01	-0.02	160	-662
	Mineral Reserve	2	12.85	0.01	-0.02	135	-764

15.1.1 Southern Oyu Open Pit Mineral Reserve

In order to estimate the Mineral Reserves, AMC relied on the study work prepared by OT LLC. Pit designs were prepared using industry standard mining software, assumed metal prices as described above and smelter terms as set forth in the IDOP Technical Report. The estimate was prepared on a simplified project analysis on a pre-tax basis. Key outstanding variables noted by AMC include: marketing matters, water supply and management, underground ventilation development and power supply. The report only considers Mineral Resources in the Measured and Indicated categories, and engineering that has been carried out to a pre-feasibility level or better to estimate the open pit Mineral Reserve.

15.1.2 Hugo North Underground Mineral Reserve

Mine planning work by OT LLC has continued during since the previous Mineral Reserve estimate in 2010 but has not indicated any significant change in the Mineral Reserve.

The Hugo Dummett underground deposit will be mined by block caving; a safe, highly productive, cost-effective method. The deposit is comparable in dimension and tonnage to other deposits currently operating by block cave mining elsewhere in the world. The mine planning work has been prepared using industry standard mining software, assumed metal prices as described above and smelter terms as set forth in the IDOP Technical Report. The estimate was prepared on a simplified project analysis pre-tax basis. The Hugo North Mineral Reserve only considers Mineral Resources in the Indicated category because there are no Measured Resources in Hugo North. The engineering has been carried out to a pre-feasibility level or better to estimate the underground Mineral Reserve. To ensure that Inferred Resources do not become included in the reserve estimate, copper and gold grades on Inferred Resources within the block cave shell were set to zero and such material was assumed to be dilution. The block cave shell was defined by a \$20/t NSR. Further mine planning will examine lower shut-offs. The Hugo North Mineral Reserve is on both the OT LLC Oyu Tolgoi licence and the EJV Shivee Tolgoi licence.

15.2 Mineral Reserves by Licence

The Hugo North Mineral Reserve contains ore covered by two licence areas: OT LLC Oyu Tolgoi and EJV Shivee Tolgoi. The EJV Shivee Tolgoi licence is subject to a joint venture agreement between OT LLC and Entrée Gold.

15.3 Key Mining Assumptions

The Mineral Reserves were estimated by AMC. Key assumptions are summarized below; other assumptions are documented in the IDOP Technical Report:

- Metal prices used for calculating the Southern Oyu open pit NSR and the Hugo North Underground NSR are copper \$1.80/lb, gold \$750/oz, and silver \$12.00/oz based on long term metal price forecasts at the beginning of the Mineral Reserve work. Analysis indicates that the Mineral Reserve is still valid at these metal prices.
- The NSR has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
- For the open pit processing and general and administration operating costs have been used to determine cut-off grades are: Southwest \$4.23/t, Central Chalcocite and Central Covellite \$3.85/t and Central Chalcopyrite \$4.03/t
- For the underground a footprint cut-off of \$20 and column height shutoff of \$20 to maintain grade and productive capacity.

- For the underground block cave, all Mineral Resource within the shell has been converted to Mineral Reserve this includes low grade Indicated Mineral Resource and Inferred Mineral Resource assigned zero grade treated as dilution.
- Only Measured Resources were used to report Proven Reserves and only Indicated Resources were used to report Probable Reserves.
- The base case financial analysis has been prepared using current long term metal price estimates of copper \$2.85/lb, gold \$1200/oz, and silver \$16.60/oz. Metal prices are assumed to fall from current prices to the long term average over five years.
- EJV is the Entrée Gold Joint Venture. The Shivee Tolgoi and Javkhlant licences are held by Entrée Gold. The EJV Shivee Tolgoi and EJV Javkhlant Licences are planned to be operated by OT LLC. OT LLC will receive 80% of cash flows after capital and operating costs for material originating below 560 m, and 70% above this depth. The Mineral Reserves are not additive to the Mineral Resources.
- The Mineral Reserves are not additive to the Mineral Resources.
- The Mineral Resource block models used for reporting the Mineral Reserves are the models reported by QG in the Mineral Resource section of IDOP Technical Report.
- The processing schedule philosophy adopted for the mine planning work assumes feeding the open pit ore into the plant at an elevated cut off grade and stockpiling low grade material for later treatment. This philosophy provides some insulation against metal price cycles and reduces the risk that the Mineral Reserve size is overestimated.

15.4 US SEC Industry Guide 7

The Mineral Reserve reported for NI 43-101 is also applicable for reporting the Ore Reserve under the US SEC Industry Guide 7. AMC estimated the Oyu Tolgoi Mineral Reserves for the NI 43-101 IDOP Technical Report March 2012 which is based on pre-feasibility study work. The definitions of the Mineral Reserve classifications under NI 43-101 are the Canadian Institute of Mining (CIM) Definition Standards - For Mineral Resources and Mineral Reserves, Prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on 11 December 2005. The definitions below are quoted from the CIM Definition Standards – For Mineral Resources and Mineral Reserves, page 5.

After consideration of guidelines and other information regarding the declaration of Reserves for the United States Securities and Exchange Commission (USSEC) reporting, AMC considers that the IDOP Technical Report is suitable for declaring a Reserve as defined in US Industry Guide 7.

Documentation underlying Mineral Reserves determined in accordance with Industry Guide 7 generally includes the following:

- A "final" feasibility study.
- Utilization of the historic 3 year average price for the commodity that expected to be mined in determining economic viability.
- Primary environmental analysis has been submitted to government authorities.

15.4.1 Bankable Study

The IDOP Technical Report is based on the technical, production and cost information contained in the OT LLC study titled Integrated Development and Operations Plan (IDOP). IDOP was completed by the Rio Tinto appointed management of OT LLC in March 2011 as the basis for the proposed project financing.

Ivanhoe has reported that on 18 March 2012, the Ivanhoe Board of Directors approved a conditional comprehensive financing plan. The financing plan contains three principal elements: project finance, bridge finance and equity.

Ivanhoe is in an advanced stage of discussion with a core lending group of financiers for a \$4.0 billion project-finance facility. The core lenders group is comprised of European Bank for Reconstruction and Development, Export Development Canada, International Finance Corporation, BNP Paribas and Standard Chartered Bank. USEx-Im Bank and its adviser, Standard Bank, the World Bank Group's Multilateral Investment Guarantee Agency and the Australian Export Finance and Insurance Corporation recently joined the lender group. Ivanhoe has indicated to AMC that the lenders have received a report from an independent engineering assessment that indicates IDOP is suitable to support the technical for consideration of the project financing. Ivanhoe has indicated that its objective is to sign loan documentation in Q3 2012. AMC therefore considered it reasonable to conclude that the bankable study test in US SEC Industry Guide 7 has been met.

15.4.2 Test Price for Commodities

The base case financial analysis has been prepared using current long term metal price estimates of:

Copper \$2.85/lb

Gold \$1200/oz

Silver \$16.60/oz.

The 2005 SME Guide Section 53 describes how the Test Price for commodities should be applied.

"If a Mineral Reserve is reported using a price lower than the test price, the forward-looking discounted cash flow must be positive, and the Reserve Sensitivity Test (based on an undiscounted cash flow) need not be performed. When applicable, a statement should be made that a Reserves Sensitivity Test was completed, or that such a test was not applicable."

The metal prices for the previous 3 years, their average and the metal prices used for the Mineral Reserve estimates are shown in Table 15.3. The only metal price that is higher than the three year average is the forecast silver price. The three year average silver price is \$13.69/oz Ag and the forecast price is \$16.60/oz Ag. The sensitivity analysis using the 3 year averages show an after tax NPV₈ of US\$13.9B and an after tax IRR of 34% with a payback period of around 5.8 years. The results are significantly improved compared to the base case financial analysis, as the averages for the copper and gold prices are higher. This indicates that the Mineral Reserve is still valid at the 3 year average prices.

Table 15.3 Metal Price Summary

Year Ended	Cu (\$/lb)	Au (\$/oz)	Ag (\$/oz)
2009	2.34	972	12.74
2010	3.42	1,225	15.44
2011	4.00	1,572	12.89
Average	3.25	1,256	13.69
Reserve NSR	1.80	750.00	12.00
Base Case Financial Analysis	2.85	1,200.00	16.60

15.4.3 Primary Environmental Analysis Submission

The 2007 SME Guide Section 56 describes how the permitting and legal requirements of US Industry Guideline 7 should be applied. It indicates that:

"To demonstrate reasonable expectation that all permits, ancillary rights and authorizations can be obtained, the reporting entity must show understanding of the procedures to be followed to obtain such permits, ancillary rights and authorizations. Demonstrating earlier success in getting the necessary permits can be used to document the likelihood of success."

Based on the understanding of the procedures and the history of permitting, it is considered reasonable to assume that the final environmental permitting will be granted without resulting in a change to the Reserve.

Holders of a mining licence in Mongolia must comply with environmental protection obligations established in the Environmental Protection Law of Mongolia, Law of Environmental Impact Assessment and the Minerals Law. These obligations include preparation of a Detailed Environmental Impact Assessment (DEIA) for mining proposals, submitting an annual environmental protection plan, posting an annual bond against completion of the protection plan and submitting an annual environmental report.

OT LLC has posted environmental bonds to the Mongolian Ministry for Nature and Environment and Tourism ("MNET") in accordance with the Minerals Law of Mongolia for restoration and environmental management work required for exploration and the development work undertaken at the site. OT LLC pays to the Khanbogd Soum annual fees for water, sand/gravel and land usage.

OT LLC has, through qualified independent consultants, prepared a DEIA for the Project consisting of three main volumes (as per instructions from the MNET), comprised of the following:

- Volume I: Oyu Tolgoi to Gashuun Sukhait Road (the "OT-GS Road")
- Volume II: Gunii Hooloi Borefield
- Volume III: Mining and Processing Facilities

Additional separate volumes have been prepared for select facilities, including the Raw Water Supply Pipeline, Temporary Airport, Permanent Airport, Chemicals and the Umdai River Diversion. Internally, an additional volume has been produced for the Coal Fired Power Plant.

Both environmental and social monitoring activities are ongoing for the Project. The revised Environmental Social Impact Assessment (ESIA) is planned for completion in April 2012 to International Finance Corporation and European Bank for Reconstruction and Development standards.

The environmental baseline assessment for the project was prepared by drawing upon the wide range of internal and independent studies that have been prepared since 2003. The existing information was reviewed and assessed for accuracy, consistency and

validity. Where additional environmental baseline data became available in 2010 before the draft ESIA was produced, these were incorporated into the ESIA. Further data collection studies have been commissioned and started and commitments have been made in the corresponding management plans to ensure that collection of baseline data continues to improve and that the results of ongoing monitoring will be integrated into updated and revised management plans and procedures.

The baseline chapters presented in the ESIA are, necessarily, a summary of an extensive body of research and assessment that has been ongoing over many years.

15.5 Mongolian Commercial Minerals

Mongolia has its own system for reporting Mineral Reserves and Mineral Resources. OT LLC has registered a Mineral Reserve with the GOM. The system is based on a review by Mongolian experts in a number of disciplines. A significant difference between the Mongolian system and 43-101 is that, under the Mongolian system, resources and reserves are not valid until registered by the GOM. A committee of Mongolian experts examines a report prepared by the Owner using a set of guidelines and then, based on a consensus of nominated experts, a recommendation is made to the Minister for Mineral Resources and Energy. The recommendation to the Minister states the resources and reserves and any conditions. The Minister then issues an order registering the resources and reserves.

The reports examined by the experts must be in Mongolian language. OT LLC national staff and Mongolian consultants prepared the report under supervision of OT LLC personnel on secondment from Rio Tinto. The Mongolian Reserve included 43-101 Inferred Mineral Resource and is similar to the IDP10 Life of Mine (Sensitivity) Case. The Mongolian Reserve and IDP10 were prepared at different times and with different metal price assumptions for the analyses.

The Minister issued an order on 10 July 2009. OT LLC prepared the following translation of the order.

“Order Of The Minister For Mineral Resources And Energy, July 10, 2009, No. 167, Ulaanbaatar

On acceptance and registration of the revised reserve estimation of Southern Oyu deposits and the reserve report of Hugo Dummett deposits and Heruga deposit.

Pursuant to Article 48.4 of the Minerals Law, Clause 8, 9 and 14.2 of the Minerals Council Charter, and the Conclusion #15-01 of the Minerals Council extended meeting #15 held on 1 July 2009, it is resolved that:

- One. Revised Mineral Reserve estimation of Southern Oyu deposits, and Mineral Reserves of Hugo Dummett deposits and Heruga deposit in 2000-2008 exploration work report by Ivanhoe Mines Mongolia Inc LLC in Khanbogd soum, Umnugobi aimag are accepted as in the attachment to Conclusion summary #15-01.

- Two. Geological Research Centre (U. Borchuluun) is authorized to register the reserves quantity accepted under the Article 1 of this Order in the state minerals reserve inventory.
- Three. In relation to registration of revised Mineral Reserve of Southern Oyu deposits, the following copper and gold reserves of Southern Oyu to be mined with open pit operation registered in the state minerals reserve inventory with Conclusion #16-01 of the Minerals Council meeting #16 held on 30 July 2007 shall be removed from the state minerals reserve registration: 127 Mt ore or 0.658 Mt copper metal and 88.1 tonne gold in Measured (A) class, 803 Mt ore or 3.371 Mt copper metal and 148.3 tonne gold in Indicated (B) class.
- Four. Ivanhoe Mines Mongolia Inc LLC and MRAM (D. Batkhuyag) are assigned to implement the resolved issues in the Conclusion #15-01 of the Minerals Council extended meeting held on 1 July 2009.
- Five. Ivanhoe Mines Mongolia Inc LLC is hereby assigned to submit exploration report and field study materials in electronic format (data CD) to Geological Information Center, within 5 working days from the Conclusion of the Minerals Council meeting and the Resolution by Minister of Mineral Resources and Energy issued in relation to it."

16 MINING METHODS

16.1 Open Pit Geotechnical

16.1.1 Pit Slope Design

In May 2009, OT LLC retained Golder Associates Limited to carry out a review of the proposed pit slope designs. A summary of the results and conclusions of the review are provided in the following paragraphs.

With the exception of extensive surface weathering that extends to depths of approximately 50 m to 60 m, the rocks are generally hard and competent. Measured mean intact rock laboratory unconfined compressive strengths range from 113 to 206 MPa.

Hydrogeology investigations in support of water supply and pit dewatering estimates were carried out in a number of previous programs and provided the following information on the hydrogeology in the vicinity of the proposed pit:

- Shallow alluvial aquifers that increase in thickness from 0 to 15 m from the south-west toward the north-west side of the proposed pit exhibit a hydraulic conductivity on the order of 6×10^{-5} to 1×10^{-4} m/s, indicating a moderate capacity to transmit groundwater.
- A shallow bedrock hydro-stratigraphic unit that exists down to a depth of approximately 150 m below ground surface, and that is not dependent on lithology, exhibits a hydraulic conductivity of between 6×10^{-5} to 1×10^{-4} m/s and 1×10^{-5} to 1×10^{-6} m/s, indicating a moderate capacity to transmit groundwater.
- A deep bedrock hydrostratigraphic unit that includes various geological units found at depths of more than 150 m below the ground surface exhibits a hydraulic conductivity on the order of 6×10^{-5} to 1×10^{-4} m/s, indicating a low capacity to transmit groundwater.

Groundwater numerical modelling was carried out along three-cross sections as part of this review for the purpose of estimating groundwater pressures for modelling ultimate pit slope stability. Two sets of models were created based on the faults acting either as pathways or as barriers to the flow of groundwater perpendicular to the faults.

The various rock types contain well-defined and pervasive geologic structures in the form of joint sets and faults. In most open pits, the systematic joint sets do not exhibit sufficient continuity to affect the stability of multiple benches. However, due to their typical close spacing, the systematic joints will usually influence the stability of the individual bench faces. In contrast, the more continuous faults are typically widely spaced and, with the exception of localized instability, do not generally influence the stability of the individual bench faces. However, the fault sets typically exhibit sufficient continuity to influence the stability of multiple benches. The results of surface and shaft mapping, together with selected oriented core logging data, have been used to determine the orientations of the more continuous fault sets and the systematic joints sets.

The current proposed design pit slope criteria established by SRK in 2005 are summarized in Table 16.1 and Table 16.2 for Phases 1 to 7 and Phase 8, respectively. The current inter-ramp slope angles are not based on a kinematic assessment of structural geology data but rather on an empirical correlation with rock mass quality. As rock mass quality appears to increase with increasing depth, the proposed inter-ramp angles also increase with depth.

Kinematic stability analyses were carried out as part of the 2009 pit slope design review to assess the stability of the individual bench faces and of multiple-benches with respect to planar and wedge failure along undercut joint sets and fault sets. The results of these analyses are independent of rock quality and consider the orientations of the more dominant structures that are expected to be undercut in the pit slopes. Consequently, these recommendations are not dependent upon depth in the pit but only on the wall orientation with respect to the dominant geologic structures.

The data in Table 16.1 and Table 16.2 include a comparison between the current proposed design inter-ramp slope angles and the maximum inter-ramp angles recommended from the results of the kinematic analyses carried out as part of the 2009 review. For the most part, the optimum inter-ramp angles recommended from the 2009 stability analyses exceed the current design inter-ramp angles for the Phase 8 Pit. This indicates that the stability of the pit walls in this pit will likely be governed by overall slope stability and groundwater considerations, and not by structural considerations.

For the most part, the optimum inter-ramp slope angles recommended from the 2009 kinematic stability analyses by Golder also exceed the current design inter-ramp angles for the upper parts of the Phase 1 to 7 slopes. In some instances; however, the current design inter-ramp angle for the lower parts of these slopes, based on the SRK work, exceeds the inter-ramp angle recommended by Golder from its limited kinematic stability analysis. Considering the average current design inter-ramp angles for Phases 1 to 7, the recommended inter-ramp angles are very close to or exceed the current designs. The exception is Sectors 7 and 8, where the current design inter-ramp angles exceed the recommended angles by approximately 2° to 5°. On balance; however, the current design angles appear to be a reasonable approximation of the those that can likely be achieved on the Phase 1 to 7 pit walls; the number of sectors in which the slopes angles can apparently be increased exceeds the number of sectors in which they may have to be decreased.

The results of the kinematic stability analyses for the individual 15 m high bench faces indicate that optimum bench face angles are likely to be lower than the current proposed design bench face angles of 65° and 75°. This is due to a large number of systematic joint sets that appear to exhibit shallow to moderate dips. Until the interim slopes are excavated and the continuity of these structures can be determined, it will not be possible to confirm the optimum bench face angles, which could in fact be lower. Therefore, it is recommended that the proposed single-bench configuration be maintained for design because this configuration and the inter-ramp angles for the Phase 1 to 8 pits are expected to provide sufficient catchment for any wedge or planar failures that may occur due to undercutting of the systematic joint sets. Once the continuity of the dominant joint and fault sets is confirmed, it may be possible to adopt a double-bench configuration and increase inter-ramp angles in some of the pit sectors in

Phases 1 to 7. However, the overall slope stability considerations will likely preclude the use of double-benching in the ultimate Phase 8 pit because of the higher ultimate walls.

Table 16.1 Phases 1 to 7 Design Inter-ramp Angles

Current Design Sector	Range of Approximate Sector Azimuths	Current Proposed Inter-ramp Slope Angle*(SRK, 2005)	Recommended Optimum Kinematic Inter-ramp Slope Angle(as per 2009 Review)
1	325° to 332°	42.3° to 53.7° Average = 49.6°	59°
	333° to 025°		45°
2	215° to 296°	42.3° to 49° Average = 46.6°	45°
	297° to 300°		53°
	301° to 312°		56°
	313° to 325°		59°
3	025° to 044°	42.3° to 49° Average = 46.2°	45°
	045° to 110°		44°
4	110° to 135°	40.6° to 49° Average = 46.4°	44°
	136° to 147°		47°
	148° to 190°		45°
5	270° to 296°	40.6° to 45° Average = 43.6°	45°
	297° to 300°		53°
	301° to 312°		56°
	313° to 332°		59°
	333° to 360°		45°
6	360° to 045°	40.6° to 45° Average = 43.9°	45°
7	140° to 147°	45° to 51.3° Average = 49.4°	47°
	148° to 296°		45°
	297° to 300°		53°
	301° to 312°		56°
	313° to 320°		59°
8	080° to 135°	45° to 50.2° Average = 49.2°	44°
	136° to 147°		47°
	148° to 230°		45°
9	065° to 135°	40.6° to 45.9° Average = 44.3°	44°
	136° to 140°		47°
10	140° to 147°	42.3° to 45.9° Average = 44.5	47°
	148° to 220°		45°

* Current proposed design inter-ramp slope angle varies with depth and is based on an assumption that rock quality improves with depth, and not on a kinematic stability assessment.

Table 16.2 Phase 8 Design Inter-ramp Angles

Current Design Sector	Range of Approximate Sector Azimuths	Current Proposed Inter-ramp Slope Angle*(SRK, 2005)	Recommended Optimum Kinematic Inter-ramp Slope Angle(as per 2009 Review)
1	325° to 332°	36.9° to 45° Average = 41.6°	59°
	333° to 025°		45°
2	215° to 296°	36.9° to 45° Average = 41.6°	45°
	297° to 300°		53°
	301° to 312°		56°
	313° to 325°		59°
3	025° to 044°	38.3° to 43.1° Average = 41.9°	45°
	045° to 110°		44°
4	110° to 135°	38.3° to 43.1° Average = 41.9°	44°
	136° to 147°		47°
	148° to 190°		45°
5	270° to 296°	38.3° to 43.1° Average = 41.9°	45°
	297° to 300°		53°
	301° to 312°		56°
	313° to 332°		59°
	333° to 360°		45°
6	360° to 045°	36.9° to 43.1° Average = 40.7°	45°
7	140° to 147°	34.3° to 38.3° Average = 37.3°	47°
	148° to 296°		45°
	297° to 300°		53°
	301° to 312°		56°
	313° to 320°		59°
8	080° to 135°	39° to 45° Average = 43°	44°
	136° to 147°		47°
	148° to 230°		45°
9	065° to 135°	36.9° to 43.1° Average = 40.7°	44°
	136° to 140°		47°
10	140° to 147°	34.3° to 38.3° Average = 37.3°	47°
	148° to 220°		45°

* Current proposed design inter-ramp slope angle varies with depth and is based on an assumption that rock quality improves with depth, and not on a kinematic stability assessment.

The overall stability of rock slopes is also dependent upon the strength of the intact rock, the spacing and continuity of the rock fractures, and the surface quality of the rock fractures. Accordingly, the intact rock strength and fracture characteristics are combined to provide an empirical measure of the overall rock mass quality, which in turn is used to estimate the overall rock mass strength for the purpose of assessing overall rock slope stability. The Laubscher (1990) system has been used in previous studies to estimate the rock mass quality of the rocks that will be exposed in the Oyu Tolgoi open pit. On a scale of zero for very poor quality rock to 100 for very competent rock, the Laubscher rock mass ratings for the rock that will be exposed in the open pit vary from approximately 35 to 50 and can be classified as poor to fair quality.

The current proposed design overall slope angles, which include the access ramps, vary from 40.9° to 50.2° in the Phase 1 to 7 pits and from 35.4° to 42° for the Phase 8 pit. The stability of the proposed Phase 8 overall slopes has been assessed with respect to potential overall circular-type failures, which involve the development of failure through intact rock and pre-existing geologic discontinuities. Limit equilibrium stability analyses have been carried out along the following three cross-sections through the ultimate Phase 8 pit:

- Section 5 – West Wall (overall slope angle approximately 38°)
- Section 9 – East Wall (overall slope angle approximately 34°)
- Section 10 – South Wall (overall slope angle approximately 34°)

With the exception of Section 10 on the south wall of the Phase 8 pit, the proposed overall slopes are expected to exhibit factors of safety slightly greater than a recommended minimum design factor of 1.3 for the predicted ultimate pit groundwater pressures. The south wall is not expected to exhibit an adequate factor of safety if it is assumed that the Solongo Fault acts as a barrier to groundwater flow and will impede the drainage of the lower half of the south wall. This is concluded from drillhole data indicating that poor-quality QMD rock will be exposed at the toe of the south wall on the north side of the Solongo Fault. The results of the stability and groundwater modelling indicate that some form of artificial depressurization will likely be required along the lower part of the south wall to achieve adequate stability. A series of 200 m long drain holes penetrating the Solongo Fault on 50 m to 100 m spacing is expected to provide adequate depressurization.

Currently, artificial depressurization is planned for most of the proposed walls but, with the exception of the south wall and other localized areas in the pit, may not be required. Consequently, with the possible exception of Sectors 7 and 8 in the Phase 1 to 7 pits, the current proposed overall slope angles are expected to be achievable on the ultimate pit. The proposed design inter-ramp slope angles for Sectors 7 and 8 are steeper than the optimum angles recommended from the results of the kinematic analyses. Until the continuity of major structures can be confirmed in the interim pit excavations, it may be necessary to reduce the inter-ramp and overall slope angles in Sector 8 and along the southwest and south sides of Sector 7 in the Stage 1 to 7 pits.

Several major, steeply dipping faults cut through the deposit. The intersections of these faults will form steeply dipping wedges that will not be undercut by the proposed pit walls. However, localized bench-scale instability can be expected along the intersection of these faults with the pit walls.

Most of the geotechnical drilling carried out to date has been near the centre of the proposed pit, and none has been done behind the proposed pit walls. The rock quality and structural conditions in the exterior rocks will probably be similar to those encountered near the centre of the pit, but it is recommended that a number of confirmatory, oriented geotechnical drillholes be drilled in the major rock types behind the ultimate pit walls. Consideration should be given to using either the Ace or Ezy-Mark core orientation tools and to carrying out packer testing at various intervals in the core holes. In addition, a core hole to confirm the existence of poor-quality QMD rock at the toe of the south wall of the ultimate pit is also recommended.

16.1.2 Proposed Field Program

A field strategy has been developed for undertaking hydrogeological and geotechnical field programs in parallel. All geotechnical drillholes will be exploited to gain greater understanding of the hydrogeological conditions. The hydrogeological outcomes will be fully integrated into the geotechnical analysis, using sophisticated 2D and 3D numerical techniques. The objective is to provide pit wall design parameters that are compatible with achievable depressurization targets relative to the mine excavation schedule.

16.1.2.1 Geotechnical

The geotechnical field program will consist of both Acoustic Televiewer logging and manual logging in addition to sampling of core from 13 diamond holes to depths of between 350 m and 610 m, for a total meterage of 6,145 m. Down hole packer testing will be undertaken to quantify the permeability of targeted structures and also the rock mass fabric. Each borehole will be completed by cementing vibrating wire pressure transducers into the rock mass at targeted intervals to help assess the pre-mining groundwater pressure regime. Laboratory assessment of rock material characteristics will be carried out to supplement historical testing of samples from both the Hugo North and South Oyu deposits. A 3D geotechnical model will be developed. The geotechnical and hydrogeological field study results will be fully integrated through numerical analyses. The outcomes will be used to assess optimal pit wall configurations for the predicted groundwater conditions over the mine schedule.

16.1.2.2 Hydrogeology

The hydrogeological study will focus on understanding the hydrogeological conditions directly within the mine footprint and also external to the ultimate pit. The key objectives of this program are to develop a predictive pore pressure capacity, which will be used in the assessment of pit wall stability and also of external impacts on, or by, the Umdai River and tailings storage facility. The hydrogeological field program will include 2,548 m of reverse circulation percussion drilling. Each hole will involve in-situ down hole testing to quantify the hydrogeological characteristics of targeted fault/dyke structures and the rock mass fabric. Each hole will be completed with either standpipe monitoring wells or cemented vibrating wire pressure transducers.

On completion of the field program, a 3D district hydrogeological model will be developed to provide a predictive capacity for assessing groundwater conditions. The requirements for depressurization of the pit walls will be assessed, water inflows will be predicted, and strategies will be developed for pit water management and depressurization.

16.2 Open Pit Mining

16.2.1 Introduction

Rio Tinto Technology and Innovation (T&I) carried out mine planning, scheduling, and costing work for the Oyu Tolgoi open pits, including integration of the underground plans, since May 2010 in support of the Integrated Development and Operating Plan (IDOP).

The open pit work has focused on the Southwest and Central orebodies of the Southern Oyu region. The deposits fall within the following pit phases:

- Phase 1 Southwest
- Phase 2 Southwest
- Phase 3 Southwest
- Phase 4 Central
- Phase 5 Central
- Phase 6 Central (predominately chalcocite) and Southwest
- Phase 7 Southwest entirely
- Phase 8 All Southern Oyu deposits

The optimization, designs, and production scheduling were completed using Measured and Indicated resources only from the current SOT resource model reported in this technical report. Additional input parameters for the Southern Oyu deposits were provided by OT LLC in April 2010. T&I updated some of the first principal assumptions for open pit mine planning.

T&I concluded that the optimization work showed a small variance in tonnage in comparison with the previous optimization work in 2007 and that the result maintains the total mineable quantity close to the quantities reported in IDP10. The variance was attributed to updated metal prices (higher), pit slope angles, and cost estimates.

The mine plan is based on conventional open pit mining using a mix of 34 m³ hydraulic shovels, 56 m³ electric rope shovels, and 290 tonne diesel trucks.

The net smelter return (NSR) for each block was calculated from the resource models and includes recovered copper, gold, and silver grades as well as dilution due to small scale dykes that intersect the mineralized zones. The calculated net smelter return (NSR) was stored in a mining model and used for open pit optimization.

The NSR is the revenue paid for the concentrate at the “mine gate.” The NSR calculation excludes costs for mining, process, and G&A. NSR is calculated as the in situ value after allowances have been made for:

- Recovery to concentrate
- Smelter deductions
- Concentrate transport
- Smelter treatment and refining charges
- Royalties

NSR is calculated for each block in the model and divided by block tonnes to show NSR in \$/t. The NSR values are used to classify the ore blocks in the pit optimization process. The NSR values are also used as economic cut-offs for the Mineral Reserve. The parameters used for calculation of NSR value are listed in Table 16.3.

Table 16.3 Base Data Template 27 – Pit Optimisation

Payment and Deductions	Cu	Au	Ag
Metal Prices	US\$2.10/lb	US\$816.00/oz	US\$13.00/oz
Payment Levels	96.5% of full Cu content	<1 g/dmt, 0%	If >30 g/dmt then 90% of full Ag content
	–	<3 g/dmt, >1 g/dmt, 90% of Au content	–
	–	<5 g/dmt, >3 g/dmt, 94% of Au content	–
	–	<10 g/dmt, >5 g/dmt, 95% of Au content	–
	–	<15 g/dmt, >10 g/dmt, 96% of Au content	–
	–	<20 g/dmt, >15 g/dmt, 97% of Au content	–
	–	>20 g/dmt, 98% of Au content	–
Deductions	Minimum 1 unit/dmt	N/A	All <30 g/dmt
Treatment and Refining			
Treatment (Concentrate)	–	\$75/dmt	–
Refining	\$0.075/lb	\$5.00/oz	\$0.45/oz
Price Participation	\$0.020/lb	–	–
Penalties	Arsenic: No penalty to 0.3%. Thereafter \$2.00/dmt penalty per 0.1%. Rejection level 0.5%.		
	Fluorine: No penalty to 300 ppm. Thereafter \$2.00/dmt penalty per 100 ppm. Rejection level 1000 ppm.		
Royalties			
Mongolian Government	5% of payable metal value		
Transport			
Moisture Content	9%		
Transportation	\$27.47/dmt		

16.2.1.1 Pit Optimization Methodology

The methodology carried out by T&I for pit optimisation was to create a single breakeven pit shell using the Mine Sight Economic Planner (MSEP). MSEP uses input data such as mining model, mining and processing costs, recoveries, and pit slope angles to calculate a pit shell based on mathematical algorithms widely known as Floating Cone (FC) and Lerchs-Grossmann (LG). The input for the breakeven pit shell was chosen from the previous mine plan (IDP10). It is standard practice for T&I to use the previous pit design as input to the LG optimization and then modified in the application of the LG process. The breakeven pit shell is shown in Figure 16.1. The break even pit shell work identified an undiscounted and discounted pit shells the inventories within these shells is shown in Table 16.4.

The mining phases were calculated using the FC algorithm. This option uses incremental revenue factors to calculate cones, which are later used for phase selection. The initial FC cones for Phases 1 to 3 were constrained geographically by limiting the easting and northing blocks available so that the cones were restricted to the Southwest zone to avoid mixing the Southwest and Central ores. The goal for Phase 1 was to achieve 45 Mt of ore with a cone value of at least US\$9.00/t. Starting from the Phase 1 pit shell, Phase 2 was then targeted for 100 Mt of ore at a net cone value of at least \$7.00/t.

Next, Phase 3 was calculated starting from Phase 2. A cone value of US\$11.00/t was the first shell that provided adequate bench width for mining. If the floating cone were not limited to Southwest region only, they would have chased central ore as well as Southwest ore.

Pit shells for Phases 4 to 7 were determined in a similar fashion by subsequently lowering the value per tonne limit while targeting an ore tonnage and strip ratio capable of feeding the concentrator. In summary, the following rationalization was considered:

- Concentrator operational constraints required that central ore be delayed until adequate production quantities of ore from Hugo North underground are available for optimum blending.
- Enough ore was to be provided from the Southwest phases to fill concentrator production until 2020.
- Ore from the Central and Southwest phases would not be blended together in the mine schedule to avoid processing recovery losses.
- Optimization for Phases 1, 2, and 3 was constrained to the Southwest area to allow Southwest mineral to be mined until 2018, while bringing on full production from underground, even though ore from Central became economic early on.

Figure 16.1 Breakeven Pit Shell

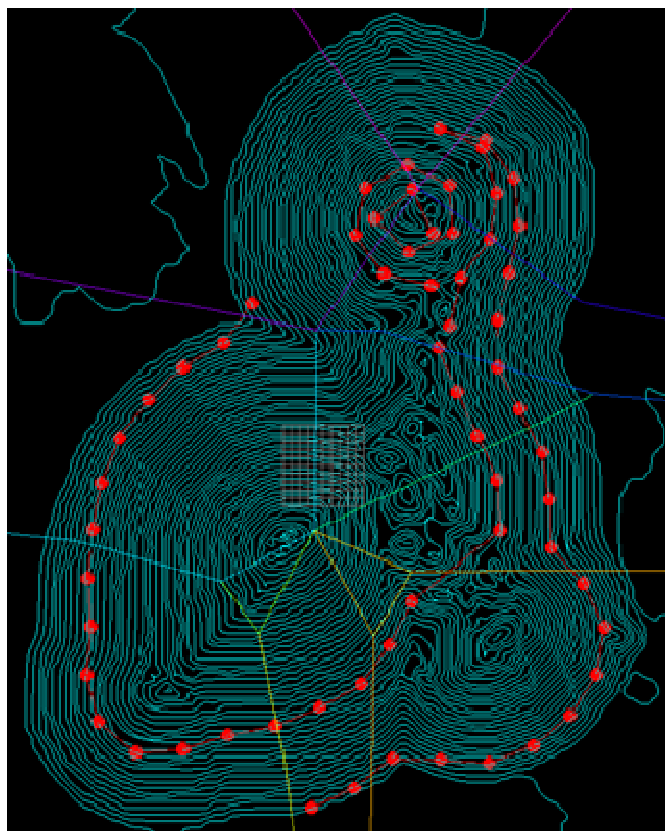


Table 16.4 Undiscounted/Discounted Pit Shells

Pit	Shell	Ore Mt	Waste Mt	Total Mt	Strip Ratio
Undiscounted	45	973	2,072	3,045	2.13
Discounted	47	924	1,563	2,488	1.69

Cross-sections of the various phases are provided in Figure 16.2 to Figure 16.4. The pit optimization results show the following general phase mining sequence for the Southern Oyu Tolgoi deposits:

- Phases 1 to 3 – include Southwest ore only.
- Phase 4 and 5 – includes the Central covellite as the majority ore, followed by chalcocite and chalcopyrite.
- Phase 6 – includes Central covellite as the majority ore, followed by chalcopyrite, chalcocite, and Southwest.
- Phase 7 – includes Southwest ore only.
- Phase 8 – includes Southwest as the majority ore, followed by chalcopyrite, chalcocite, and covellite.

Figure 16.2 Cross-Section 1 Showing Optimization Phases

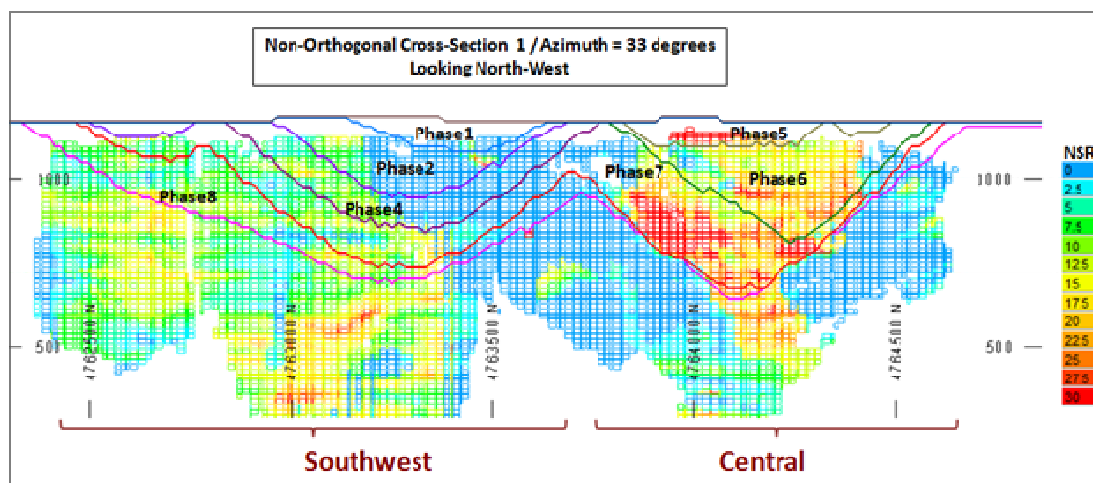


Figure 16.3 Cross-Section 2 Showing Phases 1 to 3

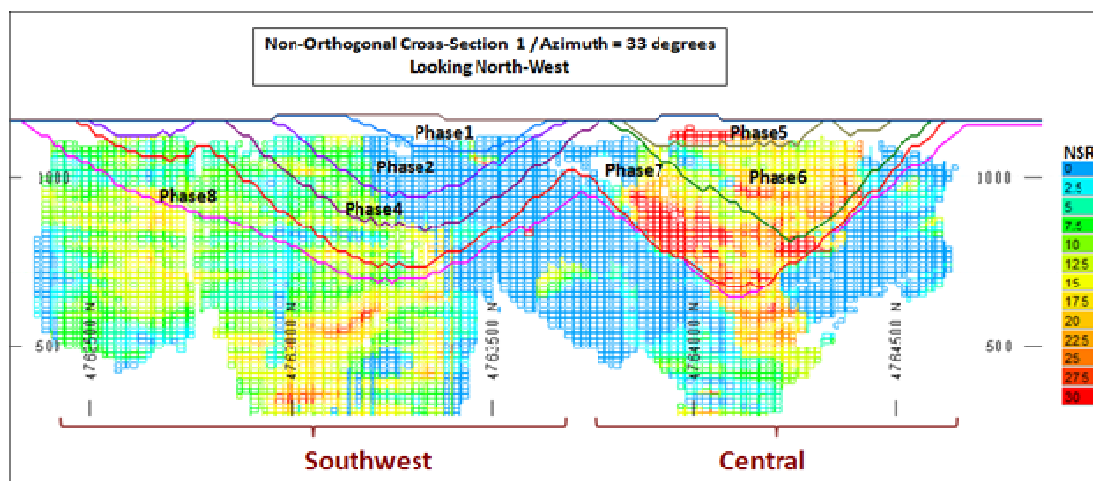
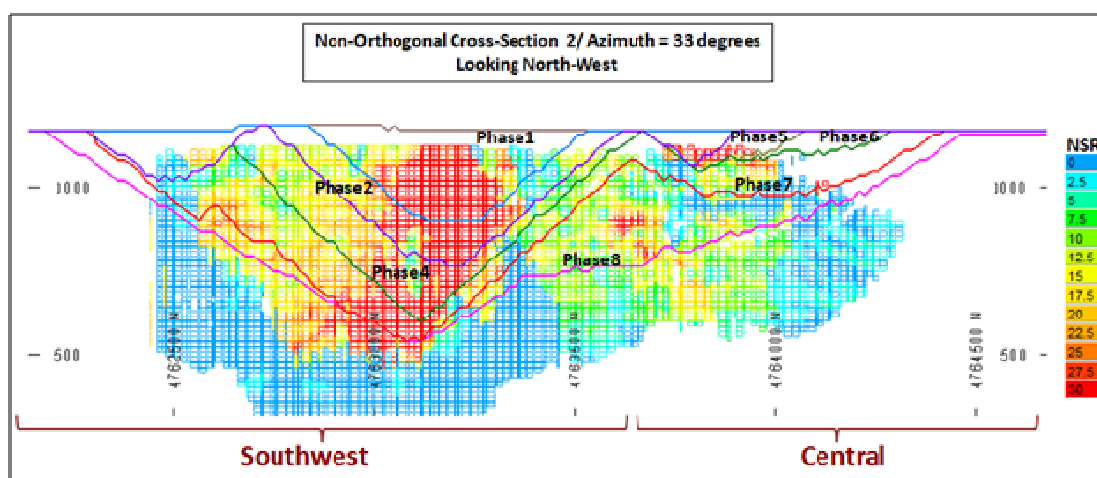


Figure 16.4 Cross-Section 2 Showing Phases 4 to 8



16.2.1.2 Mine Phase Design

The eight phase designs for the open pits were prepared using the pit shells and the following design criteria:

- Minimum operational width of 80 m between phases.
- Generally, two exit haul roads per phase
- 40 m haul road width
- 10% haul road gradient
- 15 m bench height.
- Pit slope angles and berms as recommended in the open pit geotechnical section.

Footprints and design of the eight open pit phases are shown in Figure 16.5 and Figure 16.6.

IDOP identified that there could be improvement in the overall results by making the designs of Phases 1 to 3 deeper by up to two benches. The pit optimisation and was recommended for further study.

Figure 16.5 Open Pit Phase Designs

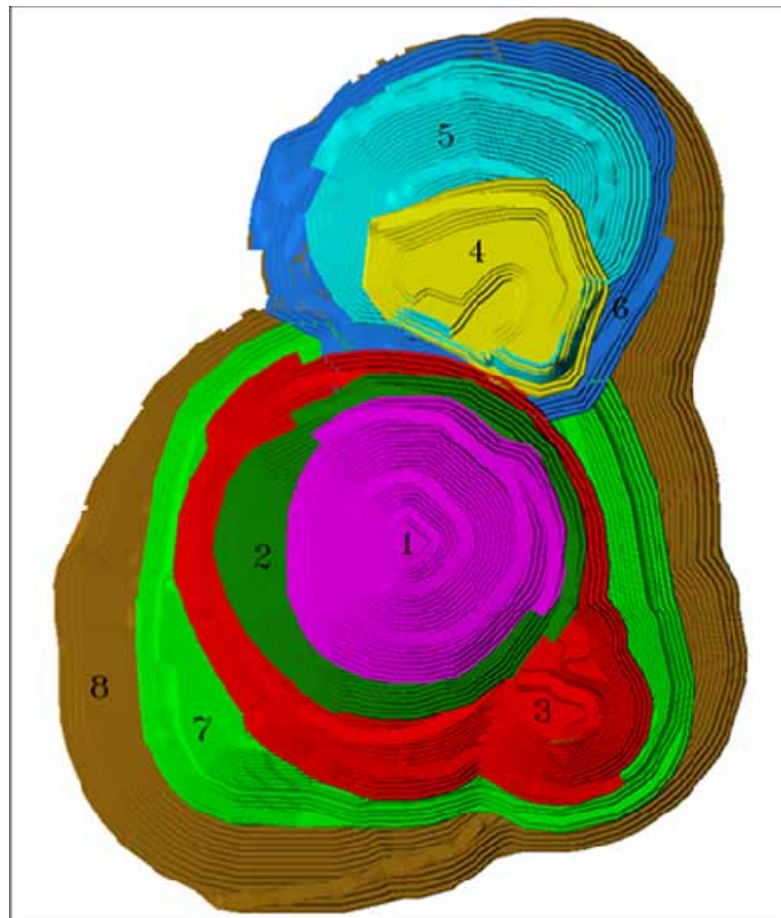
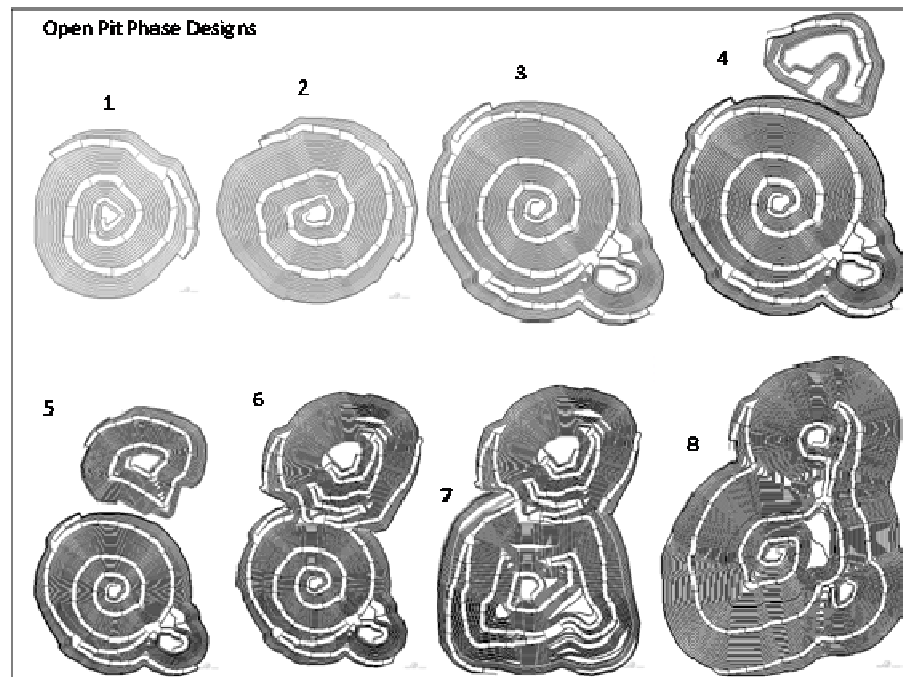


Figure 16.6 Open Pit Phase Designs



16.2.1.3 Waste Dump and Stockpile Design

The designs for active waste dumps assume a swell factor of 30% for the material delivered from pit benches, considering natural sorting and compaction in the dumps. It is also assumed that the blasted waste rock will settle at a natural angle of repose of 37°. The plan is to build the dumps in 10 m high lifts, forming 20 m high benches with 40 m wide haul roads at 10% grade. The dump height is designed to a maximum of 70 m or the 1,230 mRL. This is based on the Environmental Impact Assessment submitted to the GOM in 2007. However, an additional 10 m lift was needed on the Near South dump for additional capacity to provide flexibility for the placement of non acid forming (NAF) and the tailings stockpile.

The design criteria consider selective placement of rock on the dumps, ensuring that waste dumps are located so that potentially acid forming (PAF) waste is isolated from stream sediments to eliminate the risk of off-site migration of ARD post closure. The designs include the following:

- Placement of a 3 m thick initial NAF lift on the western half of the dumps, an area where the surface does not contain clay
- Placement of a 1 m to 3 m thick NAF cap on the reclaimed dump surface to mitigate the effects of erosion.

The following dumps were designed:

- Tailings stockpile (East dump) – This dump stores NAF material—oxide and sediment material to be used in construction of tailings storage facility (TSF). This stockpile is dynamic because tailings construction will remove dumped material from the stockpile as required.
- West waste dump – This dump will be used primarily for storing PAF material. It is not on the clay material and requires a 3 m NAF lift to prevent PAF from contaminating the water table and adjacent Umdai River.
- South waste dump Near and Far – This dump can be managed so that different material can be stored depending on cycle times and waste rock characteristics. Approximately half of these dumps are on clay and do not require the NAF bottom lift.
- The two voids between the Near South and West dumps, and between the South dump and Tailings stockpile need to be filled to achieve the final dump. This was done at the end of the mine life and assumed to be tipped from the top of the Near South dump.
- The West, Near South, and Far South dumps take a mix of NAF and PAF material. The Tailings stockpile is empty by 2032, when the third tailings cell is constructed and can be used to store both PAF and NAF rock. This Tailings stockpile is on the clay surface and does not require a NAF bottom lift.

Waste dump capacities are shown in Table 16.5, and the designs are depicted in Figure 16.7. The cross-section in Figure 16.8 shows that the waste dump and stockpile designs consider an overall slope angle of 18° after reclamation.

Table 16.5 Open Pit Waste Dump Capacities

Waste Dump	Bank Volume Mm ³
Tailings (East)	151
Near South	213
Far South	204
West	86
Void	45
Total Waste	700

Figure 16.7 Waste Dump Designs

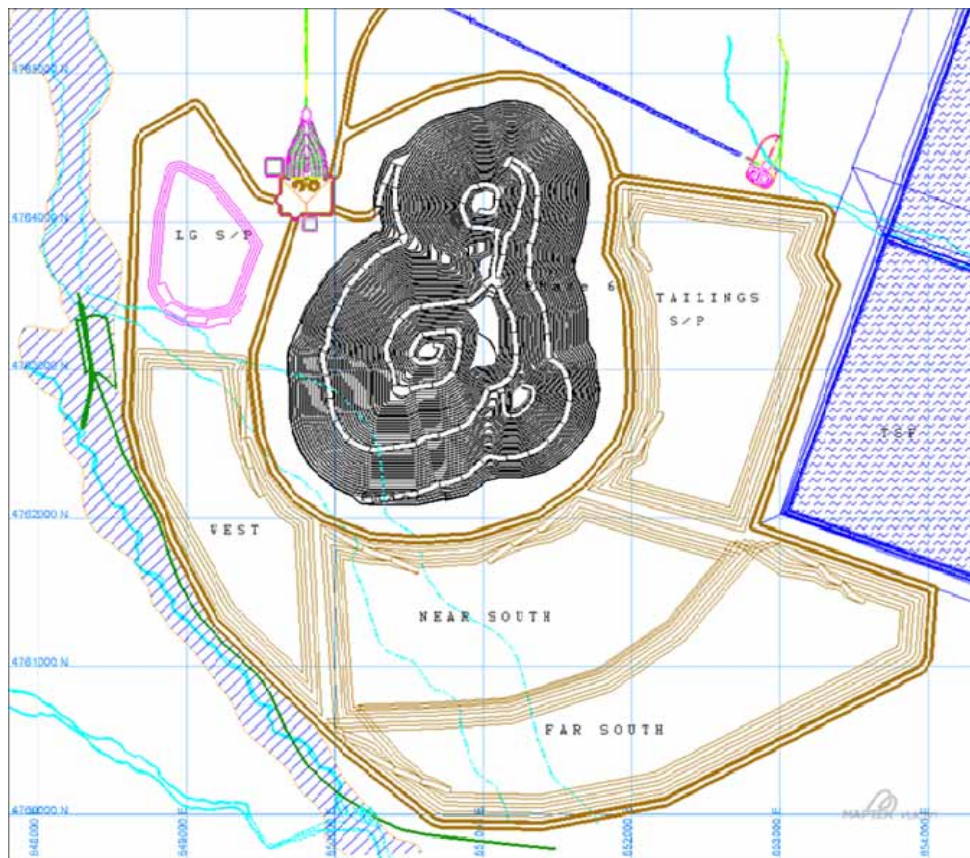
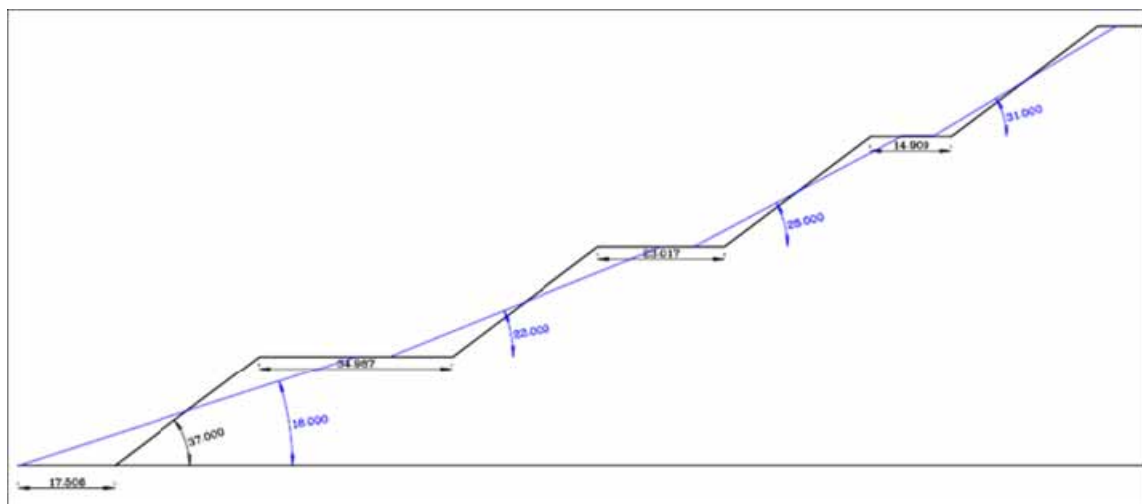


Figure 16.8 Cross-Section Showing Dump Design Geometry



16.2.1.4 Open Pit Phase Material Summary

Table 16.6 summarizes total material moved by pit phase.

Table 16.6 Summary of Total Material by Pit Phases

Phase	Ore Mt	NSR \$/t	Grade					Waste (Mt)	Strip Ratio	Total Material (Mt)
			Cu %	Au g/t	Ag g/t	As ppm	F ppm			
Phase 1 SW	63	33.68	0.55	0.82	1.40	10.37	2,094	88	1.40	151
Phase 2 SW	71	24.97	0.48	0.52	1.32	14.22	1,883	106	1.50	177
Phase 3 SW	136	22.01	0.49	0.35	1.40	37.14	1,730	225	1.65	361
Phase 4 Central	18	28.82	0.96	0.09	1.57	272.57	2,250	26	1.42	44
Phase 5 Central	92	19.47	0.60	0.12	1.37	180.01	1,959	107	1.17	199
Phase 6 Central	77	22.51	0.66	0.17	1.19	194.92	1,879	149	1.94	225
Phase 7 SW	199	16.64	0.40	0.25	1.19	54.21	1,623	343	1.73	542
Phase 8 (Final)	302	18.89	0.44	0.30	1.09	66.12	1,711	749	2.48	1,051
Total/Average	957	20.82	0.49	0.32	1.24	77.17	1,781	1,792	1.87	2,750

16.2.2 Open Pit Operating Schedule

The open pit mining operations are scheduled for nominally 350 days per annum. It was assumed that 15 days per annum will be lost due to unscheduled delays such as weather conditions; this was accounted for in the use of availability. The open pit equipment will operate 24 hours per day in two 12-hour working shifts.

Based on operation requirements, OT LLC has prepared a comprehensive labour roster that includes local and expatriate personnel. Oyu Tolgoi Project will require a significant number of key expatriate personnel to help with start up, provide training, and mentoring. There will be a significant increase in the number of local and expatriate personnel required, starting in 2012, as can be seen in Table 16.7.

Table 16.7 Open Pit Open Pit Labour Profile

Labour Category	Dec 2011	Dec 2012	Dec 2013	Dec 2019
Contractor				
Expatriate	35	29	29	12
Local	57	68	68	83
Total Contractor Labour	92	97	97	95
Employee				
Expatriate	33	35	20	11
Local	383	401	435	478
Total Employees	416	436	455	489
Total Labour	508	533	552	584
Total Expatriate	68	64	49	23
Total Local	440	469	503	561
National Hire Ratio	87%	88%	91%	96%

16.2.3 Mining Equipment

16.2.3.1 Equipment Requirements

The open pit mine will be a conventional shovel-truck operation. OT LLC's workforce will carry out drilling, loading, hauling, and dumping. Equipment maintenance will be conducted under Service Agreements with the original equipment manufacturers (OEMs) in-country dealers. Under this agreement, the contractor will bring personnel, equipment, and facilities on site to accomplish its objectives as specified in the service agreement contract. A contractor will provide blasting products and services.

IDOP was based on the use of a mixed fleet of 34 m³ diesel hydraulic shovels, 56 m³ electric rope shovels, and 290 tonne class diesel trucks. The equipment estimates are based on the specifications and data provided by the equipment suppliers in their submissions to the OT LLC tender process. The equipment fleet requirements at start-up and at five-year spans are listed in Table 16.8.

The shovel and drill hours are based on equipment productivities. Truck hours are derived from the cycle time estimates and shovel numbers. Dozers are based on the number of shovels plus additional dump requirements. Lighting towers are based on the numbers of shovels, drills, and dumping points. Numbers of light vehicles and personnel carriers are based on labour count by job.

Table 16.8 Open Pit Mining Equipment in Service

Equipment Units	2011	2012	2013	2016	2021	2026	2031
Shovel – 34 m ³ Hydraulic Diesel	2	2	2	2	2	2	2
Shovel – 495 HR Rope Electric	–	2	2	2	2	2	2
Truck 290 t – 930E-4SE	14	27	27	27	32	33	44
Production Drill – Electric	1	2	2	2	4	4	4
Production Drill – Diesel	2	2	2	2	–	–	–
Dozer – D375 – 6	4	4	4	4	4	4	4
Dozer – D475 – 7	–	2	2	2	2	2	2
Grader – 16M Cat	2	3	3	3	3	3	3
Grader – 24M Cat	–	1	1	1	1	1	1
FEL – WA1200 18 m ³	2	2	2	2	2	2	2
Wheel Dozer – WD600	1	2	2	2	2	2	2
100 t Water cart	2	2	2	2	2	2	2
Tire Changing Forklift	1	1	1	1	1	1	1
Integrated Tool Carrier	2	2	2	2	2	2	2
Excavator 4 m ³	1	1	1	1	1	1	1
Crane	1	1	1	1	1	1	1
Cable Truck	1	1	1	1	1	1	1
Blasting MMU	2	3	3	3	3	3	3
Flat Bed Crane	1	1	1	1	1	1	1
Low Loader Tractor/Trailer (110 t)	1	1	1	1	1	1	1
Lighting Towers	15	24	24	24	26	26	26
Personnel Carriers	3	5	5	5	5	7	7
Mine Lube Truck	3	3	3	3	3	3	3
Forklift	2	2	2	2	4	4	4
Secondary Drill	1	1	1	1	1	1	1
Mine Fuel Truck	3	3	3	3	3	3	3
Light Vehicles	31	32	32	32	32	28	28
Pit Pump	3	3	4	9	15	19	19
Crane (120 t)	1	1	1	1	1	1	1
Low Bed Trailer/Tractor (250 t)	1	1	2	4	4	4	4

16.2.3.2 Drilling and Blasting

Blastholes for ore and waste production drilling will be 17 m deep x 305 mm diameter. The Owner's team will conduct the blast designs, shot-firing, and monitoring, while a contractor will carry out down-the-hole services, including the supply and storage of explosives. It is assumed that ammonium nitrate fuel oil (ANFO) will be used in dry holes and heavy ANFO in wet holes. Given the prevailing conditions, 90% of blasting is expected to be dry. Powder factors of 0.66 kg/bcm for ANFO and 0.68 kg/bcm for emulsion explosives have been used for planning.

Blasting design factors are summarized in Table 16.9.

Table 16.9 Preliminary Drill and Blast Design

Variable	Unit	Dry	Wet
Bench height	m	15.0	15.0
Hole diameter	mm	305	305
Burden	m	7.6	8.5
Spacing	m	8.7	9.8
Subdrill	m	2.0	2.0
Stemming	m	6.8	7.7
Total hole depth	m	17.0	17.0

The blasting estimates are considered appropriate for IDOP, in operation, parameters will need to be refined for conditions as field data are gained. All materials are assumed to be blasted. Free digging may be possible in the first or second benches, but detailed work to define the extent of this has not been done. Delaying purchases to account for the possibility of free-digging is not recommended due to the long lead time for major equipment supply.

Because of the timing of the power station construction and the required start-up flexibility, the initial two production drills are assumed to be diesel units and the second two to be electric. As replacements are required, these drills will be replaced with electric-powered drills. In addition to the primary production blasthole drills, a secondary drill has been included in the estimates for the LOM. This will be used for pre-split blasts, and secondary drilling, and for drilling horizontal drain holes as specified by the geotechnical recommendations.

16.2.3.3 Loading and Hauling

The primary loading fleet will be two 34 m³ bucket hydraulic shovels and two 56 m³ electric rope shovels. Once in operation, one of the shovels will mine ore from the lower pit benches, and the other(s) will mine waste for the development of the next phase. Another shovel will be moving between the ore and waste phases to assist in mining ore, low-grade stockpiling, and waste stripping as required. Short-term planning will need to be coordinated with operations for moving shovels between the mining phases.

In addition to the four shovels in use during preproduction, two 18 m³ front-end loaders (FELs) are included in the estimates for the open pit mine loading fleet. This will provide additional flexibility in ore/waste loading capacity and stockpile management.

The open pit operation is planned to start with the first drill and loading unit in July 2011. Gradual production ramp-up is expected when hydraulic shovels began operating in late 2011. The larger electric rope shovels are targeted for arrival in mid 2012. At this point, the smaller loading units are considered support equipment for mining drop-cuts, cleanup, stockpile retrieval, and end-of-cut section.

This plan takes into account the increased distances of the haul roads as the pit deepens over time.

16.2.3.4 Ore Control

Preliminary plans include sampling, assaying, mine modelling, and waste/ore boundary management in the field:

- Samples of rock detritus from blastholes will be sent to the site laboratory and assayed for grade (copper, gold, silver) and deleterious elements (arsenic, talc, sulphur) every day. The laboratory will place assay results on a shared network at the end of every shift.
- The ore control system (OCS) will automatically pick up assay results and match them with their blasthole collar coordinates through their unique IDs. The OCS will then interpolate the new assays to update the grade variables and NSR values in the short-term model.
- The ore control engineer will determine the updated ore/stockpile/waste boundaries within ore shovel daily planning cuts. These boundaries (x, y, z coordinates) will be picked up by the surveying system. Surveyors will field-stake the boundaries with visible standardized colours by material destination (crusher, stockpile, waste). Alternatively, shovel operators will receive the information on ore/waste boundaries on consoles in the shovel cabin and pass the instructions on to the truck drivers for execution.

16.2.3.5 Mine Equipment Monitoring and Control

The mine equipment fleets will be monitored through a computerized system available in the industry. The information collected will be used for fleet performance management and continuous improvement. A potential supplier for this system has been identified; capital costs and ongoing operating costs for this service are allocated in the mine services section.

16.2.3.6 Open Pit Mine Dewatering

Diesel-powered pumps will be installed in sumps for open pit dewatering as the mining benches are sunk. The water will be pumped to a dam at the process plant. Pumps have been sized depending on pit depths.

16.2.4 Open Pit Production Planning

16.2.4.1 Open Pit Scheduling Methodology

The objective of the IDOP production schedule was to maximize the early cash flow from the open pit by delaying costs and bringing revenue forward with ore feed to meet concentrator throughput capacity.

Considerations for the production scheduling:

- Ensuring continuous ore supply to the concentrator by delivering the highest NSR value ore first and meeting physical mining and milling hours capacity constraints.
- Maintaining the vertical mine sinking rates according to the shovel allocation schedules.
- Maximizing utilization hours per year for mine loading equipment.
- Maintaining a balance of ore throughput hardness and mill cut-off grades that allows milling hours to be maximized.
- Transitioning open pit ore delivery from Southwest and Central in combination with the Hugo North (HN) block cave.

Stockpiling was also an important feature of the schedule. The mine schedule considers that at least two hydraulic shovels and one electric shovel will be allocated in Phase 1 during the first three years of operation. The physical limitation of the bench width in Phase 1 reduces the mining capacity to one hydraulic shovel and another electric shovel. Having greater shovel capacity in the benches of the ore phases presents the opportunity to raise mill cut-off grades. This, in turn, leads to stockpiling medium- and low-grade material and sending higher-grade ore to the mill earlier in the mine life. To achieve a balance between ore material hardness and the amount of high-grade ore available, the mill cut-off is adjusted every period, resulting in a variable cut-off policy for the first three years.

The loading fleet productivity assumptions are shown in Table 16.10. Total fleet capacity at full production used is approximately 111.6 Mtpa of ore and waste.

Table 16.10 Loading Fleet Production Assumptions

Fleet	Unit	Shovel 56 m ³ Bucket	Shovel 34 m ³ Bucket	FEL 18 m ³ Bucket
Annual production before shovels commissioned	Mtpa	–	–	10.2
Annual production after shovels commissioned	Mtpa	30.6	20.1	5.1

16.2.4.2 Mine Schedule Overview

The open pit was scheduled by month, by quarter, and by year, as follows:

- Years 2011, 2012, and 2013 by month.
- Years 2014 to 2016 by quarter.
- 2017 and on, by year.

Highlights of the production schedule results are summarized below.

- The open pit mine reaches maximum production capacity in July 2012 with the purchase of the fourth shovel. Full annual production capacity is reached in 2013 with all four shovels operating for the entire year. The FELs are maintained throughout the LOM.
- The initial sources of ores are Phases 1 to 3 of the Southwest deposit. Mine sequencing has changed from the previous schedule versions to allow waste stripping of Phase 7 to begin in year 2016. This required a small design change on the north side of Phase 7 to provide haul road access. Waste stripping of the central pit Phase 4 does not start until 2018.
- The available truck-hour requirement increases from 140,000 and 200,000 in year 2012 and 2013 to more than 300,000 in 2037. Nearly 200,000 available truck-hours are required at maximum loading capacity in 2015. The increase in LOM truck cycle times reflects the deepening of the pit mining phases and increasing distances to the waste dump. These cycle times do not include truck operator efficiencies to account for mining start-up.
- Southwest open pit ore is the main feed to the concentrator during the first nine years of operation at an average throughput rate of 32 Mtpa.
- Soft ore is processed in 2021 and 2025, harder ores from Phase 6 and 7 between 2026 and 2030. This brings the total ore feed from open pit to above 25 Mtpa, which complements the underground mine ore production. As the underground mineral source is depleted toward 2034, average-hardness minerals are processed, reaching a rate of around 50 Mtpa until the end of the mine schedule.
- This version of the schedule relies on maintaining steady ore production from Phases 1, 2, 3, and 7 in the Southwest zone. This ability depends on maintaining open dual access to these phases. Having dual access phases allows haulage profiles for ore (west-bound) and for waste (East TSF and South waste dumps) to be optimized. Post-schedule results show the need to extend the Phase 7 main haul road access up to the top benches for exit to the crusher and to the waste dumps.
- In addition to bringing copper forward from Southwest zones phases, this version of the mine plan (MP01) focuses on mining high-grade gold blocks from the lower benches of the first three phases of mining. It was found that subsequently moving mining on to the central orebody (Phases 4 to 6) as proposed in the previous schedule, delayed gold production from Southwest. Therefore, this schedule delayed the waste stripping of central zone and focused on metal production from Phases 1 to 3 and Phase 7, improving copper and gold metal production in the medium and long term.

Summaries of total open pit production, open pit ore production and head grade are illustrated in Figure 16.9 to Figure 16.11.

Figure 16.9 Total Open Pit Production Summary

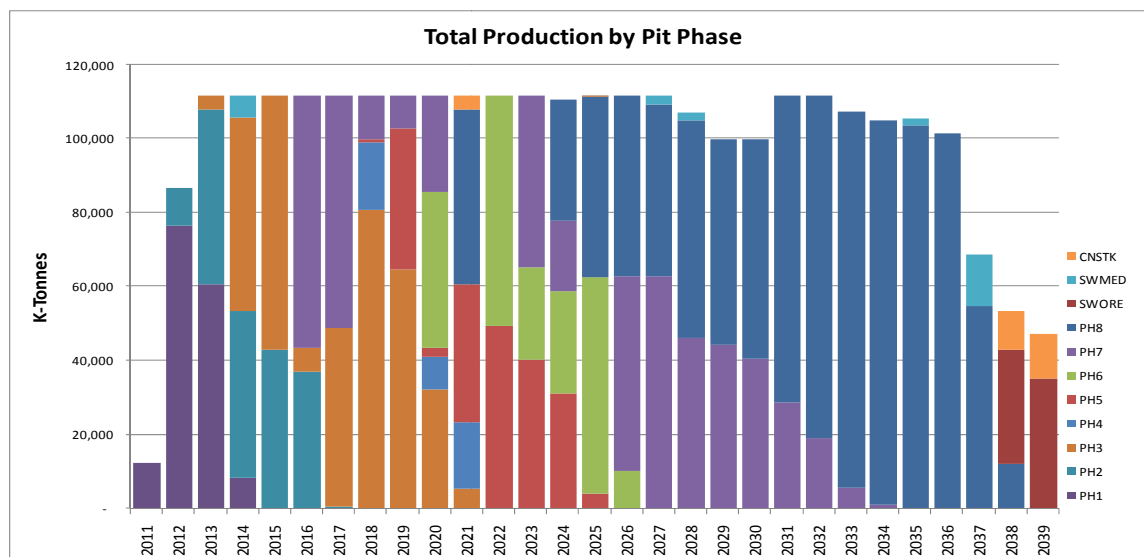


Figure 16.10 Ore Production from Open Pit by Phase

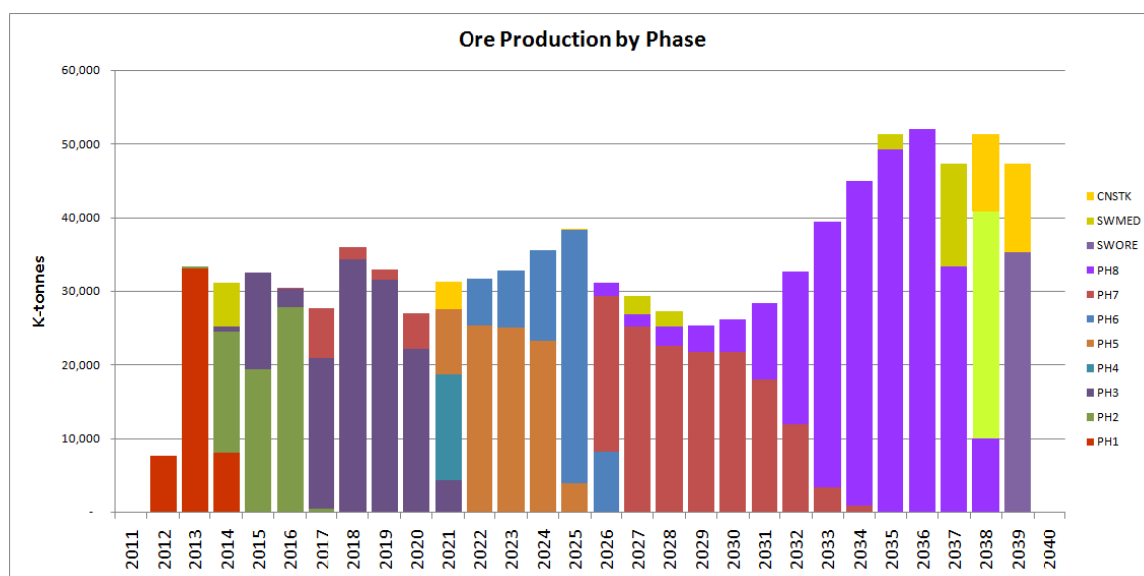
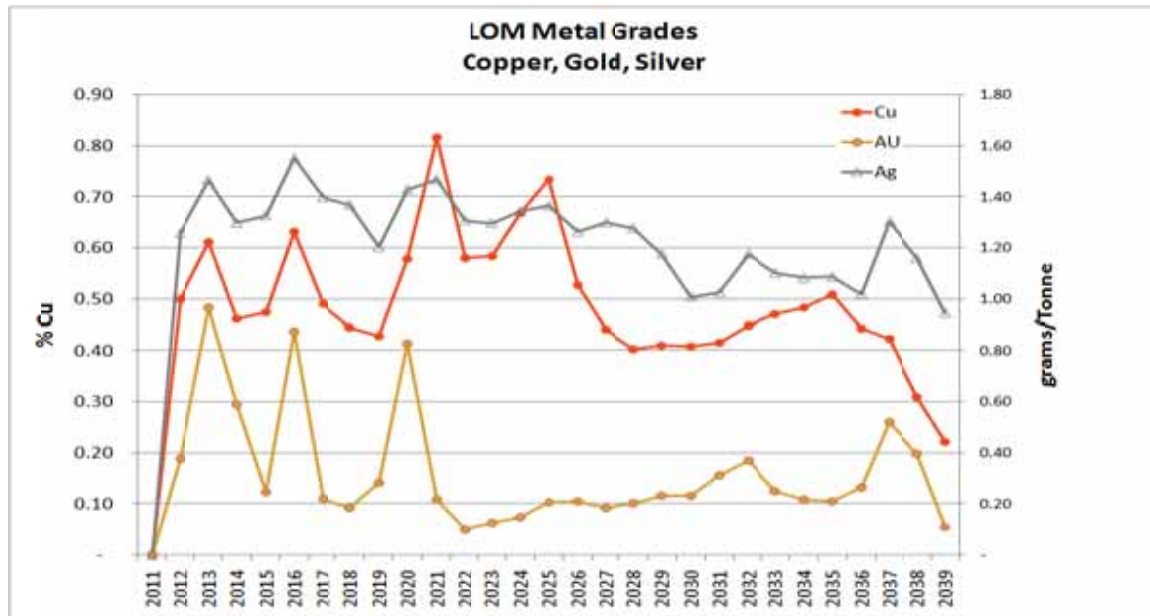


Figure 16.11 Open pit Ore Head Grades



16.2.5 Recommendations

The IDOP mining planning work resulted in the following recommendations being made for further evaluation of the open pits:

- Evaluate extra/early mining versus the improved recoveries associated with the pure Central/Hugo North feed.
- Investigate operating standby times and identify the operating delays associated with mining of bottom benches. Although the current plan reduces the number of loading units that can be allocated in the bottom benches of the mining phases, a better understanding of the operating delays associated with mining narrow areas on bottom benches is needed.
- Calculate shovel productivity rates associated with single spot, double spot, ramp, and wall cleanup loading. This work assumes an average productivity rate for the different loading units regardless of the digging conditions.
- Develop a contingency plan to manage risks due to potential geotechnical instability of the slopes during start-up mining in case the projected inter-ramp slopes angles do not hold during actual mining of Phases 1 and 2. This would affect access to ore for the first year of production.
- Set up procedures and tools for short- and medium-range mine planning at the operational level.

16.3 Underground Geotechnical

16.3.1 Introduction

This report is based on the prefeasibility geotechnical report issued in IDP10. It also discusses some of the new information based on the confirmatory tunnel. However, no new analysis was conducted for this 2012 update. New geotechnical information should be analysed in the next stage of the mining study.

Before underground exposure was available a comprehensive drillhole database had been compiled, and basic geotechnical analyses including fragmentation, cavability, and subsidence had been conducted. It was concluded that the main areas of uncertainty were in:

- In-situ stress assessment character and behaviour of large-scale structures.
- Rock mass behaviour as a reaction to excavations at depth.
- Lack of geotechnical information from northern sector of the orebody.
- Lack of hydrogeological information.

With respect to these points, it was stated that information based only on the drillholes could not reduce the uncertainty in data analyses and that it would be prudent to expose the rock mass underground.

Drillholes specifically for geotechnical purposes have been drilled since 2005. Drillholes covering the northern sector were completed, Shaft No. 1 has been sunk to 1,300 m below the surface, and comprehensive stress measurements from several depth horizons and locations in the tunnel have been successfully completed. SRK published its "Geotechnical Study Update" in October 2009 as part of IDP10.

In summary, the following was concluded:

- There is reasonable correlation between the estimated stress conditions and the results of actual stress measurements at various depths.
- There is reasonable confirmation of the ground conditions predicted from the drill core, for at least lithology exposed to date.
- New geotechnical drilling information more or less confirmed previous rock mass assessments, although there was a light shift toward less competent rock mass in some units.
- The shaft exposure indicated more-or-less dry conditions. The possibility of significant water inflow from the large-scale structures cannot be eliminated.
- The stress versus rock mass strength ratio is relatively high, and the in situ stress in approximately 80% of the rock mass is estimated to be close to or exceed the rock mass strength. This unfavourable situation will need to be mitigated by comprehensive support design and by well-designed and -executed undercutting of the orebody. Mining-induced stresses will likely exacerbate this issue.

- The structural model was reviewed and further developed by Pope (2010). One of the more significant consequences of the changes in the new resource and structural models is the change of initiation point from east to west.
- Re-analyzed fragmentation and cavability yielded similar results to the 2005 study. Consequently, the current mine design criteria did not change materially. In September 2010, as a result of additional fragmentation and draw geometry work done by OT LLC Geotechnical engineers, OT LLC decided to reduce the drawpoint spacing from 15 x 15 m to 14 x 14 m.

The underground confirmatory tunnel (termed the characterization drift) planned to transect the orebody at approximately 1,300 m below the surface is underway and has already provided valuable information for the geotechnical assessment and analyses. The comprehensive ground control and monitoring program jointly designed by OT LLC, SRK, and third-party consultants is providing information about ground behaviour and the character of the rock mass and large-scale structures.

16.3.2 Geotechnical Data

Geotechnical data collection for the Hugo North deposit is based on geotechnical logging resource delineation, geotechnical drillholes, and geotechnical mapping from the Hugo North Shaft No. 1 and drives completed by 2009. The geotechnical work consists of the following:

- The intact testing database was updated with strength data collected between 2005 and 2007. The database now contains 613 UCS tests (cf 272 in 2005). Memorandum issued by OT LLC in November 2007.
- In situ stress testing using CSIRO HI Cells was completed on six levels in Shaft No. 1 during sinking by Mining Measurement Services (MMS). These data are summarized by AMC Consultants Pty Ltd (AMC) in the report "Hugo North Shaft No. 1 – Summary of Rock Stress Measurements" (Appendix 10.4.4 of IDP10).
- The orientated geotechnical core log summary by OT LLC was a reworking of the pre-2005 orientation data and included holes drilled since 2005. The summary also included a statistical treatment of fracture frequency and RMR data for geotechnical domains at Hugo North. This memorandum was received in draft form in January 2008.

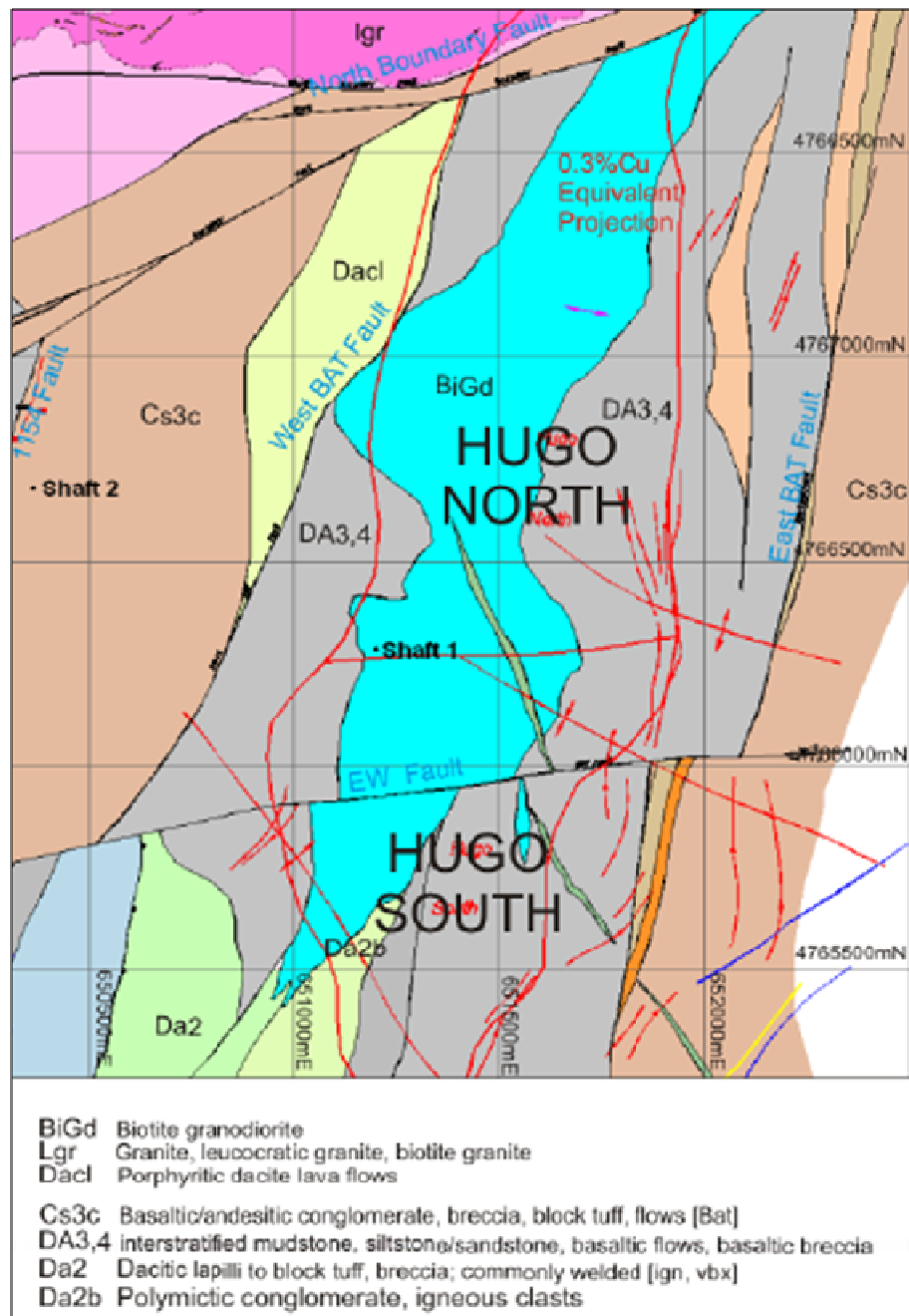
The conclusions of this work were used in the updated fragmentation, cavability, and mine design criteria analyses.

16.3.2.1 Geology and Orebody Geometry

The current geological and orebody model of the Hugo North deposit is best characterized in "Oyu Tolgoi Project, Mongolia Integrated Development Plan" by AMEC Ausenco Joint Venture from July 2005.

The lithological model is based mainly on drilling data and to some extent on mapping of the exploration block. The host rock geology is illustrated in Figure 16.12.

Figure 16.12 Geology in the Vicinity of Hugo North Orebody (SRK 2009)



The main units within the area affected by the Hugo North cave (from youngest to oldest) are listed below.

Carboniferous:

- Basaltic lava flows, minor intercalated breccia and tuff (BasL).
- Basaltic lapillic tuff (Bat).
- Basaltic andesite lava flows (AndL).
- Volcanic sandstone, carbonaceous siltstone, sandstone, and conglomerate.
- Andesitic ash flow tuff (AndIgn).

Devonian:

- Basaltic flow breccia and derived coarse volcanoclastics (unit L).
- Carbonaceous shale and laminated siltstones, conglomerates, and sandstones.
- Dacitic block ash tuff (Vbx).
- Dacite ash flow tuff (ignimbrite) units (Ign).
- Augite Basalt flow and related breccias (Va).

A wide variety of felsic to mafic dykes is found throughout the exploration block and in drillholes. Post-mineral dykes comprise basalt (Ba), rhyolite (Rhy), hornblende-biotite andesite (HiBiAnd), and most importantly biotite granodiorite (BiGd) intrusive units. The granodiorite and basalt dykes were modelled explicitly as 3D solids interpreted and simplified based on the available drillhole intersections. The property also contains variably altered and mineralized porphyritic quartz monzodiorite (QMD) dykes that may be genetically related to the copper-gold porphyry systems.

The Hugo North orebody forms the northern extension of the Hugo Dummett deposit, which lies north of latitude 4,766,400N. The separation between Hugo South and North is considered to be the 110 Fault. Hugo North extends in the Ivanhoe Mines–Entrée Gold Joint Venture property, referred to as Shive Tolgoi (Lookout Hill). The Hugo North deposit has been drill tested to 625 m north of the Ivanhoe Mines–Entrée Gold JV property boundary, onto an area of Lookout Hill known as Copper Flats.

The Hugo North orebody strikes generally north 28.5° west, with the zone plunging to the northwest. Its current base, as defined by the northern-most drilling, is near the -400 m elevation, 1,600 m below surface. The total width of the zone across all grade shells varies along strike from 200 m to more than 500 m. The mineralization dips generally to the east from as low as 40° to as high as 80° but is generally above 60°.

16.3.2.2 Structural Assessment

The structural characterization of the Hugo North orebody is based on a structural model developed by LGSi consultants and on a geotechnical structural assessment compiled by OT LLC based on geotechnical drillhole data.

The most comprehensive description of the large-scale structures is in P. Lewis' report "Structural Model for the Hugo Dummett Deposit" from July 2004 and in "Revised Fault Interpretation for the Hugo Dummett North Deposit," April 2006.

The most comprehensive analysis of the orientated drill core from Hugo North is presented in OT LLC (P. Voulgaris), "Hugo North 2007 Logging Study" January 2008. The Entrée Gold data summary were compiled in January 2006 (P. Voulgaris) "Entrée Joint Venture Geotechnical Data Review," for all resource drilling completed up to December 2005 (IDP10 Appendix 10.4.2).

Large-Scale Structures

Several large-scale faults throughout and in the vicinity of the Hugo North orebody were interpreted and explicitly modelled in 3D (Figure 16.13). Most of them are interpreted as steeply dipping. Based on the geotechnical core logging; however, a number of intersections of poor quality ground could be attributed to large-scale structures that are not explicitly modelled (Figure 16.14). This view is supported by LGSi and by OT LLC data analyses (see Lewis 2006 cited above).

For cavability and mine development strategy, the most important structures currently modelled are faults bounding the orebody. In addition to the steeply dipping structures, there appear to be a significant number of moderately and shallow dipping faults that will play an important role in the cavability assessment.

Biotite granodiorite dykes form another system of "large-scale structures" that could have geotechnical significance. One of the key objectives of confirmatory tunnel (characterization drift) is to verify character of large-scale structures. Example of two types, localized fractured zone and large-scale fault zone, is illustrated in Figure 16.15. The new structural information will be analysed in the next stage of the study.

Figure 16.13 RL 200 Plan of the Hugo North Deposit with Major Structures Highlighted

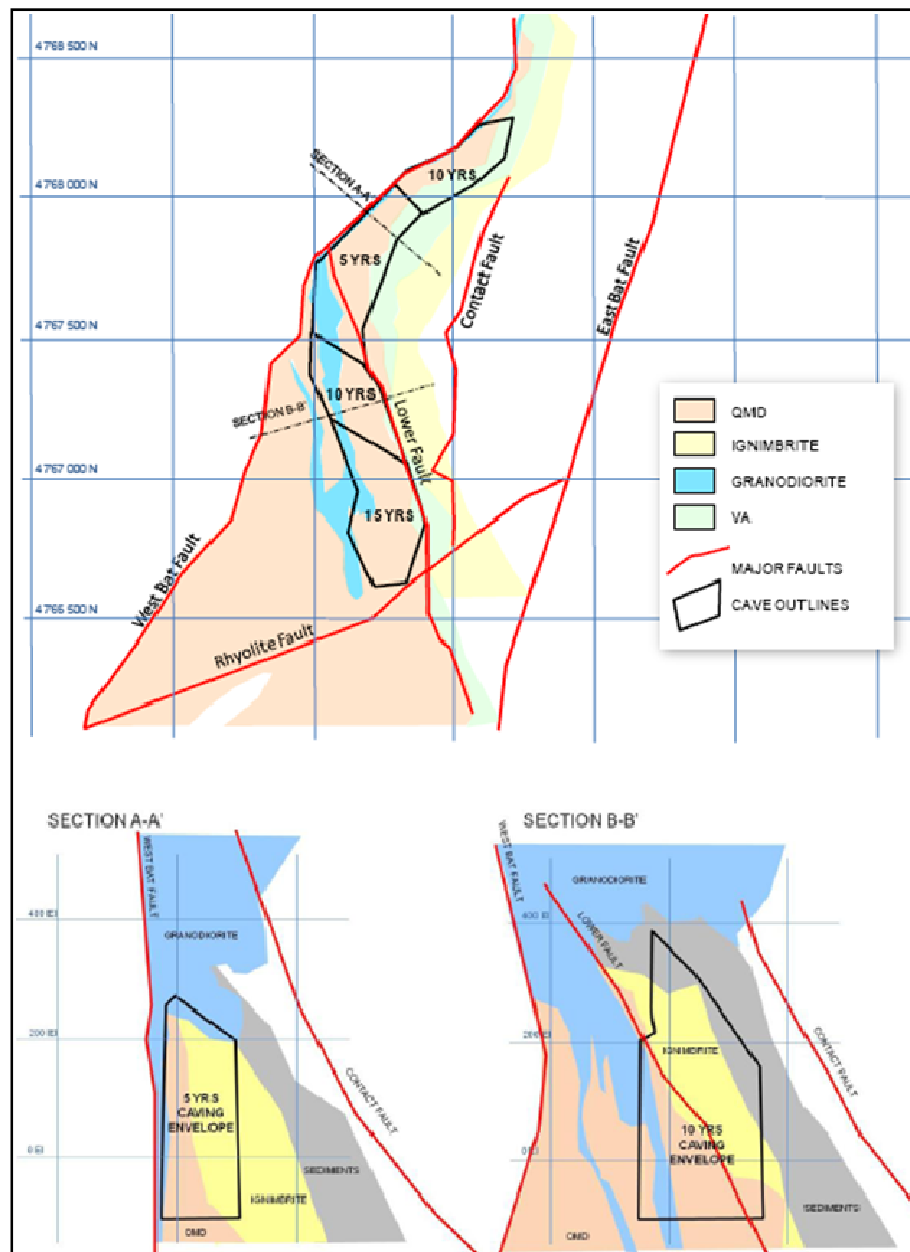
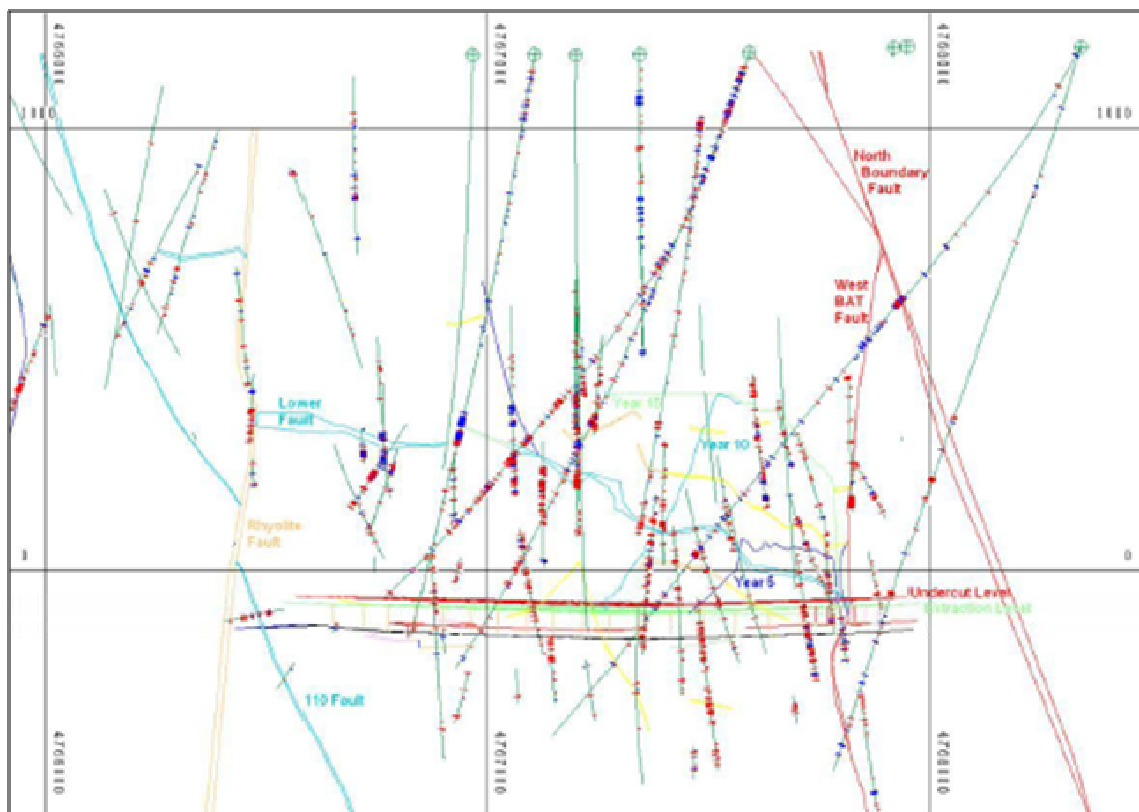


Figure 16.14 Isometric View (looking northeast) showing Amount of Faulting Logged from Core (SRK 2009)



Note: The red intervals indicate open (sheared or gauge filled) structures and yellow intervals indicate broken zones. (after SRK 2005 report).

Figure 16.15 Localized Fractured Zone (left) and Large-Scale Fault Zone (right) from Drillhole EGD 104A (depth 1,330 m and 1,503 m)



16.3.2.3 Small-Scale Structures

Several types of small-scale structures have been identified in the Hugo North orebody (joints, bedding planes, veins). OT LLC compiled a summary of these small-scale structures based on current drill core logs in March 2005, which are reworked in January 2008 (IDP10 Appendix 10.4.1).

It is apparent that most of the materials within the economic limits of the orebody could be characterized as jointed rock mass with at least four to five joint sets. Both strong (stepped rough) and weak joints has been encountered (Figure 16.16) but typically joint conditions are undulating smooth.

In general, the stereographic analysis of orebody domains (BiGd, IGN, QMD, VA) completed in 2008 shows good comparison of joint sets observed in the 2005 geotechnical review. Joint sets considered to be of “high” and “moderate” confidence (2005) are primarily represented.

Figure 16.16 Slickensided but Undulating Joint and Rough Stepped Joint Surface Drillhole EGD 119, Approximately 1,030 mRL



Underground Mapping

Underground mapping was carried out in the QMD (Shaft No. 1 from 500 to 1,300 m below surface and on the 1300 Level) and BiGD (shaft 50–500 m depth; 1,165 mRL) domains. These data are presented within the 2008 OT LLC Geotechnical Logging Summary (IDP10 Appendix 10.4.1). Mapping continues to be done through the characterization drift and other development on various levels and data will be analyzed in DIDOP.

QMD Domain

Comparison of the QMD joint orientation data shows a good representation of major subvertical to vertical joint sets in both the drillhole and mapping stereonets; random (or possibly less persistent) joint sets appear to scatter the data in the drillhole analysis. This scatter is possibly a result of data collection over significant horizontal and vertical

intervals through the domain, compared to the geotechnical mapping data that have been collected on the 1300 Level only.

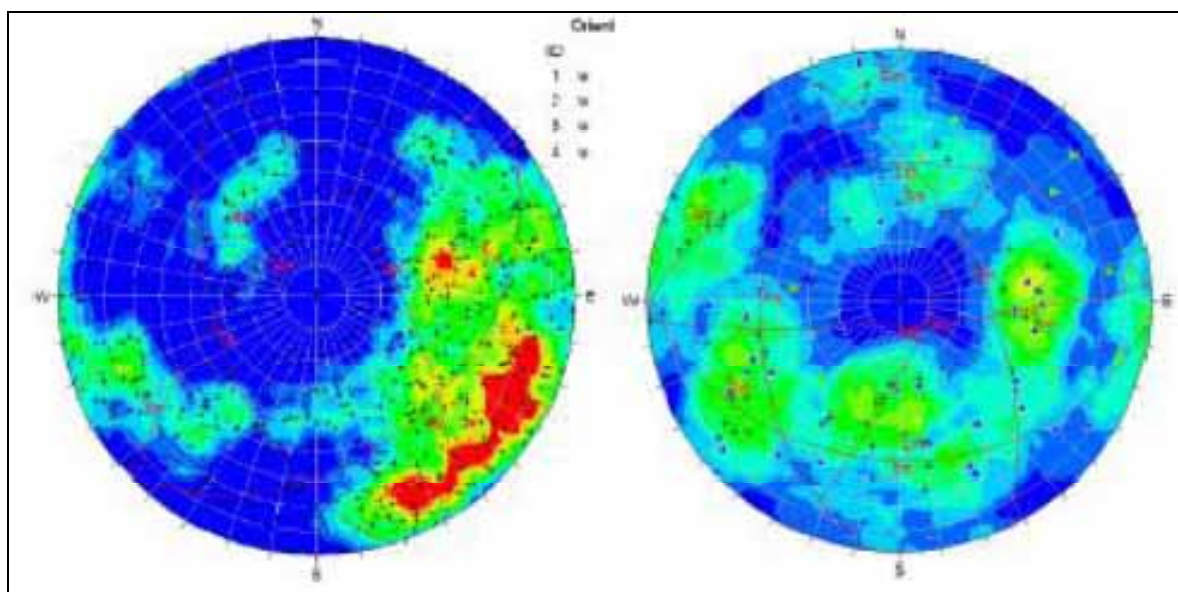
Orientated core study undertaken in 2005 identified subhorizontal sets that have not been seen in the Shaft No. 1 mapping or in the 1300 Level development. Nor were they identified the 2008 reworking of the orientated core logs. The 2005 orientated data were predominately limited to holes drilled east to west at 60°, and greater weighting was placed on flat sets in that study. The addition of the geotechnical holes drilled north to south increases confidence in the 2008 results, and the horizontal sets are more random in their distribution.

The main feature to note in Figure 16.17 is the steeply dipping set to the north-west. These sets are parallel to the Rhyolite Fault, a deposit-scale fault, and suggest that local joints will be controlled by proximity to the large-scale structures.

BiGd Main Domain

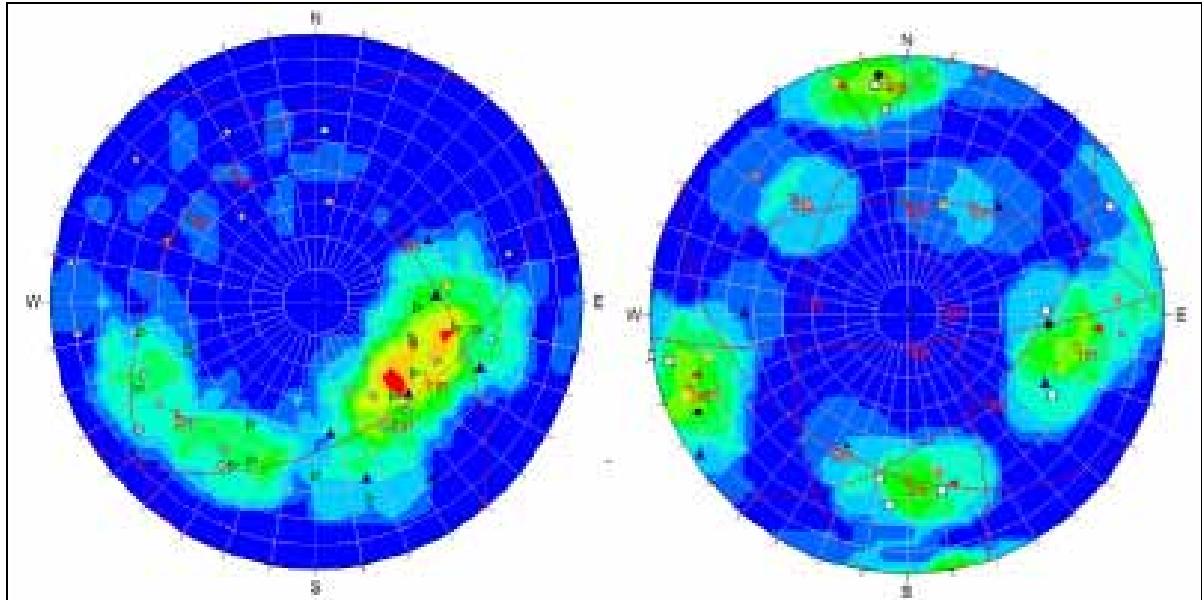
Mapping data collected from Shaft No. 1 through the BiGD main domain show generally good comparison to oriented data collected from drillholes. It should be noted that a clear bias has developed in the drillhole data, with an under-representation of southerly dipping joint sets (Figure 16.18).

Figure 16.17 Stereonets of Poles from QMD Domain Showing Comparison of Joint Sets (SRK 2009)



Note: 2008 Level 1300 mapping (lower left); 2008 combined mean orientation from drillholes (lower right).

Figure 16.18 Stereonets of Poles from BiGD Main Domain Showing Under-represented Southerly Dipping Joint Sets in Oriented Core Data



Note: 2008 combined mean orientation from drillholes (left); 2008 shaft mapping 50 to 500 m depth (right).

16.3.2.4 Rock Mass Characterization

Rock mass characterization includes an assessment of all the parameters that will affect the rock mass competency. For most assessment, the rock mass is treated as a continuum transected by large-scale structures (discontinuum) that are assessed separately. Laubscher's Rock Mass Rating system was applied to determine rock mass competency.

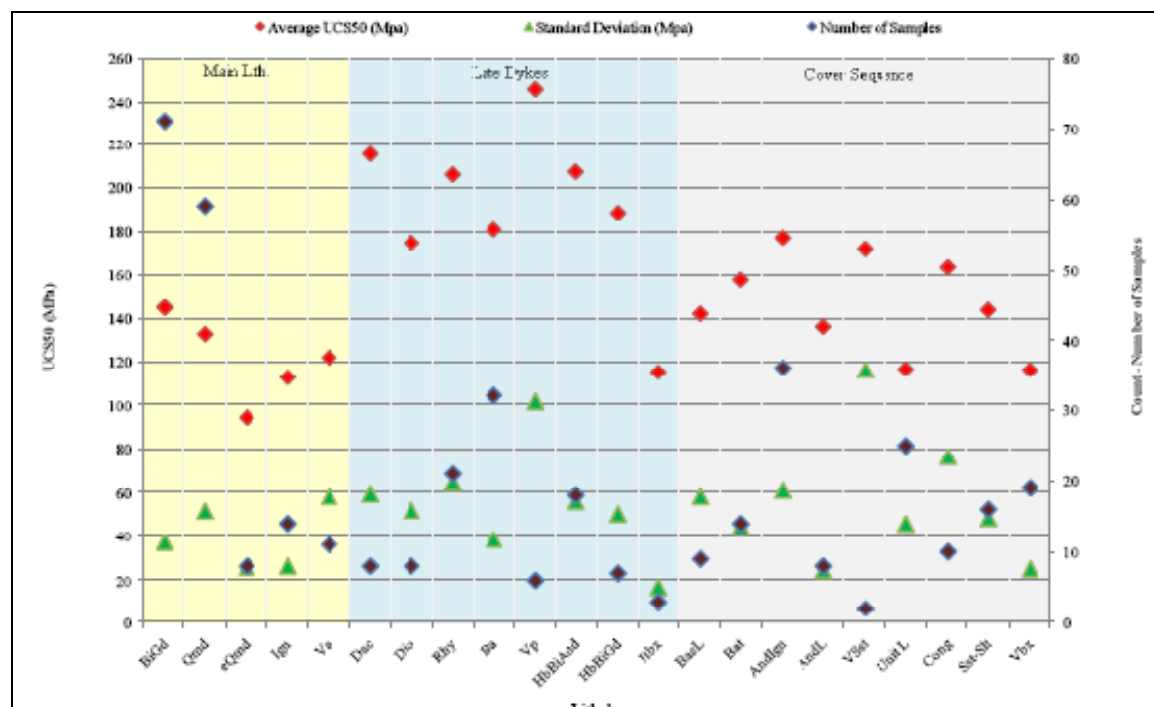
16.3.2.5 Drill Core Logging

As mentioned previously, all geotechnical logging was conducted by OT LLC staff and methodology and results were consulted with SRK. In total, geotechnical data from 89 drillholes were used.

16.3.2.6 Laboratory Testing

The intact, joint shear, and tri-axial testing completed to date were summarized by OT LLC in November 2007 (IDP10 Appendix 10.4.3). The comparison of average values for all domains is illustrated in Figure 16.19.

Figure 16.19 Average UCS Values for all Lithologies for 2004 (UCS Uncorrected) and 2007 (UCS₅₀)



Note: The mean values of BiGd, Ign, Qmd, and Va lithologies are approximately 145 MPa, 130 MPa, 113 MPa, and 122 MPa, respectively. It should be noted that the laboratory values represent the upper strength of the possible values for individual rock types. Due to the highly veined nature of the Ign and Va units, they are under-represented in the strength data.

16.3.2.7 Underground Mapping, Drilling, and Instrumentation

The main objective of the characterization drift program was to provide access for underground mapping, sampling, instrumentation, and drilling to investigate the geotechnical and geological situation in more detail than could be done from the surface. The underground development addressed, in the SRK 2009 report was in Hugo South, a different structural block than the Hugo North orebody.

Comparison of mapping and drilling results to date subsequent to the SRK 2009 report have shown no significant variation from the structural situation predicted from the surface drilling.

Results to date have provided Itasca and SRK enough information to calibrate their 2D and 3D models and verify drift dimensions, pillar sizes, and ground support requirements at the PFS level.

16.3.2.8 Geotechnical Domain Model

The preliminary geotechnical model was evaluated in 2005 using statistical spatial analyses (Leapfrog software). It was concluded that data were insufficient for such analyses at that time, but the validity of this technique for base case geotechnical

domain construction should be evaluated when all the data are available and anisotropy of the deposit is known.

On the basis of intact rock strength, joint orientations and conditions, and RMR, the geotechnical domain model is primarily the lithological model. The following domains are currently applied:

Within the Hugo North cave zone:

- QMD – the main mineralized intrusion; will be the predominate unit/domain in the undercut and extraction levels.
- VA – highly fractured unit stoped out by the QMD.
- IGN – highly fractured unit with dickite (low friction) as the dominate joint infill.
- Central – the Devonian units above the Ign, between the bounding faults: West and East BAT, North Boundary, and 110 Faults.
- BiGd Feeder – the relatively thin granodiorite dykes that appear to be the feeds for the main BiGd at higher levels. The BiGd feeders will be encountered in the undercut and extraction levels. This shows different joint orientations than in the BiGd main body.
- BiGd Main – the main mass of the granodiorite dyke in the higher levels of the cave.

Within the Hugo North subsidence zone:

- North – the Devonian and Carboniferous units between the North Boundary Fault and the West BAT fault at the north end of Hugo North
- North Granodiorite – the large Carboniferous granodiorite intrusion to the north of the North Boundary Fault
- West Carboniferous – the Carboniferous units to the west of the West BAT Fault
- West Devonian – the Devonian units to the west of the West BAT Fault
- East – the Devonian and Carboniferous units to the east of the East BAT Fault. They have not been separated because of the sparse amount of data.

The same domains were applied in the January 2006 summary of the Entrée Gold geotechnical logging from the resource drilling. This does however, show that the jointing is significantly different within the Hugo North cave zone to warrant a lateral domain boundary approximately where Hugo North turns to strike north-east.

Underground mapping experience so far suggests that there will be geotechnical subdomaining in lithological units but on a smaller scale than could be interpreted from current drillholes.

16.3.2.9 Stress Assessment

Stress information has been compiled by AMC. Several stress measurements and estimates have been completed in the past, but CSIRO HI Cell Overcoring tests during shaft construction and in lateral development on the 1300 mRL level are considered the most reliable.

The main aims of the Shaft No. 1 measurements were to provide a good depth versus stress relationship and confirm the major principal stress direction.

The information provided by AMC in January 2008 (by email) was used for fragmentation analyses. AMC made the following comments:

- The measured major and intermediate principal stress magnitudes are slightly lower than the previous “most likely thrust regime” estimate by AMC.
- The current understanding is that major principal stress σ_1 has a bearing of 050° (approximately in the north-easterly direction), dip of 00° (possible dip to the north-east), with a magnitude [MPa] of $0.052 \times \text{depth}$.
- Intermediate principal stress σ_2 has a bearing of 150° (approximately south-east), with a dip of 00° and a magnitude [MPa] of $0.033 \times \text{depth}$.
- Minor principal stress σ_3 has a bearing of 050° , dip of 90° , and a magnitude [MPa] of $0.027 \times \text{depth}$.
- As previously suggested in AMC’s stress estimate report, the measurements indicate that a more complex/non-linear depth versus stress relationship is possible (and horizontally away from Shaft No. 1).
- It is assumed that at the proposed production horizons the major principal stress is approximately twice the vertical stress.
- The orientation of the principal stresses from the six CSIRO HI cell sites correlates well with the observed borehole breakout but does not match the WASM AE orientations.
- It is understood that further rock stress measurements are planned for the 1,300 mRL Level to provide lateral information about the Oyu Tolgoi stress field and its relationship with structures.

The complete AMC report was received in the end of February 2010 and is included in IDP10 Appendix 10.4.4.

16.3.2.10 Hydrogeology Assessment

A preliminary hydrogeological assessment was completed by Aquaterra Consultants conclusions are presented in a technical memorandum dated 7 December 2004 and updated in November 2007. The groundwater model was developed using MODFLOW-96 to simulate a 35 year schedule for two scenarios of mine workings development. The approach to model the setup is reasonable and generally provides as much detail as can be expected at this level of study.

Two sets of dewatering predictions are given for each development scenario, one with and one without the effect of subsidence on bedrock vertical hydraulic conductivity. Predicted dewatering rates rise as more workings are installed and are significantly greater for simulations of major subsidence, with maximum inflow rates approaching or exceeding 100 L/s for all scenarios. This indicates a very dry mine. However, these results will be controlled to a significant degree by the distribution of hydraulic conductivity. The limitations to the equivalent porous media approach were not discussed.

To proceed with the model any further, more detailed data such as modelled head distributions over time would be required. Also, the limitations of using MODFLOW-96 in modelling mine dewatering inflow in very low permeability (K) environments may be an issue for a detailed assessment. Drilling and development of the characterization drift as at July 2010 have not shown any significant water inflows.

16.3.3 Cavability Assessment

16.3.3.1 Background

Cavability is defined as a continuous failure of the rock mass until all void space is filled with broken rock. Once draw continues, failure in the cave back continues to propagate. It is important to recognize that sustainable cave is not the same as a rock mass collapse.

Main parameters influencing cavability are:

- Rock mass competency.
- Large-scale structures (those not included in the rock mass).
- In situ stress (orientation and magnitude).
- Induced stress (mainly influenced by the direction of undercutting and rate of caving).
- Excavation geometry.

Detail descriptions of these parameters are provided in the SRK 2005 report.

16.3.3.2 Approach

The cavability of the rock mass is one of the fundamental issues for the current mining method. Two principal techniques used in the industry to investigate the cavability of the rock mass: an empirical method and a method based on numerical analysis. In the context of the Hugo North orebody geometry and level of investigation, Laubscher's empirical method was selected.

The cavability is usually not a major problem on most caving operations because the orebodies are so large that the hydraulic radius of the footprint greatly exceeds the caving hydraulic radius (such as Hugo North). In these cases, it is usually only a question of how large an area is required to meet the initial production requirements. The bulk of the tonnage mined from block caving mines comes from the lateral

extension of the cave. For orebodies where the hydraulic radius of the caving envelope is limited, several different approaches are typically required in deciding on the cavability.

Both current methods, empirical and numerical, must be treated as guidelines to place the deposit in the right “ballpark.” Because of the complexity of the geology and disturbing forces acting in the cave, it is not possible to obtain precise numbers and a sensitivity analysis must be conducted.

16.3.3.3 Input Data

As mentioned previously, the cavability of the rock mass is mainly driven by the rock mass competency (RMR) and by the disturbing forces: stress in deeper caves and by gravity closer to the surface. It is therefore important to apply mining adjustments to basic RMR values.

Laubscher’s method allows adjustments for stress, joint condition, water, weathering, and blasting. In the case of Hugo North only stress and joint orientation adjustments were applied. The weathering adjustment is applicable only to the upper, clay-rich weathered horizon. The RMR numbers were adjusted 95% for joint orientation and 90% to 97% for stress depending on depth. The RMR values for major rock types used for mining adjustments are shown in Figure 16.20.

Figure 16.20 RMR and MRMR Values for Major Rock Types, Caving Periods, and Resultant Hydraulic Radius

Rock Type	Year	Rock Mass Rating (RMR)			Modified Rock Mass Rating (MRMR)			Hydraulic Radius		
		Lower Quartile	Median	Upper Quartile	Lower Quartile	Median	Upper Quartile	Lower Quartile	Median	Upper Quartile
IGN	5									
	10	38	43	46	33	37	40	17	19	21
	15	41	45	46	35	39	40	18	20	21
	20	45	47	48	39	40	41	20	21	22
Va	5	42	48	52	36	41	45	19	22	24
	10	39	44	49	34	38	42	17	20	22
	15	39	46	49	34	40	42	17	21	22
	20	43	49	52	37	42	45	19	22	24
QMD	5	41	49	54	35	42	46	18	22	25
	10	40	46	51	34	40	44	18	21	23
	15	39	44	50	34	38	43	17	20	23
	20	39	44	50	34	38	43	17	20	23
BIGd	5	39	42	48	34	36	41	17	19	22
	10	38	43	48	33	37	41	17	19	22
	15	41	43	49	35	37	42	18	19	22
	20	20	43	44	17	37	38	8	19	20

16.3.3.4 Results for Hugo North

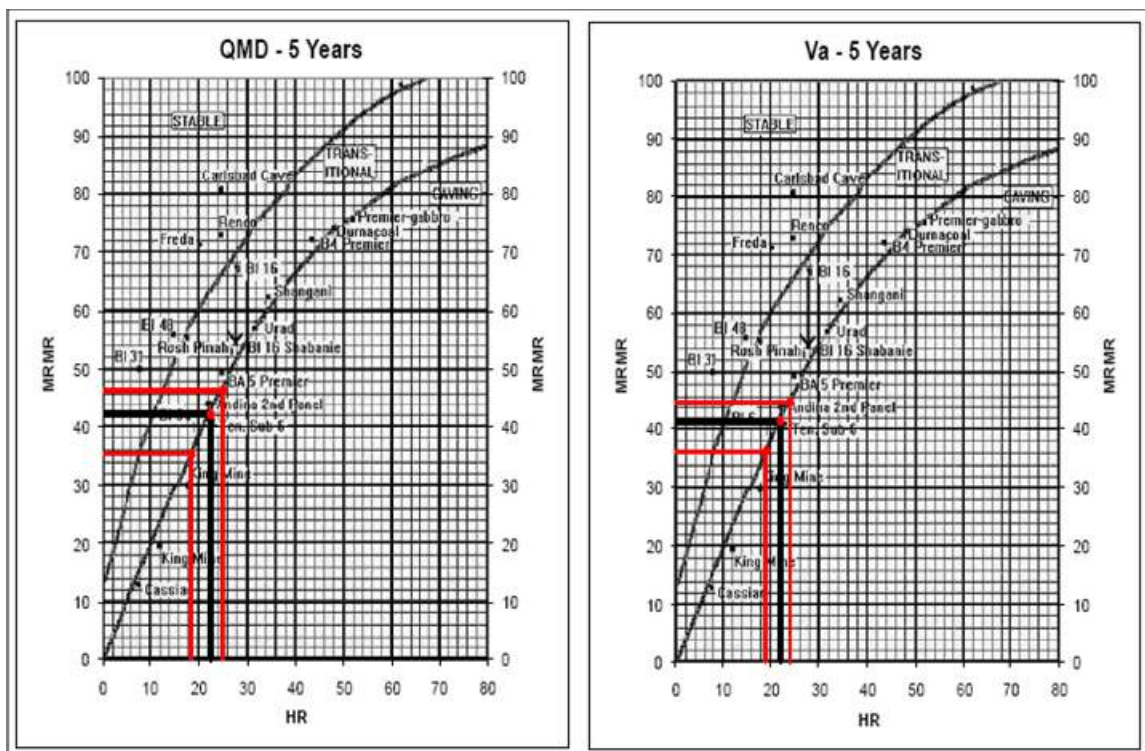
The range of the MRMR values for QMD (representing the major lithology of the orebody) and for sediments (hanging wall) were plotted against both 1990 and 2000 stability graphs (Figure 16.21).

Based on current data the hydraulic radius to initiate and sustain caving of the rock mass is approximately 20 to 23 m for the median values (5, 10, 15, and 20 year caving shapes) for QMD, Va, and Ignibrite units. This represents an area of 80 x 80 m to 90 x 90 m. The upper quartile HR values are 20 to 25 m and the lower quartile values 17 m to 20 m. Values for the QMD and Va represent most of the orebody.

Large-scale structures such as faults and shear zones will influence cavability. Geotechnical core logging indicated that such features are present in the Hugo North orebody. The large-scale structure adjustment was introduced in Laubscher's classification in 2000 and is considered to be conservative due to the lack of case studies available to date. Applying such an adjustment will further reduce the hydraulic radius by approximately 5% to 10%.

The footprint dimension of the Hugo North orebody at the base of the proposed undercut level is approximately 200 m x 1,200 m. This is equal to a hydraulic radius (HR) of 86 m. The footprint for the Entrée part of the orebody is approximately 150 m x 700 m, resulting in an HR of 62.

Figure 16.21 Cavability Assessment Using Laubscher's 1990 and 2000 Stability Graphs



Even if the critical span criterion to assess cavability were to be used for analysis, the width in the narrowest area of the orebody is at least 150 m, resulting in an HR of 38. The HR in the current footprint, including the Entrée part of the orebody, is sufficient to initiate and sustain caving.

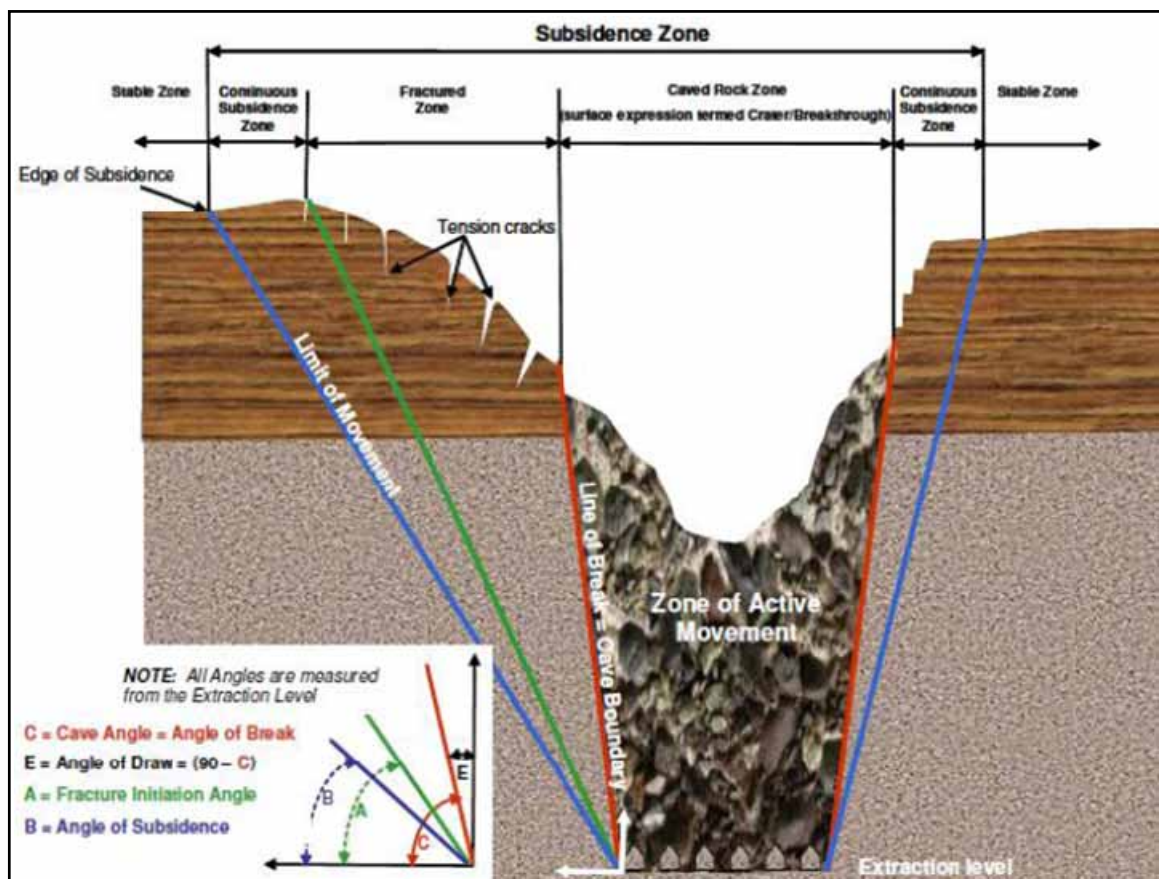
16.3.4 Subsidence Modelling of Hugo North

16.3.4.1 Background

Several subsidence predictions for the Hugo North caving have been completed in the past. Initial subsidence predictions were based on Laubscher's empirical approach. In 2005 SRK developed 2D FLAC analyses on selected sections and in 2008 Itasca undertook full 3DEC modelling.

For the SRK 2009 study, Andre Van As (Rio Tinto Copper Projects) recommended that the subsidence definition developed for the Mass Mining Technology study (MMT) be used. Figure 16.22 illustrates the term definitions.

Figure 16.22 Definition of Subsidence Zone (after MMT – Permission from Rio Tinto)



In 2009, the Itasca-modelled main large-scale interpreted structures were included. Although the complexity of the subsidence models increased as the study progressed, the level of simplification of such models in comparison to “simple” geological interpretation need to be understood; the two sections are compared in Figure 16.23.

The results of SRK’s 2005 2D numerical analyses indicated subsidence angles of between 63° and 75°, depending on the location in the cave. The results of 3DEC analyses, including major faults, indicate that the subsidence shape, and thus angles, is highly sensitive to the faults for assumed fault strength parameters. A plan view of the 3DEC model is shown in Figure 16.24. Itasca concluded that “the model results suggest that the region of large displacement coincides best with the 60° cone, although there are still some large displacements that can extend to the 50° subsidence limit” (Figure 16.25).

As discussed above, the current structural model is based primarily on drilling information and large-scale structures; intermediate and small-scale structures are not included.

SRK preliminary guidelines based on empirical design and simplified FLAC 2D sections recommend siting important structures no less than 60° from the footprint. The result of Itasca’s model support this conclusion.

Figure 16.23 Subsidence Modelling Compared to “Simple” Geological Interpretation

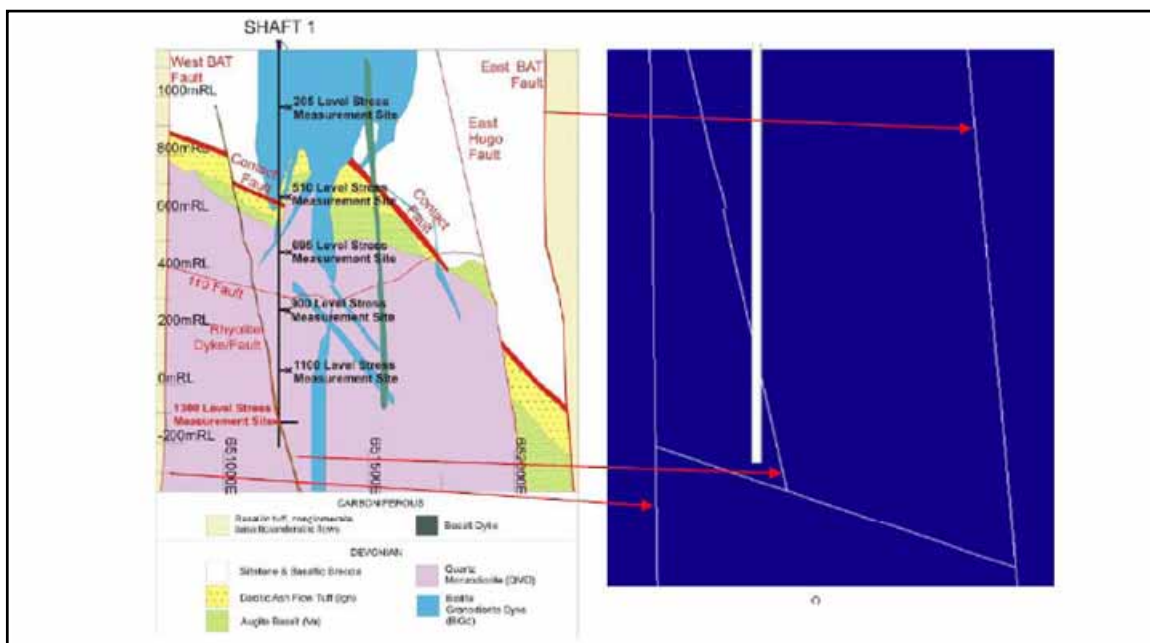


Figure 16.24 Plan View Illustrating 3DEC Model with Simplified Faults (Itasca 2009)

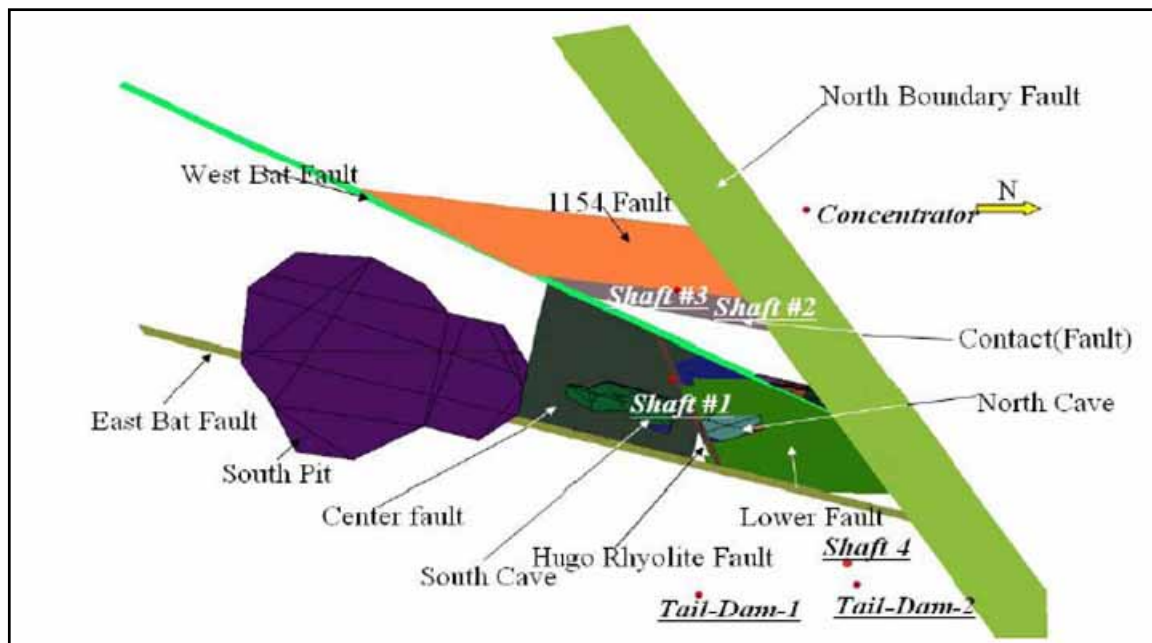
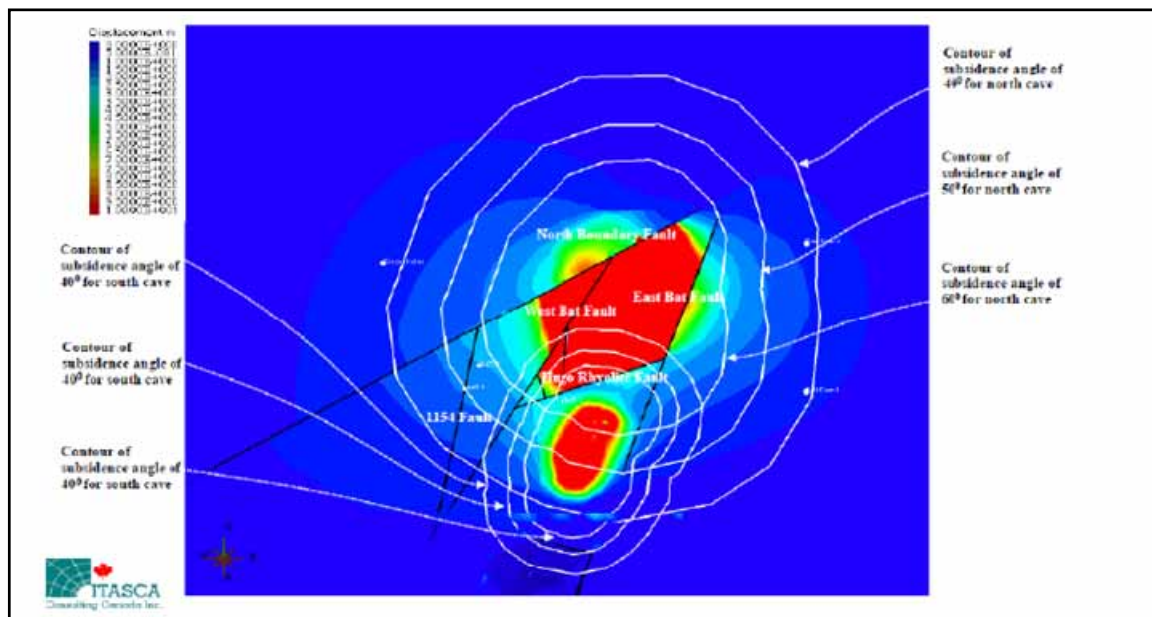


Figure 16.25 Subsidence Zone Based on 3DEC Modelling (Itasca 2009)



16.3.5 Fragmentation Analysis

16.3.5.1 Background

The fragmentation and hangup analysis for the Hugo North deposit was carried out using the Block Cave Fragmentation (BCF) software. Input data were based on the results of an ongoing geotechnical data collection program as provided (in draft) by OT LLC in January 2007 (IDP10 Appendix 10.4.5). The data contained an assessment of the rock mass properties, joint orientations, and frequencies. Results of the updated laboratory testing program are presented as IDP10 Appendix 10.4.3. Stress measurements were completed using the CSIRO HI Cell Overcoring method. Stress measurement results from AMC are presented in IDP10 Appendix 10.4.4.

The main source of the data was derived from core drilling programs, supplemented by geotechnical mapping carried out in shaft and drive excavations at Hugo North.

The geotechnical domains are based on the lithology types as defined in the geotechnical report. Owing to the inherent variability of rock mass properties and the uncertainty associated with data collected from core drilling, the effect of variations in the key input parameters was evaluated. The results are presented as a range of likely values, with the most likely outcome (based on average/expected parameters) shown for each set of results.

The BCF program uses joint set data, fracture frequencies in coreholes, stress, and rock strength properties to calculate likely fragmentation for various block caving configurations. Factors such as the draw rate, width of caving zone, cave pressure, and drawpoint width are considered. The program is based on empirically derived relationships, largely based on the experience of Dr. DH Laubscher, as well as basic rock engineering principles.

16.3.5.2 Primary Fragmentation Analysis

Primary fragmentation is the fragmentation of the rock as it separates from the surrounding rock in the cave back and falls onto the top of the draw column. The BCF program first calculates primary fragmentation and then uses the results to calculate secondary fragmentation. A number of inputs are required for a primary fragmentation analysis. Those used in the Hugo North assessment are presented below, together with a discussion of assumptions made.

Rock Mass Properties

The rock mass at Hugo North is characterized by through-going fault structures. These structures are expected to result in additional fine materials in the block cave, which can inhibit the secondary fragmentation process in the draw column by providing cushioning between larger blocks. For fragmentation analysis the rock mass strength was described without fault/dyke data. The effect of these structures was included in the analyses as additional "fines" affecting the secondary fragmentation.

The key rock mass parameters are presented in Table 16.11. Joint dip, direction, and condition data were entered as measured averages for all cases.

In addition to these key data, the rock mass contains healed fractures. The effect of healed joints was included in the analyses as part of the rock matrix, resulting in a reduction of rock strength.

BCF simulations were run for a number of different scenarios involving different times, stress scenarios, and cave angles. The various scenario inputs are listed in Table 16.12.

Secondary fragmentation analysis determines how the block sizes reduce in the draw column, as the blocks are drawn down to the drawpoint. The analysis is based on the draw height, which is the distance that a block travels down a draw column from its original position in the cave back to the drawpoint. The secondary fragmentation model inputs are reported in Table 16.13. The values in the "Additional Fines" column were determined based on fracture frequency in ore hole logs.

Table 16.11 Key Rock Mass Parameters – Average Values

Rock Type	Year	Base			Coarse			Fine			Fracture Frequency per Metre	Fracture Condition	Joint Sets
		Rock Mass Rating (RMR)	Hoek & Brown m-value	Intact Rock Strength (MPa)	Rock Mass Rating (RMR)	Hoek & Brown m-value	Intact Rock Strength (MPa)	Rock Mass Rating (RMR)	Hoek & Brown m-value	Intact Rock Strength (MPa)			
IGN	5												
	10	43	17	113 MPa	46	20	127 MPa	38	14	99 MPa	2	30	3
	15	45	17	113 MPa	46	20	127 MPa	41	14	99 MPa	2	30	4
	20	47	17	113 MPa	48	20	127 MPa	45	14	99 MPa	2	30	3
Va	5	48	17	122 MPa	52	20	180 MPa	42	14	64 MPa	2	30	5
	10	44	17	122 MPa	49	20	180 MPa	39	14	64 MPa	2	30	5
	15	46	17	122 MPa	49	20	180 MPa	39	14	64 MPa	2	30	4
	20	49	17	122 MPa	52	20	180 MPa	43	14	64 MPa	2	30	4
QMD	5	49	17	133 MPa	54	20	184 MPa	41	14	82 MPa	2	30	4
	10	46	17	133 MPa	51	20	184 MPa	40	14	82 MPa	2	30	4
	15	44	17	133 MPa	50	20	184 MPa	39	14	82 MPa	2	30	4
	20	44	17	133 MPa	50	20	184 MPa	39	14	82 MPa	2	30	3
BGd	5	42	17	145 MPa	48	20	180 MPa	39	14	120 MPa	2	30	3
	10	43	17	145 MPa	48	20	180 MPa	38	14	120 MPa	2	30	3
	15	43	17	145 MPa	49	20	180 MPa	41	14	120 MPa	2	30	5
	20												

Table 16.12 Primary Cave Simulation Inputs

Case	Input Geology Case	Cave Orientation (degrees)		Stresses (MPa)			Stress Spalling	
		Dip of Caving Face	Dip Direction of Cave Face	Dip Stress	Strike Stress	Normal Stress	% Spalling	Allow Stress Fractures?
Base Case	2a - Base	45.00	55.00	65.00	41.00	0.00	0.00	Yes
Coarse Case	2b - Coarse	45.00	55.00	44.00	20.00	0.00	0.00	Yes
Fine Case	2c - Fine	0.00	0.00	95.00	95.00	0.00	0.00	Yes

Table 16.13 Secondary Fragmentation Simulation Inputs

Rock Type	Year	Variable Inputs		Static Inputs													
		Rock Density (t/m ³)	Additional Fines (% ff > 15)	Draw Height (m)	Maximum Caving Height (m)	Draw Width (m)	Swell Factor	Rate of Draw (cm/day)	Upper Bell Width (m)	Lower Bell Width (m)							
IGN	5																
	10	2.85	2.99%	20 120 220 320 400	1250	200	1.20	20	15.00	5.00							
	15		12.92%														
	20		6.47%														
Va	5	2.53	11.43%														
	10		12.81%														
	15		16.61%														
	20		6.47%														
QMD	5	2.77	10.94%														
	10		11.11%														
	15		13.62%														
	20		12.63%														
BiGd	5	2.44	7.06%														
	10		11.97%														
	15		10.58%														
	20																

Primary Fragmentation Results

The primary fragmentation results for the Hugo North orebody, based on average parameters, are summarized in Table 16.14.

All the domains would be classified as having “Good” fragmentation. Average block linear sizes are less than 1 m (less than 0.25 m³ in volume). Variation of the input parameters showed that the results are sensitive to the joint frequency and joint conditions. The assumed 25% variation in joint frequencies can result in large variations in the block dimensions of the stronger, less-jointed rock types. However, in spite of this variation, the average block linear dimension remains below 1 m. As a general rule of thumb, the maximum linear block size is about three times the average size. This means that the maximum block size expected in any of the rock types will be less than 3 m in length. These results do not differ significantly from the previous fragmentation studies. Given the maximum particle size reported in Table 16.14, IDOP noted that hang-ups would not be expected to be a serious problem after the cave has been initiated. These results do not differ significantly from previous fragmentation studies.

Table 16.14 Tabulated BCF Results for Primary Fragmentation

Case No	Rock Type	Primary Fragmentation				Secondary Fragmentation							
						20 m Draw Height				120 m Draw Height			
		% < 2 m	Average Volume (m³)	Corresponding Cubic Dimension (m)		% < 2 m	Average Volume (m³)	Corresponding Cubic Dimension (m)	Number of Hangups	% < 2 m	Average Volume (m³)	Corresponding Cubic Dimension (m)	Number of Hangups
Base	ISN	5											
		10	98.95	0.09	0.45	99.41	0.08	0.43	1	100.00	0.05	0.37	0
		15	100.00	0.02	0.27	100.00	0.02	0.27	0	100.00	0.02	0.27	0
		20	99.69	0.14	0.52	99.72	0.11	0.48	0	99.72	0.06	0.39	1
	Va	5	100.00	0.04	0.34	100.00	0.04	0.34	0	100.00	0.03	0.31	0
		10	100.00	0.05	0.37	100.00	0.05	0.37	1	100.00	0.03	0.31	0
		15	100.00	0.11	0.48	100.00	0.10	0.46	0	100.00	0.06	0.39	0
		20	99.07	0.14	0.52	99.41	0.12	0.49	5	100.00	0.08	0.43	0
	CMD	5	100.00	0.10	0.46	100.00	0.09	0.45	1	100.00	0.07	0.41	0
		10	98.39	0.22	0.60	98.75	0.20	0.58	18	99.12	0.14	0.52	17
		15	100.00	0.13	0.51	100.00	0.12	0.49	2	100.00	0.09	0.45	0
		20	96.18	0.48	0.78	97.75	0.42	0.75	81	99.75	0.27	0.65	36
	BGD	5	100.00	0.15	0.53	100.00	0.14	0.52	2	100.00	0.11	0.48	2
		10	99.84	0.54	0.81	99.85	0.49	0.79	92	99.93	0.39	0.73	43
		15	100.00	0.07	0.41	100.00	0.06	0.39	0	100.00	0.04	0.34	0
		20											
Course	ISN	5											
		10	99.84	0.26	0.64	100.00	0.23	0.61	15	100.00	0.14	0.52	2
		15	100.00	0.05	0.39	100.00	0.06	0.39	0	100.00	0.04	0.34	0
		20	99.07	0.29	0.66	99.45	0.24	0.62	36	100.00	0.14	0.52	1
	Va	5	100.00	0.08	0.43	100.00	0.07	0.41	0	100.00	0.06	0.39	0
		10	100.00	0.09	0.45	100.00	0.09	0.45	0	100.00	0.07	0.41	0
		15	100.00	0.25	0.63	100.00	0.23	0.61	11	100.00	0.18	0.56	4
		20	98.49	0.29	0.66	98.88	0.26	0.64	30	99.52	0.20	0.58	4
	CMD	5	100.00	0.19	0.57	100.00	0.18	0.56	8	100.00	0.15	0.53	5
		10	99.35	0.32	0.68	99.55	0.29	0.66	31	99.55	0.23	0.61	25
		15	100.00	0.24	0.62	100.00	0.22	0.60	3	100.00	0.18	0.56	2
		20	87.06	0.98	0.99	91.20	0.89	0.96	270	97.64	0.57	0.88	233
	BGD	5	100.00	0.17	0.55	100.00	0.16	0.54	5	100.00	0.13	0.51	0
		10	98.70	0.81	0.93	99.31	0.76	0.91	221	99.31	0.66	0.87	146
		15	99.47	0.08	0.43	99.53	0.07	0.41	1	99.53	0.05	0.37	0
		20											
Fine	ISN	5											
		10	100.00	0.04	0.34	100.00	0.03	0.31	0	100.00	0.02	0.27	0
		15	100.00	0.02	0.27	100.00	0.02	0.27	0	100.00	0.01	0.22	0
		20	100.00	0.02	0.27	100.00	0.02	0.27	0	100.00	0.01	0.22	0
	Va	5	100.00	0.01	0.22	100.00	0.01	0.22	0	100.00	0.00	0.00	0
		10	100.00	0.02	0.27	100.00	0.02	0.27	0	100.00	0.01	0.22	0
		15	100.00	0.05	0.37	100.00	0.04	0.34	0	100.00	0.02	0.27	0
		20	97.25	0.08	0.43	99.28	0.06	0.39	0	100.00	0.03	0.31	1
	CMD	5	100.00	0.02	0.27	100.00	0.02	0.27	0	100.00	0.01	0.22	0
		10	100.00	0.03	0.31	100.00	0.03	0.31	0	100.00	0.01	0.22	0
		15	100.00	0.05	0.37	100.00	0.05	0.37	0	100.00	0.02	0.27	0
		20	100.00	0.11	0.48	100.00	0.09	0.45	1	100.00	0.04	0.34	0
	BGD	5	100.00	0.02	0.27	100.00	0.02	0.27	0	100.00	0.01	0.22	0
		10	100.00	0.11	0.48	100.00	0.10	0.46	1	100.00	0.07	0.41	0
		15	100.00	0.02	0.27	100.00	0.02	0.27	0	100.00	0.01	0.22	0
		20											

16.3.6 Mine Design Parameters

This section describes mine design parameters that are directly influenced by the geotechnical constraints. It is an update of mine design parameters from Conceptual Study in the context of 2005 data and supporting studies. For completeness all relevant information from the conceptual study are repeated. The following headings are covered:

- Mining advance strategy
- Undercut strategy
- Layout geometry
- Ground support
- Draw height
- Cave dynamics

16.3.6.1 Mining Strategy

The main geotechnical constraints dictating mining strategy and concept are draw heights, caving envelope geometry, rock mass competency (geotechnical domain model), large-scale structures, and stress orientation.

Caving Method

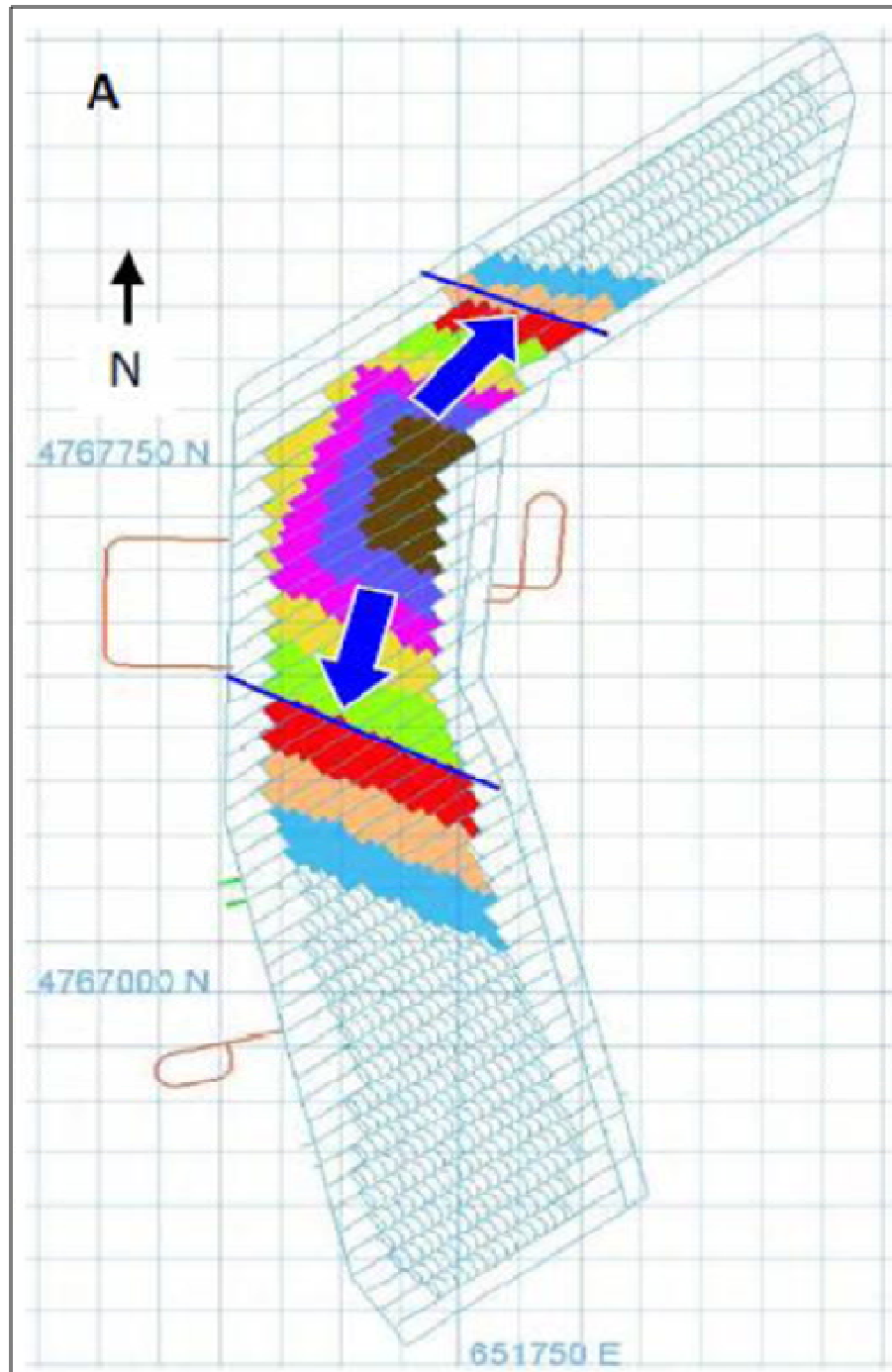
Several fundamental cave designs have been developed over the years. Grizzly and slusher versus LHD and horizontal versus incline are the most common basic types. Based on cave zone geometry, production flexibility, and the “keep it simple” rule, the standard horizontal LHD panel caving layout is recommended for the Hugo North caving envelope.

Cave Initiation

It is important to determine the number of cave initiation points (based on economics—the desired production rate) and location of the cave initiation points (advancing to weaker or stronger rocks, advancing toward or away from faults, and advancing toward or away from major principal stress). In the context of the large-scale structures (Contact and East Hugo faults), it was recommended that the cave be initiated on the eastern boundary and advance obliquely to the south-east and north-west at approximately 45° to the long axis (Figure 16.26).

As of this update (November 2010), the OT Study group is giving serious consideration to alternate cave initiation points. This is being done because of a better understanding of the structural geology and to better support proposed changes in the mine plan.

Figure 16.26 Original Cave Initiation Point and Direction of Advance. As Mentioned, Because of New Structural Information the Initiation Point can Change to the West



Note: It is recommended that cave be initiated on eastern side of the orebody to prevent formation of large wedges. In south-north direction the initiation point should be selected based on economics.

Cave Advance

The shape of the cavable part of the resource is elongated, and main large-scale structures are believed to be subparallel with the long axis of the cave shape. Two fundamental mining concepts are available: longitudinal and transversal mining advance. Based on the orientation and geometry of large-scale structures, it is recommended that the general mining advance direction be along the long axis of the caving envelope. The characterization drift and geotechnical drilling are intended to provide additional information to better define the large-scale structures and provide information needed to refine the cave plan.

As of November 2010, the OT Study Group has changed the cave advance plan to mine the higher-grade eastern “panel” of the orebody before mining the lower-grade zone against the West Bat Fault on the western side of the deposit. It is also possible that other cave mining arrangements will be evaluated in the next stage of investigation, such as separate panels.

Cave Front Profile

The correct mining concept and strategy are fundamental to successful cave mining. One of the issues is to maintain a correct cave front profile to minimize the amount of time extraction tunnels are in service—wear, consolidation, weathering, etc. Based on known structural and geotechnical model, it is recommended that a cave front profile angle of approximately 45° is appropriate in this stage. This is the approximate cave front profile used in panel caving at Henderson Mine and is also the most “successful” cave front profile based the numerical modelling recently done for a similar orebody.

16.3.6.2 Undercut Strategy

Undercut Type

Three main undercutting techniques are recognized: standard, advanced, and pre-undercut. All three have certain advantages and disadvantages. The advance undercut technique means that the drawpoints and drawbells are developed after the undercut has passed over, so that the abutment stresses are located in the massive rock mass, with only the production drift or the production drift and drawpoint take-offs developed on the production level. Damage to the extraction level is avoided by developing the drawpoints and drawbells in destressed ground. The other advantage of an advance undercut is the good condition of the rock in the brow and the significantly reduced brow wear. In the context of the Hugo North caving envelope, the advanced undercut is recommended. In the absence of detailed numerical modelling it is recommended that a 45° angle be maintained between the undercut front and the drawbell development. It is also recommended that fully supported extraction tunnels with stubs (1 round to be taken) are developed prior to undercutting.

Undercut Profile

Several types of undercut profiles were developed in the past. Principally, two main geometries are used: flat undercut and inclined undercut (crinkle cut). The main reason that the complex shape of an inclined undercut was developed is to shape the major apex over the panel drifts and to prevent formation of “pillars” on top of a flat apex. Factors that must be considered are the frictional properties and the fragmentation of the ore. Low-friction material will move at low angles, whereas high-friction material and coarse material will stack. Several types of inclined undercuts were developed (single- and double-drift types), with and without “anti-socket” drift at the top. Single-drift without “anti-socket” drift means a reduction in undercut development, but does require accurate drilling and blasting or techniques to deal with the pillars that might form at the crest. It is recommended that an inclined undercut profile be used and that one of the “proven” techniques (Freeport, De Beers) be adopted for the pre-feasibility design.

Undercut Height

The height of the undercut could influence the fragmentation in the rock masses that are anisotropic (in terms of rock mass competency). The higher the undercut, the greater the chance that the caved rock mass fragmentation will improve due to the influence of the secondary breakage (cave attrition). The benefit is; however, offset by the slower rates of undercutting and higher cost of increased drilling and blasting. It is recommended that height of undercutting should not be less than 5 m.

Shape of Undercut Front

To increase or reduce the influence of stress on the rock mass and to minimize the influence of large-scale structure undercutting, the front could be developed in the way that maintains concave, convex, or straight shape of the undercutting face. The lead between faces should be kept to a minimum. The long lead results in relaxation and failure in the exposed drift. In the case of Hugo North (no problem with cavability) the main structures are the major consideration. However, in the absence of detailed structural level maps, it is impossible at this stage to design a particular shape for the undercutting front. For prefeasibility purposes a straight undercutting front is recommended.

Direction and Orientation of the Undercut

The main geotechnical factors to consider are the geometry of the caving envelope (long and short axes of the caving envelope), rock mass competency (advancing to weaker or stronger rocks) in various domains, large-scale structures (advancing towards or away from faults), and stress orientation (advancing toward or away from major principal stress). In the context of the Hugo North caving envelope, the oblique orientation of the undercutting front—approximately 45° to the long axis—is recommended.

16.3.6.3 Layout Geometry

The main aspect of layout geometry driven by the geotechnical constraints related to MRMR and fragmentation are:

- Drawpoint layout type.
- Drawpoint spacing and extraction tunnel (panel drift) spacing.
- Undercut level/extraction level vertical distance.
- Infrastructure distance from the cave boundary.

Drawpoint Layout

The most common layouts used in today's caves are "offset herring bone" and "El Teniente" diagonal layouts. From strictly a geotechnical point of view, the El Teniente diagonal layout is recommended because it is simpler and slightly stronger. However, from an operational point of view (diesel or electric equipment and the location of the ore passes), the herring bone offset layout could have critical advantages. In this stage of investigation, both layouts are acceptable for the design.

Drawpoint Spacing

Drawpoint spacing is one of the most important and controversial items in cave mining. Often mine planners will only consider spacing that has been or is in use regardless of the mining environment or the rock mass. There are good reasons to increase drawpoint spacing—to improve the strength of the extraction level, to accommodate LHDs, and to reduce the amount of development. All of this is only permissible if an economically viable quantity of ore is recovered at acceptable dilution content. The major technical consideration is the flow of ore into drawbells, set at the optimum spacing for the interaction of ore of that fragmentation and friction. There are other important practical considerations, namely, the number of drawpoints required for production, the availability of drawpoints, the effect of closing drawpoints when the spacing is wide, changes in fragmentation with draw, and the fragmentation of the waste material.

For the purpose of this study and based on Laubscher's techniques (MRMR, block size, and interactive draw rules), the recommended drawpoint spacing is 14 m to 15 m. This will result in extraction tunnel spacing of 28 m to 30 m, less than the 2005 recommendation because the rock mass appears to be somewhat weaker.

SRK understands that since the 2009 report, OT LLC planners have elected to reduce the drawpoint spacing to 14 m x 14 m. This is based on further evaluation of cave fragmentation, internal comminution, and draw geometry.

Undercut/Extraction Level Distance

This distance is driven by several factors such as stability (greater distance, better stability), fragmentation (high percentage of oversize: shorter distance is better for hang-up blasting), development cost (shorter, cheaper). The recommended distance is 17 m.

Infrastructure Distance from Cave Boundary

This distance differs for tunnels and large-scale excavations. In the absence of numerical stress modelling, judgement and experience are used. It is recommended that perimeter drifts at the extraction level be located at least 30 m from the cave boundary, major excavations at least 50 m from the cave boundary, and any excavation above the extraction level should follow the 50° cave angle rule. Location of major structural features should also be considered, as this can affect the distribution of mining-induced stresses on excavations.

16.3.6.4 Ground Support

The ground support recommendation is based on the empirical Laubscher method and on SRK experience with other caving operations. The ground support is recommended for two basic ground conditions at depths of 80 m to 1,300 m: an MRMR of 40 to 60, which in most of the rock types for most of the caving shapes represents approximately 70% of the excavations and an MRMR of 30 to 40. The stress modelling completed by Itasca also confirmed that the proposed ground support is consistent with the modelling results and industry practice.

Undercut Drifts

The recommended maximum profile is 4.0 m x 4.0 m. Standard support for an estimated 70% of the ground conditions is fibrecrete (50 mm) over the whole profile with 2.4 m fully grouted tendons 1 m from the floor at 1 m grid. Poor ground, an estimated 30%, will have an additional layer of 50 mm fibrecrete and mesh-reinforced fibrecrete on the back. If practical, split-sets instead rebar-type tendons may be considered in the walls.

Production Drifts

The recommended maximum profile is 4.5 m x 4.5 m. Standard support for an estimated 70% of the ground conditions is mesh-reinforced 100 mm shotcrete, or fibrecrete if good penetration results are achieved, with 3 m fully grouted tendons at 1 m spacing. Tendons should cover the whole profile including floor/wall corners. Concrete floor is recommended, possibly with rails in drawpoints because of high draw columns. Intersections should have 6 m cable bolts or couple tendons at 1.5 m spacing. Drawpoint corner "bull noses" should have horizontal straps or cable slings encased in fibrecrete. Poor ground, an estimated 30%, will have additional straps connecting 3 m fully grouted tendons at 1.5 m spacing. On the walls, straps have to be protected by fibrecrete.

Drawpoint/Brow

The recommendation is rigid "Henderson type" flat-back arches and concrete 4 m from the brow anchored with 3.6 m long bolts.

Infrastructure

The recommended maximum profile is 5.5 m x 5.0 m. outside the caving envelope, the support should be designed based on the rock mass rating criteria. Standard support at

800 m to 1,200 m depth for MRMR 40 to 60, for an estimated 80% of the excavations, should have 50 mm fibrecrete with 3 m fully grouted tendons at 1.2 m spacing and mesh on the back. Poor ground, estimated at 20%, will have tendons at 1 m spacing and an additional layer of 50 mm mesh-reinforced shotcrete, or fibrecrete if good penetration results are achieved.

For the profile of 6 m x 7 m, outside of the caving envelope the support should be designed based on the rock mass rating criteria. Standard support at 800 m to 1,300 m depth for MRMR 40 to 60 for an estimated 80% of excavations should have 75 mm fibrecrete with 3 m fully grouted tendons at 1.5 m spacing and mesh on the back. Poor ground, an estimated 20%, will have tendons at 1 m spacing and an additional layer of 50 mm mesh-reinforced shotcrete, or fibrecrete if good penetration results are achieved.

Large-Scale Excavations

Large-scale excavation support will be specific to the size, life expectancy, and location. For the budgetary purposes the support consists of 100 mm mesh-reinforced shotcrete, or fibrecrete if good penetration results are achieved, with an array of 3 m fully grouted tendons at 1 m spacing and 6 m cable bolts at 2 m grid.

16.3.6.5 Draw Height

The draw column height is the economic height of diluted ore that can be drawn from that extraction level under the defined capital cost, operating cost, and profit conditions. The average draw height determines the viability of the operation. The recommended draw heights have several geotechnical constraints, although they could be compromised if the increased cost and increased dilution are accepted. The main concern is the brow wear at the drawpoint due to excess tonnage being pulled through. This, of course, depends on primary rock mass competency, drawpoint reinforcement, and the level of damage from induced stress. It is recommended for budgetary purposes that brow repair be built into the cost profiles for any drawpoint where draw heights exceed 300 m.

16.3.6.6 Cave Dynamics

The main aspects of cave dynamics are rate of caving and rate of draw.

Rate of Caving

All rock masses will cave. The manner of caving and the resultant fragmentation size distribution need to be predicted if cave mining is to be successfully implemented. The rate of caving can be slowed by controlling the draw, as the cave can only propagate if there is space into which the rock can move. The rate of caving can be increased by advancing the undercut more rapidly, but problems can arise if this allows an air gap to form over a large area. In this situation, the intersection of major structures, heavy blasting, and water influx can result in damaging air blast. Rapid, uncontrolled caving can result in an early influx of waste dilution. Good geotechnical information as well as monitoring of the rate of caving and rock mass damage is needed to fine-tune this relationship. The formula $RC > RU > RD$ means that the rate of undercutting (RU) is

slower than the rate of caving (RC) but faster than the rate of damage in the undercut drifts (RD).

Rate of Draw

Draw rates depend on the requirements for safe propagation of the cave, avoidance of seismic events, and variations in fragmentation. Production rates must be tuned to the rate of caving and should not be exceeded if major problems are to be avoided. Laubscher's techniques provide the guide to the rate of caving with respect to the MRMR. Production rates should be based on the maximum rate of draw. Based on the rock mass competency and fragmentation, the recommended rate of draw should not exceed 350 mm/d.

16.3.7 Stress and Ground Support Modelling

Itasca performed a series of FLAC3D analyses to estimate extraction-level performance at the 1,300 m level. A detailed summary of this work is found in Appendix 11A.

- Calibration – Itasca calibrated its model with actual convergence and stress measurements made in the characterization drift. Its calibrated model was used as a base for the additional work.
- Ground Support Modelling – Itasca modelled a variety of development and ventilation drifts. Ground support recommendations were found to be similar for larger headings and not as stringent for smaller headings. The models; however, did not consider mining-induced stresses or multiple heading environments. This work may be of limited usefulness.
- Extraction Level Stress Modelling – Itasca examined the performance of the extraction-level panel drifts and drawpoint bull nose pillars under mining-induced stresses. The models indicated that these excavations are expected to remain serviceable, although some rehabilitation and ground repair will be required.
- Detailed Ground Support Review – Itasca modelled ground support regimes for several different heading configurations. The modelling considered general rock structure as well as 3D wedge analysis. It was concluded that the current ground support plans are suitable for the pre-feasibility level of work.

16.3.8 Instrumentation and Cave Monitoring

SRK stressed the importance of developing a detailed instrumentation and cave monitoring program during the feasibility study and of having the system installed and tested before undercutting commences. Cave instrumentation and monitoring systems typically include a number of parallel systems to provide a comprehensive, 3D, real-time picture of cave propagation. These systems include:

- Open hole monitors (test wells).
- Direct displacement measurements (TDR, extensimeters, pressure transducers).
- Microseismic systems (also useful for predicting seismic events).

16.3.9 Geotechnical Risks

SRK included a brief review of potential geotechnical risks in its 2009 report. It was not a formal Risk Assessment but rather a summary and identification of points that should be considered as the project progresses.

16.3.9.1 Seismicity

Seismicity (rock bursts, strain bursts, sudden catastrophic failures) in block caves can result from high mining-induced stresses in the cave abutment zone and adjacent areas. SRK has recommended the use of advance undercut methods, in part to reduce the possibility of seismic events in the cave abutment zone.

Proper monitoring, planning, and execution of undercut advance and cave management can minimize the likelihood of significant seismic events.

16.3.9.2 Mud Rush

Mud rush, or wet muck, from a drawpoint can be sudden and fatal. Although the Hugo North ground is considered to be dry, and the area has very low rainfall, wet muck is still possible from rainfall or unknown groundwater intercepted by the cave propagation.

The feasibility study should address the potential for wet muck and how to prevent it from occurring.

16.3.9.3 Cave Propagation

There is very little risk that the Hugo North deposit will not cave. The behaviour/performance of the cave is less predictable. The large-scale vertical and subvertical structure can have a significant influence on the cave propagation and dilution entry. SRK recommends that more rigorous modelling be considered to gain a better understanding of the challenges facing the cave management team once Hugo North is in production.

16.3.10 Conclusions and Recommendations

16.3.10.1 Principal Conclusions

The principal conclusions from the geotechnical study are presented below.

- The Hugo North copper porphyry economic envelope of the orebody, or caving shapes as defined in this study, is approximately 1,500 m long x 200 to 500 m wide x 500 to 800 m high. From a geometry point of view, the orebody is ideally suited for mass mining caving method. In the context of current caving operations, it would be among one of the larger orebodies exploited by caving methods, such as the South American copper porphyry mines or Freeport's Deep Ore Zone. This situation offers the possibility of large production tonnages but also creates challenges from a logistical point of view.
- The Hugo North orebody plunges to the north-west and its current base, as defined by the northern-most drilling, is near the -400 m elevation (1,600 m below

surface). Proposed Lift 1 is approximately 1,200 m below surface. From this point of view, the proposed caving operation would fall into the category of deep caving mines—again South American caves could be used as an example. Existing challenges are both logistical (to “get ore from the drawpoint to the mill”) and geological (in situ stress levels approaching or exceeding the rock mass strength). Such conditions require very careful planning and scheduling around the cave strategy. Stresses must be minimized or cut off with well-planned and -executed undercuts to create a favourable mining environment and adequately reinforced ground to create an effective support system.

- Rock mass could be described as an “average caving rock mass.” Using Laubscher’s Rock Mass Classification system (RMR90), the rock mass falls into the “Fair” category, mainly class 3b. From that point of view, the rock mass is ideally suited for caving. However, the rock mass appears to be transected by numerous large-scale structures, mainly faults that pose a challenge for the effective supportability of excavations: an estimated 20% to 30% of the rock mass in the proposed Lift 1 extraction level falls into the category of “Poor” to “Fair,” highly jointed rock mass (RMR < 30). Large-scale faults also offer challenges in managing cave propagation and dilution ingress. Only main large-scale structures are explicitly modelled with some degree of certainty. The core indicates that there are more structures that cannot be modelled given the current drillhole spacing. The proposed cave initiation and advance strategy caters to major currently defined structures and structural trends but is under serious review based on recently acquired information for the characterization drift and diamond drilling.
- In situ stress knowledge is good and will be further improved by the ongoing instrumentation program. Although the currently estimated principal in situ stress is approaching (or in some cases exceeding) rock mass strength, the highly jointed and faulted nature of the orebody should have a positive impact on the stress field, i.e., stress dissipation rather than stress accumulation. Also, it should be noted that cave mining is to a large extent a stress management method, as it operates in altered stress field conditions. For example, correctly executed undercuts significantly reduce in-situ stress and enable production development in the “stress shadow.”
- An empirical and simple numerical method was used to determine the subsidence cone for Lift 1 and Lift 2. The results indicate that the expected cave angle is approximately 65° to 73°. Although the detailed geology in the potential subsidence is poorly understood, the subsidence angle is not expected to be significantly shallower based on the case studies from other caving operations (El Teniente, San Manuel, Henderson). The presence of the massive granite body to the north was not taken into consideration and could result in steeper angles. Because of the greater caving depth and resulting “bulking” of the caved and disturbed material, the influence of the major faults, mainly steeply dipping, do not appear to be of great significance to the subsidence angles.
- Based on evidence from the current excavations, groundwater should not be a problem. The issue of near-surface clays in combination with surface and underground water should; however, be investigated from a dilution and mud rush point of view.

16.3.10.2 Recommendations for the Feasibility Study

- Continue to map and sample underground exposures through the orebody to minimize uncertainty about:
 - Rock mass characterization – mainly joint continuity and geometry.
 - Structural characterization – nature, orientation, and continuity of faults.
 - Stress characterization – magnitude and orientation of the in situ stress and behaviour of excavation in the stress field.
- Geotechnical domain modelling using Leapfrog should be investigated to better subdivide the orebody rock mass.
- Expand on existing numerical modelling to further investigate:
 - caving strategy – undercutting advances versus stress field.
 - layout details – different orientation to the stress.
 - subsidence angles – with more comprehensive geology.
 - support effectiveness for various excavation scenarios.
- Conduct mudflow risk assessment.

In summary, the geotechnical pre-feasibility did not reveal any issues that could not be solved technically based on current knowledge or technology. Cavability and fragmentation do not appear to be a problem, even with uncertainty in some of the input parameters. However, at more than 1,000 m depth, the currently estimated high horizontal stress will demand comprehensive undercutting, a caving and development strategy, and sufficient ground support.

16.4 Underground Mining

IDOP confirmed block caving as the underground mining method chosen by OT LLC for the Hugo Dummett Mineral Resource. The IDOP study is based on the same mine design prepared for Hugo North Lift 1 as was presented in IDP10. The Mineral Reserve was therefore the same in IDOP as it was for IDP10. IDOP includes a revised development and production schedules and costs.

The Hugo Dummett underground deposit will be mined by block caving, a safe, highly productive, cost-effective method. The deposit is comparable in dimension and tonnage to other deposits currently operating by block cave mining elsewhere in the world.

The Hugo North Lift 1 Mineral Resource plunges to the north-east. Its current base is near the minus 100 m elevation, 1,300 m below the surface. The caving shape foreseen is nearly 2 km long x 150 to 300 m wide x 100 to 500 m high. The resource dimensions are considered suitable for block caving and place the mine into the category of “deep mining.” A critical consideration for the development of the deposit is assurance that the resource will cave and work to date by OT LLC has confirmed this expectation.

16.4.1.1 Geomechanical Factors

A preliminary hydrogeological assessment was completed by Aquaterra Consultants predicted dewatering rates rise as more workings are installed and are significantly greater for simulations of major subsidence, with maximum inflow rates approaching or exceeding 100 L/s for all scenarios. This indicates a very dry mine. Aquaterra also completed several reports for Shaft Nos. 1, 2, and 3 and conclusions support the “dry mine concept.” Although the region is arid high rainfall events after the cave has propagated to surface may introduce significant water into the mine in a relatively short period of time. OT LLC noted the issue in IDOP and plan to address it as part of the further study work and ensure sufficient pumping and/or storage capacity is designed into the system to allow for this eventuality.

SRK assigned the bulk of the rock mass for Lift 1 to two ground support categories. Good ground is defined as having an MRMR of 40 to 60. Poor ground is defined as having an MRMR of 30 to 40. SRK specified ground support requirements for excavation in each category. Mining Method

16.4.1.2 Selection Criteria

Hugo North extends from +500 m masl (metres above mean sea level) to depths well below -400 m. The deposits are extensive laterally, and the ground conditions are predominantly poorer than what is required for a non-caving mining method to be employed. The size, shape, grades, and geotechnical characteristics of the deposit necessitate the use of a bulk mining method. Caving was selected based on these factors as well as historical studies, input from third-party consultants, and economic considerations.

The advantages of using the block caving method for Hugo North include the following:

- High productivity.
- Low unit cost.
- High production rate.
- Inherent safety (no large openings standing).

The disadvantages of using the block caving method for Hugo North include the following:

- High up-front capital requirements.
- Long lead time to develop, construct, and commission the mine.
- Intermittently high secondary breaking requirements, which may result in:
 - Increased personnel exposure to open drawpoints.
 - Increased number of production interruptions.
 - Increased repairs due to the blast damage.
 - Negative impact on draw control.
- Possible ore loss and dilution if overburden fragmentation is finer than expected.
- Impact on surface facilities due to subsidence.
- Cave management and control issues related to the numerous high-angle structures that transect the deposit.
- Possible loss of developed production areas due to cave management and geologic structure problems.

16.4.1.3 Caving Parameters and Geotechnical Considerations

Information on rock mechanics and specifications for ground support and excavation orientations is provided by SRK. In February 2009, SRK provided McIntosh with preliminary caving parameters used in preparation of the production scheduling and mine designs. Listed below are the key SRK caving parameters McIntosh used in preparing the mine design and production schedules.

- The standard horizontal LHD panel caving layout is recommended for the Hugo North caving envelope, with initiation point at the eastern boundary and advance oblique to the southeast and northwest, at approximately 45° to the long axis.
- As of the November 2010 update, OT LLC is considering alternate cave initiation strategies to take into account the better understanding of the structural geology, mine grade distribution, and other factors.
- Based on the current understanding, a cave front profile angle of approximately 45° is considered appropriate. The 45° angle is maintained between the undercut front and the drawbell development.

- As of the November 2010 update, the OT LLC Study Team is actively considering reorientation of panel geometry to better suit production and ventilation requirements, as well as changes in cave initiation strategy.
- Fully supported extraction drift with stubs (one round to be taken) are developed prior to undercutting.
- Inclined undercut profile is used and, based on current design, it is recommended that one of the “proven” techniques be adopted. A straight undercutting front should be assumed.
- Either Offset Herringbone or El Teniente drawpoint layouts are acceptable.
- The recommended drawpoint spacing is 14 m to 15 m. OT LLC has recently changed drawpoint spacing from IDOP’s 15 m x 15 m to 14 m x 14 m.
- The recommended distance between extraction and undercut level is 17 m.
- Perimeter drifts at the extraction level should be located at least 30 m from the cave boundary. Major excavations should be located at least 50 m from the cave boundary. Any excavation above the extraction level should follow the 50° cave angle rule.
- Rate of draw during the cave propagation should not exceed 350 mm/d.
- Based on current data, the hydraulic radius to sustain caving of the rock mass is approximately 20 m to 22 m.
- The current understanding of principal stress is as follows:
 - Major principal stress σ_1 has a bearing of 050°, in approximately the north-easterly direction.
 - Intermediate principal stress σ_2 has a bearing of 150°, approximately south-east.
 - Minor principal stress σ_3 has a bearing of 050°.
- SRK noted that the major principal stress at the proposed production horizons is approximately twice the vertical stress.

Several subsidence predictions for Hugo North have been completed in the past. Initial subsidence estimates used Laubscher’s empirical approach. In 2005, SRK developed 2D FLAC analyses on selected sections and in 2008 Itasca undertook 3DEC modelling. SRK preliminary guidelines based on empirical design and simplified FLAC 2D sections recommended situating important structures no less than 60° from the footprint; the results of subsidence modelling suggested a range of 63° to 75°.

The cave initiation area for the footprint was selected through a coordinated analysis by OT LLC, McIntosh, and SRK. Geotechnical criteria, ore grade, and the current understanding of the geology in the area of the footprint at the chosen elevation are the driving factors in the determination to initiate the cave on the east side of the footprint. The West Bat Fault is nearly vertical and is a controlling structure of the mineralization along the western edge of the mining area. A granodiorite dyke, which is almost barren, is located in the southern area. The Contact Fault along the east side of the footprint dips to the east at approximately 60°. The recommendation from SRK is to initiate the

cave on the east boundary and to move toward the west, away from the Contact Fault. The cave initiation area is discussed further in Section 16.3.6.1. OT LLC is actively considering relocating the cave initiation area to the west side of the deposit to allow for the numerous high-angle structures in the area.

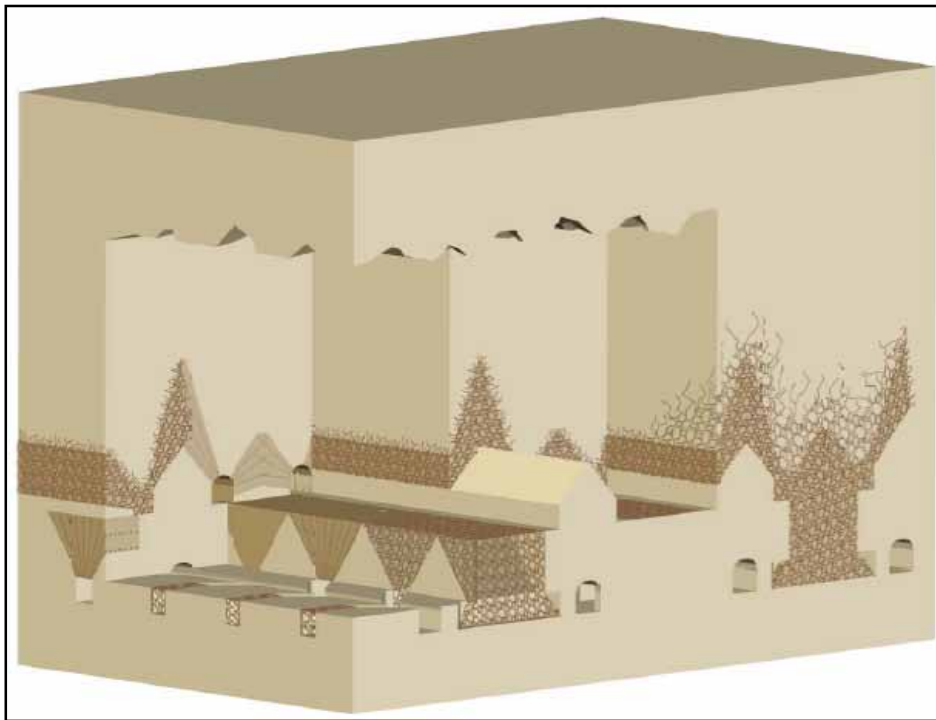
SRK recommends utilizing advanced undercut for Hugo North. The advanced undercut technique ensures that the drawpoints and drawbells are developed after the undercut has passed over, so that abutment stresses are located in massive rock pillars with only the production drift and drawpoint take-offs developed on the extraction level. Damage to the extraction level is avoided by developing the drawpoints and drawbells in de-stressed ground after the abutment stresses have passed. The other advantage of an advanced undercut is the good condition of the rock in the brow and significantly reduced brow wear.

16.4.1.4 Mining Method Description

Block caving is a high-tonnage underground bulk mining method generally applied to large homogeneous ore deposits. Ideally, the ore to be caved should be structurally weak, and the waste overburden should be weak enough to collapse over the ore without inducement as the ore is extracted.

Block caving involves excavation of natural support from beneath the ore, causing the ore to fail and collapse into the excavated void under the force of gravity and local geomechanical stresses. The broken ore is then pulled out from under the caved section through a drawpoint arrangement, subsequently removing support from ore and overburden at increasing height above the initial excavation, and eventually extending the cave upward to the surface (Figure 16.27).

Figure 16.27 Block Cave Isometric



The attractive aspect of block caving is that only a relatively small portion of the ore must be drilled and blasted prior to extraction. Once the cave initiates, production continues without further primary drilling and blasting until the ore column above is exhausted.

The block cave mining sequence begins with access and infrastructure development, followed by excavation of the extraction level and undercutting the ore. The sequence culminates in steady state production from individual drawpoints.

Ore in the column is diluted by material in adjacent columns and ultimately by overburden and lateral waste migration. When the column drawdown is complete and drawpoint grade drops below a minimum value (shut off value), the drawpoint is closed. Great care is taken in establishing and maintaining uniform draw practices throughout the mine to maximize drawpoint life and minimize dilution and stress loading from underground workings.

Block caving is a capital-intensive mining method, requiring significant investment early in the mine life for infrastructure and primary development. Once in place, the method's high up-front costs are offset by high production rates and low operating costs (relative to other underground methods) over a considerable length of time, resulting in a low overall cost per tonne. Block cave mining is among the least costly of all underground mining methods.

Initiation of the cave is dependent upon rock mass strength and local stress field orientation and magnitude. Once the cave is initiated, propagation of the cave over the adjacent panels should occur immediately.

It is not known precisely at what angle the cave at Oyu Tolgoi will propagate. A 60° subsidence design line projected from the undercut level of all mining areas is used to locate surface infrastructure. To reduce the risk of affecting Shaft No. 1, a 65° cone from the collar is used to limit the footprint boundaries. Further study is required to more accurately predict actual caving propagation.

Caving with a flat footprint is a proven, safe, and low-cost mining method currently used at several operations around the world. Examples of other companies using this method under similar conditions include Deep Ore Zone Mine in Indonesia, Henderson Mine in Colorado, Northparkes Mine in Australia, and Palabora Mine in South Africa.

Drawpoint construction and cave management are key drivers of the life-of-mine schedule and the success of caving operations; they are critical to meet the production schedule.

Draw control is the management of the cave draw down. Once undercutting has commenced—and management and coordination between development, construction, caving, and production have taken place—draw control becomes the most critical factor for safe operation of the cave and grade control. Failure to manage these activities can be catastrophic to the operation. Once undercutting commences, the cave must be kept moving to prevent compaction of the cave muck and stress loading of surrounding development and infrastructure.

16.4.1.5 Mine Layout

Production schedules were prepared using Gemcom Software International's PC-BC utility, a production scheduling program specifically designed for block caving operations. The program estimates the grade of material produced from a drawpoint as the rock is drawn. Completing the PC-BC modelling process generates a production schedule suitable for mine design and economic evaluation.

PC-BC works with an existing block model that is imported into the GEMS format. The block models provided by OT LLC contain attributes for Cu, Au, Ag, As, Tput, F, S, SPI, MB, CI P80, Fe, Mo, classification, density, and lithology. NSR, Cu Recovery, and Au Recovery are calculated from Base Data Template 21 (BDT21). BDT21 uses base metal prices of \$816/oz Au, \$2.10/lb Cu, and \$13.00/oz Ag.

The following steps were followed to create a mine layout in PC-BC. These steps are further described in the subsequent subsections.

- Determine the panel orientation
- Create a drawpoint layout
- Establish a cut-off estimate (footprint evaluation)
- Apply the production shut-off estimate (column heights)
- Run the production scheduler

A major advantage of using PC-BC for production planning is that multiple scenarios can be quickly and easily evaluated, allowing the planning team to optimize mine plans and production schedules.

Determine the Panel Orientation

The orientation of the extraction level panel drifts is determined by geotechnical factors. SRK and OT LLC have recommended an orientation of 55°, which is nearly parallel to the principal stress as determined by horizontal stress measurements taken in the characterization drifts and in Shaft No. 1 during sinking. This orientation is also nearly perpendicular to the long axis of the ore zone, thereby reducing the average LHD haulage distances. OT LLC is considering alternate orientation of the extraction level panel drifts.

Create a Drawpoint Layout

The drawpoint pattern can consist of either El Teniente or Offset Herringbone. Both patterns are deemed reasonable for use at Oyu Tolgoi.

Drawpoint access drifts in the offset herringbone layout are driven in chevron pattern between panel drifts. Each drawpoint access is offset on either side of the panel drift. The accesses are oriented in the same direction to enable extraction of ore from drawpoints on both sides of a panel drift without turning LHDs around. This layout was chosen for Lift 1 due to high efficiency with a long, narrow footprint.

The El Teniente layout is characterized by using drawpoint drifts that are at an angle from the panel drift but are straight between each panel drift. The drawpoint on one side of the panel drift faces the opposite direction of the drawpoint on the opposite side of the panel drift. The advantage of the El Teniente layout is that the drawpoints are easier to excavate and construct, the pillars are more stable, and there is better overlap of ellipsoids of draw between each drawpoint. The major disadvantage of the El Teniente layout is that the production muckers are required to operate with loaded buckets in front for 50% of their service; this makes the application of electric muckers much more difficult.

Establish a Cut-off Estimate (Footprint Evaluation)

A cut-off estimate is established to include the cost of development in deciding whether to include a drawpoint in the design. The cut-off is an NSR value used to determine the total dollar value of each drawpoint. The cut-off for Lift 1 has been evaluated multiple times since 2004. The result of these evaluations shows the best NPV at approximately -150 masl. This is a change from the previously reported production horizon of -115 masl.

In addition to the cut-off value, other criteria are considered in creating the footprint shape. The parameters used in defining mineable drawpoints are as follows:

- NSR Shut-off declining \$20.00 to \$12.54
- Minimum Height of Draw (HOD) 100 m

- Maximum HOD 500 m
- Minimum Dollar Value..... \$300,000
- Minimum Mining Width 145 m (10 drawpoints)

The minimum mining width of 10 drawpoints is applied to the perimeter of the footprint where the mineralization begins to narrow significantly. This minimum width ensures that there is sufficient area to continue caving through the section.

Apply the Production Shutoff Estimate (Column Height)

Once the block cave footprint has been digitized into PC-BC, another PC-BC best height of draw (HOD) calculation is performed, this time applying the shut-off used for production. The shut-off is the point at which production from the drawpoint is no longer economic. The shut-off is determined by estimating the marginal cost of mining (\$4.00/t), milling (\$2.85/t), and general and administrative expenses (\$0.91/t). If the NSR/t is less than or equal to the marginal cost of production (\$8.97/t) for a block in the slice file, the net revenue per tonne of that block is negative or zero. Producing tonnes that contain less than \$8.97 NSR value is uneconomic. The best HOD calculation produces grades, tonnes, and total dollar value for each drawpoint inside the mining footprint.

Run Production Scheduler

The last step is to run the PC-BC production scheduler using drawpoint tonnes, production rate, and the drawpoint construction schedule as the main scheduling parameters.

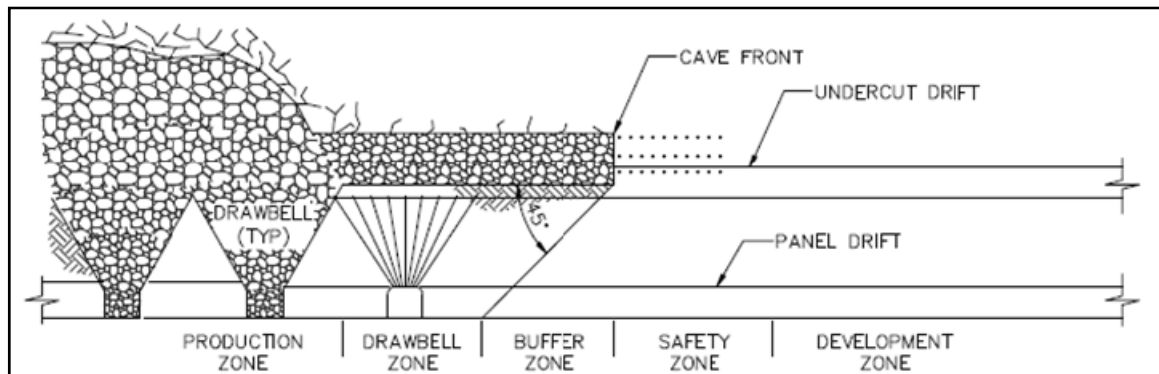
The drawpoint tonnes are calculated using the drawpoint spacing, drawpoint layout, and resulting cone geometry. The draw cone in PC-BC is defined as the material above a drawpoint. The draw cone used extends from the back of the extraction level drawpoint drift to a maximum column height of 500 m. The portion of the cone between the extraction and undercut levels simulates the volume of the drawbells. Above the undercut level, the draw cones overlap to simulate horizontal movement of rock between cones.

The production rate is based on industry experience with cave initiation and builds up to the maximum rate sustainable given the Lift 1 footprint geometry and column heights.

The drawpoint construction schedule is a result of the ability to maintain the following relationships (Figure 16.28):

- Maintaining a buffer zone (45°) between the advancing undercut above and the drawpoint construction area (no development work in this zone).
- Ensuring that enough drawpoints are constructed for the advancement of the drawbell zone.
- Advancing the production zone uniformly to preserve the distance with the advancing undercut.
- Conducting no development work in the safety zone due to the presence of abutment stress preceding the cave front.

Figure 16.28 Drawpoint Development



It is estimated that the cave will advance vertically at a maximum rate of 350 mm/d. Exceeding that rate of draw could result in the creation of a void between the uncaved rock and the drawpoints. This limit also controls the maximum rate at which the cave front can advance horizontally. The rate of advance is also limited by constructability. Industry experience indicates that the maximum rate at which drawpoints can be developed in a panel drift is ten per year. A third limitation is imposed by the return of stress after the de-stressed zone (the stress shadow) has advanced following the cave front advance. The completion of the extraction level development must be accomplished in the moving window between the 45° line following the cave front and the advance of the returning stress.

The optimal location for cave initiation is on the east side of the footprint, caving away from the contact fault, is depicted in Figure 16.29. OT LLC is actively considering changing the cave initiation point to the west side of the orebody to make allowances for the numerous high-angle structures in the “hinge” area of the deposit.

The West Bat Fault is nearly vertical and is a controlling structure of the mineralization along the western edge of the mining area. An almost barren granodiorite dyke is present in the southern area. The Contact Fault along the east side of the footprint dips to the east at approximately 65°. This relationship and caving direction is shown in section view in Figure 16.30.

Once all the data have been assembled the production scheduler is run which simulates pulling ore from each drawpoint according to the new drawpoint schedule and depletes the slice file, mixing the grade elements as the slice file is depleted. The initial production schedule produces a target production rate based on all of the input parameters. The resulting production rate allows a rock handling system concept to be designed that supports the mining activities.

The rock handling system is planned to meet the optimum production rate as well as reduce congestion between production and ongoing development and construction activities.

Figure 16.29 Cave Area – Plan View

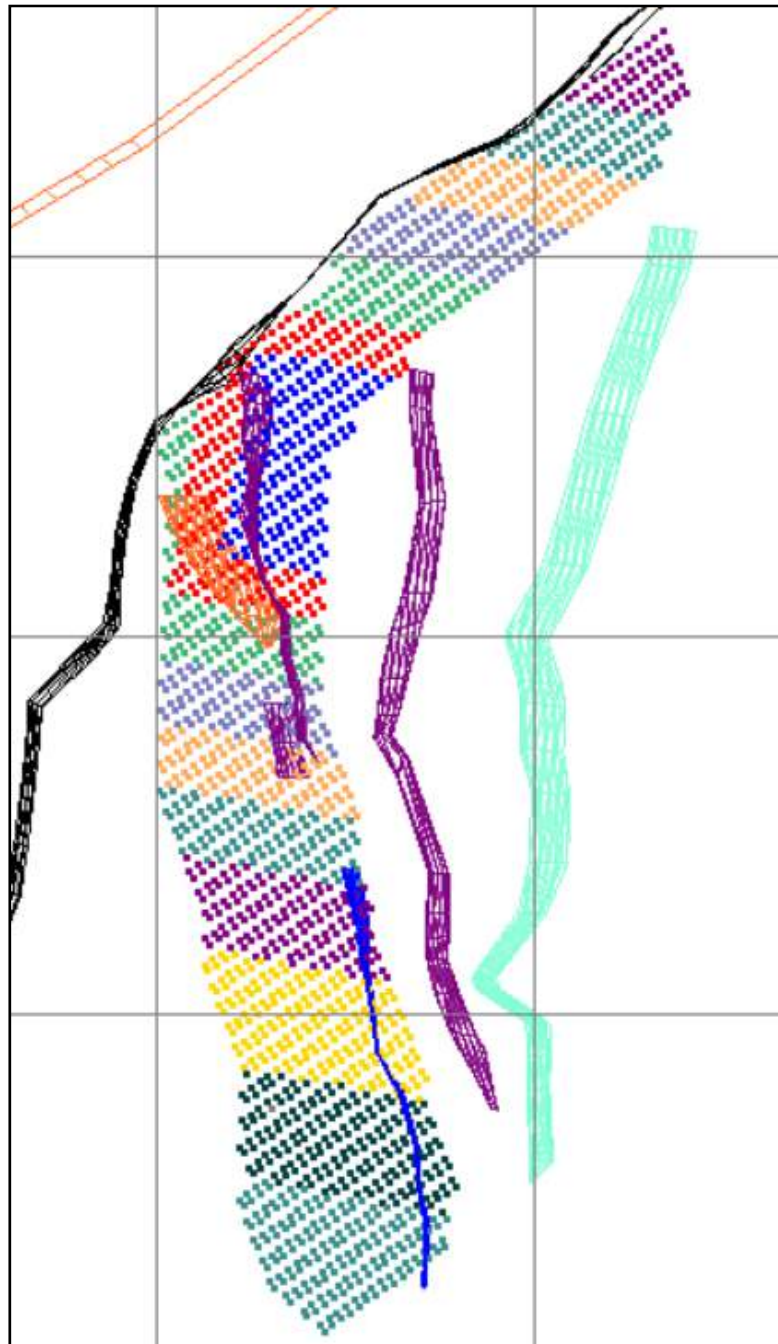
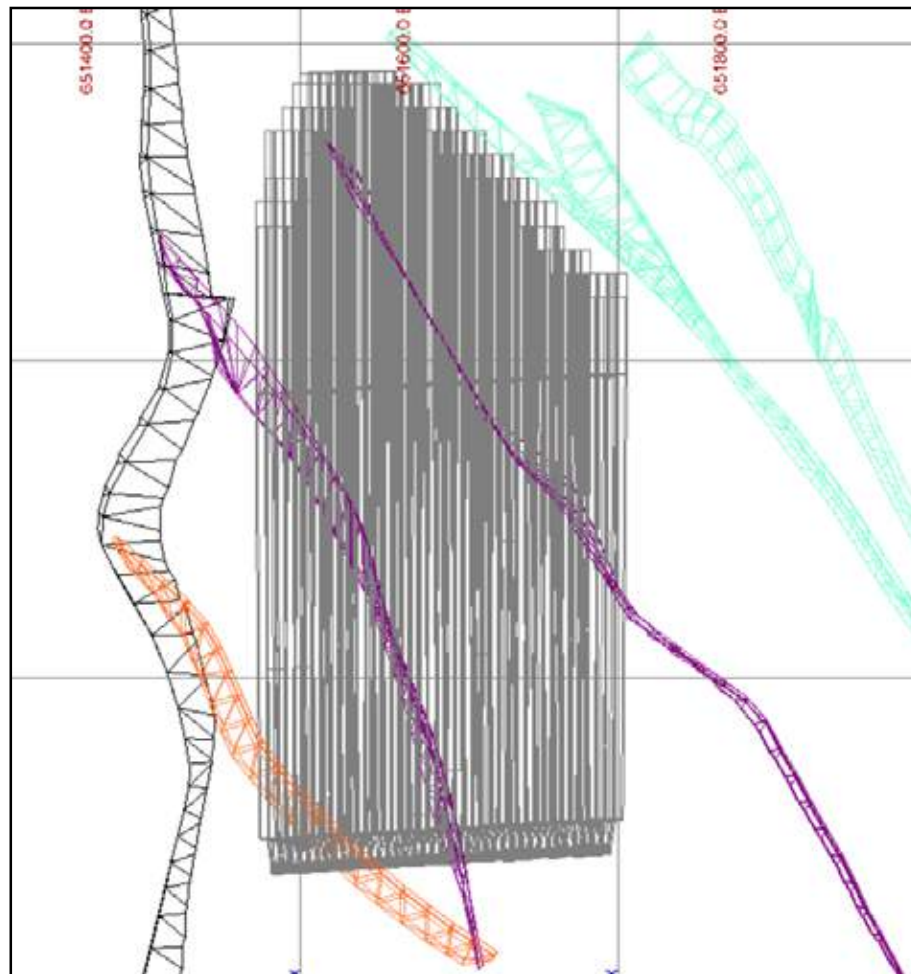


Figure 16.30 Cave Area – Cross-Section (Looking North)

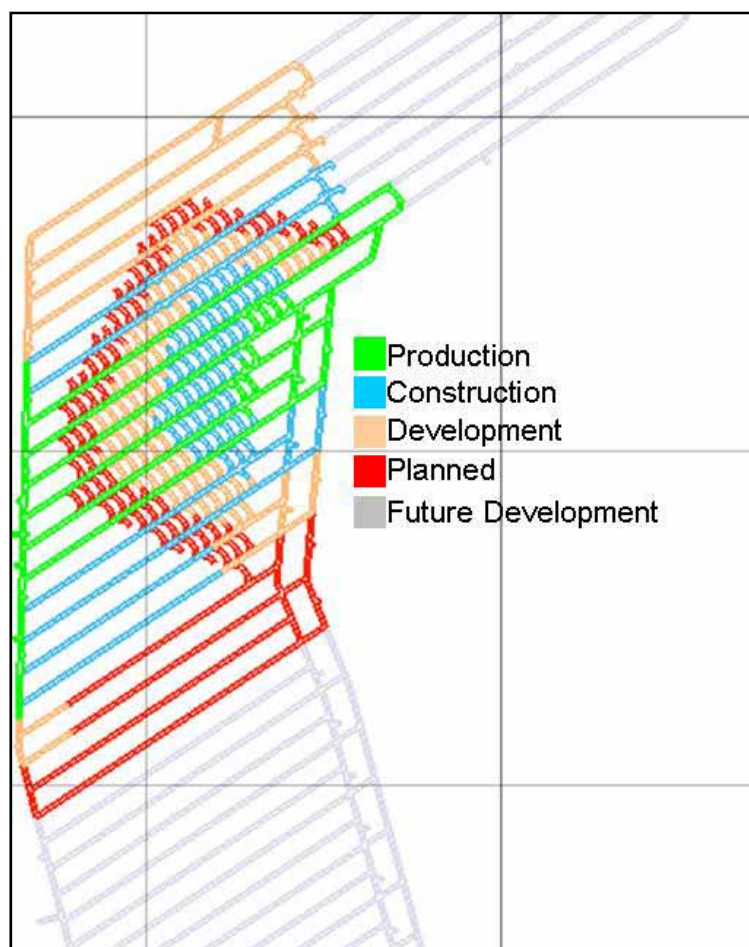


The ideal location for the LHD dump point is on the east side of the footprint, along with the cave initiation area. The production crews will be working primarily in the area between the active drawpoints and the ore passes on the east side of the footprint. The construction crews work between the drawpoints in the construction zone and the west perimeter drift. A development crew excavates drawpoints ahead of the construction crew, but the interference should be kept to a minimum with careful planning of activities. Figure 16.31 is a schematic of the activity areas on the extraction level.

Along with a change in the cave initiation point, OT LLC is actively considering relocating the production grizzlies to the centre of the production panels to reduce the average tramming distance of the production mucking units and the haulage level footprint. If this change is made, the haulage level will need to be redesigned.

The ore from an active drawpoint is hauled by an LHD to a grizzly at an ore pass near the east perimeter drift. The bottom of the ore pass is equipped with a truck chute at the haulage level. Trucks transport the ore to crushers along the west side of the footprint. The ore is then transported to the shaft bins via inclined conveyors.

Figure 16.31 Extraction Level Activities



16.4.1.6 Production Plan

Production planning is a balance between the rate at which the defined orebody will cave and the rate at which development and construction can be achieved to sustain caving. The production rate is a result of the elevation and shutoff criteria selected. The resulting production profile is reviewed and a rock handling system is designed that will support the best production rate to sustain caving.

Material above the ore is generally stronger than the ore itself and is expected to undergo coarser fragmentation than the ore while in the draw column. In theory, this phenomenon decelerates dilution of the ore, as the coarser waste fragments migrate more slowly through the draw column than the finer ore material.

The production schedule is broken into three phases: preproduction, production build-up, and sustaining production.

- Pre-production is defined as the development required prior to blasting the first drawbell. This time period begins in Year -5 and continues through Year 1. Undercutting commences before the first drawbell is blasted.
- Production build-up commences once the first drawbell is blasted. The undercut commences at the beginning of Year 1 and the first drawbell is blasted during the second half of Year 1; however, because the PC-BC draw cone geometry includes the drawbell and undercut tonnes, the undercut swell mined in advance of the caving is not calculated separately. Therefore, during Year 1, swell from the undercut and drawbell excavations is mined at an average of 2,500 t/d in the production schedule. Caving commences during Year 1 and production increases steadily to 84,200 t/d at the end of Year 8.
- Sustaining production is the period from the first year of full production through to the end of the mine life. Year 8 is the first year of full production. The last year of full production is Year 16, with output tapering off to 16,000 t/d at the end of the mine life in Year 21.

The design of the underground incorporates 4 shafts and 5 raises to ventilate the mine. The intended method of construction for the 5 ventilation raises is by raise boring. This has proven difficult and has suffered delays, the primary reason for schedule slippage has been due to poor ground conditions. Completion of the ventilation raises is on the critical path to production from the underground mine as the raises are required to meet the ventilation requirements for development and provide alternative means of egress should Shaft 1 be unavailable. In order to mitigate and minimise risk to the project production, OT LLC is investigating alternative means of establishing the vertical development required to complete the ventilation infrastructure.

Work to determine the impact of this issue is being carried out. Initial analysis by OT LLC suggests that the underground production could be delayed by around twelve months and that each month's delay could have an NPV value of \$50 M. These investigations are planned to be incorporated into DIDOP and will allow OT LLC to quantify potential costs and changes required to meet the Project schedule.

In addition possible scope changes have been indicated including:

- Additional development and excavation expenditures
- The addition of a shaft No 5
- An additional crew underground
- Increased operating cost due to design changes

Changes being implemented by OT LLC is expected to reduce the drawpoint spacing to 14 m x 14 m, increasing the number of drawpoints required from 1,690 to 1,910 and the number of panel drifts from 46 to 65. Undercut and drawbell schedules will increase to 150 drawpoints, or 75 drawbells, per year.

The drawpoint closure schedule is output from PC-BC. The drawpoint opening and closure schedules are shown in Figure 16.32. These figures have not been updated to reflect the proposed new undercut schedule and sequence, the new work will be incorporated into DIDOP.

16.4.1.7 Subsidence Evaluation

The subsidence zone from the extraction level to surface was originally projected at 60°, per SRK's recommendation (Figure 16.33). All planned infrastructure is outside this zone except Shaft No. 1, which is located inside the 60° subsidence area at the edge of the 65° subsidence line. It is assumed that Shaft No. 1 will be stripped of all conveyances once full production is achieved and will be used for ventilation only. Subsequent work by OT LLC and Rio Tinto have indicated that 50° is a more appropriate subsidence angle.

Figure 16.32 Annual Drawpoint Opening and Closure Schedules

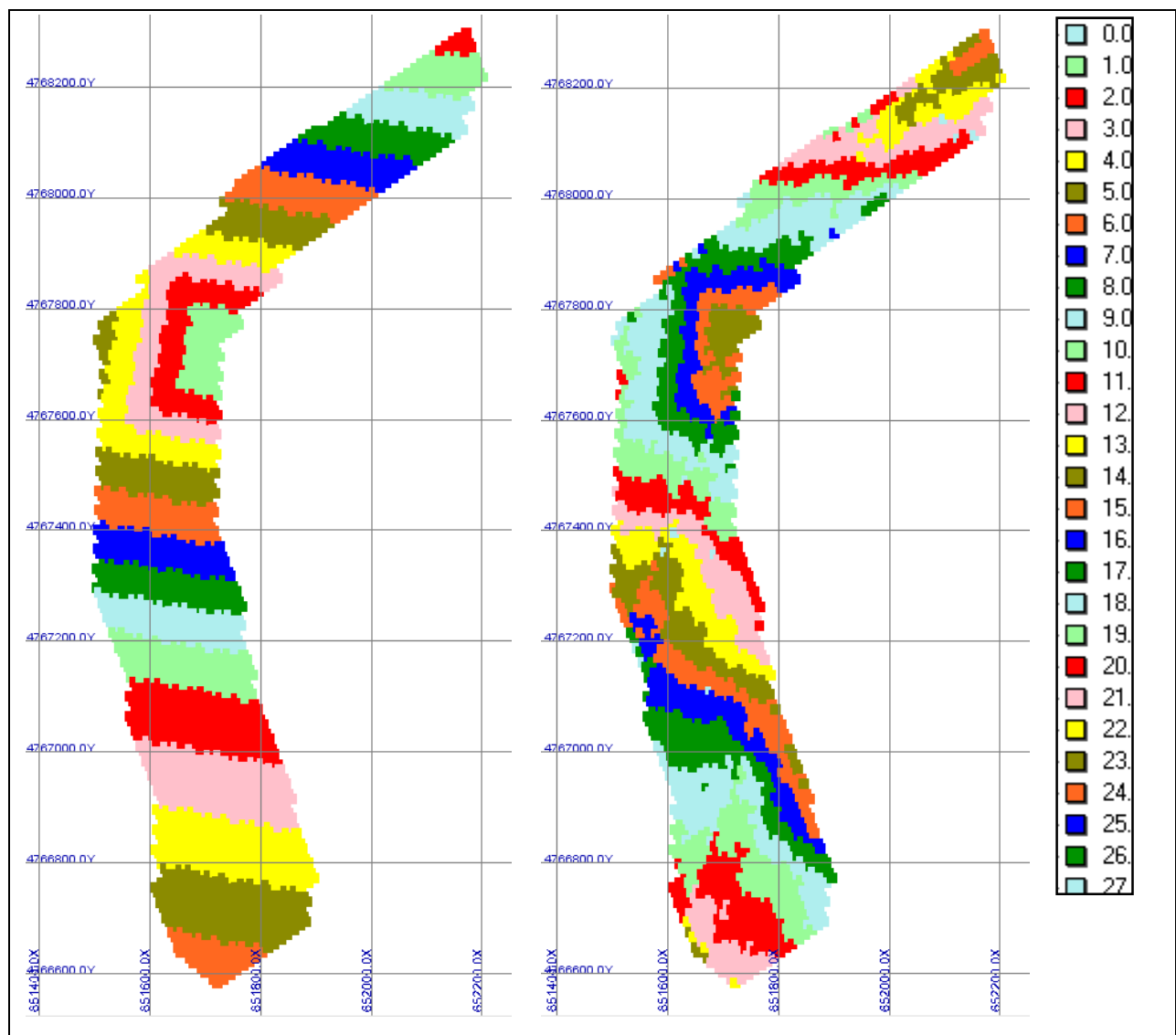
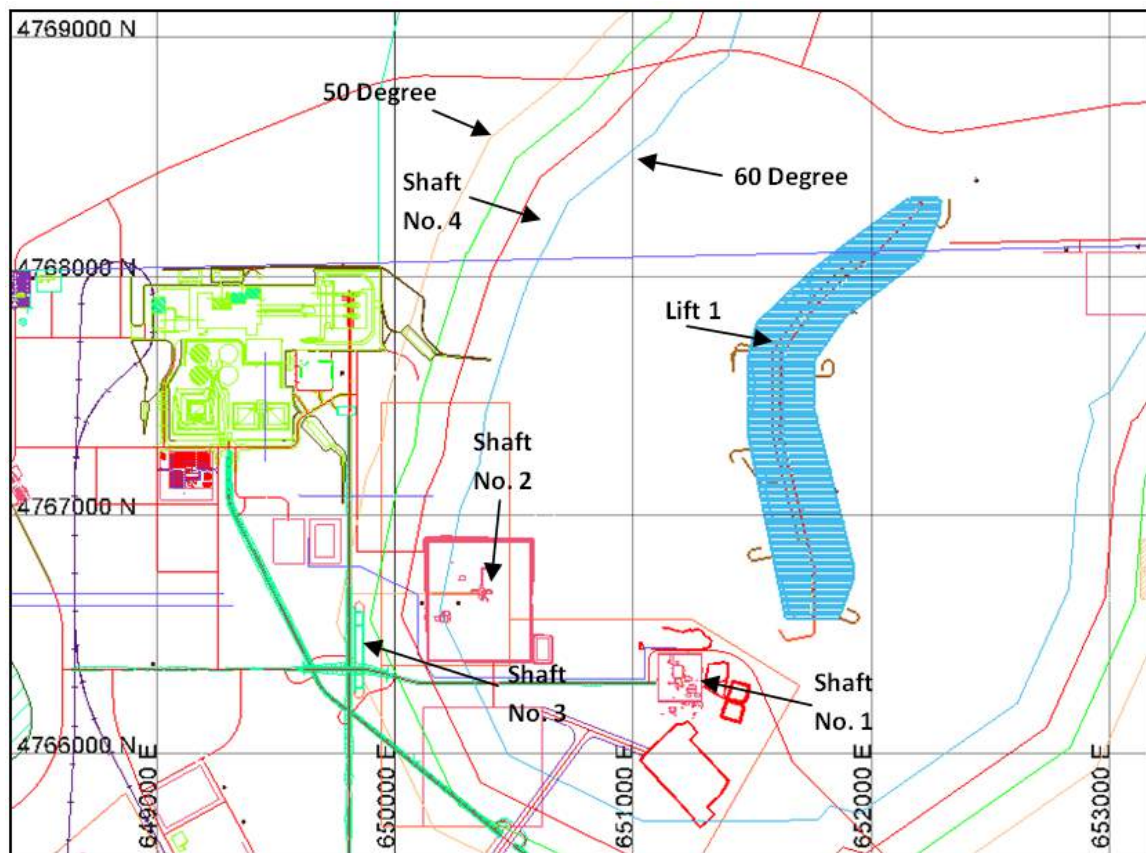


Figure 16.33 Projected Subsidence Area



It is not known precisely how the cave will propagate. The projection shown in Figure 16.33 merely illustrates the extent of surface area that the generalized subsidence projections encompass. Further study is required to more accurately predict actual cave propagation.

16.4.2 Rock Handling

The ore handling system transports cave muck from the drawpoint on the extraction level to the mill on surface. The system has been designed to meet the maximum caving rate. Panel drifts provide access to all drawbells. Each pair of panel drifts shares an ore pass; therefore, the spacing between ore passes is 56 m. LHDs transport ore from the drawbells to an ore pass located along the eastern perimeter drift to provide the LHDs with a maximum one-way haul distance of approximately 300 m. The OT LLC study team is actively considering relocating the ore passes to the centre of the panel drift. This will result in average tramming distances being reduced to about 150 m, thus increasing loader productivity.

Ore passes are raisebored to 3 m diameter. Ore passes are vertical and average 35 m high with a maximum capacity of 425 tonnes. Grizzlies will be fitted over all ore passes. Oversize rock will be broken with fixed rock breakers.

Ore passes discharge through chutes to 50 tonne trucks on the haulage level. OT LLC is actively evaluating the use of 80 tonne side-dump trucks for the haulage level. The trucks will transport ore to the primary gyratory crushers through one-way haulage loops. The haulage level is all one-way travel to reduce congestion. Haulage drifts are 40 m below the extraction level.

IDP10 assumed that trucks delivered ore to three 54 x 77 gyratory crushers. OT LLC undertook a trade off study to optimize the crusher and conveyor system design. The results of this study indicated that crushing/conveying duties could be handled by two each 60 x 89 gyratory crushers and a single inclined conveyor. This system is now being used as the foundation for mine material handling design.

The inclined conveyors transfer ore to the large bins near the shafts, and each bin feeds a set of high-speed measuring conveyors that directly load the skips. Skips are hoisted to surface to discharge into bins from which ore is transferred to the main overland conveyor and on to the mill facilities.

As part of DIDOP, the number and capacity of ore bins at the skip loaders are being reviewed to optimize flexibility and reliability and to reduce hoisting depth.

Waste and low-grade mineralization will be produced primarily from pre-production development. Prior to commissioning Shaft No. 2, all development muck will be hauled to Shaft No. 1 for hoisting to surface. At the surface, the waste is dumped on a dedicated stockpile near Shaft No. 1.

After Shaft No. 2 is commissioned, Shaft No. 1 will continue to be used for waste hoisting until the ventilation requirement increases during production build-up. Then Shaft No. 1 will be stripped and used for exhaust ventilation. After Shaft No. 1 muck hoists are decommissioned all development muck will be hauled to the production crushers for crushing and hoisting to the surface through Shaft No. 2.

16.4.3 Mine Services and Support Infrastructure

16.4.3.1 Ventilation

The main function of the mine ventilation system is to dilute and remove airborne dust, diesel emissions, and blasting gases, and to maintain temperatures at acceptable levels in the working environment.

Total mine ventilation requirements are determined from airflows specified for the production and development areas, transportation and haulage levels, and fixed facilities. Ventilation in each area must be sufficient to dilute air contaminants to acceptable limits.

Mine Ventilation Services (MVS) completed a third-party review of the ventilation design. MVS' recommendations are incorporated into the final design.

VnetPC software is used to develop models at representative stages of mine design. Several milestone points during preproduction and production build-up have been

identified; phased ventilation requirements have been developed for the development status at these milestone points.

Airflow Requirements, Criteria, and Standards

The main ventilation circuit sweeps fresh air across the mine from the intake shafts to the west of the mining areas through the mine workings. The fresh air is directed from the intake manifold drifts up to the extraction level and other working areas. The exhaust air is then collected in the exhaust manifold drifts along the east side of the footprint and is split to direct contaminated air out to the exhaust shafts.

Air quantities are based on diesel emission dilution of $0.06 \text{ m}^3/\text{s}/\text{kW}$ (Ontario Regulations R.R.O. 1990, Regulation 854 Mines And Mining Plants) of diesel engine rating based on technical specifications for major equipment. The ventilation criteria, discussed below, determine the total ventilation requirement for the mine. A total airflow volume of $1,850 \text{ m}^3/\text{s}$ ($0.022 \text{ m}^3/\text{s}$ per t/d) is required at full production to provide adequate ventilation for mobile equipment, fixed facilities, and personnel in the mine. Note that Mongolian regulations require significantly more airflow than do the Ontario Regulations. The current mine ventilation system, as designed, will not meet Mongolian regulations unless OT is able to obtain a variance.

The proposed final ventilation system consists of an exhaust network with all primary ventilation fans located on the exhaust side of the mine. The size and quantity of airways in the proposed mine layout are designed to maintain acceptable velocities in shafts, airways, access drifts, and raises. Utilization in the shaft takes account of losses due to shaft furnishings and conveyances.

Dust control is expected to be a major concern underground with the dry conditions predicted from the hydrology modelling. Dry drawpoints are a source of higher-than-average dust in the panel drifts. If the silica content in the ore is higher than normal levels, the severity of the dust problem may increase. It is impossible to predict where or when dust-generating drawpoints will be encountered. Therefore, a comprehensive dust control program is incorporated into the design to mitigate the dust problem. Water sprays are installed in every drawpoint, grizzly, truck chute, crusher, conveyor transfer, and along the conveyors.

Minimum airflow velocities were specified for different types of headings. The airflow volumes and velocity requirements are based on a combination of regulatory statutes, experience, recommendations, and prudence.

Undercut Drifts

Development of the undercut is usually one of the more difficult activities to ventilate in block cave mining. With the undercut faces retreating as the cave advances, it is not practical to provide a "permanent" exhaust system and "through" ventilation. Undercut drifts must be ventilated using the auxiliary fan-and-bag ventilation systems. Similar to multiple lateral development headings, an allocation of $35 \text{ m}^3/\text{s}$ is sufficient for each loader and/or truck, following the Ontario Regulations.

Depending on the mining rate and layout, there is some serial reuse of the air with the auxiliary systems; however, many of the undercut headings are “inactive” while drilling and mucking take place in others. In IDOP, a specific additional ventilation allocation is not made for the undercut since its production is replaced by rock removal on the extraction level. Development headings on the undercut level do require specific ventilation allowances, which are planned to be defined in DIDOP. The ventilation design allows for an additional 15% of the calculated air volume as a contingency factor to allow for leakage.

16.4.3.2 Mine Air Heating and Refrigeration

Both heating and cooling are expected to be required at Oyu Tolgoi. Mine air heating is required for fresh air through both Shaft Nos. 2 and 3. Heating design criteria are shown in Table 16.15.

Table 16.15 Mine Air Heating Design Criteria

Description	Value
Site Elevation	1,100 m
Fresh Airflow Rate (Shaft No. 2 Quantity)	933 m ³ /s
Intake Air Temperature	-31°C
Discharge Air Temperature	2°C
Heating Water Temperature	120°C
Return Water Temperature	10°C

The mine air heating system at Shaft No. 2 is composed of several finned heat exchanger coils supplied with heat by a district hot water circulation loop. The coils are arranged in parallel within an insulated building. A set of two booster fans is installed on the rear of the mine air heater to draw air into the building and through the coils; the heated air is then supplied to the headframe subcollar where it is drawn underground by the primary ventilation fans. This heating system is duplicated at Shaft No. 3.

Experience by OT LLC on site has shown that heat mitigation measures are required periodically during the summer months to maintain the development crews advance rates. The most practical solution to this problem is to reduce the temperature of the air being supplied to the work face. This cooler air lowers the area temperature and allows for the additional heat loads. The design criteria used for cooling are shown in Table 16.16.

The mine development headings are cooled by packaged in-line refrigeration units, one for each development crew. The units are installed in abandoned muck bays. Cooled air passing through the units is mixed with other intake air and ducted to the working faces. The cooling units are located 300 m to 500 m from the working faces and are advanced periodically as development progresses. Refrigeration units are only required for a few months each summer.

Table 16.16 Cooling Design Criteria

Description	Value
Maximum Outside Air Temperature	39°C
Fresh Airflow per Heading	1,400 m ³ /min (1 m/s)
Virgin Rock Temperature	34°C

16.4.3.3 Mine Access and Logistics

Personnel, equipment, and materials are transported into and out of the mine via the service cage at Shaft No. 2. The cage has a payload capacity of 38 tonnes with dimensions of 3.0 m wide x 8.75 m long. The cage is sized to allow hoisting of fixed and mobile equipment with minimal disassembly.

Personnel are bussed from a central change house facility to the staging building near the collar of Shaft No. 2. OT LLC is revisiting the concept of a local, mine-only dry house and the final decision is yet to be made. Personnel walk to the shaft via a walkway tunnel and the cage lowers them to the main station, where they disembark and walk through a personnel walkway drift to the bus-loading area on the northeast side of the station. The buses transport personnel to a locker area adjacent to the main lunchroom. From there, the miners pick up the specialty PPE, hand tools, and slickers, and board smaller personnel carriers for transportation to work areas. The auxiliary hoist at Shaft No. 2 is also able to transport personnel when the main cage is unavailable or otherwise occupied.

The main focus of the Shaft No. 2 main station is the efficient and safe flow of personnel and materials. Personnel are separated from the material handling and are in safe designated areas during loading or unloading of the personnel buses. The rail shuttle cars can be unloaded on the east or west side of the cage. A designated supply laydown area with a 10 tonne monorail is provided on both the east and west sides of the cage.

The shotcrete and concrete loading area on the northwest side of Shaft No. 2 is designed for drive-through access for the transmixer trucks to eliminate the need for backing into a drift for loading. The pipes installed on the north-west side of the shaft are located for sinking operations and will remain in place for the life of the mine.

The spill collection station is located at the main station for ease of access. There is a solid brattice in the shaft between the cage and the skip compartments from the skip-loading station to the main station. A “false bottom” on the skip compartment side at the main station allows a loader to muck out any spill from the skips. No personnel or other access will be allowed on the skip compartment side of the station.

As a safety feature, OT LLC has recently decided to install brattice between the service compartment and the skip compartments in Shaft No. 2 for the full length of the shaft. This will reduce the likelihood of spilled rock from the skips entering the cage compartment and injuring personnel or damaging equipment.

Facilities for storage and transportation of mine supplies and explosives are provided underground. A central surface warehousing facility services the entire Oyu Tolgoi

operation, including the underground mine, and is responsible for delivery of supplies and materials to the Shaft Farm. Underground storage facilities for consumable supplies are provided at the underground maintenance shops and various satellite locations throughout the mine. An underground explosives magazine is centrally located to the mine operation.

16.4.3.4 Water Management

The risk of substantial inflows is believed to be low because no major shallow aquifers exist to drain into the underground. What aquifers do exist are likely to be substantially dewatered by the open pit development prior to any subsidence. The low rainfall, together with diversion of surface water, will further limit surface water inflow risks. Barring the interception of large aquifers, OT LLC identified the greatest risk of large water inflows is from a significant rainfall event draining into the cave after the cave has propagated to the surface.

Face pumps will be used to handle water encountered during lateral development. Face pump discharge is routed to intermediate sumps and a temporary stationary pumping system to lift water out of the heading. It is advisable to carry probe holes some distance ahead of the active faces to warn the crews of possible water inflows.

The main pumping system will be located at Shaft No. 2. The mine water discharge pumping system features high-head pumps capable of pumping from the main pump station to surface in a single lift. Enough pumps will be installed to cover the peak pumping rate plus allowance for maintenance.

Service water required for mine operations will be provided from clear water sources in the Gunii Holooi surface well field. Clear water from these wells is routed to a 450,000 L service water tank at the shaft farm. The tank will supply service water to the surface facilities and down Shaft No. 2. A high-pressure water line feeds directly from surface to the underground main station to provide shotcrete/concrete pipe clean-out water.

Mine service water is used for production and delineation drilling, wash down, and dust suppression. Peak service-water usage rates of as much as 32 L/s are anticipated.

The groundwater inflow rates are expected to be negligible. No water has been intercepted during shaft sinking or development. The mine pumping system has been designed with a flow rate of 10 L/s. The cost of installing pumping systems with sufficient capacity to handle inflows from all foreseeable storms would be very expensive and the equipment would be poorly utilized. It is also difficult to precisely quantify inflow rates from the limited information available. The mine emergency electrical generation plant will supply sufficient power to operate one set of main pumps in Shaft No. 2.

The feasibility study will examine hydrogeological information being developed to update planned water balance and optimize pumping system design.

16.4.3.5 Fixed Facilities

The underground design includes facilities for:

- Underground Warehouse
- Main Maintenance Shop
- Satellite Maintenance Shop
- Workplace Storage Facilities
- Fuel and Lube Stations
- Explosives Magazine
- Underground Office and Lunchroom
- Refuge Stations
- Compressed Air (portable, fixed, and mobile equipment onboard compressors)
- Shotcrete and Concrete Distribution
- Power and Communication Systems
- Pumping and Water Management Systems

16.4.4 Mine Access and Layout

The mine design includes all design stages required to access the mining areas, build the mine infrastructure, initiate undercutting, ramp up production tonnage, and mine at full production until mineable resource depletion. An isometric view of the layout is shown in Figure 16.34.

Figure 16.34 Vulcan Layout Isometric



16.4.4.1 Mine Access

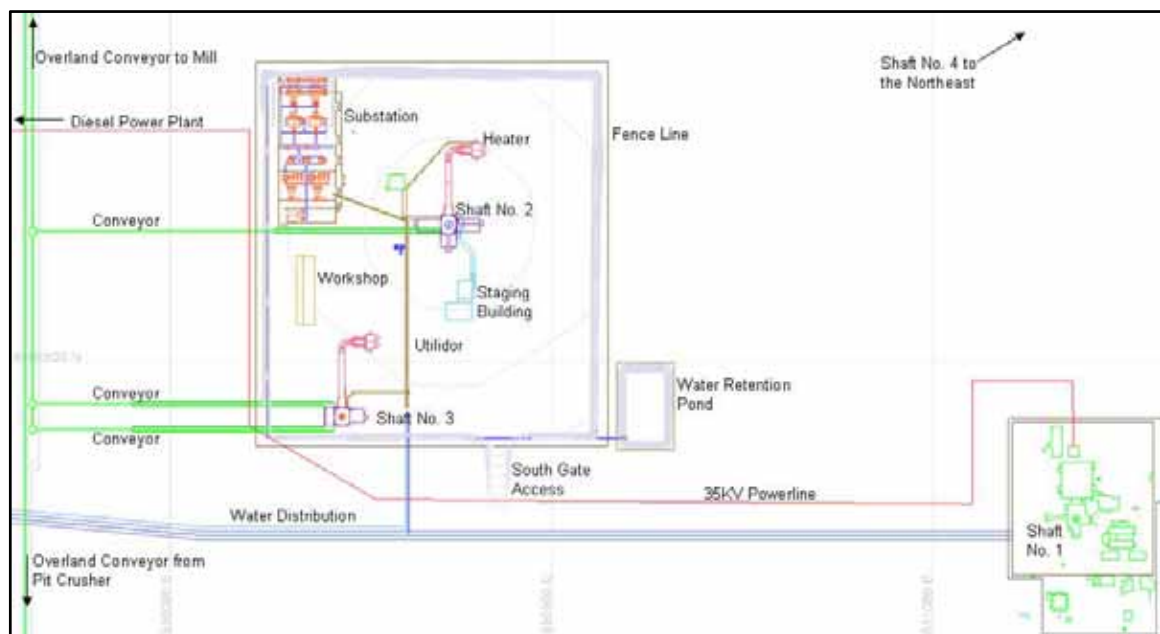
Figure 16.35 shows the Shaft Farm surface layout around Shaft Nos. 1, 2, and 3. Shaft No. 4 is approximately 3 km to the north-east of Shaft No. 2 and is used for ventilation only; therefore, no major surface infrastructure is located near Shaft No. 4 with the exception of power, water supply, and ventilation fans.

The Shaft Farm is located on the west side of the mine to promote a consistent west-to-east flow of personnel and materials into the mine, and flow of production ore from the extraction and haulage levels through the crushers and out of the mine toward the mill. Choice of location is constrained by the potential caving zone of influence on surface.

Shaft No. 1

Shaft No. 1, which has already been constructed, is required for exploration and geotechnical characterization and is also used for preproduction development, construction support, initial hoisting of ore and waste, secondary egress, and principal ventilation exhaust. The Shaft No. 1 site is west of and between the Hugo Dummett north and south zones. Shaft No. 1 was excavated to approximately 7.5 m diameter and finished to 6.7 m. The shaft collar elevation is 1,165 m and the depth of the shaft is 1,385 m.

Figure 16.35 Shaft Farm Surface Layout



Shaft No. 1 was sunk bald with rope guides. There are two hoists, the main service/muck hoist and an auxiliary hoist. The sinking hoist is a double-drum hoist with two conveyances (sinking buckets) in counterbalance. The auxiliary hoist is single-drum. The auxiliary cage is used for personnel, light service hoisting, and to provide emergency egress.

Shaft No. 1 will be reconfigured from sinking arrangement to service and muck hoisting configuration in 2012. Gear weighing up to 15 tonnes can be accommodated for mine development and movement of heavy loads. Shaft No. 1 is planned to support all development and construction activities until the Shaft No. 2 and service hoists are.

Access to Lift 1 is via the 1,300 mL station. The characterization development access is being driven 5.0 m wide x 5.0 m high. Drill stations to support exploration and geotechnical drilling are being excavated. A diamond drill program will provide further definition of mineralization and samples for geotechnical and metallurgical testing.

Shaft No. 2

Shaft No. 2 will provide for substantially all movement of personnel, materials, equipment, and services for the mine. It also provides approximately a third of the total ore hoisting capacity of the mine and serves as a primary ventilation intake.

Shaft No. 2 is a 10 m diameter, concrete-lined shaft fully equipped with steel sets on 6 m centres. The collar of Shaft No. 2 is at 1,174 m, and the shaft is sunk to a depth of 1,335 m.

A 6.75 m diameter friction hoist serves as a production hoist. Ore is hoisted using two 54 tonne skips. A similar 6.75 m diameter friction hoist serves as the service hoist. The service cage has a capacity of 38 tonnes, or 150 persons. A 3.05 m diameter single-drum hoist serves as an auxiliary hoist.

The Shaft No. 2 loading station is at the 1,195 mL. Ore is fed from two 4,000 tonne coarse ore bins onto weigh belts by apron feeders. The high-speed belts load the skips directly, with a separate loading belt feeding each skip. Bin dimensions and loading level are currently being reviewed.

Shaft No. 3

Shaft No. 3 is similar in design to Shaft No. 2 insofar as is it a 10 m diameter, concrete-lined shaft fully equipped with steel sets on 6 m centres. Unlike Shaft No. 2; however, Shaft No. 3 does not have a service hoist. Shaft No. 3 serves as a dedicated production shaft and fresh air supply. The collar of Shaft No. 3 is at 1,174 m, and the shaft is sunk to a depth of 1,273 m.

Shaft No. 3 hoisting utilizes two 6.75 m production friction hoists, each equipped with two 54 tonne skips. The hoists and skips are identical to the production hoist and skips in Shaft No. 2. The Shaft No. 3 hoisting system has a total productive capacity of 55,606 t/d.

Shaft No. 3 has two ore bins and a loading station to service each production hoist, one on the north side of the shaft and one on the south side of the shaft. Each ore bin has a capacity of 5,000 tonnes. The feeders and loading conveyors are identical to those at Shaft No. 2.

Shaft No. 4

Shaft No. 4 provides additional exhaust ventilation required after production exceeds 28,000 t/d. The ventilation capacity of the shaft at 10 m/s is 1,000 m³/s. Shaft No. 4 is used as dedicated primary exhaust ventilation and secondary escape. Shaft No. 4 has a collar elevation of 1,174 m, and is concrete-lined to a 10 m diameter. There are no conveyances in the shaft except for an emergency auxiliary cage. The shaft is sunk to a total depth of 1,153 m. There are two ventilation stations (1,117 mL and 1,153 mL). Location of Shaft No. 4 is currently under review. It may be relocated to the east of the mine to improve development and access time and to better support Lift 2 development. This work is part of the study of alternative means of establishing the vertical development required to complete the ventilation infrastructure to mitigate the issues caused by the difficulties in raise boring.

16.4.4.2 Level Development

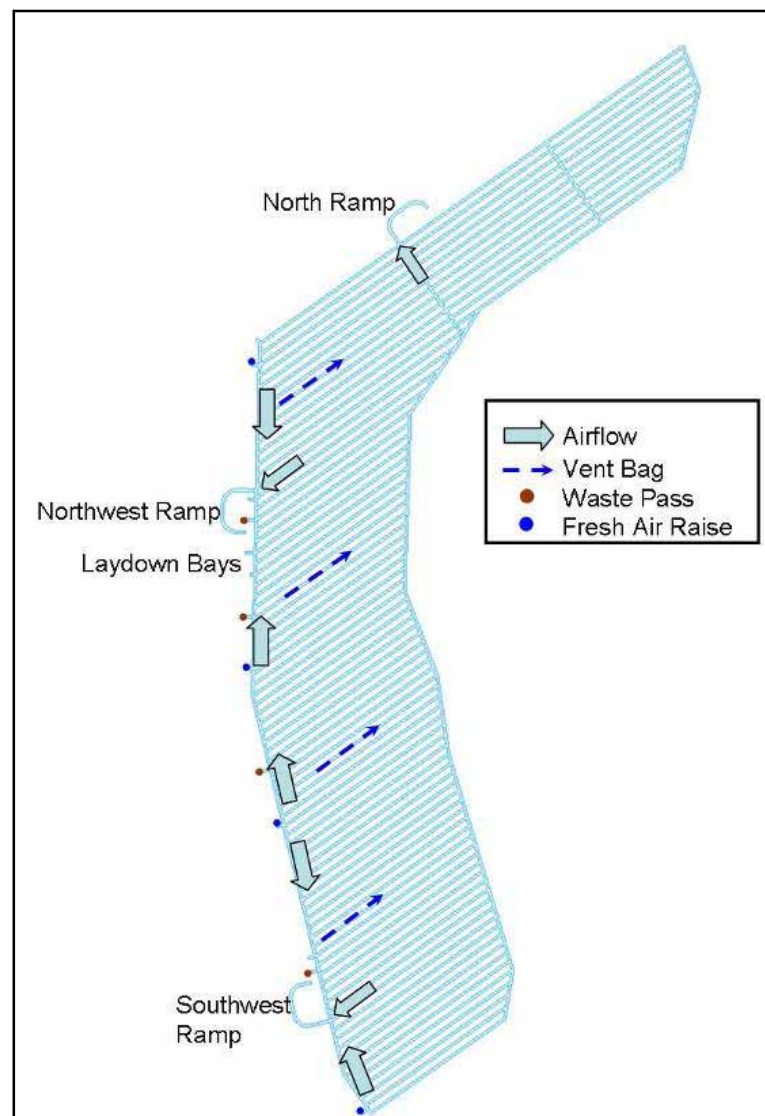
Undercut Level

Pairs of undercut drifts directly above the drawbell are driven parallel to the extraction panels, 17 m above the extraction level. The bottoms of the undercut drifts coincide with the tops of the drawpoint minor apices. A slice is then blasted off the bottom of the ore, above the undercut drift, removing support for the overlying draw column and forming the major apex of the drawpoint.

A perimeter drift is established around the footprint to provide access to the undercut drifts. The perimeter drift along the west is for permanent access and is offset from the footprint by 50 m. The perimeter drift along the east is for temporary access during development and is offset by only 15 m. Final location of the cave initiation point will likely change these details.

Due to the length of the footprint along the north end, temporary internal cross-cut drifts are driven perpendicular to the undercut drifts to provide intermediate access to the active undercutting areas. As the cave progresses to the north and the internal drifts are consumed by the cave, new temporary drifts are added to maintain close access to undercutting areas. Figure 16.36 shows the layout of the undercut level.

Figure 16.36 Undercut Level



Undercut and drill drifts are driven using conventional rubber-tired development equipment and temporary ground support measures. Panel spacing is 14 m and drift dimensions are 4.0 m high x 4.0 m wide to accommodate development equipment. Undercut drifts are connected at intervals to facilitate ventilation.

Ground support in the undercut headings is temporary, but substantial quantities of rockbolts and fibre-reinforced shotcrete are required due to the high abutment stresses experienced.

Swell muck removal is required to provide a free face for continued blasting of the advancing undercut face, and to insure removal of support from the overlying ore. Drawbell slots are driven ahead in advance and used for muck passes down to the extraction level, where ore is transferred to ore passes leading to the haulage level. When necessary, undercut muck can be dumped down small raises on the fringe

between undercut and extraction levels for rehandle, or hauled down ramps to the extraction level and dumped into ore passes.

The plan is to have one cave front advance to the north and a second advance to the south. Based on this design, two caving areas advance in opposite directions, with each cave front having the capacity to produce in excess of 42,000 t/d. The key factors in limiting the production rate from each cave front are the rate at which drawpoints can be constructed and the rate at which undercutting advances. A maximum of 150 drawpoints are constructed per year from the combination of both cave fronts.

Undercutting is initiated along the eastern side of the footprint. Once hydraulic radius is achieved, the cave advances toward the north and south. An illustration of the undercutting sequence is presented in Figure 16.37.

A crinkle-cut undercut configuration is planned for Lift 1. A series of relatively flat blastholes are drilled across the pillars over the drawbells from either side of the drawbell, and a series of inclined holes is drilled over the extraction level panel drift to form the major apex. An illustration of the undercut configuration is presented in Figure 16.38.

Extraction Level

The extraction level consists of panel drifts, drawpoints, drawbells, a perimeter drift surrounding the level, and a fringe drift along the eastern side of the footprint. The orientation of the extraction level panel drifts is determined by geotechnical factors. To manage stress abutment loads, the perimeter drift is offset from the footprint by a minimum of 50 m. The fringe drift is parallel to the east perimeter drift to allow access to the working areas without interfering with production loaders dumping in the ore passes along the east perimeter drift. Figure 16.39 shows the extraction level layout, location of raises, and direction of haulage and airflow.

The extraction level is driven at a gradient of approximately -3% from the south toward the north. This allows the extraction level to follow the plunge of the orebody to some extent.

Figure 16.37 Undercutting Sequence

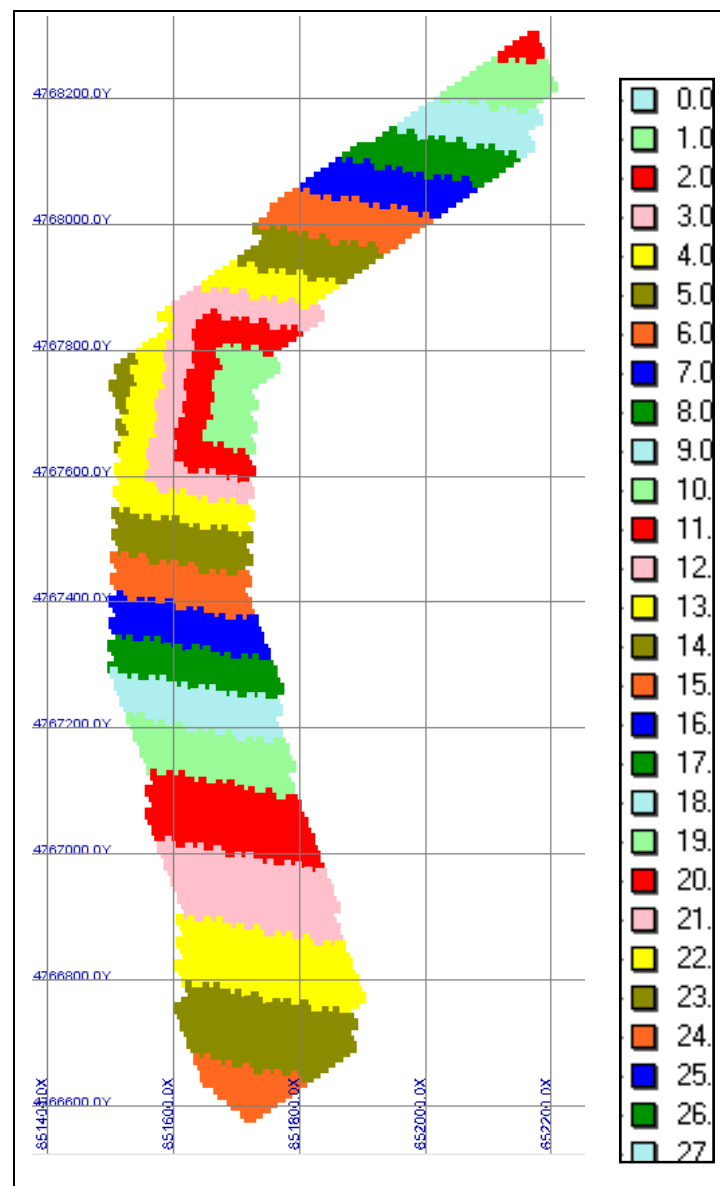


Figure 16.38 Undercut Drilling Pattern

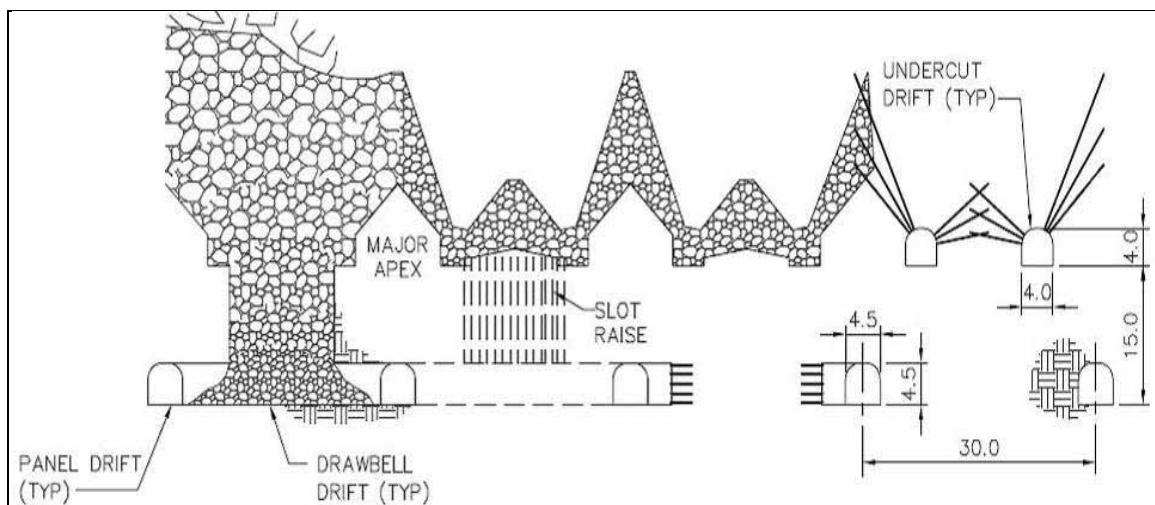
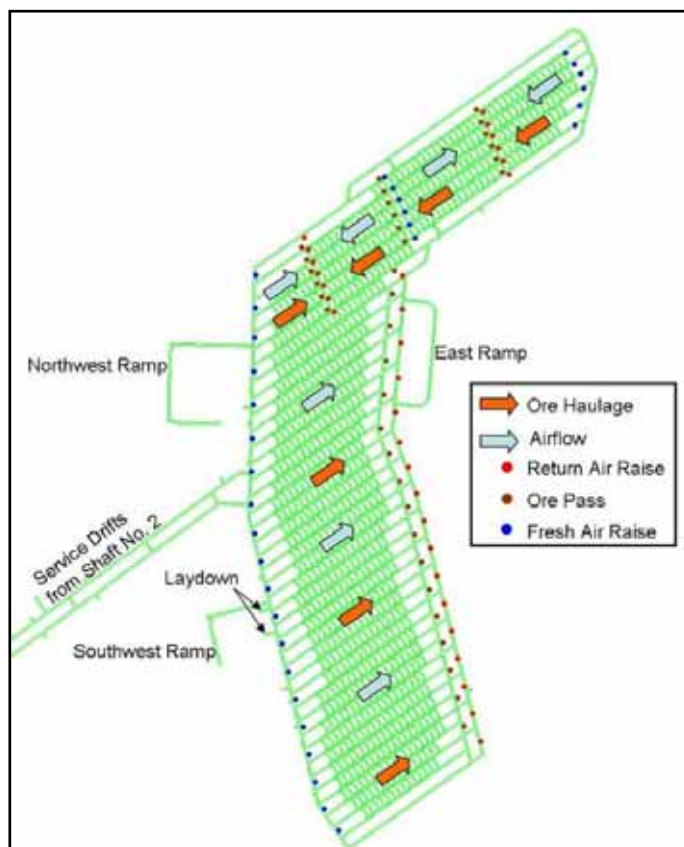


Figure 16.39 Extraction Level



All major infrastructure including the maintenance shop, lunchroom, explosives magazine, and fuel bay are located along the west side of the extraction level to allow for direct access to the service shaft. The satellite shops will be to the north and south along the east fringe drift to service areas that are further away from the central infrastructure.

Each draw area contains one drawbell, which is serviced by two drawpoints accessed from adjacent panels. The resultant drawpoint spacing along the production drift is 15 m. Production drift size is 4.5 m x 4.5 m, with a 55° turnout angle into the drawbell drift (Figure 16.40). Each drawpoint services half of one drawbell, giving it an area of influence of 15 m x 15 m. Periodic drawpoint repair is anticipated and accounted for throughout the mine life.

Panel drift spacing is 28 m, drawpoint spacing is 14 m, and drift dimensions are 4.5 m high x 4.5 m wide to accommodate production equipment.

Oversize in the drawpoints is broken using standard drill and blast techniques employing small single-boom jumbos and conventional explosives. In the IDP10 mine plan, ore would be mucked from drawpoints by LHD to an ore pass along the east perimeter drift. The ore passes are spaced at every other panel drift so the maximum one-way tramping distance is approximately 300 m per LHD. The ore passes are 3 m diameter vertical raisebores, 35 m in height extending from the extraction level to the haulage level.

OT LLC is actively considering changing the ore pass location to the centre of the panel drift. This would reduce the average tram length to some 150 m, presenting a significant savings in operating cost and increasing in production efficiency.

Each ore pass is fitted with a rock breaker and 600 mm aperture grizzlies set into the panel floor. The current mine layout is designed to incorporate a second ore pass in the event that the primary ore pass should fail. On average, approximately 7.5 Mt pass through each ore pass. Due to uncertainties associated with the strength of the rock mass, coupled with the tonnage scheduled to report to each ore pass, McIntosh assumed that each ore pass is repaired after 5 Mt has passed through it.

Extraction level ventilation was planned using 3 m diameter raises leading up from the fresh air drifts on the west and down to the return air drifts on the east. Fresh air raises are raisebored (54.5 m high) on 56 m intervals along the west perimeter drift. Return air raises are raisebored (35 m high) on 56 m intervals along the east fringe drift. Using this approach, fresh air enters the panel from the west and is drawn toward the return air raise on the east (see Figure 16.41).

Haulage Level

The haulage level consists of truck haulage drifts 5.0 m high x 5.0 m wide driven at a maximum 3% gradient to follow the extraction level. Ore reports to the haulage level truck loading chutes through the production ore passes. Loading bays at each chute allow trucks to back into the chute area for loading. This allows continuous travel on the haulage loop without interruption from loading trucks.

The image contains three architectural drawings of a herringbone drawbell structure:

- TYPICAL HERRINGBONE DRAWBELL PLAN**: A top-down view showing a herringbone pattern within a rectangular frame. Dimensions include 4300 for the overall width and 4250 for the central width. Callouts include '100 THICK SHOTCRETE 1/2 100X100 MM', 'EDGE OF CONCRETE', 'CORRUGATED STEEL LAGGING', '1000 LONG FULLY DRILLED TENDONS @ 1000 SPACING EACH WAY', and '3000 LONG ROCK BOLT AT EACH STEEL COLUMN'. A circular callout '1' points to a detail.
- SECTION A**: A cross-section showing the vertical profile. Dimensions include 315, 3200, 300, 2700, and 1340. It shows the concrete structure, corrugated steel lagging, and a circular detail '1'.
- SECTION B**: A cross-section showing the vertical profile. Dimensions include 4000, 2250, 4100, and 2250. It shows the concrete structure, corrugated steel lagging, and a circular detail '1'.

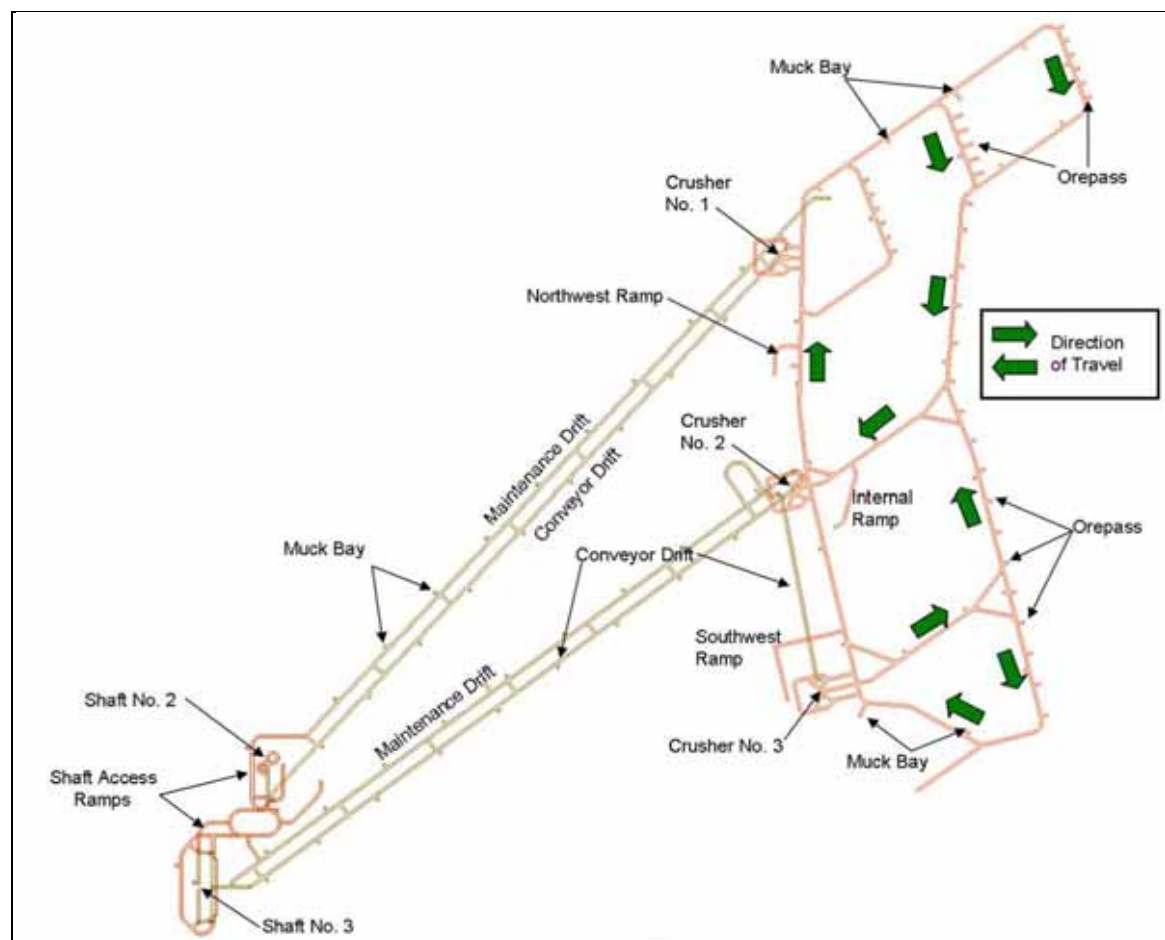
A 3D perspective diagram of a multi-level underground mine layout. The layout consists of several horizontal levels connected by vertical shafts. The top level is labeled 'Extraction Level' and is colored green. Below it is the 'Intake Level', colored blue. At the bottom is the 'Haulage Level', colored red. A vertical shaft labeled 'Orepass' connects the Extraction Level to the Haulage Level. Another vertical shaft labeled 'Exhaust Raise' connects the Intake Level to the Haulage Level. A horizontal shaft labeled 'Intake Raise' connects the Intake Level to the Extraction Level. The diagram shows a grid of tracks or roads on each level, with the Haulage Level having a more complex arrangement of tracks and supports.

The truck loops are connected to provide the flexibility of directing ore to any available crusher. Figure 16.42 shows the layout of the haulage level. The dispatch system directs the trucks to a crusher that has capacity to receive muck. The dispatch system can also direct development trucks to enter the haulage loop via the internal ramp to dump development muck into the crushers. OT LLC is actively evaluating the use of 80 tonne trucks for main line haulage. This would likely require slightly larger haulage drifts.

Two gyratory crushers are installed, as needed, below the haulage level, along the west side of the footprint. Crusher No. 1 is adjacent to the northern part of the mining footprint, where caving is initiated, and Crusher No. 2 is near the central part of the mining footprint.

Crushed ore is loaded onto a single inclined main conveyor that transports ore to skip-loading stations at Shaft No. 2. The ore can be transferred to a 500 m long conveyor for delivery to ore bins at Shaft No. 3.

Figure 16.42 Haulage Level



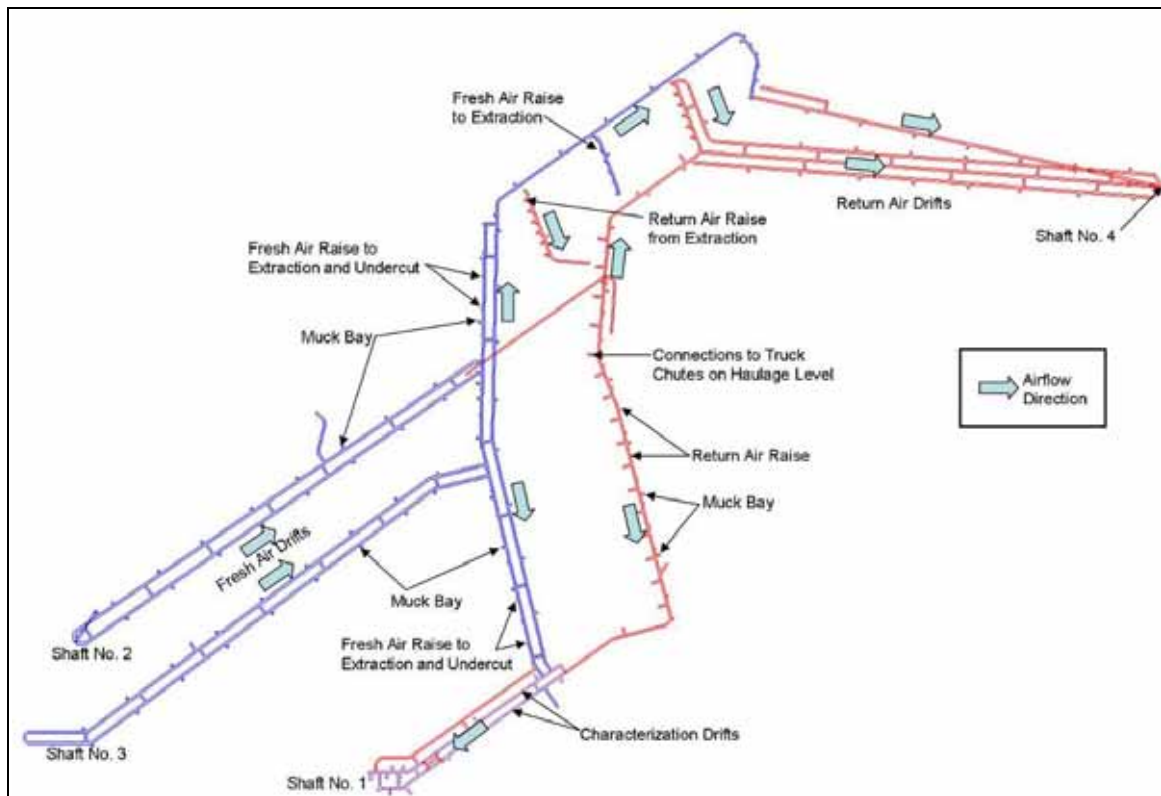
Ventilation Level

The ventilation levels consist of separate intake and exhaust manifold drifts. The exhaust level is 40 m below the extraction level, at the same elevation as the haulage level, running primarily along the eastern side of the footprint. Figure 16.43 shows the layout of the ventilation level.

The exhaust and haulage drifts are interconnected along the eastern perimeter at the truck loading bays and are driven 5.5 m high x 5.0 m wide. This provides the ability to exhaust the dust and particulates from the extraction and haulage levels directly to the

exhaust shafts. No travel will be allowed in the exhaust drift because of the high air velocities and probability of dusting in these headings.

Figure 16.43 Ventilation Level



The main ventilation drifts between the shafts and the orebody are driven 7.0 m high x 6.0 m wide. The increased size of these drifts assists in reducing the velocity in the headings as well as the pressure on the main fans.

The intake level is 60 m below the extraction level driven along the western side of the footprint (20 m below the haulage level) and is also driven 5.5 m high x 5.0 m wide. The drift dimensions are based on driving the heading with a standard set of development gear and a 1.5 m diameter ventilation duct hung from the back of the drift. During development and production, the fleet consists of 30 tonne trucks, 50 tonne trucks, 5 m³ LHDs, and 7 m³ LHDs.

Raises connect the intake and exhaust manifold drifts to various working levels to direct the air to and from the workplaces. The raises are located in stubs off the main ventilation drift. This allows the ongoing development of raises without impeding airflow or traffic along the ventilation drift. These drifts and raises are part of a ventilation system that connects the main intake and exhaust shafts to meet the ventilation requirements from start of production ramp-up to full production.

16.4.5 Mine Development Plan

Mine infrastructure is developed on an as-soon-as-possible basis, leading up to first undercutting. Production build-up drives the development schedule from this point on, leading into full production. Infrastructure excavation and construction during the build-up period is completed according to production milestone requirements.

Ventilation and hoisting constraints dictate the number of development crews that can be supported during pre-production development and construction. As such, the first few years of the schedule are resource-driven using only two development crews. The number of development crews increases as additional ventilation and hoisting capacity is brought on line to a maximum of 11 development crews in 2014.

Once additional ventilation raises to surface are completed during preproduction, limited hoisting capacity becomes the main restriction on development scheduling. Commissioning of the Shaft No. 2 service hoist and rock handling system is required to increase beyond five underground crews.

Commissioning of the rock handling system is also required before the first drawbell is blasted. If the rock handling system is not in place at the start of caving, then the risk of slowing the cave advance increases due to interference between production and construction crews. Making the rock handling system a priority increases the probability that the mine will be able to achieve the targeted production rates once caving begins.

Infrastructure and fixed facilities are constructed separately and appended to the finish dates of the excavations in the development schedules. The facilities are completed before actually being needed, thus allowing for start-up and commissioning and minimal unplanned delays in component deliveries or construction.

Development scheduling activities were divided into three key periods: preproduction, production build-up, and sustaining production. Each period was reviewed to determine the infrastructure and development required to be completed during that period. The periods and the key infrastructure associated with each period are defined as follows:

- Pre-production, prior to blasting the first drawbell.
 - Ventilation Raise No. 1, Shaft No. 2, Crusher No. 1, Incline Conveyor No. 1, all development required to construct the first 34 drawpoints.
- Production build-up, ±30,000 t/d.
 - Ventilation Raise No. 2, Shaft No. 3, Crusher No. 2, all development required to construct up to 276 drawpoints.
- Sustaining production, achieve full production through the end of life of mine.
 - Ventilation Raise No. 3, Shaft No. 4, all remaining development.

Development rates are calculated from first principles. SRK supplied ground support measures and the ratio of Category 1 and Category 2 ground types. The ground support criteria applied to each heading type and size determine their expected advance rates. Table lists all of the heading types used for the mine design and the advance rates for each.

Table 16.17 Development Advance Rates

Heading Size		Performance			
Width (m)	Height (m)	Inside the Footprint(m/d)		Outside the Footprint(m/d)	
		Single	Double	Single	Double
4.0	4.0	5.3	7.1	5.4	7.2
4.5	4.5	3.8	5.1	3.8	5.2
5.0	5.0	4.4	6.0	4.5	6.1
5.0	5.5	4.2	5.6	4.3	5.8
6.0	7.0	–	–	4.0	5.4

Development crews drive multiple headings whenever possible and by doing so increase utilization of crews and equipment. Double-heading estimates are prepared based on a 35% increase over the single-heading rate for the same drift size. These advance rates become Mine 2-4D object coding attributes.

The Lift 1 mine plan includes a number of mass excavations, some of which are very large. Table 16.18 lists mass excavations and durations. Ground support requirements for each type of excavation were developed by SRK; these requirements can have a significant impact on the time and expense involved for excavating. Timing, based on ground support type, is taken into account in scheduling each mass excavation.

All internal ventilation raises and ore passes are developed using raisebores. All raiseboring assumes that the drill rigs, drill pipe, bits, reaming heads, and crews are onsite and available as necessary. Vertical advance rates exclude mobilization and demobilization of the raiseboring rig and crews. Advance rates are applied in accordance with raise diameter and length. Table 16.19 lists a summary of the vertical advance rates.

Table 16.18 Mass Excavation Rates

Infrastructure	Excavation(months)
Gyratory Crusher	11
Shaft No. 2 Station	4
Main Shop	8
Satellite Shop	3
Explosives Magazine	3
Fuel and Lube (Extraction)	1
Fuel and Lube (Haulage)	1
Lunchroom	4
Main Pump Station	<1

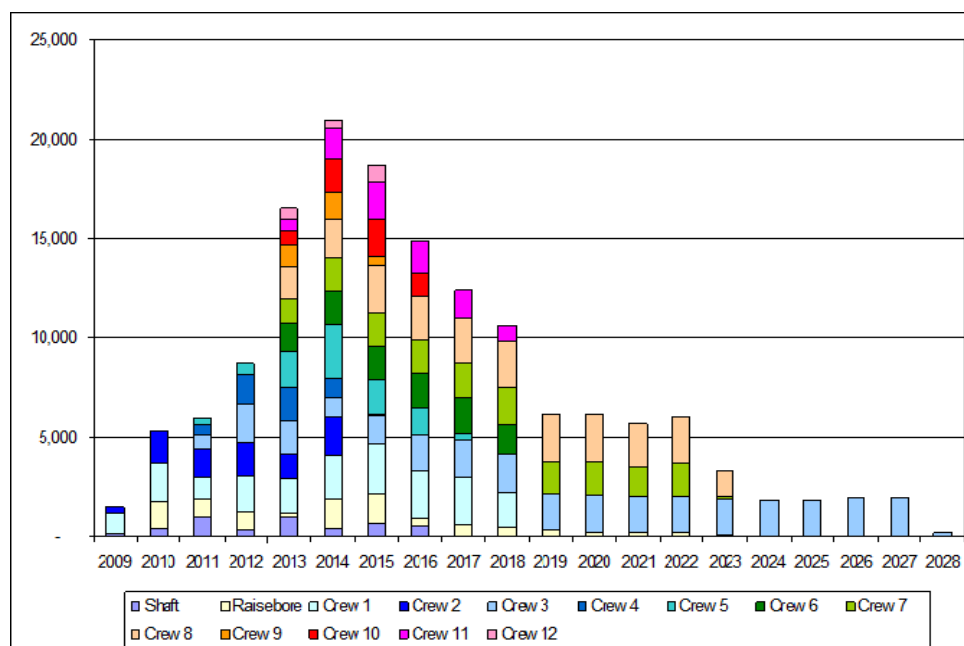
Table 16.19 Vertical Advance Rates

Heading Description	Finished Size (m)	Excavation Performance (m/d)
Shaft Sinking		
Shaft No. 2	10.0 m	2.7
Shaft No. 3	10.0 m	2.8
Shaft No. 4	10.0 m	2.8
Vertical Development		
Vent Raise – Internal	3.0 m dia.	3.0
Ore passes – Internal, Excluding Lining	3.0 m dia.	3.0

The development schedule produced for IDOP is shown in Figure 16.44. The construction durations for infrastructure were estimated from first principles based on experience from similar operations.

Criteria considered in equipment selection included suitability, equipment standardization, and cost. The equipment selection process is iterative and aimed at obtaining the optimum equipment required to achieve the planned development and production quantities and rates. The mobile equipment required for lateral development includes drill jumbos, LHDs, haul trucks, and ground support equipment. Common to all drift development is the two-boom jumbo, rock bolter, telehandler, shotcrete/concrete sprayer and transit mixer, and an emulsion truck. Apart from drifting jumbos and loaders, other mobile equipment required for development includes cable bolting jumbos and utility service equipment. The current fleet of equipment at site includes a Toro 7 LHD, Axera 7 development, and Normet shotcrete spraymec.

Figure 16.44 Development Quantity Schedule (metres)



The mine layout was analyzed and development quantities were determined from the development schedule. Underground operating conditions, spatial dimensions, and other limiting factors are taken into consideration. Roadway grades vary between +15% and -15%, with the exception of the main conveyor and maintenance drifts. All vehicles are required to have the ability to operate under loaded conditions at these maximum gradients. Production and support equipment quantities were determined from performance estimates and the production schedule. Mobile capital equipment is purchased, rebuilt, and replaced to suit fleet requirements for underground mine development and construction. The fleet requirements are summarized in Table 16.20.

Fixed equipment is purchased as dictated by the mine development and construction schedule.

Table 16.20 Mobile Equipment Units Required on Site

	Pre-production	Build-up	Sustaining
LHD, 3.0 m ³ (Cleanup)	1.0	2.0	3.0
LHD, 5.0 m ³ (Development)	6.0	6.0	4.0
LHD, 7.0 m ³ (Production)	1.0	10.0	23.0
Haulage Truck, 30 t (Development)	6.0	3.0	2.0
Haulage Truck, 50 t (Production)	0.0	8.0	17.0
Drill Jumbo, 2-Boom (Development)	7.0	6.0	3.0
Drill Jumbo, 3-Boom (Development)	1.0	2.0	0.0
Drill Jumbo, Single Boom (High Hang-ups)	0.0	2.0	4.0
Drill Jumbo, Single Boom (Cable Bolting)	2.0	2.0	2.0
Drill Jumbo, Single Boom (Rock Bolting)	8.0	8.0	4.0
Concrete/Shotcrete Transit Mixer Truck	6.0	6.0	6.0
Shotcrete Placer Truck	4.0	4.0	2.0
Concrete Pump	1.0	2.0	2.0
Drill Jumbo, Single Boom (Production)	1.0	5.0	6.0
Drill Jumbo, Commando (Secondary Breaking)	0.0	2.0	4.0
Explosives Truck/Jumbo (Development and Undercut)	4.0	4.0	5.0
Scissor Lift Truck	4.0	5.0	2.0
Lubrication Service Truck	2.0	3.0	4.0
Boom Truck	2.0	2.0	1.0
Service Truck	1.0	2.0	2.0
Electricians Service Truck	1.0	2.0	1.0
Underground Road Grader	2.0	2.0	2.0
Personnel Carrier, Underground	4.0	6.0	6.0
Heavy Duty Lowboy Trailer – 25 t+	1.0	1.0	1.0
Heavy Duty Trailer – 40 t	1.0	1.0	1.0
Underground Buses 60 men	2.0	4.0	5.0
Telehandler, Underground	5.0	7.0	6.0
Skid Steer Clean Up Loader	0.0	1.0	1.0
Crew Cab Pickup, Underground (General Purpose)	11.0	20.0	20.0
Surface Loader	2.0	1.0	1.0
Surface Flatbed – 1 t – Small Crane and Lift Deck	2.0	2.0	2.0
Fuel, Haul Truck	2.0	2.0	2.0
Wheel Loader with Tire Handler	2.0	2.0	2.0
Pallet Handler	4.0	5.0	4.0
Grove, RT700E	1.0	1.0	1.0
Backhoe, 446D	1.0	1.0	1.0

16.4.6 Personnel

Personnel requirements are developed to support planned development, construction, and operations requirements for the mine. Personnel staffing is based on production phases in the life of the operation: pre-production, production build-up, and sustaining production.

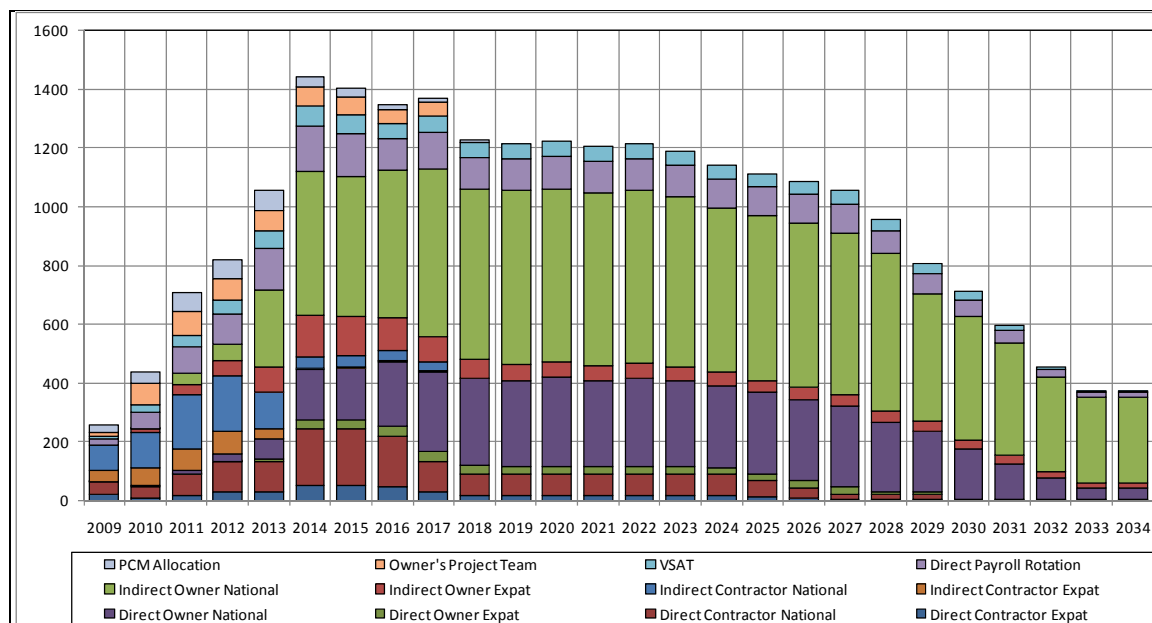
Personnel staffing is split into the following classifications:

- Owner's Project Team
- EPCM Team
- Contractor's Team
- Owner's Operations Team

Direct and indirect labour requirements are established to suit the selected mining method, support systems, and general mine requirements during mine development, construction, and operations. Personnel requirements are based on an operating schedule of 12 hours per shift, 2 shifts per day, and 7 days per week. Non-productive days are considered for holidays, dust storms, etc., in the staffing schedules. The mine is estimated to operate effectively 350 days per year of the average 365 calendar days.

The underground personnel requirements for Lift 1 are presented in Figure 16.45.

Figure 16.45 Underground Personnel Requirements



Senior staff will work a five-days-in, two-days-out rotation between the Oyu Tolgoi site and Ulaanbaatar. Owner's staff (expatriate) work a six-weeks-in, two-weeks-out rotation, 12 hours per shift. Owner's staff (nationals) work a two-weeks-in, one-week-out rotation, 12 hours per shift. Direct-hired national employees work a two-weeks-in, one-week-out

rotation; on site, they work 12 hours per shift. Individual national employees work 201 days per year.

Contractor staff and hourly personnel work a variety of different work schedules. These include:

- Expatriates – 8 weeks on, 4 weeks off.
- Nationals – 4 weeks in, 2 weeks off.

All work a 12-hour shift.

Travel

All employees associated with underground mine construction and operations work on rotation. Travel between Oyu Tolgoi and employee points of origin will be on the employees' time.

16.4.7 Further Work - Underground Feasibility Study

As part of DIDOP the underground study work is being brought to feasibility study level, the scope of work includes all activities from extraction to the shaft collars and will optimise the development and production of Hugo North Lift 1. OT LLC plans to finalise DIDOP in late 2012.

OT LLC have undertaken reviews of the underground study work and identified areas for further study that may change the designs and production in DIDOP. Key areas identified are:

Draw Point Spacing.

The recommended drawpoint spacing is 14 m to 15 m. OT LLC has considered changing drawpoint spacing from IDOP's 15 m x 15 m to 14 m x 14 m.

Ventilation Design Standards

The IDOP mine design conforms to Ontario Regulations (Canada). Where the mine design departs from Ontario regulations, an assessment of risk was undertaken to ensure that an equivalent or higher standard of safety is applied.

OT LLC has advised that it now appears that the Mongolian Regulations are, in some instances, more stringent than the Ontario regulations used in IDOP and IDP10. Such issues as backup and emergency ventilation system requirements have recently been identified.

Vertical Development and Ventilation Infrastructure

The design of the underground incorporates 4 shafts and 5 raises to ventilate the mine. The intended method of construction for the 5 ventilation raises is by raise boring. There has proven difficult and has suffered delays, the primary reason for schedule slippage has been due to poor ground conditions. Completion of the ventilation raises is on the

critical path to production from the underground mine as the raises are required to meet the ventilation requirements for development and provide alternative means of egress should Shaft 1 be unavailable. In order to mitigate and minimise risk to the project production and to OT LLC is investigating alternative means of establishing the vertical development required to complete the ventilation infrastructure.

Work to determine the impact of this issue is being carried out. The full extent of the additional regulatory requirements study is not yet known, but they will likely contribute to an increase in capital and operating costs. Initial analysis by OT LLC suggests that the underground production could be delayed by around twelve months and that each month's delay could have an NPV value of \$50 M. These investigations are planned to be incorporated into DIDOP and will allow OT LLC to quantify potential costs and changes required to meet the Project schedule.

In addition possible scope changes have been indicated including:

- Additional development and excavation expenditures
- The addition of a shaft No 5
- An additional crew underground
- Increased operating cost due to design changes

Underground Crushers

IDP10 assumed that trucks delivered ore to three 54 x 77 gyratory crushers. OT LLC undertook a trade off study to optimize the crusher and conveyor system design. The results of this study indicated that crushing/conveying duties could be handled by two each 60 x 89 gyratory crushers and a single inclined conveyor. This system is now being used as the foundation for mine material handling design in DIDOP.

Ore Handling

OT LLC is actively considering relocating the production grizzlies to the centre of the production panels to reduce the average tramming distance of the production mucking units and the haulage level footprint. If this change is made, the haulage level will need to be redesigned in DIDOP.

As part of DIDOP, the number and capacity of ore bins at the skip loaders are being reviewed to optimize flexibility and reliability and to reduce hoisting depth.

16.5 Mining Production Schedules

16.5.1 Production Scheduling Method

Both IDP10 and the present IDOP study have examined production from open pit mining of the Southern Oyu deposits and underground mining from Hugo North. Rio Tinto's Technology and Innovation (T&I) group in Salt Lake City, working with Oyu Tolgoi's open pit planning department, prepared the open pit mining work. The underground schedule for Hugo North Lift 1 was provided by the Oyu Tolgoi underground team based in Vancouver. The case adopted for IDOP assumes a 50% plant capacity expansion, is based on Mineral Reserves only, with no Inferred Mineral Resources, and includes no underground mining areas other than Hugo North Lift 1.

Hugo North Lift 1 encompasses two production areas, which are reported separately: OT LLC Lift 1 and EJV Lift 1. The latter is the OT LLC Entrée Joint Venture area. The Southern Oyu pits are contained completely within the OT LLC lease area. The mining areas for the IDOP Reserve Case are listed below.

- Southern Oyu Open Pits
 - Pit stages 1 to 3 (Southwest)
 - Pit stage 4 to 6 (Central)
 - Pit stages 7 to 8 (Southwest and Central)
- Hugo North Block Cave Lift 1
 - OT LLC Lift 1
 - EJV Lift 1

The following periods were used for production scheduling:

- Year -1 to Year 2(2011–2013)..... monthly
- Years 3 to 5 (2014–2016)..... quarterly
- Year 6 (2017) onward..... annually

The following constraints were applied to open pit scheduling:

- Maintain a constant mining rate for a relatively long period for efficient use of the mining fleet. The fleet capacity with two 34 m³ diesel shovels, two 55 m³ electric shovels, and two 18 m³ FELs is estimated to be 112 Mt/a. The haul truck capacity is 290 tonnes.
- Include re-handle from stockpiles in total movement capacity.

For process scheduling, open pit ore feed was split into three cut-off grades: Low Grade, Medium Grade, and High Grade. This allows the additional plant capacity after underground ore is depleted to be met with ore feed from the open pit at an elevated cut-off grade. This approach to open pit mine planning has consistently been shown to bring forward revenue and increase the mining rate with a net improvement in the project NPV. All ore above the marginal cut-off grade is used as plant feed.

The open pit ore grade definitions are as follows:

- Low Grade
 - South West ore type 1 >\$6.238/t
 - Central ore types 2, 3, and 4 >\$5.204/t
- Medium Grade
 - Ore types 1 – 4 >\$14.5/t
- High Grade
 - Ore types 1 – 4 >\$22.5/t

Criteria for selection were as follows:

- The low-grade cut-off is the breakeven cut-off, equal to the total processing and G&A costs.
- The medium-grade cut-off is an elevated cut-off that brings metal forward in the production schedules.
- The high-grade value cut-off allows recovery formulae to be applied at the scheduling level.

Mining costs are not included in the NSR cut-off values for the low-grade ore. This is defined as material that must be removed while accessing the medium- and high-grade ores and therefore does not need to support its mining cost. Mid- and high-value cut-off materials support both their mining costs and the associated waste stripping ratios.

The following production scheduling methodology was used to balance mine, mill, and stockpile quantities:

- Underground ore is designated the priority feed. After the available underground ore is fed to the plant, the additional capacity is met with open pit ore.
- Plant throughput capacity is determined by calculating the available mill hours after the underground ore is processed.
- Ore feed for concentrator start-up in August 2012 is exclusively from the open pit. Underground ore is only processed when the grades exceed those of the open pit material in January 2013. Any underground material hauled before this time will be stockpiled and not processed.
- Ramp-up factors were applied to the Year 1 processing production.
- The production schedule is based on Proven and Probable Mineral Reserves only. No Inferred resources were used in the open pit planning.
- The open pit schedules were based on mining inventories by bench reported within the pit stages.
- Medium- and high-grade ore mined during preproduction is stockpiled and processed in Year 1.
- The total movement from different pit stages was balanced to smooth waste and ore production rates and to match the load and haul capacity.

- Low-grade stockpiling was used to balance the mining rate where necessary.
- Central ore cannot be blended with Southwest ore and is not processed until 2021.

The throughput rate algorithm shown in Table 16.21 was developed by SGS. This formula was applied to all the blocks in the mining model. The average throughput rates are shown in Table 16.22.

Table 16.21 Plant Throughput Rates

$P_{80}=113 \cdot C_i^{0.26} \cdot SPI^{0.60} \cdot BM^{0.88}$ max P_{80} 220 μm
$t/h \text{ (instantaneous)}=29320 \cdot C_i^{0.19} \cdot SPI^{0.36} \cdot BM^{0.24}$ max TPU= 5,500 t/h

Table 16.22 Average Plant Throughput Rates

Metallurgical Ore Type	Deposit	Throughput Rate	
		t/d	Mt/a
1	Southwest (including South and Far South)	97	32.7
2	Central – chalcocite	132	44.3
3	Central – covellite	132	44.5
4	Central – chalcopryrite	119	40.1
6	Hugo North	119	40.1

The schedule was prepared using bench data reporting each ore type and waste for each 15 m bench. The flowing parameter values are carried in the detailed schedule:

- NSR \$/t.
- Cu %.
- Au g/t.
- Ag g/t.
- As ppm.
- Throughput Hours per kt
- F ppm.
- S %.
- RecCu Recovered Copper Grade %.
- RecAu Recovered Gold Grade g/t.
- SPI SAG Power Index.
- MB Modified Bond Index.
- CI Crusher Index.
- P_{80} 80% Passing microns.

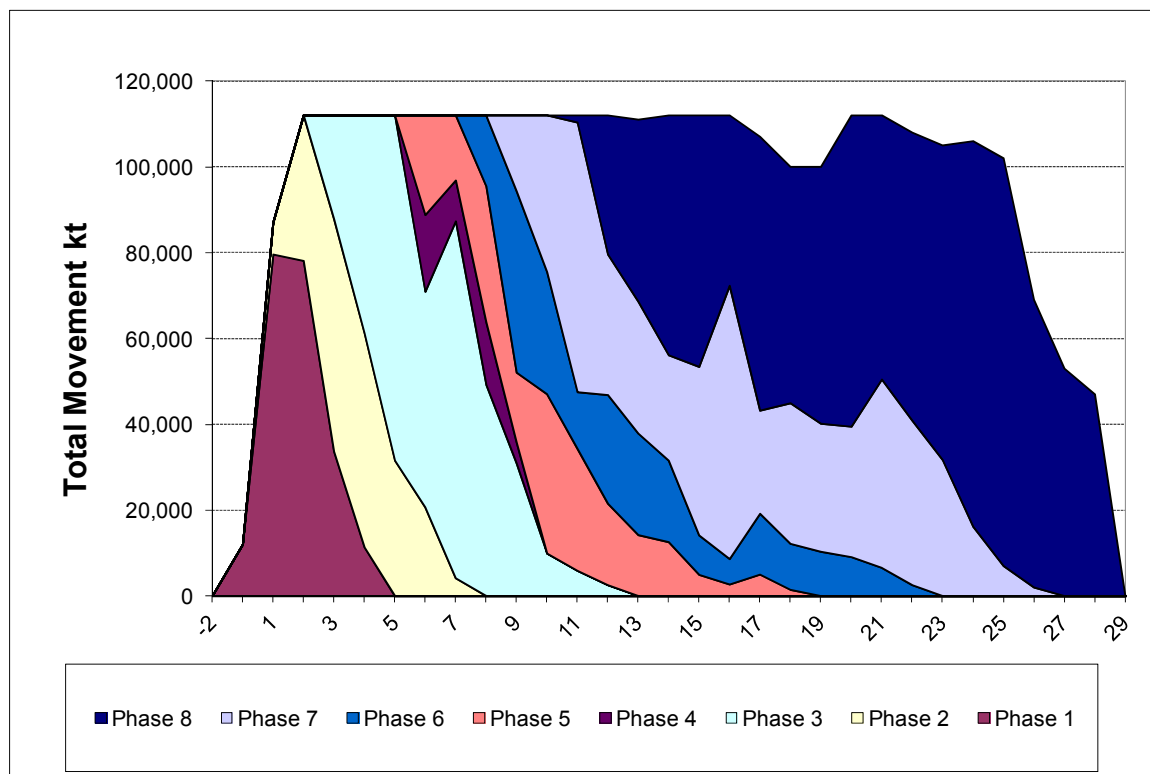
- Fe %.
- Mo ppm.
- PAF % potentially acid forming (ore 100%, waste varies).
- 27 April 2010 Base Data Template NSR.

16.5.2 Open Pit Production Schedule

Total annual open pit movement from the pits by stage, including stockpile rehandle, is shown in Figure 16.46. Stages 1 to 3 are mined from the Southwest gold zone and are the focus of initial mining. Stage 4, 5, and 6 are based on the three Central ore types to the north of the Southwest Stages 1 to 3. Stage 7 is a pushback on Stage 3, and Stage 8, or the final pit, mines the remainder of the Southwest and Central ore.

The first ten years of the schedule focus on stripping Stage 1, then 2 and 3 to extract the high-grade gold core. Stripping of Stages 7 and 8 also starts in 2016. Mining of the Central pits (Stages 4, 5 and 6) starts in 2018. Processing of the Central ore starts in 2021 and stops in 2025. Stages 7 and 8, a mix of Southwest and Central ore, are mined until the end of mine life.

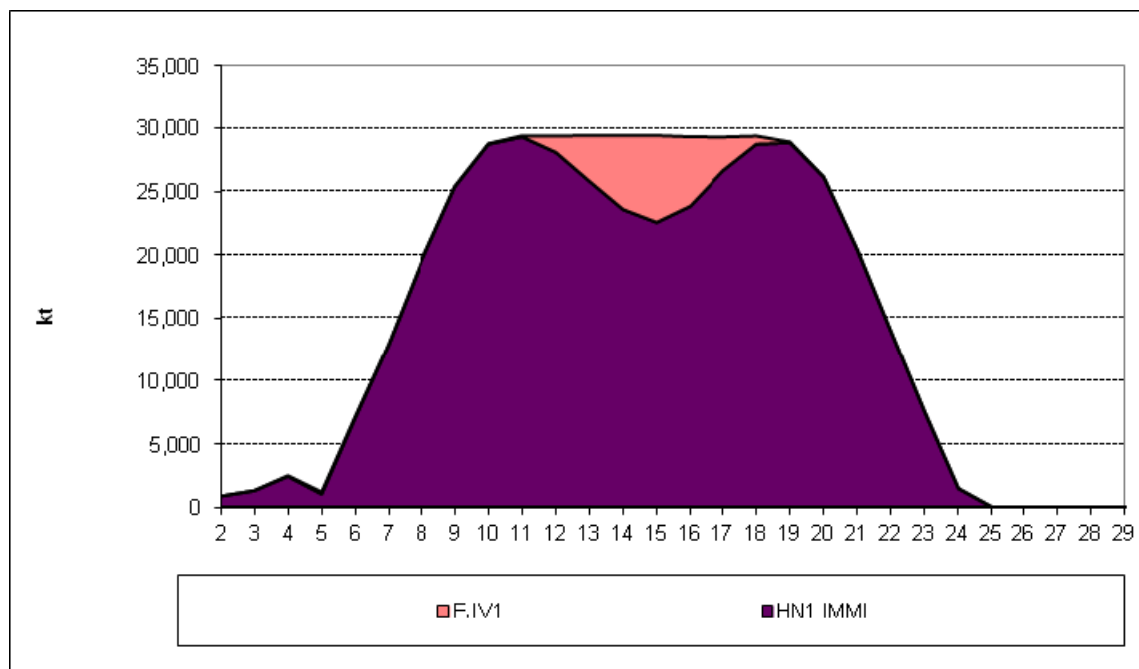
Figure 16.46 Total Open Pit Material Movement Including Stockpile Reclaim



16.5.3 Underground Production Schedule

Ore production from the Hugo North Lift 1 split between the OT LLC and EJV ore is shown in Figure 16.47.

Figure 16.47 Total Underground Material Movement



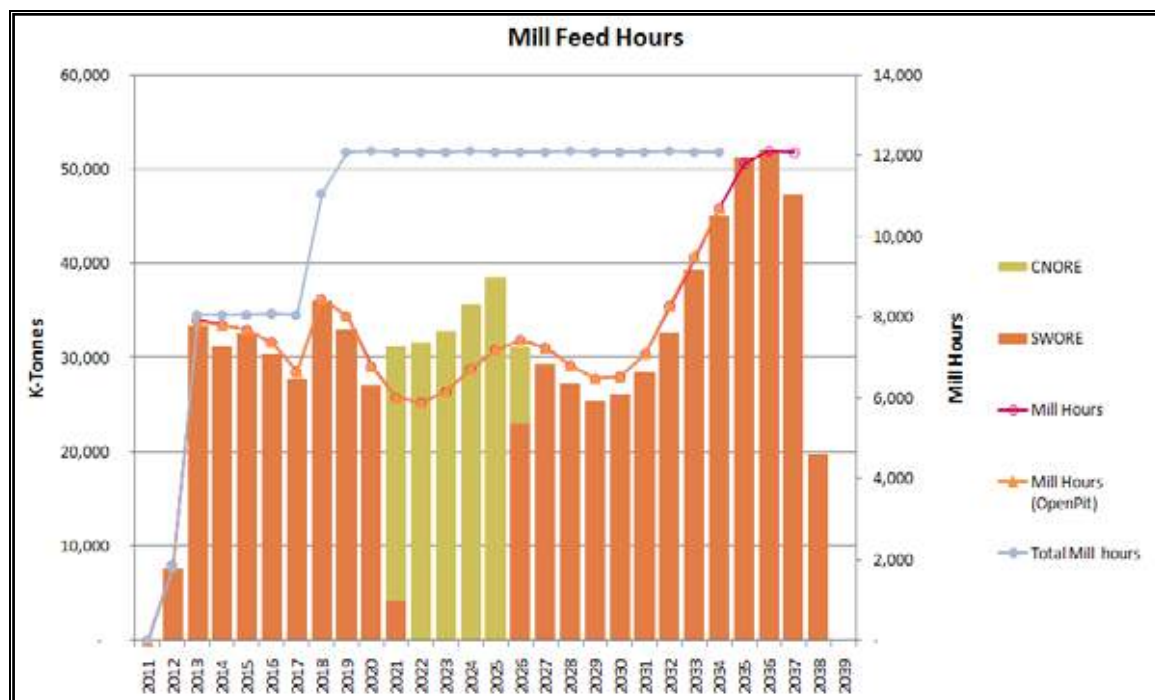
16.5.4 Processing Schedule

The processing schedule was balanced to meet the available mill hours after the underground material was processed. The ramp-up assumptions to full production are summarized in Table 16.23. Mill hours over the life of mine are shown in Figure 16.48.

Table 16.23 Process Ramp-up to Full Production

Month	Full Production (%)
Month 1	11
Month 2	34
Month 3	47
Month 4	82
Month 5	92
Month 6	100

Figure 16.48 Life-of-Mine Mill Hours



The processing schedule by metallurgical ore type and the copper and gold feed grades is shown in Figure 16.49 and total concentrate production by area is Figure 16.50. Table 16.24 details the IDOP Reserve processing schedule. There is a minor difference between the inventory used for the financial model and the Mineral Reserve. The difference is approximately 4 Mt out of 1.4 bt. This difference is due to differences in reporting from the mine designs (used for the Mineral Reserve), reporting in the production scheduling prepared by OT LLC, stockpile balances and rounding errors. The difference is not significant and if removed would not result in a material difference in the economic analysis.

The high gold grades from the open pit indicate the source of early gold production from Southwest Stages 1, 2, and 3. The decline in gold grade marks the change from Southwest to Central ores. The peaks in gold grade mark the completion of the Southwest stages. Southwest Stages 1 to 3 are centred over the gold core. Gold grades drop until the high grade in the bottom of the next stage is exposed. Annual recoveries of copper, gold, and silver are shown in Figure 16.51, Figure 16.52, and Figure 16.53.

Figure 16.49 Ore Processing and Grade by Ore Type

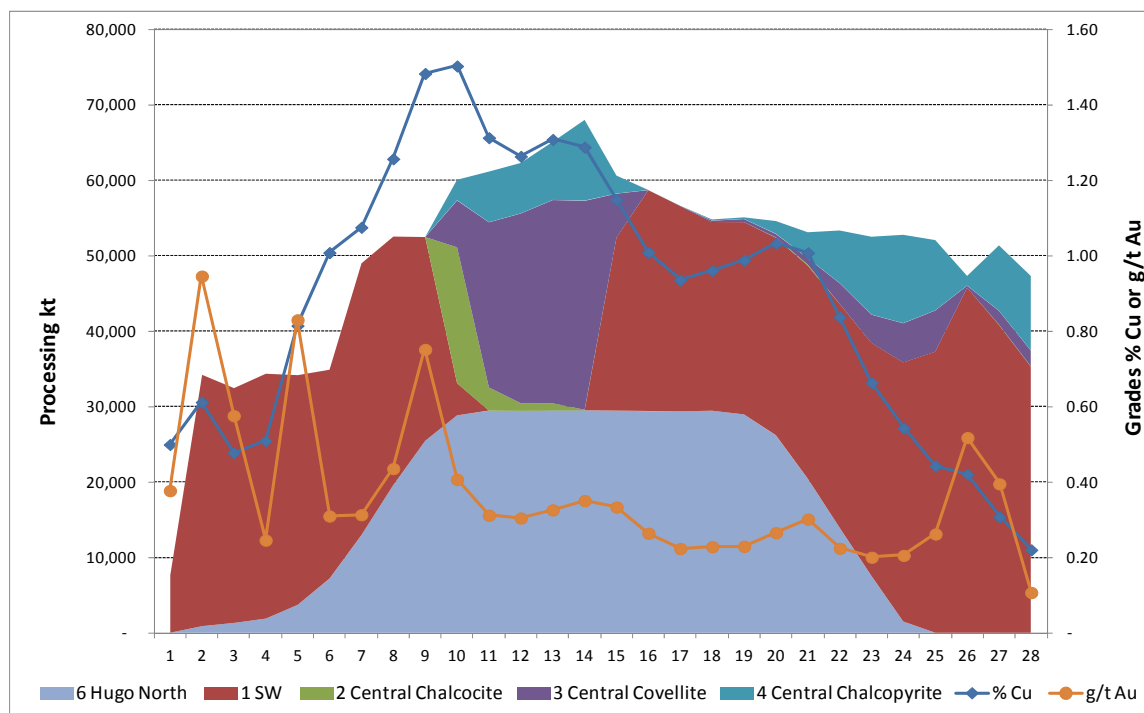


Figure 16.50 Concentrate Production by Ore Type

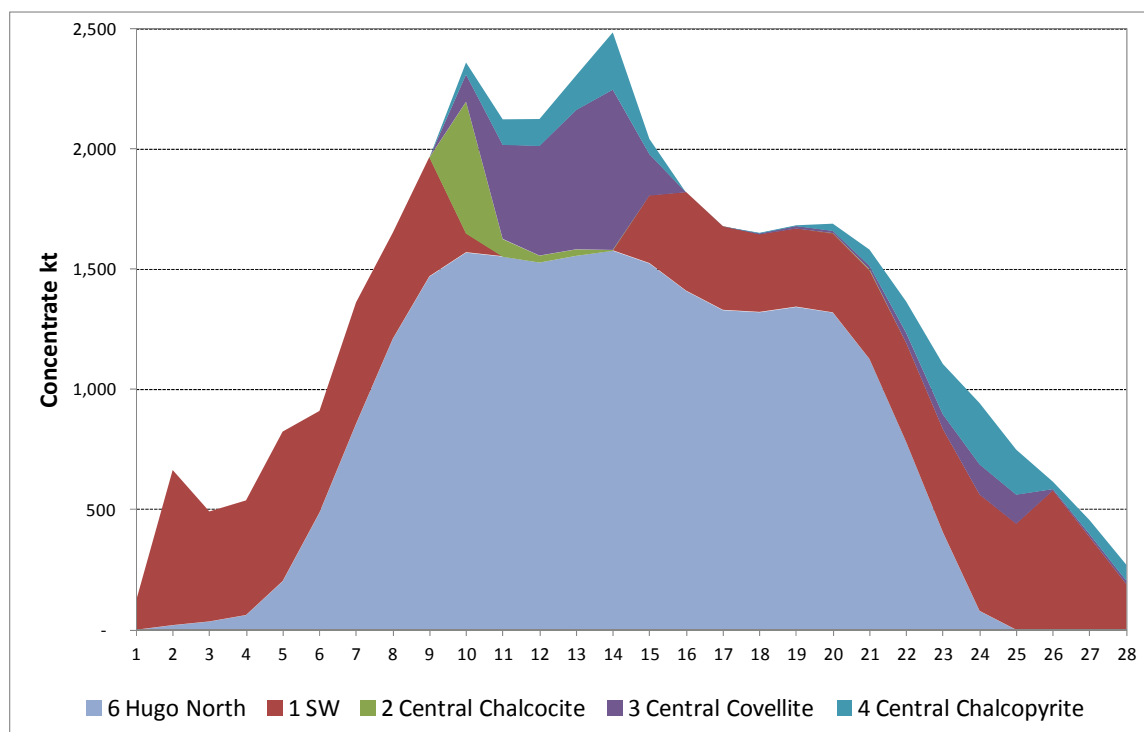


Table 16.24 Reserve Case Production Schedule Years -1 to 27

			Year												
Description	Unit	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Open Pit Ore	Mt	962	–	13	45	27	35	32	30	39	35	32	29	37	48
Open Pit Waste	Mt	1,812	12	74	67	79	76	80	82	72	76	80	79	74	64
Total Open Pit	Mt	2,774	12	87	112	106	112	112	112	112	112	112	108	112	112
U/G Ore	Mt	436	–	–	1	1	2	4	7	13	20	25	29	29	29
Total Ore Mined	Mt	1,398	–	13	46	28	37	36	37	52	55	57	58	67	77
Re-handle	Mt	119	–	–	–	6	–	–	–	–	–	–	4	–	–
Total Material including Stockpiles	Mt	2,893	12	87	112	112	112	112	112	112	112	112	112	112	112
Total Process Feed	Mt	1,398	–	8	34	32	34	34	35	49	53	52	60	61	62
Parameter Values	NSR \$/t	37.25	–	22.95	38.1	25.5	20.32	42.88	39.33	41.76	50.54	65.01	57.02	49.18	47.32
	Cu %	0.94	–	0.5	0.61	0.48	0.51	0.82	1.01	1.08	1.26	1.48	1.5	1.31	1.26
	Au g/t	0.35	–	0.38	0.95	0.58	0.25	0.83	0.31	0.31	0.44	0.75	0.41	0.31	0.31
	Ag g/t	2.09	–	1.26	1.47	1.34	1.41	1.94	2.41	2.56	2.79	3.18	2.98	2.75	2.66
	Conc kt	37,615	–	120	664	492	538	824	910	1,362	1,651	1,966	2,362	2,125	2,127
	Con Cu %	30.48	–	26.58	26.98	26.37	27.4	29.84	34.44	34.6	35.68	35.66	33.51	33.07	32.29
	Con Au g/t	10.21	–	18.43	38.46	29.9	11.97	27.42	9.64	9.2	11.09	16.19	8.39	7.26	7.21
	Con Ag g/t	63.9	–	63.81	61.84	69.77	71.29	67.07	76.87	76.68	74.39	71.93	63.24	65.34	64.26
	Con As ppm	1,365	–	291	251	354	628	383	751	686	478	546	1,965	1,809	1,847
	Con F ppm	321	–	263	302	269	233	289	289	308	308	302	312	308	321
	Copper M lb	25,276	–	70	395	286	325	542	691	1,039	1,299	1,546	1,745	1,549	1,514
	Gold koz	12,353	–	71	822	473	207	726	282	403	589	1,023	637	496	493
	Silver koz	77,282	–	246	1,321	1,103	1,233	1,776	2,248	3,358	3,948	4,546	4,802	4,465	4,394

		Year															
Description	Unit	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
Open Pit Ore	Mt	44	41	34	35	29	28	29	32	37	45	51	54	56	34	10	–
Open Pit Waste	Mt	66	71	78	74	76	72	71	79	74	63	54	50	46	21	2	–
Total Open Pit	Mt	111	112	112	109	105	100	100	112	112	108	105	104	102	55	12	–
U/G Ore	Mt	29	29	29	29	29	29	29	26	20	14	8	1	–	–	–	–
Total Ore Mined	Mt	74	70	64	64	58	57	58	59	58	59	58	55	56	34	10	–
Re-handle	Mt	–	–	–	3	2	–	–	–	–	–	–	2	–	14	41	47
Total Material including Stockpiles	Mt	111	112	112	112	107	100	100	112	112	108	105	106	102	69	53	47
Total Process Feed	Mt	65	68	61	59	57	55	55	55	53	53	52	53	52	47	51	47
Parameter Values	NSR \$/t	49.14	48.8	44.27	38.32	34.78	35.69	36.55	38.8	38.62	31.18	24.71	20.56	17.89	22.54	16.15	7.15
	Cu %	1.31	1.29	1.15	1.01	0.94	0.96	0.99	1.03	1.01	0.84	0.66	0.54	0.44	0.42	0.31	0.22
	Au g/t	0.33	0.35	0.33	0.27	0.22	0.23	0.23	0.27	0.3	0.23	0.2	0.21	0.26	0.52	0.4	0.11
	Ag g/t	2.72	2.64	2.54	2.35	2.24	2.21	2.09	2.06	2.07	1.71	1.37	1.14	1.02	1.31	1.16	0.95
	Conc kt	2,308	2,487	2,044	1,821	1,679	1,654	1,685	1,691	1,584	1,368	1,107	945	752	619	459	273
	Con Cu %	32.04	30.67	30.22	29.04	28.06	28.17	28.57	29.36	29.48	28.19	26.6	25.09	24.52	26.15	25.12	24.16
	Con Au g/t	7.38	7.7	7.98	6.89	6.06	5.98	5.86	6.65	7.81	6.77	7.33	8.7	13.33	29.86	32.85	10.87
	Con Ag g/t	63.37	59.75	62.89	63.61	62.96	60.79	56.78	55.21	56.87	54.16	51.87	49.89	53.68	77.15	92.85	108.73
	Con As ppm	1,766	2,067	1,425	1,358	1,472	1,586	1,698	1,480	1,174	1,197	1,346	1,943	1,865	430	1,058	1,483
	Con F ppm	332	341	350	365	375	363	340	344	369	367	332	277	244	215	157	81
	Copper M lb	1,630	1,682	1,361	1,166	1,039	1,027	1,061	1,095	1,029	850	649	523	406	357	254	145
	Gold koz	547	615	524	403	327	318	318	362	398	298	261	265	322	594	485	95
	Silver koz	4,702	4,777	4,132	3,724	3,399	3,232	3,075	3,002	2,895	2,382	1,847	1,516	1,298	1,535	1,370	955

Note: Minor figure differences may occur due to rounding errors.

Figure 16.51 IDOP Reserve Case Copper Production

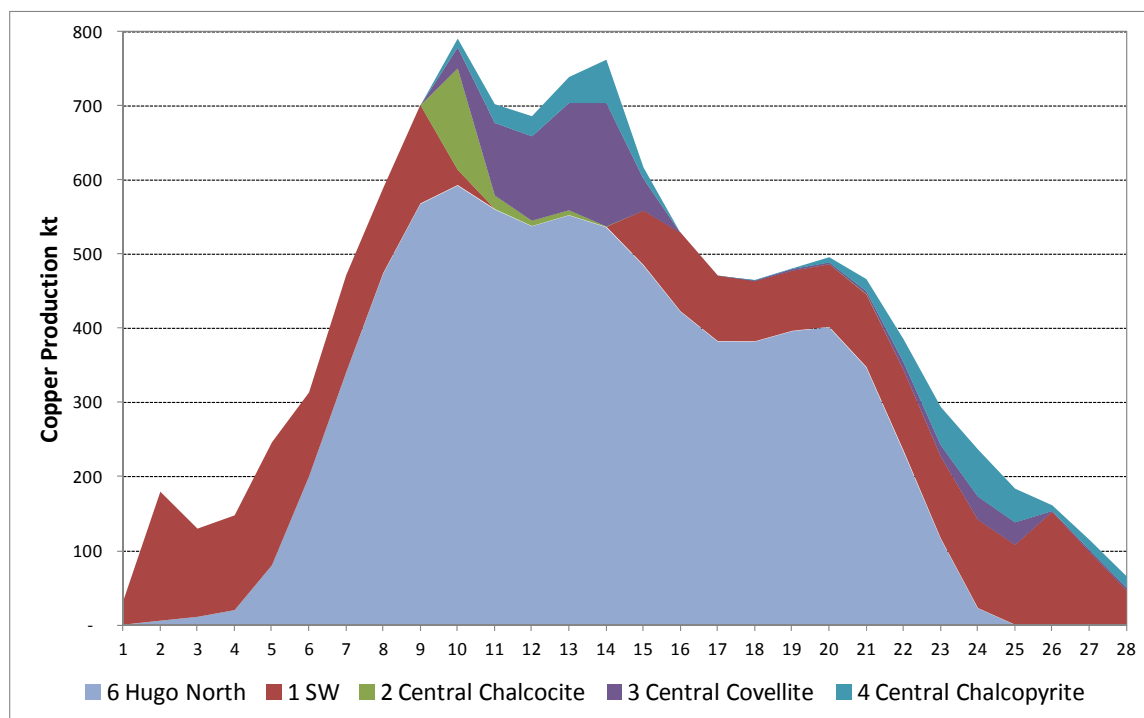


Figure 16.52 IDOP Reserve Case Gold Production

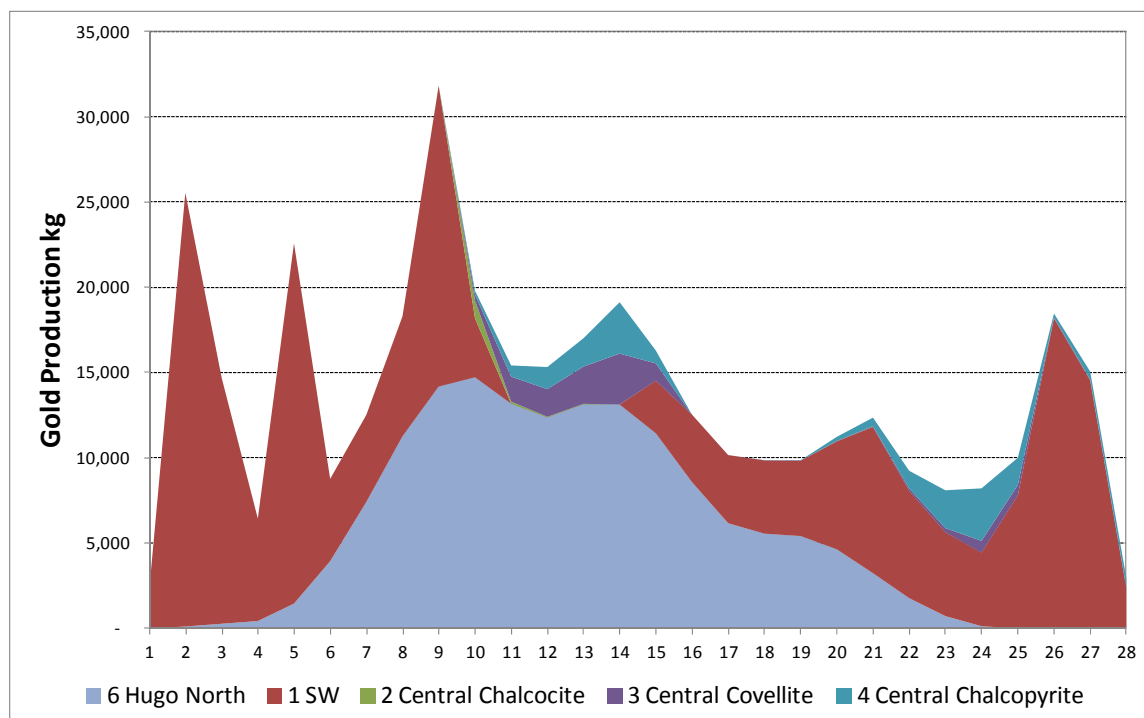
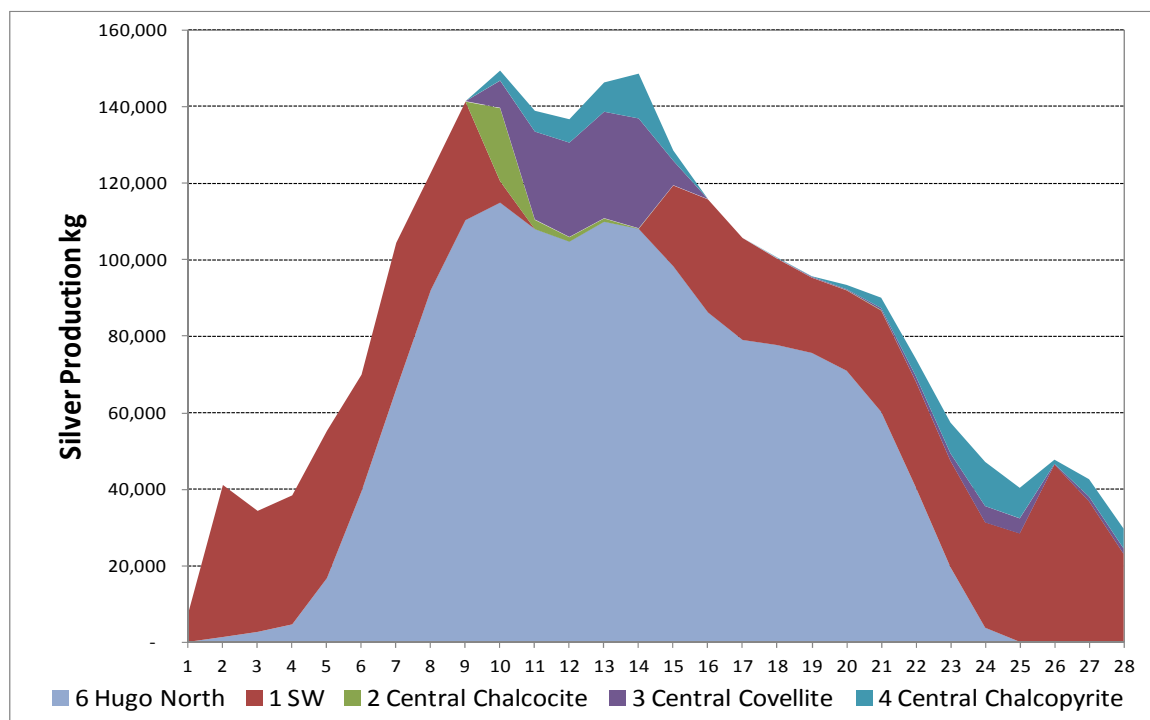


Figure 16.53 Total Silver Recovery



17 RECOVERY METHODS

17.1 Processing

17.1.1 Introduction

The Oyu Tolgoi processing plant will commence operation with two grinding lines and two flotation lines. This is sufficient for processing ore arising from the Southwest open pit. However, when Hugo North block cave Lift 1 is developed and comes on-line a third grinding line is necessary to process the additional ore.

As the copper mineralogy changes so does the concentrate grade that can be generated and the recoveries that are expected. This complexity has led to a set of equations describing grades and recoveries for the various ore types and for copper, gold and silver.

These equations have been used as part of the mine planning process and are consequently inherent in the IDOP ore delivery schedule. Equipment selection has been based around the design criteria and the duty as determined through the ore delivery schedule.

17.1.2 IDP10 Metallurgical Parameters

The metallurgical parameters used for IDP10 were issued in Base Data Template 19. However, in preparing IDOP, the predictive formulae were revised and there revisions are now incorporated in Base Data Template 27. The relevant metallurgical data and formulae are shown in Table 17.1 to Table 17.4. These parameters were used for the IDOP10 Mineral Reserve estimate.

The parameters used for Hugo North were applied to all Hugo North including the Entrée (Shivee Tolgoi) ore. Although only limited testwork has been carried out on the EJV area it is considered that the results indicate the Entrée area is similar to the rest of Hugo North.

The throughput rate algorithm is as used in IDP10 and was developed by SGS from regression analysis of CEET simulation runs for 30,000 SW blocks over Years 1 to 30 and the SGS database of projects. This formula was applied to all the blocks in the mining model and used for production scheduling.

The formulae in Table 17.1 to Table 17.4 are as used in IDP10 and have been incorporated with updated operating cost estimates in Base Data Template 27. The throughput algorithms continue to be used for mine scheduling in the IDOP (including the MP01 mine plan).

Table 17.1 Base Data Template 27 Copper Recovery

All Ores			
$a * [(b * CuF) / (1 + b * CuF)] * [1 - \exp(-b * CuF)]$			
SW and CENTRAL Chpy a = 98 b = 12.2	HN and HS a = 95 b = 15	CENTRAL Cov a = 85 b = 15	CENTRAL Cct a = 82 b = 15

Table 17.2 Base Data Template 27 Gold Recovery

All Ores
$9.8 + 0.8 * (Cu \text{ Recovery})$

Table 17.3 Base Data Template 27 Silver Recovery

All Ores
$13 + 0.8 * (Cu \text{ Recovery})$

Table 17.4 Base Data Template 27 Copper in Concentrate

SW and CENTRAL Chpy	HN and HS	CENTRAL Cov and Cct
$-3.6 * (Cu:SF)^2 + (12.8 * Cu:SF) + 22.5$	$2.9 * (CuF) + (11.4 * Cu:SF) + 15.3$	25

The throughput rate algorithm shown in Table 17.5 was developed by SGS from these relationships and the SGS database of projects. This formula was applied to all the blocks in the mining model and used for production scheduling.

Table 17.5 Plant Throughput Rates

$P_{80} = 113 * Ci^{0.26} * SPI^{-0.60} * BM^{0.88}$
max P_{80} 220 μm
$t/h \text{ (instantaneous)} = 29320 * Ci^{0.19} * SPI^{-0.36} * BM^{-0.24}$
max throughput = 5,500 t/h

17.1.3 Metallurgical Predictions

The metallurgical predictions for throughput, metal recovery and concentrate grade have been modeled by experts in their field and validated through peer review.

17.1.3.1 Throughput Modeling

60" x 113" FFE Primary Gyratory Crusher – FFE Model

The 60" x 113" Primary Crusher throughput estimate was based on a FLSmidth model and is provided in their September 2006 Performance Warranty. The ROM blasting requirement and primary crusher feed size was taken from Scott Mine Consulting Services – Blasting Requirements for the Oyu Tolgoi Open Pits 15NOV04 report.

The crusher work index was based on laboratory scale testwork conducted at Lakefield Research and AMMTEC.

- $t/h = 4,880$ (with crusher work index of 19.9 kWh/t and F80,500 mm, throughput range 4,250 to 5,350 t/h). Fluor design criteria = 5,293 t/h.
- $P_{80} = 152$ mm (range 140–152 mm).

SAG, Pebble Crusher and Ball Mills – SGS CEET Model

SGS's CEET model was used to predict throughput based on laboratory scale ore characterization carried out at Lakefield Research and AMMTEC. The CEET model predictions were verified by JKSimMet modeling and pilot plant testwork. The geostatistical block model was populated with the hardness values to predict throughput by ore type and mining period. The throughput and grind algorithms applied to the block model are shown above in Table 17.5.

AMEC Minproc concluded that the SGS predictions were justified and used their predictions in the IDOP reserve model.

17.1.3.2 Flotation Modeling

Optimum Primary Grind

In IDP05, primary grind size determination analysis was carried out for each of the production-period composites from the Southwest, Hugo South, and Hugo North deposits and the ore-type composites from the Central deposit. AMEC Minproc concluded that the IDP05 predictions were appropriate. In 2007, Aminpro validated the primary grind sizes of the existing ore types, and made additional predictions for the new HN-SLC and Entrée reserves. The criterion for P_{80} selection was the size that resulted in the highest project net present value (NPV).

The plant design is based on the optimum grind of Southwest and Hugo North. With the power requirement set to achieve an optimum grind, throughput-grind trade-offs were conducted through optimizing production scheduling.

Table 17.6 Optimum Primary Grind Size for Each Ore Type (P_{80})

Deposit/Composite	IDP05	Aminpro 2007
Southwest	180	180
Hugo South	150	–
Hugo North	140	116
Central (average)	138	179
SLC (average)	–	125
Entrée (average)	–	133

Another review of optimum grinds was conducted by OT LLC and AMEC in 2011 using updated metal prices and head grades.

The primary source document for capacity prediction for the current grinding circuit is SGS Mineral Services' "Comminution Section for Oyu Tolgoi June 2007 IDP," which states that the grinding circuits will be SAG-mill limited for 75% of the time with Southwest ore. SGS predicted that capacity could be increased by another 4,000 t/d in this period by increasing closing screen aperture from the current design of 13 mm to 17 mm, and by another 3,500 t/d by increasing ball charge from 15% to 17%. The net 7,000 t/d increase, if achieved as modelled, would be sufficient to meet the 96,000 t/d average in Years 1 to 3 after the initial ramp-up.

In the current study, it is considered unrealistic to expect a new plant with 80% to 90% national hire of generally inexperienced personnel to operate so closely to the design limits of a grinding circuit immediately after the initial ramp-up. The mill shell is structurally capable of accepting a 20% ball charge, but measurements are imprecise, and caution is recommended until media wear and liner wear are predictable and instrumentation is known to be reliable by field calibration on each shutdown and grind-out. Despite the advantages in terms of NPV and IRR, it is unusual to promote a base case in which a plant is run over the nominal capacity so soon after start-up.

The Phase 2 expansion for Hugo North is modelled as a simple 50% power expansion from a nominal 96,080 t/d with 9% HN to 160,000 t/d of blend with 85,000 t/d (53%) of HN. A 50% increase in applied power yields a 66% increase in average throughput when combined with softer underground ore. The highest capacity years are in 2025 (68 Mt), seven years after start-up of the third line on 1 March 2018. At this time, the soft Central chalcocite and covellite ores, with the contribution of 43% medium-soft Hugo North, allows for peak high milling rates at an average flotation feed sizing of 207 μm . Tonnages are constrained to 8,750 t/h for three lines and 5,500 t/h for two lines (as per Table 17.5) for the overall throughput in the mine scheduling process. With this constraint imposed, 200,000 t/d will be milled in 2025 only, equivalent to 130,000 t/d on a two-line basis.

At present, however, the design criteria have a design volumetric limit of 110,000 t/d for the two-line concentrator. Because the SAG mills are not power-limited on Central ore, it is expected that this tonnage can be processed in 2025 after 13 years of operating experience on the Phase 1 circuits and 7 years of experience with the Phase 2 circuits. When the available SAG power is not a limitation, it is possible to use the upset flow volumetric design point of 60,000 t/d for a single SAG mill and 180,000 t/d for the three line circuit post-2018. The volumetric flows in grinding and flotation in 2025 fall within design limits when ball mill cyclone feed and overflow densities are increased by about 3% solids so that the circulating loads are decreased and the grind is coarsened from 180 to 200-210 μm .

AMMTEC estimated the lost operating margin by coarsening a 180 μm grind to 200 μm at \$0.12/t for Central covellite ore and \$0.50/t for Central chalcocite ore. It will clearly be cash flow positive to mill the additional tonnage available from the mine schedule. Since the capital costs for Phase 1 are effectively sunk, these losses will be taken into account when considering overall project NPV and the overall capacity of Phases 1 and 2 in the feasibility study. A further round of primary grind optimization will be undertaken on Hugo North, Southwest, and Central ores based on updated economic parameters from the control estimate and the latest project metal price forecasts.

Optimum Regrind Size

A metallurgical prediction of the optimum regrind size was carried out for IDP05, by AAJV and later data analysis was carried out by AMEC Minproc and Aminpro. The grind size was further reviewed in 2011 by OT LLC. The optimum regrind size is based on a recovery-concentrate grade trade-off, and the production of a saleable concentrate including impurities.

Table 17.7 Optimum Regrind Size ($P_{80}\mu\text{m}$)

Deposit/Composite	IDP05*	AMEC Minproc	Aminpro 2007**	OT LLC
Southwest	25	25	45	30
Central	25	–	–	30
Hugo North	25	30	45	30
Hugo South	25	–	–	–

* Production composite sample of Southwest, Central, Hugo North and Hugo South.

**Production composite modeling of 2008 production schedules.

Aminpro's prediction of the number of regrind mills for the 96,000 tpd Phase I plant is four to six VM 1500 HP mills and this will achieve Aminpro's target regrind sizes. AMEC Minproc's prediction is eight mills based on the finer regrind sizes. AMEC Minproc has estimated that four regrind mills will be capable of achieving average regrind P_{80} values of 37 to 40 μm .

In August, Crescent Technologies Inc., the Owner's project management consultant for the concentrator design, requested Owner's input to the design criteria. The expansion study team carried out a review of past work and commissioned additional QEM. SCAN work on Southwest and Hugo North composites. It was concluded that a 30 μm regrind would initially require six mills and would yield a payback of one year, based on improved grade/recovery response and reduced fluorine penalties. An analysis of past variability testwork also indicated wide scatter around the mean value for fluorine recovery used in the mine model to predict fluorine in concentrate at a given regrind level. Regrind to 30 μm would yield a 20% reduction in mean fluorine content relative to a 40 μm regrind and would reduce the concentrate storage and blending requirements needed to avoid occasional rejections on shipments above 1,000 ppm. The work is reported in an OT LLC memorandum, with accompanying spreadsheets and Powerpoint presentation ("Review of the optimum regrind level and the initial number of regrind mills to be installed for processing of Oyu Tolgoi's Southwest Orebody," 14 June 2010).

While the Phase 1 regrind power requirement is based on the Owner's team prediction, space for an additional five regrind mills is provided in the layout. This is likely to be enough to accommodate all the regrind required in the Phase 2 expansion case but is an area of ongoing metallurgical testwork on the Hugo north composites now being tested.

Fluor has included the capital cost of six mills in the October 2010 control estimate for construction.

17.1.4 Flow Sheet Development

Flow sheet development began with IDP05 for a 70,000 tpd concentrator capable of producing 1,900 tpd of concentrate. IDP05 also included a Phase II 85,000 tpd concentrator. In 2008, the plant's processing rate was increased to 96,000 tpd with a concentrate production rate of 2,850 tpd. A Phase II plant expansion to a 160,000 tpd is planned in production Year 6.

The current Phase I 96,000 tpd plant will further be described as the IDOP flow sheet.

17.1.4.1 Comminution

IDP05

Laboratory-scale hardness testwork was conducted at MinnovEX and AMMTEC. In addition, a SAG pilot-plant test program was carried out at SGS-Lakefield. Circuit simulation work was conducted with CEET and JKSimmet. Details of the testwork and modeling are provided in IDP05. Figure 17.1 provides the overall IDP05 comminution testwork plan.

Figure 17.1 Overall IDP05 Comminution Testwork Plan

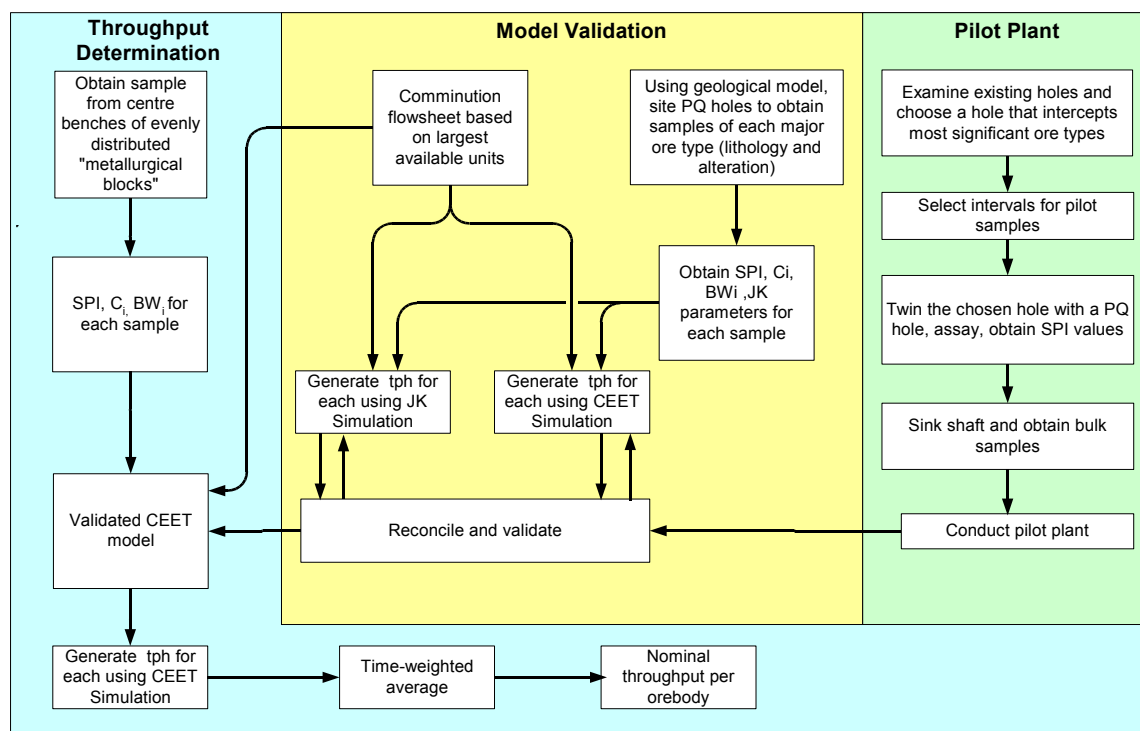


Figure 17.2 and Figure 17.3 are a summary of the ore hardness frequency distributions for Southwest, Hugo North, Central, and Hugo South.

Figure 17.2 Cumulative Frequency Distribution of SPI Values Over the Southwest, Central and Hugo Deposits(2003 to 2005 Testwork Program)

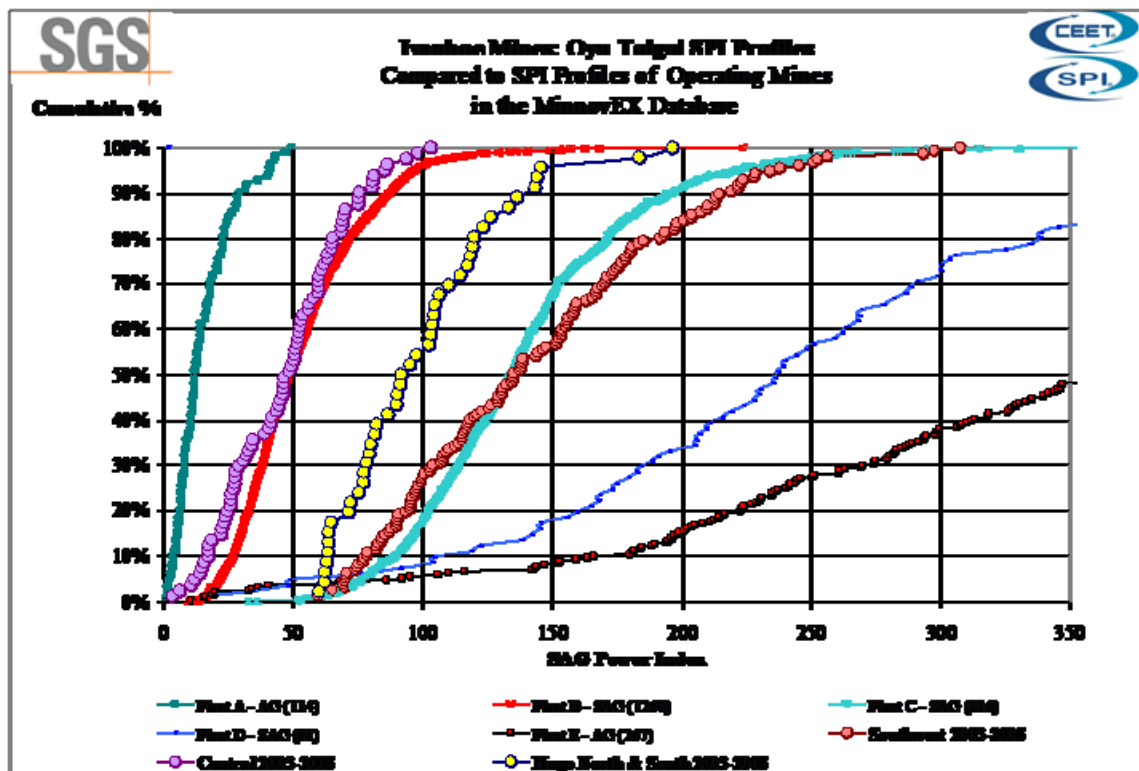
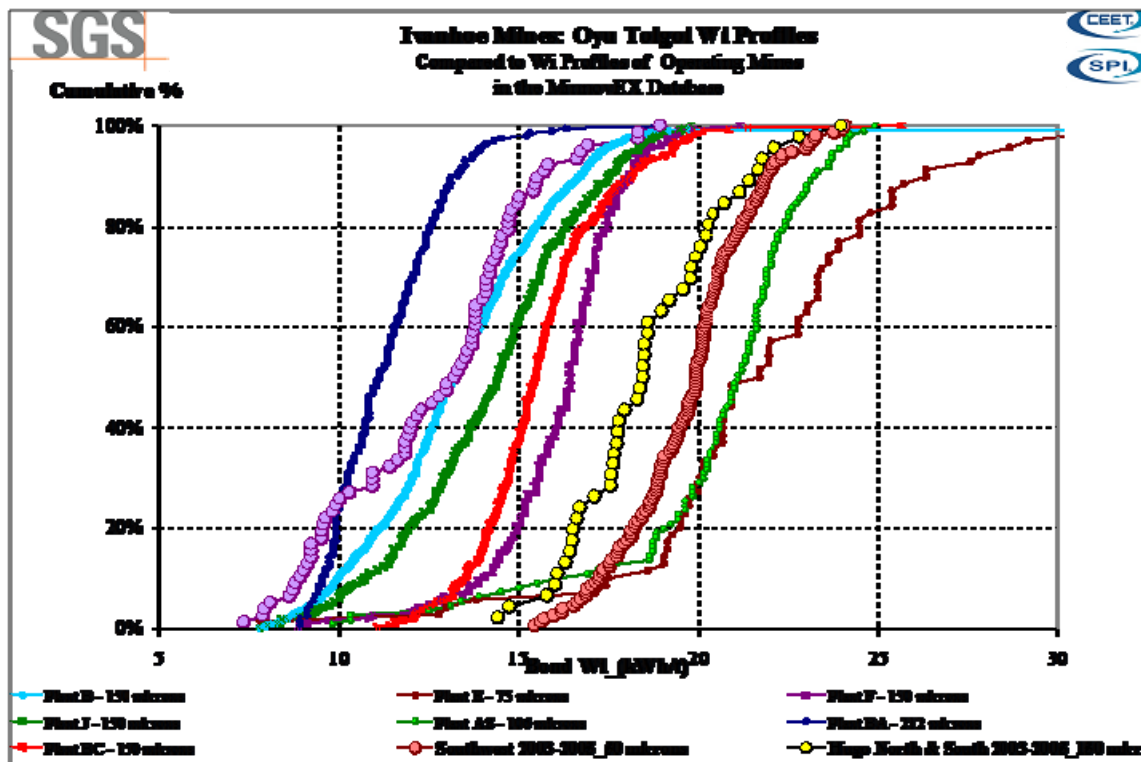
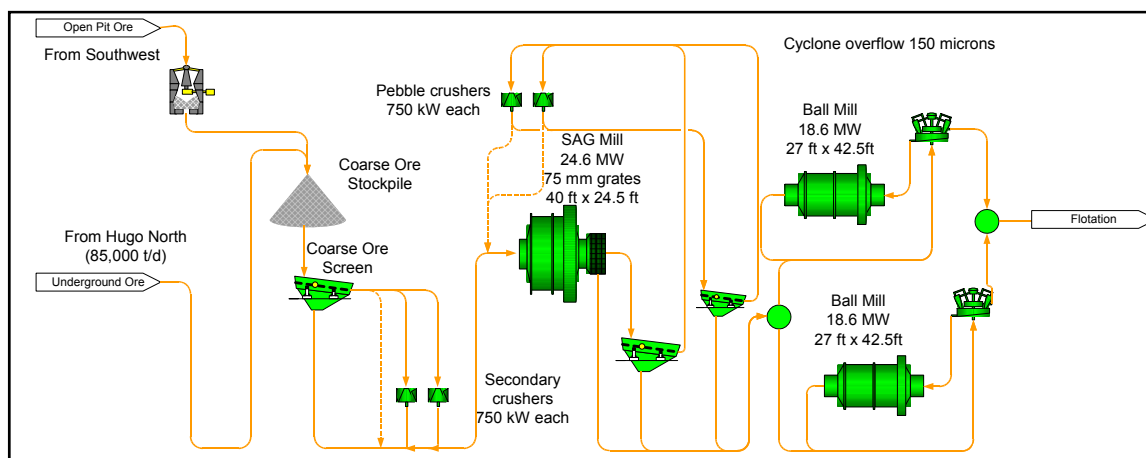


Figure 17.3 Cumulative Frequency Distribution of BWI Values Over the Southwest, Central and Hugo Deposits(2003 to 2005 Testwork Program)



After running the CEET model with a number of circuit alternatives, the optimum IDP05 grinding flow sheet was based on a single SAG line utilizing the largest commercially proven mills, with secondary and pebble crushers, and utilized the maximum installed power. Subsequent testwork and simulation was performed to validate the choice and predict the throughput at the desired primary grind for the different ore deposits. Figure 17.4 provides the IDP05 flow sheet.

Figure 17.4 IDP05 Grinding Circuit for Oyu Tolgoi Project



Production Scheduling and Geostatistics on Ore Hardness Data

SGS Minerals Services conducted a geostatistical study of the grindability data from the Southern area of the Project comprising the Projected open pit areas known as Southwest, South, Central and the Wedge (between Southwest and South). The geostatistical study was conducted by Michel Dagbert of Geostats International in conjunction with Ivanhoe geologist, Stephen Torr, who had already conducted his own study. The hardness data was distributed to each mine block and annual mining periods according to the details of the geological and mine plans.

For the Southern Open Pit area, 219 useable samples that were tested for SPI, Bond Work Index and Crusher Index measurements were available for the geostatistics study. These 219 samples were used to estimate 59,661 ore reserve blocks, representing a total tonnage of about 930 Mt. It was found that the grindability values could be attributed to most blocks with reasonable precision.

The geostatistical study determined hardness SPI and BWi values for each year of processing the ore. In this manner, the throughput and grind was determined by production period.

IDOP Flow Sheet – 96,000 tpd Concentrator

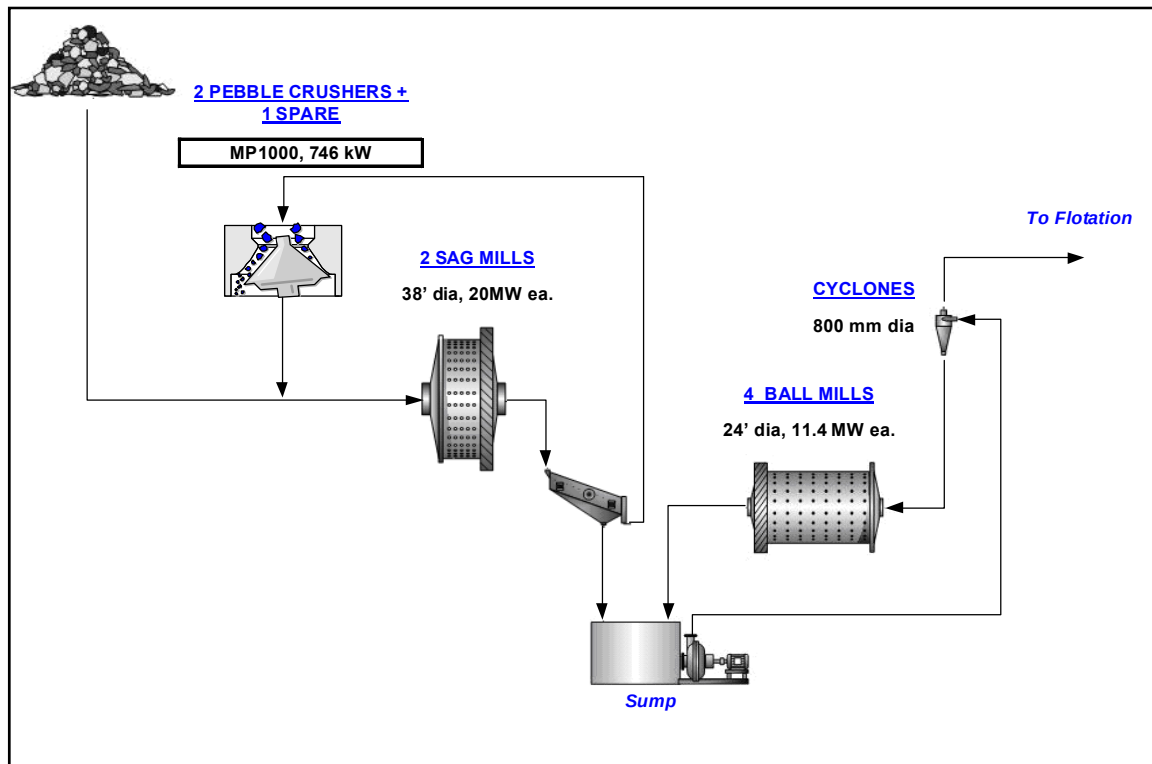
Development of the Phase I and Phase II 2009 flow sheet was conducted with the same laboratory ore hardness data from Southwest and Hugo North.

In November 2005, the comminution circuit was changed to a conventional SABC and documented in Fluor's 2005 Review and Strategic Plan (RSP); Technical Decision Memorandum, TDM003. This change led to the removal of secondary crushing and increased the 40' SAG mill motor from 24.6 MW to 28 MW. The decision reduced capital cost and increased revenue. It was also felt that the conventional SABC was a lower production risk to the less conventional secondary crushing SAG circuit. This decision was made after IDP05, and maintained the plant capacity at 70,000 tpd.

In October 2006, a decision was made to increase the plant capacity from 70,000 tpd to 96,000 tpd. The increase in plant capacity led to a second change in circuit configuration documented in Fluor's PCN 21. The circuit remained as a SABC but changed from a single grinding line to a dual grinding line. This change increased the throughput rate and capital cost, and also the revenue and economic benefit. Additionally, it was felt that a second grinding line would also reduce the production risk through increased equipment redundancy.

The above changes were modelled and again optimized by running CEET. The new optimum flow sheet consists of two SAG mills in parallel, as shown in Figure 17.5. The two SAG mills are each 38 ft diameter and most of the simulations conducted to date have assumed a 21 ft EGL mill drawing peak power of 18.2 MW and maximum sustainable power (over periods of a few hours on hard ore) of 17.25 MW. (The SAG mills are assumed to have 20 MW motors with a service factor of 1.0.) The four ball mills have been assumed to draw a total of 43.3 MW (10.83 MW each) with two 5.7 MW motors per mill (11.4 MW total installed power per mill, 45.6 MW installed power for four mills) with motor service factors of 1.0.

Figure 17.5 IDOP Grinding Circuit Flow Sheet



The Phase II flow sheet increases the grinding lines from two to three and is an identical SABC configuration of Phase I.

Predictions for Hugo North, Hugo South, and Entrée

The results from the geostatistical distribution of the grindability data from the Southern open pit areas and the CEET block output for each of the 59,661 ore blocks were used to establish useful correlations between SPI, Ci and BWI and predicted tph and P_{80} for the specific 2-line grinding circuit option, as described above.

For initial prediction purposes, these correlations present a rapid means of tph and P_{80} estimation for Hugo North (SLC S1, S2 and S3), Hugo South and Entrée samples for production forecasting purposes, prior to additional testwork and geostatistics being completed on these deposits.

Again, the throughput and grind by period were determined with the geostatistical block model populated with the ore characterization hardness data.

IDOP Equipment Requirements

A comparison of the IDP05 and IDOP flow sheet equipment is provided in Table 17.8.

Table 17.8 Flow Sheet Equipment Comparison

Flow Sheet	IDP05		IDOP	
	Phase I	Phase II	Phase I	Phase II
Ore, '000 t/d	70	85	96	160
Feed Grade, %Cu	0.67	2.28	0.66-2.28	0.92
Primary Crusher	1	1	1	2
	1.52 m x 2.26 m gyratory, 600 kW		1.52 m x 2.87 m gyratory, 746 kW	New Underground crushers but not considered in Process CAPEX
Secondary Crusher	2	2	-	-
	MP1000; 746 kW		-	
SAG Mill	1	1	2	3
	12.2 m dia x 7.47; 24.6 MW		11.6 m dia x 6.9 m, 20 MW	
Pebble Crusher	2	2	3	5
	MP1000; 746 kW		MP1000; 746 kW	
Ball Mill	2	2	4	6
	8.2 m dia x 13.1 m, 18.6 MW		7.3 m dia x 11 m, 11.4 MW	
Rougher Flotation	2 x 7	3 X 7	4 x 8	6 x 8
	160 m ³ Tank Cells		160 m ³ Tank Cells	
Regrind Mill	4	8	6 - 8	12
	Vertimill VTM 1500, 1119 kW		Vertimill VTM 1500, 1119 kW	
Cleaner Flotation	2 x 4	3 x 4	3 x 4	6 - 6 x 4
	160 m ³ Tank Cell		160 m ³ Tank Cell	
Cleaner Scavenger Flotation	2 x 4	3 x 4	3 x 4	6 - 6 x 4
	160 m ³ Tank Cell		160 m ³ Tank Cell	
Column Flotation	4	8	4-8	12
	4.5m dia x 14m Column Cell		5.5 m dia x 16m Column Cell	
Con. Thickeners	2	3	2	4
	20 m dia High Rate		20 m dia High Rate	
Concentrate Filters	2	3	2-3	4
	144 m ² Pressure Filter		144 m ² Pressure Filter	
Tails Thickeners	2	3	2	3
	125 m dia Conventional		85 m dia High Compression	

17.1.4.2 Flotation

IDP05

The IDP05 flotation flow sheet was developed based on the metallurgical test program conducted between 2001-2005 at SGS-Lakefield and AMMTEC. The results from locked-cycle tests were analyzed to derive kinetic data, and modified flotation tests (MFT) were performed on the seven production-period composites from the Southwest deposit for MinnovEX's FLEET simulator. MinnovEX also performed rougher-cleaner tests, based on a single-stage rougher and three-stage cleaner circuit, and used FLEET to determine cleaner kinetics. Conventional and Column Pilot Scale tests on Southwest and Hugo North verified the FLEET results.

- Rougher flotation – three banks of seven 160 m³ cells, 26 minute residence time.
- Regrind – (4) VTM 1500, P₈₀ 25 µm.
- First cleaner – two banks of four 160 m³ cells, 28 minute residence time.
- Cleaner scavenger – two banks of four, 160 m³ cells, 34 minute residence time.
- Recleaner columns – four columns, each 4.5 m diameter, 22 minute residence time.

Details of the metallurgical testing program and FLEET modeling are provided in IDP05 Appendix 10.A.

2009 Startup Flow Sheet

Amelunxen Mineral Processing (Aminpro) also reviewed and validated the MinnovEX flow sheet and equipment. Aminpro established flotation kinetics from the original AMMTEC locked-cycle testwork on the seven production-period composites, and the additional 2006 cleaner tests conducted at SGS. The Aminpro-Flot Simplex (AFS) model was based on rougher and cleaner kinetics derived from standard laboratory testwork and was calibrated to laboratory locked cycle testwork (LCT). The resultant kinetics indicated lower rate constants than those developed by MinnovEX. Slower kinetics lead to an increase in the size of the flotation circuit.

The AFS model was set up to treat a nominal plant feed of 100,000 t/d. The number of rows and cells per row were determined through trials observing the performance of the circuit and measuring the outcome by estimating the Net Smelter Return on a per-ton of feed basis. This approach led to the formulation of a circuit that operated at the highest economic returns.

Aminpro recommended an identical flow sheet to IDP05 with the following equipment:

- Three rows of eight 160 m³ roughers.
- Four regrind vertimills, 45 µm grind.
- Two rows of four 160 m³ first cleaner cells.
- Two rows of four 160 m³ cleaner-scavenger cells.
- Four 5.5 m diameter x 16 m cleaner columns.

Aminpro confirmed that the flow sheet proposed by MinnovEX is both adequate and robust and will handle large fluctuations in ore grades and tonnages. The AFS model was used for development of the 2009 flotation flow sheet, equipment requirements, and mass balance, and also to validate the metallurgical predictions and production schedule.

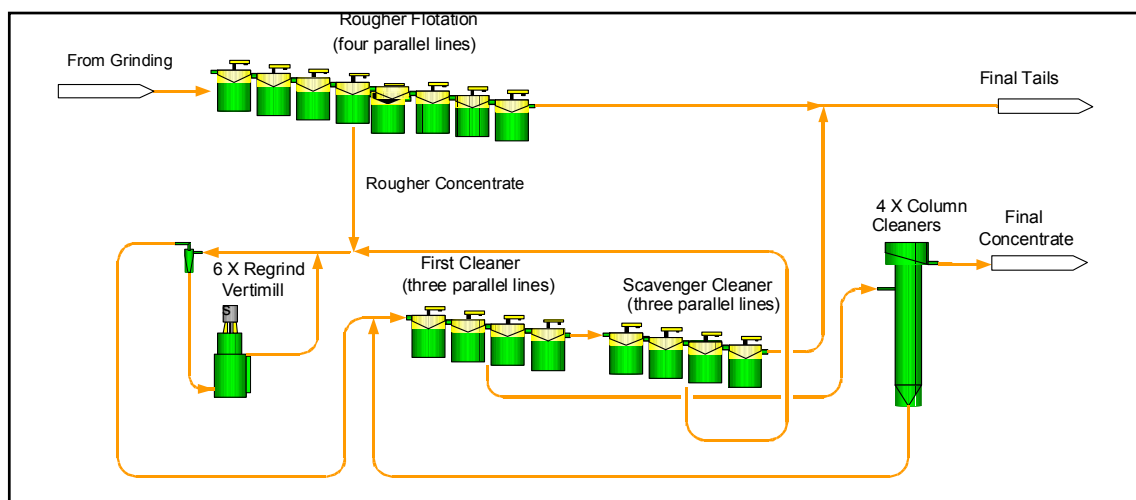
After Fluor's design review of the 2009 flow sheet, an additional row of roughers and an additional row of cleaners and cleaner scavengers was added to the flow sheet for hydraulic constraint reasons. The 2009 flow sheet major equipment is provided below, and the 2009 flow sheet is provided in Figure 17.6:

- Rougher flotation – four banks of eight 160 m³ cells, 25 minute residence time.
- Regrind – four VTM 1500, 45 µm regrind.
- First cleaner – three banks of four 160 m³ cells, 12 minute residence time.
- Cleaner scavenger – three banks of four, 160 m³ cells, 14 minute residence time.
- Recleaner columns – four columns, each 5.5 m diameter.

IDOP Start-up Flowsheet

The IDOP flowsheet is shown in Figure 17.6 and is the same as the 2009 circuit but with the addition of two more VTM 1500 regrind mills to achieve a 30 µm sizing for new feed to the cleaning circuits, as described in Section 17.1.3.2. The additional liberation minimizes the potential for rejection of concentrate due to fluorine content beyond the specification for import and allows for a reduction in circulating load around the cleaning circuits. This reduction will allow higher head grades to be treated than those in the current design, which will increase the amount of Hugo North ore that can be processed before the cleaning circuits require additional banks of cells.

Figure 17.6 IDOP Flow Sheet



AMEC Minproc has reviewed the circuit proposed by Aminpro and Fluor and found it to be adequate to achieve the design criteria at plant startup. As production schedules used by Fluor and Aminpro were superseded by ongoing AMEC Minproc activity the expansion options evaluated by Fluor and Aminpro were also superseded.

Expansion Equipment Requirements

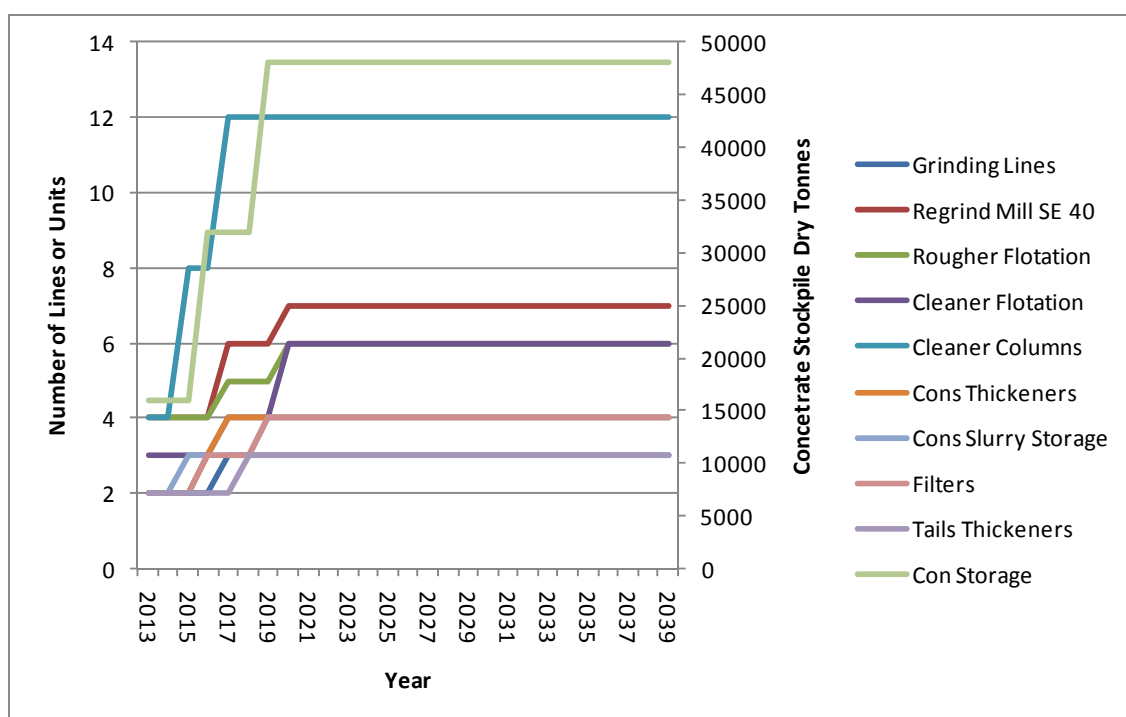
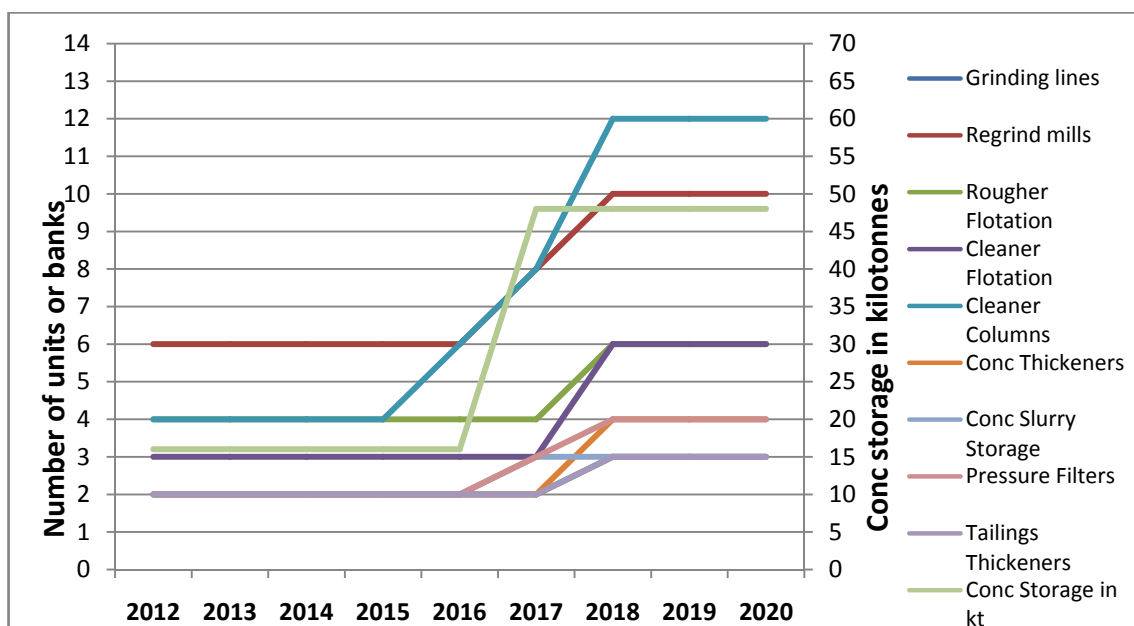
AMEC Minproc developed a calculation methodology designed to interact with the annual mine production schedule outputs to estimate year by year process equipment requirements. The basis of the calculations was to continue to meet the process design criteria specific to the process area as outlined in Fluor's Process Design Criteria DC-25-002, Rev C.

These calculations relied upon the following data from the schedule IDOP Reserve Case:

- Plant feed tonnage
- Rougher concentrate production tonnage
- Final concentrate production rate
- Final concentrate grade predictions
- Tailings tonnage

The predicted equipment requirements are summarized in Figure 17.7. Note that regrind mill predictions are based on achieving a 40 μm P_{80} . The equipment numbers in 2013 correspond to the numbers arrived at in Fluor's 2009 design review.

Figure 17.7 IDOP Equipment Requirements Based on MP01



The predicted equipment requirements are summarized in Table 17.9. Note that regrind mill predictions are based on achieving a 30 µm P₈₀ regrind on Southwest ore and a 40 µm P₈₀ regrind on Hugo North ore.

Equipment requirements for Year 1 to Year 9, the last year of predicted equipment purchase, are shown in Table 17.9. Capital costs have been distributed between one and three years in advance of operational status.

Table 17.9 Major Equipment Number Requirements IDOP Reserve Case

Year of Operation	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Grinding lines	2	2	2	2	2	2	3	3	3
Regrind mills	6	6	6	6	6	8	10	10	10
Rougher Flotation	4	4	4	4	4	4	6	6	6
Cleaner Flotation	3	3	3	3	3	3	6	6	6
Cleaner Columns	4	4	4	4	6	8	12	12	12
Conc Thickeners	2	2	2	2	2	2	4	4	4
Conc Slurry Storage	2	2	2	2	2	3	3	3	3
Pressure Filters	2	2	2	2	2	3	4	4	4
Conc Storage in kt	16	16	16	16	16	48	48	48	48
Tailings Thickeners	2	2	2	2	2	2	3	3	3

17.1.5 Further Work

A Feasibility Study will be conducted to determine the requirements and costs of the concentrator expansion to add a third grinding line and increase plant production by approximately 50,000 t/d. The work will include an analysis of comminution options and further characterization of the ore types from the various deposits. The aspects to be investigated are listed below

17.1.5.1 Plant Design/Production Scheduling

Throughput/Comminution:

- Phase 2 160,000 t/d design criteria.
- Additional HN ore hardness characterization/representation.
- Estimation of the optimum primary grind using the economic model parameters, CEET, FLEET, and liberation data.
- Estimation of the optimum SAG mill/ball mill power split.
- Primary crusher feed size (blasting) and product size (OSS) optimization, together with SAG feed size optimization (CEET).

Roughers

- Collector, frother, and pH optimization
- Blending opportunities – Southwest with Hugo North

- Arsenic/enargite depression testwork on high As zones (Hugo North and Central)
- FLEET modelling

Cleaners

- Hugo North regrind size and cleaner kinetics optimization.
- Optimization of fluorine liberation and rejection.
- Arsenic/enargite depression testwork on high As zones (Hugo North and Central).
- FLEET modelling for cleaner and cleaner-scavenger sizing optimization.
- Circuit reconfiguration to use excess SW cleaner capacity to treat high-grade ore.
- Update locked-cycle versus open circuit cleaner performance.

General

- Simulation of process water quality.
- Impact of reclaim water (process + raw water) on flotation performance.
- Refinement of the metallurgical performance algorithms in the block model.
- Extended variability work on HN composites in conjunction with block cave schedule.
- Production of additional tailings samples for further humidity cell testing on Southwest, Hugo North, Central, and blends over a wider range of ore types.

17.1.5.2 Reserve Ore Characterization/Feasibility Studies

Central

Confirmatory testing of all three Central ore types will be conducted, including locked-cycle, synergy, and variability.

Hugo South

Although Hugo South is not included in the MP01 schedule, it is a potential component of alternative schedules. A metallurgical program is required for Hugo South to bring it to feasibility study standards.

Heruga/Molybdenum Recovery

As with Hugo South, the Heruga deposit is not part of MP01 schedule. Ongoing pre-feasibility studies of Heruga are in progress. The molybdenum head grades for Heruga are substantial, and the economic viability of its separation from the other valuable minerals should be investigated.

17.2 Tailings

The current design for the tailings storage facility (TSF) at Oyu Tolgoi has been produced by Klohn Crippen Berger (KCB) in a report dated November 2010¹. The following sections provide a summary of the tailings storage at Oyu Tolgoi.

17.2.1 Design Criteria

17.2.1.1 Tailings production

The current TSF is designed to store approximately 720 Mt of tailings. The tailings production rate is expected to reach 34 Mtpa or close to 100,000 tpd after an initial ramp up period of 12 months. A production increase in 2018 is expected to increase the tailings production rate to 60 Mtpa or close to 170,000 tpd.

17.2.1.2 Tailings properties

The tailings properties integral to the design of the TSF are detailed in Table 17.10.

Table 17.10 Tailings Properties

Solids Content (w/w)	Disposal	64%
	Post Settlement	75%
PSD	Sand	42.7%
	Fines	57.3%
Specific Gravity		2.8
Density (t/m³)	Initial	1.4
	Average	1.5
Tailings Slope		1%
Angle of Friction		34°

17.2.1.3 Design Guidelines

In the absence of Mongolian standards for the design of tailings facilities or water retaining dams, Canadian standards have been adopted for guidance in selection of design criteria for the Project. The Canadian Dam Association (CDA, 2007) has developed a classification scheme, in its “Dam Safety Guidelines 2007”, that were used to provide guidance on the “standard of care” for design of dams.

The TSF has been designed with a “High” consequence classification according to CDA (2007). Corporate and political impacts of a TSF failure to the Project and the Owner are difficult to quantify and likely pose the highest consequence, which may be higher than the consequences considered by CDA (2007). An increase in consequence classification will have minor impact on seismic and flood management designs.

¹ Klohn Crippen Berger, 2010 Tailings Storage Facility Updated Feasibility Study Design , November 2010

However, the selected design earthquake Maximum Credible Earthquake (MCE) is suitable for “Very High” to “Extreme” categories. Likewise, larger flood events are not expected to cause catastrophic failure of the TSF.

17.2.1.4 Topography

The project is located in the southern Gobi desert near elevation 1180 masl. The surrounding topography is very flat with low rising hills up to elevation 1350 masl within 40 km of the site. The surface gradient within the tailings impoundment area is in the order of 0.3% and drainage is towards the south-east. The main regional drainage is the Umdai River, approximately 2 km west, which flows southward during periods of rainfall.

17.2.1.5 Climate

The climate is arid with an average annual rainfall of 64 mm and potential evaporation of 2,340 mm. The 1:1,000 year 24 hour rain event is 105 mm, whilst the 30 day Probable Maximum Precipitation (PMP) is 600 mm. Winds are generally less than 10 m/s; however, higher winds occur 8% of the time with major dust storms occurring on a weekly to monthly basis.

17.2.1.6 TSF Foundation

The soils in the TSF area are mainly ancient remnant soils, with some recent surficial alluvium and aeolian soils. The dominant soil in the TSF area is the Cretaceous clay, which was deposited in a lake environment during the Cretaceous period. The clay overlies weathered bedrock and there are transition soils consisting of colluvial clay/weathered rock and sandier clay soils.

In descending order, the subsurface soil profile within the TSF area consists of the following three main geotechnical units:

- Unit 1 - fluvial and aeolian deposits.
- Unit 2 - Cretaceous clay.
- Unit 3 - weathered bedrock and residual soils.

Unit 1 - Fluvial and Aeolian Sands:

A thin layer (20 cm) of aeolian (wind-blown) silt-sand deposits covers most of the TSF area. Fluvial (water-transported) deposits associated with the ephemeral creeks occur in topographical low areas in the northern part of the TSF and along the Center Dyke. The fluvial deposits are a mixture of well-graded sands with less than 10% fines and silty sands, locally extending up to 4 m depth. The loose, clean sands could be susceptible to liquefaction during earthquake loading if saturated.

Unit 2 - Cretaceous Clay:

Cretaceous clay is the dominant foundation soil in the TSF area, underlying approximately three quarters of the embankment and impoundment footprint. This unit is a heterogeneous blend of high plasticity clay and intermediate plasticity clay, with

discontinuous zones of clayey sand up to 2.3 m thick. The discontinuous sand zones were generally found to result in perched water tables overlying locally saturated clay. The Cretaceous clay unit is deepest towards the north, up to 30 m deep. The thickness of the Cretaceous clay thins towards the south west. The clay layer becomes discontinuous over rising bedrock near the south. The Cretaceous clay thickness has a large influence on the dam foundation strength, pore pressure response and stability. While the Cretaceous clay deposits are deepest towards the north end of the TSF, samples of equally high plasticity can be observed in the southwest and southeast embankment foundations, suggesting a relatively homogeneous distribution of the high plasticity clays. These high plasticity clays of the Cretaceous clay unit are of primary concern to the TSF design.

Unit 3 - Residual Soil and Weathered Bedrock:

Beneath the Cretaceous clay and above the intact bedrock lies a transitional unit comprising weathered bedrock, residual soils and layers of clay and sand and gravel. The unit is characterized by a gradually increasing coarsening of particle size with increasing depth. These soils are very dense, typically 2 m to 10 m thick, and vary from low permeability highly weathered bedrock to moderate permeability fractured, slightly weathered bedrock. The weathered bedrock overlies fresh, low permeability bedrock.

17.2.1.7 Seismicity

The Project is located far away from the major plate boundaries and high seismic activity areas, including the Gobi-Altai fault system located north-west of the project site. The seismic activity north, east, and south of the project site is generally low. However, two known active faults in the vicinity of the project site were identified by Research Center of Astronomy and Geophysics of Mongolia (RCAG) during field investigation. These faults, the Tavan Takhil fault and Borzon fault, are located approximately 18 km and 120 km, respectively, from the project site and are capable of generating magnitude M7 earthquakes. The return period of the characteristic earthquake on the Tavan Takhil fault estimated by RCAG is 2,250 years.

A deterministic seismic hazard assessment (DSHA) was conducted to establish ground motions corresponding to the MCE. Based on the assessment, the MCE acceleration spectrum with a Peak Ground Acceleration (PGA) of 0.32 g for the project site due to an M7 earthquake on the Tavan Takhil Fault is established. A suite of ten time histories from past earthquakes satisfying various selection criteria are selected and modified to match the MCE target spectrum. These modified time histories are used in the seismic deformation analyses of the TSF.

17.2.2 TSF Design

The current proposed location for the TSF is 1 km to the east of the pit area. It will be constructed as two adjacent cells (Cell 1 and Cell 2), each approximately 2,000 m long by 2,000 m wide. Each side of these cells will be formed by an earth fill and rock fill embankment dam, up to approximately 70 m high. This design is capable of storing the required 720 Mt of tailings, with potential to expand to store 1,500 Mt of tailings through additional cells to the north and east. Each cell will contain three internal and parallel splitter dykes forming four subcells of 500 m by 2,000 m. Cell 1 will be constructed

initially and store tailings for approximately 5 to 6 years, at which point Cell 2 will be constructed. After Year 6, tailings will be deposited into Cell 1 and Cell 2 until both cells reach an equal and final height. Tailings will be spigotted from the west side of the TSF (Southwest and Northwest embankments) and form a beach at an estimated slope of 1% towards the east side. Tailings supernatant water will be collected in a reclaim pond that will be managed in one corner of each cell, and be recycled to the mill.

A starter dam is required to commission the facility and to form the basis for the geometry of the impoundment. The starter dam will be constructed from locally borrowed earth fill. During operations, the embankment raises will be constructed from three categories of mine waste rock, in addition to local clay borrow materials. These construction materials are listed below:

- Oxide waste rock and overburden
- Sedimentary rock
- Random waste rock
- Borrow Clay

17.2.2.1 Embankment Design

All TSF embankments will be raised using the centerline or a modified centreline raising geometry with the exception of Reclaim Pond section which will be raised using downstream raising geometry. The centreline inclination angles may be adjusted during operations to suit differing tailings conditions, to provide stable crest raises and minimize dam fill.

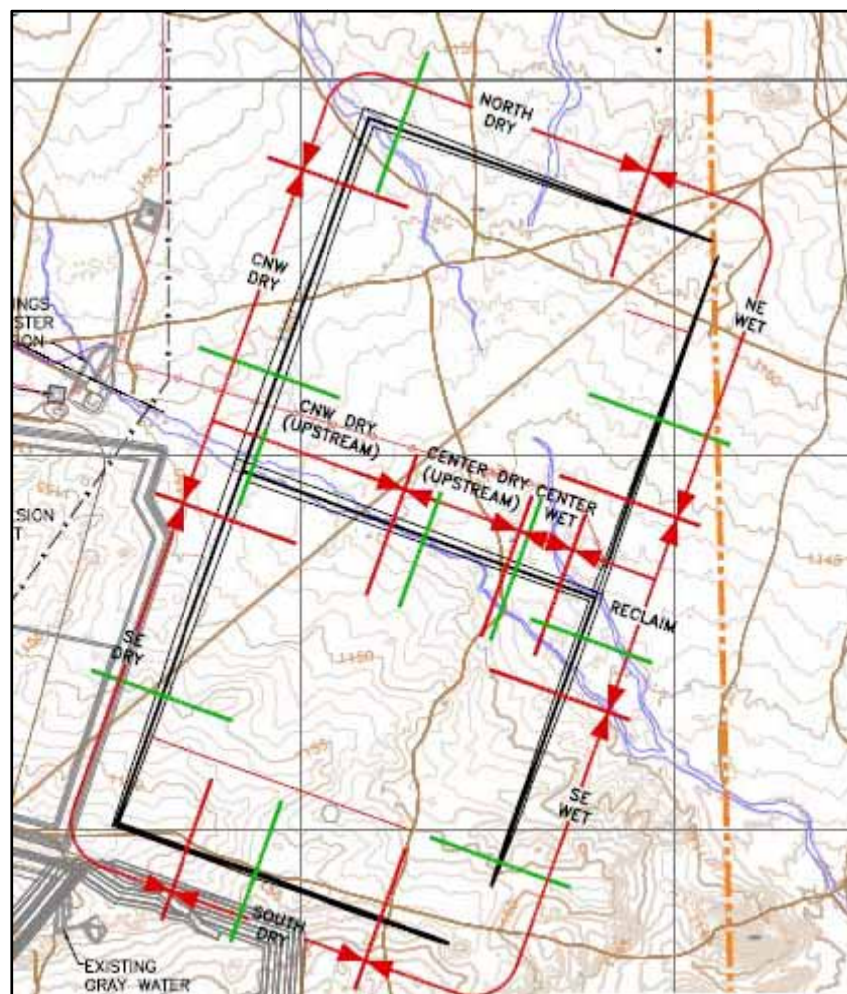
The starter perimeter embankments will have a downstream slope of 2.5H:1V and are founded on residual soil or weathered bedrock foundations. As the dams are raised, rock fill toe buttresses will be added to embankment sections over Cretaceous clay foundations. Buttresses will range from 50 m to 250 m wide depending on the clay thickness and embankment location. The resulting overall downstream slopes will vary from 3.2H:1V to 6.5H:1V (measured crest to toe). Shear strength parameters have been selected from laboratory testing and applied to the design sections to optimize fill quantities around the TSF. The final geometry may be refined further during construction based on observations made on foundation responses (pore pressure and deformation) and to results from any test fills.

The TSF perimeter embankments are divided into three primary cross-section design types shown in Figure 17.8 and summarized as follows:

1. Dry Section: The upslope-1500 m length of the South, Centre and North embankments, and the entire Southwest and Northwest embankments. This section will be constructed with an upstream-inclined centreline, using tailings to partially support each crest raise and reduce the embankment volume.
2. Wet Section: The downslope-500 m length of the South and North embankments, 250 m length of the Centre embankment between dry and wet reclaim pond sections, and the outer 1,500 m length of the Southeast and Northeast embankments where tailings supernatant and finer, wetter tailings will be against

- the embankment from 25% to 75% of the cell's full cycle time. This section will be raised using the centreline approach to improve constructability of crest raises.
3. Reclaim Pond Section: The 550 m length of the south-east and north-east embankments adjacent to the reclaim pond, and downslope-250 m length of the Centre embankment where there will be a minimum 3 m deep water pond directly against the upstream face of the embankment at all times during operation of the tailings cell.

Figure 17.8 Design Sector Locations



The design cross-sections are divided into material zones which will be constructed using different materials and placement methods. Table 17.11 summarizes the material zones used in the design cross-sections, and their primary specifications.

Table 17.11 Material Zone Specifications

Zone	Material Type	Maximum Lift Thickness	Gradation
Zone A	Clay Borrow	300 mm	>25% fines
Zone B1	Oxide and Overburden Waste	300 mm	15% to 25% fines
Zone B2	Oxide and Overburden Waste	600 mm	5% to 15% fines
Zone C	Sedimentary Waste Rock	2 m	0% to 5% fines
Zone F1	Random (NAF/PAF) Waste Rock	5 m	none specified
Zone F1	Random (NAF/PAF) Waste Rock	10 m to 20 m	none specified

Dry Section

The general and center dam dry design sections are shown in Figure 17.9 and Figure 17.10 respectively.

Figure 17.9 Typical Dry Section – General

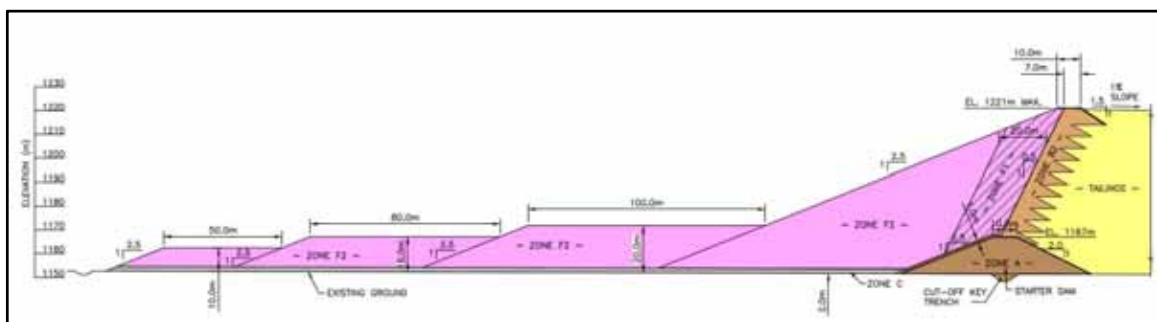
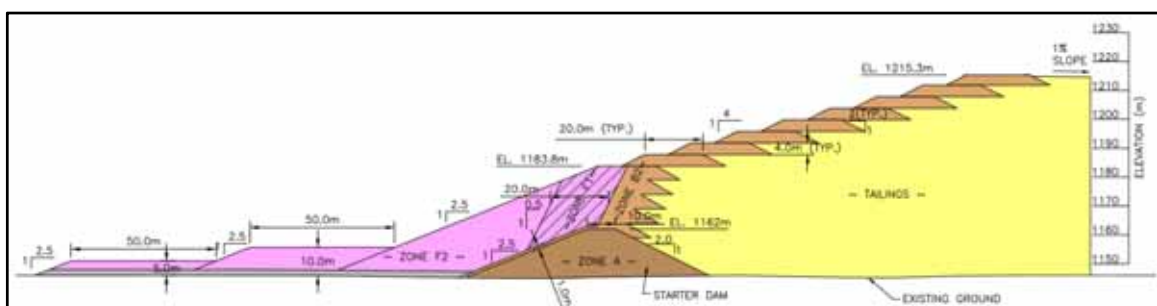


Figure 17.10 Typical Dry Section – Centre Dam



The center dam has been modified to incorporate the ongoing raising of the adjacent cell. The sections include the following zones, from upstream to downstream:

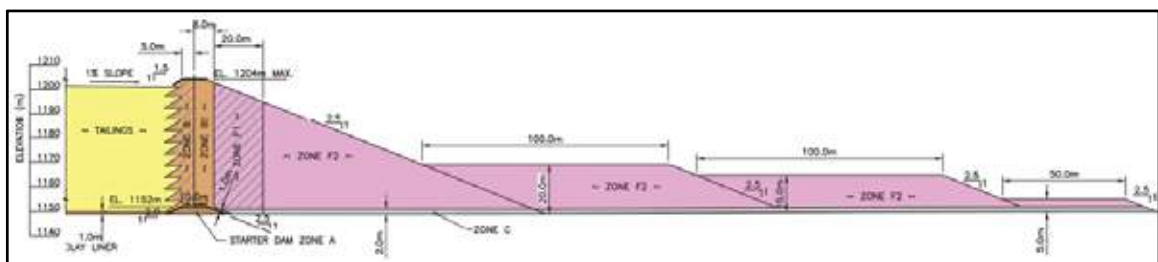
- Zone B2 (Oxide Waste Rock): 7 m wide filter zone adjacent to tailings.
- Zone F1 (Random Waste Rock): 20 m wide controlled-placement downstream shell zone to limit the lift thickness discrepancy between the bulk downstream placement of rock fill and the more controlled filter zone construction.

- Zone F2 (Random Waste Rock): Bulk downstream shell placed by the mining equipment fleet in 5 m to 20 m thick lifts, to fit with open pit production and waste schedules.
- Zone C (Sedimentary Waste Rock): Minimum 2 m thick drainage blanket to separate Random fill from PAF waste rock from the foundation, for long term closure concerns of potential infiltration of precipitation or surface run-off.

Wet Section

The wet design section is shown in Figure 17.11.

Figure 17.11 Typical Wet Section



This embankment section includes the following zones from upstream to downstream:

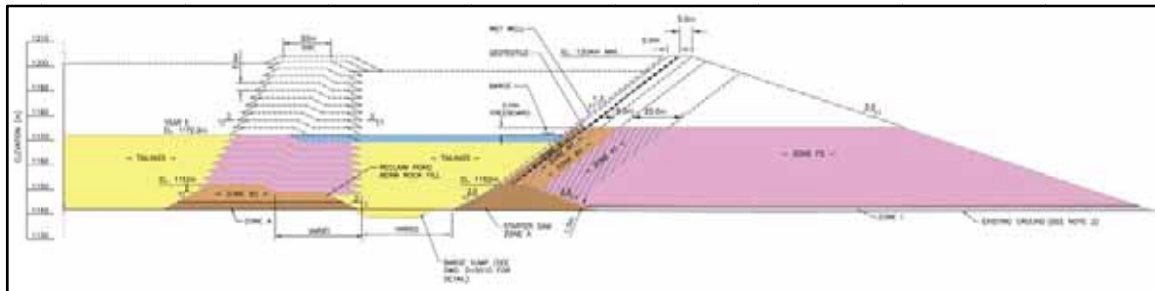
- Zone B1 (Oxide Waste Rock): 5 m wide fine filter zone.
- Zone B2 (Oxide Waste Rock): 8 m wide coarse filter zone.
- Zone F1 (Random Waste Rock): 20 m wide controlled placement downstream shell.
- Zone F2 (Random Waste Rock): Bulk downstream shell placed by the mining equipment fleet in 5 m to 20 m thick lifts, depending on the dam geometry.
- Zone C (Sedimentary Waste Rock). Minimum 2 m thick drainage blanket to separate random fill from PAF waste rock from the foundation, for long term closure concerns of potential infiltration of precipitation or surface run-off.

If gradation control proves problematic, or if significant water ponding occurs along the wet sections of the impoundment, a geotextile may be required on the upstream side of Zone B1.

Reclaim Pond Section

The reclaim pond design section is shown on Figure 17.12.

Figure 17.12 Typical Reclaim Section



The reclaim pond embankment section will include the following zones from upstream to downstream:

- Zone A (Clay Borrow): 5 m wide low permeability core zone compacted in 300 mm lifts to 95% of Std. Proctor maximum density.
- Geotextile: Required to prevent piping of tailings. If the gradation and quality control prove sufficient to provide filter compatibility between Zones B1 and Zone A/tailings, the geotextile will not be required.
- Zone B1 (Oxide Waste Rock): 5 m wide fine filter zone.
- Zone B2 (Oxide Waste Rock): 8 m wide coarse filter zone.
- Zone F1 (Random Waste Rock): 20 m wide controlled placement downstream shell.
- Zone F2 (Random Waste Rock): Bulk downstream shell placed by the mining equipment fleet in 5 m to 20 m thick lifts, depending on the dam geometry.
- Zone C (Sedimentary Waste Rock). Minimum 2 m thick drainage blanket to separate Random fill from PAF waste rock from the foundation, for long term closure concerns of potential infiltration of precipitation or surface run-off.

Embankment Stability

The embankments have been designed and modelled to achieve factors of safety as specified by Canadian (CDA) and International Committee on Large Dams (ICOLD) standards for tailings facilities.

Clay Liner

More than three quarters of the TSF footprint is lined with the natural Cretaceous clay deposit, providing an ideal barrier to groundwater seepage from the tailings impoundment. In areas of the impoundment where the Cretaceous clay is not present, primarily the southeast corner of Cell 1, a clay liner will be constructed to reduce seepage into the underlying bedrock and residual soils.

The clay liner will consist of 1 m of compacted Cretaceous clay placed in the southeast corner and in a second isolated exposure of residual soil in Cell 1. Clay fill will be borrowed from thicker deposits of Cretaceous clay primarily in Cell 2 and to a lesser degree from the northern side of Cell 1. The clay liner will be placed and compacted in

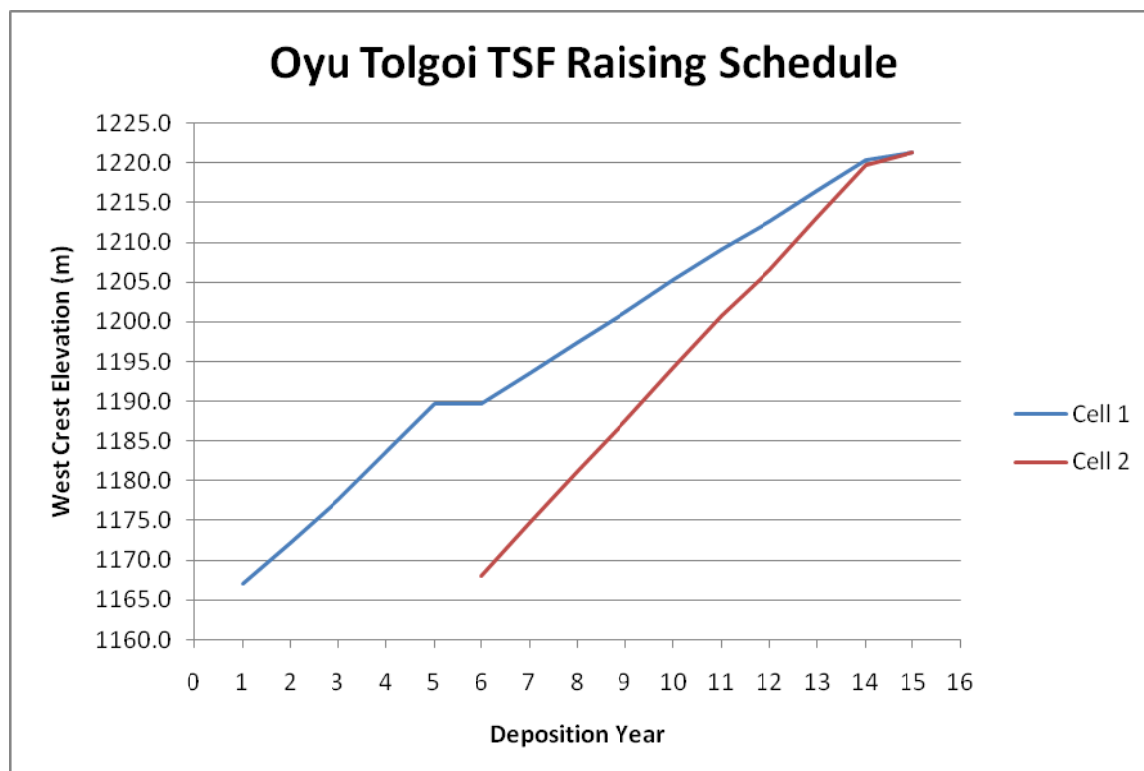
300 mm lifts, to 95% of Standard Proctor maximum density of optimum moisture content. The upstream edge of the liner will be keyed into the native clay (minimum 1 m) to maintain continuity with the natural clay deposit. The downstream edge of the clay liner will be tied into the Starter Dam Zone A – low permeability zone. The clay liner will also extend under the reclaim pond in Cell 1 and the tailings underdrainage along the east embankments.

The majority of the clay liner will be constructed during pre-production (Year 1) and covered in the first year of tailings deposition. The extreme south-east corner of Cell 1; however, will not be covered by tailings until Year 2. This area of the liner will require a strategy to avoid its desiccation and potential cracking. Strategies may involve stockpiling the clay fill in Year 1 and delaying placement until Year 2, or placement of a protective soil cover over the clay liner. Details and timing of liner construction will be reviewed prior to starter dam construction and once a tailings start-up schedule is confirmed.

17.2.3 TSF Construction Schedule

To enable commissioning, a starter dam will be constructed as a part of Cell 1 with subsequent embankment raises providing sufficient storage for tailings and water during the first 5 to 6 years. Prior to Year 6, the construction of Cell 2 will occur with subsequent embankment raises of both cells providing sufficient storage for tailings and water.

Figure 17.13 TSF Construction Schedule



17.2.4 Tailings Deposition Plan

Tailings will be deposited for a period of 1.5 months (6 weeks) successively in each of the four subcells. A full deposition cycle will take 6 months and it will occur twice each year. On start-up, cycling will be more frequent until the tailings surface rises above the irregular topography and forms a beach the full length of the subcell. This is predicted to occur in Year 2. Each deposition cycle starts with spigotting into subcell 1a (furthest from reclaim pond) followed in succession by subcell 1b, subcell 1c and finally subcell 1d (nearest reclaim pond). This order is required to promote a downhill gradient in the tailings beach along the East embankment to allow water to flow to the reclaim pond.

Tailings will be pumped from the mill to a booster station located near the northwest toe of Cell 1. Two pipes will run from the booster station up to the dam crest. One pipe will be connected to a spigot located at the midpoint of subcell 1d and the other pipeline will run along the dam crest connecting to spigots located at the midpoints of subcells 1c, 1b, and 1a. Valves located at each of these three spigots will allow spigotting to occur in a single subcell at any given time. The two independent pipes are required to ensure uninterrupted tailings deposition during dam raises as each pipe can be raised while the other is active.

The gaps at the east end of the splitter dykes are unregulated and thus both water and tailings will flow through them. Water will flow down-gradient to the reclaim pond, while the tailings will build a fan out into the adjacent cell at an average beach slope of 1%. For ore production levels of 85,000 tpd, the average thickness of the tailings layer deposited in each subcell during one cycle is approximately 2.7 m. Therefore, the tailings fan from one deposition will build out approximately 135 m along the splitter dyke and 270 m along the east dam. Once deposition switches to the next subcell, tailings will build up and eventually level out the subcell to a constant 1% beach slope.

17.2.5 Water Management

Water conservation is important due to the scarcity of water resources within the mine area and the high cost of extracting groundwater and conveyance of water to the mine site. The closest water source is an aquifer located 60 km from the mine site. A well field will be constructed and the water will be piped to the mine. The groundwater resource contains ancient groundwater in a series of interlayered aquifers. The well field will be difficult to develop and may not be recharged on a seasonal basis.

The objectives of the water management are as follows:

- Reduce water losses.
- Increase the efficiency of water recovery.
- Safe routing of flood waters.

Water recovery is mainly controlled by the wetted area of the impoundment and the settled density of the tailings. Additional influences include the amount of freezing during winter operations and higher evaporation during drier months. To maximize water recovery, the current design comprises long, narrow subcells for controlled tailings discharge to minimize evaporative losses and promote water release by consolidation.

17.2.5.1 Diversion Channels

The diversion channels have a minimum bed width of 2.0 m and are designed to route clean water run-off from the 1:1,000 year, 24 hour storm. This water is routed into the creek bed downstream of the East Seepage Pond.

Construction of Cell 2 may cause the run-off from the catchment west of the tailings facility to pond against the toe of the west tailings dam. The tailings booster station will need to be set above the flood elevation level that will apply when Cell 2 is constructed. For closure, run-off from upstream of Cell 1 and Cell 2 will be directed to the open pit. The north section of Cell 2 will be supplied with erosion protection.

17.2.5.2 TSF Water Storage and Freeboard

Precipitation on the TSF from the design flood, specified as one third between the 1:1,000 year (24 hour) storm and the 24 hour probable maximum flood (PMF) storm, will be contained within a 2.1 m deep pond, along the east dam. In the case of a flood that exceeds this volume, emergency measures can be taken to ensure controlled release of floodwater. As the dam is constructed of rock fill, it is resistant to overtopping. Due to the available storage, overtopping is expected to be minimal. For example, flood routing of the PMF would require 0.2 m depth above the pond formed by the design flood. Assuming 0.6 m of freeboard, above routed PMF level, the crest of the East dam should be kept a minimum of 2.9 m above the average of the tailings level along the East dam excluding reclaim pond portion.

17.2.5.3 Reclaim Ponds

The water reclaim ponds are located in the east-centre of the impoundment, and will operate separately in Cell 1 and Cell 2. A pump barge will be utilized to return supernatant water to process plant for reuse in the mill. A reclaim pond dyke will be constructed to control the flow into the reclaim pond to limit the extent of the reclaim pond size.

17.2.5.4 Seepage Collection

The seepage collection ditches are unlined with 2H:1V slopes and have an invert widths of between 1.0 m and 2.0 m.

The seepage dam is a very small structure with expected dry or low level conditions and hence has low impact to environment and very low economic losses in case of a breach. It is thus classified as a "Low" CDA (2007) consequence category, and is therefore to be designed to handle flows from a 100 year storm. The seepage and the 1:100 year, 24 hour run-off from the downstream slopes of the tailings dams and the catchment area between the west dam and the pit (including the waste dump) will be collected in a pond with 120,000 m³ storage capacity. The seepage pond is designed with a 10 m wide riprap spillway to route the 100 year storm.

17.2.6 Spillway

At closure, a spillway of 400 m width should be constructed 0.8 m lower than the dam crest level. The downstream slope of the dam at the spillway is assumed to be 3H:1V and would be constructed of rock fill with D50 larger than 250 mm.

17.2.7 Instrumentation and Monitoring

An instrumentation section consisting of vibrating wire piezometers, inclinometers, survey monuments and a total stress plate will be installed at various points around the impoundment to monitor embankment performance and any movement. An observational approach should be used during construction to optimize design and minimize fill volumes.

To gain a better understanding of the Cretaceous clay to loading, and to support the observational approach, a test fill loading trial will be carried out. The foundation clay will be instrumented and a rock fill dump (up to 40 m high) will be extended over the instrumented section. Measurements will be used to back analyse the pore pressure and deformation response and determine the mobilized strength of the clay in situ.

17.2.8 Closure

Structures will be stable on closure and shall not impose a risk to public health and safety, or the environment.

- The TSF will be closed as a “dry” facility.
- Consolidation and drain-down of the tailings will minimize geotechnical risk.
- Tailings Closure Cover: Dust control could potentially require the placement of a layer of NAF rock-fill over the tailings. Test plots will be carried out during operations to confirm the basis for design of the cover.
- Embankment Closure Cover: A NAF cover (low permeability or store-and-release) will be placed over PAF waste rock to minimize infiltration of surface water. All natural surface water flow will be directed away from contact with the PAF waste rock to minimize the potential for metal leaching and “contaminating” surface run-off. Test plots will be carried out during operations to confirm the basis for design of the cover.
- Major precipitation events (e.g., PMP) will be either be stored on the surface and evaporated, or safely routed down the face of the dam.

17.2.9 Independent Technical Review

In November 2010, OT LLC established an Independent Technical Review Board (ITRB) to provide review and oversight of the TSF Design and operational management. The ITRB selected by OT LLC is comprised of Professor Norbert Morgenstern, Mr. Mark Logsdon and Mr. Richard Davidson. The purpose of the Board was provided by Mr. Joergen Pilz of Rio Tinto Technology and Innovation on 23 November. The initial task of the ITRB as directed by OT LLC was to evaluate and comment on the design to ensure that TSF design, construction and operation is undertaken in accordance with

good international industry practice and review the final TSF designs prior to financial close of project financing.

The ITRB affirmed concurrence with the Draft Feasibility Report subject to addressing some recommendations. The ITRB conclusions in the third ITRB meeting in July 2011 were:

- The feasibility design of the tailings facility is suited in all material respects for the satisfactory long-term operation of the tailings facility for its intended purpose.
- The feasibility design of the tailings facility has, and is being performed, in accordance with international standards of good practice for projects of this nature and scope, and no weaknesses have been identified which could reasonably be expected to have a material adverse effect on the integrity of the tailings facility, human health and safety, or successful operation of the facility for its intended purpose.

17.2.10 Conclusions

The following conclusions are drawn from the review carried out for the current TSF design:

- The design for the TSF, as presented in the Interim Design Report (KCB, 2008) is acceptable, and ongoing work will produce a TSF that will meet Mongolian, Canadian, and US guidelines for reporting.
- The Cell 1 and Cell 2 design will store a nominal 720 Mt of tailings.
- There are opportunities to further simplify the design and operating plan to achieve a lower net present cost for tailings management.
- Removal of the constraint to remain inside the Oyu Tolgoi Licence Area 6709A boundary presented an opportunity to optimize the TSF arrangement, and options exist to expand the facility in the future either to the east or north.

17.3 Metallurgical Plant

17.3.1 Summary

The initial concentrator design is based on processing 35 Mtpa of ore from the Southwest open pit and Hugo North block cave underground deposits followed by an expansion to bring capacity to 58 Mtpa.

For the first 3 years, Hugo North development ore will be added to enrich the feed to the concentrator as it is produced. The following 2 years, development will advance and as production from the Hugo North block cave increases, ore from the Southwest open pit will be displaced. This progressive replacement of the Southwest ore with Hugo North Ore will significantly increase the copper head grade. This change in head grade requires an increment of additional flotation and concentrate handling equipment in Years 3 and 4.

This equipment upgrade, required in Year 3, has been identified and the capital requirements for these changes are considered as sustaining infusions and have been considered in the financial analysis.

By Year 6, a significant concentrator expansion is planned to increase the capacity of the plant to 58 Mtpa. This expansion adds a third grinding line together with additional flotation and concentrate handling equipment to match continued mine production from the Southern Oyu and Hugo North deposits together with ore from the Central Oyu open pit deposit.

For the next 13 years, ores from the two Southern Oyu open pits (Southwest and Central) and the Hugo North are processed at rates between 55–58 Mtpa. Approximately 50% of the ore will be provided by Hugo North. By Year 20, ore from the two open pit deposits will begin to diminish and be fully replaced by additional Southwest and Central open pit ore in Year 22. Open pit ore will feed the plant until the end of life in Year 30.

The following Concentrator Process Plant description provides details of IDOP Reserve Case including the processing schedule, design criteria, circuit, and equipment description.

17.3.2 Process Design Criteria

The process design criteria for the facilities are based on the extensive metallurgical program described in Section 13, Mineral Processing and Metallurgical Testing are summarized below.

- Geometallurgical modelling of grinding parameters jointly conducted by OT LLC and SGS Mineral Services.
- Annual mine production plan based grinding capacity and product size determinations provided by SGS Mineral Services.
- Review of the Oyu Tolgoi flotation testing data base and testwork directed by Amelunxen Mineral Processing Ltd (AMINPRO).
- Testwork performed by Process Research Associates Ltd.
- Flotation simulation balances and analysis conducted by AMINPRO.
- Testwork performed by Dorr Oliver Eimco.
- Water balances developed by Klohn Crippen.
- Testwork supervised by AAJV and performed at AMMTEC using samples obtained during the 2004/2005 sampling campaign.
- Testwork performed at MinnovEX Technologies Inc., Toronto, Ontario.
- Testwork performed by the following subconsultants, based on samples generated at AMMTEC:
 - GL&V Australia Pty Ltd, Belmont, Western Australia.
 - G&T Metallurgical Consultants, Kamloops, British Columbia.

- Metso Minerals Industries Inc., York, Pennsylvania.
- Data for site conditions supplied by Knight Piésold.

The grinding and flotation circuit capacity was modelled by SGS Mineral and AMINPRO and confirmed to match production from annual mine plans available at the time.

Optimum economic grind was estimated using AMINPRO-Flot for all ore types and used as guidance in sizing the grinding circuit. Throughput-recovery trade-offs provided the optimum economic throughput and grind for mill sizing.

The primary crusher, SAG and ball mills and pebble crusher circuits were modelled using SGS Mineral's CEET simulator and the geostatistical block model ore characterization data. CEET was used to determine the grinding power, and optimum capacity simulations were developed based on iterations of the mine production plan. These simulations determined that the IDOP Reserve Case should treat 96,000 tpd of Southern Oyu and Hugo North ores.

The flotation mass balance and equipment requirements were obtained with AMINPRO's flotation simulator. The basis for the model inputs were: OT LLC IDP05 standardized data, including locked-cycle test results from testwork at AMMTEC; kinetic data measured at MinnovEX's Toronto laboratories; and on data generated at Process Research Associates in 2007. This data was used to derive flotation response relationships for each ore type. The general flotation flow sheet defined during IDP05 was retained, with retention times, and equipment capacities adjusted, as required, to satisfy the output of the simulator.

With the flotation feed size determined for each production period, generated by CEET, the AFS simulator loaded with the flotation response parameters of each ore type, nominal and design grades for the period, and proportional contributions to mill feed of each ore type, generated full flotation circuit mass balances. The resultant flotation mass balances were then compared to the specified retention times and capacities of the circuit to determine if additional units were required. In the case of rougher, cleaner and cleaner-scavenger flotation, an additional row of cells was added to the flow sheet for each stage above that estimated by AMINPRO for reasons of hydraulic limitations.

IDOP is based on a different ore delivery schedule to that used by AMINPRO and Fluor in the above design work. Consequently the Fluor design criteria was used by AMEC Minproc to firstly confirm that the startup equipment selections remained sufficient to treat the initial ore delivery blend and, secondly, estimate a new set of plant expansion requirements to match the schedule.

The equipment requirements to achieve the production schedule, and the period in which the equipment is required to be operational, are identified in Table 17.12. Note the capital expenditure should happen in the year before the equipment is required to be operational.

The production values in the IDP10 Reserve Case were used to generate the equipment requirements. The sizes and number of units selected in Year 1 is as determined by the CEET and AFS simulations and the design factors contained in the Process Design

Criteria. The phased equipment requirements in Table 17.12 were determined by AMEC Minproc using the predicted process flows and duties that arise from the IDP10 Reserve Case schedule and the application of critical equipment design factors taken from the Fluor process design criteria.

Table 17.12 Time Phased Equipment Requirements

Equipment	Year						
	1	4	5	6	7	8	9
Primary crusher	1	–	–	–	–	–	–
Underground ore overland conveying	–	1	–	–	–	–	–
SAG mills	2	–	–	3	–	–	–
Ball mills	4	–	–	6	–	–	7
Rougher banks	4	–	–	5	–	–	6
CI and CI scavenger banks	3	–	–	4	–	–	6
No. of column cells	4	8	–	12	–	–	–
No. of vertical regrind mills	4	–	–	6	–	–	–
Concentrate filters	2	–	3	–	–	4	–
Concentrate thickeners	2	–	3	–	–	–	–
Concentrate storage tanks	2	3	–	–	–	4	–
Covered concentrate storage (t)	16,000	–	32,000	–	–	48,000	–
Tails thickeners	2	–	–	–	3	–	–

17.3.2.1 Design Factors

The primary crushing and overland conveyor facilities from the open pit mine to the coarse ore stockpile will be scheduled to operate on a continuous basis, 365 dpa, with an instantaneous capacity of 7,000 tph and utilization factor of 69%, or an average of 16.5 hours per day. The conveying facilities from the underground mine has been designed to operate on a continuous basis, 365 dpa, with an instantaneous capacity of 7,000 tph and utilization factor of 75%, or an average of 18 hours per day. The greater utilization of the underground crushing and ore handling system is achievable due to the replicated crushing and multiple hoisting shafts.

Mine drill patterns and “powder factors” were determined by Scott Blasting Consultants from the available data during IDP05. These values were used to support the capacity estimate of the primary crusher. As gyratory crusher volumetric capacity is defined by the proportion of its feed finer than the open side set, it is understood that actual blasting data, obtainable during pre-stripping, may require adaptation of the blasting plan to achieve fragmentation in the feed to the primary crusher required to produce the desired feed size distribution to the SAG mills.

The live capacity of the twin grinding line stockpile is 70,000 t with a total capacity of about 340,000 t. This provides a live capacity of about 16.8 hours during initial operation. Expansion to a three line grinding circuit will require an extension of the stockpile structure, increased storage volume, and construction of an additional reclaim tunnel.

Grinding, flotation, thickening, and tailings disposal facilities were scheduled to operate on a continuous basis, 365 dpa, with a utilization factor of 92%. For these plant unit operations, the design limits are 115% of nominal mass balance values.

The nominal head grade of 0.66% Cu was selected to be the highest annualized head grade for the first three years, and 2.28% Cu nominal for the next 4 years. The design head grade of 115% of the nominal head grade was applied for the first 5 years.

The process design criteria are summarized in Table 17.13.

Table 17.13 Summary of Comminution Process Design Criteria Years 1 to 5

Parameter	Unit	Design
Annual Ore Treatment		
- Years 1 to 5 (96 000 t/d)	ktpa	35,070
Crusher/Conveyor Utilization	%	69
Grinding and Flotation Availability	%	92
Ore Characteristics		
- Copper Head Grade	% Cu	0.766
- Moisture	% H ₂ O	3.0
- Hardness – Southwest Ore	–	–
Crushing Work Index	kWhpt	16.3
SAG Performance Index (SPI)		
Average	min	138
Range	min	59–293
Abrasion Index	–	0.08–0.274
JK "A"		
Average	–	79
Range	–	61–100
JK "b"		
Average	–	0.48
Range	–	0.21–0.86
JK "A x b"		
Average	–	36
Range	–	21–54
JK "ta"		
Average	–	0.61
Range	–	0.29–0.95
Hardness – Hugo North Ore		
Crushing Work Index	–	–
Average	kWhpt	23
Range	kWhpt	8.1–39
Bond Rod Mill Work Index		
Average	kWhpt	18.6
Range	kWhpt	14.4–24.0
SAG Performance Index (SPI)		
Average	min	100
Range	min	60–196
ROM Ore		
- Maximum size 98% passing	mm	1,200
80% passing	mm	500
Passing 150 mm	%	37
- Overland Conveyor Capacity	tph	5,293

Parameter	Unit	Design
- Stockpile Capacity, live	t	70,000
- Stockpile Capacity, total	t	340,000
Ore Reclaim Feeders (each SAG Mill Line)		
- Type	–	Apron
- Lines	–	2
- Number per Line	–	3
- Capacity, each	tph	1,123
SAG Mill Feed Conveyors (each)		
- Fresh Ore	tph	2 175
- Recycled pebbles	% new feed	30
- Recycled pebbles	tph	653
- Total mill feed	tph	2,829
- Particle size F ₈₀	mm	140
SAG Mills 1 and 2		
- Diameter	M	11.6
- EGL	M	6.9
- Ball charge	–	–
Nominal operating	% v/v	15
Maximum, mechanical design	% v/v	20
- SAG mill load	–	–
Nominal operating	% v/v	28
Maximum, mechanical design	% v/v	32
- Drive Type	–	Wrap-around
- Speed range	% critical	75–80
- Power, installed	MW	20.0
- Power utilization, average	%	90
SAG Mill Discharge Screens		
- Trommel screen aperture	mm	13 x 39
- Vibrating screen aperture	mm	14 x 40
Pebble Crusher Circuit	Each	3
- Feed to pebble crusher	% new feed	30
- Maximum feed size	mm	75
- Crusher	–	Cone
Capacity, each	tph	667
Product size, P ₈₀	mm	13
Transition Size SAG Mill Circuit to Ball Mill Circuit T ₈₀	µm	2,400
Ball Mills Lines 1 and 2		
- Number installed (each line)	–	2
- Diameter	m	7.3
- EGL	m	11.0
- Ball charge, nominal operating	% v/v	33
- Installed power, each	MW	11.4

Parameter	Unit	Design
- Power utilization, nominal, each	%	95
- Circulating load, ball mill, nominal	%	350
- Ball mill circuit, final product P ₈₀	μm	159
Primary Cyclones		
- Cyclone diameter	mm	800
- Cyclone overflow pulp density	% w/w	33

17.3.3 Process Description

The process is based on conventional technology and proven equipment, where possible single equipment streams have been chosen to minimize the number of equipment units and reduce footprint and complexity. The concentrator will be at the forefront of modern large porphyry-copper gold mills. A simplified process flow sheet is shown in Figure 17.14.

In briefest terms, the flow sheet consists of primary crushing and coarse ore stockpiling followed by SAG and ball milling with pebble crushing. Rougher flotation is followed by concentrate regrinding and two stages of cleaning, the second stage being column flotation. Final concentrate is thickened and filtered before being transported by truck or rail. Rougher and cleaner scavenger tails are thickened and disposed of in a tailings impoundment.

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18 PROJECT INFRASTRUCTURE

18.1 Power Supply and Distribution

The Oyu Tolgoi project will be energy-intensive, with electrical power requirements of more than 200 MW on start-up, increasing to around 310 MW in the longer term. A reliable and stable power supply is essential for operations and safety.

A number of studies have been completed since 2004 to investigate the advantages and disadvantages of various power supply alternatives, taking into account the terms of the Investment Agreement (IA), which allow OT LLC to obtain power from established sources in China until the end of Year 4, and important stakeholder issues related to the GOM desire to electrify the south Gobi and the substantial proportion of total country demand represented by OT LLC.

In line with the IA, the current plan is to establish a 220 kV interconnection between Bayainhanggai in Inner Mongolia, China, and the Oyu Tolgoi site to provide the power supply for Oyu Tolgoi until Year 4. A double-circuit, 170 km long transmission line will be constructed between Bayainhanggai and the site in time for the commencement of project production. Power for the relatively small construction loads will be provided through a diesel power station consisting of modular diesel generators.

For long-term power supply it has been assumed that a coal-fired power plant will be constructed at site. The power station will have a total installed generating capacity of 600 MW (gross) to meet the net peak load demand of 310 MW. This will be met by four 150 MW (gross) or four 127.5 MW (net) coal-fired units, each with an air-cooled condenser. Coal will be sourced from suppliers such as Tavan Tolgoi or Tsagaan Tolgoi within Mongolia. Space is provided in the plant design for up to six 150 MW units.

OT LLC's preferred option is to pursue the accelerated construction of a power station at OT LLC with construction to commence as soon as all stakeholders agree. OT LLC intends to seek approval from the board for an accelerated program if government permits are approved.

Currently, there is no clear date for the issuing of those permits and the IDOP Technical Report remains with the assumption that capital is expended by OT LLC to bring a power station into production by the fourth year of operation.

A Feasibility Study providing a definitive long-term power solution for the project is in preparation. It is expected that the timing of power station construction would be finalised in the preparation of DIDOP.

Power will be delivered to the infrastructure, underground mines, and concentrator via a twin-circuit 220 kV transmission line system and be distributed at 220 kV and 35 kV as required through a central substation approximately 500 m south of the concentrator facility. A second 220 kV switchyard and substation will be constructed adjacent to the production headframes for Shafts No. 2 and No. 3.

During operations, all critical process and other equipment, heating and lighting, communications, and computer systems will be kept in operation by switching

automatically to standby power supply from the diesel plant in the event of failure of the main power plant.

18.1.1 Access

18.1.1.1 Access Roads

Approximately 26 km of internal access roads for small vehicles and general normal road transport trucks will be constructed to Mongolian, international, and AASHTO Standards. Construction features will include 35 culverts, one river crossing, and three overpasses. The roads will be constructed of graded gravel. Side-drain ditches will be provided for stormwater drainage.

Access to the water borefield will be via a gravel service road from the plant site. Haul roads in the mining areas will be developed separately as part of the mine designs.

To support the project development, an existing, 105 km access road from China to Oyu Tolgoi will be upgraded using the self-perform fleet. As measured from the OT site, km 0 to km 25 will be private road, and km 25 to km 105 will be public road. Current work is progressing on obtaining the necessary water use and land disturbance permits to support road construction.

The access road development will include upgrading the roads linking the existing Mongolian customs facility at Gashuun Sukhait to the Chinese customs facility at Ganqimaodao. A working group has been established to obtain agreement on the maintenance of the "OT Road."

Beyond Ganqimaodao, a reasonable-quality provincial road approximately three years old connects to the Jingzang Expressway via the towns of Hailiutu and Wuuyan, if required. The road has deteriorated slightly but overall is acceptable for copper concentrate hauls. A railway is expected to be complete from Baotou to Gangimaodao in 2012, which will reduce traffic on the S212.

A gravel road has been completed to the town of Khanbogd; OT proposes to build a mine bypass road to detour public traffic around the project to the town of Khanbogd and beyond.

18.1.1.2 Concentrate Shipment and Handling by Truck

The project will begin producing copper concentrate at an estimated rate of 600,000 to 1,000,000 t/a. With no rail service in this undeveloped region of southern Mongolia, copper concentrate will need to be transported by truck from the Oyu Tolgoi mine to an OT transload facility at the Ganqimaodao border crossing, from where customers would normally be responsible for further transport.

The rail line in northern China is currently being extended to Ganqimaodo, and while customers would likely prefer to accept delivery to rail head here, the line is still under construction and Chinese railway officials have indicated that there is no excess capacity in the existing rail system. As a result, customers may have to arrange to

deliver the concentrates by truck to a railhead in Baotou or Hohhot within China for transfer, or possibly directly to the ultimate destination.

The trucking of this volume of product across southern Mongolia and northern China, combined with international border issues, will be a significant challenge. Border-crossing protocols and customs-clearance procedures are still developing as the border transitions from a small bilateral boarder to a large international border. The Governments of China and Mongolia have completed a bi-lateral transportation agreement, but this is being applied unevenly in favour of Chinese drivers operating in Mongolia. Neither Mongolia nor China appears to monitor or enforce load limitations with any regularity.

Safety will be a major area of concern for this operation, and a training program will need to be developed for the drivers. Skilled labour and management staff will need to be hired from other areas of the Inner Mongolia Autonomous Region (IMAR) and beyond in Mongolia.

A major support facility will need to be established for staff accommodation and to service the anticipated fleet of tractor-trailer equipment. An area of approximately five acres will be needed, which could be set up on the Mongolian side of the border crossing. The manufacturer's willingness to assist in providing a high level of parts and service support will be a major consideration in the final procurement decision for the equipment. There are no other maintenance facilities between OT and the Ganqimaodao border crossing, although the existing level of service improves slightly on the Chinese side of the border.

To ensure a reliable and efficient transportation strategy, OT could manage and operate the haul internally, or it could help a local haulage contractor that has provided services to OT in the past to develop into a bulk commodity transportation company. The latter is likely to be more successful in dealing with border issues to prevent serious delays or transportation stoppages.

18.1.1.3 Airstrip

A new airport is required at Oyu Tolgoi to replace the existing temporary facility for the transportation of people and goods to the site from Ulaanbaatar. The new airport will be 11 km north of the Oyu Tolgoi mine lease area and has been sited to avoid flight paths over existing camp areas and structures. The runway will be concrete-surfaced, 3,250 m long, be aligned with the prevailing north-west-south-east wind direction, and have a projected life of 30 years. Design criteria were established to suit commercial aircraft up to the Boeing 737-800 series arriving and departing twice per day.

Facilities at the terminal include standard check-in counters, a security checkpoint, a departure lounge sized for 100 persons, and an adjacent car park for 40 vehicles. Electricity will be provided from an independent diesel generator equipped with a standby unit. No aircraft refuelling capability will be provided.

18.1.2 Main Site Infrastructure Buildings

18.1.2.1 Accommodation Facilities

Accommodation facilities for the first 10 to 15 years of operations will be constructed adjacent to the existing main construction camp. These will be pre-fabricated, dormitory-style buildings, all equipped with sprinkler and fire-detection systems and central utility and housekeeping rooms on each floor. Seven buildings with single-occupancy rooms for senior staff and seven buildings with double-occupancy rooms for junior staff will be provided.

The complex will be complete with kitchen-dining area, TV/Internet, and other recreation facilities.

The existing construction camp will be used as long as possible, including the ger VIP, senior staff, and junior staff rooms, modular camp mess, laundry facilities, modular construction offices, and HSES, fire-fighting, and emergency medical facilities.

18.1.2.2 Truckshop Complex

The truckshop complex will be approximately 1 km north-west of the primary crusher, within the maintenance complex, adjacent to the bonded customs storage yard to the north-east and the main fuel storage facility to the south-east. It will cover a land area of approximately 225 m x 175 m, or 4 ha. This space will be occupied by outdoor facilities and three self-contained structural steel, pre-engineered buildings designed with the required facilities for repair, maintenance, and rebuild of the open pit mining equipment; storage space for spare parts and consumables; and administration offices. No mine personnel change facilities are included in this complex.

The maintenance building proper will have ground floor space for 10 service bays and related workshops and a second-floor mezzanine for IT, office, and meeting rooms.

Cast rails will be embedded in the floor of two of the repair bays for protection when repairing tracked equipment. Serviced with parallel, dual-rail 100 tonne and 25 tonne overhead cranes, the bays will be sized to provide for the full dump height of a 360 tonne capacity haul truck. (The current open pit mine plan is based on the use of the smaller Komatsu 930-4SE, 290 tonne, haul trucks). A 6 m aisle-way down the centre of the building, between the bays, will provide forklift access directly from the tool crib and warehouse.

Space is provided for lube storage, vehicle wash, and a welding/machine/tire shop. A pipe rack will connect the shop to the lubricant storage building. Recovery systems and holding tanks will be provided to store waste oil and coolant products for recycling or disposal.

The wash bay will be a stand-alone building north-east of the workshop. Floor- and catwalk-mounted monitors will provide high-pressure water streams to thoroughly wash the mobile equipment. Wash water will be recirculated through a large sump outside the building, where solids will settle out before the water is recycled back to the wash bay.

Any oil in the recycled water will pass through an oil skimmer to remove hydrocarbons prior to re-use.

18.1.2.3 Administration Building

The administration building will serve as the central, primary office space for all operations staff on site, including executive, supervisory, and support personnel. The building will be equipped with all necessary communication support and will be the main data centre for the site, including most of the ICT management and storage equipment. An emergency response area in the main conference room will serve as the command point in the case of an area-wide emergency. The building will be two-story, pre-engineered, and steel-framed, with a total floor area of 5,000 m². The building will be cooled in summer by the air-conditioning system and heated in winter by the same system using hot water rather than by chilled water. The data centre will need to be cooled in both summer and winter.

18.1.2.4 Mine Dry

The mine dry is a centralized, permanent operations facility whose primary purpose is to provide all site staff with a place to shower and change into work clothes and back into clean street clothes. Spaces are also available for various other activities such as office/meeting room, laundry service, and muster rooms where mining crews will assemble to receive shift instructions before departing for their respective work sites.

The male change rooms will consist of “clean” side and “dirty” sides with approximately 1,200 wire mesh lockers and bench-seating areas on each side, separated by showers and other toilet amenities designed to support the largest number of staff per shift change. A separate women’s change room will accommodate approximately 125 women. VIPs and visitors will have access to a separate room with single lockers that can support up to ten people at a time.

18.1.3 Water Systems

18.1.3.1 River Diversion

The Umdai River channel runs through the proposed open pit area and so must be diverted. Subsurface flow in the river channel is constant, but surface flows are also present occasionally, though usually only after heavy rainfall. The proposed diversion system consists of a dam, diversion channel, and subsurface diversion designed to divert all subsurface and surface flows around and to the north of the South Oyu open pit. Potential seepage from the diversion will be cut-off by keying the main dam below the fluvial gravel. The dam and associated hydraulic structures are designed to ICOLD standards. Both groundwater and surface flood flows will be returned to the Umdai River channel downstream.

18.1.3.2 Site Water Supply System

Raw Water

Raw water for project requirements will be met by pumping groundwater to the site from the Gunii Hooloi aquifer basin. The raw water borefield and pipeline designs are at "Issued for Construction" (IFC) status. Detailed engineering design of the pipeline—inclusive of bore pump stations, collector tank pump stations, break-tank pump stations, emergency storage lagoons, access road, and 6.3 kV power distribution—is underway.

Water Storage and Treatment

A permanent water treatment and bottling plant has been constructed to treat raw water from the Gunii Hooloi borefield to drinking (potable) and domestic water standards.

One section of the building houses two raw/firewater storage tanks, two domestic water tanks, the camp fire water pump station, and domestic water pump station. Each raw water/firewater tank has a capacity of 400 m³ for a total volume of 800 m³, including a combined independent firewater capacity of 324 m³. The remaining water can feed the treatment plant for up to 10 hours at full demand rate in the event of a loss of raw water supply. The two domestic water tanks have a combined capacity of 800 m³, sufficient to supply water at full demand for 16 hours in the event of a loss of raw water supply.

The other area of the building houses the water treatment plant, bottled water storage, a laboratory/office, equipment storage, and chemical storage. Waste water will be automatically and regularly discharged into the sewer system by a dedicated pump station in this room.

Potable water intended for routine drinking will meet Mongolian Standards and WHO guidelines for drinking water quality. Raw water will be pumped to the treatment plant for dosing, multi-media and granular-activated carbon filtration, micro-filtration, reverse osmosis treatment, ultraviolet sterilization, and ozone disinfection prior to bottling in 20 L bottles and onward distribution. The total output of the treatment facility is 8 m³/h.

Domestic water for use in washing, cleaning, flushing toilets, etc., although not intended for routine drinking, will be treated to potable standards. Domestic water will be treated at an average flow rate of 70 m³/h (peak rate 125 to 150 m³/h) before distribution to the construction camp and the rest of the Oyu Tolgoi site via utility pipes.

Firewater Distribution System

Firewater storage tanks will be installed in the concentrator, and pump stations and dedicated fire mains complete with hydrants will be established in the main functional areas of the site. The fire mains also serve sprinkler systems and fire hose reels at the warehouse, operations camp, north gatehouse, and concentrator office. Fire extinguishers will be provided at all facilities as a first line of defence. Fire detection and alarm systems will be installed at key facilities and will report alarms to the mill area control room in the process plant or to the main gatehouse, which will be manned 24 h/d.

Wastewater Treatment Plant

An existing wastewater treatment plant, installed within the construction camp, was upgraded from 800 equivalent people (EP) capacity to 4,000 EP capacity in 2006 and is now fully operational. All sewage generated on site will either be pumped directly to the plant or transported there by truck for treatment by mechanical biological activation using a sequencing batch reactor (SBR). The plant is designed for an inflow of 600 m³/d at a daily pollution load of 240 kg BOD₅. Maximum inflow is expected to be between the hours of 5 pm and 9 pm each day.

Treated wastewater will eventually be recycled into the industrial processes to minimize water make-up from natural resources. Sludge will be aerobically stabilized in storage tanks and will require no further processing. Thickened sludge will be finally disposed of in the waste management centre.

18.1.4 Information and Communications Technology (ICT) Systems

Communications systems throughout the Oyu Tolgoi infrastructure and operations facilities will be state of the art complete with information, security, data, and voice communications. The principal components of the ICT systems will include a fibre-optic communications backbone, local area network (LAN), and Voice over Internet Protocol (VoIP) system to support the voice mail system and auto attendant function. Security systems will consist of a closed-circuit television (CCTV) system connected to the local server and control centre and access security system based on card access for building entrance, security turnstiles, vehicle gates, and fence surveillance.

The fire alarm system will consist of thermal flame detectors, ionization smoke detectors, and manual pull stations in site-wide fire-detection zones. The central fire alarm panel will be installed in the north gatehouse, and local fire alarm panels will be provided in ICT rooms or electrical rooms throughout the plant, interconnected via the Ethernet LAN network.

A digital trunk radio system will permit voice communications between individuals carrying hand-held radios, base stations, and mobile vehicles. A designated emergency channel, GPS tracking, and monitor display of individual radios will allow users to notify support authorities in the event of an emergency or safety issue.

Additional communications and control systems will be provided for the open pit and underground mines. The wireless open pit truck dispatch system will include links to each truck, GPS transmitters, automatic vehicle locator display, and two-way radio transceivers. The underground leaky-feeder coax system will provide network connectivity in underground shafts and tunnels, emergency paging systems, personnel and underground rail dispatch and tagging, remote control of rails and LHDs, and Gai-tronics phones in rescue chambers.

18.1.5 Other Support Facilities and Utilities

- Gatehouse– The north gatehouse complex will be constructed at the northern boundary of the mine lease area and serve as the main security control point for the Oyu Tolgoi mine. The complex will be a single-story building with facilities for a security and logistics checkpoint, site-wide security monitoring, and fire response. Safety and orientation sessions for new employees and/or visitors, as well as special community relations functions, will be held here. A separate building will serve as a bank for employees of the mine and the general public. Depending on final agreements for road construction, a south gatehouse may be required to support concentrate transport traffic.
- Warehouse – The main site warehouse for operations will be constructed adjacent to the construction warehouse to provide heated storage for process equipment parts, spares, critical piping, and valves. A fenced, external storage area will include a dedicated space for hazardous chemicals (HAZCHEM).
- Medical Centre – The Oyu Tolgoi medical centre is designed for both walk-in patients and ambulance access for stretcher patients. It will have a reception area, consultation offices with diagnostic equipment, two resuscitation/treatment beds, two ward beds, and a pharmacy. All controlled prescription drugs will be registered to monitor use and stock level. Hard and electronic copies of medical reference material will be available for all clinic staff and medical providers on site.
- First aid posts are stationed throughout the site to provide initial response in the event of an incident. If required, personnel will be moved to the medical centre under the supervision of senior medical staff. Two Toyota ambulance vehicles fully equipped with emergency beacons/sirens are available.
- The Medical Mongolia clinic in Ulaanbaatar and the International SOS Global Medical Services (GMS) in Beijing will provide ongoing support for patients requiring medical evacuation.
- Fire Station – The fire station will be in the warehouse compound and be in continuous communication with all operations via a digital trunk radio within a 15 km radius. A fully equipped, dedicated fire crew of four fire-fighters will provide fire and emergency response on the project and be trained to meet the Mongolian statutory requirement for Airport Fire Response. The primary fire truck will be fitted with a 7,000 L water tank, an 800 L foam tank, and a roof-mounted foam monitor, supported by 6,000 L water truck. A dedicated fire water supply line with central fire pumps will provide large-fire response support in the main camp.
- Heating – A central coal-fired boiler plant is under construction to provide hot water heating for all surface facilities at the plant site plus the mine air heating systems in the Shaft No. 1 and No. 2 areas. Hot water will be supplied and returned through a primary circulation loop to the various secondary circulation and heating loops complete and dedicated hot water/glycol heat exchangers to provide heating to the end users.
- Underground Utilities – Basic engineering design of underground utility services for raw water, fire water, domestic water, sewer water, return water, and treated water has been completed and battery limit interfaces tentatively agreed among all

- involved parties for infrastructure facility areas, the concentrator, mining, and the power plant.
- **Waste Disposal Facilities**– Wastes generated during the development and operation of the Oyu Tolgoi mine will be collected and disposed of in accordance with Mongolian and international laws. The waste management centre will include a non-hazardous waste landfill, leachate management, and a waste incinerator/oil burner.
 - The landfill will be prepared with a clay-geomembrane composite liner system overlain by a protective geotextile fabric. Exposed waste will be routinely covered with soil to satisfy hygienic and visual needs and surrounded by secure perimeter fencing. Leachate will be collected and gravity-fed to adjacent evaporation ponds fully contained by a composite clay-geomembrane liner system on prepared ground with engineered embankments. Given the arid climate at Oyu Tolgoi, no other treatment will be required.
 - The small amount of suitable hazardous wastes, waste oils, and other combustibles that cannot be directly placed in the landfill will be burned in an incinerator designed and operated to international standards to adequately pacify, neutralize, or vaporize the varieties of waste being burned. Ash residue will be properly disposed of in the landfill.
 - **Fuel Storage** – During the construction period a contract agreement was established with a Mongolian supplier, Petrovis, to store 25 ML of pre-purchased fuel to reduce site storage requirements. The supplier will truck the diesel fuel to site and unload it into storage tank farms. Three diesel fuel storage and dispensing facilities will be strategically placed close to the users:
 - A general vehicle fuel facility immediately south of the main construction camp, containing two 50 kL gasoline fuel tanks and two 400 kL diesel fuel tanks, to cater to four-wheel drive light vehicles, road haulage trucks, mobile cranes, and general purpose site trucks.
 - A second facility with two 400 kL diesel fuel tanks for fuelling the mine fleet south of the truckshop, in an area that provides access for the heavy mine trucks operating in the open pit.
 - A third fuel facility with four 50 kL diesel fuel tanks at the diesel power station just south of the central substation.
 - Construction of the fuel storage facilities and dispensing systems is well progressed and scheduled for completion in December 2011.
 - **Core Storage Facility** – The site engineering department has completed the conceptual design of the pre-engineered building that will serve a core storage shed.
 - **Toyota Workshop** – Toyota has constructed a light-vehicle maintenance facility for its fleet of vehicles at the site. The facility is complete and fully functional.
 - **Fencing** – The entire site boundary will be surrounded by a mine lease perimeter fence with security gates at entrance/exit points. The fence will be a conventional post-and-chain mesh, wide-type, approximately 2 m in high. Supplementary security fencing may be required at individual infrastructure facilities. Temporary

security fencing is already established at the project, but it does not cover the entire site boundary, including some development facilities.

18.1.6 Temporary Construction Facilities

Considering the various construction locations for project facilities, five separate construction camps will be provided, three on-site and two remote, off-site camps, "south" and "north," near the new China access road and the raw water pipeline from the borefield, respectively. In addition, the operations accommodation facility will be brought forward in the schedule to reduce burden on construction facilities from commissioning and operations teams.

The main construction camp is at the north-west corner of the mining site, adjacent to the Umdai River and within 1 km of the centre of the concentrator site to the immediate east. The camp consists of gers arranged in various blocks, together with all the services and amenities of a small, self-contained community. The blocks and other facilities are connected by service roads. The camp is sized to house approximately 2,450 persons and contains an extensive array of dining and recreation facilities.

The Khanbogd camp is sized for about 1,500 people and consists of gers arranged in various blocks, together with the services and amenities, connected by service roads.

The consolidated contractor's camp is about 1 km south of the concentrator and is sized for 4,700 residents. Standard dormitory units, kitchen/mess hall, waste disposal, and recreation facilities are provided.

The remote south camp, for 1,500 workers engaged in constructing the new road to China and the power line from China to the OT site, will consist of dormitory-type units. The north camp will house a maximum of 1,350 workers involved in construction of the raw water pipeline for the Gunii Hooloi borefield to the process plant. This camp will consist primarily of Weatherhaven fabric tents.

The design and construction of the construction water supply facilities are complete and the system is operational. A total of 19 bores have been sunk and are operating. The Oyu Tolgoi Environmental department maintains a demand forecast and also monitors site consumption each week. Water use is monitored by flow meter, and a charge per cubic metre is paid to the Khanbogd governor in accordance with an annually renewable agreement. Chemical testing is carried out quarterly on all bores, and bacteriological testing is completed every two weeks on all bores and at significant points of use such as the camp kitchen.

Detail engineering of the construction warehouse has been completed and has been approved by the Mongolian Expertise Agency. The warehouse has been completed and is in daily use.

Batch plants 1, 2, and 3 are complete and operational. No further batch plants are planned at this stage.

19 MARKET STUDIES AND CONTRACTS

OT LLC will conduct market analysis continuously as the project is developed and afterwards when it becomes a producing operation. CRU Strategies prepared a market study in November 2010 on behalf of potential lenders and OT LLC.

Non-binding memoranda of understanding for concentrate sales to two large Chinese smelters were agreed to during Q3 2011. Contracts are expected to be finalized with these smelters over the coming months. In addition, non-binding agreements on principal sales terms have been reached with two international trading companies; conversion of these agreements to binding contracts is under discussion. Most of the concentrate initially produced at Oyu Tolgoi is expected to be delivered to customers in China.

OT LLC's Sales and Marketing will be supported by Rio Tinto Copper Marketing, led by its Chief Marketing Officer. The marketing team will oversee and execute all sales and marketing activities on behalf of OT LLC.

A value model has been created and will be further informed by the CRU study, customer visits, and market information. The marketing plan is under development and already includes consideration of the following factors:

- Location value to customer compared to imported material landed at Chinese ports.
- Precious metals recovery and payment.
- Length of contract.
- Percentage of off-take to smelters versus traders.
- Percentage of tonnage on contract versus spot.
- Percentage of feed for any one smelter.
- Number of smelters for a given scale of operation.
- Management of concentrate quality and volume during commissioning and ramp-up.
- Alternate off-shore logistics and costs.
- Delivery point and terms.
- Packaging.

A detailed timeline has been developed for marketing, logistics, and contract-to-cash functions.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Government and Community Relations

OT LLC intends to implement systems to manage and mitigate social and environmental impacts of the Project in cooperation with the GOM, local administration and local communities. A series of assessments and studies have been done on social and economic impacts of the Project by independent research institutions, on which Oyu Tolgoi's socio-economic management plans are based. All assessments and planning processes have included local community members and other regional and national-level Mongolian stakeholders.

The socio-economic impacts of the Project and main risks are defined by economic and industrial growth, social sector service delivery needs, influx into the South Gobi region, public infrastructure and urban development, relocation and displacement, social aspects of environmental impacts, local administration capacity, tradition, cultural heritage, and changes of living standards.

20.2 Human Resources and Training Strategy

OT LLC has stated that key to its human resource and training strategy is a corporate vision of ensuring all Oyu Tolgoi staff and contractors meet and exceed international best practice standards. OT LLC, in partnership with all relevant GOM and non-government agencies, intends to work together to ensure that there is a suitably qualified Oyu Tolgoi Project workforce available to meet the timeframe of the Project.

All staff will be covered by a comprehensive human resources policy and will be paid salary packages in compliance with Mongolian Labour Law. For the Project, OT LLC is committed that during operations 90% of the workforce will be Mongolian nationals, and during the construction and expansion periods, this is relaxed to 60%. For all mining activities, the requirement is for at least a 75% Mongolian employee content. Penalties paid for exceeding the allowed percentage of foreign employees under these regulations amount to 10 times the average Mongolian salary (amounting to about \$10,000 per excess foreign national). The penalties are to be paid into a Mongolian Employment Promotion fund, from which contributions can be made to the OT Training Strategy and Plan.

The OT Training Strategy and Plan is planned is to be tabled with the GOM within 150 days of the effective date of the investment agreement. As part of this plan, OT LLC is committed to supporting 150 undergraduate scholarships, 30 of which are to be at international universities, over a 6 year period.

Within 5 years of the commencement of production, OT LLC will use its best endeavours to ensure that no fewer than 50% of its employed engineers shall be Mongolian nationals and, within 10 years of the commencement of production, that no fewer than 70% of its employed engineers shall be Mongolian nationals.

20.3 Occupational Health, Hygiene and Safety

Oyu Tolgoi Health & Safety Management System (H&S MS) has been developed to provide Management with clear direction on health and safety management and to ensure compliance and provide the basis for driving improvements. The Oyu Tolgoi H&S MS consists of 17 elements:

1. Health and safety policy has zero harm as its goal.
2. Training competency and awareness will be facilitated by the use of training matrices, competency-based training for extreme risk tasks, and using knowledgeable trainers.
3. Hazard identification and risk assessment follows a three-tiered approach (pre-task; qualitative and quantitative).
4. Disaster management and recovery is based on a structured threat analysis risk assessment and involves an emergency response team, area emergency coordination, and a disaster management and recovery process.
5. Legal and other requirements – cultivate a culture of compliance with relevant laws and regulations, standards and procedures.
6. Documentation and document control via an established document structure and naming convention.
7. Data and records management consistent with Oyu Tolgoi standards, legislative requirements and privacy laws.
8. Operational control will be facilitated by safe systems of work.
9. Management of change – In accordance with the maximum reasonable consequence associated with a change, the appropriate risk management approach is implemented.
10. Supplier and contractor management to ensure conformance to Oyu Tolgoi H&S standards, procedures, and values.
11. Management review via several performance measurement processes.
12. Communication and consultation – written and verbal.
13. Health and safety management improvement planning at various organizational levels.
14. Measuring and monitoring of exposure and emissions.
15. Performance assessment and auditing – both 3rd and 2nd party (i.e. external and interdepartmental).
16. Organizational resources, accountabilities and responsibilities by active leadership support of health and safety processes.
17. Non-conformance incident and action management to ensure continuous improvement.

20.4 Environmental Studies

Holders of a mining licence in Mongolia must comply with environmental protection obligations established in the Environmental Protection Law of Mongolia, Law of Environmental Impact Assessment and the Minerals Law. These obligations include preparation of a Detailed Environmental Impact Assessment (DEIA) for mining proposals, submitting an annual environmental protection plan, posting an annual bond against completion of the protection plan and submitting an annual environmental report.

OT LLC has posted environmental bonds to the Mongolian Ministry for Nature and Environment and Tourism ("MNET") in accordance with the Minerals Law of Mongolia for restoration and environmental management work required for exploration and the development work undertaken at the site. OT LLC pays to the Khanbogd Soum annual fees for water, sand/gravel and land usage.

OT LLC has, through qualified independent consultants, prepared a DEIA for the Project consisting of three main volumes (as per instructions from the MNET), comprised of the following:

- Volume I: Oyu Tolgoi to Gashuun Sukhait Road (the "OT-GS Road")
- Volume II: Gunii Hooloi Borefield
- Volume III: Mining and Processing Facilities.

Additional separate volumes have been prepared for select facilities, including the Raw Water Supply Pipeline, Temporary Airport, Permanent Airport, Chemicals and the Umdai River Diversion. Internally, an additional volume has been produced for the Coal Fired Power Plant. Summaries of the status of each of the main DEIA volumes, together with other in progress DEIAs, are given below.

Volume (I): OT-GS Road

The first main DEIA document, for the transport corridor running south of the Project to the Chinese border, was submitted in April 2004 and approved in May 2004. An amendment to the approved DEIA was submitted in December 2006 to allow for an alternative road alignment to the Chinese border and approved in March 2007. A further amendment DEIA for the OT-GS Road was submitted to the MNET in February 2011 to reflect the changed entrance location to the Project and reflect new GOM requirements for paved roads and has been approved by the MNET. A further amendment to the OT-GS Road DEIA report is currently being prepared to reflect road alignment adjustments back to northern mine site entrance and around a border military post as well as updated borrow pit and quarry locations.

Volume (II): Gunii Hooloi Borefield

The second main DEIA document, for the Gunii Hooloi Borefield, was submitted and approved in 2005. This DEIA was amended in January 2011 to reflect updated site water demand estimate and updated hydrogeological understanding and reserve approvals in the Gunii Hooloi Borefield and remains valid.

As a supplementary DEIA to the Gunii Hooloi Borefield volume, the Raw Water Supply pipeline DEIA was approved in March, 2010. An amendment to the raw water supply pipeline DEIA is currently being prepared to reflect updated alignments around tree grove and paleontological findings areas.

Volume (III): Mining and Processing

The third main DEIA document incorporates the results of IDP05, and was first submitted in January 2006. Following review by the GOM, the submission was amended in May 2006. An expert committee review of the Mine and Processing DEIA was completed in September 2006, and further information was submitted in November 2006 to address issues raised. The third volume of the DEIA was approved by the MNET in December 2007. An addendum to the Mining and Processing DEIA which incorporates ongoing updates that have occurred to the site facilities was prepared and approved in 2011 that reflects ongoing updates to facilities in the mine licence area and remains valid.

The Undai River Diversion was originally included as part of the approved Mine and Processing DEIA approved in December 2007 and, at the request of the MNET, is now being presented in a separate DEIA report. This Undai River Diversion DEIA has been prepared and is currently under review by the MNET.

Volume (IV): Coal Fired Power Plant

Power Plant DEIA reports have been prepared and approved by the MNET in 2011 for up to 5 (3 initially) x 150 megawatt coal fired steam power plant located on the OT Licence.

Volume (V): Airports

The Temporary Airport DEIA was approved in September 2007 and remains valid. The Permanent Airport DEIA report was approved by the MNET in 2011 and remains valid.

Volume (VI): Others

A number of supplementary DEIA assessments have been performed which cover current development work associated with underground Shafts 1 & 2, waste water treatment, diesel power supply, emulsion plant, chemicals import and usage, and the quarry and batch plant have been prepared and approved by the MNET. An addendum to the Chemicals DEIA was prepared and has been approved by the MNET in 2011 to reflect new chemicals that have been identified for use during operations.

OT LLC has contracted the Institute of Archaeology at the Mongolian Academy of Science to complete archaeological studies of the Project. The studies have resulted in the excavation and removal of sites of historical and cultural significance within the Project area in accordance with the relevant Mongolian laws and customs.

Environmental Social Impact Assessment

Both environmental and social monitoring activities are ongoing for the Project. The revised Environmental Social Impact Assessment (ESIA) is planned for completion in April 2012 to International Finance Corporation and European Bank for Reconstruction and Development standards.

The environmental baseline assessment for the project was prepared by drawing upon the wide range of internal and independent studies that have been prepared since 2003. The existing information was reviewed and assessed for accuracy, consistency and validity. Where additional environmental baseline data became available in 2010 before the draft ESIA was produced, these were incorporated into the ESIA. Further data collection studies have been commissioned and started and commitments have been made in the corresponding management plans to ensure that collection of baseline data continues to improve and that the results of ongoing monitoring will be integrated into updated and revised management plans and procedures.

The baseline chapters presented in the ESIA are, necessarily, a summary of an extensive body of research and assessment that has been ongoing over many years.

21 CAPITAL AND OPERATING COSTS

21.1 IDOP Reserve Case Cost Summary

OT LLC prepared the operating and capital costs presented in IDOP. The operating costs are based on the designs of the mines and plant in IDOP. The cost estimates in IDOP were generally based on actual costs expended to date and on contracts and supplier quotations for the construction and operation of the project and were applied to the design quantities, equipment schedules, labour numbers and consumption rates and productivity assumptions. Labour costs reduce over time as it is assumed that expatriate labour is reduced and more roles are carried out by Mongolian nationals. The methodology for the cost estimation is described in the following section.

In order to provide an analysis of IDOP for the IDOP Technical Report, costs to December 2011 were treated as sunk and escalation was applied to the operating and capital costs base (excluding Phase 1) prepared for the IDOP study to produce the cash flow. The operating and capital costs in IDOP were increased by 8.1% for the IDOP Reserve Case. The capital costs for the Project are detailed in Table 21.2 and Table 21.1 provides a breakdown of operating costs and revenue in the IDOP Reserve Case.

Table 21.1 IDOP Reserve Case Operating Costs and Revenues

	US\$M	\$/t Ore Milled		
	Total IDOP Reserve Case	First 5 Years	First 10 Years	IDOP Reserve Case
Revenue				
Gross Sales Revenue	87,105	65.71	78.41	62.33
Less: Realization Costs				
Concentrate Transport	960	0.45	0.72	0.69
Treatment and Refining	4,453	2.39	3.55	3.19
Government Royalty	4,355	3.29	3.92	3.12
Total Realization Costs	9,768	6.12	8.19	6.99
Net Sales Revenue	77,337	59.58	70.22	55.34
Less: Site Operating Costs				
Mining (all sources)	7,531	6.69	6.46	5.39
Processing	6,311	6.90	5.20	4.52
Tailings	922	0.92	0.79	0.66
G&A	6,738	8.39	6.25	4.82
Entrée JV Fees	539	0.02	0.02	0.39
Government Fees and Charges	2,495	2.84	2.22	1.79
Management Fees	1,789	3.11	1.90	1.28
Total Site Operating Costs	26,325	28.86	22.82	18.84
Operating Margin	51,012	30.72	47.40	36.50

Table 21.2 Total Project Capital Cost– IDOP Reserve Case

US\$M	Phase 1 100 ktpd	Expansion 160 ktpd	Sustaining	Total
Mining				
Southern Oyu	484		490	974
Hugo South	0	0	0	0
Hugo North Lift 1	944	2,828	830	4,601
Hugo North Lift 2	0	0	0	0
Heruga	0	0	0	0
Subtotal	1,428	2,828	1,319	5,575
Processing and Infrastructure				
Concentrator	1,310	645	121	2,076
Infrastructure	1,404		129	1,534
Power Station	17	1,156	96	1,270
Tailings Storage Facility (TSF)	37		50	86
Subtotal	2,768	1,801	397	4,966
Indirects & EPCM				
Construction Indirects/Site Services	433			433
Project Management	434			434
Subtotal	868			868
Construction & Operations				
Construction Owner's Team	305			305
Operations Owner's Team	828			828
Subtotal	1,133			1,133
Mongolian Government Payments				
Mongolian Customs Duties		127	43	170
Mongolian VAT		360	133	493
Subtotal		487	176	663
Total Development Program	6,197	5,115	1,892	13,204

Notes: Phase 1 capital includes only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

21.2 Capital Costs

Capital costs have been allocated into the following categories:

- Mining
 - Open Pit
 - Underground
- Processing and Infrastructure
 - Indirects & EPCM
 - Construction & Operations
- Mongolian Government Payments

Customs and VAT costs in the Phase 1 capital estimates have been included in direct costs and shown separately for the remainder.

21.2.1.1 Open Pit Capital Costs

The open pit mine capital costs include the following:

- Heavy mobile equipment (HME)
- Minor equipment
- Explosives and diesel
- Mining software and systems
- Spare shovel bucket
- Spare FEL bucket
- Truck tray re-liner package
- Dewatering
- Pit dispatch system
- Global positioning system for mine surveying
- Blasting contractor set-up
- Maintenance system
- Pit power distribution
- Training simulator

HME capital costs are based on supplier quotations and allowances for the number of equipment units required. The HME prices include start up spares, delivery to Oyu Tolgoi site, assembly, and commissioning. Prices for small support equipment, such as light vehicles, personal carriers, cranes, and the mine fuel truck, include start up spares, and on site delivery costs, and exclude assembly and commissioning costs. As a result of the update on HME productivity assumptions, additional equipment units were allowed for in IDOP. Final contract prices were agreed for the equipment and the equipment purchase price assumptions were updated, showing a decrease in drilling

and loading and a slight increase in hauling price compared to the previous analysis completed in early 2010.

21.2.1.2 Underground Capital Cost

Underground capital costs are divided into the following two categories:

- Capital Development – includes all preproduction and production build-up development and infrastructure required to initiate production and undercut the cave until the end of 2015.
- Sustaining Capital – includes work required for long-term development, facilities, or systems to sustain full production operations, but not direct production costs.

Capital development began in Year -5 and are planned to end in Year 2 (pre-production). Production starts in Year 2 and production build-up continues through Year 8. The first year of full production is Year 8, and full production continues through Year 16. Production ramps down from Year 17 through Year 21. All planned development following first production, but not included in major life-of-mine infrastructure, is categorized as sustaining development. Many of the tasks falling into the capital development and sustaining capital categories are identical. Task classification into either of the two capital categories is purely a matter of timing relative to production status.

The design of the underground incorporates 4 shafts and 5 raises to ventilate the mine. The intended method of construction for the 5 ventilation raises is by raise boring. This has proven difficult and has suffered delays, the primary reason for schedule slippage has been due to poor ground conditions. Completion of the ventilation raises is on the critical path to production from the underground mine as the raises are required to meet the ventilation requirements for development and provide alternative means of egress should Shaft 1 be unavailable. In order to mitigate and minimise risk to the project production, OT LLC is investigating alternative means of establishing the vertical development required to complete the ventilation infrastructure.

Work to determine the impact of this issue is being carried out. Initial analysis by OT LLC suggests that the underground production could be delayed by around twelve months and that each month's delay could have an NPV value of \$50 M. These investigations are planned to be incorporated into DIDOP and will allow OT LLC to quantify potential costs and changes required to meet the Project schedule.

In addition possible scope changes have been indicated including:

- Additional development and excavation expenditures
- The addition of a shaft No 5
- An additional crew underground
- Increased operating cost due to design changes

The components of capital development and sustaining capital are further defined below.

- Drifting and Excavating
 - Rock handling – truck haulage, conveyor drift, and crosscut development and associated rock handling excavation.
 - Ramps and accesses – access ramp, level access, and level development.
 - Fixed facilities drifting – infrastructure excavation (i.e. material staging area, shops, lunchrooms, etc).
 - Extraction and undercut drifts – perimeter drift, panel drift, drawpoint drift, and drawbell drift development.
 - Ventilation drifts and accesses – dedicated intake and exhaust airways (lateral only).
 - Mass excavations – large excavations required for facilities and installations.
- Shafts and Raises
 - Intake and exhaust raise development.
 - Ore pass and waste pass development.
 - Service and production shaft development.
 - Ventilation shaft development.
- Equipment and Infrastructure
 - Truck haulage infrastructure – supply and installation of truck haulage-related fixed equipment and infrastructure.
 - Ore flow infrastructure – supply and installation of conveyors grizzlies, ore chutes, crushers, etc.
 - Drawpoint construction – drawpoint related excavation and construction activities, including construction of drawpoints, slot raise excavation, drawbell and undercut drilling and blasting, and undercut swell mucking.
 - Mobile equipment – production LHDs, production drills, haulage trucks, development jumbos, bolters, trucks, utility vehicles, etc. This includes the initial supply, materials for major rebuilds, and replacement of mobile units.
 - Fixed facilities infrastructure – ventilation facilities, pumping systems, shops, offices, lunchrooms, storage areas, fuel and lube areas, explosives magazine.

21.2.1.3 Project Capital Costs

The Phase 1 construction of the Phase 1 plant and infrastructure is substantially complete.

21.2.2 Scope of Work

The scope of work for the Phase 2 PEP relates primarily to the underground mine, on which development has commenced. The Phase 2 execution plan will be developed during the underground mine Feasibility Study and will include lessons learned and advancements in ongoing project activities from Phase 1.

The scope of work for Phase 2 includes the following elements:

- Completing infrastructure not finished under Phase 1.
- Conducting work related to the power plant.
- Carrying out regional development work.
- Expanding the concentrator to a capacity of 150 kt/d.

21.3 Operating Costs

- Mining
 - Open Pit
 - Underground
- Processing
- Tailings
- G&A
- Other
 - Government Fees and Charges
 - Management Fees

21.4 Open Pit Operating Costs

The detailed monthly and LOM production plans have been developed to optimize open pit mine production and to determine the necessary equipment hours.

Benchmark productivity assumptions were used to determine the unit costs for the various mining activities. Labour costs are based on the salary and wages structure provided by OT LLC Human Resources. Ore control costs are based on estimates for assaying and required consumables.

The main operating cost driver is haulage, followed by loading and labour. Additional haulage costs result from dividing the TSF into two cells. It is currently assumed that Cell 1 will be constructed with NAF waste material as soon as open pit mining starts. The East waste dump behaves as a temporary storage for NAF waste. The production plan uses this dump as final destination. Therefore, additional haul profiles have been added to the cycle times to account for the travel times between the East NAF dump (TSF stockpile) and the cell construction dikes as mining progresses.

As the pits deepen, haulage costs increase with greater haulage distance. Labour costs reduce over time as it is assumed that expatriate labour is reduced and more roles are carried out by Mongolian nationals. The unit cost in the last production year is higher because mining operation is shut down before the end of the year.

21.4.1.1 Underground Operating Costs

Operating costs include direct and indirect costs associated with the production of ore. Production is defined as pulling ore from drawpoints and delivering it to the main overland conveyor and all activity directly supporting this process. Operating costs include the following items:

- Production
 - Production mucking – production haulage by LHDs from drawpoint to extraction level ore passes.
 - Truck haulage and ore flow – truck operation, chute maintenance, apron feeder maintenance, crushing and conveying system operation, and conveyor spill cleanup.
 - Secondary breaking – fixed and semi-mobile rock breakers, rock breaker operators, breaking of drawpoint oversize, and relieving low and medium hang-ups.
- Ground Repair
 - Drawpoint repair.
 - Sill concrete repair.
 - Ore pass repair.
- Indirects – costs not directly allocated to production.

21.4.1.2 Process Operating Costs

Concentrator operating costs were developed using the template used by AMEC for IDP10. Data for all areas were updated to the latest available as of November 2010. All prices were in Q4 2010 US dollar terms, except where noted. Process plant operating costs have been developed for the main ore sources on an annual basis. The cost areas making up the process operating costs are:

- Workforce
- Reagents and Consumables
- Maintenance Materials
- Concentrator Administration Costs
- Miscellaneous and Contractor Costs

Labour has been modelled on an annual basis from Year 1 to Year 7 (the first full year after expansion). Administration and miscellaneous costs have been considered 100% fixed in both phases, given that the increase in manpower is projected to be much lower than the increase in tonnage in Phase 2, as is normal in expansion of a mature plant.

Power, maintenance consumables and reagents, grinding media and filter cloths have been treated as fully variable on an annual basis for the IDOP Reserve Case.

Concentrator operating costs were calculated based on testwork data, calculation of consumables, and unit rates developed from various sources, including:

- IDP10
- updated quotations for supply of goods and services from AMEC and Rio Tinto Procurement
- OT LLC's and AMEC's database and experience from similar projects
- labour rates and numbers provided by OT LLC

OT LLC has created a detailed labour estimate on a period-by-period basis in order to apply labour costs over the life of mine. This includes changes in personnel numbers as production levels change, and maintenance of at least 90% Mongolian national hire for all of OT LLC in advance of the IA mandated schedule.

21.4.1.3 General and Administration Operating Costs

The G&A costs cover all costs not directly associated with mining and processing. The areas included were:

- Executive Management
- Human Resources and Training
- Business Readiness and Integration
- Health, Safety and Security
- Regional Development & Communications
- Government Relations
- Commercial

The following is a list of cost elements included in the cost estimate of each area:

- Employee Related Costs
 - FIFO Travel
 - FIFO Accommodation
 - Relocation Cost
 - Recruitment Cost
 - R&R Travel
 - Housing Cost
 - In-country Costs
- External Services
- Inward freight

- Consultants
- Legal Fees
- General Consumables
- Mobile Equipment
 - Site Transport
 - UB transport
- Safety, First Aid & Medical Supplies
- Utilities
- Fuel and Energy
- Contractors
- Operating Leases and Rent
- Telecommunications
- Office Accommodation Costs
- Advertising, Promotions and Donations
- Subscriptions & Memberships
- Rates, Taxes & Licences
- Travel Expenses
- Asset Purchases

21.4.1.4 Other Operating Costs

Other costs that are treated as operating costs in the IDOP Reserve Case are:

- Government Fees and Charges
- Management Fees
- Entrée JV fees
- Ex BHP 2% NSR based payment

22 ECONOMIC ANALYSIS

22.1.1 Key Assumptions

IDOP is an update of the Reserve Case previously presented in IDP10. The results of the IDOP Reserve Case show an after tax NPV₈ of US\$11.4 B. The case exhibits an after tax IRR of around 31% and a payback period of around 6 years. An initial capital cost of US\$6.0 B will be expended before production can commence.

The key economic assumptions for the analyses are shown in Table 22.1. Metal prices are assumed to fall from current prices to the long term average over five years.

Table 22.1 Economic Assumptions

Parameter	Financial Analysis Assumptions	
Discount Rate	8.00%	per annum
Copper Price	\$2.90 average (\$2.85 long-term)	per lb
Gold Price	\$1,272 average (\$1,200 long-term)	per oz
Silver Price	\$17.41 average (\$16.60 long-term)	per oz
Treatment Charges	\$70.00	per dmt concentrate
Copper Refining Charge	\$0.070	per lb
Gold Refining Charge	\$5.00	per oz

22.1.2 Investment Agreement and Taxation Assumptions

Both the process of negotiation and the final agreement of the IA presented an opportunity to confirm how the laws of Mongolia should be interpreted in their application to the Project and provided for some specific terms to apply to the Oyu Tolgoi Project. For OT LLC, the agreement has provided the confidence in the stability of the terms the Project will operate under and reliably assess its intended investment in the Project. The agreement itself is effective for an initial term of 30 years with an available extension of a further 20 years.

In accordance with the requirements outlined in the 2006 Minerals Law of Mongolia, upon execution of the IA and the fulfillment of all conditions precedent, the GOM has become a 34% shareholder in OT LLC through the immediate issue of OT LLC's common shares to a shareholding company owned by the GOM. Upon a successful renewal of the IA after the initial 30 year term, the GOM also has the option to increase its shareholding to 50%, under terms to be agreed with Ivanhoe Mines at the time.

A number of conditions precedent were set down in October 2009 and were required to be met before the IA terms came into effect. These were met and confirmed by the GOM in March 2010, triggering the issue of the GOM's equity share in the Project and bringing the IA into full effect.

Despite its role as an equity owner, the GOM will not be required to contribute to the initial capital cost of the Project. Ivanhoe Mines, as the parent company, retains the right to fund the Project by way of a combination of debt, redeemable preference shares and common shares provided the debt to equity ratios fall within the 3:1 ratio required by Mongolian law.

In the case of shareholder debt, loans (including existing shareholder loans at the time of the agreement) initially attracted an interest rate of 9.9% (real) per annum with corresponding adjustments to be made to the outstanding balance to reflect increases in US CPI during each period. The coupon rate applicable to redeemable preference shares was to be 9.9% (real) and carry the same escalation terms. All principal and interest outstanding on shareholder debt, outstanding coupon payments on redeemable preference shares and the face value of those redeemable preference shares must be paid in full prior to any dividends on common shares being paid.

In 2011 an Amended Shareholders Agreement was concluded which reduced the applicable rate from 9.9% to LIBOR plus 6.5%. In addition, an in principle agreement was reached to convert the balance of preference shares into ordinary shares. Both adjustments were to take place based on the 31 January 2011 balances, although the preference share conversion had not occurred by 31 December 2011.

Under the authority of the Shareholders Agreement, Ivanhoe has the right to act as or appoint a Management Team to oversee the construction and operation of Oyu Tolgoi. The management team is compensated with a Management Services Payment equal to 3% of total operating and capital costs prior to commencement of operations and 6% of operating and capital costs during operations. This payment is included in the economic analysis as a project expense and is confirmed as tax deductible in the IA.

OT LLC is required to achieve commencement of production within 7 years of the effective date of the IA.

Under the terms of the IA, a range of key taxes has been identified as stabilized for the term of the agreement at the rates and base currently applied. The following taxes comprise the majority of taxes and fees payable to the GOM under Mongolian law and are shown with their stabilized rates:

- Corporate income tax 25%
- Royalties 5% (gross sales value)
- Value added tax 10%
- Customs duties 5%

In accordance with the Excess Profits Tax invalidating law, as from 1 January 2011, the taxpayer will not be subject to the excess profits tax or any similar windfall tax.

OT LLC is also only subject to those taxes currently listed in the General Taxation Law and not taxes introduced at any future date. These taxes are collectively noted as non-discriminatory taxes and as such cannot be imposed on OT LLC in any manner other than that applied to all taxpayers. OT LLC may also apply to take advantage of

any future law or treaty that comes into force and which would apply any rates lower than those specified in the IA.

The GOM recently enacted amendments to the legislation governing the carry forward of income tax losses. The loss carry forward period has been extended to 8 years and if sufficient, can be applied to offset 100% of taxable income. This contrasts with the previous law in which losses carried forward for 2 years and were subject to a 50% limit.

The agreement also provides OT LLC with the benefit of a 10% tax credit for all capital investment made during the construction period. The amount of this credit can be carried forward and credited in the three subsequent profitable tax years. It is noted in the agreement that if VAT payments, which are currently non-refundable, become refundable in the future, the availability of the investment tax credit will cease from that point. In that event, past earned investment tax credits will still be applied.

22.1.3 Operating Assumptions

Although it has a requirement to make its self discovered water resources available to be used for household purposes, it is confirmed in the agreement that OT LLC holds the sole rights to use these water resources for the Project. The contract for the utilization of water with the GOM water authority is in effect for 30 years with subsequent 20 year periods of renewal.

The supply of power has been recognized as being critical to the execution of the Oyu Tolgoi Project in the IA. OT LLC has been given the right to import power initially but must secure power from sources within Mongolia from the fourth year of operation.

OT LLC also has the right to construct roads for the transport of its product and airport facilities to suit the Project's needs. The GOM has committed to providing OT LLC with non-discriminatory access to any railway constructed between Mongolia and China if such a railway is constructed.

22.1.4 Project Results

A summary of the IDOP Reserve Case project financial results is shown in Table 22.2 and the mining production statistics are shown in Table 22.3.

The estimates of cash flows of the Project have been prepared on a real basis based at 1 January 2012 and discounted back to a current day NPV at a rate of 8%. Metal prices used for the analysis are copper \$2.90/lb (\$2.85/lb long-term), gold \$1,272/oz (\$1,200/oz long-term) and silver \$17.41/oz (\$16.60/oz long-term).

Table 22.2 IDOP Reserve Case Financial Results

		Before Taxation	After Taxation
Net Present Value (US\$M)	Undiscounted	41,080	33,105
	5.0%	20,435	16,773
	6.0%	17,893	14,730
	7.0%	15,693	12,953
	8.0%	13,783	11,403
	9.0%	12,119	10,046
	10.0%	10,665	8,856
Internal Rate of Return		32%	31%
Project Payback Period (Years)		6.14	6.14

Table 22.3 IDOP Reserve Case Mining Production Statistics

		IDOP Reserve Case	5 Year Average	10 Year Average	IDOP Average
Quantity Ore Treated	Mt	1,398	34.0	44.5	51.8
Copper Feed Grade	%	0.94	0.69	1.09	—
Gold Feed Grade	g/t	0.35	0.58	0.49	—
Copper Recoveries	%	87%	85%	85%	—
Gold Recoveries	%	79%	79%	79%	—
Copper Concentrate	Mt (dry)	37.62	0.69	1.29	1.39
Copper Concentrate Grade	%	30%	30%	33%	—
Contained Metal in Concentrate					
- Copper	Mt	11.47	0.20	0.43	0.43
- Copper	Mlb	25.28	0.45	0.94	0.94
- Gold	Moz	12.35	0.50	0.57	0.46
- Silver	Moz	77.28	1.54	2.88	2.86

IDOP Reserve Case Processing and concentrate and metal production are summarized in Figure 22.1 and Figure 22.2 respectively. There is a minor difference between the inventory used for the financial model and the Mineral Reserve. The difference is approximately 4 Mt out of 1.4 bt. This difference is due to differences in reporting from the mine designs (used for the Mineral Reserve), reporting in the production scheduling prepared by OT LLC, stockpile balances and rounding errors. The difference is not significant and if removed would not result in a material difference in the economic analysis.

Figure 22.1 IDOP Reserve Case Processing

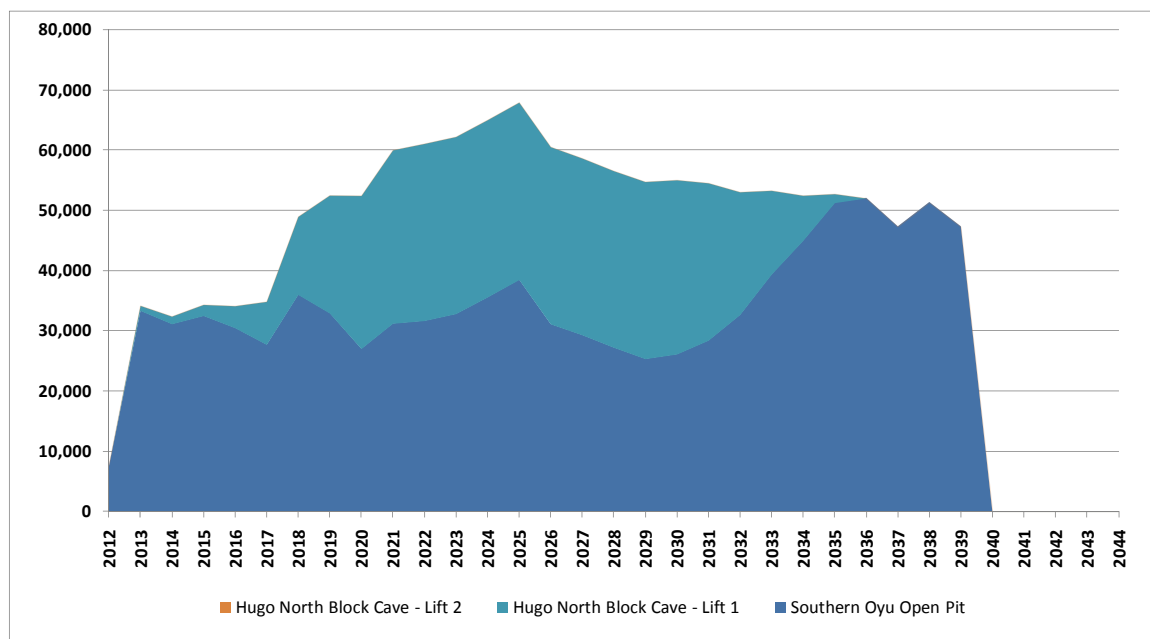
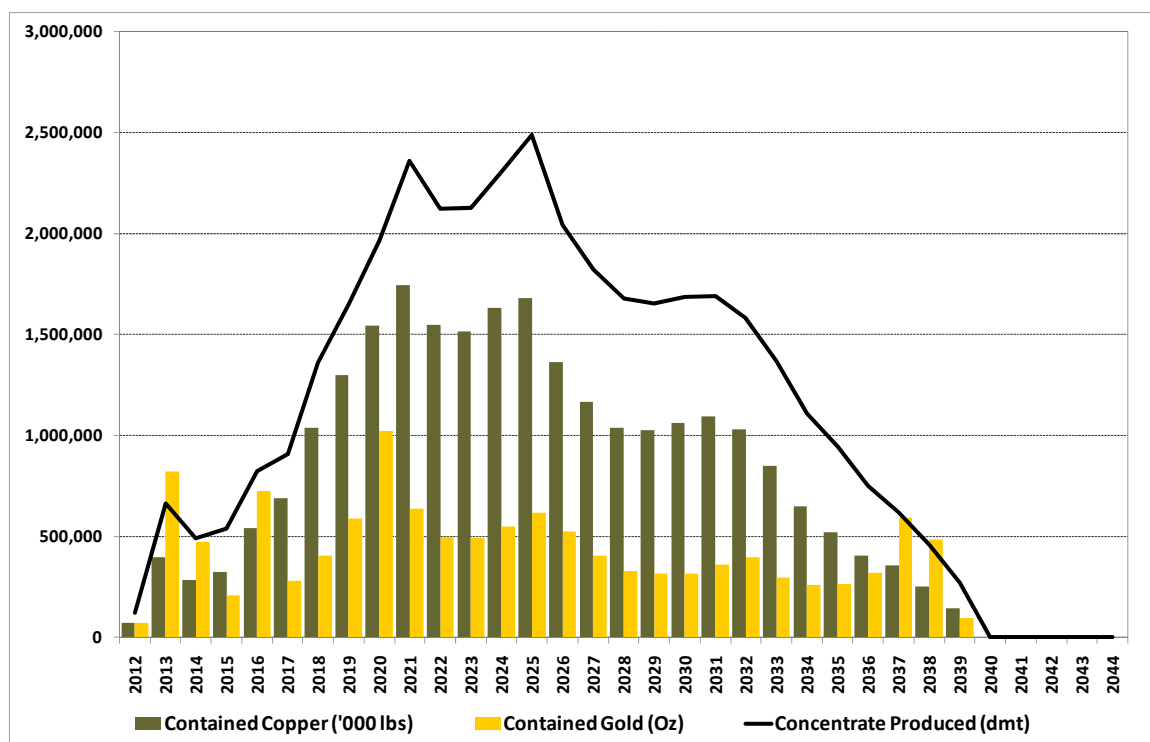


Figure 22.2 IDOP Reserve Case Concentrate and Metal Production



Mine site cash costs are shown in Table 22.4. Those costs relating to the direct operating costs of the mine site, namely:

- Mining
- Concentration
- Tailings
- General and administrative (G&A) costs
- Government fees and charges (excluding corporate taxation)
- Management fees

In addition, realization costs are shown which depict the actual realizable value of Payable Copper produced after accounting for the transport treatment and royalties payable on these sales.

Table 22.4 IDOP Unit Operating Costs by Copper Production

	\$/lb Payable Copper		
	IDOP Reserve Case	First 5 Years	First 10 Years
Mine Site Cash Cost	1.01	2.17	1.05
TC/RC, Royalties and Transport Cash Cost	.40	0.58	0.47
Total Cash Costs Before Gold Credits	1.48	2.75	1.52
Gold Credits	(0.62)	(1.73)	(0.82)
Silver Credits	(0.05)	(0.08)	(0.05)
Total Cash Costs After Credits	0.81	0.95	0.65

Notes: Payments made directly to Ivanhoe from the Project, principally the management fee as specified in the terms of the IA are included in the Mine Site Cash Cost per pound of payable copper. These payments due to Ivanhoe average around 0.10 \$/lb.

The revenues and operating costs have been presented in Table 22.5, along with the net sales revenue value attributable to each key period of operation.

Table 22.5 IDOP Reserve Case Operating Costs and Revenues

	US\$M	\$/t Ore Milled		
	Total IDOP Reserve Case	First 5 Years	First 10 Years	IDOP Reserve Case
Revenue				
Gross Sales Revenue	87,105	65.71	78.41	62.33
Less: Realization Costs				
Concentrate Transport	960	0.45	0.72	0.69
Treatment and Refining	4,453	2.39	3.55	3.19
Government Royalty	4,355	3.29	3.92	3.12
Total Realization Costs	9,768	6.12	8.19	6.99
Net Sales Revenue	77,337	59.58	70.22	55.34
Less: Site Operating Costs				
Mining (all sources)	7,531	6.69	6.46	5.39
Processing	6,311	6.90	5.20	4.52
Tailings	922	0.92	0.79	0.66
G&A	6,738	8.39	6.25	4.82
Entrée JV Fees	539	0.02	0.02	0.39
Government Fees and Charges	2,495	2.84	2.22	1.79
Management Fees	1,789	3.11	1.90	1.28
Total Site Operating Costs	26,325	28.86	22.82	18.84
Operating Margin	51,012	30.72	47.40	36.50

The Phase 1 direct capital investment program since project inception is depicted in Table 22.6 as the capital cost required prior to reaching commercial production at the end of 2012 (pre-production). The total cash cost estimated to be expended at the end of 2012 is estimated at \$US6.0 B which includes \$1.0 B invested in underground development.

The Total Project direct capital costs required from 1 January 2012 are shown in Table 22.7, divided between the investment required to commence production at 100 ktpd and complete Shaft No 2 (Phase 1), the concentrator expansion to 160 ktpd, power station and completion of the underground mine to 54 ktpd (Phase 2), and the sustaining capital to support ongoing operations.

The Phase 1 capital shown in Table 22.6 and Table 22.7 include only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

The changes in financial results for a range of copper and gold prices are shown in Table 22.8. Cumulative cash flow for the IDOP Reserve Case is depicted in Figure 22.3 and a complete cash flow is provided in Table 22.9.

Figure 22.3 IDOP Reserve Case Cumulative Cash Flow

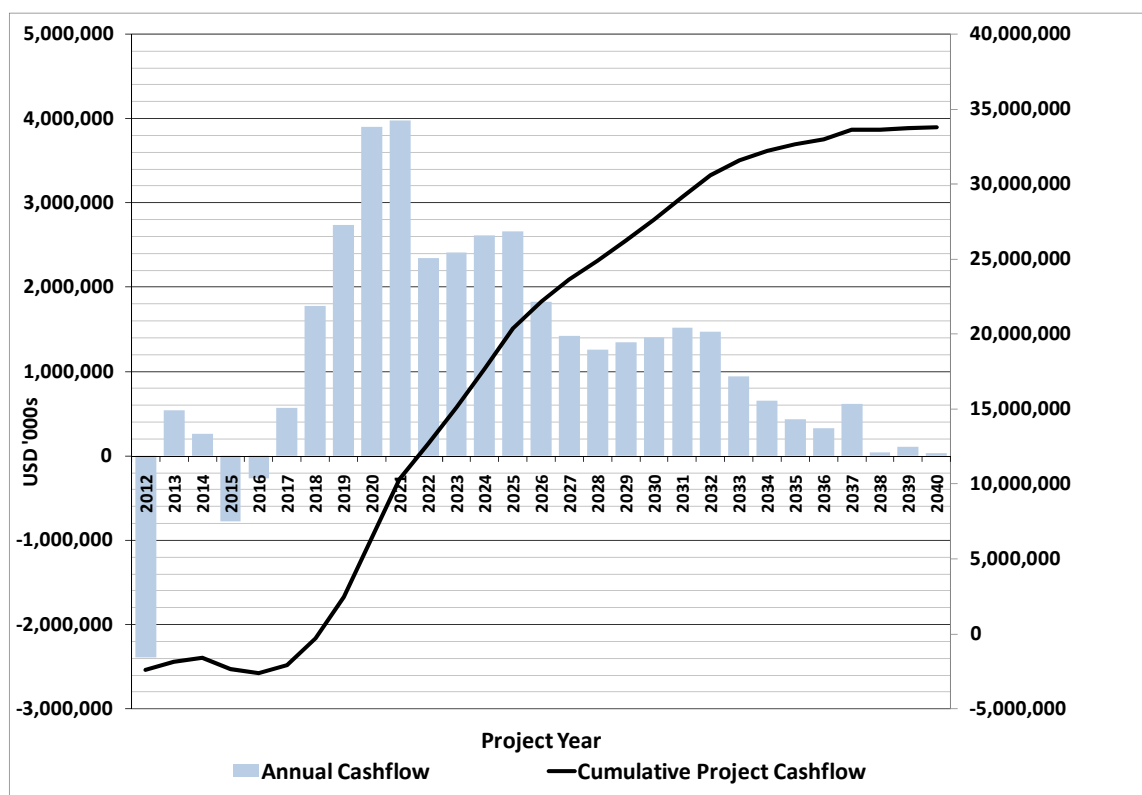


Table 22.6 Pre-Production Capital Cost

	Phase 1		Phase 2	Total
US\$M	2000 to 2011	Costs Remaining		
Mining				
Southern Oyu	308	201	–	509
Hugo South	–	–	–	–
Hugo North Lift 1	548	299	184	1,032
Hugo North Lift 2	–	–	–	–
Heruga	–	–	–	–
Subtotal	855	501	184	1,541
Processing and Infrastructure				
Concentrator	950	260	–	1,210
Infrastructure	945	240	–	1,186
Power Station	17	–	–	17
Tailings Storage Facility (TSF)	11	27	–	38
Subtotal	1,924	528	–	2,452
Indirects & EPCM				
Construction Indirects/Site Services	430	73	–	503
Project Management	332	134	–	466
Subtotal	762	207	–	969
Construction & Operations				
Construction Owner’s Team	137	120	–	256
Operations Owner’s Team	287	486	–	773
Subtotal	424	605	–	1,029
Mongolian Government Payments				
Mongolian Customs Duties	–	–	5	5
Mongolian VAT	–	–	14	14
Subtotal	–	–	19	19
Total Development Program	3,966	1,841	204	6,010

Notes: Phase 1 capital includes only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

Table 22.7 Total Project Capital Cost– IDOP Reserve Case

US\$M	Phase 1 100 ktpd	Expansion 160 ktpd	Sustaining	Total
Mining				
Southern Oyu	484	–	490	974
Hugo South	0	0	0	0
Hugo North Lift 1	944	2,828	830	4,601
Hugo North Lift 2	0	0	0	0
Heruga	0	0	0	0
Subtotal	1,428	2,828	1,319	5,575
Processing and Infrastructure				
Concentrator	1,310	645	121	2,076
Infrastructure	1,404	–	129	1,534
Power Station	17	1,156	96	1,270
Tailings Storage Facility (TSF)	37	–	50	86
Subtotal	2,768	1,801	397	4,966
Indirects & EPCM				
Construction Indirects/Site Services	433	–	–	433
Project Management	434	–	–	434
Subtotal	868	–	–	868
Construction & Operations				
Construction Owner's Team	305	–	–	305
Operations Owner's Team	828	–	–	828
Subtotal	1,133	–	–	1,133
Mongolian Government Payments				
Mongolian Customs Duties	–	127	43	170
Mongolian VAT	–	360	133	493
Subtotal	–	487	176	663
Total Development Program	6,197	5,115	1,892	13,204

Notes: Phase 1 capital includes only direct project costs within the scope of Phase 1 as approved by the OT Technical Committee and update at 31 December 2011. They do not include non-cash shareholder interest, management fees, foreign exchange gains or losses, tax pre-payments, T Bill purchases or exploration phase expenditure. All GOM payments for the Phase 1 capital program have been included within the direct or indirect project cost area to which they relate.

Table 22.8 After Tax Metal Price Sensitivity – IDOP Reserve Case

After Tax Values	Gold (\$/oz)							
Copper (\$/lb)	800	1,000	1,200	1,500	1,750	2,000	2,500	3,000
Project Net Present Value at 8% (\$M)								
2.00	4,998	5,563	6,169	7,085	7,779	8,441	9,750	11,053
2.50	8,266	8,791	9,310	10,103	10,765	11,418	12,719	14,031
2.85	10,350	10,875	11,403	12,188	12,840	13,496	14,808	16,114
3.00	11,246	11,769	12,293	13,079	13,736	14,392	15,702	17,004
3.50	14,219	14,748	15,277	16,065	16,716	17,367	18,668	19,977
4.00	17,206	17,729	18,250	19,031	19,682	20,338	21,650	22,959
4.50	20,175	20,699	21,224	22,011	22,667	23,322	24,627	25,932
5.00	23,159	23,684	24,207	24,990	25,643	26,296	27,599	28,903
Project Internal Rate of Return (IRR%) After Tax								
2.00	21%	22%	23%	25%	26%	27%	29%	31%
2.50	26%	27%	28%	29%	30%	31%	33%	35%
2.85	29%	30%	31%	32%	33%	34%	36%	37%
3.00	30%	31%	32%	33%	34%	35%	37%	38%
3.50	34%	35%	35%	36%	37%	38%	40%	41%
4.00	37%	38%	39%	39%	40%	41%	42%	44%
4.50	40%	41%	41%	42%	43%	44%	45%	46%
5.00	43%	43%	44%	45%	45%	46%	47%	48%
Project Payback (Years)After Tax								
2.00	7.48	7.31	7.16	6.95	6.74	6.55	6.25	6.00
2.50	6.76	6.61	6.48	6.31	6.18	6.06	5.79	5.53
2.85	6.35	6.25	6.15	6.02	5.88	5.75	5.51	5.31
3.00	6.21	6.12	6.04	5.88	5.75	5.63	5.41	5.23
3.50	5.81	5.72	5.63	5.51	5.41	5.32	5.16	5.03
4.00	5.48	5.41	5.34	5.25	5.18	5.11	4.98	4.75
4.50	5.25	5.19	5.14	5.07	5.01	4.92	4.72	4.55
5.00	5.08	5.03	4.98	4.86	4.77	4.69	4.53	4.39
Total Cash Costs (after Credits) – Life of Project								
2.00	0.90	0.82	0.74	0.63	0.53	0.43	0.24	0.04
2.50	0.94	0.86	0.78	0.66	0.57	0.47	0.28	0.08
2.85	0.96	0.88	0.81	0.69	0.59	0.50	0.30	0.11
3.00	0.97	0.90	0.82	0.70	0.60	0.51	0.31	0.12
3.50	1.01	0.93	0.86	0.74	0.64	0.54	0.35	0.16
4.00	1.05	0.97	0.89	0.78	0.68	0.58	0.39	0.19
4.50	1.09	1.01	0.93	0.81	0.72	0.62	0.43	0.23
5.00	1.12	1.05	0.97	0.85	0.75	0.66	0.46	0.27

Table 22.9 IDOP Reserve Case Cash Flow

US\$M	Year									Total
Cash Flow Statement	-1	1	2	3	4	5	6 to 10	11 to 20	21 to Life of Mine	
Gross Revenue	386	2,913	1,766	1,438	2,795	2,255	23,728	40,192	11,633	87,105
Realization Costs	36	238	156	145	256	245	2,604	4,744	1,344	9,768
Net Sales Revenue	350	2,675	1,610	1,293	2,539	2,009	21,124	35,448	10,289	77,337
Site Operating Costs										
Mining	73	174	158	168	324	313	1,737	3,396	1,188	7,531
Processing	59	260	250	252	251	161	1,140	2,353	1,585	6,311
Tailings	0	37	30	21	25	43	193	380	192	922
G&A	221	341	276	257	291	261	1,353	2,397	1,341	6,738
Other	0	0	0	1	1	1	4	531	0	539
Total Site Operating Costs	353	812	714	699	891	780	4,427	9,058	4,307	22,040
Operating Surplus / (Deficit)	(3)	1,863	896	594	1,648	1,230	16,696	26,390	5,982	55,296
Indirect Costs (inc Depreciation)	12	224	195	224	271	207	933	1,835	983	4,884
Net Profit Before Income Tax	(15)	1,638	701	371	1,377	1,023	15,763	24,555	4,999	50,412
Income Tax	0	0	0	0	0	0	804	5,744	1,428	7,975
Net Profit After Income Tax	(15)	1,638	701	371	1,377	1,023	14,959	18,811	3,571	42,437
Capital Expenditure	2,674	968	653	1,234	1,511	610	866	1,123	207	9,845
Other Cash Adjustments	(302)	135	(237)	(128)	26	(65)	179	(304)	182	(514)
Net Cash Flow After Tax	(2,387)	535	286	(735)	(160)	478	13,914	17,993	3,182	33,105

23 ADJACENT PROPERTIES

This section is not used.

24 OTHER RELEVANT DATA AND INFORMATION

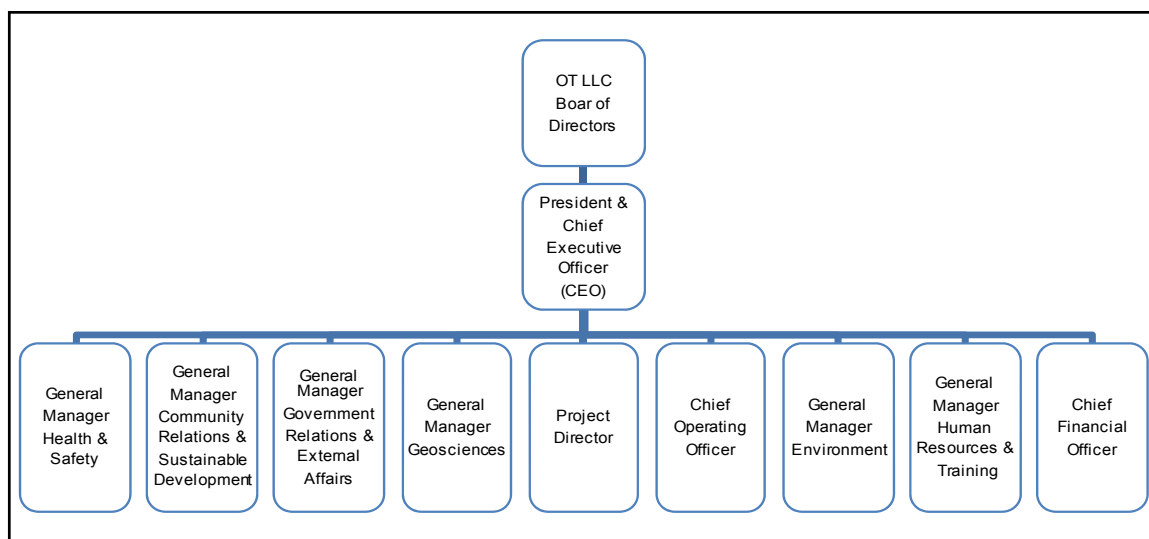
24.1 Management

The operations plan sets out the various roles and responsibilities required to successfully manage the operating aspects of the Oyu Tolgoi project. It includes the overall workforce development strategy and other operations planning aspects related to supply and logistics; environment, health, and safety policies and procedures; and concentrate shipment and handling. OT LLC has developed an Operational Readiness Plan to ensure that all facilities and personnel are in place to support the successful transition from project construction to mine operations. The operations plan is current as of December 2010 and may be subject to change as planning advances for operations.

24.1.1 The Organization

OT LLC's aim is to develop a proficient and reliable workforce possessing the productivity and skills associated with international best practice to support its overall goal of high operations efficiency. The organizational structure of the Oyu Tolgoi project is expected to evolve from a more hierarchical structure to one that is a flatter, more global-type structure. This evolution will be tied to the development of professional management ability and advanced skills within the OT LLC workforce. The overall OT LLC management organizational structure is presented in Figure 24.1.

Figure 24.1 Organization Chart for Overall OT LLC Management



24.1.2 Training and Human Resources

OT LLC recognizes that employment, benefits, and training are vital to meeting project goals for operations. OT LLC has made commitments and developed plans related to training and human resources as part of its corporate commitments and stewardship of the project.

Human resources functions for national, regional, and site requirements will be managed centrally in Ulaanbaatar, supported by permanent recruitment offices in Dalanzadgad and Khanbogd, to maintain consistency through the recruitment, hiring, and training processes. Recruitment and training will be linked to the project development schedule and strategy to ensure successful workforce ramp-up.

To meet the needs of construction and operations, fixed rosters have been developed as part of the work rotation strategy. There are four principal rosters for construction and two principal rosters planned for operations. These rosters will guide the work rotation of the OT LLC workforce at site, regional offices, and in Ulaanbaatar.

24.1.3 Workforce Development

Present planning provides for an operations workforce consisting of both expatriates and Mongolian nationals. This is largely owing to a lack of Mongolian exposure to the particular technical and trades skills that will be required for the Oyu Tolgoi operation. It is anticipated that the requirements for an expatriate workforce will decline as the operation matures and Mongolian staff skill levels improve to international best practice standard through training and on-the-job experience. This phased reduction will follow the commitments outlined in the Investment Agreement with the GOM.

24.1.4 Supply and Logistics

The volumes of incoming freight for the operations phase have not yet been established, but quantities will be much lower than those required for construction.

OT LLC's procurement standards and processes have been developed to ensure that the manner in which the procurement function is conducted is transparent and understood by all key stakeholders. The Statement of Procurement Practice (October 2010) sets out a range of expectations and commitments relating to how business will be conducted between the Oyu Tolgoi Procurement team, its internal customers (or other departments within the Oyu Tolgoi business), and suppliers to the business. An annual external audit will ensure OT LLC is at all times meeting the commitments outlined under the Statement of Procurement Practice.

24.1.5 Concentrate Shipment and Handling

Copper concentrate will be sold in-bond free-on-board at a bonded yard on the Chinese side of the border in Ganqimaodao. The proposed copper concentrate transport strategy assumes the existence of a bi-lateral concentrate export/import agreement to allow concentrate to be shipped across the border directly and without having to operate a cross-border shuttle service for full and empty containers.

The Chinese railway is planning to construct an extension of its network to Ganqimaodao. Once this rail line becomes operational, copper concentrate will be exported in bulk to a small truck reload facility at Ganqimaodao for those customers requiring truck delivery.

24.1.6 Environment, Health, and Safety

Oyu Tolgoi Health & Safety Management System (H&S MS) has been developed to provide Management with clear direction on health and safety management and to ensure compliance and provide the basis for driving improvements.

24.1.7 Operational Readiness Plan

The detailed Operational Readiness Plan will cover all operating aspects of the Oyu Tolgoi project. It includes an overview of OT LLC's operational framework and operating philosophy and descriptions of the pertinent strategy, plans, and procedures for 21 identified functions across the project, both on- and off-site. The associated strategies are currently under preparation.

24.2 Risk Assessment

Risk analysis was undertaken for IDOP with the assistance of Rio Tinto Technology and Innovation (RT T&I) which updated the previous risk analysis that was conducted in September 2011. The risks identified in IDOP remain current, the summary below includes discussion based on the current status of the Project.

24.2.1 Summary

RT T&I was asked by OT LLC to facilitate a series of risk analysis workshops to assist in the process of identifying and evaluating risks associated with various key aspects of the Oyu Tolgoi project. The workshops were conducted on and between 25 and 29 October 2010 at the Ivanhoe office in Vancouver, Canada. The scope of work for these activities is described in a September 2010 document to Rio Tinto Copper Projects (RTCP)/OT LLC from RT T&I. Key project staff and consultants who were unable to attend the risk workshops at Ivanhoe's Vancouver office were remotely accessed via telecommunications hook-ups, primarily from the project administration office in Ulaanbaatar, Mongolia.

RTCP/OT LLC requested the semi-quantitative risk analysis to identify and analyse the various threats and opportunities associated with the project as viewed and perceived at the time the risk workshops were conducted. The October 2010 risk analysis represents an update to a previous risk analysis conducted by Rio Tinto in September 2009. Earlier risk analyses were also carried out in November 2008 and September 2007. For continuity purposes, all risks identified in the September 2009 risk register were carried over in the current risk register. All September 2009 risks that were deemed to be still applicable were reassessed for likelihood and consequence impacts; if the risk was deemed to be no longer applicable, it was so indicated by being greyed out in the latest risk register.

24.2.2 Key Assumptions

The risk analysis focussed on threats rather than opportunities as the significant opportunities at Oyu Tolgoi were included within the risk analysis assumptions. Five key assumptions were made in the risk analysis of Oyu Tolgoi:

- Business values, in terms of capital expenditures, operating costs, and productivity, can be achieved by the Project.
- The Project will be delivered on time and on a budget that is in an acceptable plus/minus range under the current status of the Project.
- Acceptable safety, health, and environmental performance maintained throughout the Project.
- Maintain excellent relationship with GOM and approval bodies.
- Uphold acceptable management of community and social concerns and expectations of the Project.

24.2.3 Recommendations

To achieve the objectives of the risk analysis, Rio Tinto recommended the following:

- Fully understand the results of the risk analysis and manage the identified risks – particularly the Class III and Class IV risks, which require active management – in a manner that is consistent with the candidate risk-management measure(s) recorded in the risk register.
- Confirm Risk Owners assigned to each risk and ensure that the designated individuals are provided the financial resources, authority, and managerial support to successfully execute the agreed upon candidate risk-management measures. If needed, Risk Owners need to assign Task Owners to assist them with their risk-management duties and responsibilities.
- Appoint a senior-level person as a “Risk Champion” to assure that all assigned Risk Owners fulfil their risk-management responsibilities.
- At the appropriate time, develop implementation plans and schedules for each candidate risk-management measure that has been assigned; particularly, the Class III and Class IV risks, which require active management.
- Conduct periodic and detailed reviews of the risk register, as applicable, to gauge progress and identify and address any gaps in the risk-management program or problems with the proposed risk-management actions that were identified in the risk register. Typically, such reviews are carried out at:
 - A quarterly basis
 - Upon any developments that alter the nature and/or size of the risks, and /or
 - As key project milestones are reached.

Such reviews and updates can be used to evaluate and refine the risk register due to real or anticipated changes implemented since the original risk analysis was completed. This is particularly important for Oyu Tolgoi, since a number of key

assumptions made during the focused risk workshop may need to be altered and or revised as the Project progresses forward at a relatively rapid pace.

24.2.4 Risk Areas

The following key development features identified for IDOP identify the areas for which the main risks have been reported by the risk analysis process:

- Investment Agreement
- Environment
- Marketing
- Water Supply and Management
- River Diversion
- Infrastructure
- Workforce for Operations
- Schedule Risks
- Power Supply
- Financing

24.2.4.1 Investment Agreement and Taxation Assumptions

The risk analysis was carried out prior to the signing of the IA and many of the risks related to unknowns in relation to the terms of the IA at the time of the analysis work and addressing the issues that are now part of the IA conditions precedent. The signing of the IA has allowed many of the issues to be clarified and will provide a better understanding of potential risks as well as identifying opportunities in the next stage of the continuous risk analysis process. The IA has been signed by all parties and became effective at 31 March 2010 as a number of conditions precedent before terms come into effect were completed to the satisfaction of all parties.

24.2.4.2 Environment

To date, no environmental issues have been identified that cannot be managed through normal mining practices. Both environmental and social monitoring activities are ongoing for the Project. The revised ESIA is planned for completion in April 2012 to International Finance Corporation and European Bank for Reconstruction and Development standards.

The environmental baseline assessment for the project was prepared by drawing upon the wide range of internal and independent studies that have been prepared since 2003. The existing information was reviewed and assessed for accuracy, consistency and validity. Where additional environmental baseline data became available in 2010 before the draft ESIA was produced, these were incorporated into the ESIA. Further data collection studies have been commissioned and started and commitments have been made in the corresponding management plans to ensure that collection of baseline data

continues to improve and that the results of ongoing monitoring will be integrated into updated and revised management plans and procedures.

The baseline chapters presented in the ESIA are, necessarily, a summary of an extensive body of research and assessment that has been ongoing over many years.

24.2.4.3 Marketing

Current consumption trends for copper suggest that Oyu Tolgoi is well positioned to sell its product. A diversified strategy will be needed to evaluate and encourage the development of greater smelter capacity in geographically favourable regions of China and elsewhere in Asia as concentrate production at Oyu Tolgoi increases beyond 1.5 Mtpa.

As part of the Investment Agreement, OT LLC has made a commitment to undertake a feasibility study for the construction of a copper smelter in Mongolia to service Oyu Tolgoi. This study must be completed within 5 years of commencement of production. OT LLC is not committed to finance or build a smelter itself under the terms of the Investment Agreement.

Non-binding memoranda of understanding for concentrate sales to two large Chinese smelters were agreed to during Q3 2011. Contracts are expected to be finalized with these smelters over the coming months. In addition, non-binding agreements on principal sales terms have been reached with two international trading companies; conversion of these agreements to binding contracts is under discussion. Most of the concentrate initially produced at Oyu Tolgoi is expected to be delivered to customers in China.

24.2.4.4 Water Supply and Management

The development of a borefield to access groundwater reserves within the Gunii Hooloi aquifer basin has been established as the most cost-effective option to meet the raw water demand for the project. Water from the borefield will be required for process water supply, dust suppression in the mining areas, and potable use.

Close monitoring of aquifer behaviour, calibration of models, and monitoring of relevant surface features will become priorities as operations commence.

Entry into the region by other resource companies and increased demand due to population increases will heighten the competition and sensitivities associated with water supply.

Preliminary estimates of total site water demand for a future processing rate of 160 ktpd are 918 L/s average and 1,081 L/s peak, both figures incorporating a conservative design development contingency of 25%. These estimates assume a Phase 1 peak design processing rate of 110 kt/d at 92% plant availability, 64% tailings density, and 0% mine dewatering. Although some mine water could become available at certain stages of mining, the amount would be limited and has been conservatively excluded from design assumptions. Water consumption in the process plant represents more than 90% of the total demand, Estimates of total site water demand for the Phase 2 processing

rate of 160 kt/d are 918 L/s average and 1,081 L/s peak, both figures incorporating a conservative design development contingency of 25%.

OT LLC currently has an approved GOM licence for the abstraction of 870 L/s from Gunii Hooloi and plans to seek an increase for the project expansion based on revised yield numbers that were developed after pump-testing the borefield in 2011.

The Oyu Tolgoi Environmental Management System, currently under development in accordance with ISO14001 standards, will include water conservation as one of the key areas of continuous improvement during the operations period.

24.2.4.5 River Diversion

The existing Umdai River channel runs through the future open pits. Subsurface flow in the river channel is constant, but surface flow is sporadic, occurring only after heavy rainfall. The river must be diverted to prevent the mining hazard represented by water inflows to the pit and to ensure continued supply to downstream users. Because the near-surface water in the river flows in a perched water table, separated from the underlying groundwater table, the river flow is not expected to be affected by the drawdown associated with mine dewatering.

24.2.4.6 Infrastructure

Oyu Tolgoi is located in an isolated region of Mongolia, with little developed infrastructure. The site is; however, only 80 km from the Chinese border, where resources do exist to support the Project in the energy, transportation, manufacturing, and construction areas. The development plan for the Project is based on the principle of maximizing Mongolian content while involving and realizing the benefits of the resources in China. Balancing the dual objectives is seen as achievable.

It is also assumed that the Chinese road and rail transportation systems can accommodate the movement to site of imported materials required for construction and operations and the shipment off site of all concentrate produced at the process plant. This will need to be confirmed.

24.2.4.7 Tailings Storage Facility

The TSF analysis concluded that the current design is acceptable for reporting purposes and there are opportunities to further simplify the TSF design and operating plan to achieve a lower net present cost for tailings management.

24.2.4.8 Workforce for Operations

OT LLC plans to maximize Mongolian employment levels and achieve a 90% national operating workforce. Mongolian exposure to the specific technical and trade skills required for the Project has been limited, and an extensive training program will be required to achieve this level of local participation.

24.2.4.9 Schedule Risks

IDOP noted that there are risks associated with large scale mining (both open pit and underground) and processing development, construction, and operating that may occur because of changes to the assumptions and could cause delays to the project.

The two most significant schedule risks that have currently been identified are from delays in power and delays to the underground production. OT LLC is actively managing these issues and has plans in place that are mitigating the issues.

Commissioning with ore of the concentrator Line 1 circuit will be dependent on power available from China as there will be insufficient temporary power on site to undertake this activity. If power is available from China by 15 August 2012, the plan date for production of first concentrate would be achieved.

The design of the underground incorporates 4 shafts and 5 raises to ventilate the mine. The intended method of construction for the 5 ventilation raises is by raise boring. There has proven difficult and has suffered delays, the primary reason for schedule slippage has been due to poor ground conditions. Completion of the ventilation raises is on the critical path to production from the underground mine as the raises are required to meet the ventilation requirements for development and provide alternative means of egress should Shaft 1 be unavailable. In order to mitigate and minimise risk to the project production and to OT LLC is investigating alternative means of establishing the vertical development required to complete the ventilation infrastructure.

Work to determine the impact of this issue is being carried out. Initial analysis by OT LLC suggests that the underground production could be delayed by around 12 months and that each month's delay could have an NPV value of \$50 M. These investigations are planned to be incorporated into DIDOP and will allow OT LLC to quantify potential costs and changes required to meet the Project schedule.

In addition possible scope changes have been indicated including:

- Additional development and excavation expenditures
- The addition of a shaft No 5
- An additional crew underground
- Increased operating cost due to design changes

24.2.4.10 Financing

Timely development of the Project depends upon Ivanhoe's ability to maintain an adequate and reliable source of funding. Ivanhoe's existing financial resources are insufficient to meet all anticipated development expenditures required to advance the Project to the commencement of commercial production and Ivanhoe will require access to additional sources of funding during 2012 in order to maintain the project's current development schedule. The scale of the Project is such that a significant amount of financing is required. Such financing would typically be sought from debt and equity capital markets. A comprehensive financing plan for the completion and start-up of the Project, which includes three basic elements: project finance, bridge finance and equity,

remains under discussion with Rio Tinto, but there can be no assurance that an agreement with Rio Tinto will be reached. Failure to reach agreement with Rio Tinto on a comprehensive financing plan could have a material adverse impact on Ivanhoe.

In addition, volatility in debt and equity capital markets and other factors may adversely affect Ivanhoe's ability to implement a comprehensive financing plan particularly as it relates to project finance. Ivanhoe operates in a region of the world that is prone to economic and political upheaval and instability, which may make it more difficult to obtain sufficient debt financing from project lenders. Failure to obtain sufficient additional financing would likely have a materially adverse impact on Ivanhoe's ability to maintain the current development schedule for the Project and could jeopardize Ivanhoe's ability to meet its contractual commitments to third parties in respect of the Project, including those in respect of the Investment Agreement and the Amended Shareholders' Agreement. Ivanhoe may be able to partially mitigate the risk of failing to obtain additional financing by selling some or all of its non-core assets but there is no assurance that the proceeds of any such sale would be sufficient to meet all Oyu Tolgoi Project expenditure requirements.

There are covenants in the Head of Agreement that restrict Ivanhoe's ability to incur debt and the Right of First Offer that applies, except in limited circumstances, to equity financing transactions that Ivanhoe proposes to undertake. Accordingly, Ivanhoe's ability to implement those aspects of its comprehensive financing plan involving project financing and bridge financing is dependent on approvals from Rio Tinto, which approvals cannot be guaranteed. Equity financing does not require Rio Tinto's approval but, depending on the terms of the equity financing, may be subject to Rio Tinto's Right of First Offer, which right expires on 24 October, 2012. Rio Tinto's Right of First Offer may operate to delay Ivanhoe's ability to complete an equity financing and such delays may become protracted if Ivanhoe and Rio Tinto disagree as to how the Right of First Offer should operate or whether the Right of First Offer applies to a particular type of equity financing transaction.

Although Ivanhoe and Rio Tinto are currently engaged in discussions aimed at implementing the comprehensive financing plan proposed by Ivanhoe, Rio Tinto's ability to effectively veto any and all debt financing transactions proposed by Ivanhoe means that any financing plan that is ultimately approved and implemented may be materially different from the comprehensive financing plan that Ivanhoe has presented to Rio Tinto for discussion. If Rio Tinto declines to approve a proposal for third party debt financing and, as an alternative, offers to provide its own debt financing, Ivanhoe may have no effective alternative than to accept Rio Tinto's proposal to provide such financing even if the terms proposed are less favourable than those available from a third party. If Rio Tinto declines to approve a proposal for third party debt financing and does not offer to provide any debt financing itself, Ivanhoe may not be able to obtain any debt financing at all, which could limit Ivanhoe's financing alternatives to equity capital, the terms of which may be dilutive and otherwise unfavourable, and the sale of non-core assets at potentially distressed prices.

24.2.4.11 Power Supply

The Investment Agreement recognized that reliable supply of electrical power is critical to the Project. The agreement also confirmed that Ivanhoe has the right to obtain electrical power from inside or outside Mongolia, including China, to meet its initial electrical power requirements for up to 4 years after Oyu Tolgoi begins commercial production. The agreement further established that (i) Ivanhoe has the right to build or sub-contract construction of a coal-fired power plant at an appropriate site in Mongolia's South Gobi region to supply the Project; and (ii) that all of the project's power requirements would be sourced from within Mongolia no later than 4 years after the start of commercial production.

OT LLC is proceeding with arrangements to ensure that electrical power from China will be available for the start of initial production that presently is expected in the third quarter of 2012. The construction of the transmission towers along the 95 km section of the power line from the Oyu Tolgoi mine site to the Mongolia-China border was completed in October 2011. In early March 2012, Chinese contractors began construction of the power line and switching station as part of the 87 km link in the 220-kilovolt power transmission line to be built from the electrical distribution grid in Inner Mongolia, China, to the China-Mongolia border.

A separate power-purchase agreement establishing a supply arrangement between Mongolian and Chinese authorities is required before Chinese electrical power can be imported into Mongolia. OT LLC will be a party to any agreement for the purchase and supply of electrical power.

Subject to negotiations and a final agreement, the commercial arrangements and power-purchase tariffs are expected to be expedited to ensure that imported power will be available at the Oyu Tolgoi mine site by the third quarter of 2012. In the meantime, additional diesel-powered generating capacity is being provided with expansion planned for April 2012 to meet the project's more immediate requirements during the remaining stages of construction.

In November 2011, the GOM provided OT LLC a cabinet resolution allowing for the future construction by OT LLC of a coal-fired power plant in Mongolia dedicated to the Project. Such a plant would require certain GOM permits, the negotiation of commercial agreements with the GOM and coal suppliers, and the arrangement of financing for the accelerated construction. If the establishment of a power plant is required for the early production at Oyu Tolgoi, the required revisions to the construction schedule for the Project could adversely affect the project's ability to achieve full commercial production in 2013. Although construction of a power plant is expected as part of the Project's future development, there is no provision for a plant in the current capital cost estimates for 2012, and the additional financing that would be required for such a plant is not contemplated as part of the Company's current financing plan. The HOA provides that if construction of a 50 megawatt, or greater, power plant was started before 1 January, 2015, the construction would be funded by loans from Rio Tinto, with 40% of the outstanding balance to be repaid in 2015 and the remainder in 2016.

25 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCE

- The geology of the Project is well understood.
- The deposits are considered to be examples of a copper–gold porphyry system and related high-suffixation types of deposits.
- The deposits are grouped into three areas: Heruga, Southern Oyu and Hugo Dummett.

The exploration program relies strongly on geophysical survey data (IP and magnetics), and other target anomalies still remain within the Project land holdings.

25.1.1 Southern Oyu Deposits

Four deposits are known in the Southern Oyu system, each of which are potentially mineable by open pit methods. A resource estimate was reported above a 0.3% Cu equivalent cut-off in the report by AMEC Minproc, 2006 and above 0.6% Cu equivalent cut-off in Blower, 2006. In the current Technical Report, the resource estimate has not been re-estimated. The base case assumes a Cu equivalent cut-off of 0.3% but includes the results of alternative scenarios down to 0.2% Cu equivalent cut-off. The base case CuEq cut-off grade assumption was determined using cut-off grades applicable to mining operations exploiting similar deposits.

The Southwest Deposit consists primarily of pyrite–chalcopyrite mineralization related to biotite–magnetite alteration, overprinted by chlorite–sericite alteration. Mineralization is characterized by high gold contents with gold–copper ratios (g/t Au to % Cu) of about 1:1 in the main part of the deposit, rising to 3:1 in the core of the system and at depth. Gold in the Southwest Oyu deposit is closely associated with chalcopyrite and occurs intergrown with chalcopyrite, as inclusions and fracture infills within pyrite, or on grain boundaries of pyrite. The deposit is essentially hosted in augite basalts.

Oyu South Deposit mineralization is hosted in quartz monzodiorite in the south-west and basalt throughout the central portion of the deposits. Chalcopyrite is the principal copper sulfide, but in higher-grade areas bornite locally exceeds chalcopyrite. Alteration in basaltic rocks consists of chlorite, biotite, hematite–magnetite, and weak sericite. Quartz monzodiorite has been altered by advanced argillic alteration. Small zones with elevated gold values occur locally.

Mineralization in the Central Deposit is characterized by an upward-flaring high-sulfidation zone that overprints and overlies porphyry-style chalcopyrite–gold mineralization. A secondary enriched supergene chalcocite blanket tens of metres in thickness overlies the high-sulfidation covellite–pyrite zone. The high-sulfidation portion of the Central Oyu deposit contains a mineral assemblage of pyrite, covellite, chalcocite–digenite, enargite, tennantite, cubanite, chalcopyrite, and molybdenite. Dominant host rocks are dacite tuff and quartz monzodiorite. Higher-grade mineralization is associated with disseminated and coarse-grained fracture-filling sulfides in zones of intensely contorted quartz stockwork veins and anastomosing zones of hydrothermal breccias. Chalcopyrite–gold mineralization is dominant on the south and

western margins of Central within either basalt or quartz monzodiorite adjacent to intrusive contacts with basalt. The high-sulfidation part of the Central deposit lacks significant gold. Alteration in the Central deposit shows a close spatial relationship to mineralization and original host lithology. Biotite–chlorite and intermediate argillic alteration coincide with chalcopyrite–gold mineralization within basalt. Advanced argillic and sericite alteration coincide with the high-sulfidation mineralization within quartz monzodiorite, and ignimbrite.

The Wedge Deposit contains a zone of high sulfidation mineralization hosted principally in dacite tuff, grading downward and southward into chalcopyrite mineralization in basalt and quartz monzodiorite host rocks. High-sulfidation mineralization consists of pyrite, chalcopyrite, bornite, enargite, covellite, and primary chalcocite in advanced argillically-altered host rocks. The high-sulfidation mineralization grades downward into chalcopyrite, with lesser bornite within basalt host rocks, and pyrite + chalcopyrite mineralization in quartz monzodiorite. Gold is absent, except locally in drillholes adjacent to the South Fault.

25.1.2 Hugo South Deposit

Copper mineralization at the Hugo South deposit is centred on a high-grade (typically >2% Cu) zone of intense quartz stockwork veining, which in much of the deposit is localized within narrow quartz monzodiorite intrusions and extends into the enclosing basalt and dacite tuff. The intense stockwork zone has an elongate tabular form, with a long axis plunging shallowly to the north-north-west, and an intermediate axis plunging moderately to the east. Copper grades gradually decrease upwards from the stockwork zone through the upper part of the basalt and the dacite tuff, and a broader zone of lower copper grades occurs below and to the west in basalt and quartz monzodiorite.

25.1.3 Hugo North Deposit

The highest-grade copper mineralization in the Hugo North Deposit is related to a zone of intense stockwork to sheeted quartz veins. The high-grade zone is centred on thin, east-dipping quartz monzodiorite intrusions or within the upper part of the large quartz monzodiorite body and extends into the adjacent basalt. In addition, moderate- to high-grade copper and gold values occur within quartz monzodiorite below and to the west of the intense vein zone, in the Hugo North gold zone. This zone is distinct in its high gold (ppm) to Cu (%) ratios (0.5:1). Bornite is dominant in the highest-grade parts of the deposit (3% to 5% Cu), and is zoned outward to chalcopyrite (2%). At grades of <1% Cu, pyrite–chalcopyrite ± enargite, tennantite, bornite, chalcocite, and rarely covellite occur, hosted mainly by advanced argillically altered dacite tuff.

Elevated gold grades in the Hugo North deposit occur within the up-dip (western) portion of the intensely veined high-grade core, and within a steeply dipping lower zone cutting through the western part of the quartz monzodiorite. Quartz monzodiorite in the lower zone exhibits a characteristic pink to buff colour, with a moderate intensity of quartz veining (25% by volume). This zone is characterized by finely disseminated bornite and chalcopyrite, although in hand specimen the chalcopyrite is usually not visible. The sulfides are disseminated throughout the rock in the matrix as well as in quartz veins. The fine-grained sulfide gives the rocks a black “sooty” appearance. The red colouration is attributed to fine hematite dusting, mainly associated with albite.

The Hugo North Deposit is characterized by copper–gold porphyry and related styles of alteration similar to those at Hugo South. This includes biotite–K-feldspar (K-silicate), magnetite, chlorite–muscovite–illite, albite, chlorite–illite–hematite–kaolinite (intermediate argillic), quartz–alunite–pyrophyllite–kaolinite–diaspore–zuniyte–topaz–dickite (advanced argillic), and sericite–muscovite zones.

25.1.4 Current Resource Estimation

The current Hugo North, Hugo South, and Southern Oyu resource models were developed using industry-accepted methods and are considered acceptable for incorporation into resource estimates.

The Mineral Resources were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the Project satisfies sufficient criteria to be classified into Measured, Indicated, and Inferred Mineral Resource categories.

25.1.5 Heruga Deposit

The Heruga project, which lies within the Javkhlant MEL, contains a large zone of porphyry copper-gold-molybdenum mineralization that has been subject to ongoing systematic drilling by Ivanhoe Mines.

The copper–gold-molybdenum porphyry-style mineralization at Heruga is hosted in Devonian basalts and quartz monzodiorite intrusions, concealed beneath a deformed sequence of Upper Devonian and Lower Carboniferous sedimentary and volcanic rocks. The deposit is cut by several major brittle fault systems, partitioning the deposit into discrete structural blocks. Internally, these blocks appear relatively undeformed, and consist of south-east-dipping volcanic and volcanoclastic sequences. The stratiform rocks are intruded by quartz monzodiorite stocks and dykes that are probably broadly contemporaneous with mineralization. The deposit is shallowest at the south end (approximately 500 m below surface) and plunges gently to the north.

The alteration at Heruga is typical of porphyry style deposits, with notably stronger potassic alteration at deeper levels. Locally intense quartz-sericite alteration with disseminated and vein pyrite is characteristic of mineralized quartz monzodiorite. Molybdenite mineralization seems to spatially correlate with stronger quartz-sericite alteration.

Copper sulfides occur at Heruga in both disseminations and veins/fractures. Mineralized veins have a much lower density at Heruga than in the more northerly Southern Oyu and Hugo Dummett deposits.

Modelling of mineralization zones for resource estimation purposes revealed that there is an upper copper-driven zone and a deeper gold-driven zone of copper-gold mineralization at Heruga. In addition, there is significant (100–1,000 ppm) molybdenum mineralization in the form of molybdenite. Locally high gold grades (exceeding 50 g/t) appear to be associated with base metal $\pm$ molybdenite in late stage veins.

The Mineral Resource estimate for the Heruga deposit was prepared by Stephen Torr of Ivanhoe Mines under the supervision of Scott Jackson of Quantitative Group. A close-off date of 21 June 2009 for all survey (collar and down hole) and assay data incorporated into the database.

Ivanhoe created three dimensional shapes or wireframes of the major geological features of the Heruga deposit. To assist in the estimation of grades in the model, Ivanhoe also manually created three dimensional grade shells (wireframes) for each of the metals to be estimated. Construction of the grade shells took into account prominent lithological and structural features, in particular the four major subvertical post-mineralization faults. For copper, a single grade shell at a threshold of 0.3% Cu was used. For gold, wireframes were constructed at thresholds of 0.3 g/t and 0.7 g/t. For molybdenum, a single shell at a threshold of 100 ppm was constructed. These grade shells took into account known gross geological controls in addition to broadly adhering to the abovementioned thresholds.

QG checked the structural, lithological, and mineralized shapes to ensure consistency in the interpretation on section and plan. The wireframes were considered to be properly constructed and honoured the drill data.

Resource estimates were undertaken using Datamine® commercial mine planning software. Interpolation domains were based on mineralized geology, and grade estimation based on ordinary kriging. Bulk density was interpolated using an inverse distance to the third power methodology. The assays were composited into 5 m down hole composites; block sizes were 20 m x 20 m x 15 m.

QG also built a model from scratch using the same wireframes and drill data used in the Ivanhoe model. Gold, copper, and molybdenum were interpolated using independently generated variograms and search parameters. QG compared the two estimates and consider they are well within acceptable limits thus adding additional support to the estimate built by Ivanhoe.

The Mineral Resources for Heruga were classified using logic consistent with the CIM definitions required by NI 43-101. Blocks within 150 m of a drillhole were initially considered to be Inferred. A three dimensional wireframe was constructed inside of which the nominal drill spacing was less than 150 m. The shape aimed to remove isolated blocks around drillholes where continuity of mineralization could not be confirmed. Within the 150 m shape there were a small number of blocks that were greater than 150 m from a drillhole. These were included because it was considered that geological and grade continuity could be reasonably inferred within the main part of the mineralized zone. Of the total tonnes classified as Inferred, approximately 95% are within 150 m of a drillhole while the average distance of the Inferred blocks is approximately 100 m.

25.2 MINERAL RESERVE

The Mineral Reserves in this Study are based on pre-feasibility study level work carried out for Oyu Tolgoi in IDOP Technical Report. Measured and Indicated Resources have been converted to Proven and Probable Mineral Reserves. Pre-feasibility level study work, Measured and Indicated Mineral Resources meet the data density adequacy requirements for Mineral Reserve reporting under the NI 43-101 Standards of Disclosure for Mineral Projects of the Canadian Institute of Mining and Metallurgy (CIM). This result meets the main objective of the Study.

26 RECOMMENDATIONS

Recommendations have been stated throughout the text of the report. The key recommendations have been identified in the Future Work section of IDOP. These are:

26.1.1 Detailed Integrated Development and Operating Plan

The next phase in the project planning process is the preparation of DIDOP that builds on IDOP. The DIDOP will integrate a number of detailed capital expansion studies into an overall project report. The key additional work to be assessed in DIDOP includes: the Phase 2 Project Expansion, infrastructure, power supply, water permitting, concentrate marketing, the underground feasibility study and further work on mine closure and reclamation plan.

26.1.2 Phase 2 Project Expansion Plan

A detailed execution plan will be developed for Phase 2 that includes lessons learned and incorporates tools and advancements from the Phase 1 project execution.

OT LLC plans to undertake engineering studies of the concentrator expansion. The focus of the study will be the addition of a third grinding line and other requirements for the Phase 2 plant production.

26.1.3 Power Supply Determination

The supply of power has been recognized as being critical to the execution of the Oyu Tolgoi Project in the IA. OT LLC has been given the right to import power initially but must secure power from sources within Mongolia from the fourth year of operation.

Power is currently assumed in the IDOP Technical Report to be initially imported from Inner Mongolia and sourced from a power station constructed by OT LLC by the fourth year of operation in accordance with the terms of the Investment Agreement.

OT LLC is currently considering a range of options to ensure a reliable and efficient power supply after Year 4. Central to their considerations are two options:

- Option 1 – Build a coal-fired power plant (four 150 MW units) at site. A feasibility study and detailed drawings for power plant with three 150 MW units capacity have already been completed. This study will need to be re-worked to reflect the addition of another generator and auxiliaries.
- Option 2 – Build a coal-fired power plant (four 150 MW units) at the Tavan Tolgoi project site and erect a double-circuit 220 kV transmission line to the Project site. With the power plant at this location, the feasibility study would need to address changes in capacity, water supply, and other design issues, as well the 140 km long 220 kV transmission line.

OT LLC's preferred option is to pursue the accelerated construction of a power station at OT LLC with construction to commence as soon as all stakeholders agree. OT LLC intends to seek approval from the board for an accelerated program if GOM permits are approved.

Currently, there is no clear date for the issuing of those permits and the IDOP Technical Report remains with the assumption that capital is expended by OT LLC to bring a power station into production by the fourth year of operation.

It is expected that the timing of power station construction would be finalised in the preparation of DIDOP.

26.1.4 Water Permit

The current estimate of average water demand for the concentrator expansion to 160 kt/d is 918 L/s, which is marginally above the rate of 870 L/s that has already been approved by the GOM.

OT LLC's strategy is to obtain approval for increases to the currently approved water reserve ahead of any mine expansion plans. The objective of the study will be to assess the impact of the concentrator expansion on water demand and to determine the need for obtaining GOM approval for any substantial increase in the approved water demand from the Gunii Hooloi aquifer.

26.1.5 Concentrate Marketing

OT LLC has developed a marketing plan and currently includes consideration of the following factors:

- Location value to customer compared to imported material landed at Chinese ports.
- Precious metals recovery and payment.
- Length of contract.
- Percentage of off-take to smelters versus traders.
- Percentage of tonnage on contract versus spot.
- Percentage of feed for any one smelter.
- Number of smelters for a given scale of operation.
- Management of concentrate quality and volume during commissioning and ramp-up.
- Alternate off-shore logistics and costs.
- Delivery point and terms.
- Packaging.

A detailed timeline has been developed for marketing, logistics, and contract-to-cash functions. Finalisation of the letters of intents and execution of contracts is planned to be completed in the near future.

26.1.6 Underground Feasibility Study

As part of DIDOP the underground study work is being brought to feasibility study level, the scope of work includes all activities from extraction to the shaft collars and will optimise the development and production of Hugo North Lift 1. OT LLC plans to finalise DIDOP in late 2012.

The design of the underground incorporates 4 shafts and 5 raises to ventilate the mine. The intended method of construction for the 5 ventilation raises is by raise boring. There has proven difficult and has suffered delays, the primary reason for schedule slippage has been due to poor ground conditions. Completion of the ventilation raises is on the critical path to production from the underground mine as the raises are required to meet the ventilation requirements for development and provide alternative means of egress should Shaft 1 be unavailable. In order to mitigate and minimise risk to the project production and to OT LLC is investigating alternative means of establishing the vertical development required to complete the ventilation infrastructure.

Work to determine the impact of this issue is being carried out. Initial analysis by OT LLC suggests that the underground production could be delayed by around twelve months and that each month's delay could have an NPV value of \$50 M. These investigations are planned to be incorporated into DIDOP and will allow OT LLC to quantify potential costs and changes required to meet the Project schedule.

In addition possible scope changes have been indicated including:

- Additional development and excavation expenditures
- The addition of a shaft No 5
- An additional crew underground
- Increased operating cost due to design changes

26.1.7 Socio-economic Aspects of Mine Closure Plan

The preliminary mine closure and reclamation plan includes provisions to ensure that adverse socio-economic impacts of mine closure are minimized and positive impacts are maximized. To this end, OT LLC has planned that allowances will be incorporated into the annual mine operations budget starting 10 years before mine closure to address the costs of:

- Lost employment by the mine workforce.
- Adverse effects on supply chain businesses and downstream businesses, affected communities, public services, and infrastructure.
- Promoting ongoing sustainability among affected stakeholders and communities.

The details of additional socio-economic aspects of a conceptual mine closure plan have not yet been fully developed and are the subject of work to be done in the near future.

26.1.8 Infrastructure

As part of the preparation of DIDOP, work is continuing on further defining the ongoing infrastructure needs for the project. OT LLC has advised that it expects future additions to the project scope which may include:

- Operations camp expansion
- Border facilities upgrade
- Concentrate bagging plant upgrade
- Power substation expansions
- Central maintenance complex
- Central control room
- Borefield expansion
- Operations warehouse expansion
- Core storage warehouse

There may be additions to scope beyond these items and all items and updated cost estimates will be included in DIDOP.

27 REFERENCES

References to other reports and information have been identified throughout the report in the sections to which the references relate.